BRANCH OF THE FEDERAL STATE AUTONOMOUS EDUCATIONAL INSTITUTION OF HIGHER EDUCATION

"National Research Technological University "MISIS" in Almalyk

(Branch of NUST MISIS in Almalyk)

DEPARTMENT OF "MINING"

EDUCATIONAL AND METHODICAL COMPLEX BY SUBJECT

"Technology and complex mechanization of open-pit mining"

(for students of the field of education: specialty
210504 - "Mining", profile: open-pit mining)

Almalyk-2022

МИСиС
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АЛМАЛЫК / филиал

Made up:

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Educational and methodical complex of the subject

Technology and complex mechanization of open-pit mining

Developed in accordance with the OS in:

Independently established educational standard of higher education Federal State Autonomous Educational Institution of Higher Education "National Research Technological University "MISIS" in the specialty 21.05.04 MINING (Order No. 602 O.V. dated 02.12.2020)

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Introduction

This work program covers information about a set of interrelated processes, methods and techniques of mechanized mining, based on fundamental knowledge of the laws of development and capabilities of technical means.

The technology of the open-pit mining method includes two aspects: the technology of production processes (excavation, movement and storage of rocks) and the technology of open-pit mining (construction and development as the field is developed in time and space of the quarry as a mining complex)

The main purpose of teaching the discipline is the formation of knowledge and skills of students in the field of technology and complex mechanization of open-pit mining.

The objective of the course is to develop students' skills and abilities to calculate the main elements and parameters of a quarry, the formation of cargo flows, the choice of rational methods of opening and development systems, the selection and calculation of complexes of basic mining and transport and auxiliary equipment.

As a result of studying the discipline, students should master mining terminology, get an idea of the structure of world mining, the current state of the mineral resource base of Uzbekistan and mining production, methods of field development, features and main scientific and technical problems of open-pit mining.

Knowledge should be obtained about the properties of rocks – objects of development, about methods of calculating parameters and indicators of technological processes, the basics of completing equipment complexes, the main characteristics of modern and promising mining equipment, about the features of opening quarry fields and justification of development systems.

For students of the specialty "Open-pit mining", the acquired complex of knowledge is only a necessary basis for studying the main special disciplines "Processes of open-pit mining", "Technology and complex mechanization of open-pit mining", "Quarry Design", "Planning of open-pit mining", during which appropriate skills and abilities will be acquired.

Students of related specialties face a somewhat different task. It is during the study of this discipline that they should acquire the skills of operational calculations of mining and transport machines in various technological schemes, justification of their choice for specific mining and geological conditions; formation of technological cargo flows, schemes for opening and designing development systems, practical analysis of technological processes as control objects; justification of decisions made. The acquired skills and abilities will be consolidated during the course design.

BRANCH OF THE FEDERAL STATE AUTONOMOUS EDUCATIONAL INSTITUTION OF HIGHER EDUCATION
"National Research Technological University "MISIS" in Almalyk

COLLECTION OF LECTURES
on the subject

"TECHNOLOGY AND COMPLEX MECHANIZATION OF OPEN-PIT MINING"

Lecture 1

Introduction. general information about the technology of open-pit mining.

The purpose of the lesson: to give general information about the technology of mining in an open way and
to outline the goals and objectives of this course.

Plan:
1. Introduction. The subject and objectives of the discipline.
2. The essence of open-pit mining.
3. Basic concepts. Terminology.

The economic development of our country is inextricably linked with the further development of the mining industry. Our Republic has large reserves of minerals. Currently, the largest quarries for the extraction of gold, silver, copper and coal are operating. The development of open-pit deposits in comparison with underground provides significantly better technical and economic indicators. At the same time, it is associated with a number of negative consequences: land disturbance, changes in microclimate and water balance, etc.

The rapid development of open-pit mining became possible thanks to the achievements of mining science and technology, which are based on the works of N.V. Melnikov, V.V. Rzhevsky, A. O. Spivakovsky, E.V. Sheshko, M.G.Novozhilov, A.I.Arsentiev, B.P. Yumatov, G.A. Nurka, P. I. Tomakov, V.S. Khokhryakov, Yu. I. Anistratov and many others.

The term "Technology" generally means a set of knowledge about the methods, means and organization of performing any production and technical work.

The technology of field development is a set of interrelated processes, methods and techniques of mechanized mining, based on fundamental knowledge of the laws of development and capabilities of technical means.

The technology of the open-pit mining method includes two aspects: the technology of production processes (excavation, movement and storage of rocks) and the technology of open-pit mining (construction and development as the field is developed in time and space of the quarry as a mining complex).

The technology of production processes includes the principles, means, mechanization complexes and organization schemes of the main production processes: preparation of rocks for excavation, excavation and loading operations, transportation, transshipment, warehousing and unloading of rock mass.

Technology and complex mechanization of open-pit mining considers the parameters of quarries in their dynamics, methods of mining workings, schemes for the development of mining operations in a quarry, methods of opening and development systems, methods and means of product quality management, organization and planning of mining operations in a quarry.

Mining operations, in which all the main production processes that ensure the extraction of minerals from the bowels of the Earth are carried out in open-pit mine workings, are called open-pit mining. A mining enterprise engaged in the extraction of minerals by open-pit mining is called a quarry. In the practice of open-pit mining of coal deposits in placer deposits, the names "cut" and "mine" are used instead of the term "quarry".

During the production of open-pit mining, the natural surface of the mineral deposit is disturbed and a recess is formed in the Earth's crust, limited by an artificially created surface. This excavation, representing the totality of individual mine workings, is also called a "quarry". Thus, the concept of "quarry" can be used in two meanings — economic and technical. The formation of significant recesses in the Earth's crust (modern quarries reach a depth of several hundred meters) disrupts the natural balance of the rock mass surrounding the quarry. The redistribution of internal stresses in the array in this case can lead to undesirable deformations of the side surface of the quarry (landslides and collapses), which leads to disruption of normal mining operations and can cause accidents and accidents. To avoid this, the side surfaces of the quarry are given a certain angle of inclination, which ensures their stability. In this regard, there is a need to excavate significant volumes of rocks covering and containing minerals, which are called overburden or overburden. The annual volumes of overburden moved in modern quarries amount to tens of millions of cubic meters and often many times exceed the volumes of extracted minerals. Minerals and overburden are removed from the quarry to the surface. In favorable conditions of occurrence of mineral overburden, separated from the array, may not be removed from the quarry, but placed in its developed space.

The development of an array of rocks (overburden and minerals) within the boundaries of the quarry is carried out by horizontal or slightly inclined layers. Layers are usually worked out in parallel with some lag of work in space and time on the underlying layer. Thus, the side surface of the quarry acquires a stepped shape. The need to divide the developed rock mass into layers is determined by the following factors:

limited parameters of mining machines engaged in the excavation (development) of rocks;

the presence in the developed array of layers having various physical, mechanical and qualitative characteristics;

increased risk of collapse of an exposed rock mass of considerable height.

The significant size of open-pit mine workings and the developed space and the absence of height restrictions create favorable conditions for the use of powerful mining and transport equipment in open-pit mining operations, providing high technical and economic indicators. Effective use of the equipment is possible only with a clear organization of the work of all parts of mining production and the availability of highly qualified personnel.

There are excavator and hydraulic methods of production of open-pit mining. In the excavator method, various equipment is used — excavators, scrapers, bulldozers, wheeled and conveyor transport. With the hydraulic method, the main production processes are carried out by the energy of moving water. For this purpose, special equipment is used — hydraulic monitors, dredgers, etc. The excavator method is universal. It can be effectively applied in any conditions. The hydraulic method of development is used only in favorable geological and climatic conditions (mainly in the development of loose rocks in the presence of sufficient water and areas for the placement of empty rocks in hydraulic dumps).

When conducting open-pit mining, the side surface of the quarry acquires a stepped shape. Part of the side surface of the quarry, which has the shape of a step, is called a ledge (Fig. 1.1). The surface bounding the ledge from above or below is called the upper or lower platform of the ledge. The vertical distance between these platforms is called the height of the ledge. The platforms of the ledges are called horizons. Each horizon in the quarry is characterized by an absolute or conditional mark (for example, the horizon +135 has an absolute mark of 135 m above sea level). Horizon +135, on which the transport routes are located, is the transport (working) horizon (Fig. 1.2).

The ledge is one of the main technological elements of the quarry and the efficiency of production processes largely depends on the correct determination of its height marks in the thickness of the developed array. When dividing the thickness of the developed massif into ledges, it is necessary to take into account both the operating parameters of mining equipment and the physico-mechanical properties of the composing rocks, their conditions of occurrence, mining-geological and climatic conditions of the deposit. The defining feature of the ledge is the presence of a transport horizon. The height of the layer, worked out for one transport horizon, is one ledge. When the transport horizon is located in the middle of the ledge, the latter is divided into two approaches — upper and lower (see Figure 1.2).

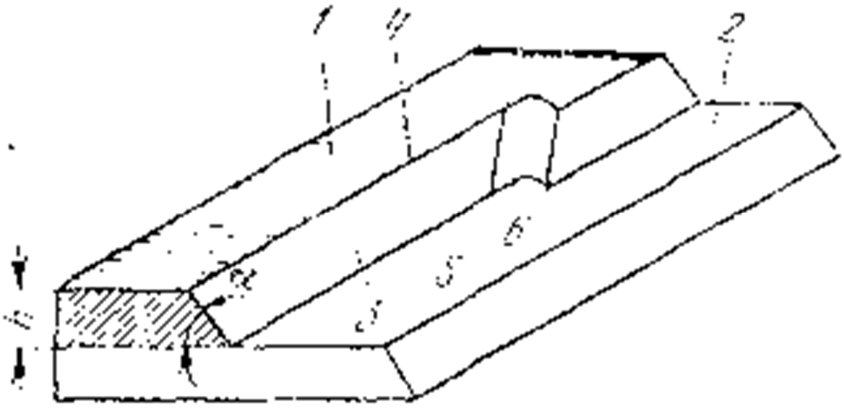

Figure 1.1. The ledge diagram:

1 – the upper platform of the ledge; 2 – the lower platform of the ledge; 3 – the slope of the ledge; 4 – the upper edge of the ledge; 5 – the lower edge of the ledge; 6 – the face of the ledge; h – the height of the ledge; α – the angle of the slope of the ledge.

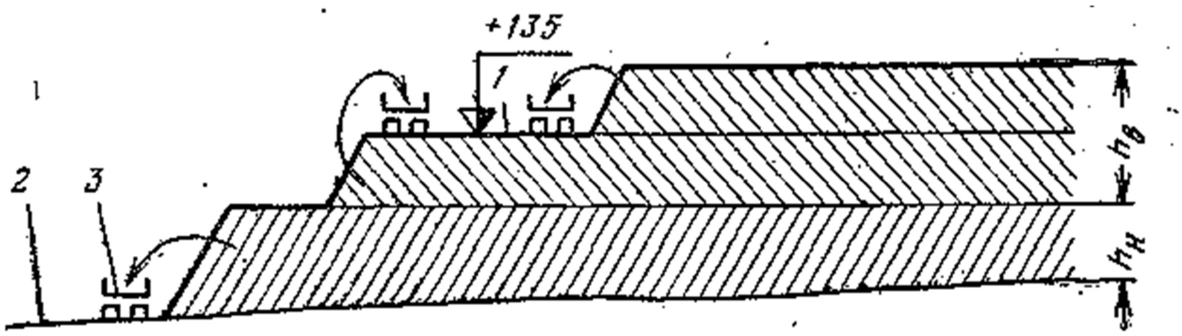

Figure 1.2. Cross section of two ledges:

1 – working platform of the upper ledge; 2 – working platform of the lower ledge; 3 – vehicle; hb – height of the upper ledge (the upper ledge is divided into two steps); hh – height of the lower step.

The platform of the ledge, on which the main equipment for its development is located, is called the working platform of the ledge. The width of the working platforms is usually several times (two to four) higher than the height of the ledge.

The site where the work is not carried out is called a berm. Depending on the purpose, there are safety and transport (connecting) berms. Safety berms are designed to increase the stability of the side surface of the quarry and to detain pieces of rock falling from the slopes of ledges. The width of these berms is 20-30% of the height of the ledge. In soft rocks, safety berms are left every 15 m vertically. In rocks, it is possible to leave wider berms (cleaning berms) after 30 m vertically. In this case, their width is at least 6 m. Bulldozers, small excavators, loaders are used to clean such berms. Transport berms are designed to accommodate transport communications of the quarry. Their width is determined by the type of vehicles and the intensity of their movement.

The inclined surface bounding the ledge from the side of the worked-out space is called the ledge slope. The line of intersection of the slope of the ledge with its upper and lower platform is called the upper and lower eyebrows, respectively. The angle of the slope of the ledge is called the angle formed by the slope of the ledge and the horizontal plane (see Fig. 1.1). Depending on this angle, the slope of the ledge may have a stable or unstable position. The stability of the slope of the ledge is crucial in ensuring safe mining conditions. There are angles of short-term and long-term stability of slopes of ledges. The angle of short-term stability ensures a stable position of the slope of the ledge for a short period (several months), which is sufficient for the safety of work on workers constantly moving ledges. The angle of long-term stability of the slopes of the ledges should ensure their stability for almost the entire lifetime of the quarry, which is typical for ledges that are not in operation.

The conditions for ensuring long-term stability are less favorable, since in this case, with prolonged exposure to external factors (precipitation, wind, variable temperature, etc.), an intensive decrease in their mechanical strength occurs on the rocks.

The slope of the ledge, which is in short-term stability, collapses over time and acquires a slope angle corresponding to long-term stability for this type of rocks and their conditions of occurrence. Density, adhesion and coefficient of internal friction are the main physical and mechanical characteristics of rocks that determine the stability of the slope of the ledge. The permissible slope angle of the ledge is also influenced by the height of the ledge (Table 1.1).

Of great importance for the stability of the slope of the ledge is the position of the planes of stratification of the rocks composing the ledge. If the planes of the stratification are located at some angle to the horizon, then in order to increase the stability of the slope of the ledge, they tend to work out the ledge so that the planes of the stratification have a fall in the opposite direction from the slope.

When developing loosened rocks to ensure a stable slope position, its angle should not exceed the angle of the natural slope. This condition is true in the development of dry rocks. The presence of water in the rocks of the ledge requires a reduction in the slope angle of the ledge by 10-20 ° or more (Table 1.2).

Table 1.1

Breeds	Slope angle (degrees) at the height of the ledge, m			
	5-12		15-25	
	Working ledge	Non-working ledge	Working ledge	Non-working ledge
Greasy clay, light loam, gravel, loess, vegetable soil, sand, sandy loam with crushed stone	40-50	30-40	32-45	25-35

Heavy clay, heavy loam with an admixture of pebbles and crushed stone, clay with boulders, shale clay, large pebbles with cobblestones, developed without loosening by drilling and blasting	45-65	40-55	45-60	40-50
Also, developed using drilling and blasting loosening	55-65	40-55	50-60	40-50
Ordinary sandstones, strong clay shale, weak limestones, dense marl, iron ores, soft conglomerates	65-75	60-65	60-70	55-60
Granite rocks and granites, very strong sandstones and limestones, quartz ore veins, pyrites, strong marble and dolomites	75-80	70-75	75-80	70-75
Quartzites, basalts, granites, quartz rocks, the strongest sandstones and limestones	До 90	80-85	До 90	75-80

Table 1.2

Породы	Angle of natural slope (degrees) for breeds		
	dry	wet	damp
Plant layer	40	30-35	20
The sand is large	32-35	32-40	20-27
Medium sand	28-32	32-35	20-25

Fine sand	25-30	30-35	12-20
Loam	40-50	35-40	20-30
The clay is greasy	40-45	35	12-20
Gravel	35-40	35	15-20
Peat without roots	40	25	10-15

When using analytical methods for determining the permissible angles of the slopes of ledges, they proceed from the need to ensure a certain margin of their stability, which is characterized by a coefficient of stability margin pu. The latter is understood as the ratio of holding and shifting forces for the upper part of the ledge (collapse prism), prone to collapse. The coefficient of stability margin is assumed to be equal to 1.1—1.2 and 1.5—2, respectively, for short-term (working ledges) and long-term (non-working ledges) stability.

The part of the slope of the ledge that serves as the object of the impact of mining equipment during its development is the face of the ledge (see Figure 1.1).

Reference words: technology, Technology and complex mechanization of open-pit mining, quarry, mining, overburden, preparation of rocks for excavation, excavation and loading operations, transportation, dumping, ledge, access, face, work site, vehicle, ledge height.

Lecture 2

Topic: Types of developed deposits and deposits. Types of open-pit mining

Plan:

1. Types of deposits being developed.

2. Types of open-pit mining.

Types of fields and deposits being developed

The objects of open mining are mineral deposits. Open-pit mining of coal and ore deposits, deposits of building rocks, cement raw materials, mining and chemical raw materials, etc. are distinguished by industry.

The mineral deposits being developed are located in a very diverse natural environment.

The types of deposits differ primarily in their characteristic geometric features.

1. Mineral deposits in the form can be:

isometric - developed more or less equally in all directions (massive deposits, stocks, nests, etc., fig. 2.1, b, h);

plate-shaped - elongated mainly in two directions at relatively low power (layers and formation-like deposits, Fig. 2.1, a, b, d, g);

tube-shaped and columnar - elongated mainly in one direction;

intermediate and transitional between these forms (lenses, veins, saddle-shaped deposits, folds, kinks, tectonically disturbed formations of layers) (Fig. 2.1, d, e).

The shape of deposits determines the shape of quarry fields.

2. The relief of the surface of the deposit can be flat (see Figure 2.1, a), in the form of a slope of elevation (see Figure 2.1, b), in the form of elevation (see Figure 2.1, c), hilly (see Figure 2.1, d) and, finally, the deposit can be under water. The order of development and possible means of mechanization depend on the relief of the surface.

3. Depending on the position relative to the prevailing surface level and depth of occurrence, deposits are distinguished:

Surface type - directly coming to the surface or located under sediments of low power (up to 20-30 m, see Fig. 2.1, a);

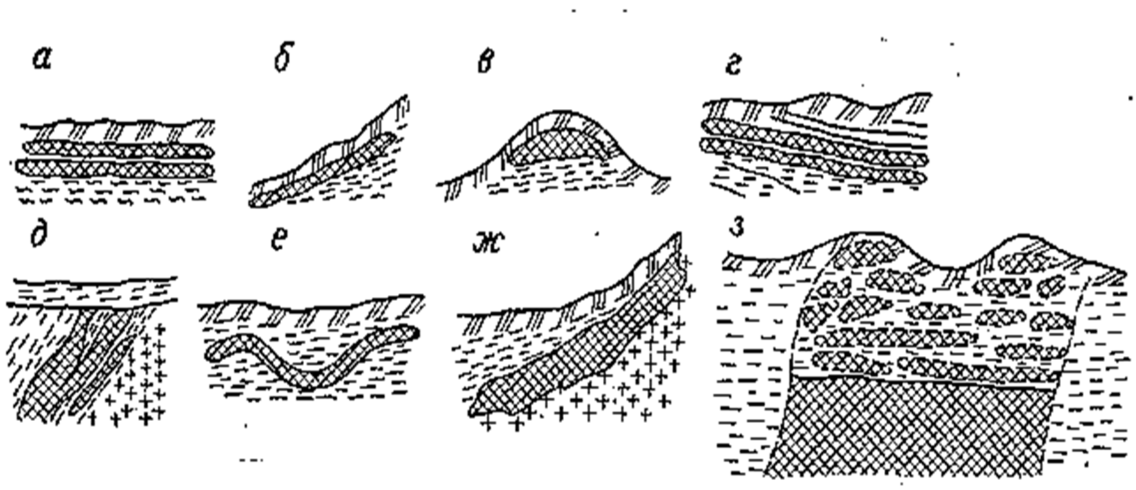

Fig. 2.1. Schemes of developed deposits and deposits.

deep type - located well below the prevailing surface level, the thickness of the thickness of the empty rocks can range from 40 to 250 m (see Fig. 2.1, d, e); such deposits can be developed in an open or underground way, which is economically justified;

high-altitude type - located above the dominant surface level (see Fig. 2.1, b, c); deposits can be objects of open or underground development;

altitude-depth type - partially located above and below the dominant surface (see Fig. 2.1, g).

The occurrence may be in agreement or disagreement with the relief of the surface; the deposit may occupy all or part of the elevation (mountain slope). The size of the quarry in depth and in plan, as well as the technical means used, especially transport, depend on the position of the deposit relative to the Earth's surface.

4. According to the angle of incidence, deposits are distinguished:

flat, characterized by slightly inclined (up to 8-10 °) and wavy occurrence of the main part of the deposit (see Fig. 2.1, a, d); their special case is horizontal deposits;

inclined - with angles of incidence from 8-10 to 25-30 ° (see Fig. 2.1, b);

steeply inclined - with angles of incidence of more than 25 - 30° (see Fig. 2.1, g);

steep - with angles of incidence 56 - 90° (see Fig. 2.1, d);

complex occurrence, characteristic of anticlinal and synclinal folds (see Fig. 2.1, e) and sharp geological disturbances; it is characterized by a variable direction of fall of the deposit.

Such separation of deposits is accepted on the basis of open-pit mining technology. Thus, the placement of dumps in the worked-out space of the quarry is possible during the development of horizontal and shallow deposits (Fig. 2.2, a) and in special cases - during the development of elongated inclined and steeply inclined deposits. When developing inclined deposits, according to the conditions of stability of the end sides of the quarry and the placement of opening workings, it is usually not necessary to excavate the overburden rocks of the lying side of the deposit (Fig. 2.2, b). In case of a steep fall, it is necessary to develop the host rocks of both the hanging and lying sides of the deposit (Fig. 2.2, c).

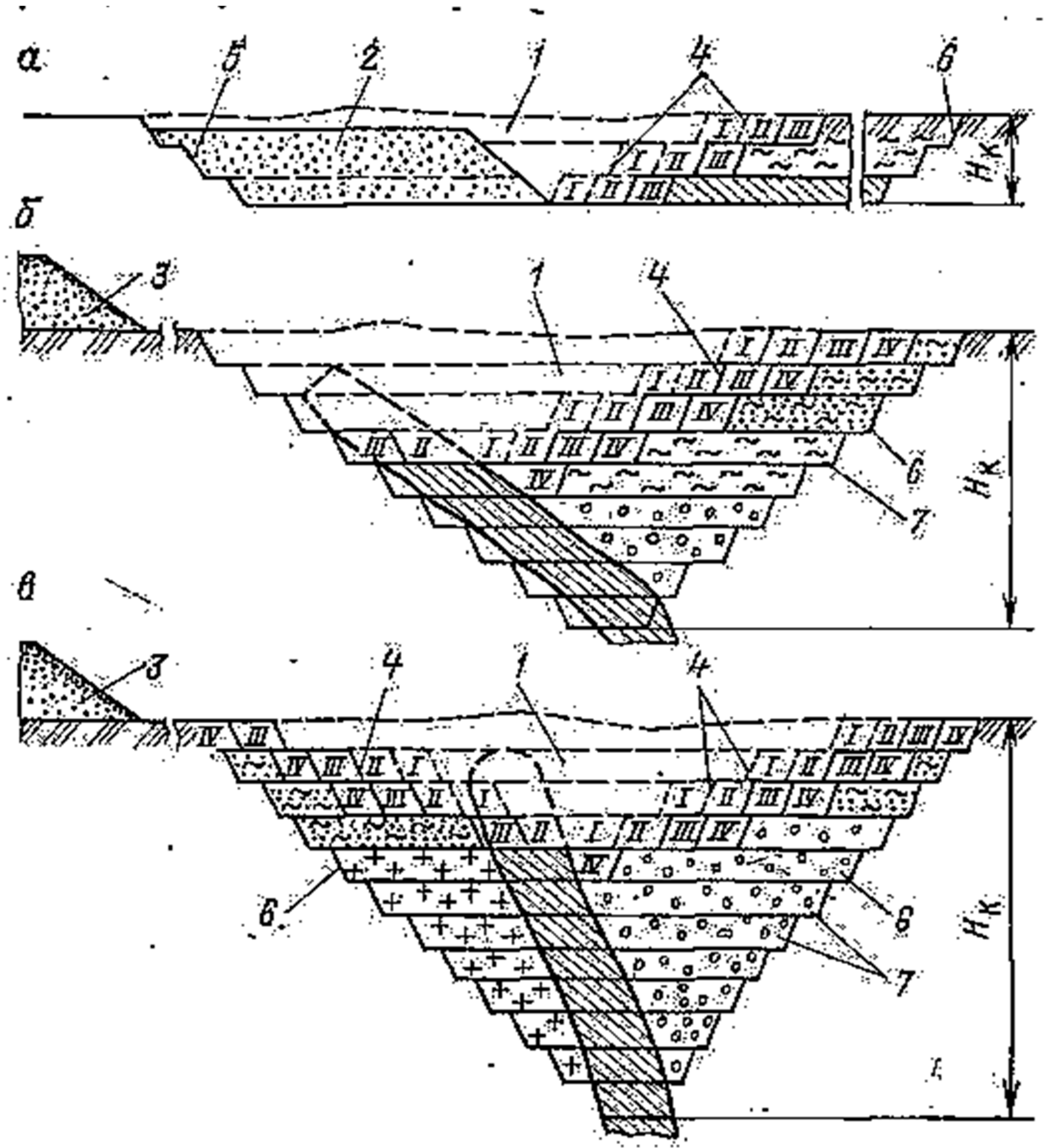

Fig. 2.2. Schemes of open-pit mining:

1 - the developed space; 2 and 3 - internal and external dumps, respectively; 4 and 5 - working and non-working board, respectively; 6 - the final contour of the quarry; 7 - berms; I - IV - the sequence of development of work on ledges.

5. According to the capacity of the deposits are divided into very low-power, low-power, medium-power, powerful and very powerful. This separation is due to the dependence of the number of simultaneously worked mining ledges on the capacity of the deposit. The conditions and procedure for the development of horizontal and inclined (steeply inclined) deposits are not the same, therefore, the indicators of the same power classes are numerically different for these deposits.

6. The structure of the deposit. On this basis , there are:

simple deposits (see Fig. 2.1, b, g) with a homogeneous structure, without significant layers and inclusions; in this case, all the minerals of the deposit are removed together (gross method of excavation);

complex deposits (see Fig. 2.1, a, d) containing, along with conditioned minerals, substandard grades of it, as well as layers or inclusions of waste rocks with clearly defined contacts; in this case, separate (selective) development of conditioned and substandard minerals and waste rocks is necessary;

dispersed deposits (see Fig. 2.1, h) having a complex structure, in which conditioned and substandard minerals and waste rocks are distributed in the thickness of the earth's crust without a clear pattern and pronounced contacts; the choice of a separate or gross method of extraction of minerals is made after detailed operational exploration.

7. The quality of the mineral in the deposit can be distributed:

evenly, when the quality of the mineral that meets the requirements of the consumer is approximately the same within the deposit; in this case, excavation (gross or separate) in different areas of the deposit can be carried out independently, without averaging;

it is uneven when the quality distribution is not the same in depth or in terms of the deposit; in this case, it is necessary to plan simultaneous excavation in different parts of the deposit, have several working excavation sites and average the quality.

8. According to the prevailing types of rocks, deposits can be represented by:

rock overburden and minerals;

heterogeneous covering rocks and rocky (semi-horizontal) minerals and host rocks; in this case, the thick layer covering the deposits is represented by alternating soft, dense, semi-horizontal and rocky rocks;

soft and dense covering rocks and rock , or semi - rock minerals and host rocks;

semi- fossilized overburden rocks and semi -fossilized or very dense minerals;

soft overburden rocks and heterogeneous minerals;

soft overburden rocks and soft or dense minerals.

These factors have a decisive influence on the choice of technical means, the procedure for conducting and the possibility of open-pit mining.

Types of open-pit mining

The main types of open-pit mining are classified according to the position of the deposit relative to the surface (Fig. 2.3).

1. Surface view developments. These include the majority of placer developments, natural building rocks, a significant part of coal and a small part of ore developments in horizontal and shallow deposits. The quarries are shallow (up to 40-60 m) and have a relatively constant depth. Overburden rocks and minerals are diverse, often soft and semi-hard.

2. *Development of a deep view. These include most of the ore and partially coal mining with an inclined and steep fall of deposits. In this case, the quarries gradually deepen; their final depth can reach 800 m. All types of rocks are developed in such quarries.*

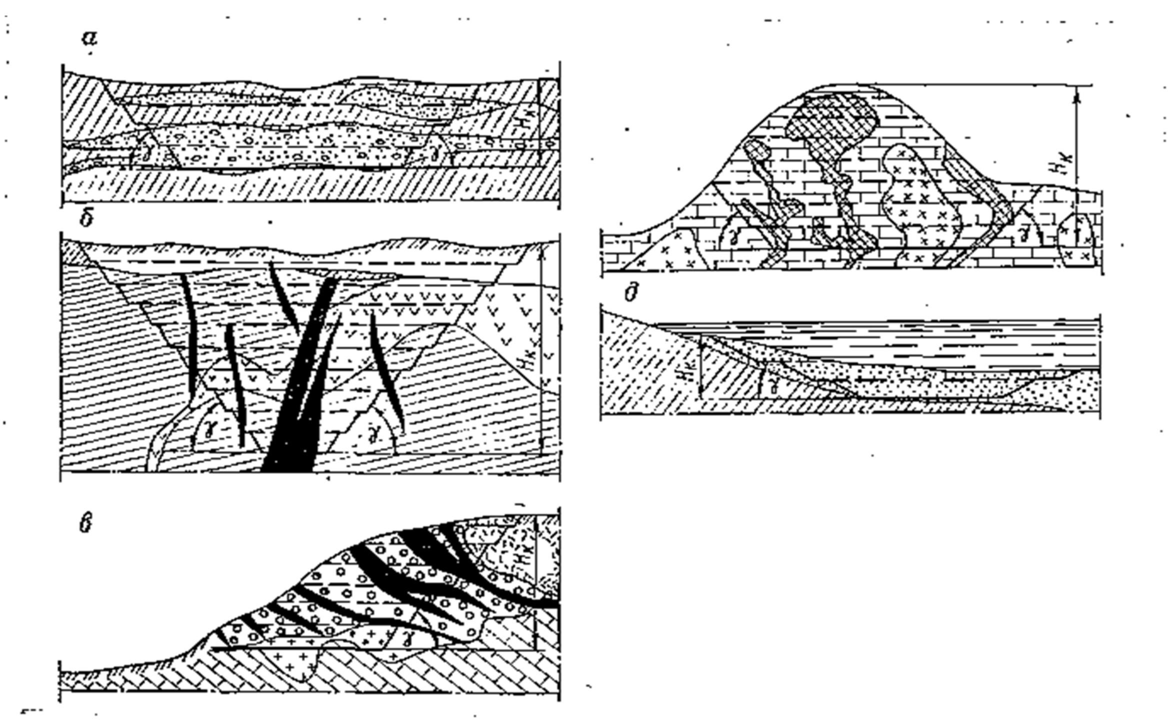

Figure 2.3. Schemes of open—pit mining:

a, b, c, d and d - surface, deep, upland, upland-deep and underwater, respectively.

3. Developments of the upland view. These mainly include open-pit mining of various ores, mining and chemical raw materials, construction rocks and sometimes coal. The deposits are located well above the prevailing surface level; the number of working ledges and the size of the quarries are varied in terms. Minerals and overburden rocks are mainly rocky.

4. Developments of the upland-deep view. These include open-pit mining of various ores, mining and chemical raw materials, construction rocks and coal mining with a complex relief of the surface of the quarry field. Minerals and overburden rocks are rocky or semi-rock, sometimes heterogeneous.

5. Underwater view developments. The deposits are located under water, the covering rocks usually have a relatively small capacity. This type includes, in particular, developments in floodplains of rivers and from the bottom of seas and lakes. The rocks are soft, dense, semi-oval or heterogeneous.

Each of these types of open-pit mining differs from the other in the preparation of the deposit for operation, the order of its development, the opening of working horizons, the location of dumps and, accordingly, the nature of complex mechanization of mining operations.

Developments of the first type are the most economical. At the same time, the extraction of minerals is carried out immediately at full capacity and overburden rocks are placed in the developed space.

Overburden and mining of mineral deposits during deep-type mining are carried out in layers in descending order. The rock mass, as a rule, is moved up to the surface, and the overburden is stored in external dumps. Mining and preparatory work is preceded by the development of each new layer, ensuring the opening of working horizons. The depth of the quarry gradually increases to the limit determined by the boundaries of the career field.

Open-pit mining of the upland type is characterized by the movement of covering and enclosing rocks and extracted minerals using transport down to the location of dumps and processing complex.

The development of deposits of the upland-deep type has the characteristic features of the second and third types of open-pit mining.

The types of deposits (with flat terrain) developed by the open method, in relation to rounded, elongated and elongated forms and the general angle of their incidence with the corresponding letter designations are shown (for educational purposes) in Fig. 2.4 and 2.5.

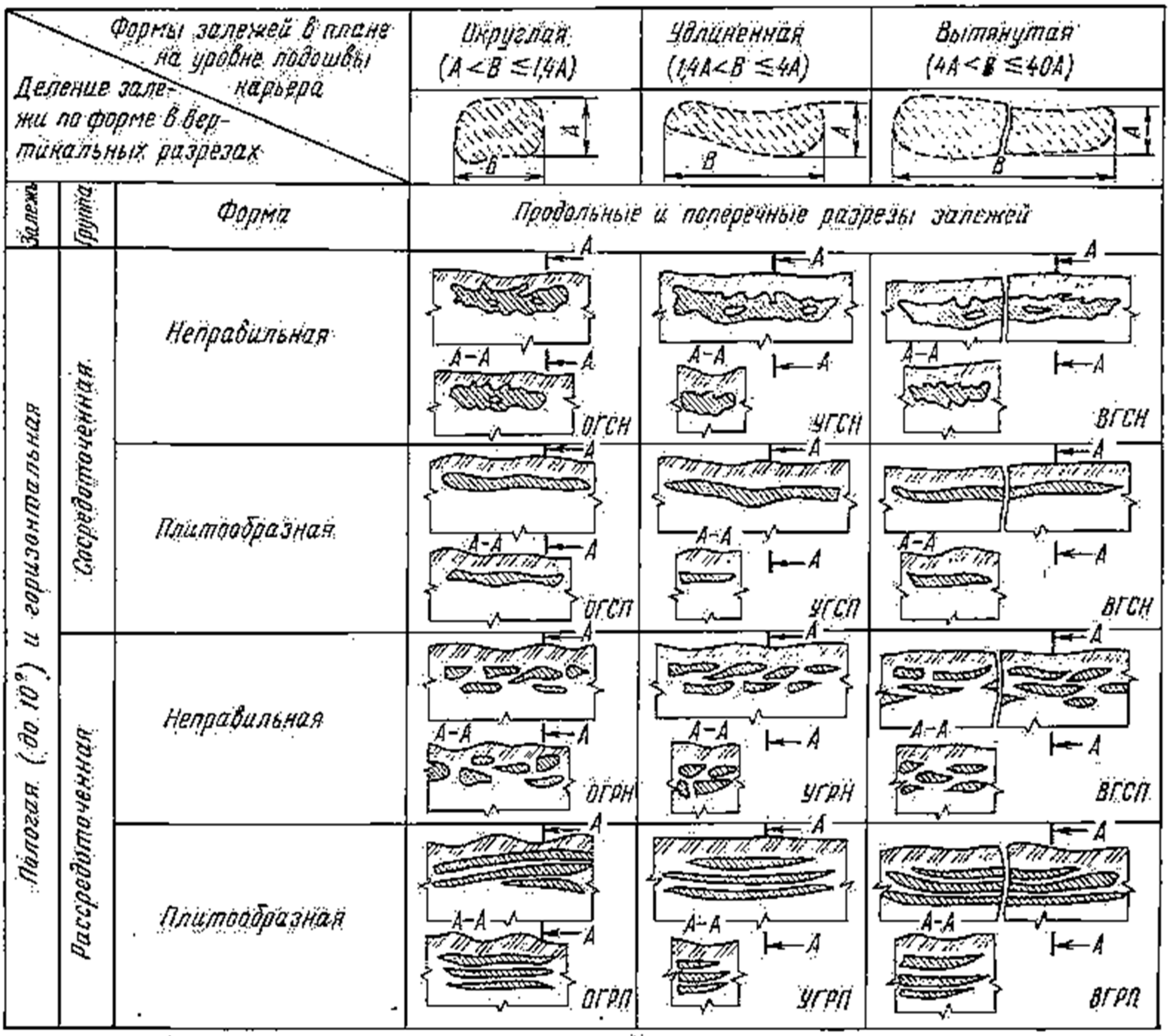

Figure 1.4. Schemes of horizontal and shallow deposits developed by the open method:

O - rounded; U - elongated; B - elongated; G - horizontal; C - concentrated; H - irregular; P - tile-shaped; P - dispersed; UGSP - elongated horizontal concentrated irregular shape; UGSP - elongated horizontal concentrated tile-shaped; UGRN - elongated horizontal dispersed irregular shape; OGRP - rounded horizontal dispersed tile-like.

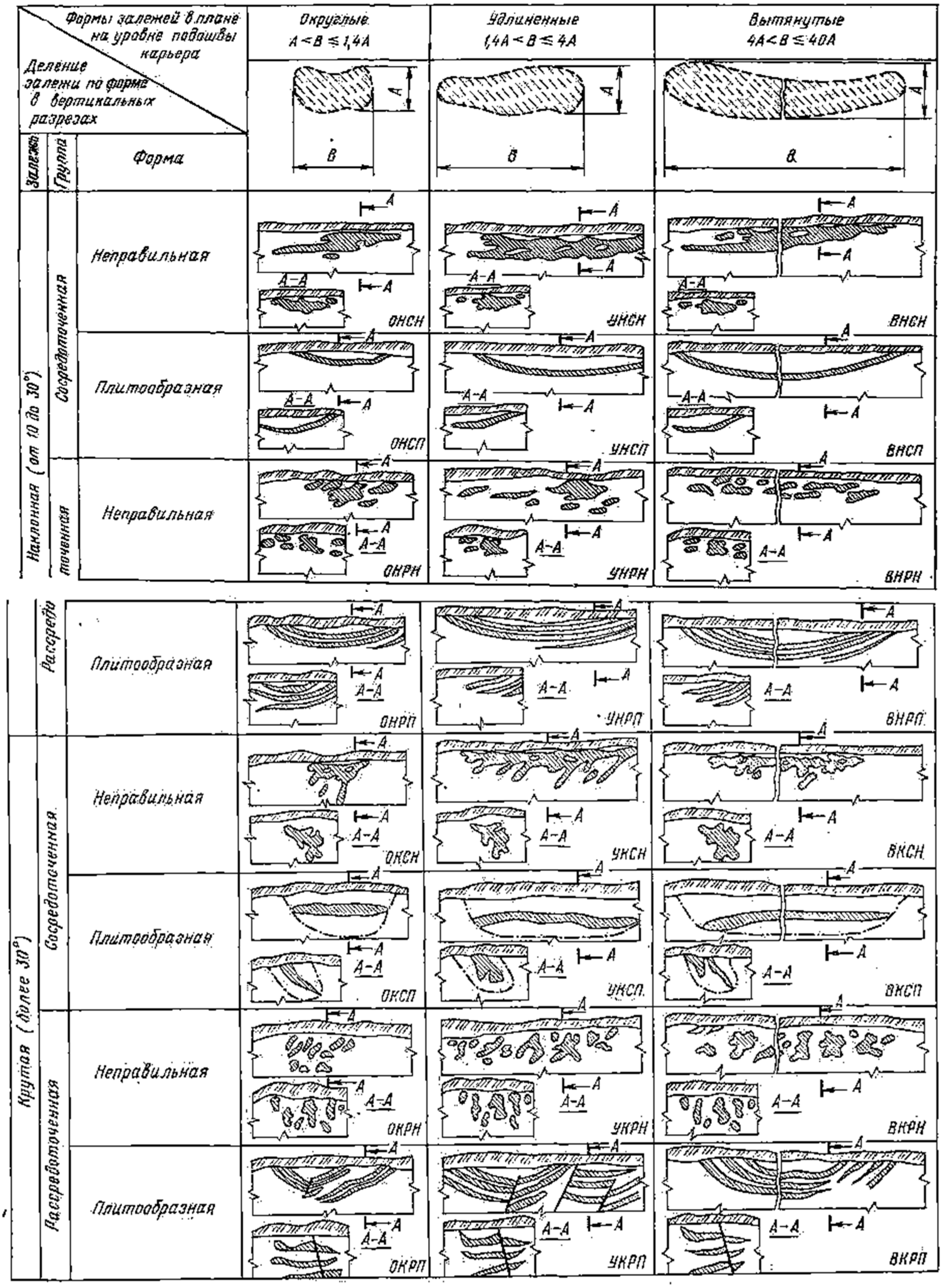

Figure 2.5. Schemes of inclined and steep deposits developed by the open method:

ONSN – rounded oblique concentrated irregular shape; UNSP – elongated oblique concentrated tile–shaped; WRN - elongated oblique dispersed irregular shape; the other symbols are the same as in Figure 2.4.

Reference words: isometric, plate-shaped, tube-shaped and columnar, intermediate and transitional, surface, deep, high-altitude, high-altitude, shallow, inclined, steeply inclined, steep, complex occurrence, low-power, medium power, powerful, very powerful, simple, complex, dispersed, upland, upland-deep, underwater.

Security questions:

1. What are the deposits classified by shape?
2. What determines the shape of the deposits?
3. According to the angle of incidence, deposits are distinguished…
4. Name the main types of open-source development.
5. What is the essence of surface view development?

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Lecture 3

Topic: Types and periods of mining operations. The procedure for the development of open-pit mining.

Plan:
1. Surface preparation and drainage of the rock mass.
2. Mining and capital works.
3. Mining and construction works.
4. Operational mining operations.
5. Reconstruction of the quarry economy.
6. The procedure for the development of mining operations.

Types and periods of mining operations

The development of new deposits or the next sections of the quarry field begins with the preparation of the surface. It consists in carrying out special, sometimes expensive and large-scale engineering works to divert rivers, streams, in some cases lakes, deforestation and uprooting of stumps, fencing of a quarry field from surface water runoff through a network of drainage ditches. Surface preparation also includes the removal and storage for subsequent use of the soil layer, surface leveling, the creation of special sites for the installation of mining equipment, the construction of primary access roads or railways to mountain areas and dumps.

Usually, at the same time as surface preparation, special work is carried out to drain the rock mass within the quarry field or individual sections. If necessary, if the rocks are prone to landslide and collapse, special work is carried out to strengthen the instrument sections of the rock mass.

Surface preparation and drainage of the deposit, completed in whole or in part, allow you to start mining and capital works. These include the removal of covering rocks, the creation of capital, split trenches and pits, as well as embankments that allow the systematic production of stripping and mining operations to begin in strict accordance with the project.

Mining and capital works performed during the construction of a quarry before its commissioning are called mining and construction works. They also include mining operations during the construction of a quarry (associated mining) and a complex of works on the construction of transport communications.

Economic features of mining and construction works:

all costs of mining and construction work relate to capital investments;

the unit costs of mining and construction work (by 1 m3) are higher than for mining operations during the operation of the quarry, especially after it reaches the design production capacity.

With this in mind, it is advisable to carry out mining and construction work in the minimum volume that is necessary to ensure the extraction of minerals: either in the volume of the full design capacity of the quarry, or part of this capacity (most often from 30 to 60%) provided for in the approved project.

Operational mining operations are divided into:

stripping operations, consisting in the excavation and transfer of waste rocks and substandard minerals to dumps with the creation of mineral reserves prepared for development and uncovered;

mining operations, consisting in the extraction and delivery of the extracted mineral to warehouses or to the consumer.

The structure of operational mining operations also includes work on the stripping of uncovered mineral reserves, the arrangement of transport communications, carrying out regular sections of split trenches on uncovered ledges to increase the length of the front of mining and stripping operations and work on the development of the dump economy of the quarry.

Mining and capital works are financed according to the project by the Construction Bank in accordance with the procedure established for facilities under construction; operational mining works are financed by the industrial bank in accordance with the procedure established for operating enterprises.

After the quarry is put into operation with incomplete design capacity, all mining operations are operational or, along with operational work, mining and capital work continues to be carried out simultaneously on the next sections of the quarry field. As the length of the mining front increases, the next stages of the quarry are put into operation, prepared for development and uncovered mineral reserves. Thus, the production capacity of the quarry is gradually increasing to the design level. The period from putting the quarry into operation until it reaches its design capacity is often called the period of mastering the design capacity of the quarry. With a change in the input capacity (full design or a separate stage — the launch complex for minerals) from 5 to 30 million tons / year or more, the standard period for its development increases from 9 to 24 months.

The work on the creation of opening and split mine workings is called mining preparatory. Depending on the period of the quarry's activity (construction or operational) and the source of financing (capital investments or at the expense of the main activity of the operating enterprise), mining preparatory work refers to mining capital or operational work. In some cases, mining and preparatory work carried out during the operational period after the development of the design capacity of the quarry is classified as mining and capital.

The mining and capital works also include those continuing during the operational period related to drainage, in particular, drilling of the next water-lowering wells, construction of underground drainage workings and roads.

When establishing additional exploration data on the deposit and reasserting mineral reserves, especially during the transition from one stage of mining operations to another according to the project schedule, there is a need for the reconstruction of the quarry with the replacement of mining and transport equipment, reconstruction of opening workings and dumps and changing the production capacity of the quarry. Reconstruction works are related to mining and capital and are carried out according to specially approved projects.

The final stage of open-pit mining, usually associated with depletion of reserves or with the need to switch to an underground mining method, is a period of "damping" (repayment) of mining operations, sometimes lasting several years.

Procedure for the development of open-pit mining

The procedure for the development of open-pit mining operations cannot be established arbitrarily. It is a logical consequence and primarily depends on the type of deposit being developed, the topography of the surface, the shape of the deposit, the position of the deposit relative to the prevailing surface level, the angle of its incidence, power, structure, distribution by quality of minerals and types of overburden. The next logical consequence is the choice of the type of open-pit mining: surface, deep, on

-mountain, upland-deep or underwater.

Another stage of judgments is a fundamental (preliminary) decision about the quarry field — its possible depth, dimensions along the bottom and surface, slope angles of the sides, as well as the total reserves of rock mass and minerals in particular. Possible locations of consumers of minerals, dumps, tailings dumps and their approximate capacities are also being established, which makes it possible to outline possible directions and ways of moving quarry cargo. Based on these considerations, the possible dimensions of the quarry field, its location in conjunction with the relief of the surface, as well as the approximate contours of the mining allotment to the future enterprise are established. Only after that, taking into account the capacity of the quarry required by state plans, they begin to solve the problem of the procedure for the development of mining operations within the quarry field.

Figure 3.1 shows the development schemes of mining operations and pit ledges in profile and plan. The arrows depict the directions of mining operations for deposits of various shapes in conditions of a flat surface. In order to accelerate the commissioning of the quarry and reduce the level of capital costs, mining operations begin to be carried out where the mineral deposit is closer to the surface with the minimum possible volume of mining and construction work, taking into account possible solutions for opening working horizons for future periods and taking into account the development system that provides a high level of integrated mechanization of mining operations.

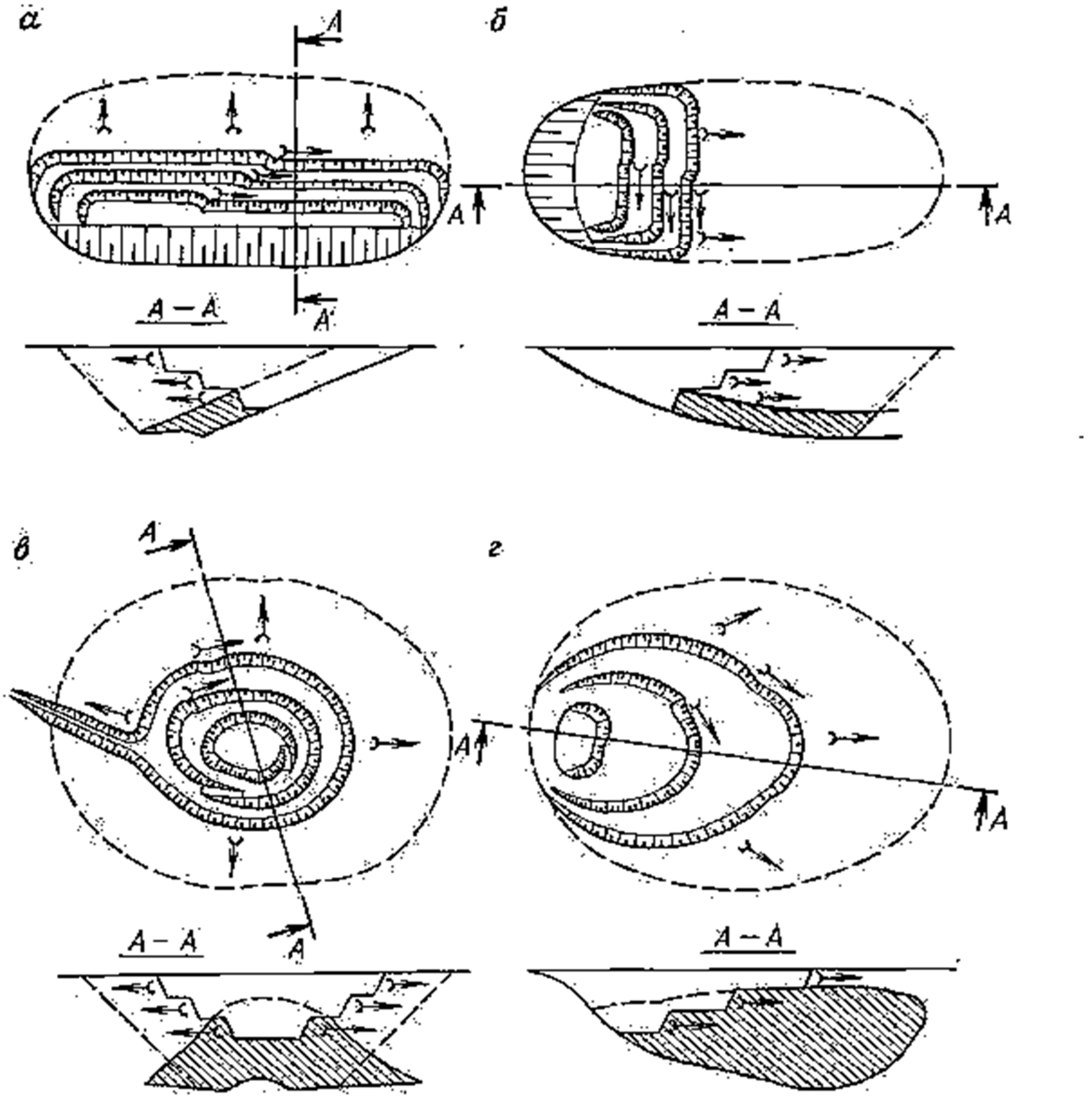

Figure 3.1. Mining development schemes:

a, b, c and d — the work front is located respectively along the long axis of the quarry, along the short axis, concentrically and ellipsically

The main goal of open-pit mining — extraction of minerals from the subsoil with simultaneous excavation of a large volume of overburden covering and containing deposits — is achieved with a clear and highly economical organization of the leading and most expensive process of open-pit mining - moving rock mass from the faces to reception points in warehouses and dumps. The efficiency of the movement process is achieved by the organization of stable cargo flows of minerals and overburden rocks, in relation to which the issues of opening the working horizons of the quarry field, as well as the capacities of the vehicles used, are solved.

Key words: surface preparation, drainage of rock mass, mining and capital works, maintenance works, mining and preparatory works, reconstruction of quarry facilities, the period of "attenuation", the choice of the type of open-pit mining, the purpose of open-pit mining.

Security questions:

1. What is the preparation of the surface?
2. What works are related to mining and capital?
3. What works are operational mining operations divided into?
4. Name the final stage of open development.
5. What is the main purpose of open-pit mining?

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Lecture 4

Topic: The concept of the mode and stages of mining operations. Preparing a career field for development

Plan:
1. Types of stripping coefficients.
2. Mining mode.
3. Stages of mining operations.
4. Methods of drainage of deposits.

The concept of the mode and stages of mining operations

Technical solutions for open-pit mining and its economic results are determined primarily by the ratio of the volumes of stripping and mining operations in general and by the periods of activity of the quarry. Quantitative evaluation of these ratios is carried out using stripping coefficients.

Average overburden coefficient of Csr (m3/m3)—the ratio of the volume of overburden rocks Vb.k in the contours of the quarry to the reserves of the mineral Vi.k in these contours:

$$K_{ср} = \frac{V_{в.к}}{V_{и.к}}.$$

The average operational stripping coefficient Ksr.e (m3/m3) is the average stripping coefficient for the period of operational work in the quarry. It is determined by the ratio of the total volume of overburden rocks Vb. to in the quarry minus the volume Vb.c, which was removed during the construction of the quarry, to the total mineral reserves of Vi.k minus that part of the Vi.c, which was extracted during the construction of a quarry*:*

$$K_{ср.э} = \frac{V_{в.к} - V_{в.с}}{V_{и.к} - V_{и.с}}.$$

The current coefficient of overburden Ct (m3/m3) is the ratio of the volume of overburden rocks Vb.t, actually moved from the massif to the dumps for any period of time (month, quarter, year), to the extracted volume of minerals Vi.t for the same period:

$$K_T = \frac{V_{в.т}}{V_{и.т}}.$$

The boundary coefficient of overburden Kgr determines the volume of overburden rocks per unit volume of mineral, which is permissible to move from the array to the dumps under the condition of profitability of open-pit mining.

The planned stripping coefficient Kp is used when planning the current production cost of the mineral St (sum / m3); it characterizes the amount of stripping work, the costs of which are repaid during the current production of open works:

$$C_т = C_{и.т} + K_п C_{в.т},$$

where $C_{и.т}$ и $C_{в.т}$ are, respectively, the current costs of developing 1 m3 of minerals and 1 m3 of overburden.

Overburden coefficients at many quarries are measured by the ratio of the volume or mass of overburden rocks to 1 ton of mineral.

The ratio of the current volumes of stripping and mining operations primarily determines the production capacity of the quarry by rock mass, which is not constant, primarily due to changes in the annual volumes of stripping operations for individual periods. This change is a consequence of the unstable capacity of overburden and mineral deposits, the conditions of its occurrence, the presence of various geological disturbances, uneven content of useful components in the deposits. Changes are also determined by economic reasons. At the same time, enterprises — consumers of minerals are designed for a certain production capacity and must receive strictly defined volumes of minerals of established quality. These provisions are taken as a basis for choosing the mode of mining operations at the quarry.

The mining regime is understood as the sequence of execution of the volumes of stripping and mining operations established by the project or study in time, which ensures the planned, safe and cost-effective development of the deposit over the lifetime of the quarry. The mining regime is estimated according to the schedule, which shows the changes in the volume of mining and stripping operations by year for the entire period of the existence of the quarry (Fig. 4.1).

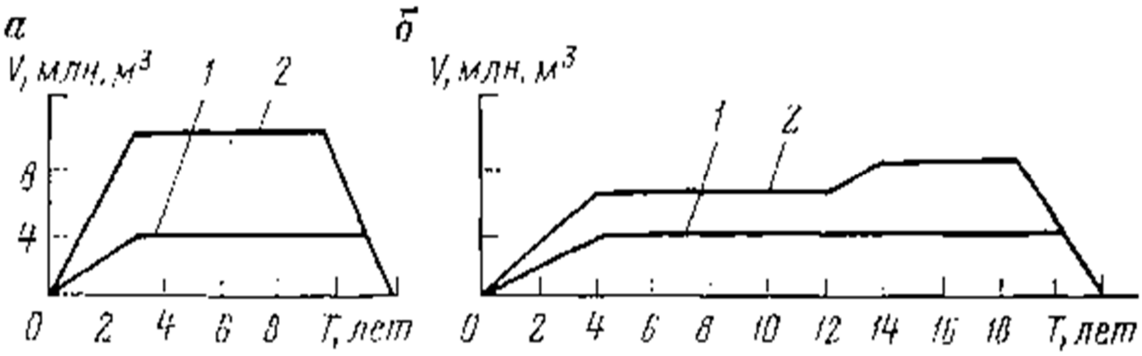


Fig. 4.1. Graphs of changes in the volumes of V extraction (1) and stripping (2) by year T: a and b, respectively, for the duration of the existence of quarries for 10 and 20 years.

For a relatively short period (up to 5 years) the mode of mining operations at existing quarries is established when planning mining operations for the fifth year. The mining mode is economically effective, which ensures maximum profit from the development of the deposit with the production of minerals of the required quality.

When the duration of the quarry is 8-12 years (which corresponds to the depreciation period of the main quarry equipment), economic efficiency is achieved by the fact that the development is carried out for a possibly longer period of time with constant annual volumes of stripping work (Fig. 4.1, a); with a longer duration of the quarry, in general, it is advisable to divide the entire period of work into separate periods, each of which is characterized by a constant annual volume of stripping works; the volume of these works is increased or decreased during the transition to the next period (Fig. 4.1, b).

The periods of operation of a quarry with significantly different volumes of stripping work are called development stages. With a short period of existence of a career, they tend to develop without dividing into stages, and with a long period, it is desirable to allocate several stages.

In the first case, it is advisable to carry out work with a constant current stripping coefficient close to the average operational one. In the second case, a stepwise increasing schedule of mining operations is obtained (see Fig. 4.1, b). The duration of each stage is linked to the depreciation periods of the main equipment; the transition from stage to stage is usually timed to the period when it becomes necessary to reconstruct the quarry and replace physically and morally obsolete mining and transport equipment.

The uneven mode of mining operations within the stage leads in some years to the performance of "peak" volumes of stripping operations. At the same time, the economic indicators of the development are deteriorating, since in a relatively short period there is a concentration of a large number of mining and transport equipment, energy capacities, which leads to overstaffing of workers and employees, as well as to the additional construction of auxiliary workshops and household facilities. The disadvantages of the uneven mode of work at quarries with a relatively short life span and when constructing them in insufficiently developed areas of the country are particularly acute.

Maintaining uniform volumes of stripping operations at each stage contributes to the sustainable economic activity of the enterprise.

The choice of a rational mining regime at the quarry is of great importance for increasing the profitability of enterprises and accelerating the turnover of funds, it allows reducing premature and inefficient costs during those periods of the quarry when the stripping coefficient and the cost of minerals change due to changes in natural conditions. The calendar stages correspond to the volumetric stages of the quarry development, i.e. certain intermediate contours of the quarry in depth and in plan (Fig. 4.2). Establishment of such phased contours, and within them annual contours (mining regulations) for each ledge, it is the task of establishing a rational mining regime.

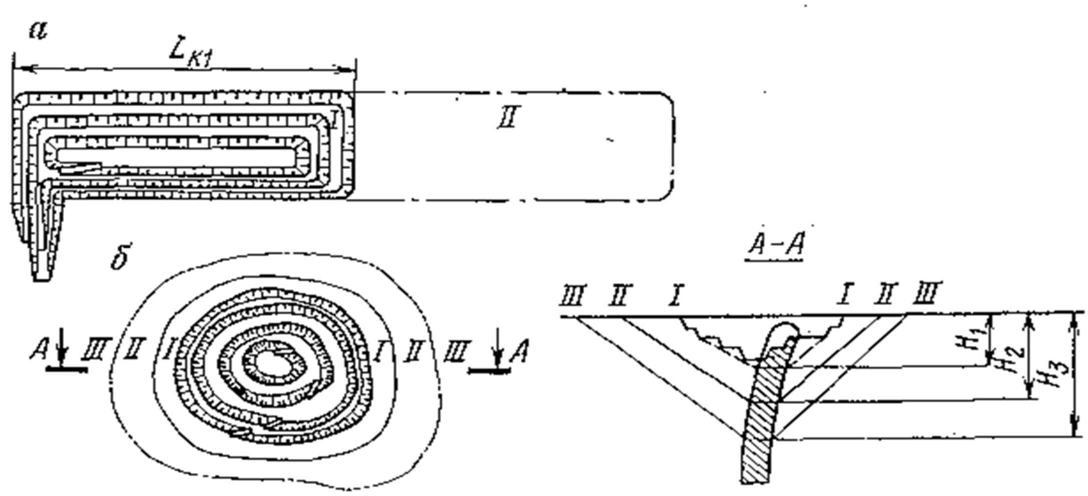


Fig. 4.2. Schemes of stage contours of career development:
a and b - on quarries of elongated and rounded shapes, respectively

Preparing a career field for development

For the normal conduct of mining operations and the possibility of placing technical and economic structures, transport communications and dumps, all natural obstacles and artificial structures within the quarry field and in the area of transport approaches to it are removed or transferred. Natural obstacles include: forests, large shrubs, streams, rivers, lakes, swamps — in lowland deposits; overhangs, stabs — in the mountains. Roads and railways passing within the technical boundaries of the quarry, as well as various industrial and domestic structures are considered artificial structures.

The forest and shrubbery are removed first of all on the territory of capital and split trenches and the placement of the industrial site, and then, as mining operations develop, completely within the final contours of the quarry. These works are carried out in a mechanized way using electromechanical saws, brush cutters, bulldozers and other means. In areas with heavy snow drifts and in steppe arid terrain, vegetation around the quarry and industrial site should be preserved as long as possible. It protects these objects from snow and sand drifts.

The waters of swamps, lakes, streams and rivers are diverted immediately beyond the limits of the mountain drainage. For the descent of water, channels with runoff are arranged in the direction of lowered terrain areas, and for the diversion of streams and rivers, a bypass channel is constructed behind the contour of the mountain branch. The old channel is usually blocked by a dam to create the required backup, since the route of the bypass channel, as a rule, runs along higher absolute marks. The dimensions of the cross-section of the bypass channel should ensure the passage of water during the flood period. To prevent water from seeping into the quarry, the channel slopes are concreted or lined with stone. The bypass channel is given a slope equal to the natural slope of the riverbed in this section.

The waterlogging of deposits sharply reduces the stability of the slopes of mine workings carried out in sandy, soft, dense and fractured rock and semi-rock formations, complicates and increases the cost of construction and maintenance of transport communications in the quarry, sharply reduces the productivity of the main mining and transport equipment.

The field drainage system should provide normal conditions for mining, capital and operational work in the quarry. Drainage measures provide for the protection of the quarry from surface and underground water inflows through special workings and organization of drainage.

The method of drainage of the quarry is chosen depending on the water-physical properties of rocks, the number, location, capacity and water content of aquifers. There are surface, underground and combined methods of drainage.

In any hydrogeological conditions, upland channels are constructed to protect the quarry from surface water runoff in areas where relief marks are lowered, through which water flows to the catchments. The cross-section of the upland ditches is calculated by the possible inflow of water, and the longitudinal profile of the ditches is given a slope of $i = 2 \div 3\%$.

When developing deposits with simple hydrogeological conditions, drainage trenches are carried out and a system of quarry drainage is created. With such a surface drainage method, the quarry itself is a drain.

Simple hydrogeological conditions are characterized by:

deposits composed of rocky and semi-horizontal low- and medium-fractured rocks and aquiferous sediments with a capacity of up to 10-15 m with an inflow of groundwater into the quarry up to 300-500 m3/h;

deposits composed of soft and sandy unstable rocks with local water saturation and inflow of groundwater into the quarry up to 100 m3/h.

The drainage of the remaining deposits located in complex and very complex hydrogeological conditions should be carried out with the creation of a system of special drainage workings to lower the groundwater level in the contour of the quarry field.

The surface method of draining these deposits sometimes consists in creating a system of drainage trenches or trenches in combination with horizontal drainage wells, but much more often — in creating a system of vertical water-lowering wells of large diameter (250-500 mm), which are arranged in one, two or three rows at a distance of 30-50 to 200-250 m from each other, depending on filtration coefficient of the drained rocks. Pumping water from such wells is carried out, as a rule, by centrifugal submersible pumps.

With an underground drainage method, drainage shafts are usually constructed with a network of underground workings, which are carried out on minerals or waste rocks. Drifts are carried out in stable rocks along a permeable rock layer every 200-250 m along the sides of the quarry, prone to deformation. Water enters the drainage workings through through or clogged filters. From the workings, water flows into the drainage basin of the drainage shaft and is pumped to the surface.

The combined drainage method uses a system of wells drilled from the surface and drainage drifts with the necessary devices. Drainage underground workings during the construction of the quarry are carried out by special construction organizations, and during operation - by quarry services allocated to specialized sites.

The water removed from the quarry should be discharged into the nearest watercourse or catchment, eliminating the possibility of its re-penetration into the quarry through cracks, sinkholes or permeable rocks. There should be no waterlogging of the territory adjacent to the quarry. Measures are also envisaged to conserve groundwater resources, excluding pollution and mineralization of water supply sources and reservoirs of workers' recreation areas.

During the development of the field, the drainage system of the quarry, as a rule, changes: new contours of water-lowering wells, underground workings, water collectors, etc. are created. Changing the system allows you to drain rocks in advance before their development and at the same time avoid premature construction of expensive water-lowering facilities.

Reference words: overburden coefficient, average, average operational, current, boundary, planned, mode, period, stage, natural and artificial obstacles, drainage system, surface, underground, combined.

Security questions:

1. *Name the main types of stripping coefficients.*
2. *What determines the boundary stripping coefficient?*
3. *What do you mean by the mining regime?*
4. *What is called the development stage?*
5. *What are the ways of draining the quarry.*

Lecture 5

Topic: The order of formation of cargo flows. Types of cargo flows

Plan:

1. Formation of cargo flows.
2. Types of cargo flows.

The variety of forms of deposits and conditions of their occurrence in the subsurface, on the one hand, and the basic principle of open—pit mining - layered (step-by-step) excavation of both overburden rocks and minerals, on the other hand, predetermine the need to form cargo flows in such a way as to ensure minimal costs for moving the rock mass from the faces to the dumps and warehouses and thereby achieve maximum savings when conducting open-pit mining. The solution to this problem is to create cargo flows of the quarry and on this basis to open the working horizons of the quarry. The order of formation of cargo flows is shown below on the example of an elongated quarry field with two deposits of different quality I and II (Fig. 10.1, a) and relative horizon marks from +20 to -60 m. Mining operations begin from stage 1, closer to deposit I, at elevations of -10 and ± 0, at which rock mass is excavated in the faces and cargo flows begin. Two sides (right and left) are planned for development; on each of them, in each layer under development, the volumes and quality of rocks are different both for individual stages (1-6) of mining operations, and for the entire development period. With the order of development of mining operations shown in Fig. 10.1, the volumes of overburden rocks and ores by grades for each horizon are estimated (calculated) separately on the right and left sides and are depicted in the form of a step-by-step schedule of the mining regime (Fig. 10.1, b). When constructing a step-by-step schedule of mining operations, it is necessary to provide for the minimum time for the start of mining and the appropriate attribution to later periods of excavation and movement of the bulk of overburden rocks. According to a phased schedule, it is possible to assess the economic efficiency of the adopted mining development option by comparing it with possible other options. If this order of development is taken as a basis, they begin to consider and form cargo flows.

For this, a summary table is compiled (Figure 6.1, c) receipts from each horizon of volumes of various cargoes for each stage of development (1-6) and for each working side of the quarry.

Based on the data in the table, cargo flows can be formed. However, in order to judge the mining calendar by the accepted productivity of the quarry (by ore), it is necessary to transform step-by-step graphs and tables into calendar ones (Fig. 6.2, a and b), on which the years of the quarry's existence are postponed along the ordinate axis. The order of transformation of graphs is described below.

The given example of plotting shows how, both by stages of mining operations and by the years of existence of the quarry, the required volumes of quarry cargo to be removed and moved to ensure production development plans are determined. Using the method of variants, step-by-step and calendar schedules can be improved in order to optimize the economic results of the open-pit development of this field. At the same time, even approximate calculations performed in this way make it possible to justify the formation of quarry cargo flows at all stages of mining operations and, consequently, to prove the economic efficiency of the accepted method of opening.

Schedules for the formation of cargo flows should be built for all types of deposits with mandatory consideration of the surface relief. If necessary, it is necessary to divide the volumes of overburden rocks by their types, and minerals by grades, in order to then make better decisions on the choice of a complex of mining and transport equipment and the duration of operation of each cargo flow. At the same time, the general surface mark is fixed on the graphs and the upland and deep parts of the quarry field are highlighted.

Types of cargo flows

Each excavation layer can generally be represented by:

overburden rocks (rocky, semi-horizontal, dense or soft);

substandard and off-balance sheet minerals stored in separate dumps for use in subsequent periods; minerals, in which, according to the planned tasks, types and grades are allocated for separate transportation and use.

The flow of goods of a certain quality, characterized by a relatively stable (in time) direction and a certain volume of traffic per unit of time (shift or day), is called an elementary cargo flow.

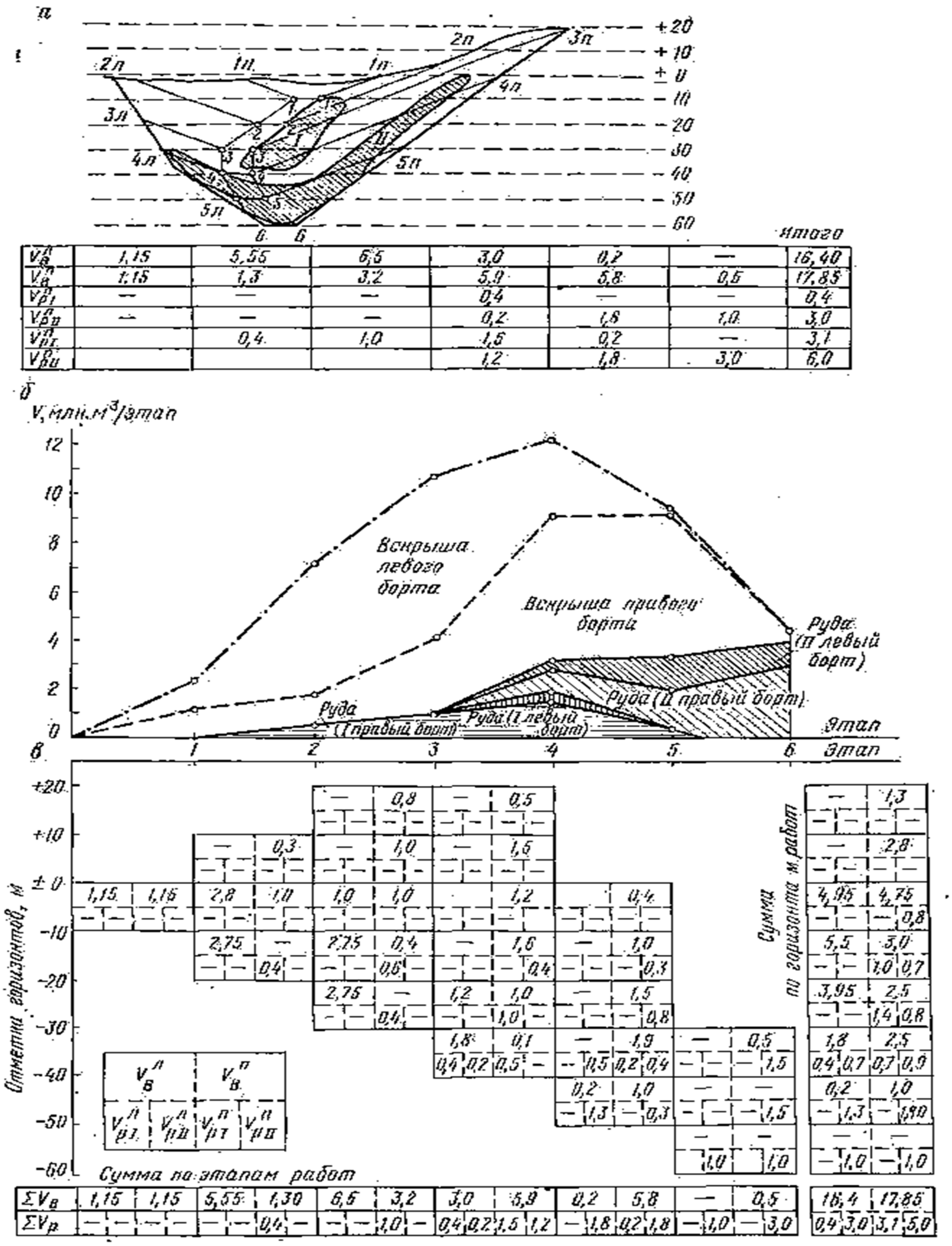
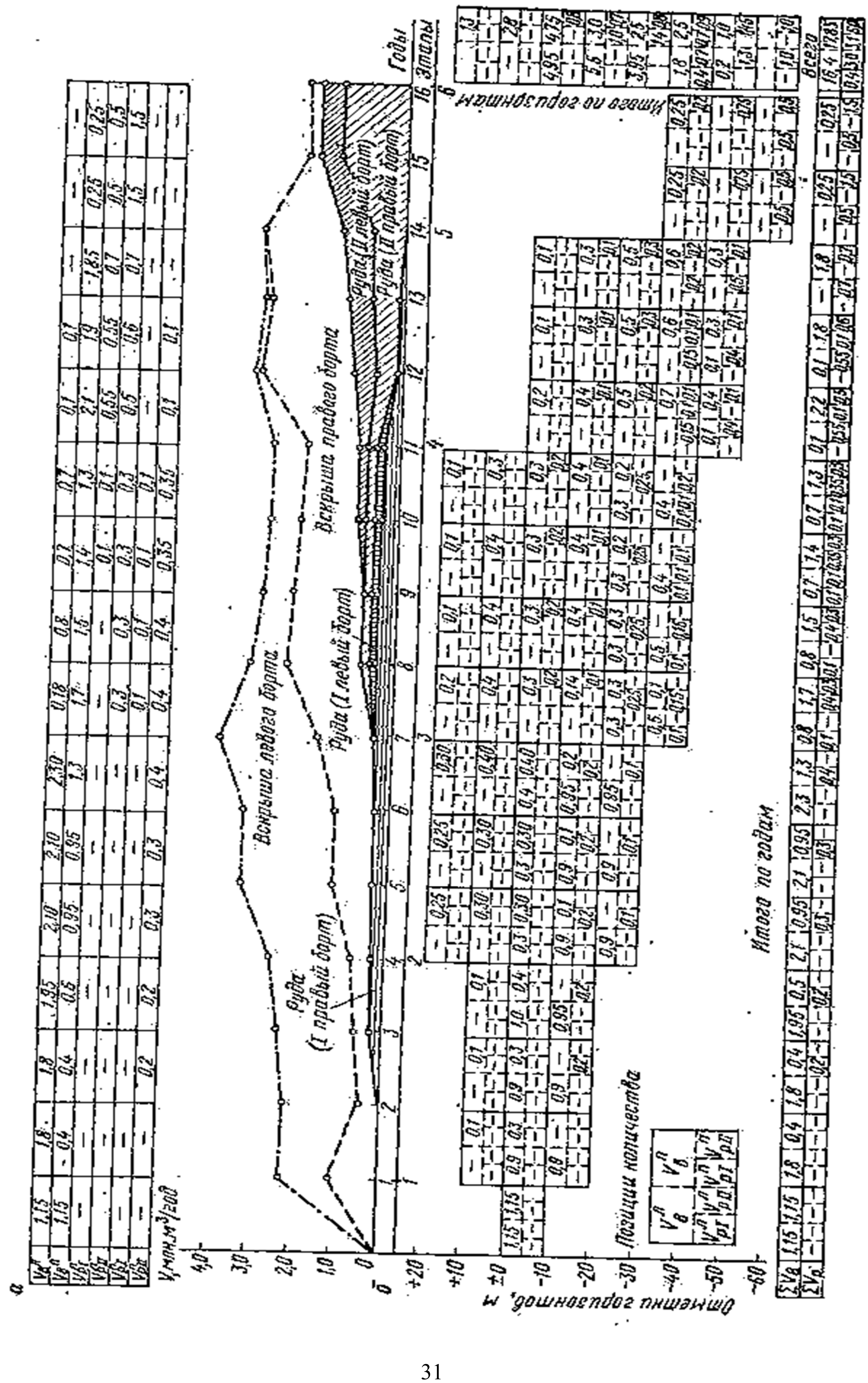


Figure 10.1. The scheme of staged cargo flows (a) and schedules of the mining regime (b) and the phased distribution of cargo flows (c).


а
б
V, млн.м³/год
Отметки горизонтов, м
Годы
Этапы
Вскрыша левого борта
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Руда (II правый борт)
Позиции количества
Итого по годам
Итого по горизонтам
Всего

If the rocks in the face are homogeneous (simple face), then one elementary cargo flow begins from it; two or three elementary cargo flows begin from a complex face (with heterogeneous rocks and separate excavation). Thus, the number of elementary cargo flows on the ledge depends on the number of faces and the method of excavation of rocks in them, and it is usually greater than the number of active faces.

Elementary cargo flows may differ in their directions (Fig. 10.3, a and b), as well as by type of transport (see Fig.10.3, b), transport communications (Fig. 10.3, c) or models of one type of quarry transport. For example, elementary rock and ore cargo flows from a complex ore face when using vehicles and one highway often differ only by moving ore and rock in different dump trucks of the same size (Fig. 106.3, d). When using conveyor transport in such conditions, separate conveyors are already required, i.e. elementary cargo flow10.3, c).

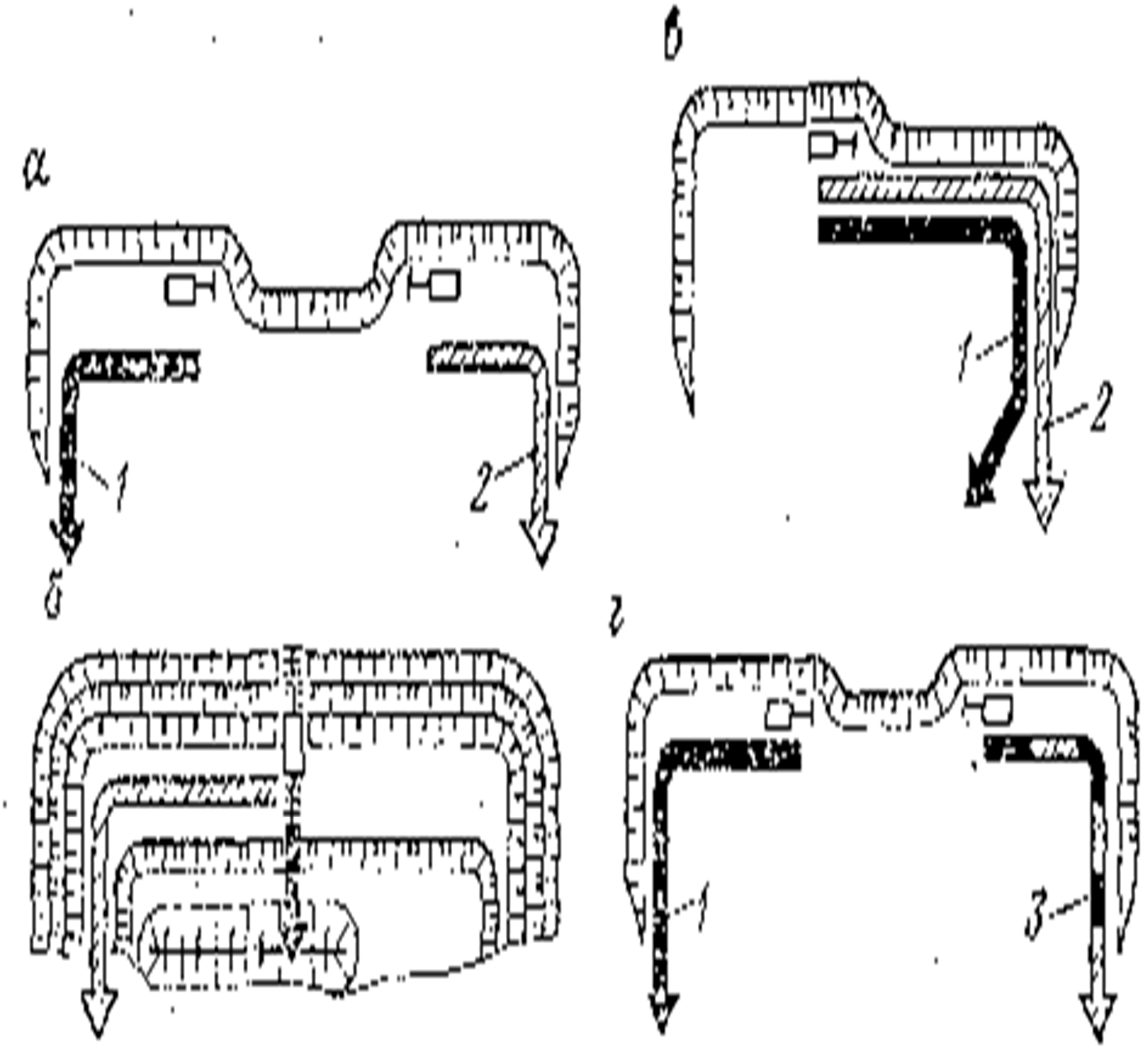


Fig. 10.3. Schemes of elementary cargo flows:
1 - overburden rocks; 2 - minerals; 3 - alternately empty rocks and minerals.

Elementary cargo flows from the faces with homogeneous rocks in order to reduce their number tend to combine into one cargo flow from the ledge (Fig. 10.4). According to the same principle, cargo flows of ledges are combined into homogeneous cargo flows of a group or all ledges of a quarry (Fig. 10.5, a and d).

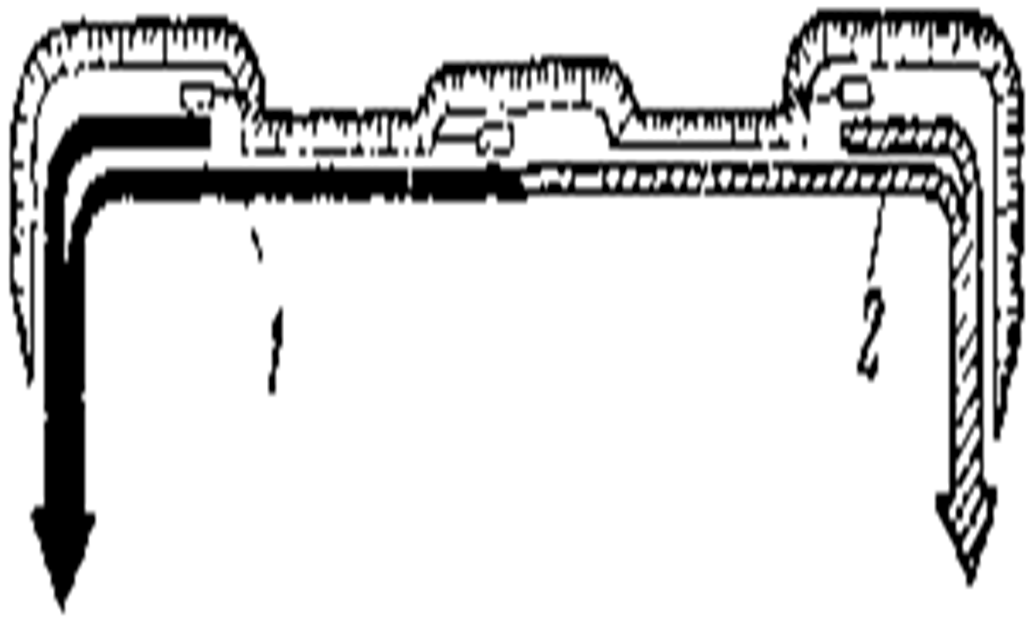


Fig. 10.4. The scheme of cargo flows from the ledge:
1 - overburden rocks; 2 - minerals.

A group of connecting elementary cargo flows having common communications forms a converging cargo flow (see Fig. 10.4 and 10.5, a). The total cargo flow of a quarry or its section, then divided into separate cargo flows, is called divergent cargo flow (Fig. 10.5, b). Mainly cargo flows of overburden rocks and minerals are divided, less often — heterogeneous breeds and rarely — homogeneous breeds.

The total cargo flow formed by elementary cargo flows converging at first, and then (more often on the surface) diverging, is called a complex cargo flow (Fig. 6.5, c). If there are transshipment or sorting points along the cargo route, the cargo flow is called combined. In the practice of open development, complex and combined cargo flows prevail.

If cargo flows consist of heterogeneous rocks, they are called heterogeneous cargo flows.

The total cargo flow of a quarry is called concentrated if its constituent cargo flows move along one output transport communications from the quarry (see Figure 10.5, a), and dispersed (see Figure 10.5, d) if cargo flows move along different communications.

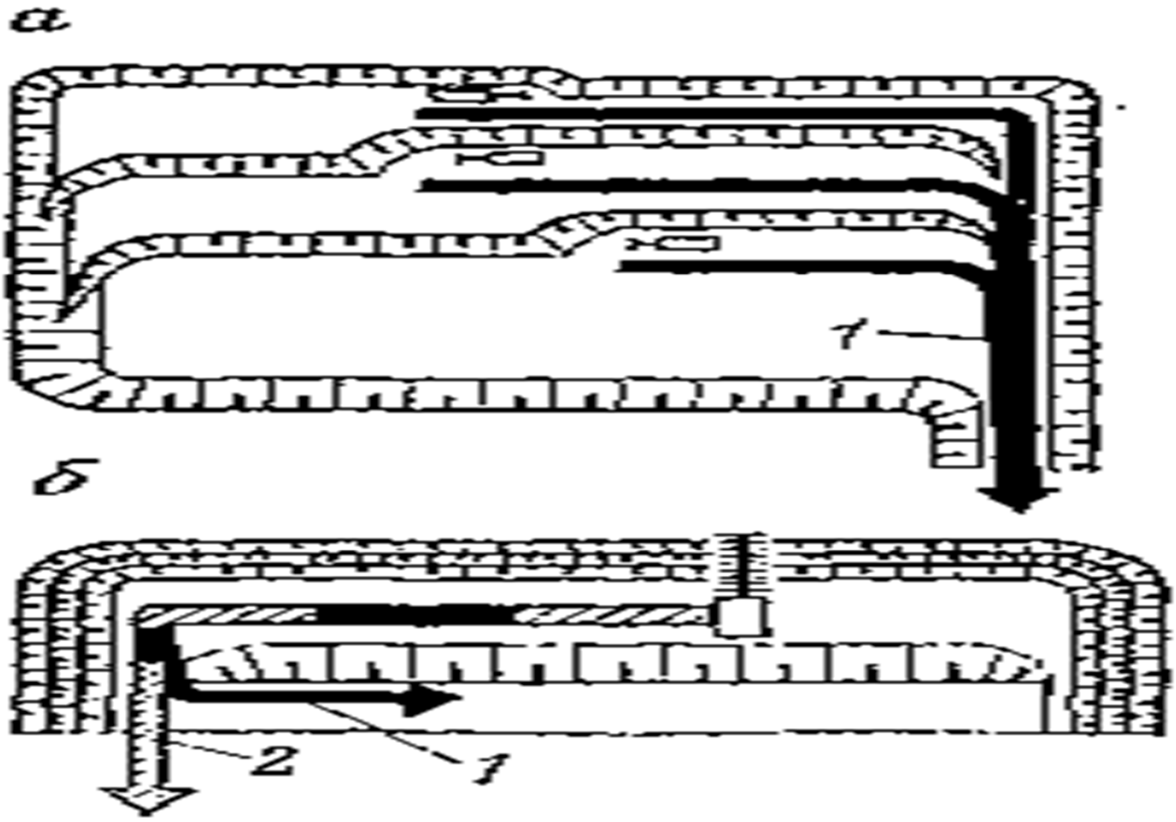
Reducing the number of cargo flows in the quarry makes it possible to use equipment more economically, improve the quality of roads, as well as reduce the number of opening workings and the cost of their construction.

Several cargo flows in the quarry can be:

independent from each other, if the operation of the complex of equipment servicing this cargo flow (from its beginning to the end) does not depend on the operation of equipment servicing other cargo flows, and the equipment is strictly assigned to a certain cargo flow;

dependent on each other, if it is necessary to periodically redistribute equipment, in particular vehicles, along adjacent cargo flows for more complete use of it; such redistribution is carried out by the dispatching service;

rigidly dependent, if the dispatching service constantly, in accordance with the schedule, changes the loading of equipment, redistributes equipment and regulates the volume of elementary cargo flows (for example, to achieve the desired averaging of minerals coming from the quarry to the concentrator).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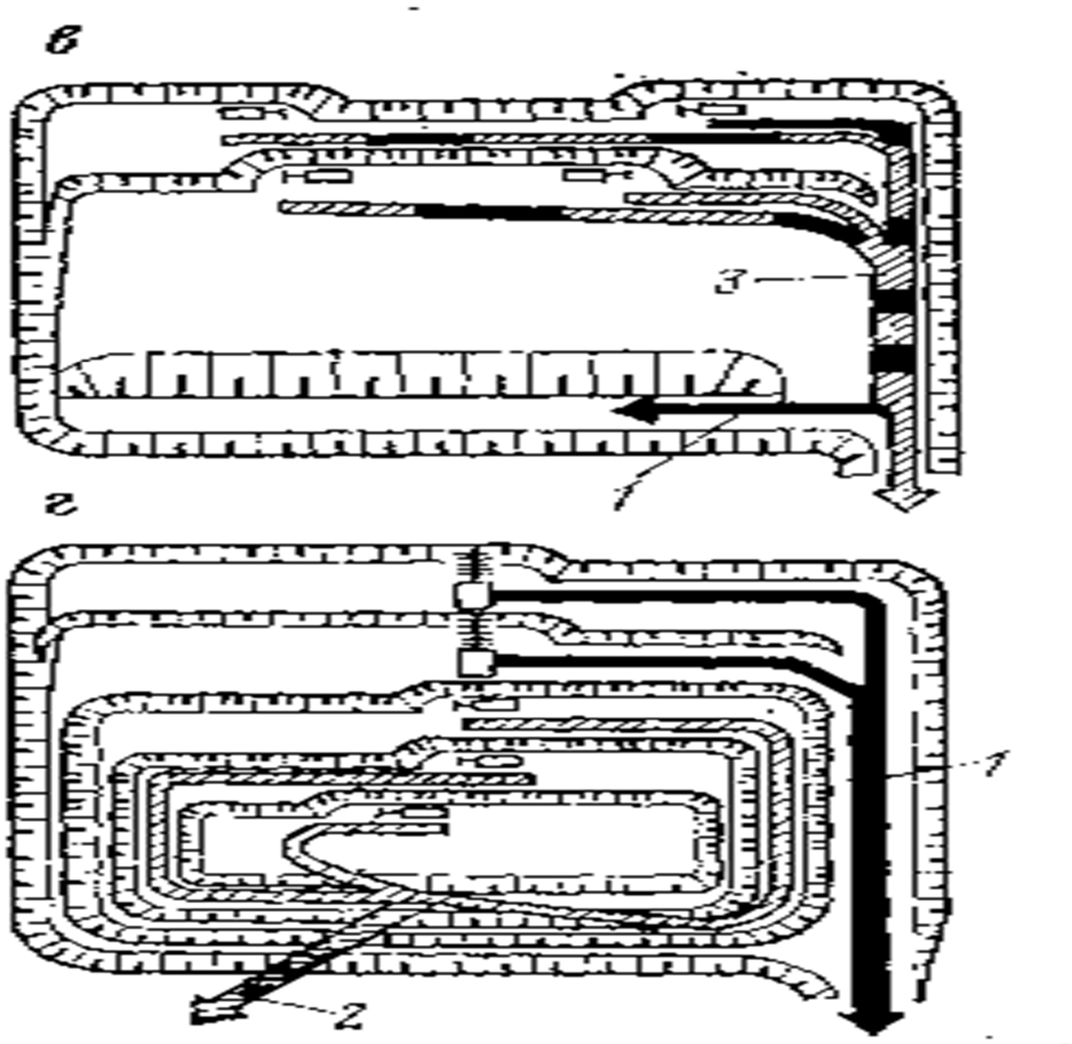


Fig. 10.5. Schemes of cargo flows from the quarry:
1 - overburden rocks; 2 - minerals; 3 - alternately empty rocks and minerals.

Dependent cargo flows are the most common. Cargo flows organizationally combine all processes: preparation of rocks for excavation, their excavation and loading, moving, dumping and warehousing. The precise functioning of cargo flows determines the efficiency of mining operations and the efficiency of equipment use.

Key words: cargo flow, development of mining operations, schedule of mining operations, elementary cargo flow, cargo flow from a ledge, cargo flow of a group, converging, diverging, heterogeneous, concentrated, dispersed, independent, dependent, rigidly dependent.

Control questions:
1. What determines the need for the formation of cargo flows?
2. For which fields do they build schedules for the formation of cargo flows?
3. What is called elementary cargo flow?
4. What cargo flows are called heterogeneous?
5. Cargo flows in a quarry can be…

Lecture 6

Topic: Opening the working horizons of a career

Plan:

1. The initial stages of mining development.
2. Opening mine workings.
3. Methods of opening the working horizons of the quarry.

Initial stages of mining development

The opening of working horizons is carried out through the construction of workings specially designed for this purpose. To ensure the transportation of rock mass, each horizon must be opened with a capital trench (Fig. 11.1, a), usually inclined, since it connects the mark of the horizon being opened with the mark of already existing horizons and surfaces.

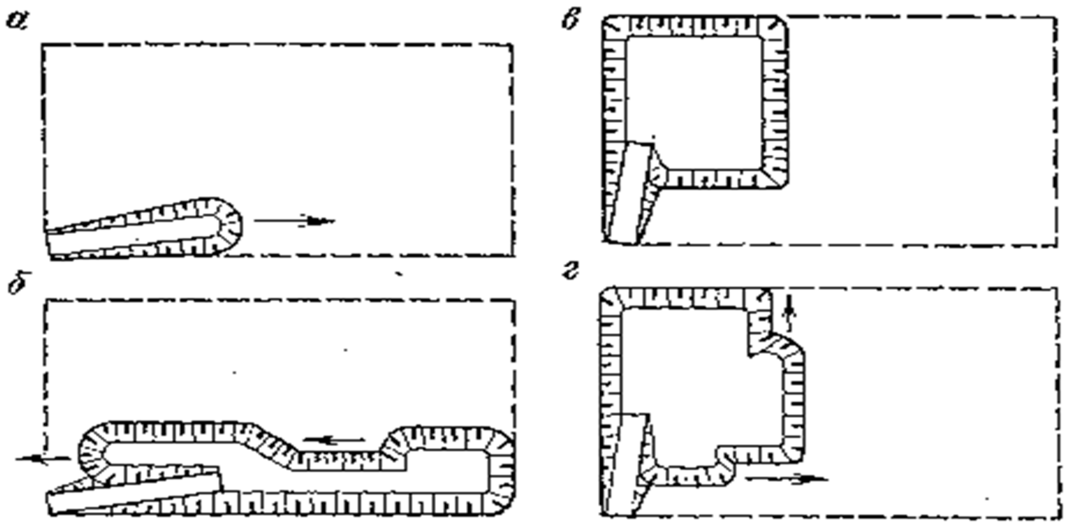


Fig. 11.1. Schemes of the initial period of mining development on the horizon.

Mining operations on the horizon begin with the creation of the initial front, for which a split trench is carried out (Fig. 11.1 b) or a split pit (Fig. 11.1, c). Sometimes the rock, if the parameters of excavators allow, is placed in the instrument dump, but more often it is transported to an external dump. Next, one or two sides of a split trench (see Fig. 11.1, b) or a split pit (Fig. 11.1, d) are spaced. After the necessary advance of the upper ledge, it becomes possible to open the lower horizon and carry out split work within it. The longitudinal slope of the working horizons should be established taking into account the safety of the operation of vehicles during loading.

The choice of the location of the split trenches is mainly influenced by the relief of the surface and roof of the deposit and the need to reduce the volume of mining and construction work for the fastest commissioning of the quarry. Usually, split trenches in the development of horizontal and shallow deposits are carried out along the strike of the deposit. This provides a sufficient front of work for high-performance machines and allows you to obtain significant uncovered mineral reserves. In small quarries, preparatory work can be carried out sequentially in several relatively short sections. In this order, deposits of construction rocks are often developed, which reduces the initial costs of stripping and the equipment used.

When developing formations of shallow formations and deposits of complex structure, the direction of mining development should provide the possibility of separate excavation of minerals and waste rocks. When developing reservoir deposits, work is developed by falling and very rarely by rising. If, according to the opening conditions, the work front is located along a short axis or diagonally of the quarry field, the required power of the quarry is achieved at a high speed of moving the work front.

At any location of the work front and the direction of mining development, the rock thickness of the ledge with an area of F (m2) (on its surface) and an average capacity of H (m) should be worked out according to the calendar plan for T months. The average monthly value of the opened area can be taken as a comparable indicator of the intensity of development

$$F_{м}=F/T=L_{ф.у}v_{ф},$$

when $L_{ф.у}$ - the accepted average length of the work front of the ledge, м; $v_{ф}$ — the average monthly rate of movement of the work front, m/month.

The rate of advance of the work front is determined primarily by the intensity of mining reserves.

Opening mine workings

The division of capital trenches is shown in Table 11.1. Stationary external and internal capital trenches are used for a long time. Their parameters (initial and final depth, longitudinal slope, length, slope angles of the sides) are strictly regulated depending on the specific conditions, properties of the surrounding rocks and technical conditions for the design of transport communications.

Table 11.1. Division of capital tranches (according to E. F. Sheshko)

Признак разделения	Основные различия	Траншеи
Расположение траншей относительно контура карьера	Расположение вне контура карьера	Внешние
	Расположение внутри контура карьера	Внутренние
Число уступов, обслуживаемых системой траншей	Один уступ	Отдельные
	Несколько (группа) уступов	Групповые
	Все уступы карьера до конечной глубины	Общие
Основное назначение траншей	Для движения груженых и порожних поездов (маятниковое движение транспорта)	Одинарные
	Для движения только груженых или только порожних поездов (поточное движение транспорта)	Парные
Стационарность траншей	Постоянное расположение за контуром или на бортах в конечном положении	Стационарные
	Временное расположение внутри конечных контуров на бортах, подлежащих разработке	Скользящие (временные)

The cross section of individual capital trenches is trapezoidal or triangular. When the transport and safety berms are located on the sides of the trenches, they have a stepped shape. The depth of capital trenches usually varies from zero to a value equal to the height of one or more ledges. The rises (slopes) of capital trenches depend on the type of transport used (Table 11.2).

Table 11.2. Characteristic rises of capital trenches

Вид карьерного транспорта	Величина подъема в направлении движения транспортных сосудов, %	
	груженых	порожних
Наклонные траншеи		
Железнодорожный:		
паровая тяга	0,02—0,03	0,025—0,035
тепловозная и электрическая тяга	0,025—0,04	0,025—0,06
моторные вагоны	0,04—0,05	0,06—0,08
Автомобильный	0,05—0,1	0,08—0,12
Крутые траншеи		
Бесклетевой подъем с тягачами	0,12—0,25	—
Ленточные конвейеры	0,25—0,33	—
Клетевой подъем	0,25—0,5	—
Скиповой подъем	0,50—1,0	—

The angles of the slopes of the sides of the capital trenches are determined by their service life, the properties of the rocks, their water content. The side of the trench with a long service life should have long-term stability; its slope angle in sandy, soft,

in dense and semi-horizontal rocks, no more than the angle of the natural slope is accepted, and in rocks - up to 50 - 60 °. Both sides of the external capital trenches have a permanent position, and in a stationary internal capital trench only one side has a permanent position. The minimum width of the bottom of the capital trenches is determined by the sum of the dimensions of the vehicles, the safe gaps between them, the transverse dimensions of the platforms and the cuvettes located on the bottom. The width of the bottom of the capital trench, set according to the conditions for the placement of transport communications, is checked according to the conditions for the possibility of carrying out a trench.

The cross-sectional area of the underground opening workings is determined by the dimensions of the transport equipment and the schemes of track development (taking into account the observance of the necessary gaps). For conditions when broad gauge railway transport is used (dumpcars, gondolas and industrial electric locomotives), the section of the workings (tunnel) is regulated by GOST.

Ways to open the working horizons of a career

The opening of the working horizons is carried out to provide the cargo flows formed on the ledges with transport communications that allow moving cargo from the working horizons to reception points on the surface or on intermediate horizons. The opening workings begin from the surface or from the intermediate working horizon already opened and end at the level of the working area of the horizon being opened.

The method of opening is determined by a number of signs, primarily by the type of opening workings.

In some cases (the use of tower excavators and cable cranes), the development of the entire field and the movement of quarry cargo are carried out without opening workings. It is possible to create transport access to individual working horizons of the quarry even in the absence of opening workings: for example, when transporting overburden rocks to the overburden dumps of upland or upland-deep quarries, when using conveyors located on a non-working board, etc. This method of opening is called trenchless.

In most cases, the working horizons of the quarry are opened with capital trenches or semi-trenches. Less often, the opening is carried out by underground workings (inclined and vertical shafts, tunnels, tunnels), as well as by a combined method.

Trenches designed for the movement of wheeled vehicles (rail and road transport) should be inclined; trenches equipped with lifts - steep.

Depending on the number of ledges (one, a group or all the ledges of the quarry) served by trenches with a common route, separate, group and general trenches are distinguished, respectively (see Table 11.1).

External trenches are stationary or semi-stationary. Internal trenches can be stationary (located on non-working sides of the quarry), semi-stationary, temporary and sliding. Temporary and semi-stationary internal trenches on the working sides of the quarry are used to reduce the volume of mining and capital works and when the volume of stripping works is redistributed over time.

Table 11.3. Classification of autopsy methods

Признак способа вскрытия	Способ вскрытия		
	открытыми выработками (траншейными)	подземными выработками	комбинацией открытых и подземных выработок
Положение вскрывающих выработок относительно конечного контура карьера	Внешними, внутренними или смешанными траншеями и полутраншеями	Внешними, внутренними или смешанными	Внешними, внутренними или смешанными
Стационарность выработок	Стационарными, полустационарными и временными (скользящими) траншеями или полутраншеями	Стационарными	Стационарными или комбинацией стационарных с полустационарными (временными)
Наклон выработок	Крутыми или наклонными траншеями и полутраншеями	Вертикальными, крутыми, наклонными или горизонтальными	Комбинацией вертикальных, крутых, наклонных или горизонтальных
Число обслуживаемых горизонтов	Отдельными, групповыми или общими траншеями и полутраншеями	Отдельными, групповыми или общими	Отдельными, групповыми или общими
Характер движения транспортных средств на уступе (поточное или маятниковое)	Одинарными или парными траншеями и полутраншеями	Одинарными или парными	Одинарными или парными

On the working horizon, opened by one (single) capital development, the pendulum (return) movement of vehicles is most often used. If the working horizon is opened by two workings (cargo and empty), then the through movement of vehicles on the ledges is ensured and in this case the use of mining equipment increases in time, as a result of which the increase in costs for the construction of opening workings is compensated. Such workings are called paired, they can have an external or internal foundation and consist of a pair of separate, group or general trenches or semi-trenches. Single and paired tracks are allocated accordingly. Paired trenches and tracks are used mainly in shallow quarries with intensive cargo turnover.

In accordance with these main features of the division of capital trenches, Table 11.3 provides a classification of the main methods of opening, based on the classification of Prof. E.F. Sheshko. When opening horizons located below the dominant level of the earth's surface, the longitudinal profile of the capital trenches is characterized by a rise in the direction of movement of loaded vehicles, and when opening horizons located above the dominant level of the earth's surface — a rise in the direction of movement of empty transport vessels. According to the location of the opening workings relative to the quarry field and the deposit, there is an opening by flanking and central trenches (or underground workings), an opening from the recumbent or hanging side of the deposit, as well as from the end of the quarry.

Reference words: excavation, trench, semi-trench, capital, split, external, internal, separate, group, general, paired, stationary, sliding (temporary), inclined, steep, opening method, opening by underground workings, combined method.

Security questions:

1. How is the opening of working horizons carried out?
2. What determines the speed of the front of work?
3. What are the signs of the division of capital tranches.
4. What is the method of opening called trenchless?
5. What are the methods of opening.

Lecture 7

Topic: Routes of opening workings

Plan:

1. Routes of opening workings.
2. Forms of routes of capital workings.

The route of a trench or other work-out is a line whose position in space is determined by the plan and profile of the roadbed of the transport route. The horizontal projection of the route is the path plan, and its vertical projection is the longitudinal profile of the path. The path in the plan consists of rectilinear and curved sections, and in the profile - of horizontal and inclined sections, as well as mating sections between them, providing the necessary smoothness of transitions.

Tracing consists in establishing the axis of the transport path on the plan and in the profile. The points through which the route should pass are determined by a combination of topographic, geological, construction and other factors.

According to the position of the route relative to the contour of the quarry, external, internal and mixed routes are distinguished according to the workings. According to the service life, stationary, semi-stationary and sliding (temporary) routes are distinguished; the first are located on the non-working sides of the quarry, the second - on temporarily preserved sections of the working sides of the quarry, sliding (temporary) - on the developed sections of the working sides of the quarry.

The basis for tracing capital trenches is the intermediate or final position of the sides of the quarry, depicted on the plan by isolines of the same elevation marks with an interval equal to the height of the ledge. The route of the external trenches is carried out from the surface to the horizontal that determines the position of the ledge to be opened; the route of the internal trenches runs along the side and crosses the horizontal lines that limit the ledges (Fig. 12.1).

Usually, the route is introduced into the contour of the quarry from its end in low places of the surface relief, which simplifies tracing inside the contours of the quarry field and reduces the amount of mining and construction work. When choosing the position of the route, they also take into account the need to ensure the stability of those sections of the sides where capital trenches are located, the possibility of increasing their service life, the convenience of placing stations and dumps on the surface and approaches to dumps, the length of tracks on the surface, as well as connecting paths between trenches and downhole tracks in a quarry, etc.

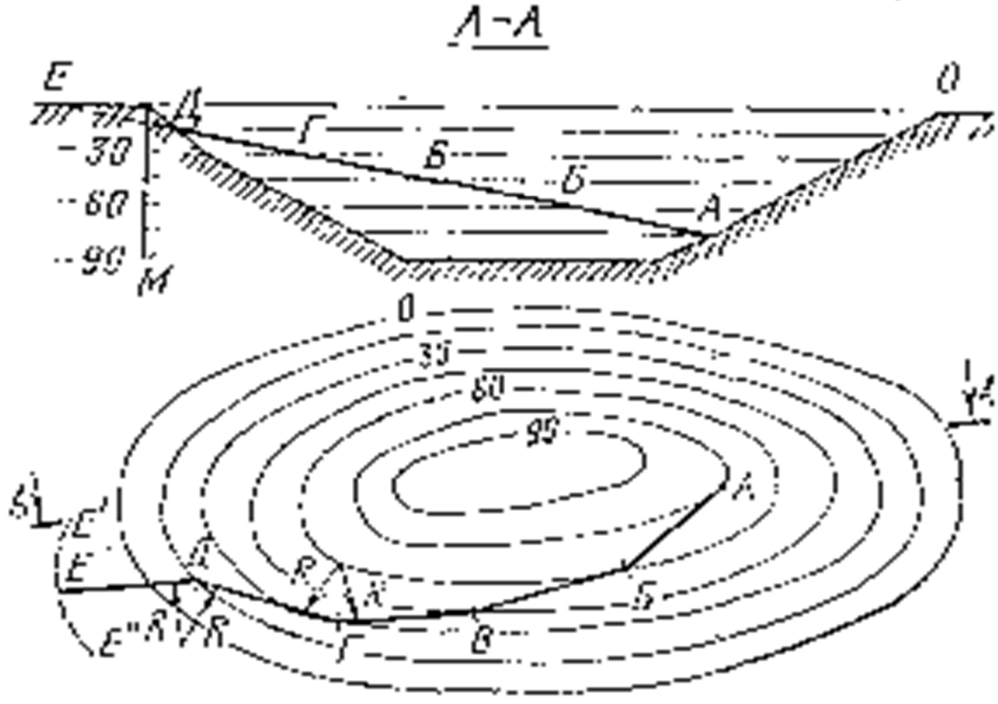


Fig. 12.1. The scheme of tracing capital trenches:

А, Б, В, Г, Д- points of alignment of the route to the horizons; E - the beginning of the route)

The main parameters of the route are the magnitude of the guiding lift, the difference in elevation marks of the beginning and end of the route, the radii of curved sections, the theoretical and actual length of the route, the number and design of points of junction of horizontal paths to inclined ones.

The theoretical length of the route D (m) is determined by the difference in the elevation marks of the No and Nh through which it passes, and the angle I of the slope of the route to the horizon (degree):

$$L_T=(H_o-H_x)/tg\ I=H/i_p,$$

when i_p - guiding ascent (slope) of the route.

The actual length of the route Ld (m) is greater than the theoretical one due to its elongation caused by a decrease in the slope angle of the route in curved sections and in areas adjacent to the trenches to the working horizons. Therefore, Ld = KuLt, where Ku is the coefficient of lengthening of the route.

On curved sections of the route, when using wheeled vehicles, the resistance to movement increases by the value of wk (N/t) and it is necessary to soften the rise of trenches to the value of id = ip - wk /g, q. The value of wk depends on the radius of the curve R. The smallest radius of the curve Rmin is set depending on the structural patency of the rolling stock. The value of Rmin affects the amount of separation of the sides of the quarry required for laying curves, as a result of which it is advisable in general to use rolling stock that allows the smallest radii of curves.

In railway transport, the smallest length of the profile element (a segment of track with a constant amount of lift) is determined from the condition of safe train movement. Constant movement is ensured if the train passes no more than one fracture of the track profile at any given time. Therefore, the length of one profile element should not be less than the length of the train.

Forms of routes of capital workings

The shape of the capital development route in the plan is simple if the route is located on one side of the quarry and does not change its direction along the entire length. A track is difficult if it consists of two or more sections of different directions connected to each other, or if it runs along all sides of the quarry. The routes of external trenches are always simple, internal trenches usually have complex routes.

The shape of the route in the plan is set in accordance with the size of the quarry field, the guiding rise and the profile elements.

If the actual length of the route of the internal trenches does not exceed the length of the quarry along the stretch on the corresponding horizon Lk, then the simple route will completely fit on one side. However , the condition $L_{д} = K_y H_{к}/i_p \leq L_{к}$ it is performed only with a favorable ratio of the length of the quarry field $L_{к}$ and the depth of the Nk quarry with this guiding rise ip and the coefficient of lengthening of the route K_y.

If $L_{д} = K_y H_{к}/i_p > L_{к}$, then the following two cases are possible when tracing.

1. The track is placed on one side of the quarry and its direction is changed from forward to reverse as many times n1 as it is necessary to place the track:

$$L_{д} = K_y H_{к}/i_p = n_1 L_k.$$

The value of n1 can be an integer or a fractional number. Straight sections of the route are connected at the same time by means of dead ends or loops of small radius. The loop connection (Fig. 8.2, a) is usually used for motor transport, and the dead-end connection (Fig. 8.2, b) is used for railway transport.

The placement of the entire route on one side of the quarry is rational when developing a deposit from the supine to the hanging side and moving the front in parallel. However, the presence of dead ends dramatically reduces the carrying capacity of the route, since the direction of movement of the train changes in dead ends, which requires its braking and stopping. The organization of traffic is also becoming more complicated. Therefore, dead-end routes should not be used, at least on the group of upper horizons of the quarry.

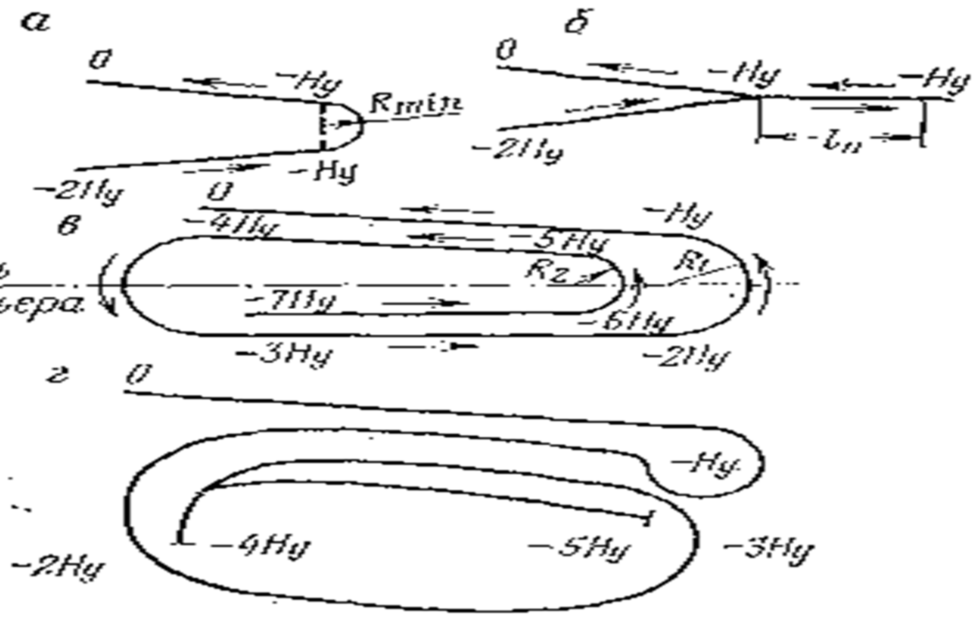


Fig. 12.2. Route diagrams in the plan:

lp – length of the abutment area

2. The route is carried out from one side to the other as many times n2 as necessary for its placement on the corresponding horizons of the sides with an average length of their perimeter P(м):

$$K_y H_к / i_p = n_2 P.$$

In this case, the track encircles the quarry in the form of a spiral (Fig. 8.2, b). The spiral route includes curved sections that are located on the end sides of the quarry and usually have a large radius. The placement of curves in this case does not cause difficulties and, as a rule, it is not necessary to create special half-mounds or half-recesses.

Often the internal route includes both straight, spiral and dead-end (loop) sections (Fig. 12.2, d). With the construction of such complex routes, the conditions for opening individual horizons, the efficiency of quarry transport and the use of a rational development system are improved.

The inner highway is a direct continuation of the outer one. Such a mixed route is usually used for opening in deep quarries: several upper horizons are opened using an external route, and an internal route is brought to the underlying horizons of the quarry.

The deepening of the route of internal capital trenches is determined through the average value of its slope and the actual length.

A simple route is used in the development of deposits with a significant strike at a small depth of the quarry, and a dead—end one - with a relatively small size of the field along the strike, especially with a steep drop, when the size of the quarry is small across the strike. A loop track is created when opening internal trenches, if vehicles are used, and, when possible, when rail transport. A spiral route is arranged if the use of a loop or dead-end route is impossible or irrational due to the conditions of the occurrence of ore bodies, the separation of the sides, the required carrying capacity, the efficiency of the quarry transport. The reconstruction of railway tracks with a spiral route is very difficult and therefore it should be stationary in this case. In road transport, periodic reconstruction of highways is quite acceptable.

Reference words: trench route, path plan, longitudinal profile of the path, tracing, base for tracing, theoretical length of the route, actual length of the route, simple, complex, mixed.

1. ***Control questions:***
2. ***1. What is called the trench route?***
3. ***2. What is the tracing of the path?***
4. ***3. Which route is called simple?***
5. ***4. Which route is called difficult?***
6. ***5. In what cases are mixed routes used?***

Lecture 8

Topic: Division of the quarry field into excavation layers. Height and stability of ledges.

Plan:

1. Horizontal excavation layers.

2. Inclined excavation layers.

3. The influence of the height of the ledge on a number of general career indicators.

4. Stability of the slope of the ledge.

Open-pit mining is characterized by a certain order of excavation and movement of minerals, covering and enclosing rocks. For the systematic development of rocks and rational use of equipment, the quarry field is divided into separate excavation layers, in most cases horizontal (see Figure 2.2). The excavation of the layers is carried out sequentially from top to bottom, regardless of the direction of the stratification of rocks.

The possible number of layers depends on the depth and size of the quarry in the plan. The thickness of the layers in the depth of the quarry can be different. With simultaneous working out of layers, ledges are formed.

The number of ledges along the deposit in the profile of the quarry field depends on the thickness of the deposit (Fig. 5.1), the angle of its fall, the difficulty of developing rocks used by excavation and loading vehicles.

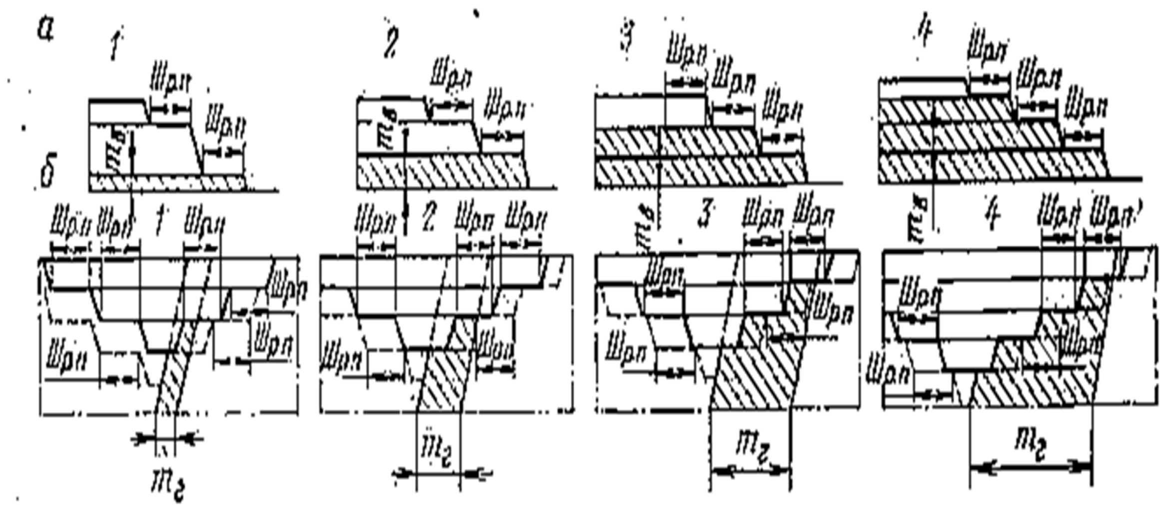


Figure 5.1. Schemes of mineral deposits:

a - horizontal; b - inclined and steep; 1, 2, 3 and 4 - deposits of very small, small, medium power and powerful, respectively; Wr.p - width of the working platform of the ledge.

Deposits of very low power are worked out with one ledge; at the same time, horizontal deposits with a vertical capacity of mv ≤ 2÷ 3 m are inefficient to develop with single-bucket excavators, and with inclined and steep deposits with a horizontal capacity of mg≤ 20÷ 40 m, cutting a new ledge is associated not only with the complete excavation of minerals on the overlying horizon, but also with additional by moving the ledge along the host rocks (Fig. 5.1, a, 1 and b, 1). The extraction of minerals from horizontal deposits of low power (mv = 4-20 m) is carried out by one ledge of normal height, and with inclined and steep deposits (mg = 20-40 m), the cutting of the next ledge is possible after the excavation of the deposit on the above horizon (Fig. 5.1, a, 2 and b, 2). Deposits of medium capacity (mv = 15÷40 m, mg = 50÷120 m) in one profile of the quarry can be simultaneously developed by two ledges (Fig. 5.1, a, 3 and b, 3). Powerful deposits (mv>20÷40 m, mg >80÷150 m) are developed by three or more ledges or sub-steps.

Sometimes , with gentle and inclined deposits , development is carried out

inclined layers (ledges) of different thickness (depending on the thickness of the layers) on the stratification of rocks (Fig. 5.2, a). Individual layers are developed sequentially, ahead of schedule. In rare cases, mining is carried out in steep (more than 25-30 °) layers, starting from the middle of the quarry field to its borders (Fig. 5.2, b). Such excavation is possible only when developing steep deposits and homogeneous stone massifs. It allows in stable arrays to provide steeper slopes of the working sides of the quarry in these conditions and reduce the current volume of stripping operations. However, with such a excavation, the opening of horizons and the transportation of rock mass are significantly complicated.

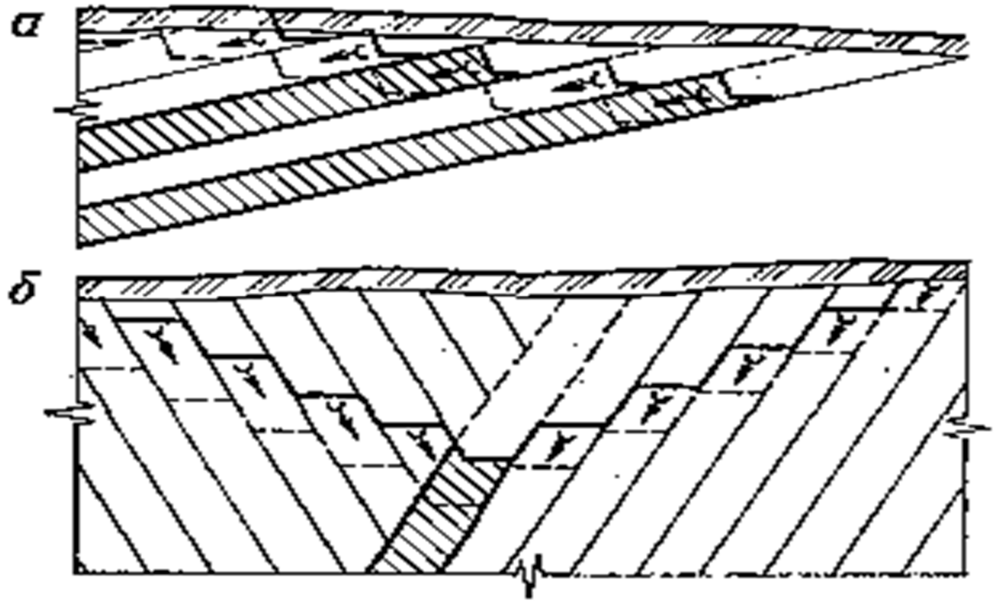


Fig. 5.2. Schemes of development of deposits by inclined and steep layers.

The ledge is one of the most important elements of open development. The height of the ledge is rational, at which, under these conditions, the following are ensured: safety of mining operations, high productivity of equipment, minimum volumes of auxiliary work, established annual volumes of mining and stripping operations and minimum costs for them.

The height of the ledge directly affects a number of general career indicators: the quality of the extracted mineral; the rate of advance of the front; the rate of deepening of mining operations and, consequently, the production capacity of the quarry; the construction period of the quarry; the volume of mining and capital works; the total length of the work front, intra-quarry paths and roads; the angle of slope of working and non-working sides.

The rational height of the ledges cannot be determined by any one factor; it must be chosen on the basis of determining the cumulative influence of all the factors listed above in specific natural conditions and taking into account the possibilities of opening working horizons. Analytical methods for determining the height of a ledge cannot take into account the totality of these factors. Safety of mining operations is a basic requirement.

When developing horizontal and shallow deposits, the thickness of the deposits and the covering rocks usually determines the height and number of ledges. When alternating horizontal and shallow layers, the height of the ledge is determined depending on the thickness of the individual layers and the layers of empty rocks lying between them, taking into account the necessary quality of the mineral.

The stability of slopes in soft rocks is crucial. In such rocks, according to the Safety Rules, the height of the ledge should not exceed the maximum height of the excavator scooping; otherwise, "visors", "overhangs" remain in the upper part of the ledge and rock collapse is possible.

When developing inclined and steeply falling deposits, represented mainly by rocky and semi-rock formations, the height of the ledge is determined mainly by indicators of technological processes, losses and dilution of minerals, the required production capacity of the quarry and the conditions for opening working horizons. The costs of preparing the rocks for excavation and transporting the blasted rocks decrease with an increase in the height of the ledge. The minimum costs for excavating the blasted rocks correspond to the height of the ledge 15-20 m.

At the same time, according to the Rules of Technical Operation (PTE), the height of the ledge in rock and semi-rock formations should not exceed the maximum height of the excavator's scooping by more than 1.5 times (Table 5.1), provided that the height of the collapse will not exceed: with one— and two-row blasting - the maximum height of the scooping, excavator, and with multi—row blasting - one and a half times the maximum height of scooping. When excavating blasted rocks from such ruins, additional measures should be taken to prevent the formation of "visors" and "overhangs". Only in rocks of I and II fracturing categories with normal and increased explosive consumption and inclined drilling of wells, when the blasted rocks will be in a loose state, with the permission of the Gosgortehnadzor authorities, the height of the ledge is allowed $H_у > 1,5\ H_{ч.мах}$.

Table 5.1

Maximum height of ledges in rock and semi-rock formations when excavated by quarry-type mechlopats

Экскаватор	Максимальная высота черпания экскаватора, м	Максимальная высота уступа, м
ЭКГ-2	8,5	12,5
ЭКГ-3,2	10,0	14,5
ЭКГ-5	11,0	16,5
ЭКГ-8	12,5	19,0
ЭКГ-12,5	15,6	23,5
ЭКГ-20	18,0	27,0

In cases where excavation is carried out by excavators with elongated working equipment with top loading, the height of the ledge should correspond to the parameters of the tunneling equipment (Table 5.2).

During the development of complex deposits, losses and dilution of minerals are almost directly proportional to the height of the mining ledge. Therefore, it is advisable to take the height of mining ledges no more than 10-12 m and use quarry-type excavators with bucket capacity under economic conditions when excavating minerals separately, especially in shallow deposits. 3—5 $м^3$.

Table 5.2

The maximum height of the ledge when excavated by mechanical shovels with elongated working equipment with top loading

Экскаватор	Высота уступа (м) в породах		
	мягких *	полускальных **	скальных ***
ЭКГ-2у	5	7	10
ЭКГ-3,2у	5,5	8	9
ЭКГ-4у	8	13	13
ЭКГ-6,3у	13	18	19

* Угол устойчивого откоса уступа 34°.
** То же, 45°.
*** То же, 70°.

The speed of the trenches is approximately inversely proportional to the height of the ledge. The rate of deepening of mining operations depends on this speed. The higher the height of the ledge, the smaller the possible production capacity of the quarry for minerals. This position is especially important in the first period of the career.

At the same time, the required volumes of stripping work decrease with an increase in the height of the ledge as a result of a reduction in the number of working ledges and an increase in the angle of slope of the working side of the quarry. At the same time, the required speed of movement and the length of the mining front are also reduced. Therefore, when developing deposits of a simple structure in the middle zone of the quarry (in depth), it is sometimes advisable to increase the height of the ledge.

From the experience of conducting open works in the development of inclined and steep deposits of a simple structure, it was found that the optimal height of the ledge when using excavators with E = 3 - 5 m3 is 12 - 15 m and 17 - 20 m for excavators with E = 8 - 12.5 m3. With multi—row blasting of vertical borehole charges in rocks of III-V fracturing categories, often under safety conditions Nu = Lf.max + h (h = 1 ÷ 3 m).

In the open-pit mining of mineral deposits, it is very important to ensure the stability of the ledges and prevent their deformation during the entire period of construction and operation of the quarry.

Of the many factors on which the stability of slopes depends, the determining one is a group of geological factors (composition, condition, structure and properties of rocks). They determine the conditions of deformation of the array and the choice of calculated schemes of slope stability, the nature of anti-deformation measures and the values of calculated indicators.

Of the group of hydrogeological factors, the main one is the influence of groundwater that changes the properties of the massif (due to leaching of fractured carbonate rocks, swelling of clay rocks, etc.) and its stressed state; under the action of hydrostatic and hydrodynamic forces, filtration destruction of slopes can occur (flooding and suffusion). The waterlogging of contact zones and structural disturbances leads to deformations of slopes (as a result of a decrease in the strength of rocks at the contacts) and a sudden breakthrough of waters.

The third group consists of technological factors.

It is necessary to take into account that the parameters of the opening workings, their position relative to the contour of the quarry and their service life determine the intensity of the development of rheological processes and weathering of rocks in the array, the development of deformation processes in the array (pruning of layer contacts or violations, etc.). At a high rate of movement of the mining front in the array, deformation and rheological processes do not have time to develop, which allows you to give the slopes of the working ledges steeper angles of inclination. The placement of dumps in the worked-out space increases the resistance to the shear forces of the instrument array of rocks.

The sides of quarries may have areas of concave, convex and rectilinear shape in plan. It was found that, other things being equal, slopes having a concave shape in plan are more stable than flat ones.

Blasting operations cause a seismic effect, the formation and development of fracturing and zones of reduced strength in the slope of the ledge, as well as the unstable surface of the slope of the ledge itself. To reduce the harmful effects of explosions when setting the ledges in the final position, it is necessary to: change the parameters of drilling and blasting operations; apply (taking into account the specific situation) short-delayed blasting of borehole charges of the required diameter and contour blasting, charges with inert cores; place rows of wells at an angle of 60-90 ° to the contour of the side; apply shielding; use artificial reinforcement of ledges; introduce an increased coefficient of stability margin into calculations.

There are short-term and long-term stability of slopes, which should have working and non-working ledges, respectively. The coefficient of stability margin of working ledges is pu = 1.15 ÷ 1.2, and non-working ones in clay and fractured rock and semi-rock rocks are pu = 1.5 ÷ 2.

When pre-selecting the angles of slopes of working and non-working ledges, it is advisable to use the data given in Table 5.3. To clarify the values of the angles, especially with unstable rocks or unfavorable occurrence of weakening surfaces, it is necessary to conduct field studies and calculations of slope stability.

Table 5.3

Slope angles of ledges

Группа пород	Характеристика породного массива	Высота одиночного уступа, м	Угол откоса уступа, градус		
			рабочего	нерабочего: одиночного	нерабочего: сдвоенного или строенного
Скальные породы, $\sigma_{сж} > 8 \times 10^7$ Па	Весьма крепкие осадочные, метаморфические и изверженные породы	15—20	До 90	70—75	65—70
	Крепкие малотрещиноватые и слабовыветрелые осадочные, метаморфические и изверженные породы	15—20	До 80	60—75	55—60
	Крепкие трещиноватые и слабовыветрелые осадочные, метаморфические и изверженные породы	15—20	До 75	55—60	50—55
Малопрочные скальные, полускальные породы, $\sigma_{сж} = 8 \cdot 10^6 \div 8 \cdot 10^7$ Па	Осадочные, метаморфические и изверженные породы зоны выветривания, относительно устойчивые в откосах известняки, песчаники, алевролиты и другие осадочные породы с кремнистым цементом, конгломераты, гнейсы, порфириты, граниты, туфы	10—15	70—75	50—55	45—50
	Значительно выветрелые осадочные, метаморфические и изверженные породы и все породы, интенсивно выветривающиеся в откосах (аргиллиты, алевролиты, сланцы и др.)	10—15	60—70	35—45	35—40
Мягкие и сыпучие породы, $\sigma_{сж} < 8 \times 10^6$ Па	Глинистые породы, а также полностью дезинтегрированные разности всех пород	10—15	50—60	40—45	35—40
	Песчано-глинистые породы	10—15	40—50	35—45	30—40
	Песчано-гравийные породы	10—15	До 40	30—40	25—35

Примечание. При падении слоев, рассланцованных толщ, тектонических трещин и других поверхностей ослабления в сторону карьера под углом 30—65° (если трещины заполнены глиной, то под углами более 25°) угол откоса уступа должен соответствовать углу падения этих поверхностей ослабления, но быть не более приведенных в таблице.

The width of the prism (m) of the possible collapse of the ledge (Fig. 5.1) in the absence of weakening surfaces can be determined depending on the angle of slope and the strength of the rocks by the formula

$$Z = \frac{2H_y\left[1 - ctg\alpha tg\left(\frac{\alpha+\rho}{2}\right)\right] - 2H_{90}}{ctg\left(45^0 - \frac{\rho}{2}\right) + tg\left(\frac{\alpha+\rho}{2}\right)},$$

when α — slope angle of the ledge, degree; ρ — the angle of internal friction of the rock, degree; H_{90} — height of vertical separation crack, m,

$$H_{90} = \frac{2K}{\gamma g} tg\left(45^0 + \frac{\rho}{2}\right),$$

K — rock adhesion, Па; γ — rock density, кг/м^3.

Depending on the structure of the rock mass, the value of Z is (according to G. L. Fisenko):

(0,1 ÷ 0,2) H_y — when the attenuation surfaces fall towards the array (Fig. 11.1, a);

(0,25 ÷ 0,3) H_y — with a steep and inclined fall of the attenuation surfaces towards the developed space (Fig. 5.1, b);

(0,3 ÷ 0,4) H_y — in case of horizontal occurrence or gentle fall of the attenuation surfaces towards the developed space (Fig. 11.1, b).

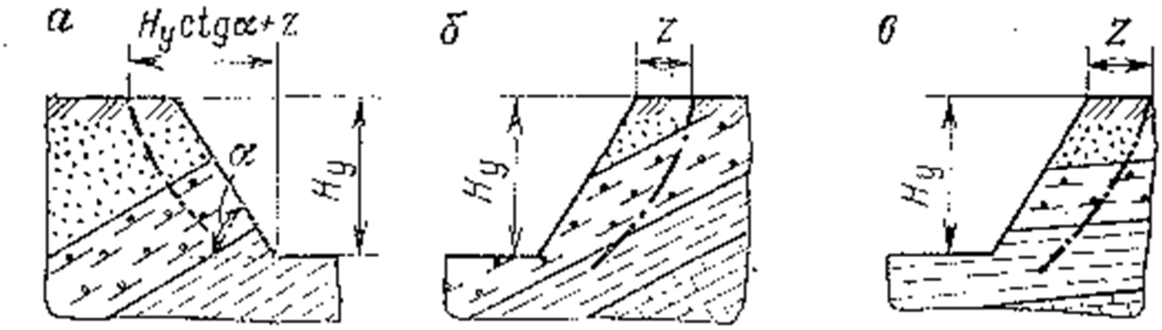


Fig. 5.1. Schemes for determining the width of the prism of a possible collapse of the rocks of the ledge.

Reference words: excavation layers, horizontal, inclined, steep, excavation of layers, number of layers, simultaneous development of layers, deposit capacity, deposits of very small, small, medium power and powerful, width of the working platform of the ledge, open-pit element, rational height, factor, analytical methods for determining the height of the ledge, development of horizontal and shallow deposits,

Security questions:

1. Why is the quarry field divided into excavation layers?
2. Which layers are called horizontal?
3. Which layers are called inclined?
4. What is the angle of inclination of the steep layers?
5. How many ledges work out deposits of very low capacity?
6. What is the height of the ledge considered rational?
7. What should be the height of the ledge in the development of soft rocks?
8. What should be the height of the ledge in the development of rocky and semi-horizontal rocks?
9. What factors affect the stability of the slopes of the ledge?

Lecture 9

Topic: Basic concepts of the mining front

Plan:

1. **1. The front of mining operations by location and structure.**
2. **2. The front of mining operations in the direction of movement of the rock mass.**
3. **3. The front of mining operations for loading rock mass and the number of transport cargo exits.**

The direction of development of mining operations on the ledge is not chosen arbitrarily. The location of the split trench (pit) should correspond to the design plan of mining operations in order to ensure the necessary number of overburden and mining faces during the operational period of layer development, the regularity of overburden and mining operations.

The front of the ledge works differs in the following features:

1. By location.

The work front is located along the long axis of the quarry field (Fig. 6.1, d, e, i and 6.2, a). A significant length of the work front and transport communications is achieved, and the speed of its movement is small (30-60 m/ year). Favorable conditions are being created for the separate extraction of minerals of various grades, there are large reserves for increasing the intensity of field development and quarry capacity. This location of the front causes a large amount of mining and capital work during the construction of the quarry. It is expedient and common in cases where the thickness of the rocks covering the deposit is relatively small.

The work front is located along the short axis of the quarry field (Fig. 6.1, a, b, c, w and 6.2, b). The length of the work front and transport communications are small, and the speed of its movement reaches 70-300 m/ year. Reserves for increasing the production capacity of the quarry, the possibility of separate excavation and the creation of large uncovered mineral reserves are small. Such an arrangement of the front provides relatively small amounts of mining and capital works, but complicates the opening of horizons and the operation of transport communications due to the need for frequent reconstructions. It is common in the thick thickness of the covering rocks, as well as in the development of powerful steep deposits using mobile means of transport.

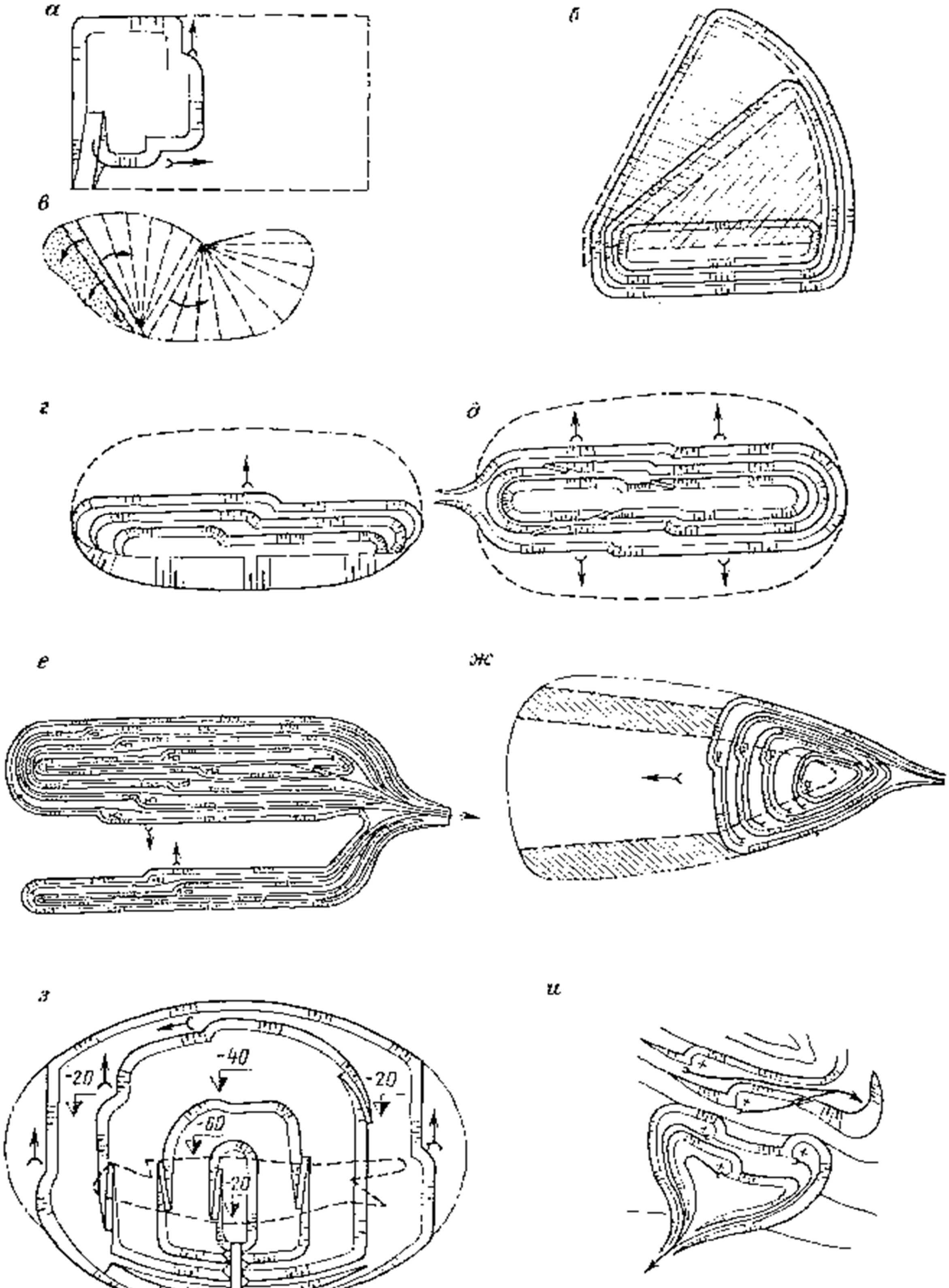


Figure 6.1. Schemes of various options for the direction of development of mining operations (arrows indicate the direction of movement of individual faces and working sides of quarries).

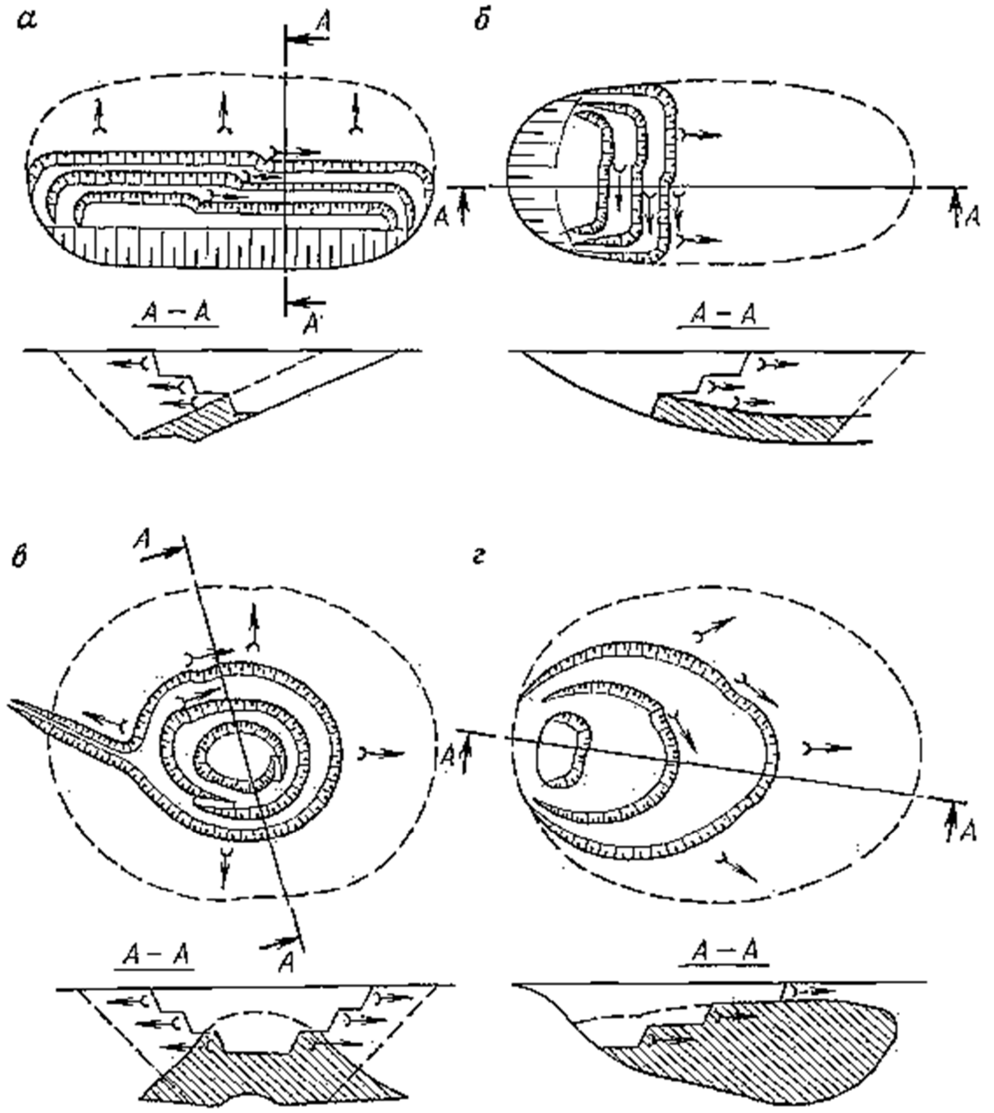


Fig. 6.2. Schemes of mining development:

a, b, c and d — the work front is located respectively along the long axis of the quarry, along the short axis, concentrically and ellipsically

The work front is located concentrically (see Fig. 6.2, c) or in an ellipse (Fig. 6.1, h and 6.2, d). The length of the front, transport communications and the number of faces at different stages of the development of work on the ledge are different. Such an arrangement of the front ensures minimal amounts of mining capital and preparatory work when cutting new ledges and a high rate of deepening of mining operations. However, at the same time, periodic changes in the position of the opening workings are inevitable. Opportunities to increase the production capacity of a quarry are usually limited.

2. By structure.

A homogeneous front of work — if it is composed only of overburden rocks or only minerals of the same grade. At the same time, the gross excavation of the rock mass is carried out. A homogeneous front can be solid (Fig. 6.3, a) and divided into blocks with independent faces (Fig. 6.3, b and c). The division into blocks is due to the need to place the required number of excavators and other technical means on the ledge. It is economically efficient to install one powerful excavator on the ledge. However, with large volumes of work and the absence of excavators of the required capacity, it is necessary to place two or even three excavators on the ledge.

A heterogeneous front of work — if blocks of waste rocks, minerals and their various grades alternate within it (Fig. 6.3, d). The excavation in the faces with a heterogeneous front is gross. Division into blocks (using two or three excavators), as a rule, is necessary to ensure continuous extraction of minerals.

A complex heterogeneous front of work is when it is practically impossible to isolate blocks with only empty rocks or only with minerals of the same grade within its limits (Fig. 6.3, d). In this case, separate excavation of the rock mass is carried out.

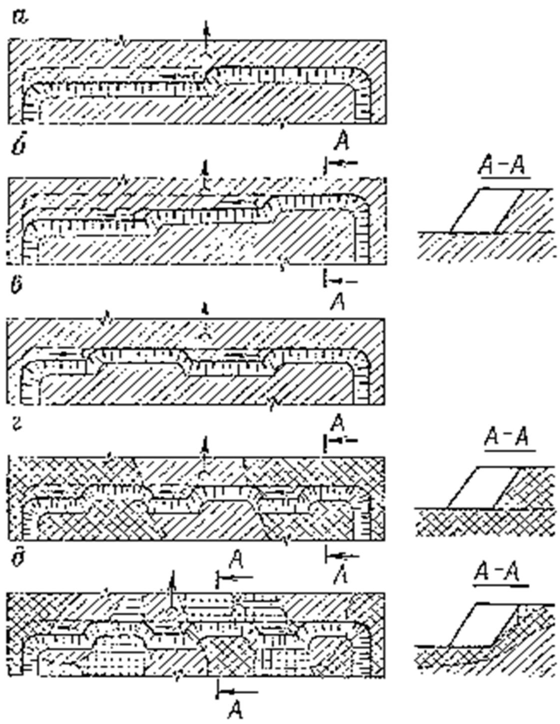


Figure 6.3. Schemes for dividing the front of the ledge work by structure.

3. In the direction of movement of the rock mass.

The front of work with transverse movement of rock mass — when storing overburden rocks in the worked-out space using overburden excavators and transport dump units (Fig. 6.4, a), as well as when excavating the rock with bulldozers or scrapers, front face and moving it along the shortest distance to the inner or outer dump (Fig. 6.4, b).

The front of work with the longitudinal movement of rock mass — when moving it from the faces using quarry transport (Fig. 6.4, c).

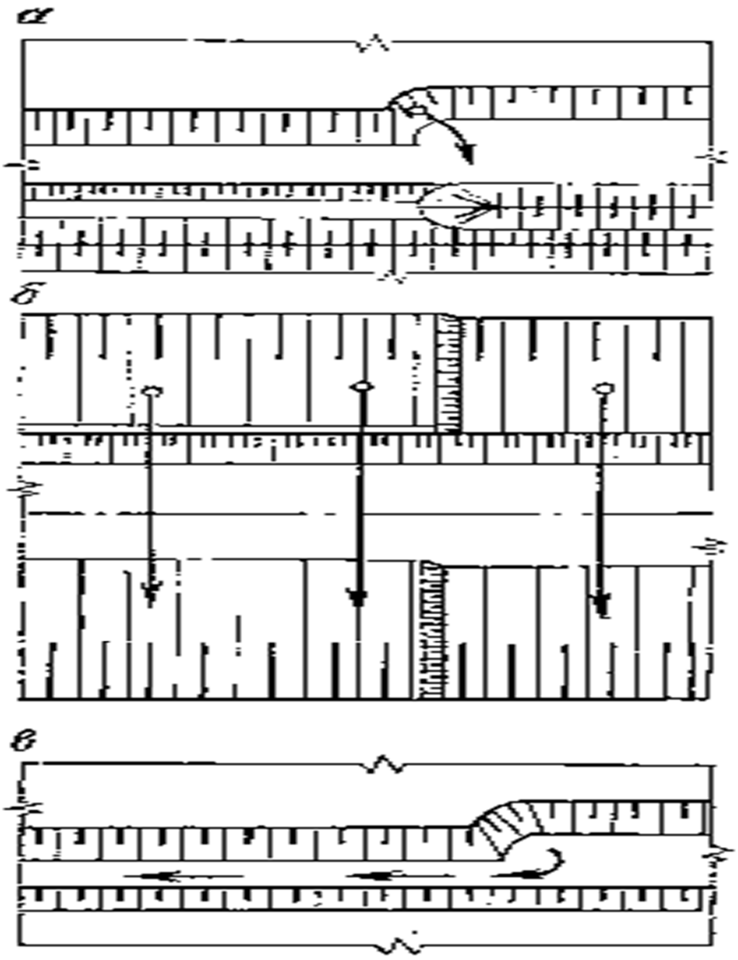


Fig. 6.4. Schemes of movement of rock mass relative to the front of the ledge work

4. On loading of rock mass.

The bottom loading of the rock mass on the horizon of the installation of the excavation and loading equipment (Fig. 6.5, a) is widespread and provides the most economical and productive use of each section of the mining front of the ledge.

The upper loading of the rock mass (Fig. 6.5, b) is necessary and expedient in cases where it is difficult or unprofitable to arrange transport communications on the soil of the ledge being developed, — when carrying out a trench, small amounts of work on the horizon, etc.

The upper excavator transshipment of rock mass (Fig. 6.5, c) is characterized by the fact that the rock or mineral is stored by an excavator on the upper platform of the ledge and then loaded into vehicles by another excavator. Such a scheme is used in special cases, for example, when finalizing the lower horizons of deep quarries, on slopes with a small amount of work, when using hydraulic transport, draglines, etc.

The lower excavator transshipment of rock mass (Fig. 6.5, d) is characterized by the movement of rock or minerals by an excavator to the underlying horizon and loading it with another excavator into vehicles. The use of such a scheme is advisable on slopes, to reduce the height of the ledge, with small amounts of work, to improve the working conditions of transport, etc.

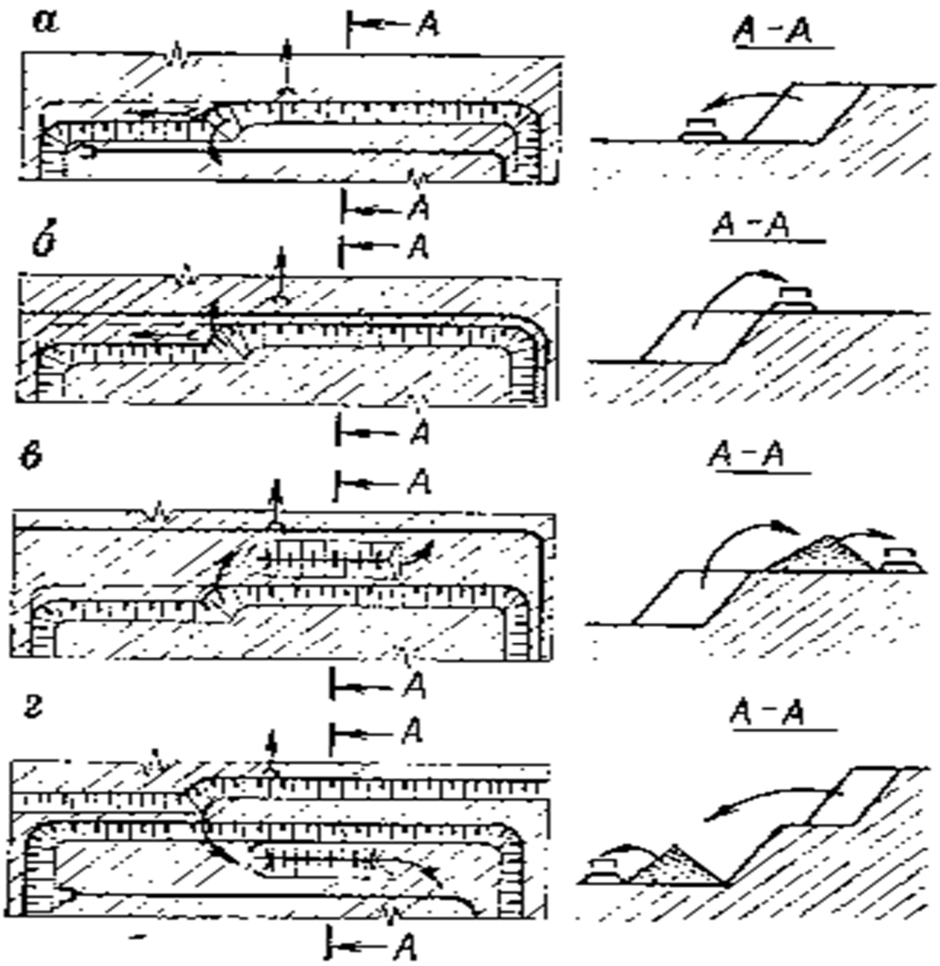


Figure 6.5. Schemes of loading rock mass on the ledge.

5. By the number of transport cargo exits.

Single front — if it has one cargo transport exit from the ledge (Fig. 6.6, a, b and c). Such a front is typical for most quarries when using various mining and transport equipment.

A double front is if it has two cargo transport exits from the ledge (Fig. 6.6, d, e). The front of such a design is two single fronts and can be used for a large length of surface—type quarries, as well as for a group of upper ledges of powerful deep-type quarries.

In rare cases, a built-up front is possible (Fig. 6.6, e).

A dead—end front (with return traffic) - if a single front on a ledge has one common transport exit serving for the supply of empty trains or cars and for the delivery of goods (Fig. 6.6, a, b, d, d and e). The dead-end front has become the most widespread in all types of quarry transport.

Through front (with flow traffic) — if a single front on a ledge has two or more specialized transport exits: separately for the supply of empty and separately for cargo (Fig. 6.5, b). A double dead-end front can also be used periodically as a single through front (see Fig. 6.6, d), and a built-in a dead—end front is like a double through front (see Fig. 6.6, e).

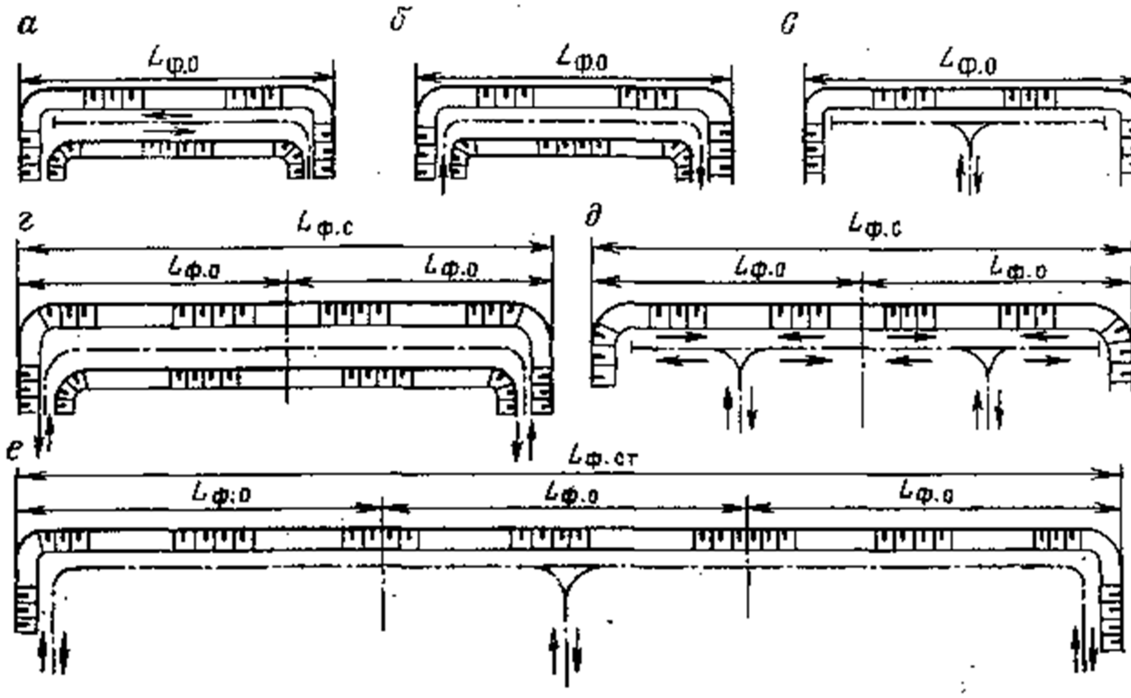


Fig. 6.6. Schemes of structures of the mining front:
Lf.o, Lf.c and Lf.st – the length of the work front, respectively, single, double and built.

1. *6. By the nature of the movement of vehicles:*
2. *a) a dead-end front, with return traffic, - if a single front on a ledge has one common transport exit, serving for the supply of empty trains or cars and for the delivery of goods (Fig. 6.6, a, b, d, e, e). A dead-end front it has become the most widespread in all types of quarry transport.*
3. *b) Through-front, with flow-through traffic, - if the single ledge front has two or more specialized transport exits: separately for empty feed and separately for cargo (Fig. 6.6, b). The double dead-end front can also be used periodically as a single through-front (see Fig. 6.6, d), and the constructed dead–end front is like a double through front (see Fig. 6.6, e).*
4. *7. By the position of the transport exit.*
5. *Flank front — if the transport exit is located on the flank of the ledge front (Fig. 6.6, a, b and d); it is used when opening working horizons with stationary workings.*
6. *Central front — if the transport exit is located within the front (Fig. 6.6, b and e). Such a front is used when opening workings are located on the working side of the quarry and on the mining ledge when developing horizontal or shallow deposits.*
7. *The listed characteristics of the work front of the ledge serve as the basis for the correct choice of the system, development, opening and application of technical means.*
8.
9. *Reference words: signs, along the long axis, along the short axis, concentrically, homogeneous front, heterogeneous, composite, with transverse displacement of rock mass, with longitudinal displacement of rock mass, lower loading, upper loading, single front, double front, dead-end, through, flanking, central.*
10.

11.

12. Security questions:

13. 1. By what signs does the front of mining operations differ?

14. 2. What can be the front of work by location?

15. 3. What can be the front of work on the structure?

16. 4. What can be the front of work in the direction of movement of the rock mass?

17. 5. What can be the front of work on the position of the transport exit?

Lecture 10

Topic: Directions of movement of the front of work. The length and speed of the work front movement.

Directions of movement of the work front

The ledge, as a rule, is divided into panels along the front of the work (Fig. 7.1). The panels can also be tabs at the same time. One or more panels can be worked out on the ledge at the same time. As the panels are being worked out, the working front of the ledge moves. After working off the panel, it is necessary to rewire the transport communications located along the front of the work.

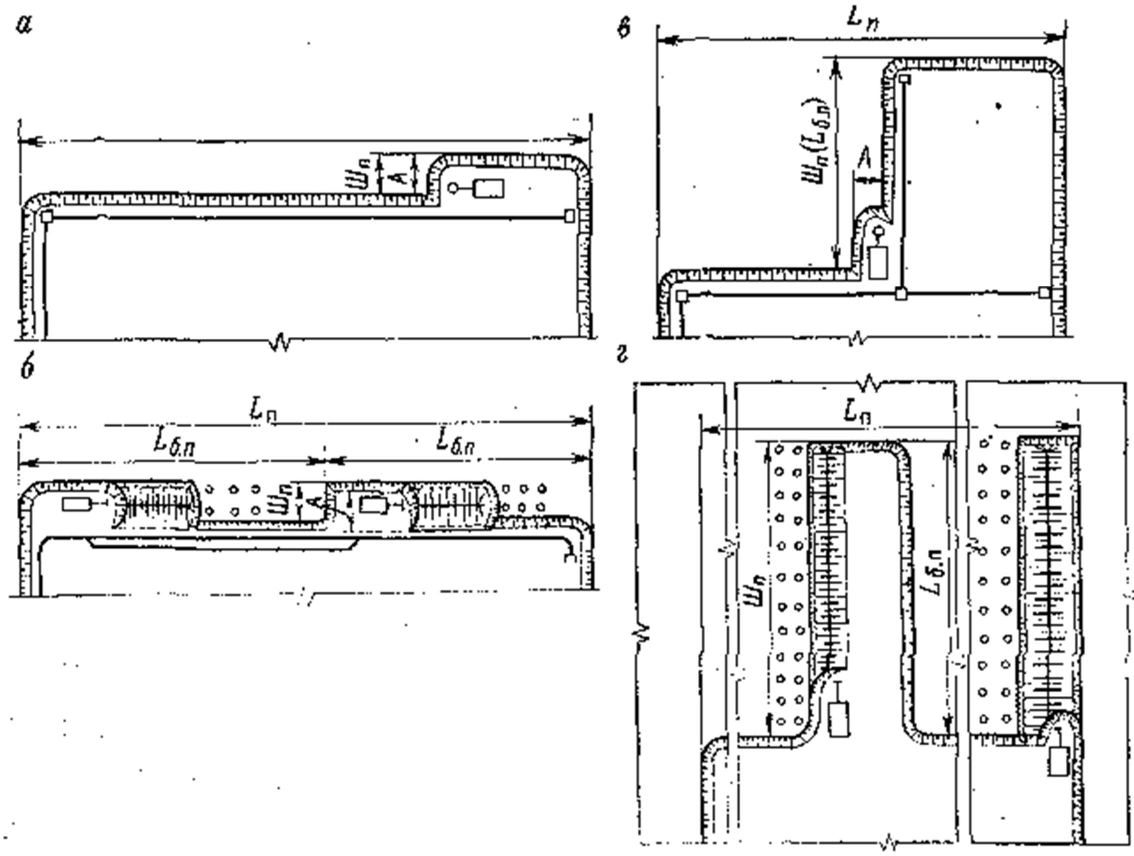


Figure 7.1. Diagrams of panels and panel blocks:
a and b, respectively, for longitudinal blocks and entrances; c and d, respectively, for transverse blocks and entrances (wide panels)

The panel is characterized by the height of the ledge Well, length Lp and width Sp; with one panel on the ledge, its length is equal to the length of the work front of the ledge Lf.y.

The part of the panel allocated for development by one excavation machine is called a panel block (for example, an excavator block); within the panel, one or more such blocks with a length of Lb.p. can operate simultaneously (see Fig. 7.1). Panel blocks, in turn, can be divided into working blocks; within each of they perform any one workflow, such as drilling, blasting, excavation (Fig. 7.2). The excavation of rocks in each block is carried out in narrow strips, called excavation approaches. In some cases, the entries are also panel blocks (see Figure 7.1). The panel blocks and recess entries, depending on their location relative to the ledge front, can be longitudinal (along the ledge front, □ = 0°, Fig. 7.3, a, g, w and k), transverse (across the ledge front, □ = 90 °, Fig. 7.3, b, d, z and l) and diagonal (0°<□<90°, fig. 14.3, b, e, i and m). Longitudinal panel blocks and dredging are used in the application of all types of transport, transverse — usually in automotive and conveyor.

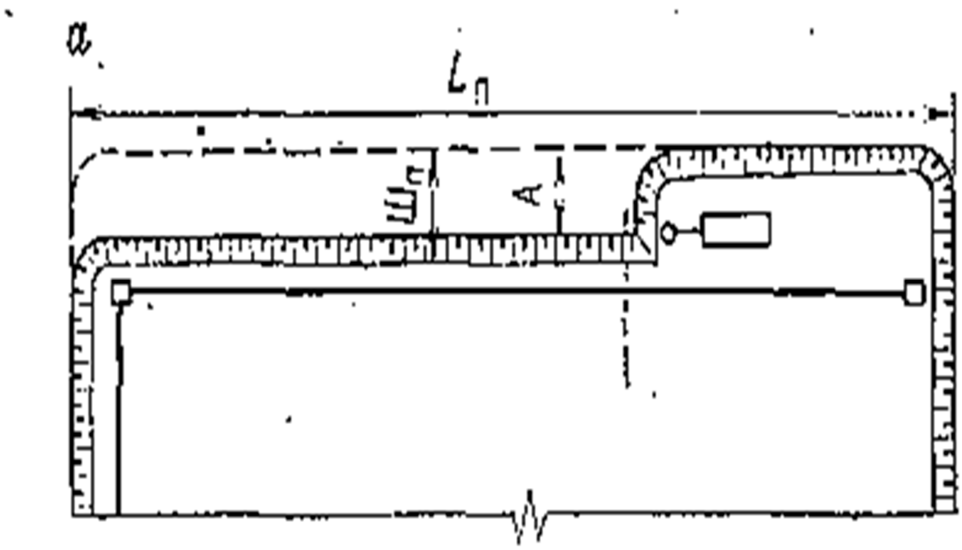


Fig. 7.2. Schemes for dividing the panel blocks into working blocks:
Lv (Le), LB and Lp.b — the length of the blasted (excavated), drilling and prepared for drilling blocks; 1 — the downhole path.

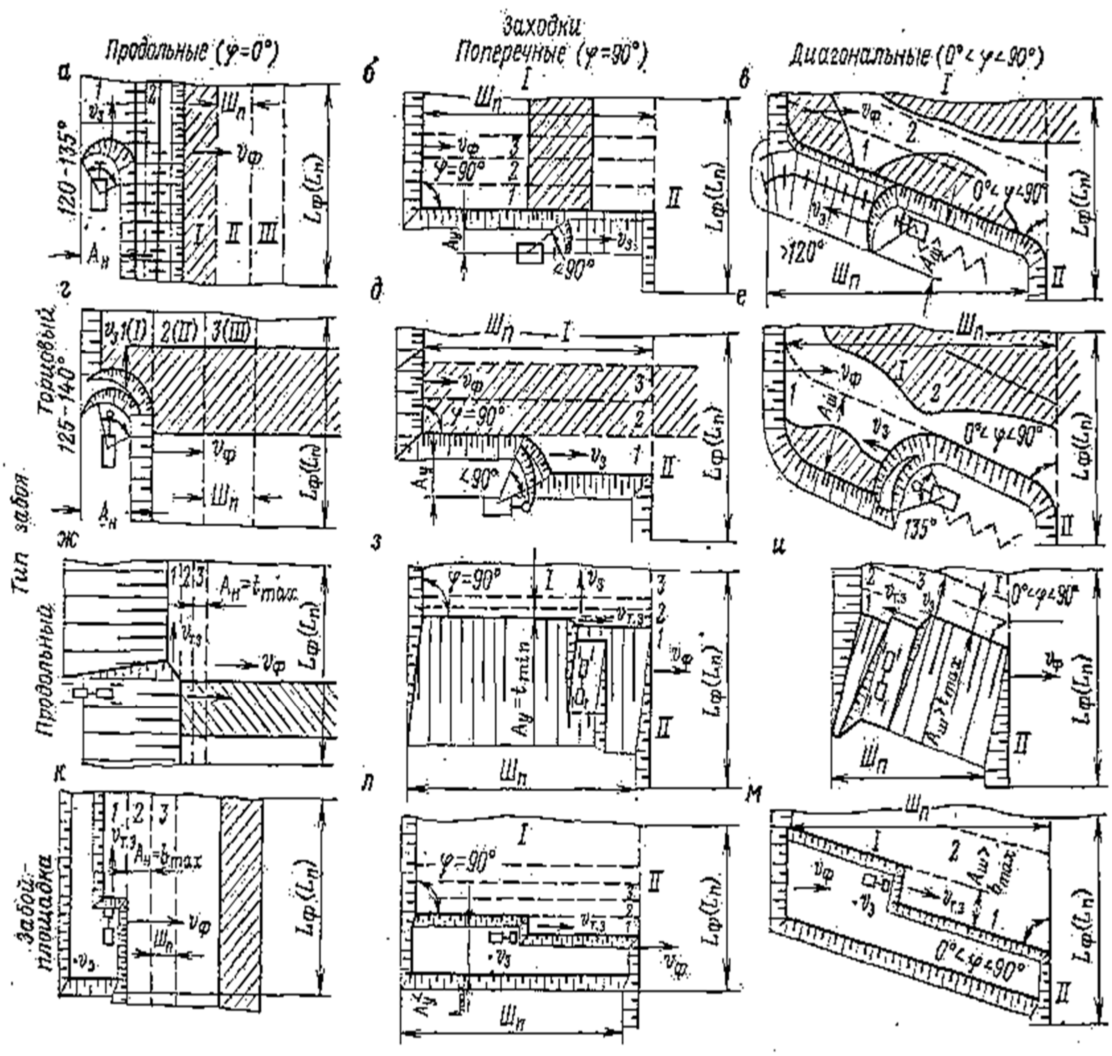


Figure 7.3. Panel and entry block diagrams:
I, II, III — panels; 1, 2, 3 — panel and entry blocks.

The width of the entrance A at the end face and the face-site corresponds to the width of these faces. There are normal, narrow and wide approaches (see Figure 7.3). Under normal approaches, the rock is excavated when the machines move along a rectilinear axis within the entire length of the approach, provided that the linear parameters of the machines are used to the maximum. Narrow z a stroke k and differ from normal by incomplete use of the working parameters of the excavation machines. Wide approaches are characterized by a variable direction of movement of cars in the plan (zigzag axis).

The work front within the boundaries of the layer can move:

1. Parallel to the long or short axis of the career field from one border to the other (opposite) (Fig. 7.4, a). In this case, the ledge has one working slope (single-side recess), the second side of the ledge is non-working.

This option is used for the development of horizontal and shallow deposits with a significant length of the quarry field. It is characterized by large volumes of mining and preparatory work, even with a small capacity of the covering rocks.

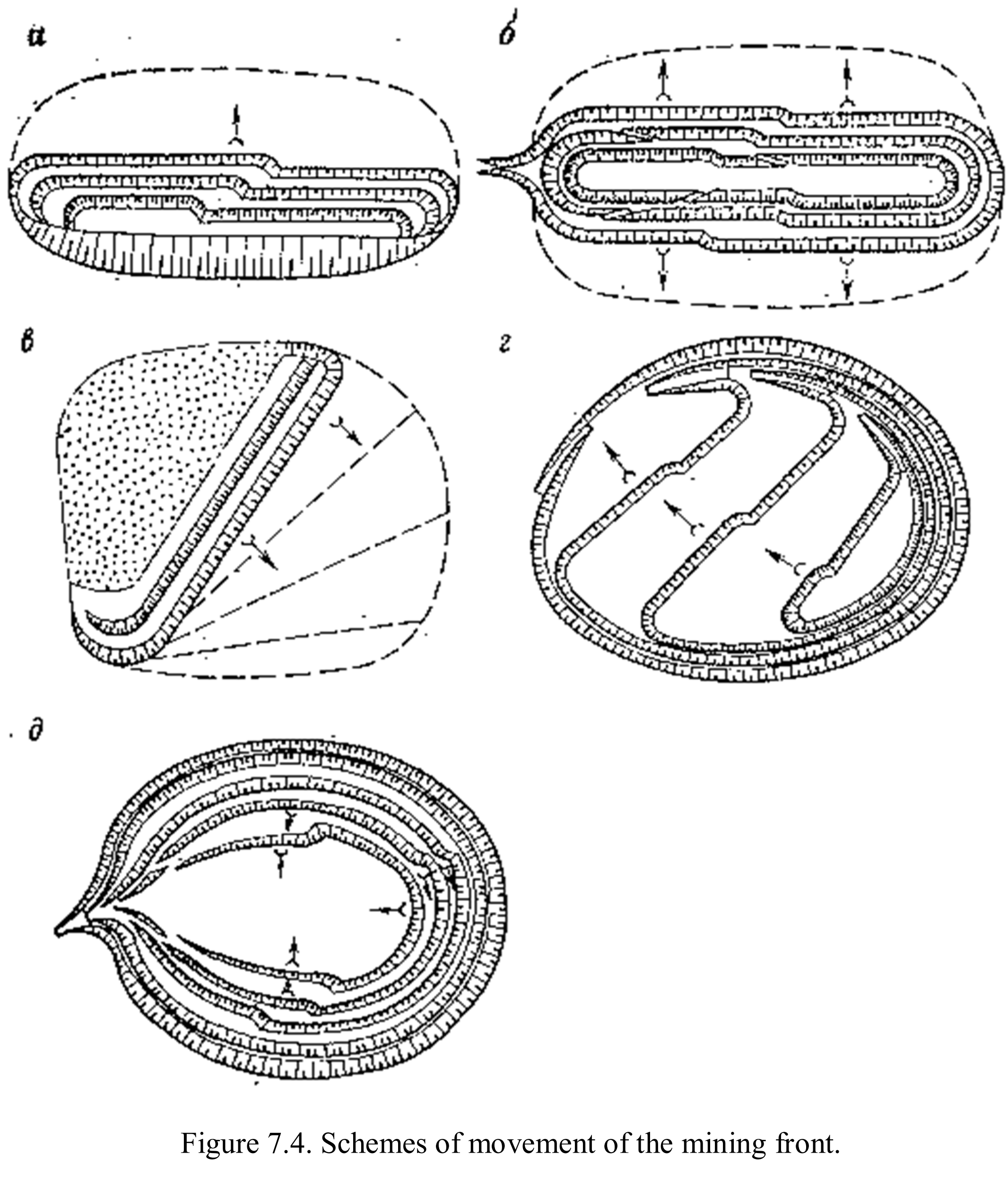


Figure 7.4. Schemes of movement of the mining front.

2. Parallel to one of the axes of the field from the intermediate position between the boundaries of the excavation layer to its contours (double-sided recess) (Fig. 7.4, b). In this case, the opposite (or all along the perimeter) slopes of the ledge are permanently or periodically active (working), the speed of movement of individual sections of the work front of the ledge decreases.

This option is used for the development of oblique and steeply falling deposits elongated along the strike, especially with a large final depth of the quarry and a powerful thickness of the covering rocks.

3. Along a fan with a turning point located on or near the border of the quarry field (Fig. 7.4, c). In this case, the ledge has, as a rule, one working slope. When developing horizontal deposits, one turning point is arranged for all the ledges of the quarry; when developing steep deposits, a

separate turning point is created for each ledge (Fig. 7.4, d). The rate of movement of various points of the work front of the ledge during its development along the fan is variable (see Fig. 7.1, b and c).

This option of moving the work front of the ledges is possible when developing quarry fields of rounded shape in plan and small capacity of soft covering rocks, when horizontal layers and overburden rocks are developed using continuous-action equipment (often transport-dump bridges), as well as in cases of development of steep rod-shaped ore deposits with a spiral shape of the opening route.

4. Radially from the center of the excavation layer to its contours (the work front is located concentrically or sickle-shaped). This option can be used when working out horizons with wide approaches in specific conditions of occurrence (see Fig. 7.2, b and 7.1, h).

5. In a spiral, starting from the peripheral sections of the quarry field and ending in the center, which may occur during the development of horizontal and shallow layers (Fig. 14.4, e).

The length and speed of the advance of the work front

The length of the mining front of the quarry, which consists of the length of the fronts of individual ledges, should be sufficient to ensure the installed production capacity of the quarry for minerals and rock mass, as well as to prepare new horizons.

The initial front of the ledge may be equal to the length of the Lc or the width of the Vc quarry field, or more often it is less than the Lc (Vc). This front increases with the development of mining operations, so its length Lf.y is not constant — it is smaller at the beginning and end of the development period of this horizon. When a split trench is located in the middle of a quarry field and its bilateral development, the length of the work front of one ledge can reach 2Lf.y.

When using powerful dredging and loading equipment on a ledge, it is desirable to have a single dead-end or through front of work using a single excavator, the performance of which corresponds to the planned amount of work on this horizon. This improves the organization of work and the use of equipment.

With a low intensity of development, the number of excavators may be less than the number of working ledges. In this case, work on ledges or a group of ledges is carried out by one excavator. When using powerful excavators (weighing more than 500-600 tons), their frequent stages from ledge to ledge are undesirable according to technical conditions.

The annual operating capacity of the excavator (m3/year) should be equal to the planned amount of work on the ledge

$$Q_{э.г}=H_y L_{ф.у} v_ф,$$

when $L_{ф.у}$ и $v_ф$ — averaged, respectively, the length of the ledge front (m) and the speed of its movement, m / year.

Thus, for the specific length of the ledge front and its speed of movement, only one excavator model can be selected that provides the best technical and economic development results.

Only with a large length of the work front (2-3 km or more) it is advisable to use several excavators on the ledge. The need for this arises with a high intensity of mining operations, a significant height of the ledge and in the absence (or impossibility of using more powerful excavators under transport conditions). In such cases, the single-panel work front of the ledge is divided into blocks.

The length of the panel blocks is set so as to ensure the continuity and mutual independence of work in the faces of adjacent blocks. If the rock mass is heterogeneous, it is necessary to allocate blocks, respectively, by grades and types of rocks and minerals. In such cases, the length of individual panel blocks may be different. With a small length of adjacent blocks, they are developed by one excavator sequentially.

The independence of the development of panel blocks represented by rocks is ensured with sufficient volumes, and therefore the length of the working blocks — blown up, prepared for blasting (drilled) and drilled. The faces of adjacent panel blocks must have the same direction of movement and are significantly removed from each other.

The intensity of development is characterized by the rate of movement of excavator faces. The rate of movement of the end faces (m /day) with the width of the entry A (m) and the daily productivity of the excavator Qe.s. (m3/day) is

$$v_з=Q_{э.с}/(AH_у).$$

Time (day) of working out the panel block length $L_б$ (м)

$$t_б=L_б/v_з.$$

With several panel blocks on a ledge, the time of their working out in equal conditions can be assumed to be the same. The movement of the faces usually ranges from several meters to several tens of meters per day.

The rate of advance of the work front per unit of time (usually per year) depends on the capacity of the quarry and a number of other factors. In modern quarries, it varies from 30 to 250 m/year, and in some cases reaches 400-600 m/ year; its usual value is 80-120 m/ year. A high rate of movement of the mining front is achieved when developing horizontal layers of low power with the movement of minerals by road or conveyor transport, and overburden rocks — into the developed space by overburden excavators or transport dump units.

In railway transport, no more than three excavator blocks are allowed within a single work front due to difficulties with transport and exchange operations, and in motor transport - up to six blocks. In conveyor transport, the number of blocks is limited, as a rule, by the capacity of excavators and conveyors used.

In long-range quarries, if it is necessary to intensify the mining of the upper horizons by excavators of relatively small capacity working in conjunction with rail transport, a double front of work is used, which allows up to four or five excavators to be installed on the ledge. When using vehicles, the device in such conditions of several transport exits from the ledge allows you to reduce the distance of transportation in the quarry, as well as on the surface.

The minimum length of the panel block is usually set based on the conditions of transport and drilling and blasting operations. Thus, during rail transportation, the length of the block and the distance between adjacent faces should be at least 2.5—3 train lengths to ensure the independence of the supply and loading of trains in each face. The volume of the exploding block currently usually amounts to at least two weeks (and often a month) of the excavator's productivity in the development of overburden rocks. Usually, the minimum length of blocks in railway transport is 300-500 m when mining rocks and 200-400 m when excavating soft rocks.

When using road transport, the minimum length of the panel block is reduced to 80-150 m according to the conditions of drilling and blasting operations and traffic safety. The smallest length of the panel block is thus an almost constant value for a certain type of transport.

Each excavator must perform the planned annual volume of work Qe.g. At a given speed of movement vf and the height of the ledge, Well, a certain front of work is required for this.

Reference words: ledge, panel along the work front, entry, working front, panel block, working blocks, excavation approaches, longitudinal, transverse, diagonal, work front within the boundaries of the layer, parallel, fan, radial, length, speed, initial front, large extent, minimum length.

Security questions:

1. What is the characteristic of the ledge panel?
2. What is called dredging?
3. How can the work front move within the boundaries of the layer?
4. What should be equal to the initial front of the ledge?
5. Based on what is the minimum length of the panel block set?

Lecture 11

Subject: The working area of the quarry. Prepared, opened and ready to be excavated stocks**.**

Plan:

1. 1. Working area of the quarry.
2. 2. Prepared stocks.
3. 3. Opened stocks.
4. 4. Stocks ready for extraction.

Both during the construction period and during the operation of the quarry, several ledges are being developed simultaneously.

Each of them has a working and non-working front, i.e. that part of the ledge within which rock excavation is not carried out for a long time (at least a year).

Each lower ledge is separated from the overlying one by safety and transport berms. Such berms are mandatory both in the working and non-working part of the ledge front. Between the working fronts of the ledges, work platforms must be left, the width and length of which are set by the project.

The zone in which the main technological processes of open-pit mining are carried out is called the working area of the quarry. Examples of working areas are shown in Figure 8.1.

The working area can cover one, two or all sides of the quarry. It is a surface that moves and changes in size and shape, having a diverse spatial configuration and a different position in the space of the quarry field over time.

During the construction period, the working area of the quarry includes only overburden ledges, and by the end of mining and capital works - and mining. When operating in the working area of the quarry, zones of stripping, mining and mining-preparatory (rifling) works are allocated (see Figure 8.1).

The number of stripping, mining and mining-preparatory blocks of panels and faces is not set arbitrarily, since it depends on the implementation of plans for certain types of work and the regularity of field development according to the project and schedule of the mining mode.

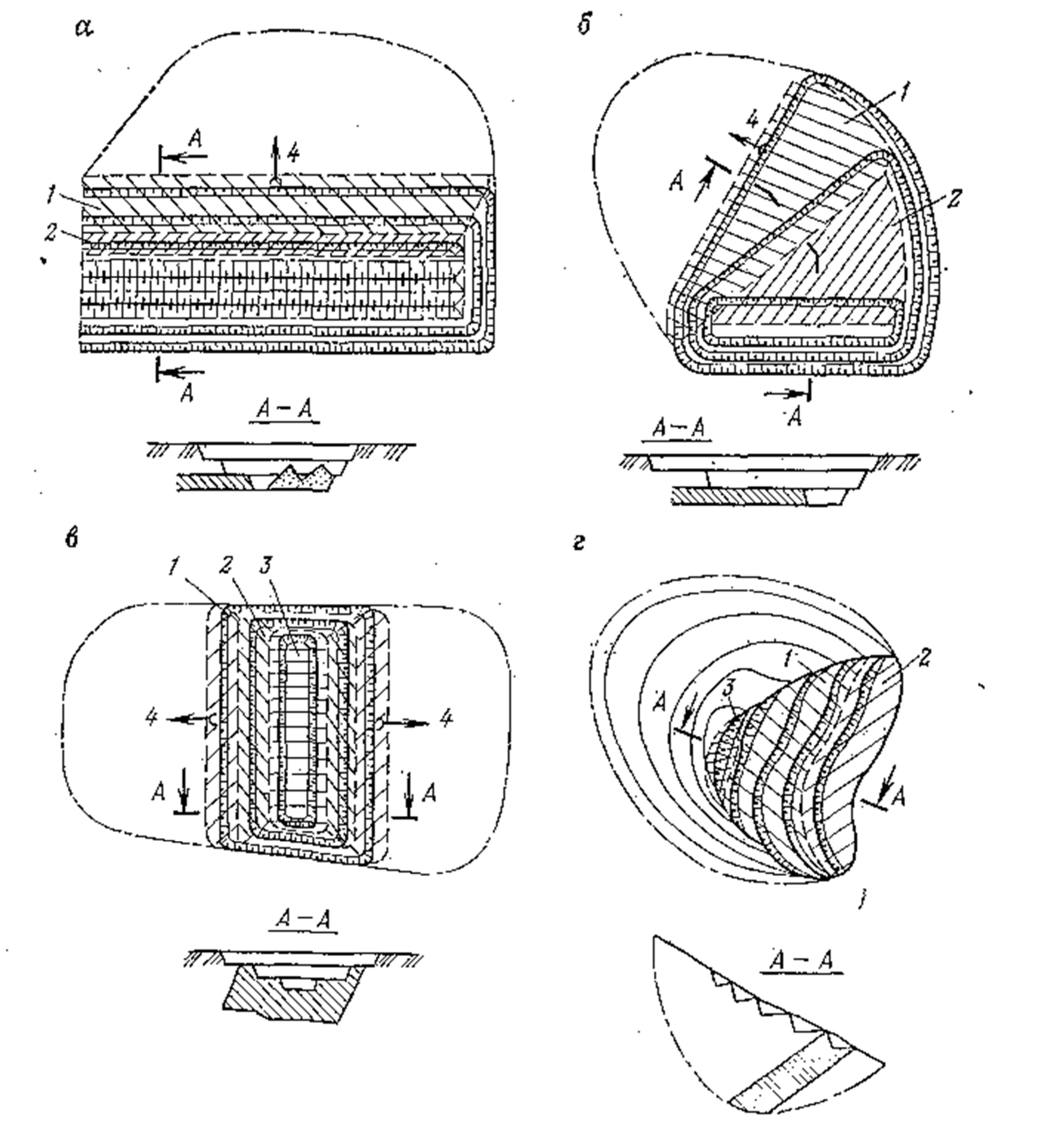


Figure 8.1. Schemes of working areas of the quarry:

a and b — in the development of horizontal deposits; c and d — in the development of steep deposits; 1 — overburden working zone; 2 — mining working zone; 3 — zone of mining and preparatory work; 4 — the direction of the movement of the work front.

Within the working area of the quarry, for each working excavator, there is a certain horizontal area S b, characterized by the average width of the working area Sh.p. and the length of the panel block Lb allocated to the excavator along the work front. Usually the area of the Sb ranges from 15 to 40 thousand m2 when using rail transport and from 3 to 15 thousand m2 for other types of transport. In each specific case, with the known technology of work, the minimum value of Sb can be calculated more accurately.

The number of panel blocks placed in the work area of a given size,

$N_б=S_{р.з}k_o f k_и/S_б$,

где $S_{р.з.}$ — the area of the horizontal projection of the working area, m2; co — coefficient, taking into account the presence of slopes of ledges on the area of the Sp.z. (under normal conditions, co = 0.85 □ 0.93); f — coefficient, taking into account the presence of reserve (non-working) blocks (f = 0,75 ÷ 0,8); $k_и$ — the coefficient of use of the working area area, which determines the correspondence of the working edge of the ledge to the length of the block $L_б$ ($k_и$=0,7 ÷ 0,9).

Changes in the areas of horizontal and vertical projections of the working area as a whole, overburden rocks and minerals as mining operations develop can be depicted on the graphs of the mining regime.

On each ledge of the working area , there are:

intensive mining operations, when each excavation machine on the ledge has a relatively short front of work and the annual advance of the front is approximately equal to or greater than the length of the front of the machine;

non-intensive mining operations, when the annual advance is significantly less than the length of the front of the excavation machine.

In different periods of development of each ledge, the degree of intensity of mining operations varies significantly. It is the greatest during the opening of the horizon and the carrying out of split trenches, and then decreases due to an increase in the length of the work front. When the work fades, the number of active faces and the length of the work front are gradually reduced.

The maintenance of a larger than necessary number of active faces on the ledge is associated with an irrationally large size of the working area of the quarry, an increase in transport and energy communications and costs. In principle, the smaller the size of the working area, the more concentrated and intensive mining operations are carried out, the more economical the development is. However, at these sizes, the regularity of mining operations and the reconstruction of the development system should not be disrupted.

The size of the working area depends on the development period, the type of deposit being developed, the angles of the slopes of the working sides changing with the depth, the angles of the slopes of the sides at the time of repayment of open works, the size of the quarry field and the accepted direction of mining development (Fig. 8.2). In general, the height of the working area of the quarry is equal to the sum of the heights of the ledges being developed. During the construction and development of the design capacity of the quarry, the working area continuously increases in plan and height during the development of deposits of any type. Further, during the period when the design capacity of the quarry is reached, the size of the working area reaches its maximum values, if the intensity of mining operations does not decrease.

In subsequent periods, during the development of horizontal and shallow deposits, the working area, having full development in height and in plan, shifts in a given direction, its dimensions change only partially as a result of changing the configuration of the quarry field and creating additional advanced ledges in areas of increased surface relief. At the same time, there is no need for mining and preparatory work. Working areas in the development of horizontal and shallow deposits are usually continuous in both overburden and mining operations and are relatively stable in size; therefore, they are called solid zones.

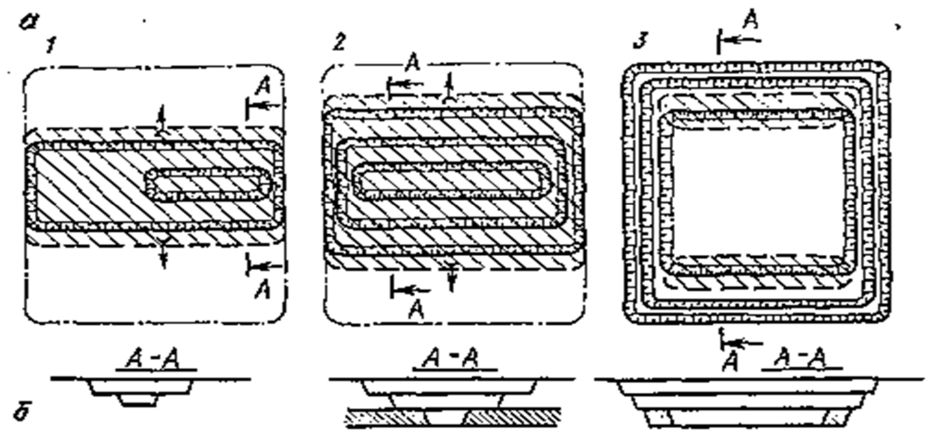


Figure 8.2. Dynamics of the working area of the quarry:

a and b — when developing horizontal and steep deposits, respectively; 1, 2, 3, 4 and 5 — stages of changing the position and size of the working area.

When developing inclined and steep deposits, the working area increases in plan and height due to the separation of the sides and the opening of new horizons until the upper ledges reach the intermediate (stage) or final boundaries of the quarry field on the surface. At the same time, the annual volume of work on the rock mass is increasing.

After reaching the final contours, mining operations on the upper ledges stop and the working area shifts vertically. At the same time, its dimensions are usually reduced in plan and the annual volume of work on the rock mass is reduced.

Working zones in the development of inclined and steep deposits are called deepening working zones. The characteristic features of such working areas are the variable number of working ledges and their sizes.

Prepared, opened and ready-to-be-dredged stocks

The ledge to be developed must be prepared for the implementation of the main (technological) processes: preparation of rocks for excavation, excavation, loading and transport operations. For example, prior to the development of the upper ledge of the quarry, surface preparation, drainage work and drainage of the array are carried out at least within the first working panels. For the lower ledges, the preparation consists primarily in the excavation of the rocks of the upper ledges in compliance with the design dimensions of working and non-working sites, as well as in the dismantling of equipment, various communications, in the cleaning of rock piles from the upper platform (if they are left for any reason), the elimination of overhangs dangerous to maintenance personnel and equipment, etc. D.

The prepared reserves of the rock mass of the ledge are understood as those volumes that can be involved in the initial technological processes (drilling, blasting, mechanical loosening, etc.) preceding the excavation and loading operations, or at least in one of the initial processes.

To perform the main processes — excavation, loading and movement of rocks from the faces to the points of reception of goods — it is necessary to open the ledge, i.e. to carry out the opening work, lay transport communications, create an initial face for the excavation of rocks. Part of the prepared reserves of rock mass, to which transport access is provided, necessary for the excavation and movement of rocks, is called the opened reserves of rock mass of the ledge.

The implementation of a full range of technological processes is possible only within the discovered reserves of rock mass. Usually the volume of opened stocks is less than the prepared stocks, in some cases they may be equal.

Part of the uncovered reserves are ready-to-excavate reserves of the rock mass of the ledge. These include stocks that are ready to be excavated, loaded and moved directly from the massif (soft and often dense rocks) or after blasting, mechanical loosening, etc. (rock, semi-rock and sometimes dense rocks).

In particular cases, for example, when excavating soft rocks without prior preparation, the stocks opened and ready for excavation are the same. Figure 8.3 shows examples of the location of prepared, opened and ready-to-excavate reserves of rock mass of the ledge.

Naturally, reserves of minerals of various grades and types, respectively prepared, opened and ready for excavation, and, if necessary, reserves of substandard and off-balance ores and minerals extracted along the way, are allocated from the reserves of the rock mass.

After determining the reserves within each ledge, the total prepared, opened and ready for excavation reserves of rock mass and minerals for the quarry as a whole are found by summation.

As the work front of the overlying ledges moves forward and the position of transport communications changes, the volumes of the rock mass of the underlying ledges move into prepared reserves, and then sequentially or simultaneously into opened and ready for excavation (see Figure 8.3).

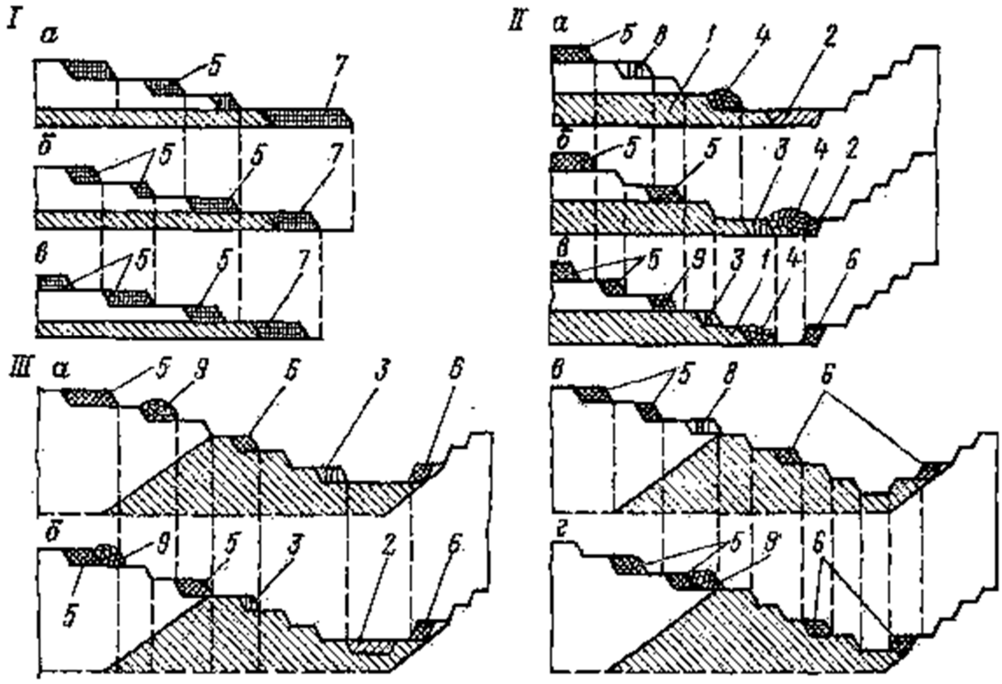


Figure 8.3. Schematic representation of rock mass reserves (current) during the development of horizontal deposits with soft (I) and rocky (II) rocks and during the development of steep deposits (III):

a, b, c, d — sequential change of reserves; 1 — mineral; 2 — prepared mineral reserves; 3 and 4 — respectively, the drilled and exploded volume of the prepared mineral; 5 and 6 — the uncovered volume of rock and uncovered mineral reserves, respectively; 7 — mineral reserves ready for excavation; 8 and 9 are respectively the burnt and blown volumes of the rock.

It is necessary to distinguish between reserves of rock mass and minerals at a fixed point in time — current and for a certain period of operation of the quarry (most often annual) — planned.

The prepared and uncovered reserves of rock mass on the ledges mentioned above belong to the current ones. These reserves should be sufficient to perform all preparatory work, and this provision determines the volumes of the corresponding working blocks: prepared for drilling, drilling, exploding, etc.

Reserves ready for extraction should be sufficient for the rhythmic operation of the excavation and loading equipment, and in the mining zone - also to ensure the necessary range of minerals and current quality targets. Currently, at powerful quarries, ready-to-excavate reserves of rock mass per excavator are, as a rule, not less than its monthly productivity. Prepared and opened stocks at any time should provide ready-to-be-dredged stocks.

Planned prepared and uncovered reserves of rock mass and minerals are necessary to ensure the fulfillment of the design volumes of stripping, mining and mining preparatory work under the accepted procedure for their production. The volume and location of these reserves are established during the annual planning of mining operations, taking into account the possibility of temporary cessation of work on individual or all overburden ledges, changes in the quality of minerals in the subsurface, etc. Planned reserves of rock mass are determined by the movement of all lower ledges relative to any upper, stopped (Fig. 8.4).

By the time the quarry is put into operation, the prepared mineral reserves for year—round operation should be sufficient to fulfill at least a three-month production plan for the first year of operation, and for seasonal work - at least a six- or seven-month plan.

The prepared and uncovered reserves are calculated by the method of horizontal sections on a mining plan made on a scale of at least 1: 1000. To do this, on each horizon between the lower and upper brow of adjacent ledges, the areas of prepared, uncovered and ready for excavation reserves, the corresponding average height of the ledge and the density of overburden rocks and minerals are determined. The results of calculations are determined in volumes and tons of reserves of rock mass, ore (and tons of metals) by their types.

In the practice of open-pit mining of coal, various ores and building rocks, the concepts of uncovered, prepared and ready-to-excavate reserves are widely used in relation only to minerals. Even in one department, it is not yet possible to strictly link the definition of reserves and their graphical interpretation in relation to the entire variety of conditions for the occurrence of minerals, especially with intermittency within the ledge of overburden rocks, off-balance sheet and balance sheet minerals.

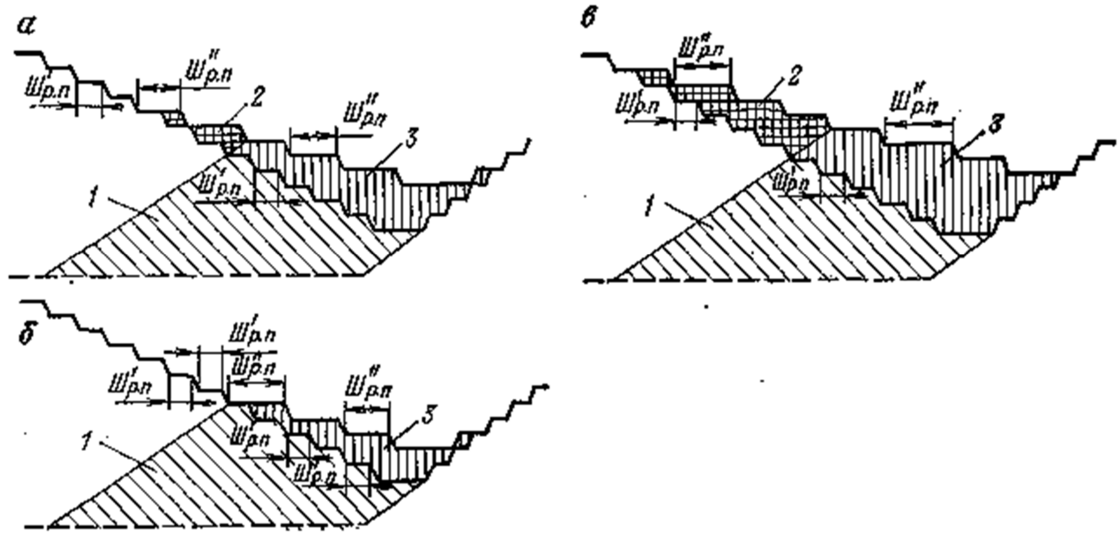


Figure 8.4. Schemes for determining the planned reserves of rock mass:

a, b and c, respectively, when stopping the intermediate, all and upper overburden ledges; 1 — minerals; 2 and 3 — respectively, the volumes of rocks and mineral reserves; Sh'r.p and Sh'r.p —

respectively, the minimum and actual width work sites.

Reference words: working and non-working front, ledge, safety and transport berm, front, working area, quarry side, overburden ledges, mining ledges, number of panel blocks, intensive and non-intensive mining operations, continuous zones, volume, prepared reserves, opened, ready for excavation, current, planned.

Control questions:

1. What is the working area of the quarry?
2. What mining operations are distinguished on each ledge of the working area?
3. What stocks are called prepared?
4. What reserves are called uncovered?
5. What do you mean by ready-to-excavate reserves of rock mass?

Lecture 12

Topic: Classification of open-pit mining systems.

Plan:

1. **1. General concepts.**
2. **2. Classification of development systems according to V.V.Rzhevsky.**
3. **3. Classification of development systems by E.F.Sheshko.**
4. **4. Classification of development systems according to N.V. Melnikov.**

The open-pit mining system refers to the order and sequence of performing open-pit mining operations within a quarry field or its site. The system must ensure safe, systematic and economical integrated development of all minerals, the required production capacity of the enterprise, full extraction of reserves, protection of the subsoil and the environment.

In general, mining operations include mining, stripping and mining preparatory work.

When extracting low-value minerals that come directly to the surface, stripping operations may be absent or not of significant importance. In such cases, the development system is the order and sequence of mining operations and work on the opening and preparation of horizons within the quarry field.

Sometimes, when developing horizontal deposits after the completion of mining and construction work, there is no need to open new horizons; in such cases, the development system is characterized only by the order and sequence of overburden and mining operations.

Naturally, the development system is connected with the equipment complexes used at the quarry. If the development system determines the order and sequence of mining operations, then the equipment complexes determine the types, capacity and arrangement of equipment that ensures the production of mining operations in the prescribed volume and order. Below, their classifications are considered separately, since development systems, regardless of the means of mechanization of mining operations, have their own sequence and regularity of overburden, mining and mining preparatory work.

According to the degree of mutual dependence of stripping, mining and mining preparatory works, development systems are distinguished:

dependent (rigidly dependent), in which there is a strict dependence between stripping, mining and mining preparation works in relation to the sequence of their execution in time and space; at the same time, planned uncovered mineral reserves are very limited (usually for a period of no more than 15-45 days) and the procedure for conducting mining operations is strictly regulated by the calendar plan;

semi-dependent, in which stripping, mining and mining preparatory work is carried out without strict mutual alignment in time; planned uncovered reserves can be significant (for a period of up to 3-6 months); the procedure for conducting work is regulated by an annual calendar plan that provides for significant time reserves between these types of work, which allows them to be performed with varying intensity; independent, in which stripping, mining and mining preparatory work is carried out almost independently of each other; at the same time, the uncovered mineral reserves are almost not limited to the organization of work and the time reserves in their conduct are very significant.

When developing horizontal or shallow deposits, at the end of mining and preparatory work, the primary front of overburden and mining operations of the quarry is created; the resumption of mining and preparatory work is possible during the reconstruction of the quarry. Thus, systems for the development of horizontal and shallow deposits during operation are characterized only by the order and sequence of overburden and mining operations and changes in the length of the work front or the height of individual ledges and the size of work sites. Such development systems are called continuous.

During the development of inclined and steep deposits, mining and preparatory work is carried out both during construction and during the operation of the quarry to create a front for mining and stripping operations. The composition of mining preparatory work during the operational period includes the opening and cutting of new working horizons. Thus, the development systems for inclined and steep deposits are characterized by the order of overburden, mining and regular mining preparatory work. Such systems can be called deepening.

When developing upland deposits, systems of the first group are used. With steep slopes and steep inclined fall of deposits, systems of the second group are used. When developing deposits that are difficult in terms of topographic and mining-geological conditions within the same quarry field, systems from both groups can be used simultaneously.

According to the direction of the movement of the mining front in the plan, there are development systems:

longitudinal, in which the single-side or double-side front of overburden and mining operations moves parallel to the long axis of the quarry field;

transverse, in which the single-side or double-side front of stripping and mining operations moves parallel to the short axis of the quarry field;

fan-shaped, in which the front of stripping and mining operations moves along a fan with a central (common) or dispersed (two or more) turning points;

annular, in which the working area covers all sides along the perimeter of the quarry and the development is carried out in annular strips from the center to the boundaries of the quarry field or from the boundaries to the center.

For all variants of development systems, the location of dumps (external, internal or mixed dumps) determining the direction of movement of overburden rocks is of primary importance.

The classification of mining systems in accordance with these main features is given in Table 9.1, and the graphical one is shown in Figure 9.1. This classification, which is based on mining—geological and geometric prerequisites, characterizes the essence of open-pit mining technology and facilitates the subsequent calculation of development systems. The justification of the development systems provides for the establishment of quantitative dependencies between the main dimensions of the deposit, the quarry field, the parameters of the elements of the development system, the parameters and arrangement of equipment and the production capacity of the quarry for mining, stripping and mining preparatory work.

The choice of development systems is based on the following provisions:

1. Establishment for specific conditions of the maximum possible production capacity of a quarry for mineral resources according to natural and technical conditions. The maximum capacity of the quarry depends on the nature of the complex mechanization of mining operations, which is the basis for the calculations of the development system. Such calculations are made during the design of new and reconstruction of existing quarries.

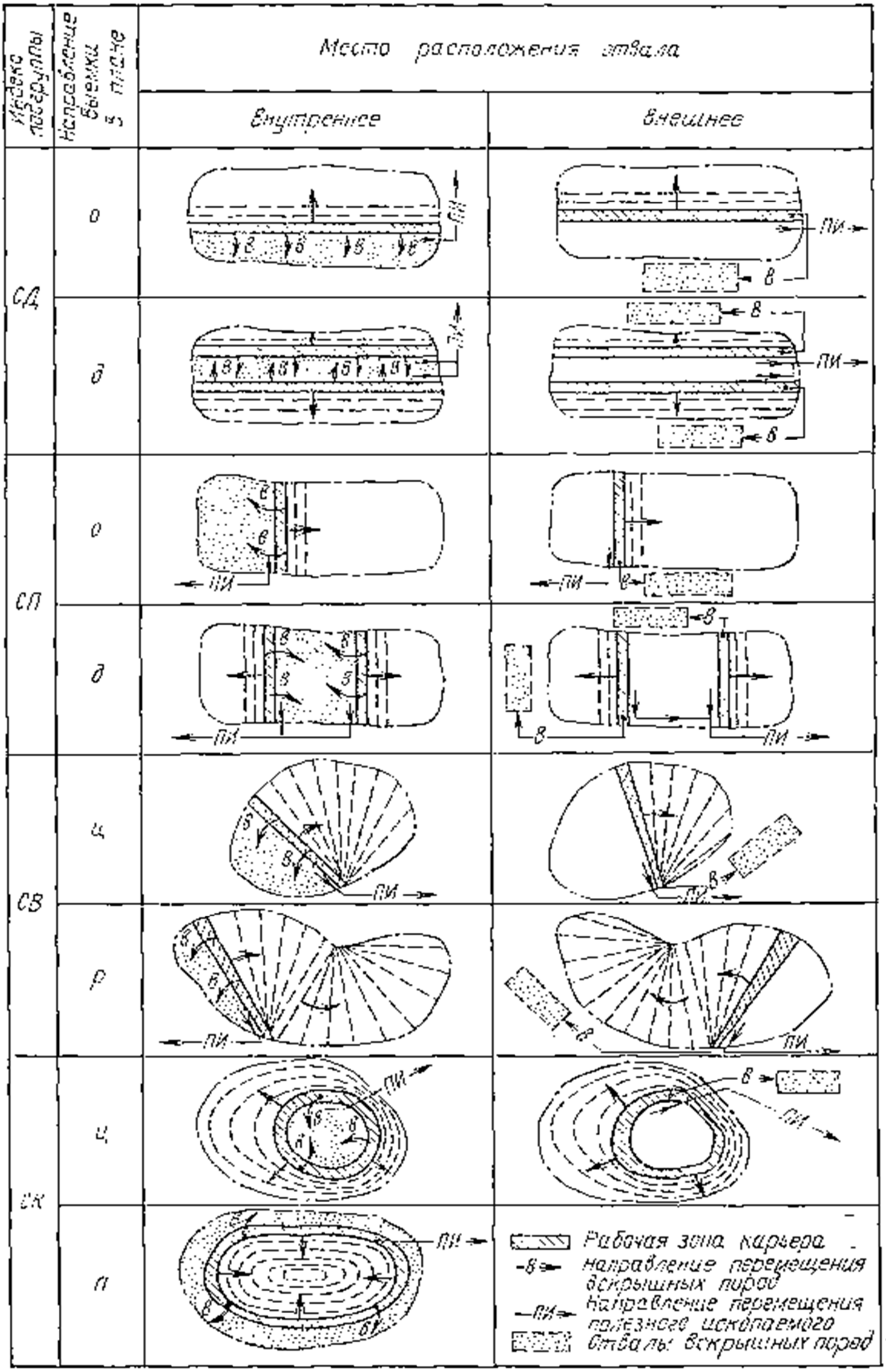
1. 2. Ensuring the specified planned production capacity of the existing quarry for minerals. When calculating, the possible equipment complexes are also set. The tasks of this direction are solved in the design of quarries and in the technical justification of mining and stripping plans at existing enterprises.

Таблица 16.1. Классификация систем открытой разработки месторождений.

Индекс группы	Группа систем	Индекс подгруппы	Подгруппа	Индекс системы	Система разработки
С	Сплошные	СД	Сплошные продольные	СДО СДД	Сплошная продольная однобортовая Сплошная продольная двухбортовая
		СП	Сплошные поперечные	СПО СПД	Сплошная поперечная однобортовая Сплошная поперечная двухбортовая
		СВ	Сплошные веерные	СВЦ СВР	Сплошная веерная центральная Сплошная веерная рассредоточенная
		СК	Сплошные кольцевые	СКЦ СКП	Сплошная кольцевая центральная Сплошная кольцевая периферийная
У	Углубочные	УД	Углубочные продольные	УДО УДД	Углубочная продольная однобортовая Углубочная продольная двухбортовая
		УП	Углубочные поперечные	УПО УПД	Углубочная поперечная однобортовая Углубочная поперечная двухбортовая
		УВ УК	Углубочные веерные Углубочные кольцевые	УВР УКЦ	Углубочная веерная рассредоточенная Углубочная кольцевая центральная
УС	Смешанные (углубочно-сплошные)	—	То же, в различных сочетаниях		

Примечание. К наименованию системы добавляется: «с внешними или внутренними отвалами».

а


Индекс подгруппы
Направление выемки в плане
Место расположения отвала
Внутреннее
Внешнее
СД
о
д
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о
д
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Рабочая зона карьера
направление перемещения вскрышных пород
Направление перемещения полезного ископаемого
Отвалы вскрышных пород

б

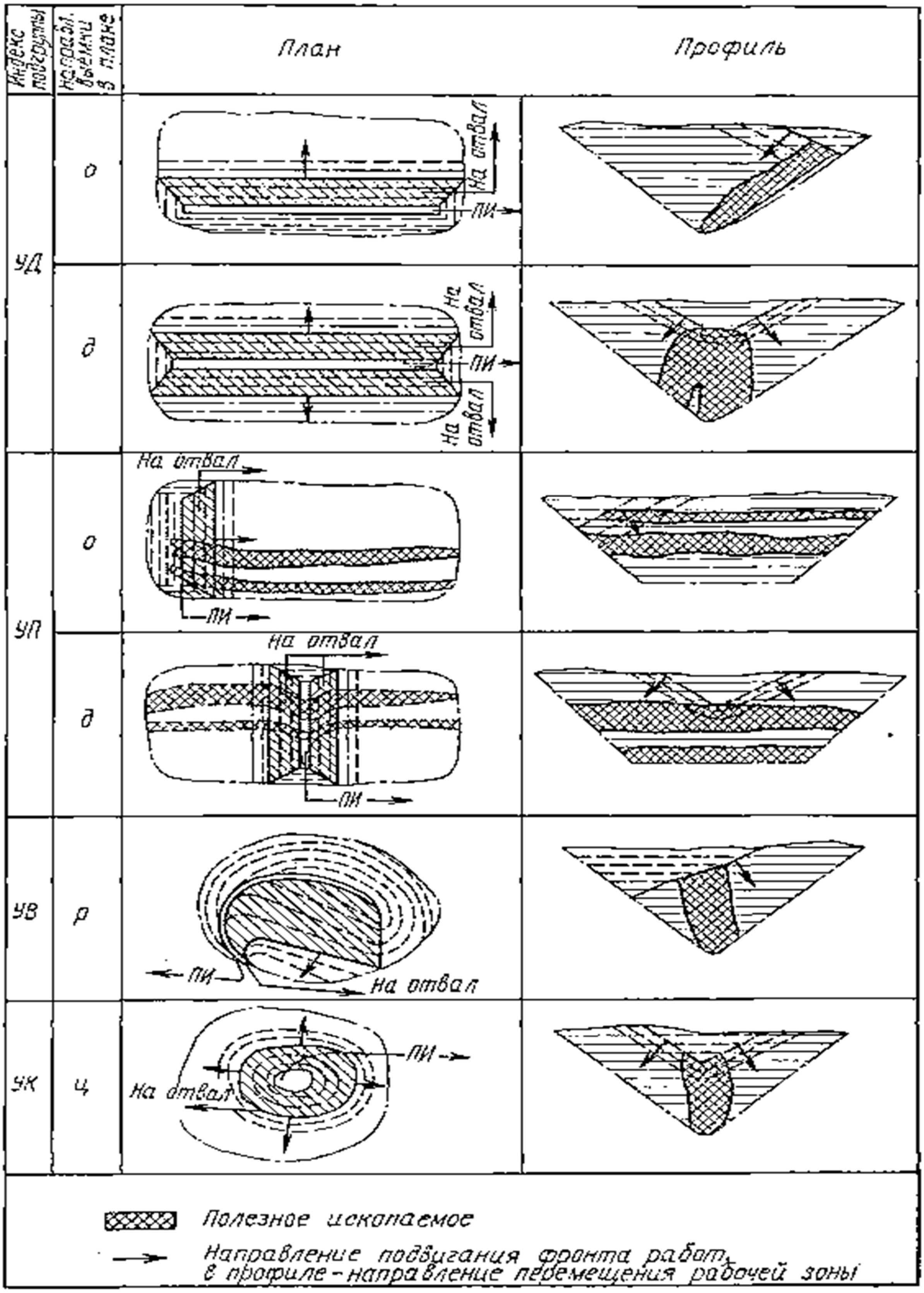


Fig. 9.1. Systems of open-pit mining of mineral deposits:

a – solid; b – recess; o, d, c, p and p – the direction of the recess in the plan is, respectively, single-sided, double-sided, central, peripheral and dispersed.

3. Reducing the costs of stripping and mining operations to a minimum (with known complexes of equipment and production capacity for minerals). Such a calculation can be made for

several power options and equipment complexes.

The intensification and concentration of mining operations contributes to the fullest use of mining and transport equipment. Therefore, the realization of the maximum possible mining production capacity of the quarry is associated in most cases with the achievement of optimal technical and economic results of development.

The initial data for the justification and research of the development system are the initial information about the deposit and the quarry field, about the mining regime used, methods of opening and possible equipment complexes for use.

Classification of development systems according to the direction of movement and the method of overburden production

In 1947, Prof. E. F. Sheshko proposed a classification of mining systems in the direction of movement of overburden rocks into dumps. According to this feature, the following are distinguished (Fig. 9.2):

A. Systems with transverse movement of rock into dumps without the use of vehicles; these development systems can also be called transportless.

B. Systems with longitudinal (frontal) movement of rock into dumps using vehicles; these systems can also be called transport.

B. Combined systems with transverse and longitudinal

by moving the rock into the dumps; these development systems have at the same time signs of transportless and transport systems.

Further, the division of these groups (A, B, C) into independent development systems is based on production methods and the degree of difficulty in performing transport and dump work.

Group A is divided into systems A-1, A-2 and A-3 according to the method of production of transport and dump operations. The L-0 system is singled out separately with a small amount of stripping work. Group B is divided into systems B-4, B-5 and B-6 according to the relative complexity of rock transportation.

Group B includes two development systems — one of each of the transportless and transport groups. This group is divided into B-7 and B-8 systems based on the relative predominance of transportless or transport movement of overburden rocks.

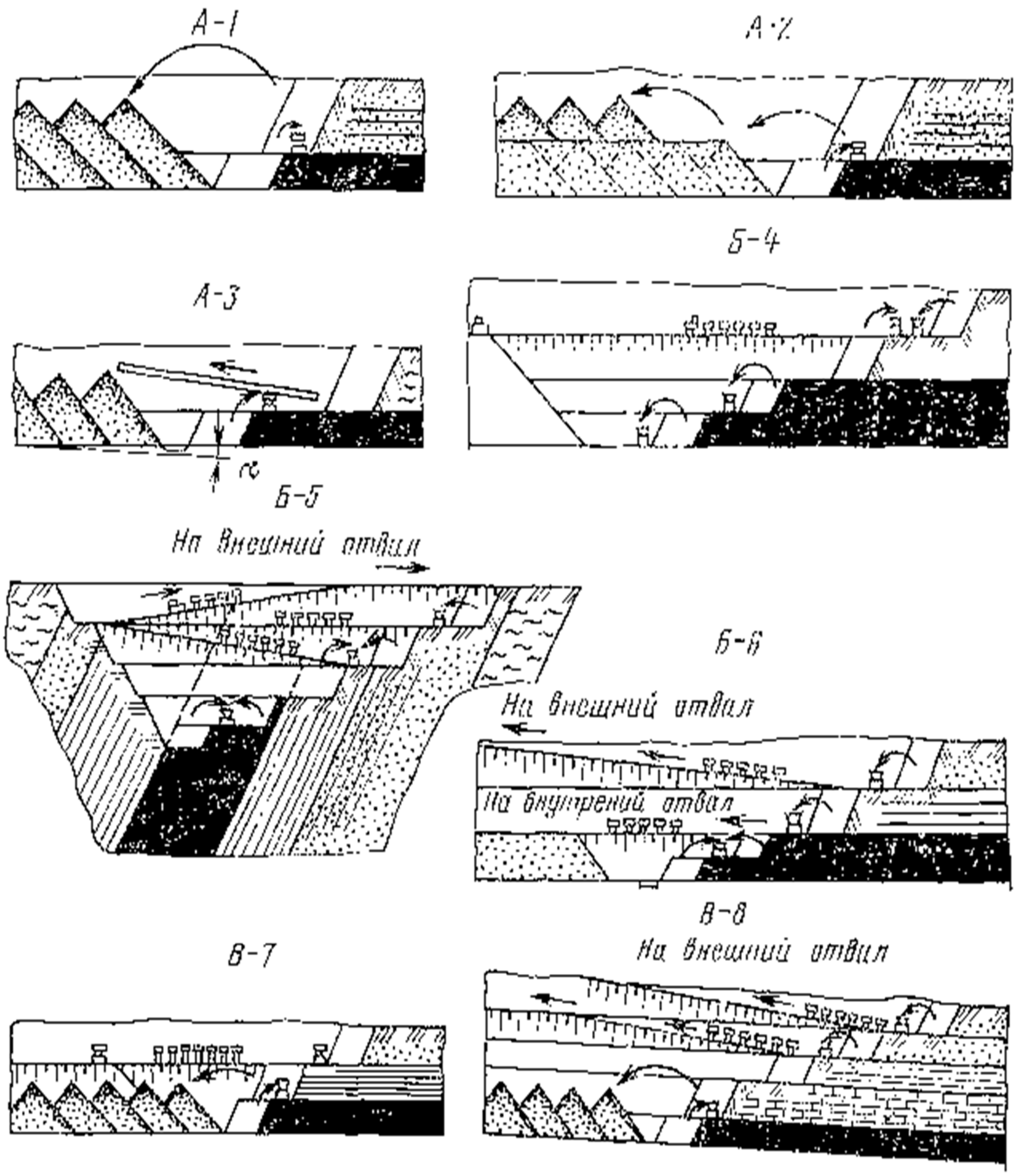


Fig. 9.2. Systems of open-pit mining (according to E. F. Sheshko)

In 1952, Academician N. V. Melnikov proposed a classification of development systems according to the method of overburden production. According to this classification, all development systems are divided into five groups.

With a transportless mining system, the movement of rock from the face to the internal dump is carried out by overburden excavators (mechlopats or draglines).

The transport dump group includes development systems in which overburden rocks are moved to internal dumps by means of transport dump bridges and cantilever dumpers.

The special group includes development systems in which overburden rocks are removed by tower excavators, wheel scrapers, hydro-mechanized method or cable cranes. These can also include the development of overburden rocks by bulldozers, rope scrapers and other special equipment.

Transport systems include development systems in which overburden rocks are moved to dumps by means of transport. These systems are more complex and less economical compared to transportless ones, but they can be used under any conditions of occurrence of the deposit and therefore are the most common.

Combined systems can be used in the development of horizontal and shallow deposits with a thick layer of covering rocks.

Reference words: order, sequence, dependent, semi-dependent, independent, continuous, deepening, longitudinal, transverse, fan, ring, positions, direction of movement of overburden rocks into dumps, method of overburden production.

Security questions:

1. What do you mean by a development system?

2. Which systems are called solid?

3. Which development systems are called in-depth?

4. Describe the classification of development systems in the direction of movement of overburden rocks into dumps.

5. Describe the classification of development systems according to the method of overburden production.

Lecture 13

Topic: General information about the complex mechanization of open-pit mining. Principles of complex mechanization

Plan:

1. General information.
2. About the development of rocks.
3. In-line technology.
4. Basic requirements for equipment complexes.

The technological essence of the main processes consists in changing the aggregate composition and spatial position of the rock. The breed is involved in separate processes sequentially or almost simultaneously. When performing several technological processes with the same equipment, an elementary volume of rock (for example, 1 m3) participates in them sequentially (for example, when developing rock with a scraper) or simultaneously when separate processes are physically combined, for example, loading, moving and storing rock with an overburden excavator into the developed space.

Drilling, blasting and excavation processes can be performed with a significant advance in space and time relative to each other. Excavation and loading operations and transportation are much more tightly linked. The interconnection of all technological processes and their degree of dependence is determined by a common goal — to create elementary, intermediate and quarry cargo flows of a certain capacity.

For uninterrupted elementary cargo flow, it is necessary to have the initial and final warehouses of these goods. Therefore, in reality, the concept of elementary cargo flow is broader than previously given, and includes the flow of rock mass itself, its reserves that are at various stages of involvement in technological processes, as well as a certain capacity of the dump or other unloading point.

Cargo flows are created by a chain of interconnected machines and mechanisms that consistently carry out a full cycle of basic (technological) and auxiliary processes. Each such chain of machines and mechanisms is a complex of mining and transport equipment.

With the complex mechanization of mining operations, heavy manual labor is displaced not only from the main, but also auxiliary processes.

The completeness of mechanization is understood as qualitative (by types and models) and quantitative compliance of the means of mechanization included in the complex according to the main and auxiliary processes. The complete construction of mechanization is characterized by the total capacity of the equipment for related technological processes corresponding to the required capacity of the cargo flow.

Complex mechanization and automation of the main and auxiliary processes, the development of more advanced and productive mining and transport equipment together with the improvement of technology and organization of open—pit mining are the main factors of technical progress of mining production.

About the development of rocks

The essence of the technological processes of mining consists in overcoming the resistance of rocks to separation from the massif, their destruction and displacement.

The resistance of rocks for each subsequent process is explained by a number of factors.

1. When performing technological processes from the original natural state (in the array) the breed

goes into an artificially altered state: rock and semi—rock rocks, when excavated, pass into a group of destroyed rocks, and dense and soft rocks - into a loosened state.

2. The condition of the already artificially modified rock is not the same after each technological process. For example, different lumpiness and degree of loosening of the destroyed rock located in the collapse and in the dump hopper cause different resistance to development and different technical performance of the same type of excavators in quarry and dump faces. A different number of free surfaces and, accordingly, the degree of crushing is characterized by primary and secondary rock blasting, etc.

3. Each process is characterized by the application of heterogeneous external efforts to the breed. The resistance of a rock of the same composition and physical condition cannot be the same in every process.

4. The resistance of the rock when performing individual processes is not the same. In different volumes, the aggregate composition of the rock is unstable.

5. The resistance of this breed when performing the same process changes when using different types of mechanization tools and their different capacities. The characteristics of the rocks being developed should not be considered in isolation from the means of mechanization of processes. For example, the indicators of rock resistance to spherical and pneumatic impact drilling, excavation by mechanical shovels and rotary excavators, etc. are not the same.

The mining and technological characteristics of rocks when performing processes can be controlled by choosing in a certain way the means of mechanization of processes and technological parameters of workplaces (faces, transport communications, etc.). It is important not just to separate from the array, destroy and move rocks, but also to ensure high productivity of mechanization tools in each process, their reliability in performance and durability of operation. For example, it is technically possible to move very large rock pieces in wagons, but this is abnormal, since it leads to an increase in the duration of loading and unloading wagons, causes their rapid wear.

With all the variety of rocks being developed, the conditions of their occurrence and the technique used, it is possible to establish a number of general provisions for an approximate relative assessment of the difficulty of developing rocks:

1. Rational mining technology is characterized by both the minimum number of technological processes and the minimum total energy and labor costs for their implementation. At the same time, it should be ensured: the restoration of normal conditions for performing the next cycle of mining operations, obtaining products of the required quality, the maximum possible preservation of the surrounding natural conditions.

2. The energy and labor costs attributed to 1 m3 of rock mass depend on: indicators of the difficulty of rock destruction; climatic, topographic and hydrogeological conditions of development; the scale of mining operations and the size of the quarry field; the technical means used to perform technological processes; the required size of the pieces of separated and moved rock; the organization of production of individual processes and mining works in general.

3. The condition and properties of rocks during the execution of individual technological processes can be controlled by selecting the composition and technology of previous processes (for example, the choice of the size of the primary destruction in conjunction with the choice of the type and parameters of the mining and transport machines used). Such a choice is mandatory, since it is as a result of it that the minimum total costs per unit of production are achieved. The increase in the degree of crushing and grinding of rocks achieved during the performance of previous technological processes is usually associated with a decrease in

the costs of performing subsequent technological processes.

At the same time, the management of rock properties in the construction of rational technology has technical, organizational and economic limitations. For example, an increase in the degree of crushing of rocks by explosion with an increase in explosive consumption has limits due to the parameters of collapse, the distribution of explosive energy for crushing and moving the rock, the size of the zone of expansion of rock pieces, etc.

4. The costs of moving the rocks being developed depend mainly on the parameters of the quarry and the location of receiving and reloading facilities on the surface and in the quarry. Since transportation costs account for 40 to 70% of the total costs of rock development, all previous and subsequent technological processes usually tend to be mechanized and performed in such a way as to create the most favorable conditions for the operation of quarry transport. Therefore, the mining and technological characteristics of the rock achieved as a result of the processes preceding the movement should ensure the effective operation of a particular type of transport.

5. Work experience shows that, depending on the type of rock being developed and the requirements for transport, the total costs of energy, labor and materials (in value terms) for performing all production processes preceding the relocation (without taking into account the scale of development), attributed to 1 m3 of rock, vary in the range of 1:25. The accuracy of calculating the economic indicators of mining technology usually also does not exceed 5%.

Taking into account the above provisions, as a physico-technical and generalized technological basis for comparing rocks in terms of resistance to the execution of processes (preceding displacement), it is proposed to take a relative indicator of the difficulty of developing a rock Pt.p. The Pt.p indicator characterizes the rock in its natural state (in the massif) and at the same time takes into account subsequent changes in the mining and technological characteristics of the rock after performing the processes of preparation for excavation, excavation and loading.

With the specified prerequisites $П_{т.р}$ approximately can be defined from the expression

$П_{т.р}=1/3(П_б+П_в+П_э)$,

where $П_б$, $П_в$ и $П_э$ — accordingly, the indicators of the difficulty of drilling, blasting and excavating rock.

Numerically indicators $П_б$, $П_в$ и $П_э$ they are characterized by rock categories for drillability, explosivity and excavability.

According to the relative difficulty of development, rocks are divided into 5 classes and 25 categories in accordance with the value of $П_{т.р}$:

I class — *easily processed* ($П_{т.р}$= 1÷5); categories of development difficulties 1, 2, 3, 4, 5;

II class — *average development difficulty* ($П_{т.р}$ = 6÷10); categories 6, 7, 8, 9, 10;

III class — *difficult to work out* ($П_{т.р}$= 11÷15); categories 11, 12, 13, 14, 15;

IV class — *very difficult to work out* ($П_{т.р}$=16÷20); categories 16, 17, 18, 19, 20;

V class — *extremely difficult to work out* ($П_{т.р}$=21÷25); categories 21, 22, 23, 24, 25.

Distinctive features of breeds of each class;

I class — возможность их разработки без предварительной подготовки к выемке;

II class — the possibility of development without blasting, but with mandatory preliminary preparation for excavation, for example, drainage, mechanical loosening, dynamic separation from the array, etc.;

III class — the need for pre—detonation with a relatively small explosive consumption (rocks of classes I-II in terms of explosiveness);

IV class — the need for blasting at high explosive consumption (III—IV classes of rocks by drillability and explosivity);

V class — exceptional drilling and blasting difficulty (rocks of class V for drillability and explosivity and extra-mountainous).

The same breed, with different requirements for preparing it for movement, can be characterized by different indicators of Pt.p. The Pt.p indicator is introduced for an approximate relative estimate of the costs of performing individual processes with enlarged economic and technological calculations and the initial choice of equipment complexes.

In-line technology.

The complex of the main mining and transport, auxiliary and crushing and sorting equipment should ensure systematic, in accordance with the capacity of the cargo flow, preparation of rocks for excavation, their excavation and loading, movement, warehousing and sometimes primary processing within each technological zone of the quarry in which the cargo flow is formed.

In mining technology, there may be no need to perform separate processes (mainly preparation of rocks for excavation, transport movement), and in the complex of equipment — appropriate special means of mechanization. When loading minerals into MPC wagons, there may, of course, be no means of mechanization of warehousing. In all cases, the complex of equipment includes machines and mechanisms that ensure the excavation and movement of rocks.

Complex mechanization of mining operations at quarries is developing on the basis of the development of in-line technology, as well as the maximum possible combination of individual operations when performing basic processes. In-line technology is achieved more easily when using continuous machines. However, it is possible to create a rhythmic flow with cyclic excavators, as well as with rail and road transport.

It should be remembered that most of the production processes of open-pit mining, including the work of chain and rotary excavators, are performed in repetitive cycles of shorter or longer duration.

It is generally believed that the technology using chain and rotary excavators and conveyors is continuous and it is called in-line. However, the full technological cycle of open-pit mining is always cyclical, since cargo flows are formed as a result of mining operations.

The main requirements for equipment complexes are as follows:

1. The complex of equipment should include only machines whose passport characteristics correspond to the mining and technological characteristics of rocks during each process (their drillability, explosivity, excavability, transportability).
2. The complex of equipment must comply with climatic and mining-geological conditions of development (occurrence,
3. the structure of the deposit, water availability, topographic conditions, etc.); mining and transport vehicles must equally ensure the technical feasibility of technological processes when mining and geological conditions of work change, the difficulty of developing rocks and the quality of minerals.
4. The complex of equipment must comply with the accepted systems of development and opening, the size and shape of the quarry, its capacity, the duration of construction and operation, the organizational conditions of mining operations, as well as the means of mechanization installed by consumers of raw materials — at the crushing and processing plant, CHP, warehouse, etc
5. The smaller the number of operating machines and the mechanisms included in the complex, the more reliable, productive and economical its work is.
6. Individual machines and mechanisms of the complex should correspond to each other in their parameters (loading and unloading height, ratio of geometric capacities, dynamic loads, etc.), as a rule, be standard and serial, so that replacement is possible. The equipment manufactured according

to special requests should be used only in special cases — with the unique scale of mining operations or specific conditions of occurrence of the deposit, when the use of standard equipment does not ensure the achievement of the desired effect.

7. The coefficient of reserve capacity and technical performance of individual machines in comparison with the average hourly

8. indicators of their work in accordance with the nature of mining production should be at least 1.2—1.3 (in the development of soft rocks) and not more than 1.5—1.7 (in the development of rocky and heterogeneous rocks).

9. Complexes should, if possible, be provided with machines and mechanisms of continuous operation.

10. If possible, preference should be given to one powerful machine instead of several machines of lower power.

11. However, the use of a high-performance powerful machine with high energy and metal consumption with insufficient annual load worsens the economic performance of work compared to the performance of two machines, smaller in weight and power, but able to perform the required amount of work. The best economic effect is always achieved under the condition of full use of the power and productivity of the machines and mechanisms included in the complex, primarily the leading machines of the equipment complex.

12. The leading machines, to which other elements of the complex are subordinate, are, as a rule, excavation and loading machines and means of transport; with exceptionally difficult-to-develop rocks, drilling rigs can limit the productive work of the entire complex; in most cases, productivity is limited by the capabilities of quarry transport.

13. Preference should be given to equipment complexes, when using which the number of labor-intensive and poorly mechanized auxiliary processes and operations is minimal. The complete set of means of mechanization of auxiliary works and processes should ensure a minimum time for their execution. It is necessary to focus on the use of powerful auxiliary equipment for the maintenance of several equipment complexes with clear planning and management of their work.

14. Any equipment complexes must fully meet the requirements of mining safety, ensure the completeness of extraction of mineral reserves from the subsoil, the required quality of products and the possibility of integrated use of all types and grades of minerals.

In most cases, it is advisable to use various sets of equipment for stripping and mining operations. In cases where it is not possible to allocate independent ledges or blocks within a mineral deposit, a single set of equipment can also be used, having only various means of mechanization of the storage of rocks and minerals.

Thus, the basic principles on which the formation of equipment complexes is based are: in-line production, possible combination of processes, the shortest distance of movement of rock mass, reduction of the number and volume of auxiliary work. In specific complexes, these principles are implemented in a certain way to obtain the best technical and economic indicators of development, primarily for labor.

Key words: cargo flow, complex of mining and transport equipment, completeness of mechanization, drilling, blasting and excavation processes, qualitative, quantitative, complex mechanization and automation, rock resistance, relative indicator of the difficulty of rock development, classes and categories of development. the main mining and transport equipment, auxiliary equipment, crushing and screening equipment, machines.

Security questions:

1. What do you mean by the completeness of mechanization?
2. What is the essence of the technological processes of mining?
3. What factors explain the resistance of rocks for each subsequent process?
4. What are the distinctive features of the breeds of each class, divided according to the relative indicator of the difficulty of development?
5. What are the main requirements for equipment complexes?
6. What requirements should the equipment complexes meet?
7. What should the complete set of the means of mechanization of auxiliary works and processes provide?.

Lecture 13

Topic: Technological classification of equipment complexes

Plan:

1. Dredging and dump complexes of equipment.
2. Excavator-dump complexes of equipment.
3. Excavation, transport and dump complexes of equipment.
4. Excavator-transport-dump complexes of equipment.
5. Removal and transport-unloading complexes of equipment.

The equipment complexes used and implemented in quarries can be divided into six technological classes (Table 14.1).

In the presence of continuous-acting dredging and loading equipment, equipment complexes are called dredging, and with cyclic-acting dredging and loading equipment — excavator.

Complexes of equipment for stripping operations necessarily include means of mechanization of dump operations, and complexes of equipment for mining operations — means of mechanization of unloading operations.

Dredging and dump complexes of equipment (VO) include rotary and chain excavators, cantilever dumpers or transport dump bridges (Fig. 14.1, a).

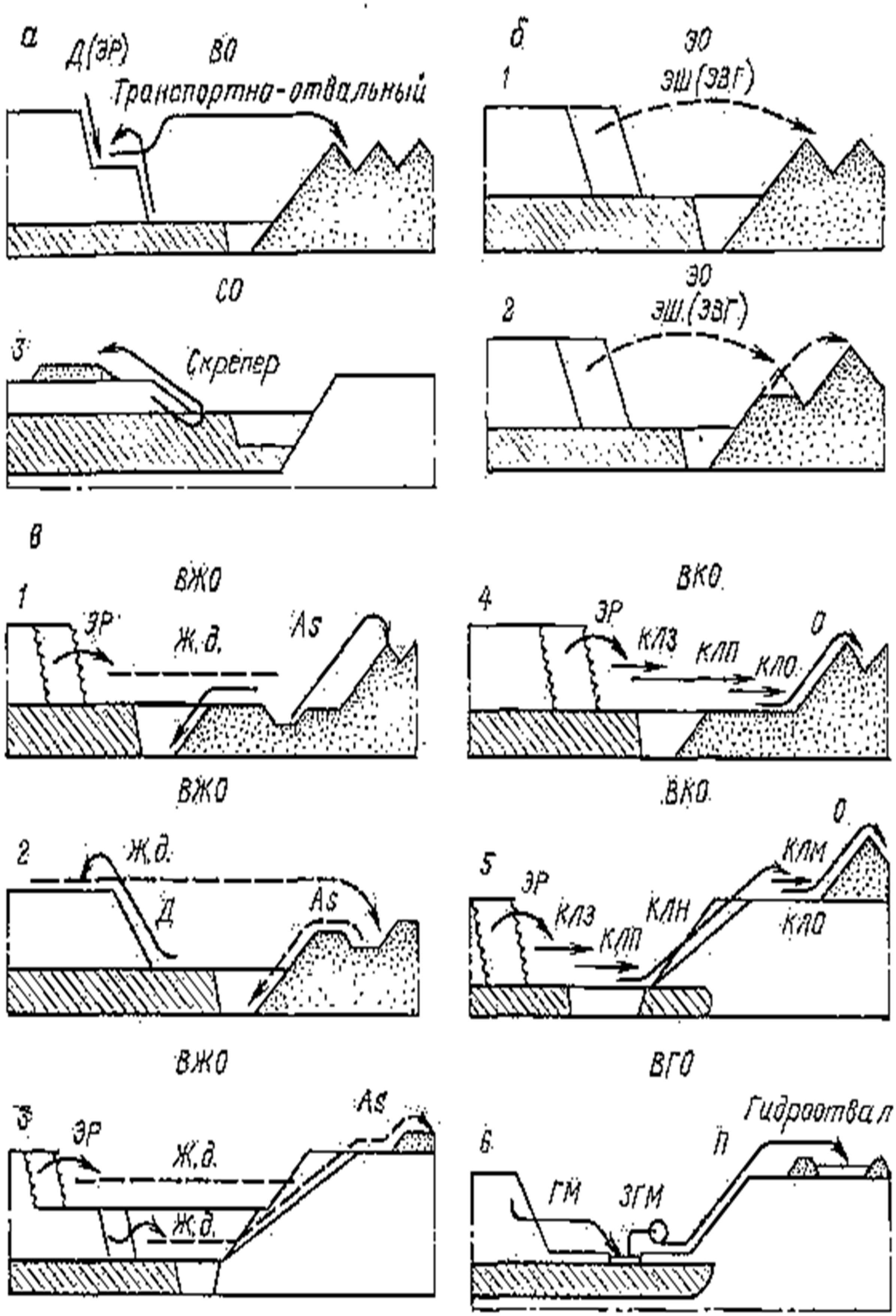
The main machines of excavator-dump equipment complexes (EO) are powerful overburden mechlopats or draglines used for transshipment of overburden rocks into the worked-out space (Fig. 14.1, b, 1, 2). Complexes of scraper equipment (CO) belong to the same class (Fig. 14.1, b, 3).

A characteristic feature of the excavation and transport dump complexes of equipment (WTO) is the continuity of the excavation of soft or finely-torn rocks and the transportation of overburden rocks (Fig. 14.1, c).

For excavator-transport-dump complexes of equipment (THIS) is characterized by the use of cyclic excavators during excavation and loading, and for moving — almost all known modes of transport (Fig. 14.1, d).

Excavation (excavator)-transport and unloading equipment complexes (VTR and ETR) differ in the presence of unloading devices on the surface or at consumers (Fig. 14.1, d, e).

Further differentiation of equipment complexes is carried out in close connection with the technology of mining operations by types of equipment of the leading process (excavation and loading operations, cargo movement and dumping). In this case, the determining role, as a rule, belongs to the type of transport used, the name of which is included in the name of the complexes (EKO, EZHO, EAR, etc.).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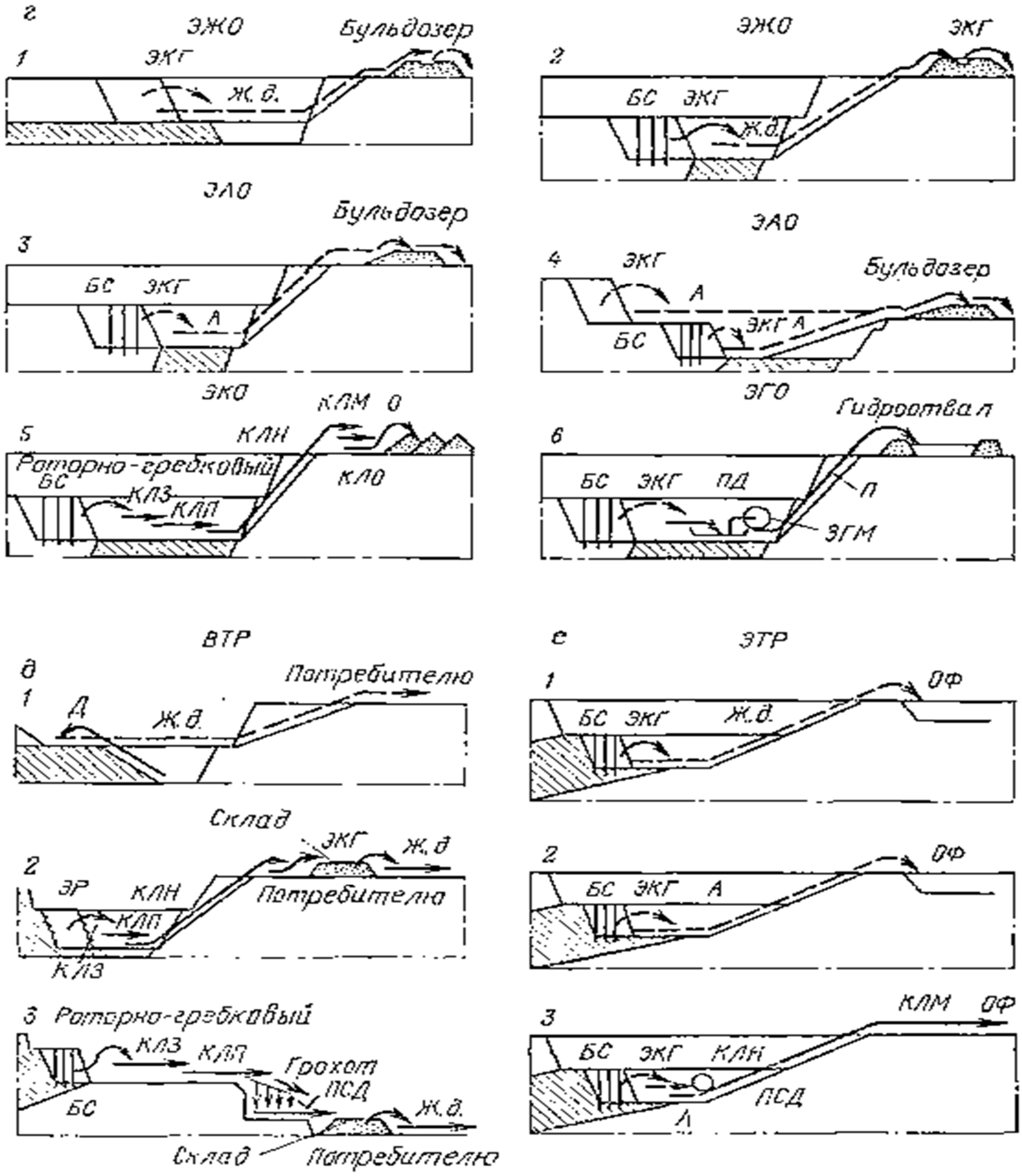


Fig. 14.1. Schemes of equipment complexes used in open development:

a — excavation-dump; b — excavator-dump (1 and 2 — with simple and multiple transshipment; 3 — scraper); c — excavation-transport-dump (1, 2 and 3 — with railway transport; 4 and 5 — with conveyor transport; 6 — with hydraulic transport); d — excavator-transport-dump (1 and 2 with railway transport; 3 and 4 — with road transport; 5 — with conveyor transport; 6 — with hydraulic transport); d — excavation-transport-unloading (1 — with railway transport; 2 and 3 with conveyor transport); e — excavator-transport-unloading (1 — with railway transport; 2 — with road transport; 3 with motor conveyor transport); D — chain excavator; ER — rotary excavator; ASH — dragline; EVG and ECG — respectively stripping and quarry mechlopats; O — cantilever dumper; As — abzetzer; KLZ, KLP, KLN, KLM and KLO — belt conveyors, respectively, downhole, transfer, inclined (lifting), trunk, dump; GM — hydraulic monitor: ZGM — dredger; P — pulpovod; BS — drilling rig; A — motor transport; PD and PSD — mobile and semi-stationary crushers, respectively; OF — processing plant.

Table 19.1. Technological classification of equipment complexes used in open-source development

Класс комплексов	Комплекс оборудования	Тип оборудования комплекса		
		Выемочно-погрузочные работы	Транспортирование	Отвалообразование и складирование
1	Выемочно-отвальный (ВО)	Роторные и цепные экскаваторы	Нет	Транспортно-отвальные мосты, консольные отвалообразователи
2	Экскаваторно-отвальный (ЭО, СО)	Вскрышные экскаваторы, скреперы	Нет	Вскрышные экскаваторы, скреперы
3	Выемочно-транспортно-отвальный (ВТО)	1) Роторные и цепные экскаваторы, гидроразмыв (м. п.) * 2) Скальные комбайны, специализированные экскаваторы (с. п.) **	Конвейеры, гидротранспорт, железнодорожный транспорт и автопоезда	1) Консольные отвалообразователи, гидроотвалы (м. п.) 2) Отвальные машины (с. п.)

Since the most expensive and time-consuming process in open-pit mining is the movement of rocks, the minimum cost of funds is achieved either by moving rocks to the final position over the shortest distance, or by using cheap modes of transport.

When developing horizontal and shallow deposits, often all or part of the overburden rocks are moved to the developed space along the shortest distance — across the front of the ledge, while combining all or part of the technological processes. With dense and soft overburden rocks, the combination of technological processes is achieved by using:

excavation machines with the necessary dimensions of working equipment, when the complex includes only one type of main equipment — usually single-bucket excavators;

dredging machines and transport dump units, when equipment complexes include rotary excavators and cantilever dumpers or chain multi-bucket excavators and transport dump bridges.

In addition to the partial combination of processes, the use of dredging equipment complexes ensures the continuity of all processes.

When moving soft overburden rocks along the work front of ledges into internal or external dumps, typical equipment complexes are:

rotary excavators — conveyor transport — cantilever dumpers;

chain multi—bucket excavators - railway transport — abzetzer;

single—bucket excavators - conveyor transport with hopper feeders — cantilever dumpers;

scrapers or bulldozers.

The equipment complexes used in the development and longitudinal movement of rock and semi-rock are very diverse, as are the types and properties of bedrock and the conditions of their occurrence. Usually complexes include as the main equipment drilling rigs of various types (in the preparation of semi—rock — sometimes mechanical rippers), single-bucket excavators of the mechlopat type (sometimes when excavating small-blown rocks - single-bucket loaders), various vehicles, dumpers, the choice of which depends primarily on the type of transport used.

The most common (up to 1/2 of the volume of mining operations) complexes with railway transport and single-bucket excavators on dumps. Complexes with road transport and bulldozers on dumps are also widely used.

Complexes with automobile and railway transport are widely used in deep quarries. Complexes with motor-conveyor and motor-skip transport are promising, as well as (with an acceptable size of the blasted rocks or additional mechanical crushing of them at loading points on working ledges) complexes using only conveyor transport, and as excavation and loading machines - respectively continuous—acting equipment and single-bucket excavators.

When developing deposits of the upland type, in addition to those listed, complexes with combined transport are used, including in various combinations automobile transport, ore passes, cable-suspension roads, railway transport.

In the development of rock and semi-rock formations, equipment complexes with mechanization vehicles are characterized by relative independence of the processes. The degree of independence varies for different processes and is determined primarily by the technical feasibility and economic feasibility of creating reserves (reserves) of rock mass necessary for the smooth execution of the next process. For example, drilling and blasting can be largely independent of each other and of excavation and loading operations when creating sufficient reserves of rock prepared for development and ready for excavation; road and rail transport (in combination) - if the transshipment point is a warehouse. The operation of dump excavators and railway transport is characterized by a much lesser degree of independence, provided due to the relatively small volume of rock remaining in the receiving bunker after unloading the train. Complexes with single-bucket excavators and conveyor transport are characterized by minimal independence, the connection between which is carried out through hopper feeders (due to their small capacity).

In addition to the reserves of rock mass, the independence of the processes is ensured by the technical possibility of their implementation in case of failure of some basic means of mechanization, for example, one or two railway trains, several cars, a dump excavator. In this case, in the course of work, it is possible to redistribute technical means, which, of course, leads to a decrease in productivity or downtime of equipment, but not to the termination of work.

Reference words: excavation, excavator, means of mechanization of dump operations, means of mechanization of unloading operations, main machines, excavation-dump, excavator-dump, excavation-transport-dump, excavator-transport-dump, excavation-transport-unloading.

Security questions:

1. What classes are divided into equipment complexes used and implemented in quarries?
2. Which equipment complexes are called dredging?
3. Which equipment complexes are called excavating?
4. Which machines and mechanisms include dredging-dump complexes?
5. What is the difference between dredging-transport-unloading equipment complexes from other complexes?

Lecture 14

Topic: Longitudinal, transverse, fan and ring development systems

Plan:

1. Longitudinal development systems.
2. Transverse development systems.
3. Fan development systems.
4. Ring development systems.

Longitudinal and transverse mining systems are preferred for elongated deposits having a shape close to a rectangle or an elongated oval in plan.

The longitudinal single-board system of development by horizontal layers is widespread in large open-pit fields of elongated shape; it allows the use of:

HE and EO complexes at the shortest distance of movement of overburden rocks into internal dumps;

WTO complexes and THIS is with the movement of rocks by vehicles along the work front;

the same complexes are simultaneously used for transshipment of rocks of the lower ledge into internal dumps and moving rocks of the upper ledges by vehicles to external or internal dumps.

Mining operations are carried out using, as a rule, independent excavation and loading and transport equipment.

The longitudinal single-side system is characterized by parallel movement of the work front of the ledges. At the same time, the width of the panel being developed or the entry along the entire length of the front is the same.

Transport communications include downhole paths or roads Z, connecting paths on the berms M and the paths of the capital trench K (Fig. 15.1, a). The point of junction of mobile (downhole) paths to stationary paths is transferred as the front moves, the junction itself is carried out on connecting berms left on the non-working side of the quarry, and connecting paths periodically lengthen. The traffic pattern of means of transport does not change when moving the junction point.

When using the complexes of THIS and ETR, longitudinal parallel excavator approaches (along the strike of the deposit) provide a sufficient front for placing two or three excavators on the ledge, following one after the other with some lag. In railway transport, it is necessary to systematically transfer the curved part of the track (see Fig. 15.1, a).

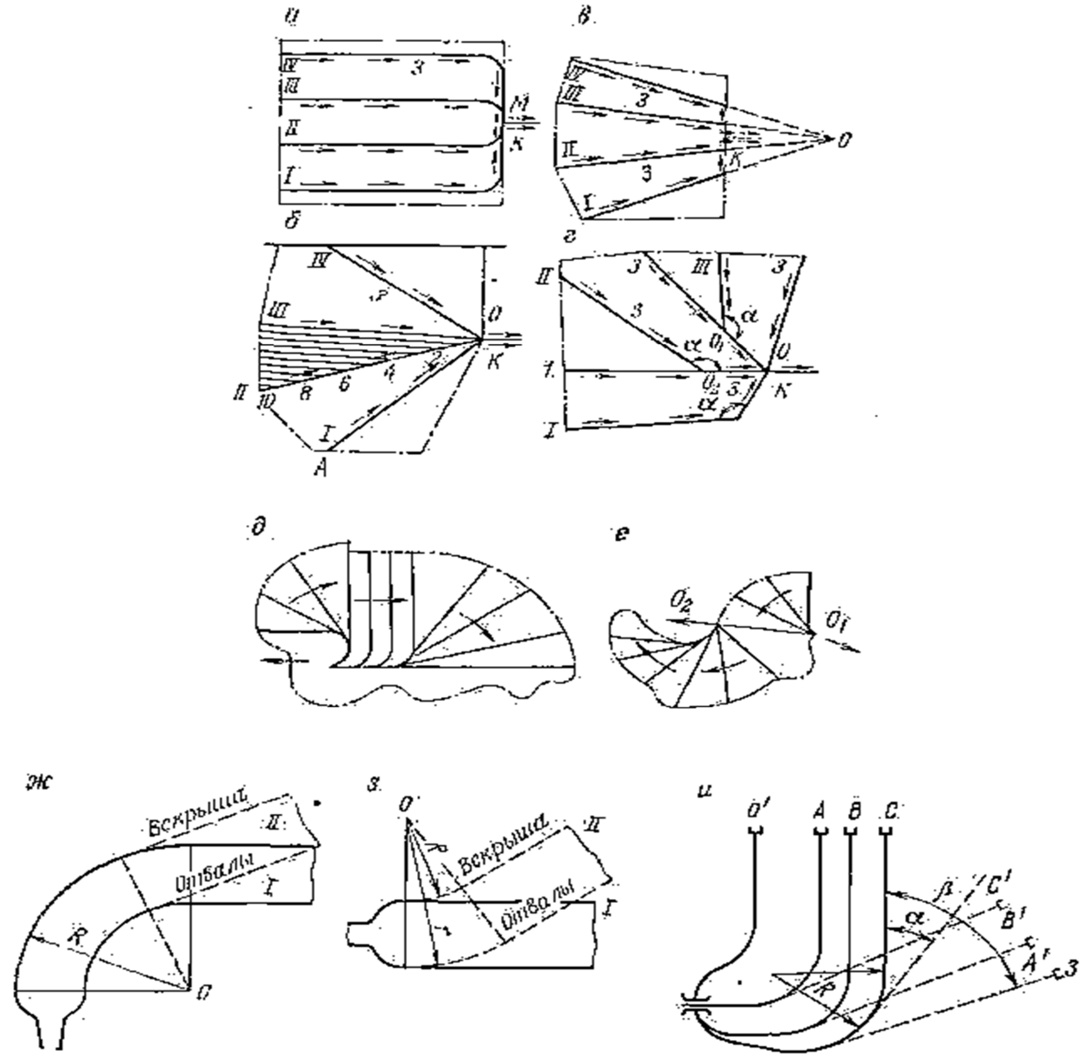


Fig. 15.1. Schemes of moving the work front:

I – IV – the sequence of the position of the work front; A, B, C and A', B', C' — the positions of the downhole tracks, respectively, before and after turning the angle □; O' — the dump path.

Parallel movement of the mining front is typical when using mechanical shovels and rotary excavators in combination with wheeled and conveyor transport. It is relatively rarely used with chain excavators and transport dump bridges; at the same time, a large number of tracks are arranged on the sites, and their transfer on curved sections is difficult and time-consuming.

The longitudinal two-board system of horizontal layers mining is sometimes used for very large quarry fields and significant mineral reserves. With this system, enterprises of large production capacity are created, especially in cases where the lowest capacity of overburden falls in the middle of the quarry field.

Transverse single-board development systems use:

with relatively narrow and elongated or dispersed deposits, when the mining front is located parallel to the long axis of the quarry field, it is impractical due to the large volume of mining and capital works and a short period of operation;

when the length and width of the quarry field are large (and close in size), when the location of the work front parallel to the short axis of the quarry is sufficient to accommodate one powerful complex of stripping equipment.

In the first case, EAO complexes are usually used, and in the second case, EO, HE or EKO complexes are used.

The fan-shaped central development system is effective with a rounded and close to triangular configuration of the quarry field, which makes it possible to conveniently locate a permanent turning point. In rare cases, a two-board fan development system is used.

With fan development, the movement of the work front occurs as the ledges are worked out so that its beginning is always at a constant turning point O, and the end describes a part of the circle with a radius equal to the length of the ledge (Fig. 15.1, b). At the same time, the speeds of movement of individual places of the front vary from zero at the turning point to a maximum at the end of the ledge.

The ledges are worked out with variable-width approaches having the shape of a triangle or trapezoid in plan, or with constant-width approaches, but with a different number of them on separate sections of the work front and periodic excavation of "wedges" at the beginning of each approach.

The fan system is usually used in the development of soft rocks by complexes with chain excavators and railway transport when moving railway tracks with continuous-acting track motors. During a certain number of shifts, excavators excavate the rock at the corresponding pickets (0-10) in such a way that the initial pickets (0-1) account for one unit of front movement, while at the final pickets (9-10) the movement is nine units.

The position of the turning point during the mining of the quarry field remains unchanged, only the "deployment" of the curves of the turning point is performed, working out one end of the quarry. The length of the work front remains unchanged. With the fan movement of the front, only one-way transport access to the ledges and a dead-end train traffic pattern within the horizon in railway transport are possible. Thanks to the permanent turning point, the connection of the paths of the capital trench to the paths of the working horizons of the quarry is facilitated and the need for systematic labor-intensive work on the transfer of curved sections of the tracks is eliminated. The distance of movement of the rock mass is reduced, and in some cases, the volume of mining and preparatory work. It is convenient to place industrial structures (traction substations, depots, workshops, etc.) and permanent drainage installations at the junction of the tracks. The presence of a minimum number of switches allows the use of continuous track-moving machines.

When using complexes with chain excavators, seasonal overburden operations and year—round mining operations are characteristic. Therefore, it is necessary to have a significant amount of mineral reserves opened and ready for extraction for the winter period. To increase it, sometimes the center of the turning point is moved beyond the contour / quarry (Fig. 15.1, c) or a mixed fan and parallel movement of the front is used (Fig. 15.1, d). In this case, the whole of the opened reserves in the plan takes the form of a trapezoid, and its relative volume increases.

During the period of the career, the development system may change: one part of the career field is worked out using a longitudinal development system, and the other — a fan development system (Fig. 15.1, e). With the fan movement of the front, the turning point is moved with a change in the direction of the fan turn (Fig. 15.1, e).

The design of the turning point is chosen based on the requirements of the completeness of mining of the quarry field, reliable operation of transport during the entire period of operation of the quarry and the minimum amount of mining and capital works.

The center of rotation can be placed on the side of the non-working side of the quarry (Fig. 15.1, g) and on the side of its working side (Fig. 15.1, h). In the first case, as the rotation increases, the length of the work front increases and the area of the quarry field area that is worked out at one position of the turning point increases. However, at the same time, the volume of work on its construction increases.

As the work front moves, the rails are moved, but they are constantly located tangentially to the corresponding curves of the turning point. When the work front is moved and the downhole tracks are rotated by an angle □ (Fig. 15.1, and), part of the curve of the turning point straightens and the length of the front increases. The angle of rotation of the fan □ is chosen from the condition of the largest possible area of the quarry field, worked out without rearranging stationary paths; at angles □> 180 °, certain difficulties arise.

The ring central mining system is used in some cases when areas with a small capacity of overburden or with minerals of better quality are confined to the middle of the quarry field, as well as with its favorable outlines (Fig. 15.2, a and b). Overburden rocks are moved to internal and external dumps, since the capacity of internal dumps is insufficient to accommodate the entire volume of overburden rocks. It is most convenient with such a system to move overburden rocks and minerals using vehicles (see Fig. 15.2, a).

If the mineral deposit has a rounded shape in plan and the overburden capacity is minimal in some areas of the field and maximum in its middle, it is economically advantageous to use an annular peripheral development system (Fig. 15.2, c).

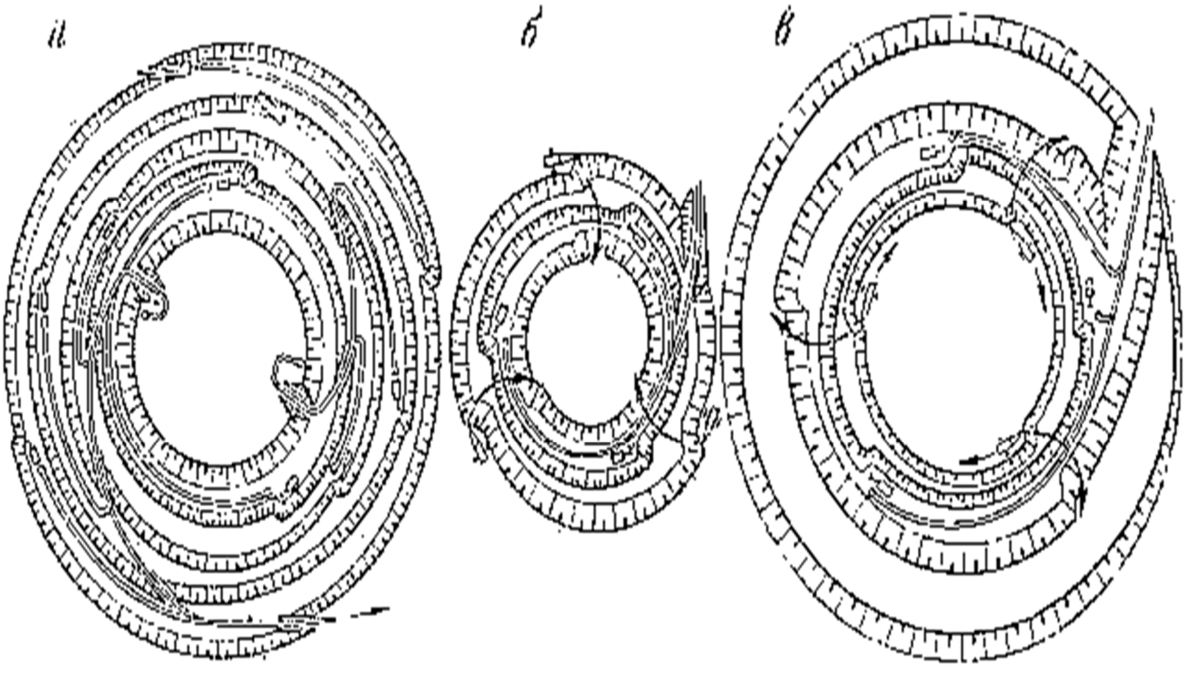


Fig. 15.2. Solid ring development systems.

Reference words: elongated deposit, large quarry fields, longitudinal single-sided, longitudinal double-sided, transverse single-sided, central fan, complexes of VO, EO, WTO and THIS.

Security questions:

1. Describe the longitudinal development system.
2. In what cases are transverse single-board development systems used?
3. What equipment complexes are used for longitudinal development systems?
4. In what cases are fan development systems used?
5. In what cases are ring development systems used?

Lecture 15

Topic: Opening working horizons with continuous development systems

Plan:

1. Trenchless autopsy.
2. Opening by external trenches.
3. Opening by internal trenches.
4. Opening routes.

Trenchless opening is typical: when developing deposits using tower excavators and rope scrapers; for overburden horizons when using equipment complexes IN, EO, EKO and ECO during internal dumping (Fig. 16.1, b, c and e), as well as for mining horizons when using inter—stage reloaders - conveyor bridges;

when cutting advanced ledges as a result of increasing the relief of the surface, transport access to which is carried out directly from the surface (see Fig. 16.1, b);

when using bulldozer and scraper complexes, if the movement of machines on the ascent (descent) is carried out along the slope of the ledge;

in cases of application of hydromechanized and drazhny complexes of the equipment.

External separate trenches are typical for opening:

one mining horizon during the development of horizontal (see Fig. 16.1, b, c, d and h) and shallow deposits, if the angle of incidence of the latter □ does not exceed the permissible rise i for the accepted mode of transport (tg □≤i) (Fig. 16.1, o);

one overburden horizon when using scraper and bulldozer complexes;

one or two overburden horizons in automobile or conveyor transport and external dumping (fig. 16.1, d);

one overburden horizon for railway transport and other means of transport for mining operations.

External group trenches in similar conditions are used to open:

two (rarely more) mining horizons;

of all overburden horizons when using railway, and in mining operations — conveyor, automobile or railway (with the dispersal of overburden and mining cargo flows) transport, when tg □≤i;

groups of upper overburden horizons in railway transport, when the lower overburden ledge is worked out using an excavator-dump complex, and mining — by another type of transport.

External general trenches are used to open horizontal shallow (tg □≤i) deposits when using railway transport (Fig. 16.1, a).

Internal trenches are often used to open all or a group of upper overburden horizons in road transport (Fig. 16.1, d, e, n and n).

Trenches of mixed laying (external and internal) are typical for opening shallow deposits using railway transport (Fig. 16.1, w, z, i, k, l, m, o and p). They are often used in the development of horizontal and shallow deposits of limited size using road transport.

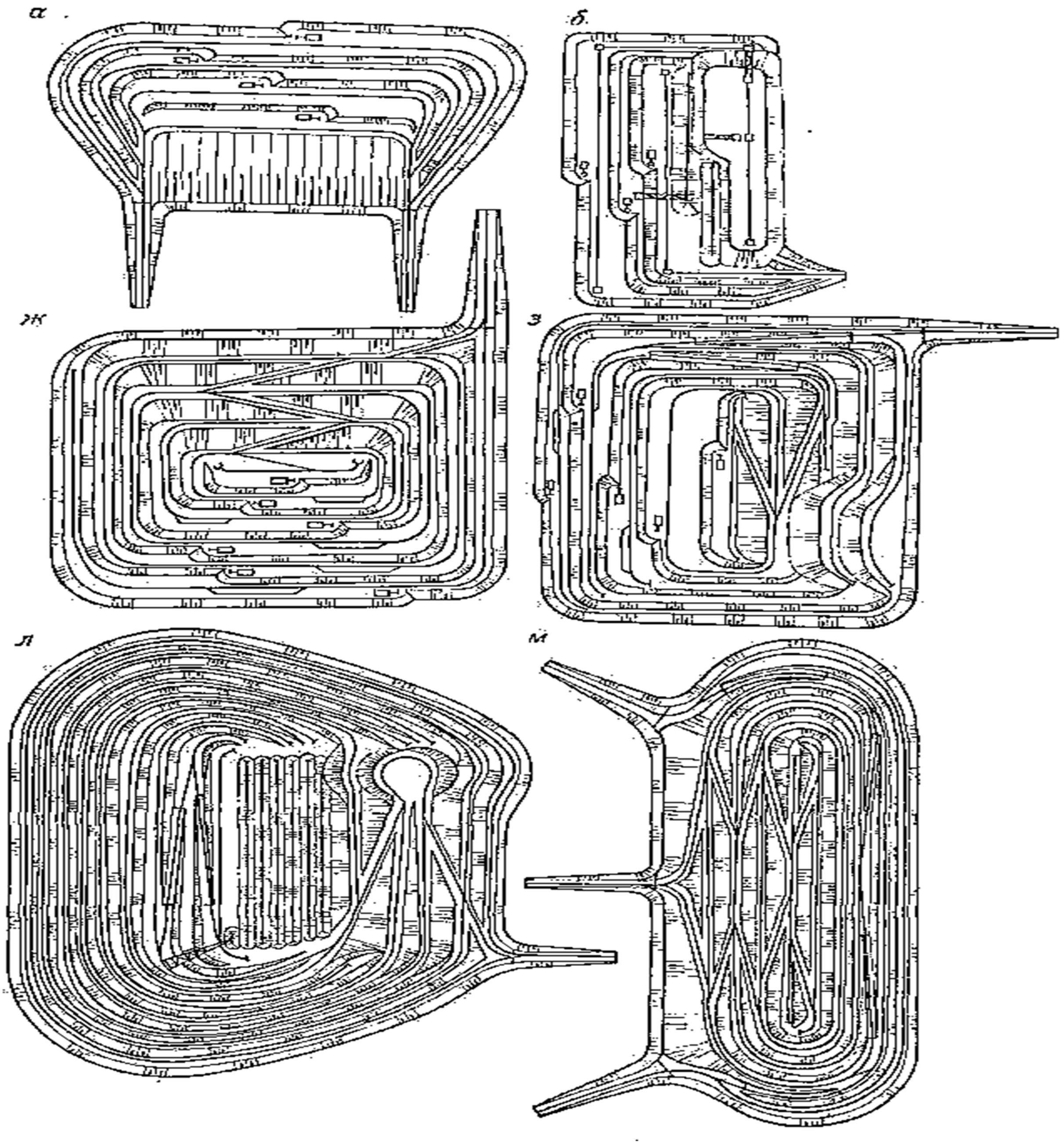
Auxiliary automobile exits are usually carried out to the working horizons where railway and conveyor transport is used, as well as the HE and EO complexes (see Fig. 16.1, b).

Schemes of opening routes for horizontal and shallow deposits differ: types of opening workings used; the number of working horizons opened by a common route; the number of trenches with different routes; the location of external and internal trenches relative to the contour of the quarry field (working and non-working longitudinal or endboard, internal dumps, combinations thereof); the shape of the trenches and the number of horizons revealed by a straight section of the route.

Specific methods of opening are characterized by a combination of the above factors, taken into account mining-geological and mining-technical conditions at the beginning, at individual stages and at the end of field development. Therefore, these methods of autopsy are individual and practically do not repeat each other.

When developing horizontal and shallow deposits, the main groups of schemes of opening routes differ in the position of the axis of the opening workings relative to the contour of the quarry.

Opening with the use of a single flanking external common trench is typical for a single dead-end front with longitudinal single-side and fan systems for the development and use of EJO and EJR complexes. Two flanking trenches are characteristic of a through single and double dead-end mining front (Fig. 16.1, a). In some cases, placers and deposits of construction rocks are opened with one central trench when using vehicles and preparing horizons with split trenches or pits.



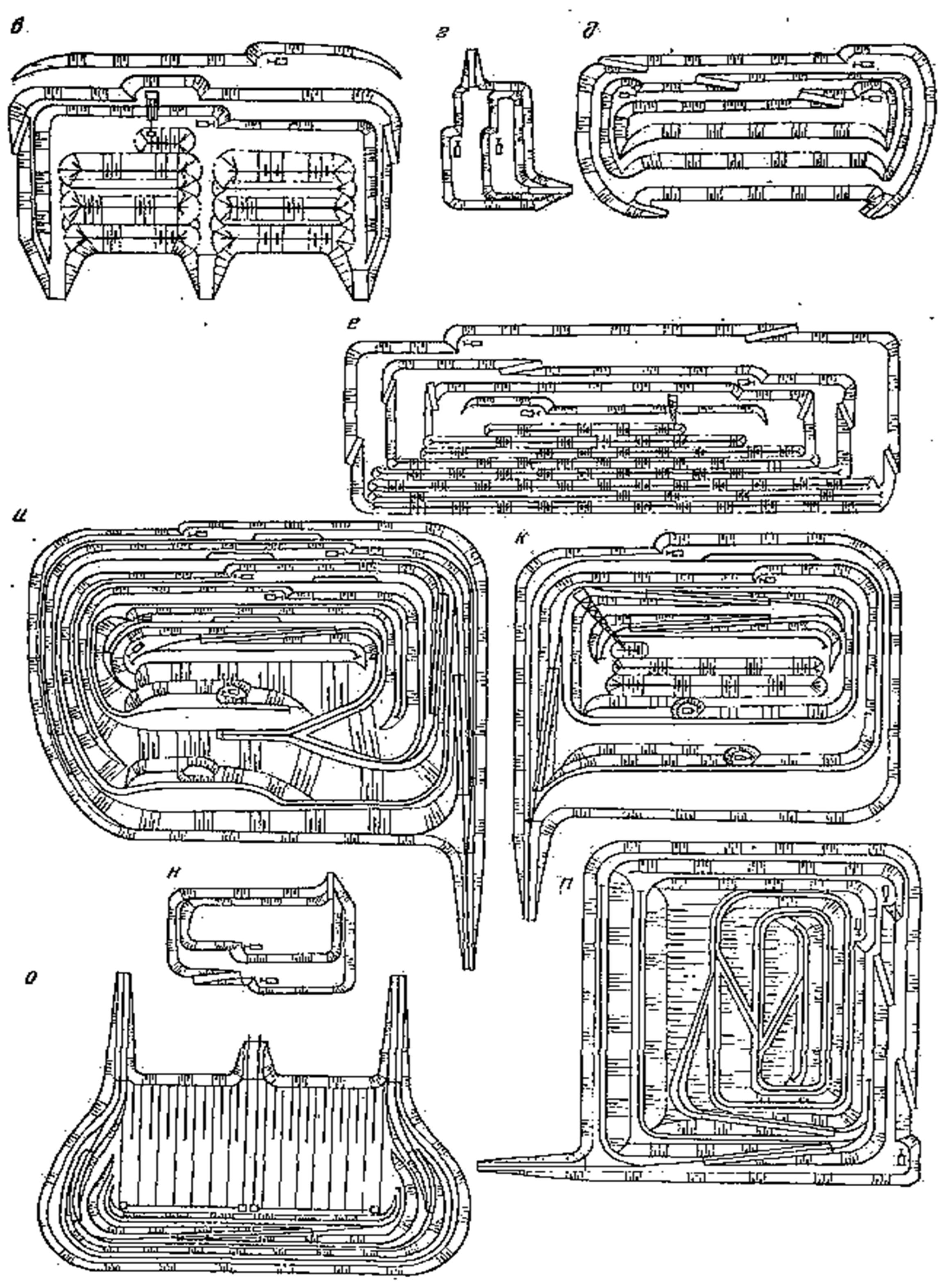


Fig. 16.1. Methods of opening horizontal and shallow deposits.

External group and separate trenches are used when working out the entire or lower part of the overburden strata in horizontal deposits with complexes of equipment for EO and HE both with longitudinal (see Fig. 16.1, b and c) and with fan (Fig. 16.2) development systems. The number of trenches (one, two or three) depends on the size of the quarry field.

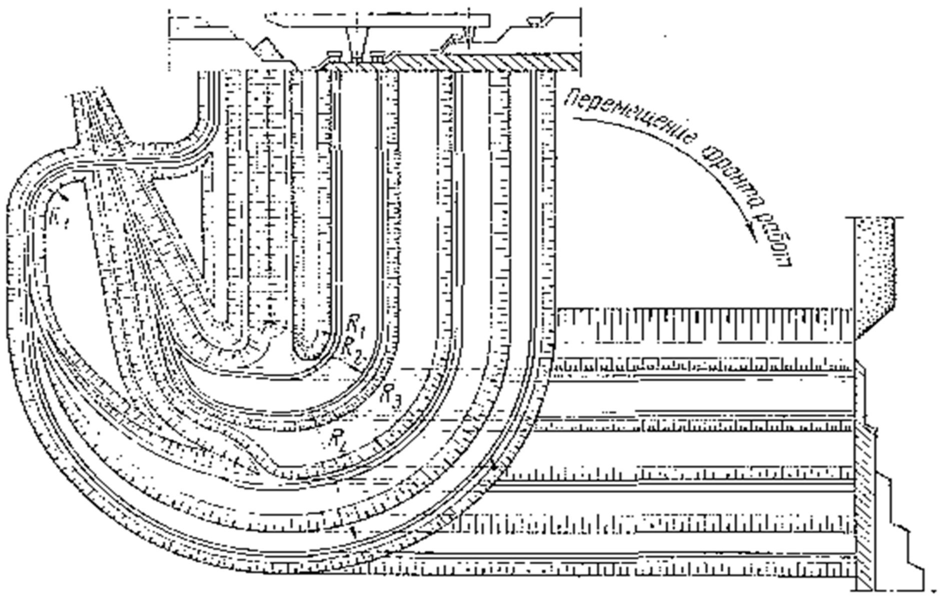


Fig. 16.2. The opening scheme and the design of the turning point for the fan system and the use of the HE, HE and VZHR complexes

Separate and group external trenches are used in the development of deposits of small size in plan and depth (placers, construction rocks) when using scrapers, loaders, motor vehicles, less often conveyor transport (see Fig. 16.1, d).

Schemes of opening routes of internal laying are widely used during the construction of quarries during the operation of EAO complexes. Trenches are mainly group or general; temporary exits placed on the working side of the quarry are also used (see Fig. 16.1, e). When transshipment of rocks of the lower overburden ledge into the worked-out space, the opening of the mining horizon with motor transport in some cases is carried out using internal semi-stationary exits along the end sides of the quarry (see Fig. 16.1, e).

Routes of mixed (external and internal) laying are used in the development of shallow deposits (tg □>i), primarily when using railway transport for stripping operations.

Common one or two flanking trenches of mixed laying are characteristic when transporting the entire mountain mass by rail, respectively, with a single dead-end or single through and double dead-end front of mining operations (see Fig. 16.1, w, w and i).

With external dumping, the inner part of the route is usually located on the longitudinal non-working side of the quarry. The shape of the internal route with one trench is a dead—end multi-stage (see Fig. 16.1, g). A single-stage dead-end route is characterized by the arrangement of dead ends on each horizon, and a multi-stage (translational-dead-end) is characterized by the arrangement of dead ends through n horizons

$$n \approx \frac{L_{к.ср}}{l_T} = \frac{L_{к.ср} i_p}{K_y H_y},$$

where $L_{к.ср}$ — average career length, м; $l_т$ — the length of the route section when opening one horizon, м; K_y— the coefficient of elongation of a simple internal route when adjacent to the sites.

When moving overburden rocks into internal dumps, the track is often located at the end of the quarry and the lower part of the longitudinal non-working side, free of dumps (see Fig. 16.1, h). Often the lower part of the internal track is placed on the working side of the quarry (see Fig. 16.1, and).

When forming internal multi-tiered dumps with partial transshipment and transportation of rocks by rail, the inner part of the route is usually located at the end and on the working side of the quarry (see Fig. 16.1, k). When only transport-free dumps are placed in the developed space, the upper part of the inner route is arranged on the same dumps (see Fig. 16.1, l). One or two flanking group trenches are usually used.

In very powerful quarries of great length, the opening of overburden horizons can be carried out by two flanking trenches with the placement of the inner part of the route at the end and on the working side of the quarry, and mining horizons — by a central trench of mixed laying with one or two internal routes (see Fig. 16.1, m). The opening of one common central trench in wheeled transport is rarely used due to the increase in the mileage of vehicles along the horizons compared to the flank opening.

Schemes with parallel use of opening routes of external, internal and mixed laying are used for:

opening of mining and overburden horizons, respectively, external and internal by separate trenches (see Fig. 16.1, k), which is typical for many placers and deposits for the extraction of building rocks;

opening of overburden horizons by flanking trenches of mixed laying, and mining horizons by external central ones (one or several). This is typical when using railway and conveyor transport, respectively, in stripping and mining operations (see Fig. 16.1, o);

autopsy, when railway and road transport are used simultaneously during stripping operations (see Fig.16.1, p).

Schemes of this group are widely used when using both different and one type of transport in a quarry to reduce the distance of transporting rock mass, accelerate the opening and preparation of horizons, etc.

Systems of opening routes in the development of horizontal and shallow (tgp^t) deposits are often adequate to the schemes of opening routes (due to the invariability of opening during the life of the quarry). When opening internal trenches with a sliding or semi-stationary route, the system of opening routes is characterized by a regular change in the position of opening workings in the plan. The position of only a part of the opening workings often changes. Periods of unchanged position of individual tranches range from three to four months to several years.

In general, systems of opening routes of shallow deposits are more dynamic when tg □>i. Their outer segments, as a rule, remain unchanged. When the inner part of the route is located on the longitudinal non-working side of the quarry, it gradually deepens during new half-trenches and the device of dead ends or loop connections. The lower part of the inner track, when located on the end side of the quarry and the longitudinal non-working side (when moving rocks to internal dumps by rail), is semi-stationary. Along with the transfer of dead-end exits on the lower horizons, with an increase in the width of the quarry, some of the dead-end connections on the medium-depth overburden horizons are eliminated. With the depth of the quarry, the route of opening workings as a whole becomes more complicated: the number of turns of the route and the length of its part located on the working side of the quarry increase; the number of ledges opened by a straight section of the route decreases. At the same time, the possible number of trenches for opening the lower horizons decreases.

Reference words: trenchless opening, external individual trenches, external group trenches, external common trenches, internal trenches, trenches of mixed laying, schemes of opening routes, internal laying, mixed laying, parallel use of opening routes.

Security questions:

1. Which deposits are characterized by trenchless opening?
2. In what cases are external separate trenches used?
3. What is the difference between the schemes of opening routes for horizontal and shallow deposits?
4. What is the characteristic of a single-stage dead-end highway?
5. In what cases are schemes with parallel use of opening routes used?

Lecture 16

Topic: The procedure of excavation by excavator-dump technological complexes

Plan:

1. Conditions of application of technological complexes.

2. Calculations and analysis of technological complexes and schemes of excavation.

The use of technological complexes for transshipment of overburden rocks into the developed space is very economical and desirable in all cases when possible, in particular under the following conditions:

horizontal or gentle fall of a mineral deposit (usually no more than 10-12 °, sometimes up to 15-17 °);

limited capacity of deposits (up to 20-30 m, in some cases up to 50-60 m) and overburden rocks (up to 40-45 m, sometimes up to 60 m).

These technological complexes are also used when working out the outputs of inclined and steep deposits or narrow, elongated and shallow mineral lenses. At the same time, direct or multiple transshipment of overburden rocks to the sides of the quarry is carried out.

Calculations of rigidly interdependent stripping and mining technological complexes are related to:

the choice of the type and capacity of stripping excavators and the excavation scheme;

with the choice of the type of mining complex, primarily the transport of minerals, and the establishment of the order of transport maintenance of mining faces during the operation of the excavator-dump complex of equipment;

with the determination of the width of the approaches, berms, sites of stripping and mining ledges and the calculation of the uncovered mineral reserves.

The mutual arrangement of equipment should be calculated comprehensively: in terms of and according to several typical geological profiles. Calculations for only one transverse profile, without taking into account the arrangement and sequence of equipment operation in the plan (along the work front), can lead to significant errors.

An important condition for the correct calculation and construction of overburden and mining technological complexes is the equality or multiplicity of the width of the entry for minerals to the width of the entry for overburden rocks in order to evenly move the front of overburden and mining operations.

With an increase in the capacity of overburden rocks placed in internal dumps, as well as the capacity of the deposit, or in the absence of sufficiently powerful overburden excavators, simple and multiple transshipment of rock becomes either ineffective or technically impossible. The requirement to ensure sufficiently large uncovered mineral reserves during seasonal stripping operations leads to a significant complication of technological complexes.

Calculations and analysis of technological complexes and excavation schemes, as well as design solutions of the elements of the development system, are given below in relation to the most common longitudinal single-board development system.

After working off one overburden 1 and one overburden on the mineral u (Fig. 23.1, a), the rock from the next overburden 2 can be placed on the free area of the developed space (with the exception of the bottom-hole strip P) in the dump 20 (Fig. 23.1, b).

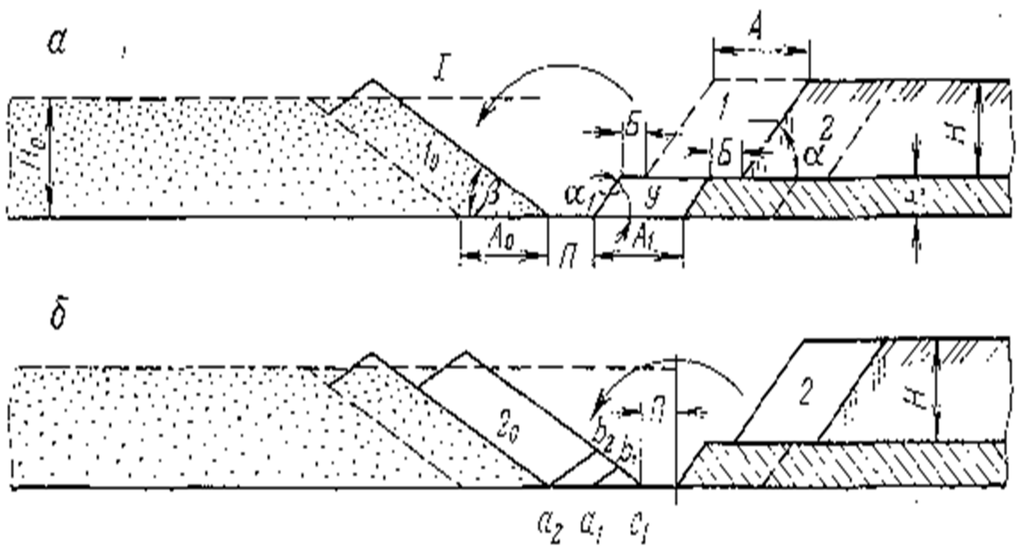


Fig. 23.1. Schematic diagram of simple transshipment of overburden rocks:

a and b – respectively, the position before and after the overburden 1; 1 – the average level of the blade.

Regardless of the order of excavation of the overburden, it is most economical to place the rock as close as possible to the strip P, filling in the triangular area a1, b1, c1 sequentially, and then the quadrilateral a2b2b1a1. At the same time, the slope of the dump of the previous approach is not filled with rock.

With an insignificant height of the overburden ledge H1 (Fig. 23.2, a), the capacity of the dump shown by the triangle / o may be sufficient, and a minimum unloading radius R0.1 is required for transshipment of overburden rocks.

With an increase in the height of the overburden to H2, H3, H4, etc., the volume of the rock being handled increases (areas 2, 3, 4; see Fig. 23.2, a). The rock should be placed higher up the slope of the dump in areas 20, 30, 40, etc., and the corresponding unloading radii should increase to sizes R0.2, R0.3, R0.4, etc..

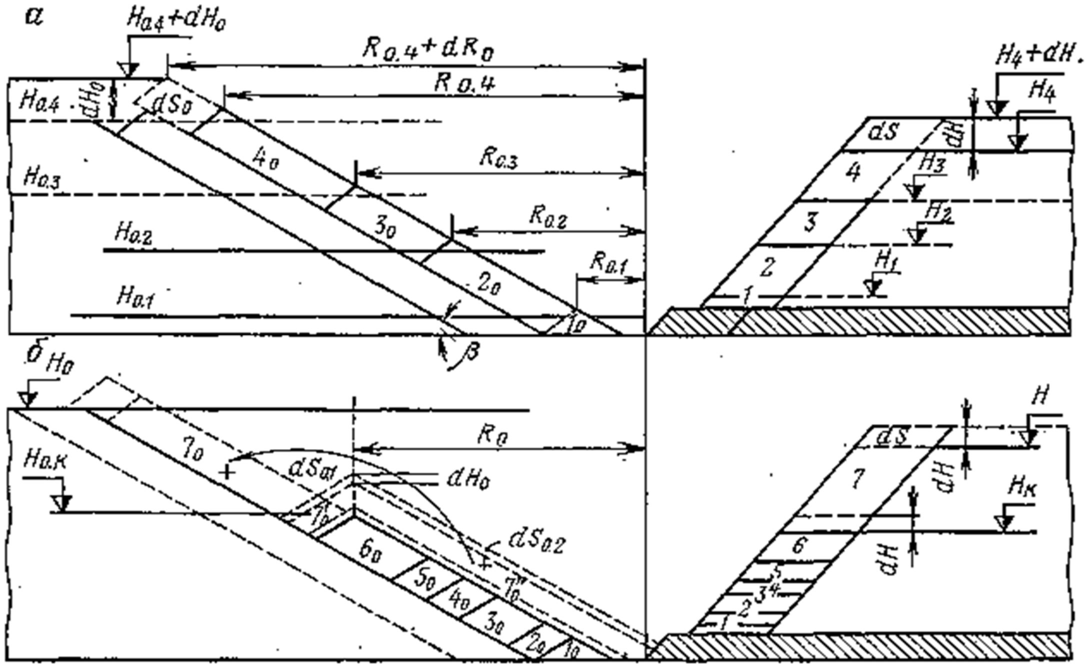


Fig. 23.2. Schemes for calculating the volumes of transshipment of overburden rocks

Such a simple transshipment, in which the entire rock is excavated only once (from the array) and is directly placed in the dump embankment, is in principle possible at any height of the overburden ledge.

However, the required unloading radius (m) of the overburden excavator increases significantly at a high height of the blade, since

$$R_0=П+H_0 ctg\ \beta.$$

Therefore, with a significant capacity of overburden rocks, multiple transshipment is used, in which the rock from the overburden in volumes 1, 2, 3, 4, 5 and 6 is poured into the dump in the same sequence and forms areas of 10, 20, 30, 40, 50 and 60. This is possible until the maximum unloading radius of the overburden excavator is fully used (Fig. 23.2, b). With an increase in the capacity of overburden rocks, the volume of the dump increases (70 = 70' + 70") due to the transshipment of the volume of rocks 7 without increasing the unloading radius of the overburden excavator, but with partial or complete filling of the bottom-hole strip and the slope of the mining ledge.

Practically, the excavation of the rock of the overburden is carried out at its entire height H, and in most cases the rock is poured at the maximum unloading radius R0 (see Fig. 23.2, b). At the same time, as a result of free fall and subsequent shedding along the slope, the rock fills the dump not in the described sequence (10, 20, ..., 70), and by increasing the area of the dump with thin layers dS0.1 + dS0.2 (see Fig. 23.2, b).

However, the originally described dumping procedure is more economical as a result of reducing the distance of movement of the rock. At the same time, the calculation methodology and economic evaluation of excavation schemes are simplified.

Part of the rock 70’ poured into the dump can remain in place, since its position corresponds to a simple transshipment. The second part 70" fills the bottom-hole strip and partially seals the slope of the mining ledge. To create conditions for the excavation of the deposit, this part of the dump rocks should be re-excavated and placed above the area of 70’ (in Fig. 23.2, b is shown by an arrow).

The ratio of the re-excavated volume of rock 70" to the total volume of the primary excavated rock 70'+70" is called the coefficient of multiplicity of transshipment (reexcavation):

$$K_{пер} = \frac{7_0^{"}}{7_0^{'} + 7_0^{"}}.$$

With the correct construction of the technological complex, always a larger or smaller volume of rock placed in the developed space will not be subsequently reexcavated, therefore, the coefficient of multiplicity of transshipment should be less than one. In specific mining and geological conditions, with small unloading radii of overburden excavators and especially with the development of landslides of dump rocks, the coefficient of multiplicity of transshipment may be greater than one, in some cases $К_{пер}$ = 3÷4 and more.

The economically acceptable coefficient of overexcavation is approximately determined from the expression

$$K_{пер} = \frac{C_T - C_б}{C_{пэ}},$$

where $С_т$ – costs per 1 m3 of stripping operations when using transport, sum; $С_б$ – costs per 1 m3 of stripping operations with simple transshipment, sum; $С_{пэ}$ – the cost of reexcavation of 1 m3 of rocks, sum.

By the value of the economically permissible coefficient of overexcavation for the accepted type of overburden excavators and the excavation scheme, it is possible to determine the maximum height of the ledge worked out with excavator transshipment of overburden rocks.

Reference words: fall of a deposit, choice of type and capacity, choice of type, calculation conditions, simple transshipment, multiple transshipment, dump embankment, unloading radius, rock excavation, order of dumping, coefficient of multiplicity of transshipment, economically permissible coefficient of reexcavation.

Контрольные вопросы:

1. Name the conditions for the use of technological complexes for transshipment of overburden rocks.

2. How is the simple transshipment of overburden carried out?

3. How is the multiple transshipment of overburden rocks carried out?

4. What coefficient is called the coefficient of the multiplicity of transshipment?

5. By what formula is the economically permissible coefficient of overexcavation determined?

Lecture 17

Topic: Methods of opening at the excavator-dump technological complex

Plan:

1. Opening of one and two flanking capital trenches.
2. Opening of one central and two flanking capital trenches during the development of two blocks.
3. Opening with three capital trenches.

For the movement of minerals, automobile and conveyor transport is more often used, less often – automobile-railway and railway. Conveyor plants can compete with road transport for moving soft minerals mainly at high quarry capacity. When transporting minerals from the faces by rail, it is difficult to organize work on the flanks of the quarry, where in this case the front of the work is bent or it is necessary to load with uncoupling of wagons, as well as with the laying of an exhibition dead end.

When using EO and ETR complexes in a quarry, it is common to open external individual (with one mining horizon) and group (with two mining horizons) trenches or internal trenches used for motor transport and located mainly on the end sides of the quarry.

The opening schemes are directly related to the number of EA equipment complexes in the quarry and the organization of their interaction with mining equipment complexes.

One or two interconnected complexes of stripping and mining equipment are usually operated within the quarry field. When using two complexes, the general front of work at the quarry is divided into blocks. At the same time, each block must have its own transport exit (one or more).

There are the following methods of opening mining horizons, interconnected with the organization of stripping and mining operations.

1. Opening of one flank capital trench during the development of rocks and minerals in one block (Fig. 24.1, a). The stripping complex follows ahead of the mining one with an advance, the value of which is regulated by safety requirements.
2. After working off each entry, the stripping and mining equipment returns to its original position..

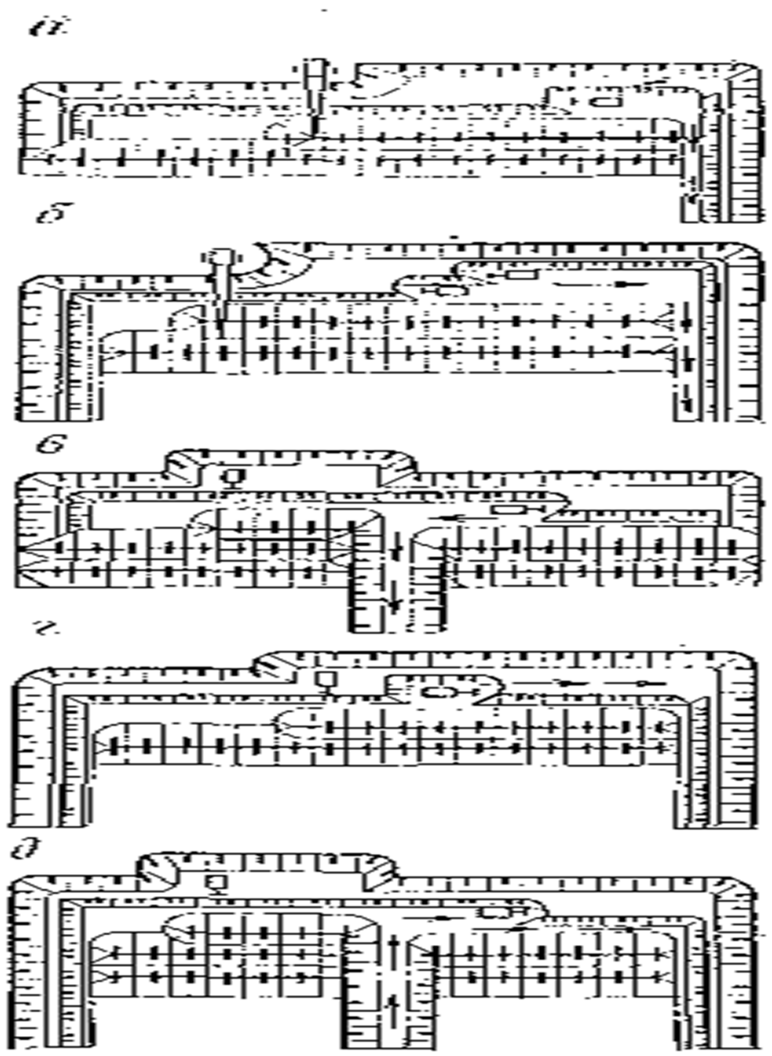


Fig. 24.1. Schemes of opening horizontal deposits when using EO complexes

1. Opening with two flanking capital trenches during single-block mining of the rock mass (Fig. 24.1, b). The extraction of minerals in this case can be carried out according to two options: the mining complex can follow the overburden or work ahead of it. With this scheme, the working stroke of excavators in both directions is possible.

3. Opening of one central capital trench during the development of two blocks (Fig. 24.1, c). Transshipment of overburden rocks is carried out alternately in both blocks. The reserves ready for excavation are limited by the opened and cleaned strip of the mineral for the width of the overburden and the full length of one block. The stripping and mining equipment returns to its original position at idle after each block has been worked off.

4. Opening with two flanking capital trenches during development with two blocks (Fig. 24.1, d). Mining and stripping operations are carried out simultaneously in different blocks. Overburden and mining equipment is distilled to the starting position by idling.

5. Opening with three capital trenches (two flanking and one central) during the development of two blocks (Fig. 24.1, d). This scheme provides for the possibility of continuous traffic and the working stroke of excavators in both directions.

Of the considered methods of autopsy, the last two are preferable. It is not recommended to use a scheme of opening one flank trench. Opening with two flank trenches in single-block development is used in conditions when the total length of the front is insufficient to divide it into two blocks and with a small production capacity of the enterprise.

Key words: the type of transport used, an interconnected complex, stripping and mining equipment, mining horizon, organization of work, flanking capital trench, central.

Security questions:

1. What is the reason for the autopsy scheme?
2. What are the methods of opening mining horizons?
3. How is the opening of one flank capital trench carried out during the development of one block?
4. How is the opening of two flank capital trenches carried out during the development of two blocks?
5. Under what conditions is the opening of two flank trenches used in single-block development?

Lecture 18

Topic: Technological complexes with cantilever dumpers

Plan:

1. General information about technological complexes with rock movement by dumpers.
2. Characteristics of technological complexes with cantilever dumpers.

Technological complexes with rock movement by dumpers are characterized by the presence and combination of three overburden processes - excavation, displacement and dumping. These complexes are mainly used in the development of two types of deposits:

with soft and dense overburden rocks and minerals;

with soft and dense rock and rocky (semi-horizontal) minerals.

When developing deposits of the first type, mining equipment complexes include rotary excavators with normal digging efforts and conveyor transport means. During the development of deposits of the second type, the extraction of the exploded mineral is carried out by mechlopats in combination with automobile, railway or conveyor transport; the rock mass enters the conveyors through a self-propelled crushing unit. When moving exploded rock overburden by cantilever dumpers in the process chain, a self-propelled crushing or crushing-screening unit with a feeder hopper is also required.

The development system is most often longitudinal single-board. With a fan system of development, the complex of stripping equipment, as a rule, should include an additional loader between excavators and a dumper. The annual advance of the work front can reach 300-350 m . Advantages of technological complexes with the movement of rock by conveyor dumpers: the possibility of transporting rocks to the dump along the shortest path, the continuity of the production process, the best use of the equipment complex in time (daily productivity with overburden excavators of the same capacity is 20-35% higher than when using railway transport), high labor productivity, simple organization of overburden work, no need for special works on rock dumps, except for reclamation, improvement of the stability conditions of the dump slopes due to the possibility of controlling the general angle of laying their system.

Technological complexes with the movement of rock by cantilever dumpers are used in areas with a relatively dry, warm climate or during seasonal overburden operations under conditions: good exploration and systematic drainage of the quarry field; horizontal or slightly inclined occurrence of layers or formation-like deposits with a slight change in the hypsometry of the soil and the roof of the formation to ensure acceptable slopes; smooth outlines of the contours of the quarry field, which avoids a sharp reduction and increase in the length of the mining front; significant mineral reserves in the contours of the quarry. Complexes with rotary and chain excavators, as a rule, are not effective if there are hard inclusions in the developed thickness of soft overburden rocks (in the form of boulders, interlayers, etc.), if it is impossible to work out strong rocks with a separate ledge using a cyclic excavation technique.

Characteristics of technological complexes with cantilever dumpers

The simplest technological complex of stripping and mining operations includes one rotary or chain multi-bucket excavator with a built-in non-rotating console equipped with a belt conveyor (Fig. 25.1). Soft overburden rocks and deposits with the use of such an excavator are worked out alternately. The rock moves through the unloading conveyor into the worked-out space (see Fig. 25.1, a). The mineral is loaded onto a conveyor or into wheeled vehicles, while the console is deployed at an angle of 25-30 ° to the front of the work (see Fig. 25.1. b). With the next and separate excavation of rock and minerals, the possible production capacity of the quarry decreases due to periodic mining operations, the organization of work becomes more complicated and the periodic use of transport is also conditioned. The use of the complex is possible in the development of non-watered deposits in conditions of relatively small (up to 20-30 m) total capacity of overburden rocks and mineral deposits.

The continuity of overburden and mining operations is achieved by dividing the complex into separate technological complexes of overburden and mining operations by using an additional excavator in the extraction (Fig.25.2, a).

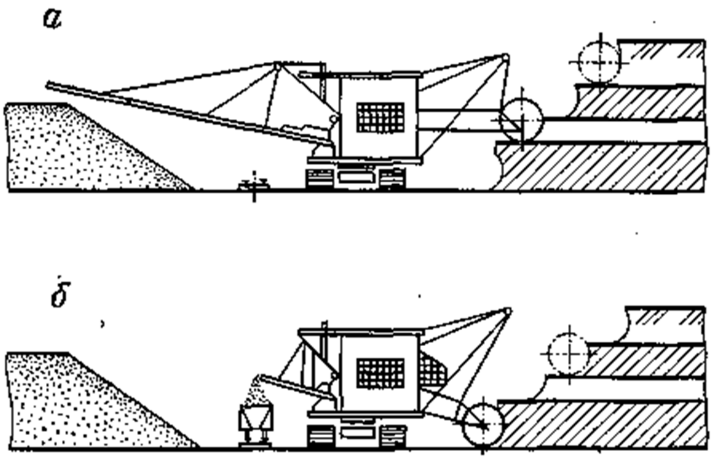


Fig. 25.1. The scheme of the technological complex when using one rotary excavator as a set of equipment in and the head machine of the VTR complex.

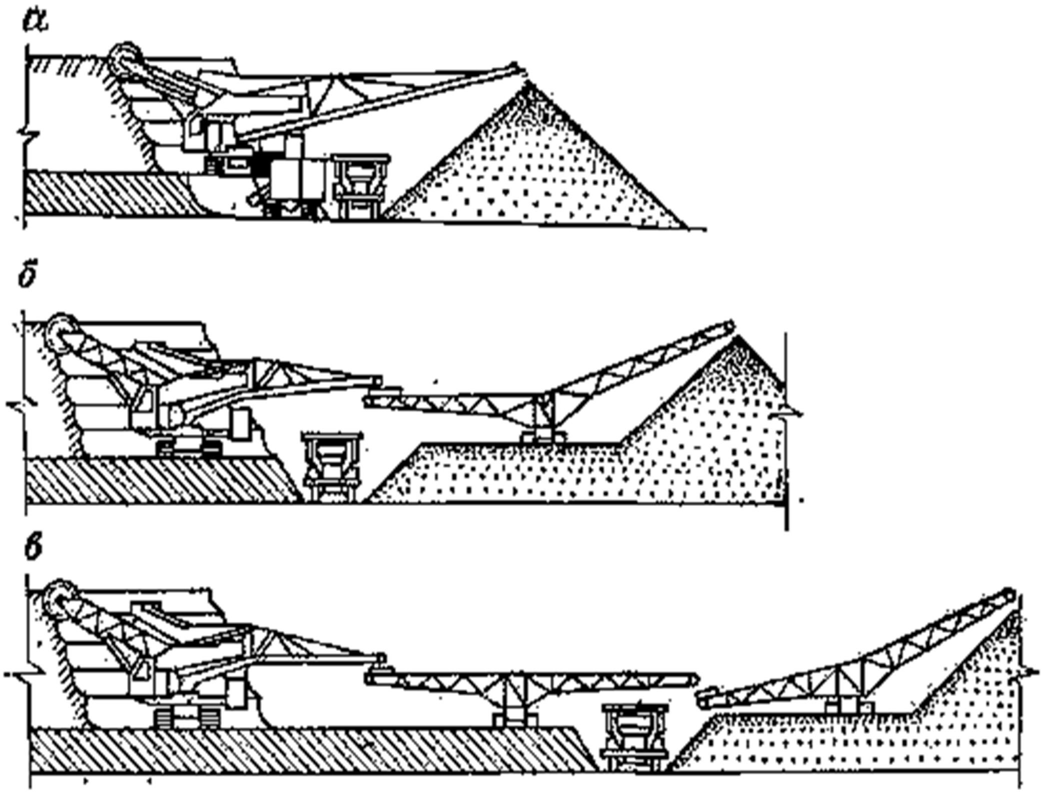


Fig. 25.2. Schemes of stripping technological complexes using multi-bucket excavators and cantilever dumpers.

A rotary excavator excavates and moves overburden rocks into the worked-out space, and a mining excavator loads minerals into vehicles. Transport communications are placed on the roof, soil or intermediate horizon of the deposit.

With an increase in the capacity of the deposit and overburden rocks, as well as if it is necessary to create significant uncovered mineral reserves, the length of the console built into the rotary excavator may be insufficient and a special cantilever dumper is included in the complex of overburden equipment, to which the rock comes directly from the excavator (Fig. 25.2, b) or through a conveyor loader (Fig. 25.2, in).

The excavation schemes differ in the installation location of the cantilever dumpers in the plan and profile of the quarry field, which determines the parameters of the elements of the development system, the composition of the equipment complex and the parameters of the dumpers themselves. The dumper can be installed on the roof of the mining ledge (Fig. 25.3, a), on an intermediate platform (Fig. 25.3, b) or on a rock (Fig. 25.3, c and d). Schemes with periodic changes in the installation location of the dumper are possible.

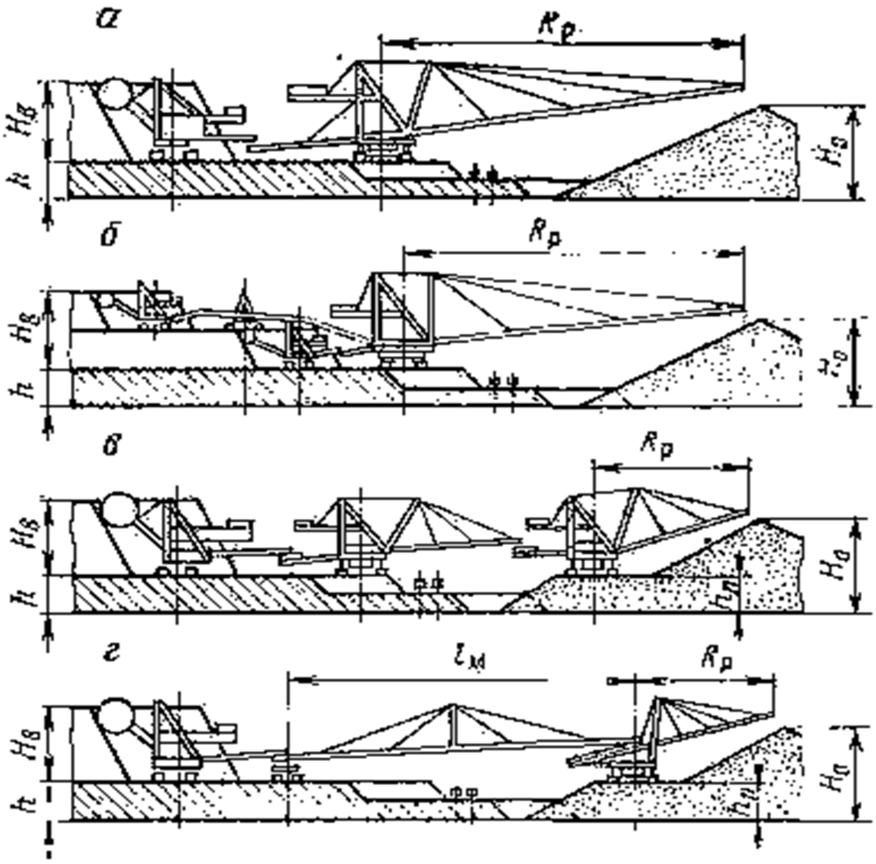


Fig. 25.3. Excavation schemes when using cantilever dumpers.

In the scheme of excavation with the installation of a dumper on the roof of the deposit (see Fig. 25.3, a), the rock is fed to the dumper by an excavator located on the same horizon. The dump is poured off without turning the dumper console of the dumper. The organization of the operation of stripping and mining equipment complexes (Fig. 25.4) is rigidly dependent.

With this scheme, it is rarely possible to create a backup mining approach under the dump console of the dumper, as a result of which the complexes of overburden and mining equipment must move one after another, working out the next overburden and mining approaches with the same width. Mining operations are ahead of stripping operations along the front, while idle transitions of excavation machines are required after working off the next stripping and mining approaches (see Fig. 25.4, a, b, c and d), otherwise large downtime of stripping and mining equipment complexes or one of them is inevitable. The need for such an organization of stripping and mining operations arises when developing a powerful deposit with two high mining ledges, even when using a powerful complex with a loader and a dumper during stripping operations.

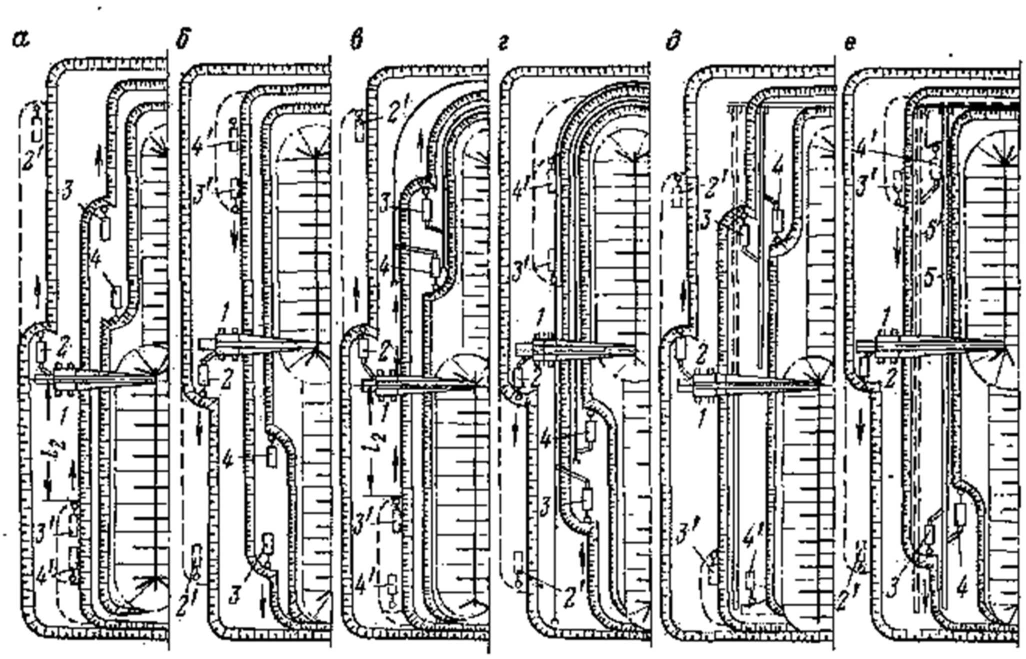


Fig. 25.4. Excavation schemes with the installation of a dumper on the roof of the deposit:

a and b – for EAR or VAR complexes; c and d – for VZHR or EJR complexes; d and e – for WRC complexes (a, b and d – when working off approaches in the direction of the capital trench; b, g and e – when working off approaches in the opposite direction); 1 – dumpers; 2 and 2' – overburden rotary excavators; 3 and 3' – mining rotary excavators of upper scooping; 4 and 4' – mining rotary excavators of lower scooping; 5 and 5' – positions of downhole conveyors.

The connection between stripping and mining operations is less rigid when using EAR complexes, when there are no difficulties with the organization of its work at any position of the equipment complex in (see Fig. 25.4, a and b). When using the VZHR or EJR complexes, as the stripping complex moves along the front of the ledge, it is necessary to transfer the curved section of the path (see Fig. 25.4, c and d).

If there is a backup approach, overburden and mining complexes can excavate in different directions; idle transitions of equipment are excluded, bending of downhole railway tracks is not required, conveyors can be used when transporting minerals (Fig. 25.4, d and e) and the organization of work is simplified. However, to create a backup approach, it is necessary to increase the unloading radius of the dumper by an amount equal to the width of the mining approach.

The insertion of a rotary excavator into a new entry is advisable on the end section opposite to the junction of the capital trench, since this facilitates the placement of overburden rocks extracted during insertion in the inner dump and allows the use of a dumper with a shorter (by 8-10%) console length. The working stroke of the equipment complexes in the direction of the transport exit from the mining ledge also allows, when moving minerals by conveyors, to move conveyor staves in advance within the spent part of the entry.

With the scheme of excavation with the installation of a dumper on different horizons with an excavator (see Fig. 25.3, b), the presence of a loader in the complex is mandatory. With the use of this scheme, it becomes possible to alternately work out the upper and lower overburden approaches with one rotary excavator, while the loader is used only when working out the upper overburden approach. All the main technological provisions specified for the first scheme of the dumper installation also apply to this scheme.

With the scheme of excavation with the installation of a dumper on the dump, its unloading radius is reduced. The rock intake from the excavator is carried out by a loader (see Fig. 25.3, c) or a connecting bridge (see Fig. 25.3, d). The mutual connection of stripping and mining operations is similar to the one described above. The dumping of the blade is usually carried out with the rotation of the console, which leads to the leveling of the surface of the blade and a decrease in its height due to the absence of "ridges".

With the scheme of excavation with a change in the standing position of the dumper when filling a multi-tiered dump (Fig. 25.5), the linear dimensions of the rotary excavator decrease, since the overburden ledge of great height is worked out by two or three steps. Such a scheme is possible when using powerful HE complexes (with a passport equipment capacity of 5-10 thousand m3/h or more) for mining a powerful (up to 70-90 m) thickness of soft overburden rocks with a small (less than 10 m) mineral deposit capacity. When working out two or three approaches with one rotary excavator, the dumper is sequentially installed on the roof and soil of the deposit (see Fig. 25.5, a and b) and the rock or on the intermediate overburden horizon, the roof of the deposit and the rock.

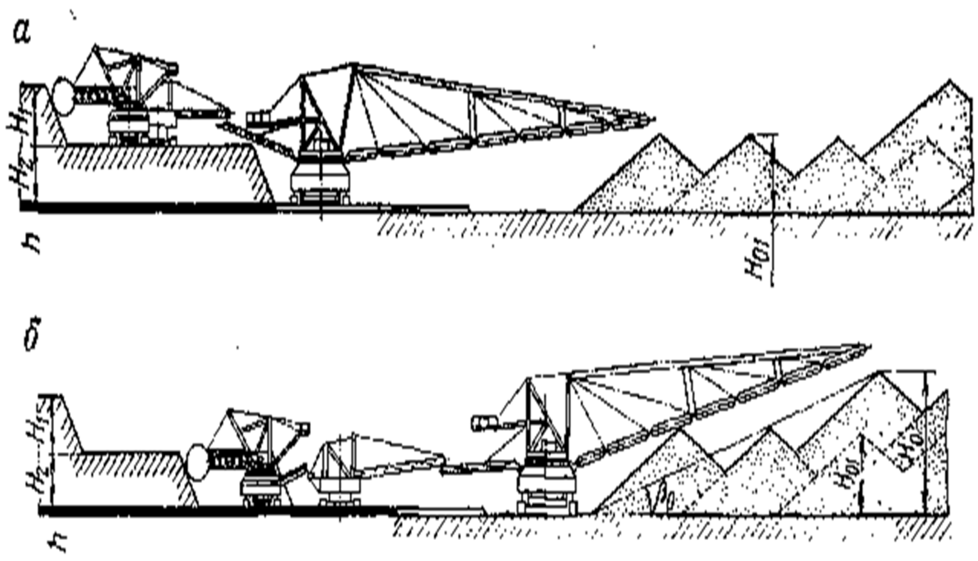


Fig. 25.5. Two-stage scheme of excavation with a change in the standing horizon of the dumper.

Key words: excavation, displacement, dumping, rotary excavators, chain excavators, development system, shortest path, continuity, use in time, length of the mining front, head machine, excavation scheme, roof of the deposit, standing place.

Security questions:

1. What are the characteristics of technological complexes with rock movement by dumpers?

2. In the development of which deposits are dumpers used?

3. Specify the advantages of technological complexes with the movement of rock by conveyor dumpers.

4. What is the difference between the excavation schemes when using a dumper?

5. How does the excavation take place when installing the dumper on the roof of the deposit?

Lecture 19

Plan:

1. General information about transport technological complexes.
2. Technological complexes with conveyor movement of rock mass.

General provisions

Transport technological complexes are used in the development of horizontal and shallow deposits of any capacity. With continuous development systems, these complexes are typical for the development of the upper part of a powerful overburden layer in horizontal deposits (with the creation of advanced ledges).

The costs of excavation and loading operations, movement and dumping during the development of soft, dense and heterogeneous rocks are usually characterized by a ratio of 4:4:2. Therefore, the cost-effectiveness of development depends simultaneously on the means of excavation used, the type of transport and the distance of movement of the rock mass, primarily overburden rocks.

For transport technological complexes, separate execution of the processes of excavation, loading, as well as transportation of rock mass along the work front of the ledges is mandatory.

As a rule, the process of dumping is also carried out separately.

To reduce the distance of intra-quarry transportation with large sizes of quarry fields, the following can be used: a transverse single-board development system; a longitudinal single-board development system with a double work front of ledges with one or two transport exits;

a longitudinal development system with a built-up work front of ledges with three transport exits.

The transverse system is used in the development of horizontal deposits using EAO equipment complexes, sometimes EKO complexes.

A double front with two flanking transport exits is widespread in internal dumping (Fig. 17.1, a), the length of the work front is 3-4 km or more and the use of railway and conveyor transport. The opening of one ledge with two temporary exits is used during the operation of EAO complexes, when the length of the front of the ledges worked out with the movement of rock into external dumps (usually dispersed) is 1.5—2 km or more (Fig. 17.1, b).

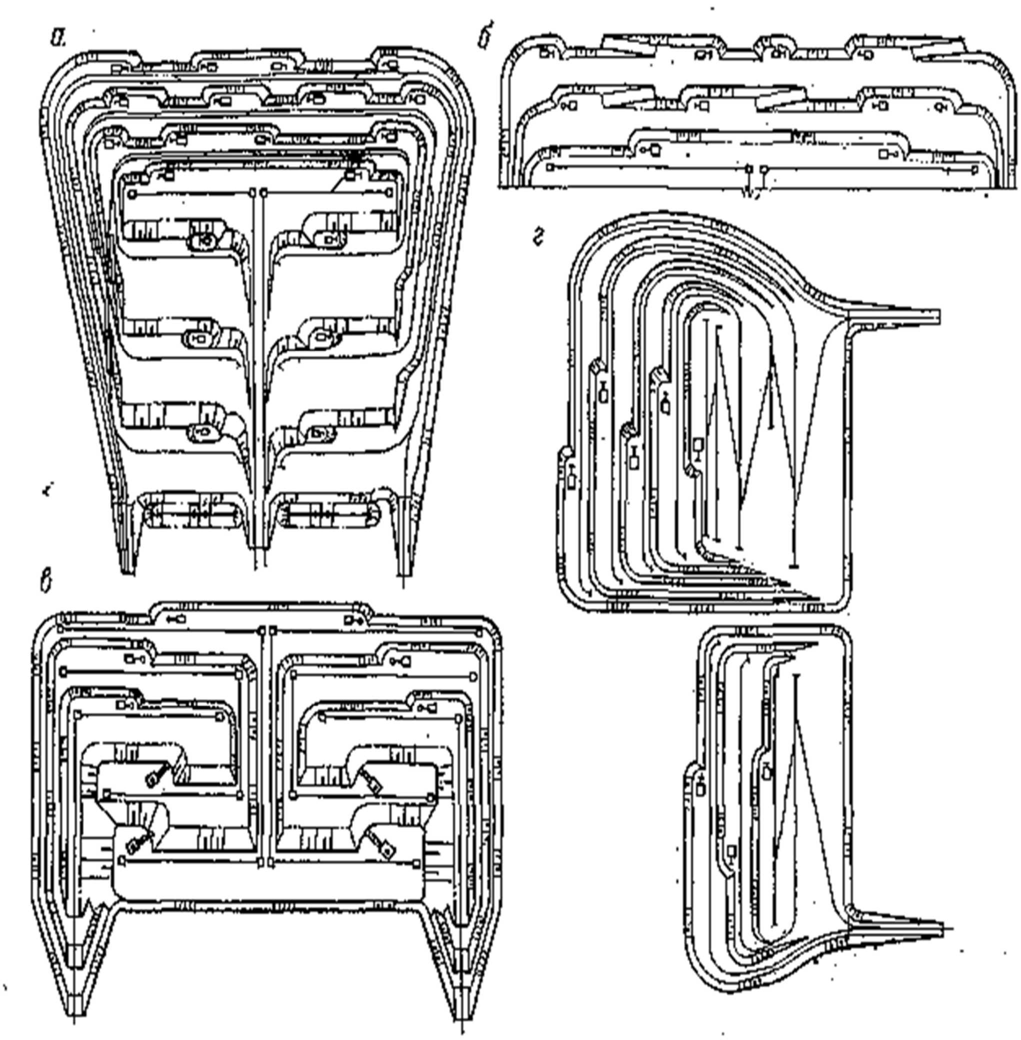


Fig. 17.1. Construction of the work front of the ledges when using the complexes THIS and the WTO.

The built-up front of overburden ledges during internal dumping necessitates the abandonment of temporary mineral pillars and overburden dams to the soil of the ledge in question (Fig. 18.1, c). Such a design of the front may be appropriate for small capacities of horizontal deposits and overburden rocks when using equipment complexes of EAO or East Kazakhstan Region.

The abandonment of a temporary or permanent rock bridge between individual sections of the quarry field along the strike of the deposit is characteristic when they are alternately put into development with a large time interval (Fig. 17.1, d). When developing shallow deposits in horizontal layers, as the work front moves forward due to an increase in the overburden capacity, the width of the rock bridge and the volume of the whole mineral are constantly increasing, and the front of the internal dumps is reduced; therefore, the constructed design of the front, as well as the advanced development of individual sections of the quarry field, are in most cases ineffective in these conditions.

With external dumping, several temporary exit routes are possible when EAO complexes are operating, usually only on the upper horizons when developing shallow deposits.

The movement of rocks by transport along the work front does not limit the height of the working area of the quarry and the capacity of the overburden rocks being worked out. Therefore, the parameters of development systems, including the volumes of discovered mineral reserves, depend on the working dimensions of the equipment used to a lesser extent than when using the EO and HE complexes.

Technological complexes with conveyor movement of rock mass

The rational distance of moving rocks by conveyors during the development of soft and medium-dense rocks in high-capacity quarries reaches 6-8 km. In such conditions, conveyor transport is quite competitive with railway transport in terms of costs related to 1 m3 of transported rock.

The length of conveyor lines and the number of overloads are minimal when developing one overburden ledge of an elongated quarry field with the movement of rock into the inner dump and the same speeds of movement of the overburden and dump work fronts (Fig. 17.2, a). The rotary excavator 1 loads the rock onto the downhole conveyor 2 directly or through the downhole loader. In the latter case, the step of moving the bottom-hole conveyors increases (the width of the panel) and the conditions for working off dead ends and tapping into a new overburden are facilitated. Next, the rock enters the transfer conveyor 3, mounted on a connecting berm at the end of the quarry, from which it is delivered directly or through an inter-stage loader to the dump conveyor 4 and the cantilever dumper 5.

Under similar conditions, in the case of external dumping (Fig. 18.2, b), the rock is fed from the transfer conveyor 3 through the inter-stage loader 6 to the connecting conveyor 7 located on the surface, and then transported along the dump conveyor 4 to the dumper 5. Cantilever dumpers or double-support conveyor bridges can be used as inter-stage loaders.

In the complex (see Fig. 17.2, a), downhole and dump conveyors move simultaneously, and in the complex also a connecting conveyor on the surface. Therefore, with the complex shown in Fig. 17.2, b, the volume of auxiliary work is very large; despite the savings obtained by reducing capital costs for the installation of conveyors, the operating costs for additional movement of them increase and the productivity of powerful equipment decreases due to downtime.

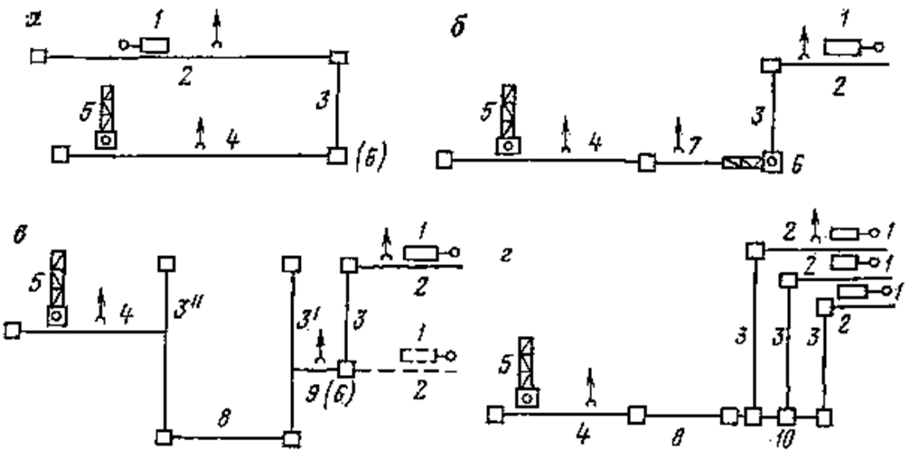


Fig. 17.2. Schemes of transportation of overburden rocks by conveyors.

With external dumping, in cases of unequal speeds of movement of the overburden and dump work fronts, different directions of their development, as well as to reduce the volume of movement with a significant length of connecting conveyors on the surface, instead of them, the complex includes (Fig. 17.2, c) a horizontal main conveyor 8, a transfer conveyor 3" mounted at the end of the dump on the roof the lower dump ledge, and the transfer conveyor 3' on the surface at the end contour of the quarry. Instead of inter-stage reloaders in the quarry and on the dump, it is rational to use inclined main conveyors 9.

When developing a thick layer of covering soft rocks with several ledges, the complex includes (Fig. 17.2, d) an assembly inclined main conveyor 10, from which the rock enters the horizontal main conveyor 8.

During internal dumping, the grouping of cargo flows of the same (at the place of unloading) rocks is usually carried out by installing common transfer (Fig. 17.3, a and b) or downhole (Fig. 30.3, c) conveyors. When moving rocks to various unloading points, it is necessary to maintain elementary cargo flows and have several downhole, transfer and dump conveyor lines. For these reasons, the number of downhole conveyor lines may be less and more than the number of serviced working horizons or equal to it (Fig. 17.4).

Thus, the complex of equipment can include: downhole, transfer, dump, trunk, inclined and horizontal conveyors, downhole and inter-stage reloaders. The movement of conveyor lines is usually carried out by turnodozers. Downhole conveyors are equipped with self—propelled loading bins, and dump conveyors are equipped with self-propelled unloading trolleys. Individual designs of transfer conveyors have telescopicity, which reduces downtime and ensures the independence of the movement of adjacent conveyors.

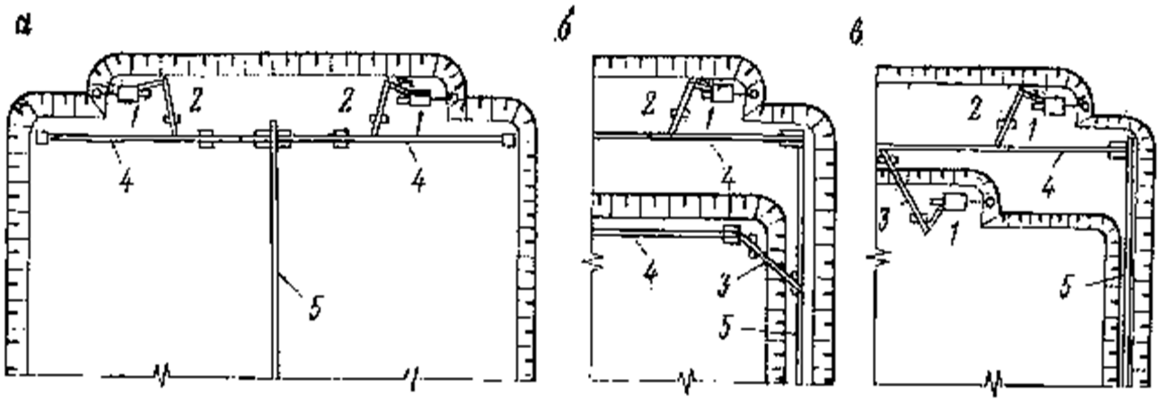


Fig. 17.3 Schemes of grouping of cargo flows in conveyor transport:

1 — rotary excavator; 2 and 3 — downhole and inter—stage reloaders, respectively; 4 and 5 - downhole and transfer conveyors, respectively.

When moving overburden rocks by conveyors into internal dumps and the presence of elementary cargo flows, in case of equality of the marks of the dumping horizons and the working sites of overburden ledges, the installation of additional dumpers or inter-stage loaders is excluded.

Grouping of cargo flows, and therefore horizons, allows for their maintenance to use one downhole, transfer and dump conveyors (see Fig. 17.4) or two downhole and one transfer and dump conveyors (see Fig. 17.3, b). With these schemes of excavation, both capital costs for downhole and transfer conveyors and operating costs, including their movement, are reduced; the number of horizons decreases and the height of the ledges of internal dumps increases. The disadvantage of these schemes is the presence of inter-stage reloaders.

Excavation schemes involving the development of high overburden ledges are also used. The ledge is divided into sub-steps, which are worked out using a single set of continuous equipment, while the linear parameters of rotary excavators, their weight and cost are reduced. The downhole conveyor in such a technological complex is located on the roof of the lower approach. After working out the entry of the lower approach within the entire or main part of the front, the rotary excavator arranges a ramp with a slope of up to 5 ° (respectively at the end of the quarry or within the remaining part of the front of the lower approach) and goes to the upper platform of the approach; the loader is located on the lower platform of the lower approach. Then the excavator fulfills the approach on the upper approach, idles back to its beginning, descends along the exit to the working area of the lower approach and performs the exit, after which it follows to the place of insertion into the new entry of the lower approach and begins a new technological cycle of development.

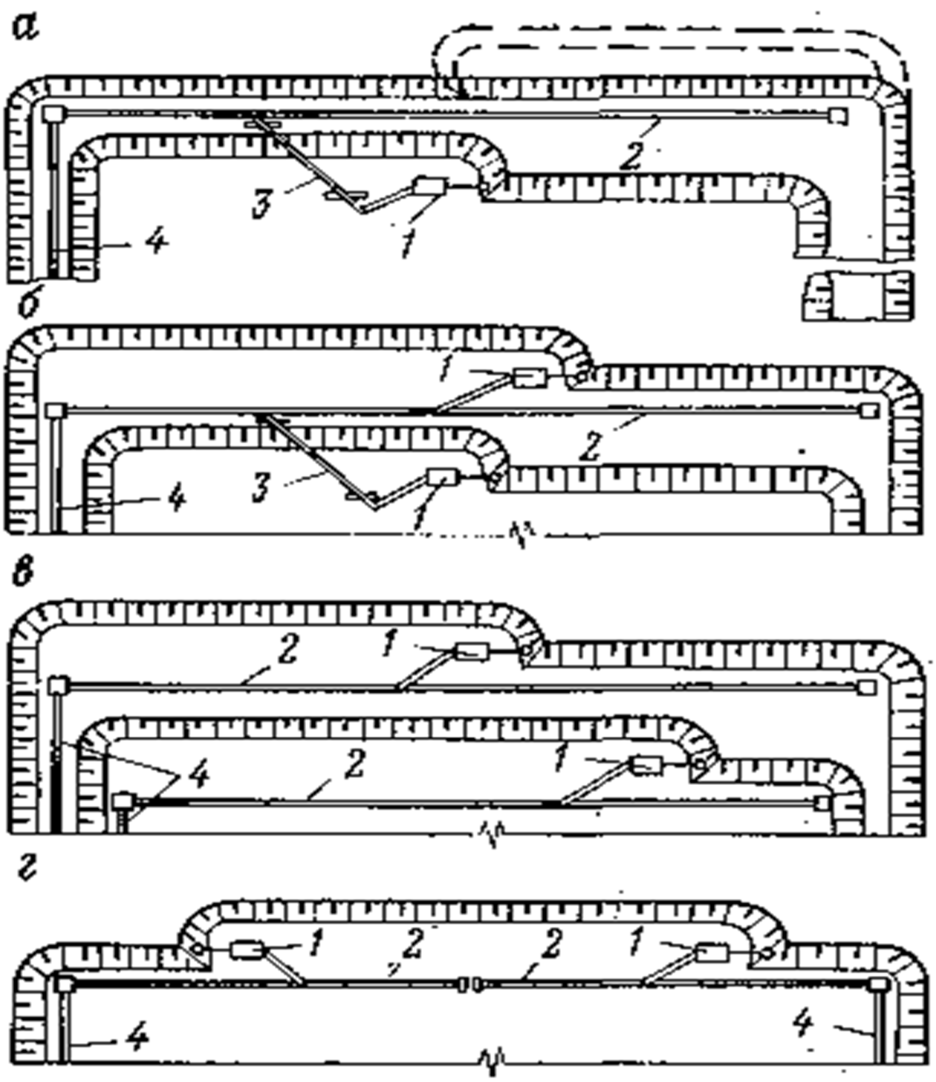


Fig. 17.4. Schemes of excavation when using complexes of East Kazakhstan region:

a — with the development of a ledge with two approaches by one rotary excavator; b - with the development of approaches by separate excavators with a common downhole conveyor; c - with the development of separate ledges without grouping cargo flows; d — with two excavators and downhole conveyors on the ledge; 1 — rotary excavators; 2 — downhole conveyors; 3 — inter-stage loaders; 4 —transfer conveyor.

Reference words: horizontal and shallow deposits, costs, excavation, loading, dumping, transverse, longitudinal, EAO, East Kazakhstan region, double front, THIS, WTO, travel distance, length of conveyor lines, transportation scheme, downhole, dump, trunk, transfer, grouping of cargo flows.

Security questions:

1. In the development of which deposits are used transport technological complexes?
2. What development systems are used to reduce the distance of intra-barrier transportation?
3. In what cases is the length of conveyor lines minimal?
4. Describe the schemes of transportation of overburden by conveyors.
5. How is the grouping of cargo flows of the same rocks carried out during internal dumping?

Lecture 20

Topic: Technological complexes when moving rock mass by motor transport with continuous development systems

Plan:

1. Conditions for the use of technological complexes with the movement of rock mass by motor transport.
2. Autopsy schemes when using motor transport.

Technological complexes with the movement of rock mass by motor transport are widely used in the development of sand-gravel and carbonate deposits, as well as in the development of horizontal and shallow ore deposits and coal seams of limited size and irregular configuration or with relatively stable parameters of occurrence, but uneven ore quality. With long distances of transportation to the consumer, it is characteristic to use automobile and railway transport with the device of transshipment points on the surface or at the end of the quarry in front of the capital trench.

The development system is transverse (Fig. 18.1), longitudinal (Fig. 18.2), transverse-longitudinal or radial with an irregular configuration of the front and uneven movement of its individual sections. Dumping is internal, external or combined. When developing relatively powerful horizontal deposits, the dumping of internal dumps begins after the formation of several mining ledges and reaching the soil of the deposit (see Fig. 18.1, a and b).

In case of alternate development of dispersed small deposits, which are sections of the same quarry field or nearby quarries, it is advisable to place overburden rocks within the spent areas or quarries in order to reduce the size of the land allotment and reduce the distance of transportation.

For motor transport, a single-sided longitudinal system of overburden development is possible in shallow deposits with split trenches in contact with the hanging side of the deposit (see Fig. 32.2), and a transverse development system is used for the extraction of minerals.

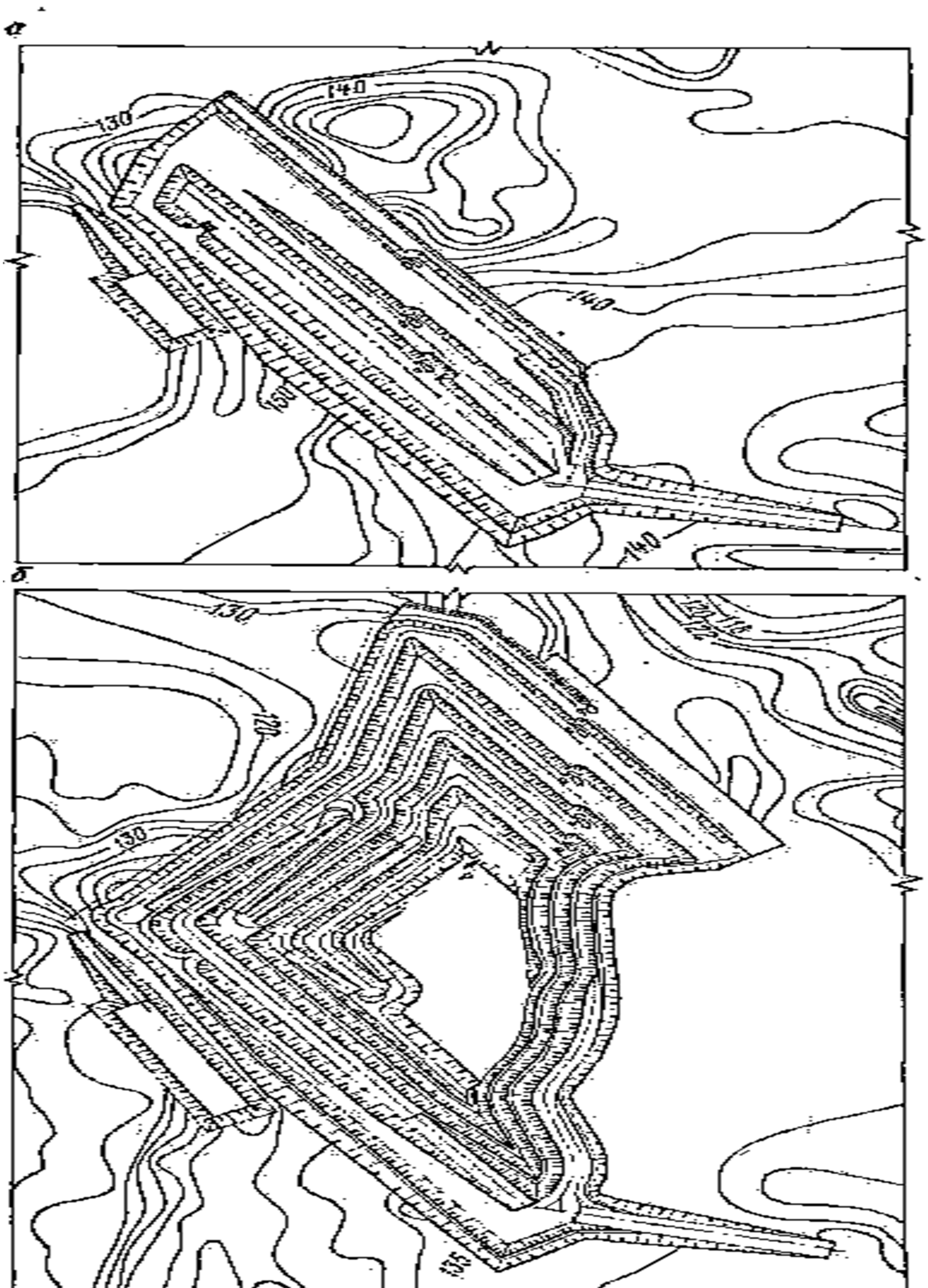


Fig. 18.1. Design schemes for the development of mining operations in a sand and gravel quarry:

a — when the quarry is put into operation;

b — for the 4th year of operation.

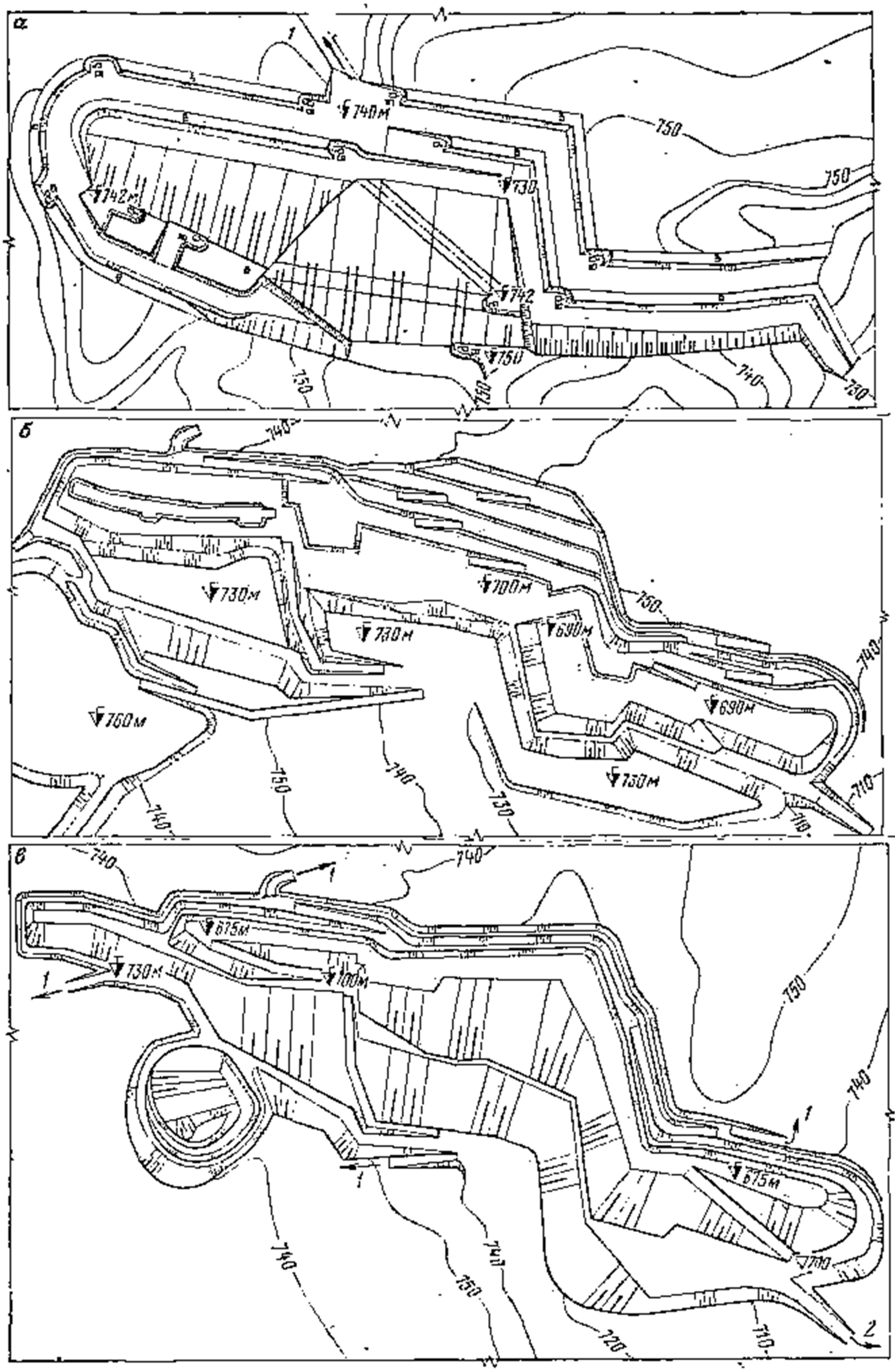


Fig. 18.2. Design schemes for the development of mining operations at the quarry:
a, b and c, respectively, when the quarry is put into operation, for the 5th year of operation, at the end of mining; 1 — for the dump; 2 - for the industrial site.

The autopsy schemes in the technological complexes under consideration are characterized by a wide variety. As a rule, one or two upper horizons are opened by an external trench on the flank from the side of the non-working side of the quarry (see Fig. 18.1, 18.2). With a relatively large number of ledges (four to five or more) and limited quarry sizes in terms of the development of horizontal deposits, the route of permanent or semi-stationary internal exits is usually looped and located on one or two non-working sides of the quarry, changing until the end of the deepening of mining operations (see Fig. 18.1, a and b). When developing elongated shallow deposits, overburden horizons are opened by one or two systems of temporary exits along the working side of the quarry (see Fig. 18.2) with the transportation of rocks to dispersed external dumps; the shape of the routes of such exits is simple or looped, depending on the number of routes, the length of the work front and the number of horizons (see Fig. 18.2, a, b and c). Working horizons during the development of shallow deposits can also be opened by a system of internal exits along the non-working side of the quarry in the absence of internal dumps (see Fig. 18.2, b and c). With the device of the exits, both mining and lower overburden horizons are opened; their number and position in the plan and the shape of the route depend on the angle of fall of the deposit.

The width of the approaches and work sites, the height of the ledges, the speed of moving the front of the work, the productivity of the complexes are calculated in the same way as with the development deepening systems.

The technological complex of layer-by-layer mining is also used in the development of elongated steep deposits of large extent (Fig. 32.3). Within the layer, a continuous transverse development system is used with advanced split trenches on the mining horizons. The layer is divided into several ledges. Overburden rocks are moved by motor transport to external dumps. The opening of the working ledges is carried out by a system of semi-stationary internal exits.

Minimum width (m) of the stripping panel on the lower stripping horizon of the layer

$Ш_п = H_у(\text{ctg } \beta + \text{ctg } \alpha) + b_п$,

where $H_у$ — ledge height, м; β — the angle of incidence of the formation, degree; α — slope angle of the ledge, degree; $b_п$ — width of the safety berm, м.

On the overlying horizons within the mining zone, the width of the overburden panels increases (with each horizon by the value of Shp). Within the stripping zones, the width of the panels remains unchanged (see Fig. 32.3). The use of such a technological complex in favorable conditions makes it possible to reduce the volume of mining and capital works and the current stripping coefficient at the beginning of the field operation.

With a transverse single-sided system for the development of elongated steep deposits, a technological complex with internal dumping, characteristic of continuous development systems, is also used. The main part of the overburden (after working off part of the quarry field - the first stage quarry with external dumping) can be moved to internal dumps by motor transport or sometimes by conveyors. The quarry of the first stage deepens to the final design mark.

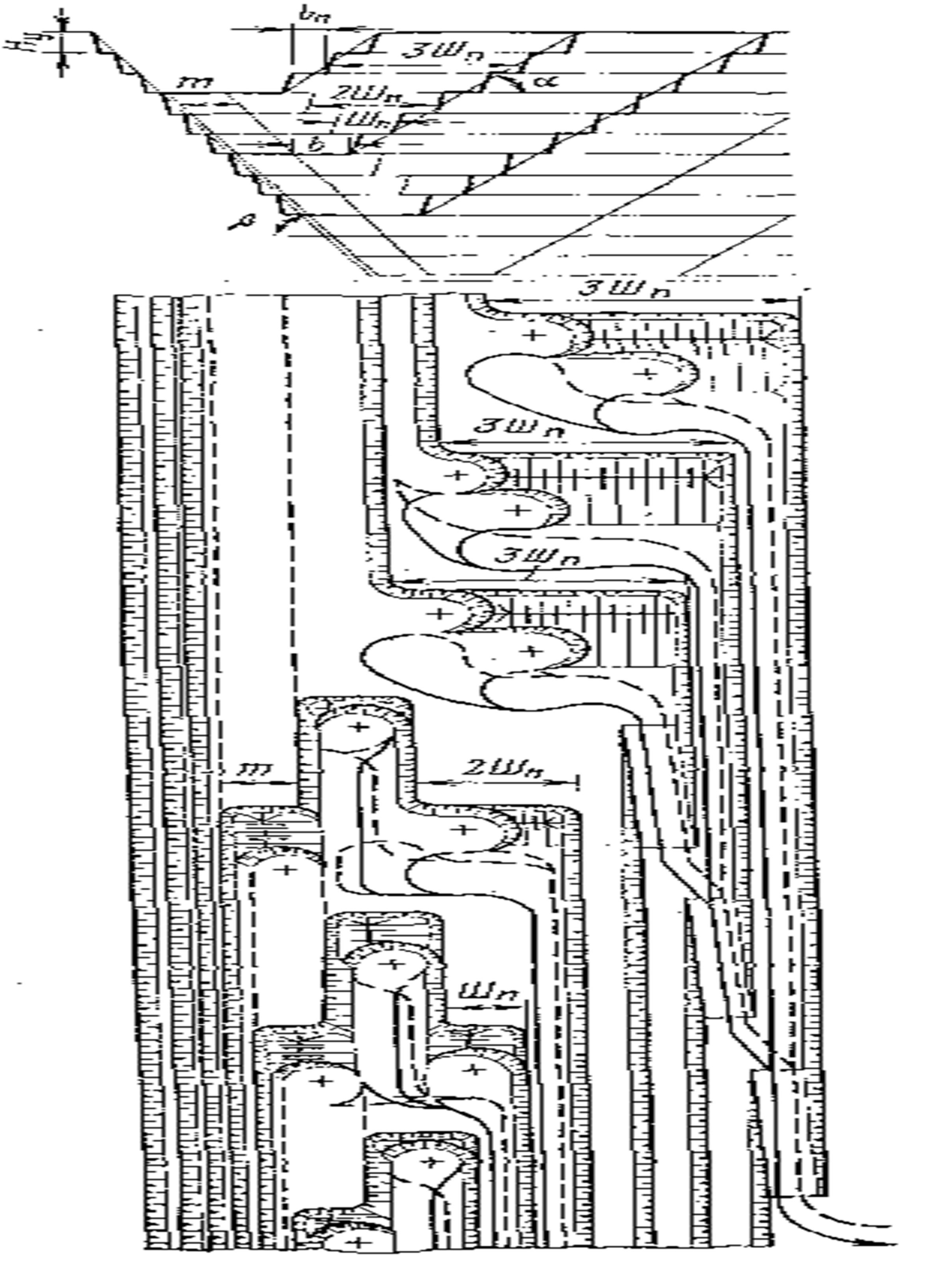


Fig. 18.3. Scheme of layer-by-layer mining of a steep deposit

As the internal dumps form and the overburden front moves along the strike, the front moves accordingly

dump works. The ledges are worked out simultaneously on all horizons of the quarry (Fig. 18.4). Overburden rocks are transported by dump trucks to high-altitude dumps on transport farms.

At the same time, the distance of transportation is reduced, the movement of vehicles occurs without lifting, cargo flows are dispersed and the productivity of dump trucks increases significantly compared to transportation to external dumps. The mineral is transported to the surface via internal semi-stationary ramps on board the quarry from the hanging side of the deposit. As the work front moves forward, the congresses are alternately covered with the rock of the internal dump of the corresponding horizon. By the time the congress is liquidated, a new congress (semi-trench) should be prepared on the same horizon. The work front can be through or dead-end.

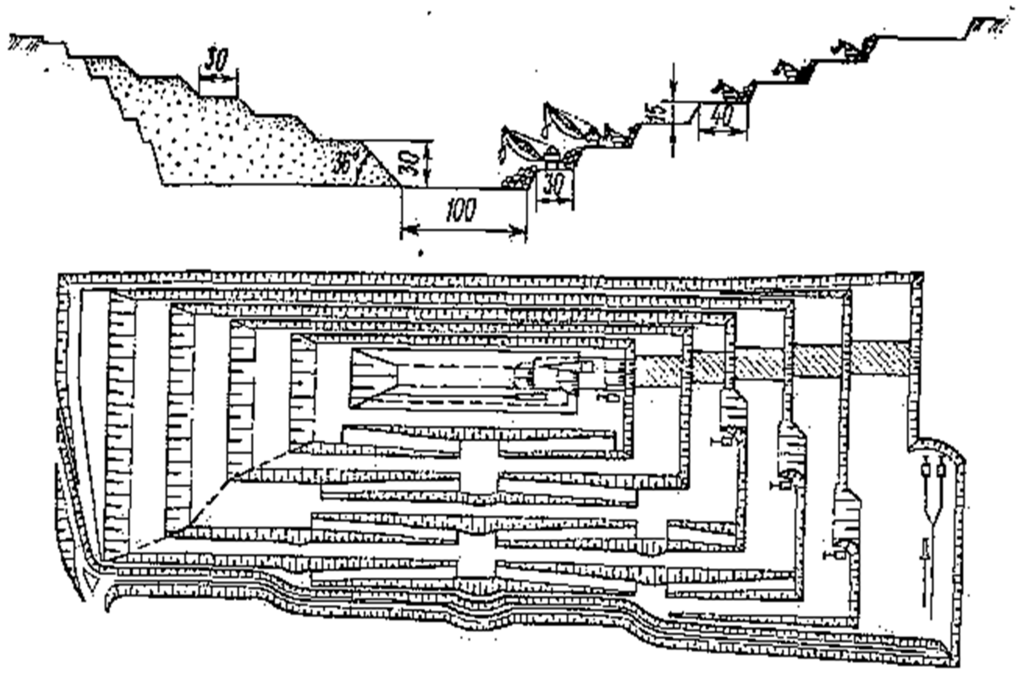


Fig. 18.4. Scheme of development of steep deposits with internal dumping.

When moving the front of the work along the strike of the deposit, the extraction of minerals and stripping work on the ledges alternately alternate and are carried out by the same excavators. The uncovered mineral reserves are provided with a uniform advance by overburden mining operations on all horizons. Required advance (m) by overburden on each ledge

$B=Q_н/[(H_к-H_н)m\gamma_и\eta_и]$,

where $Q_н$ — regulatory reserves, т; $H_к$ — career depth, м; $H_н$ — sediment capacity, м; m — horizontal capacity of the formation being opened, м; $\gamma_и$ — mineral density, т/м^3; $\eta_и$ — mineral extraction coefficient.

The required volume of reserves ready for excavation on each ledge is created when mining operations are ahead of mining operations by one or two 15-30 m wide approaches. With simultaneous work on all horizons, the deposits are developed with a more uniform distribution of overburden volumes over time.

Complete placement of the rock in the internal dumps is possible under the condition

$К_{ср}=1/[(К_{р.о}-1)\gamma_п]$,

where $К_{ср}$ — average stripping ratio, м^3/т; $К_{р.о}$ — residual coefficient of loosening of the rock in the dump (for semi-horizontal rocks at high dumps $К_{р.о}$= 1,08÷1,15); $\gamma_п$ — rock density, т/м^3.

To prevent landslides of internal dumps, sediments represented by moistened clays must be transported to external dumps or stored on the upper tier of internal dumps. The total slope angle of the internal dumps (at the height of the dump tier of 15 m) usually does not exceed 17-18 °.

The use of this technological complex is advisable in the development of inclined and steep deposits to full depth (synclinal folds and muldoobraznye deposits with a relatively small depth of the lock parts, individual sections of formations cut off in depth by disjunctive disturbances), as well as in the development of the upper horizons of deposits developed underground, and in quarries subject to reconstruction, where the use of external dumps for one reason or another are uneconomical or impossible.

Reference words: sand-gravel and carbonate deposits, horizontal, flat, transverse, longitudinal, radial, alternate development, opening scheme, width of approaches and work sites, technological complex, overlying horizon, formation, placement of rock in internal dumps.

Security questions:

1. In the development of which rocks are technological complexes used with the movement of rock mass by motor transport?
2. In the development of which deposits is the technological complex of layer-by-layer mining used?
3. How is the minimum width of the stripping panel determined on the lower stripping horizon?
4. What kind of technological complex is used for the transverse single-side system of development of elongated steep deposits?
5. How is the required advance for stripping on each ledge determined?

Lecture 21

Topic: In-depth development system. conditions for the use of in-depth development systems.

Plan:

1. **The shape and structure of deposits.**
2. **prevailing types and power of rocks.**
3. **Water availability and temperature regime.**
4. **Surface relief.**
5. **The shape and size of quarries.**
6. **Production conditions and mining volumes**.

The shape and structure of deposits. Formations, formation-like deposits and formations of formations are characteristic of coal, iron ore, apatite and phosphorite, copper ore and other deposits.

Isometric deposits, mainly of massive and stockwork types, are characteristic of many deposits of non-ferrous metal ores, ferruginous quartzite, chrysotile asbestos, etc. Tube-shaped deposits are characteristic of diamond deposits. Deposits of transitional forms are also being developed.

Most of the formation-like deposits have clear contacts, but uneven quality both in individual deposits and within the same deposit in depth and in plan. Many deposits, primarily of the stockwork type (ores of non-ferrous metals, chrysotile-asbestos, a number of ores of chemical raw materials, etc.), are complex-structured, characterized by the absence of clear contacts of deposits, numerous inclusions of waste rocks, the presence of several (up to a dozen or more) ore bodies of complex shape, uneven ore quality in areas, the distance between which is measured in several meters, etc. In general, inclined and steep deposits are characterized by numerous geological disturbances that cause changes in the spatial position, shape and size of deposits, as well as the quality of minerals.

The prevailing types and power of rocks. In all inclined and steep deposits, overburden rocks are primarily sediments covering deposits, containing rocks, interlayers and inclusions. In coal deposits, the host rocks are usually semi-rock and rock (of the first and second classes according to the difficulty of development), and the coal itself is a dense or semi-rock. Many ore deposits are characterized by metamorphosed, sedimentary and igneous rock host rocks and minerals with a wide range of changes in the indicator of the difficulty of developing rocks (Ptr varies from 4-5 to 20 or more). Frozen semi-basement and rocky (permafrost) host rocks and minerals are typical for deposits in the northern and northeastern regions.

*The usual capacity of coal seams varies from several to tens of meters; such a power range is also characteristic of formation-like deposits of non-ferrous metal ores, mineral chemical raw materials, chrysotile asbestos, etc. The capacity of iron ore deposits varies from tens to hundreds of meters.*Характерным является:

simultaneous development of breeds with different indicators of Ptr, differing by 3-5 categories or more;

increasing the difficulty of developing rocks with the deepening of the quarry due to the increase in strength and reduction of fracturing of rocks, even of one mineralogical composition.

The thickness of the covering rocks (mainly quaternary deposits) is usually small (from a few meters to 30— 40 m). At the same time, deposits with a capacity of covering rocks up to 100 and even 150 m are being involved in open-pit mining on an increasing scale. The covering rocks in such deposits are soft, dense, heterogeneous and semi-oval.

Water content and temperature conditions. Deposits of deep and high-altitude-deep types, as a rule, are flooded (from one to six aquifers). The negative temperature regime of permafrost soft, dense and semi-permafrost rocks with a clay skeleton adversely affects the performance of technological processes and ensuring the stability of the slopes of individual ledges in the summer.

The relief of the surface. The choice of technological solutions (mainly for opening, layout of the master plan) is significantly influenced by the hilly terrain and especially by the complex relief of the surface of high—altitude deposits. At the same time, it affects the avalanche risk and stability of dumps, the location of processing plants and dumps depends on it, and consequently, the distance of transportation of minerals and overburden rocks, and is also an initial factor in choosing the order of development of upland deposits, complexes of overburden and mining equipment, the location of receiving points of rock mass.

With the development of mining operations in upland quarries, the surface relief also changes, which determines the expediency in some cases of changing overburden and mining technological complexes.

The shape and size of quarries. The final shape and dimensions in terms of a deep-type quarry are determined by its Nc depth, the angles of laying non-working sides of the un and the size of the deposit at the bottom level. The size of the quarry field may be limited by: the presence of areas where the deposit capacity is less than permissible, or areas with non-industrial content of useful components; the presence of natural or artificial barriers; a large distance between individual deposits of the deposit.

The shape of the surface contour of deep quarries is usually rounded, regardless of the shape of the deposit in the plan. At the same time, the shape and size of the contour of each horizon and the quarry as a whole in the initial period of field development are determined by the shape and size of the deposit and the development system used (Fig. 19.1) and to a lesser extent by the size and shape of the final contours of individual horizons and the quarry field as a whole.

Production conditions and mining volumes. In order to systematically deepen mining operations at a certain speed, it is required to move the work front on all exposed ledges at an appropriate speed. With any mining system in a deep-type quarry, the length of the work front of each overlying ledge is greater than the underlying one, as well as the final dimensions of the ledges. Therefore, large volumes of stripping work should be carried out on the upper horizons and their working time is longer than the underlying horizons. At the same time, new ledges are cut, the total number of working ledges increases for a long time. In this regard, the volume of stripping works is also increasing.

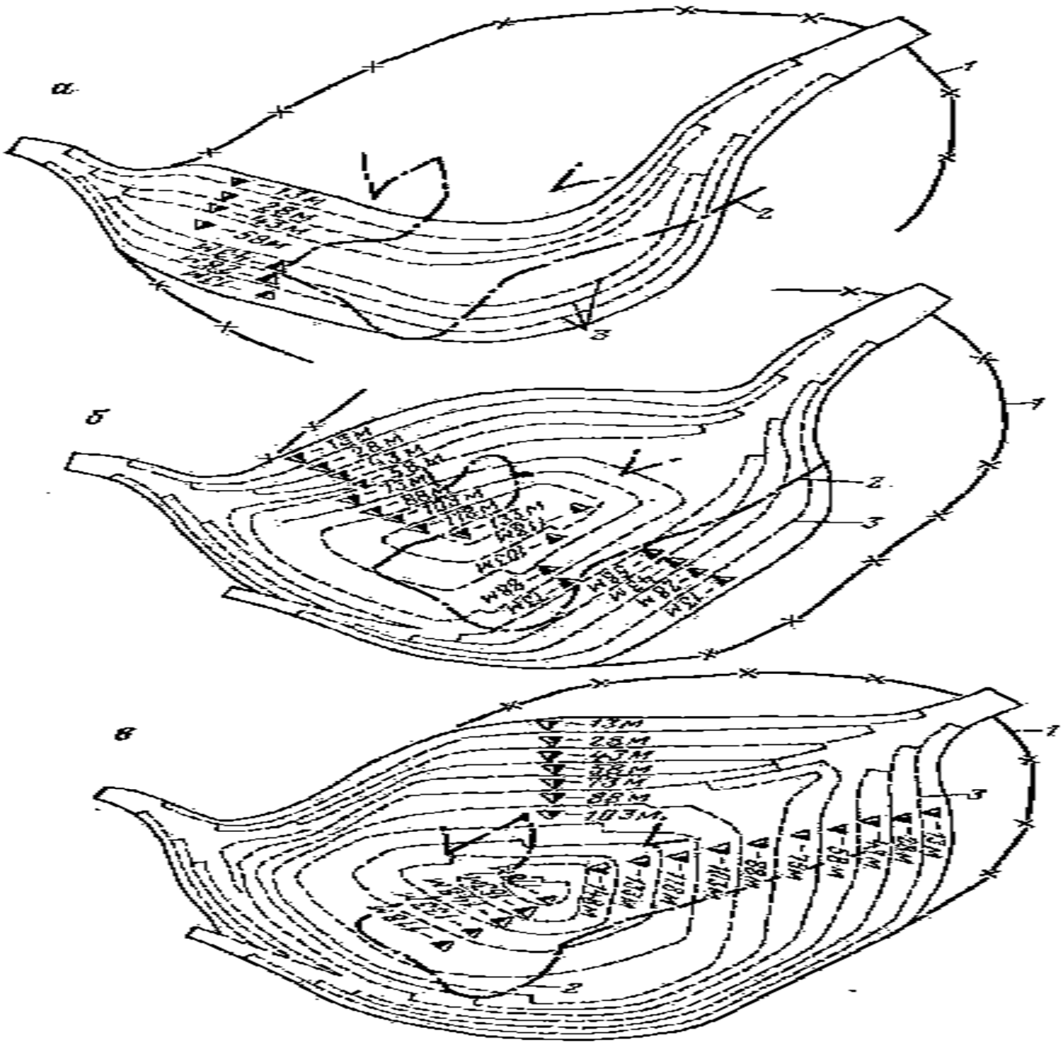
With the deepening of the quarry, the difficulty of developing rocks increases, the height of lifting the rock mass and the distance of transportation of overburden rocks increases. The reduction in the size of the lower horizons causes cramped conditions for the operation of equipment complexes, primarily transport. The management of the quality of the extracted mineral is also becoming more complicated, the water inflow is increasing. Mining conditions are especially complicated when a quarry reaches a depth of 150-200 m or more.

Ensuring the planned volumes of mineral extraction is achieved:

selection of mining and stripping technological complexes that best correspond to the natural and organizational conditions of each stage of development and ensure continuity between the complexes at adjacent stages and during the reconstruction of the quarry;

by changing the schemes of opening routes and the method of opening both with each new stage of development (usually during the reconstruction of a quarry), and within one stage, observing the generally accepted system of opening routes;

by regulating the parameters of the development system in order to manage the current volumes of stripping work both by stages and within the development stages.



19.1. Schemes of changing the shape and size of the quarry and horizons as mining operations deepen:

a, b and c — stages of mining development; 1 — the final contour of the quarry; 2 — the contour of the ore deposit; 3 — the contours of the horizons.

Reference words: formation-like, formations of formations, isometric, tube-shaped, rocky semi-horizontal, deep, altitude-deep, hilly, difficulty of development, regulation of the parameters of the development system.

Security questions:

1. Which deposits are characterized by layers and formation-like deposits?
2. Which rocks are overburden in all inclined and steep deposits?
3. What influences the choice of technological solutions?
4. What determines the final shape and size in terms of a deep-type quarry?
5. Conditions of production of mining operations at the deepening systems of development.

Lecture 22

Topic: Options for the development of mining operations, designs and parameters of berm in deep development systems

План:

1. Варианты развития горных работ.
2. Транспортные бермы.
3. Предохранительные бермы.

In general, when developing an inclined or steep deposit with parallel front movement in a quarry, seven variants of the initial position and direction of mining development are possible (Fig. 36.1): variants 1 and 2 characterize the use of a transverse single–sided development system, 3 and 4 – longitudinal double–sided, 5 and 6 - longitudinal single-sided, 7 - transverse double-sided development systems. Each variant is characterized by certain methods of opening and the mode of mining operations.

Options 1 and 2 in the simplest development conditions are equivalent in terms of the amount of stripping and the method of opening. In both cases, the opening workings and the corresponding transport communications are stationary.

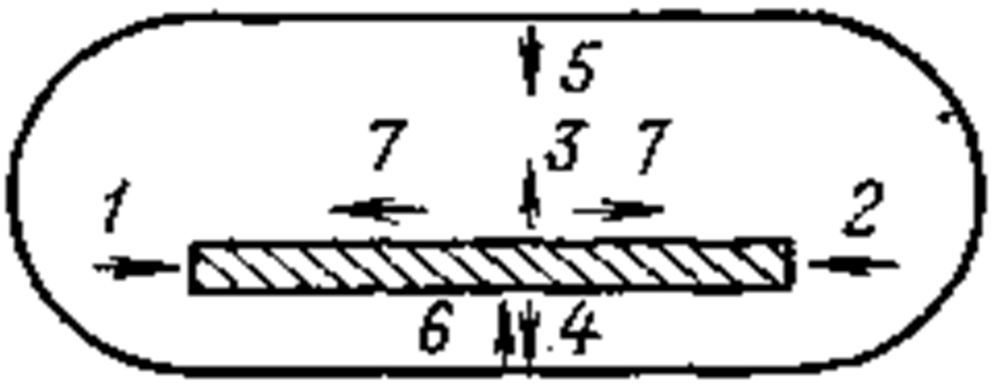


Fig. 36.1. Schemes of variants of the initial position and development of mining operations

In variants 3 and 4, the volume of mining and capital works is small, but the operating conditions of transport are complex, since the opening workings are non-stationary at least for the group of lower working horizons. A split trench can be carried out along the enclosing rocks from the side of the hanging or lying side of the deposit or along the deposit. In the first case, the separate development of minerals is facilitated, its losses and dilution are reduced; such trenches are mandatory for longitudinal systems of low-power (up to 30-40 m) deposits. When developing powerful deposits (200 m or more), split trenches are more often carried out along the deposit near its recumbent side to achieve a more uniform overburden mode and accelerate the transfer of temporary exits to permanent ones.

The development of mining operations according to option 5 is associated with the performance of large volumes of mining and construction work from the contour of the quarry from the hanging side of the deposit, and therefore with large capital expenditures and a long construction period of the quarry.

At the angles of incidence of the deposit β up to 30-35 ° with option 6 of the development of work directly from the recumbent side of the deposit, stationary exits can be arranged on the non-working side of the quarry (with an angle of un) without additional separation (un ≤ β).

With oblique elongated deposits, mining operations develop most often according to option 6 – from the recumbent side of the deposit using a longitudinal single-sided development system.

With an increase in the angle of fall of the deposit (β> un), the volume of stripping (mining and capital) works in the initial period of development according to option 6 increases (see Fig. 36.1). Therefore, in steep deposits, mining operations are developed from the middle of the quarry field to the hanging and recumbent sides of the deposit according to options 3 and 4 using a longitudinal double-sided development system. Working out of overburden rocks from its hanging side can be carried out evenly with a smaller number of excavators or forced for accelerated entry of stationary opening workings.

With very elongated quarry fields and the use of motor vehicles, as well as skip lifts, it is unprofitable to have an excessive mining front due to the large volumes of mining and capital works and the length of transport communications. In this case, it is possible to use variants of transverse development systems. With the transverse arrangement and bilateral development of the front (option 7), relatively small volumes of mining and capital works and transportation distances along the horizons are characteristic - 20-40 and 30-40% less, respectively, than with a longitudinal single–side development system. However, when using a transverse double-sided development system, it is necessary to ensure high speeds of movement and deepening of mining operations, to have increased slopes of intra-barrier roads and sometimes to construct steep trenches equipped with inclined lifts. When using vehicles, a transverse-longitudinal development system is possible.

Muldoobraznye deposits in most cases begin to develop from the wings of the deposit with the movement of the front of the cross strike (Fig. 36.2, a), which allows you to reduce the volume of stripping operations in the initial period. The development system in this case is longitudinal double-sided. When developing muld, it is also possible to move the front of the work along the stretch (Fig. 36.2, b), which improves the stability conditions of the sides and sometimes allows partial placement of rocks in internal dumps.

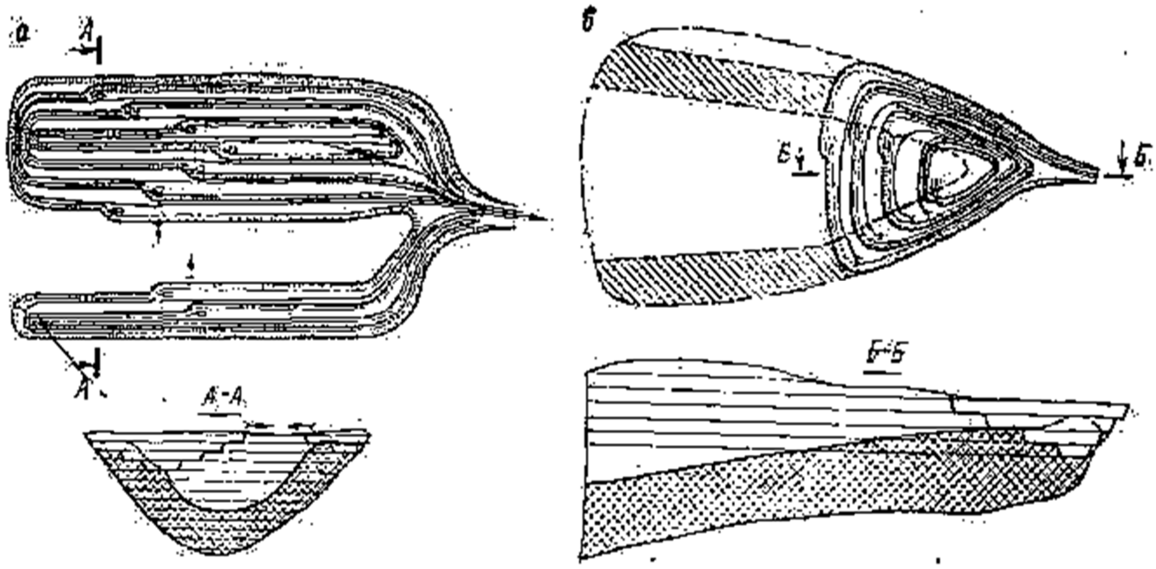


Fig. 36.2. Longitudinal double-sided (a) and transverse single-sided (b) systems for the development of muldy deposits.

When developing relatively short ore bodies, when the quarry has a rounded shape in plan from the very beginning, as well as many deposits of construction rocks, radial-circular development of mining operations is possible on each horizon from the middle in all directions; preparation of horizons is carried out by pits. Circular development of mining operations on the horizons is also advisable when developing deposits confined to a domed hill, while the direction of mining development is from the boundaries of the quarry field to the center. The use of annular central and transverse-longitudinal development systems allows to achieve a high rate of deepening of mining operations in a short time, with a minimum amount of mining and capital work to reach the deposit and start mining operations, reduce the volume of stripping operations in the first operational period of development. With the full development of mining operations, the further use of these systems does not always give positive technical and economic results.

With steep and relatively short deposits in terms of deposits, it is fundamentally possible to fan-develop mining operations on working horizons using a fan-dispersed mining system. At the same time, as a rule, the route of the opening workings is stationary or semi-stationary and has a spiral shape. The fan axis for each horizon is located at the point where the horizontal section of the route adjoins the opening trench. A fan-dispersed development system is characterized by specific features.

In many cases, when rationally developing deposits in difficult conditions, it is necessary to apply different systems or separate variants of systems at different sites, depending on changes in mining and geological conditions and the scale of mining operations. Quite often, as mining operations develop in a quarry, it turns out to be expedient to consistently (rarely simultaneously) use various development systems.

Berm designs and parameters

The ledges of the non–working side of the quarry are separated by platforms (berms) - transport and safety.

Transport (connecting) berms connect the capital trenches with the working horizons on the corresponding ledges. These berms are always horizontal when working out the quarry field in horizontal layers. The minimum width of the transport berm B (Fig. 36.3) consists of the width of the cuvette K (K = 0.5-0.7 m), the transport lane T and the safety lane Z (the width of the prism of a possible collapse). In easily weathered rocks, the width of the safety lane on the side of the worked-out space is at least 2-4 m and, in addition, a scree platform with a width of 4-6 m is provided.

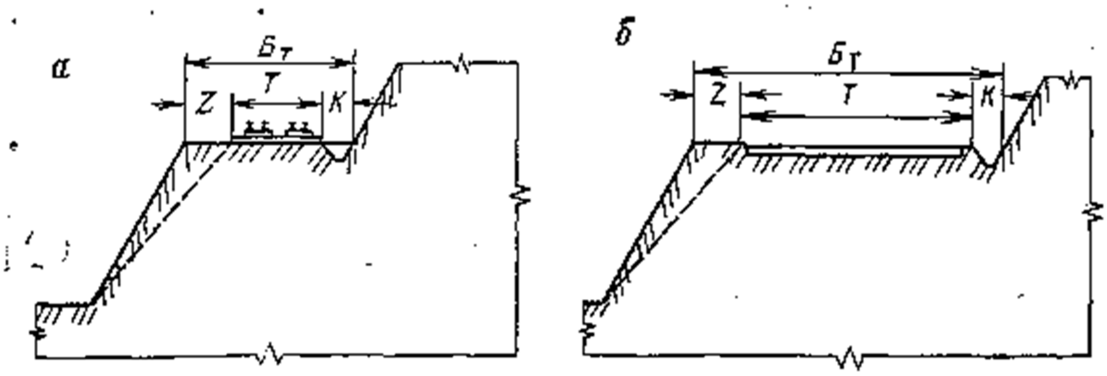


Fig. 36.3. Elements of connecting berms:
a and b – respectively for rail and road transport

In railway transport, T is equal to 3 m for one and 7.5 m for two tracks. In case of motor transport, the width of the carriageway and road verges in two-lane traffic is 11, 13, 15, 18, 22 and 30 m, respectively, for dump trucks with lifting capacity 10-12, 27-30, 40-45, 65– 75, 100-120 and 160-180 t . With tractors with semi-trailers , the T increases by 1-2 m . Often, a fence in the form of a rock shaft with a height of 0.7–1.2 m is arranged on the side of the road, and with a load capacity of more than 75 tons – up to 3.5 m.

The total width of the transport berm for one railway track should be at least 6.5 m, and for two tracks – 10.6 m; in practice, the width of the berm is assumed to be at least 8 and 12-14 m. With dump trucks with a lifting capacity of 27 and 40 tons, the width of transport berms at quarries is 16-18 m, and for more powerful ones – up to 30 m.

The separation of non-working sides for the device of horizontal connecting berms is arranged mainly in shallow quarries. In deep quarries, connecting berms are left only on flat sides, when their additional spacing is not required; with steep stability conditions, connecting berms are almost not provided.

The ramps (semi-trenches) carried out on non-working sides are in fact inclined transport berms, so their width is determined in the same way as the connecting berms.

Safety berms are used to reduce the angle of laying the side of the quarry and increase its stability. The width and location of the safety berms (on each or through two or three ledges) are set based on the accepted angles of slopes of non-working sides and ledges.

During the mass explosion of rocks in the contour zone by vertical borehole explosive charges, rocks outside the exploding block are destroyed; the stabs extend into the depth of the massif 5-10 m from the upper edge of the ledge, the zone of noticeable crack development is 20-30 m, and the zone of concussions and deformations is up to 40-60 m from the wells (Fig. 36.4). For this reason, as well as due to weathering, the berm of small width is destroyed after 3-4 years and a continuous slope of great height is formed. This is dangerous even at the slope angles of the side up to 30 °, since large blocks roll down on the working ledges, In these conditions the width of the safety berms increases to 8-12 m or more.

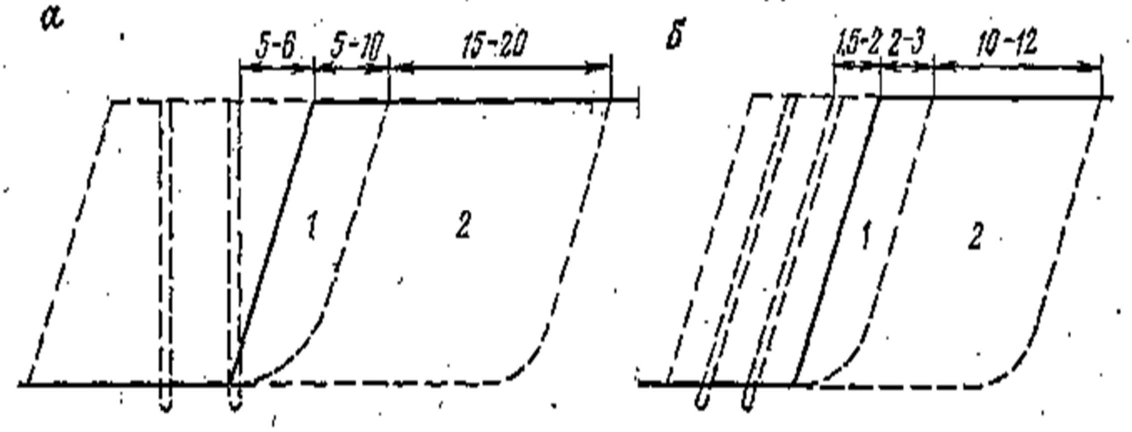


Fig. 36.4. Zones of violation of the ledge during the explosion of vertical (a) and inclined (b) borehole charges:

1 – the zone of stabbing; 2 – the zone of concussion.

When extinguishing the sides, it is advisable to use special blasting methods. If the angles of incidence of rock layers are greater than 26-30 °, then the slopes of the extinguished ledges should coincide with their contacts.

In rocks, it is desirable to build a non-working side with double and built ledges (height 30-45 m) with more gentle slopes and a width of 10-15 m safety berms.

Wide safety berms on each ledge (Well ≥b≥0.5 Well) are typical for soft water–saturated rocks, and for rocks - if the final position of the side is not definitively established.

The arrangement of the ramps and with stable sides leads to their flattening (Fig. 36.5, a) and an increase in the volume of overburden rocks in the contours of the quarry. An increase in the slope angle of the non-working side can be achieved when it is rebuilt with inclined safety bars, the longitudinal slope of which is equal to the slope of the exits (Fig. 36.5, b). The main disadvantage of such structures is the increase in the cost of mining operations during the construction of inclined berms compared to horizontal ones (by at least 20-25%). Such non-working boards are advisable mainly in deep round-shaped quarries with their relatively small dimensions in plan and with a spiral track of stationary exits (Fig. 36.5, c). Inclined safety berms can be used for moving mining equipment between ledges, and in short–term operation - also for transporting rock mass..

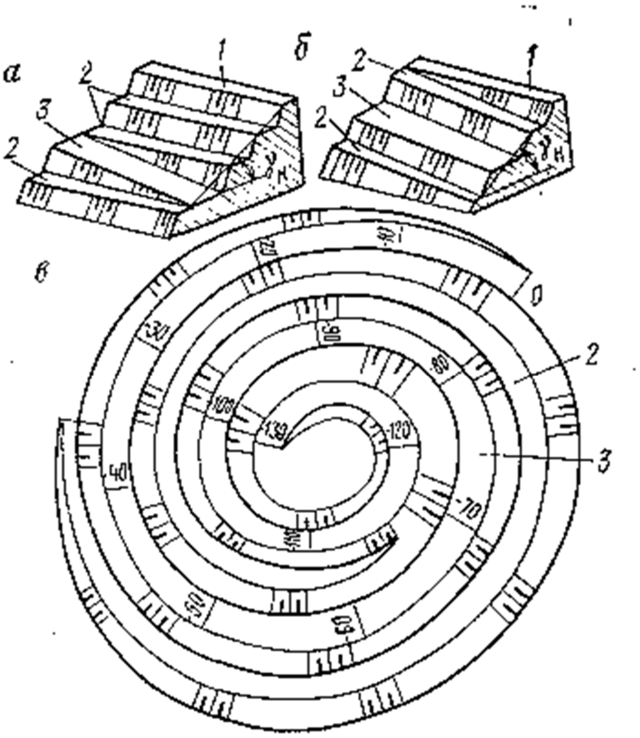


Fig. 36.5. Schemes of boards with horizontal and inclined safety berms:
1 – day surface; 2 – safety berms; 3 – exit.

Reference words: transverse single-sided, longitudinal double-sided, longitudinal single-sided, transverse double-sided, volume of mining and capital works, angle of fall of the deposit, ledges of the non-working side, connecting berm, width of the ditch, transport lane, safety lane, collapse prism, separation of non-working sides, exit.

Security questions:

1. Specify according to the scheme the options for the initial position and the direction of development of mining operations.

2. Which development system is used to characterize variants 1 and 2 (Fig. 36.1)?

3. How are the ledges of the non-working side separated?

4. What are the transport berms designed for?

5. What are the safety berms for?

Lecture 23

Topic: Opening by external capital trenches

Plan:

1. The required throughput of the route of the opening workings.
2. The main economic indicators of the volume of work on opening and preparation of the horizon.
3. Options for the depth of the external trenches.

Required throughput of the route of opening workings (trains/day)

$$N_T = \frac{fW_c}{V_c} = \frac{fL_{\phi.y}H_yN_yv_\phi}{V_cT_p},$$

where f – reserve ratio (f=1,2÷1,25); W_c – average daily cargo turnover of working horizons serviced by one route, м^3; V_c– capacity of the composition (in a dense body), м^3; $L_{ф.у}$ – average length of the work front of the ledge, м; $N_у$ – the number of working ledges serviced by this route; $v_ф$ – the rate of advance of the mining front, m/year; $T_р$ – the duration of the career in a year, day.

When opening the working horizons with external capital trenches, a high throughput of the route is ensured, as well as the independent conduct of capital trenches from mining operations in the quarry, as a result of which the construction period of the quarry is shortened and cargo flows are divided already in the initial period of its operation. However, as the depth of the outer trench H increases, its volume increases proportionally to H2 and H3, and this, in turn, limits the final H.

Variants of different depth of laying of external trenches, providing the required cargo turnover of the quarry and the productivity of excavators, are compared in terms of capital and operating costs, taking into account the construction period of the quarry. If the depth of the external trench limits the production capacity of the quarry, this should be taken into account in technical and economic calculations*.

One of the main economic indicators of the performance of a certain amount of work on the opening and preparation of the horizon are depreciation costs (sum / m3) for mining and construction work, attributed to 1 m3 of operational work:

$$C_a = \frac{V_{г.п}C_{г.п}}{V_{г.э}},$$

where $V_{г.п}$ – the volume of mining and preparatory work, м^3; $C_{г.п}$ – unit costs for mining and preparatory work, сум/м^3; $V_{г.а.}$ – the volume of mining and operational work, м^3.

When opening internal trenches, depreciation costs for mining and preparatory work (sum/m3) are determined by the expression

$$C_a^{'} = \frac{V_{г.п}C_{г.п} + V_p C_в}{V_{г.э}},$$

where V_p – the volume of additional side spacing, $м^3$; $C_в$ – unit costs for mining and operational (stripping) work, сум/$м^3$.

When opening several horizons simultaneously by the system of external trenches, the construction volumes of the upper trenches (and, consequently, the costs for them) are attributed only to the corresponding volumes of the rock mass of the horizon being opened, with the exception of the last trench in depth and the largest in volume, the paths of which then pass into the inner trenches. Its construction volume is distributed over the volume of the rock mass of both the exposed and the underlying horizons.

Thus, external trenches can have a deeper foundation with a large size of the quarry field in terms of, mainly at the intersection of the deposit, and a large final depth of the quarry.

When laying an external trench with a depth of H in the direction of the reception points of the rock mass, the total distance of transportation decreases compared to opening internal trenches, which is especially important when using vehicles.

With the height of the ledge Nu = 10 ÷ 15 m, the small size of the quarry and the volume of cargo flows, the final depth of the outer trench H is 15-20 m.

For medium - sized and large - sized quarries , H is 25-30 and 40-50 m , respectively .

When opening trenches of mixed laying according to the technical conditions of tracing, an increase in the depth of the outer trench by one horizon is advisable if the route is simplified, for example, one dead end is reduced to the lower horizon opened by internal trenches. In this case, the savings in operating costs will significantly exceed the additional costs of deepening the external trench.

During the development of deposits, the outputs of which under sediments have a limited reach, the size of the quarry in terms of the first period is relatively small. The internal laying of capital trenches from the side of the recumbent side under such conditions may not provide the required carrying capacity of the route or is associated with a large additional spread of the sides for laying dead ends and loop joints. In these cases, the outer trenches are deepened to 60-80 m, which makes it possible to open the deposit faster and significantly improve the ratio of overburden and mining operations in the first period of the quarry's existence. At the same time, the total volume of mining and construction works increases by the time the quarry is put into operation, but the necessary advance of stripping works on the upper ledges decreases.

In large quarries with a large (more than 80 m) capacity of low-resistant sediments, the depth of external trenches can be increased if they directly open a mineral deposit. With this solution, a large cargo turnover of the quarry is ensured, the cargo flows of minerals and overburden rocks are separated and the reliability of the transport scheme increases.

Typical for the development of deposits with a thick layer of covering rocks (80-200 m) is the construction of a first-stage quarry ("initial quarry") with limited dimensions along the bottom and surface in order to reduce the volume of mining and capital works. At the same time, during the construction period, it is necessary to rebuild a section of the non-working side (along the final or intermediate contour of the quarry) to accommodate the internal route. The creation of such a board is mandatory when opening mining horizons with a steep trench for the operation of combined, for example, automobile-conveyor-railway transport. The creation of "semi-stationary" external trenches makes it possible to accelerate the construction of a quarry and the development of its capacity, as well as reduce the amount of stripping work in the first period of operation.

Reference words: throughput, route, reserve ratio, cargo turnover, working ledge, duration of the quarry, economic indicators, volume of mining preparatory work, unit costs, volume of mining operations, laying.

Security questions:

1. By what formula is the required throughput of the route of the opening workings determined?
2. What is provided when opening working horizons by external capital trenches?
3. By what formula are depreciation costs for mining and construction work determined?
4. By what formula are depreciation costs for mining and preparatory work determined?
5. How can I increase the depth of external trenches?

Lecture 24

Subject: Simple, dead-end and loop routes

Plan:

1. A simple route.
2. Dead-end routes.
3. Loop routes..

The most widespread for opening deep horizons were various types of internal semi-trenches (congresses). They are most often a continuation of the trenches of external laying.

The angle in terms of φ between the exit axis and the slope of the ledge can be within arcsin ctga≤φ≤π/2 (Fig. 20.1). At a minimum angle φ = arcsin i ctga (see Fig. 20.1, a), the additional spacing of the non-working side decreases. This angle is increased when using vehicles to reduce the length of the route, the distance of transportation on the horizons. At the same time, the exits have an upper trench and a lower semi-trench part (see Fig. 20.1, b) or an upper trench part, and the lower part in the form of an embankment (see Fig.20.1, c). Often inclined internal trenches are also called exits.

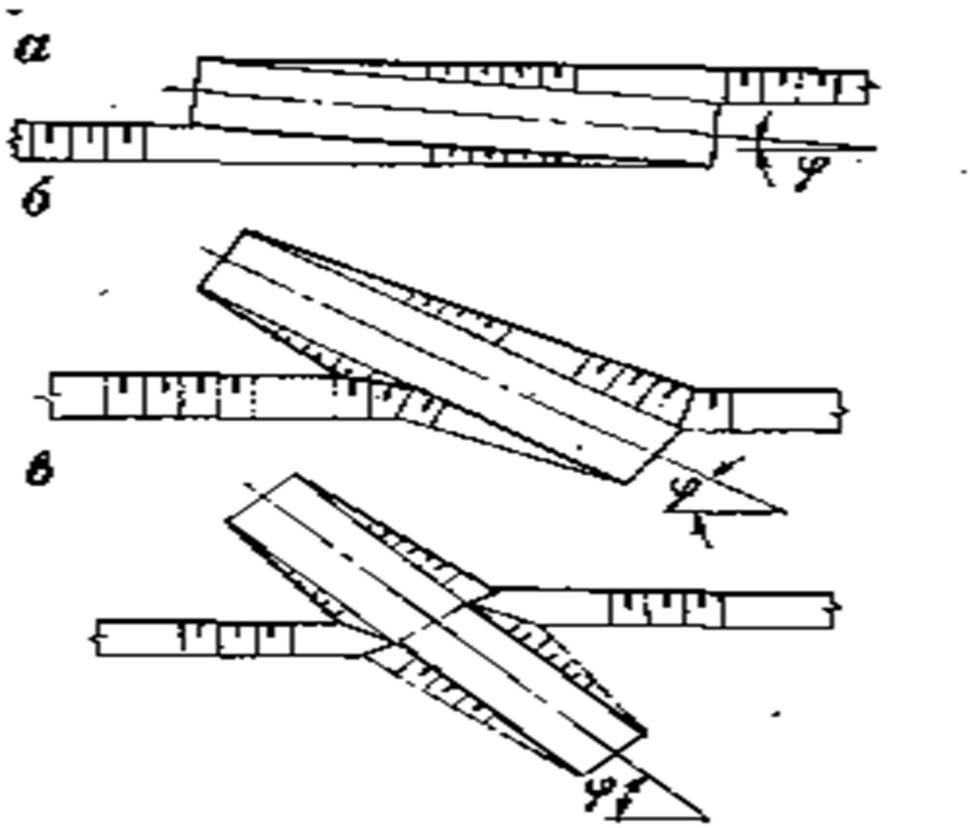


Fig. 20.1. Exit diagrams.

A simple track is placed on one or two adjacent sides of the quarry. The track can be stationary, semi-stationary, sliding, or have stationary and sliding parts. A simple highway is characterized by a constant direction of movement of vehicles within its limits and the least complex design of points of abutment to the horizons. The consequence of this is the maximum speed of traffic and the capacity of the route for internal routes, the minimum spacing of the side of the quarry.

Sections of the stationary route are formed immediately as the working horizons are opened or when the sections of the sliding route exit to the non-working side of the quarry, and the horizon with a new section of the stationary route is working or already non-working. When forming a stationary simple route in the zone of the spent upper horizons, it is advisable to arrange the junction of the exits to them on the guiding ascent. At the same time, the number of horizons opened by a straight segment of the route increases, and the spacing of the sides decreases. In case of motor transport, according to the requirements of traffic safety on a prolonged ascent, the device of flat inserts is necessary.

The abutment of stationary exits to the working horizons is usually carried out at intermediate sites.

For rail transport, the length of the Lp platforms is 150-400 m; for motor transport—15-30 m and mainly depends on the required width of the transport berms.

Internal tracks or trenches of mixed laying with simple tracks in railway transport are usually possible in quarries up to 60-100 m deep . In motor transport, such stationary routes are widely used in elongated quarries for opening both all and groups of upper horizons.

With semi-horizontal rocks, the slope angle of the side, which ensures the placement of transport communications, is 26-38 °. If it is less than the angle determined by the slope stability conditions, then an additional side spacing is required to accommodate stationary exits. In the general case, the volume of additional side spacing (m3) can be approximately determined for all forms of the route as the volume of the semitrism (Fig. 38.2):

$$V_p = \frac{K_y b H_k^2}{2i_p},$$

where Hк — the final depth of the laying of the inner route, м; Kу — the coefficient of lengthening of the route; b — trench bottom width.

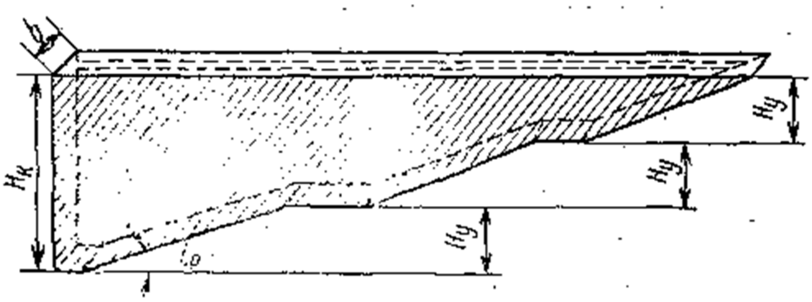


Fig. 20.2. Scheme for the calculation of additional side spacing for the placement of the internal route

Dead-end routes can be stationary, semi-stationary and sliding. The length of the dead-end platforms of the junction of the Lp is determined from the conditions for placing the train on them and the possibility of braking it before stopping; at the same time, the conditions of the exchange of trains are taken into account, i.e. the scheme of the track development of dead-end sidings and junction posts. The value of Lp varies from 250 to 600 m . The width of the dead-end platform is determined by the number of tracks laid, the dimensions of the rolling stock and the stability of the slopes of the ledges, usually it is 8-20 m.

Typical for a dead-end highway is the opening of working horizons with single internal *trenches (Fig. 20.3, a and b). It is possible to use paired trenches with a dead-end route (Fig. 20.3, b).*

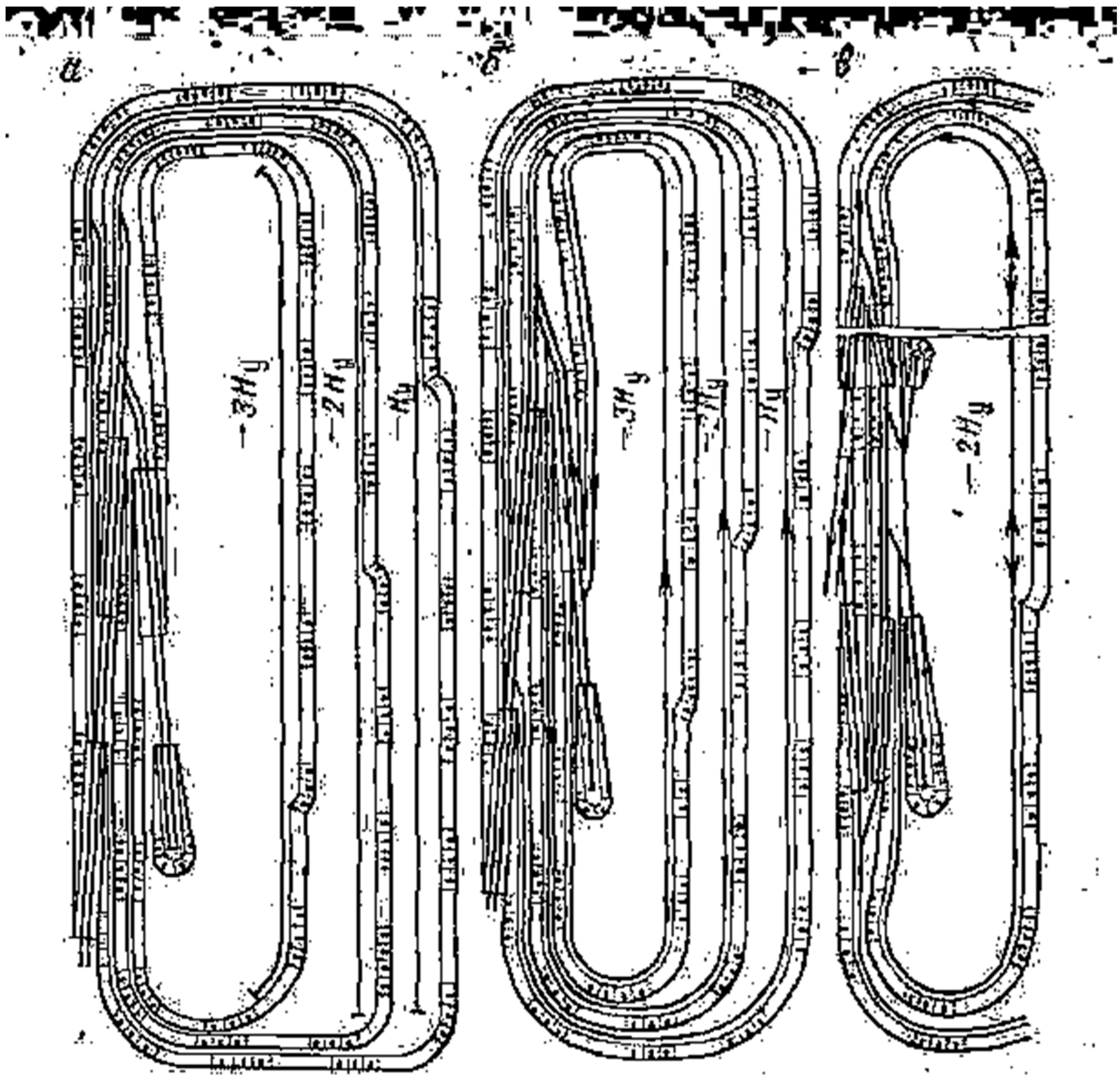


Fig. 20.3. Schemes of dead—end routes:
a and b - double—track, respectively, with one- and two-way abutment;
c - single-track (paired trenches).

Dead-end routes are divided into single-stage and multi-stage (translational-dead-end) accordingly, when opening a straight section of the route of one or more horizons, and by the number of paths — on single-track and double-track.

In deep quarries (Nc =170-200 m or more) with their large dimensions in plan, the route is often three- and two—stage at the upper and middle horizons, and single-stage at the lower ones. With a small length of the side of the Lc, the maximum possible height (m) of the exposed ledges depends on the slope of trench I and the length of the dead-end areas of the junction of the Lp:

Well max=(Lk-2Lp)i.

The track development of dead-end junction points is mainly determined by the number of tracks at the exits, the number of sides of the junction of the working horizon tracks to the dead end (one- and two-way), the presence or absence of "hostility" of the routes of loaded and empty trains.

The track development of exit ramps, intermediate and dead-end junction points is established in accordance with the cargo turnover of the working horizons serviced by individual sections of the route.

The continuous movement of trains allows to increase the capacity of the dead-end route. This requires the installation of telescopic dead ends with a two-way junction or two routes, respectively, when opening single and paired trenches (Fig. 20.4).

With flow-through train traffic schemes, a large length of the quarry field is required. So, even with a single-stage telescopic track, the length of one of its sections is equal to 1300-1500 m. The length of the side should be much longer to accommodate a multi-stage telescopic track. Usually, no more than two upper ledges are opened with a straight section of the route. With schemes with two routes, the preparation of horizons is accelerated due to the simultaneous carrying out of trenches in two directions.

A two-track dead-end highway, even with telescopic dead ends, has less capacity than a simple one. The arrangement of the third and fourth paths does not lead to an increase in the capacity of the route due to the intersections of the paths on the horizons, therefore multi-track routes are not used

With in-line traffic schemes and auto-blocking, the capacity of double-track dead-end tracks can be 200-280 pairs of trains per day, and the annual production capacity of a quarry on a mountain mass can reach 16-30 million tons with single-track two routes and 25-40 million tons with double-track telescopic routes. Schemes of track development of simple and dead-end routes, in which the flow movement of trains is provided, are possible practically only with a longitudinal single-board development system.

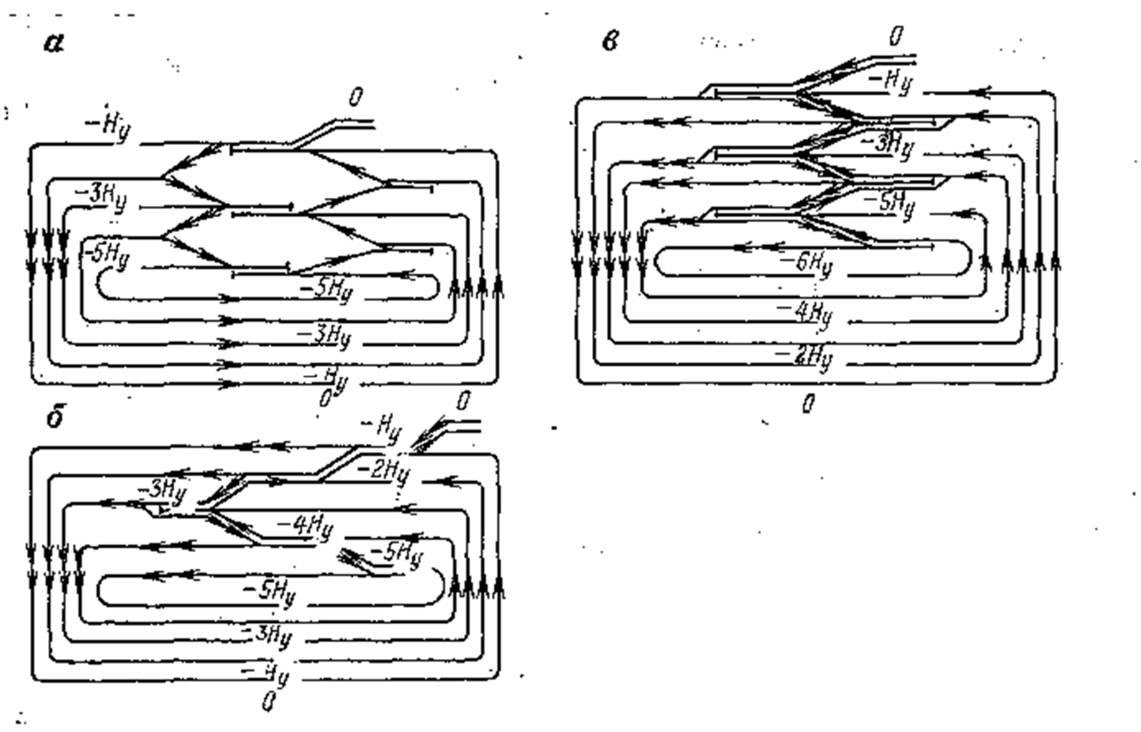


Fig. 20.4. Schemes of track development of dead—end tracks in the flow of trains:

a — with two single-stage dead-end tracks (freight and empty): b and c - respectively with multi-stage and single-stage with telescopic dead ends.

Loop routes are characterized by high throughput. When installing tracks on a slope or on the side of a quarry, the placement of a turning platform is possible in a recess, on an embankment, or simultaneously in a half-recess and on a half-embankment (Fig. 38.5, a and b). The volumes of mining operations (m3) for the construction of a excavation or embankment are approximately calculated according to the formulas proposed by E. I. Vasiliev:

for semi-excavation

$$V = \frac{2}{3} K_{в} \psi R^3 \lambda;$$

for a half - sleep

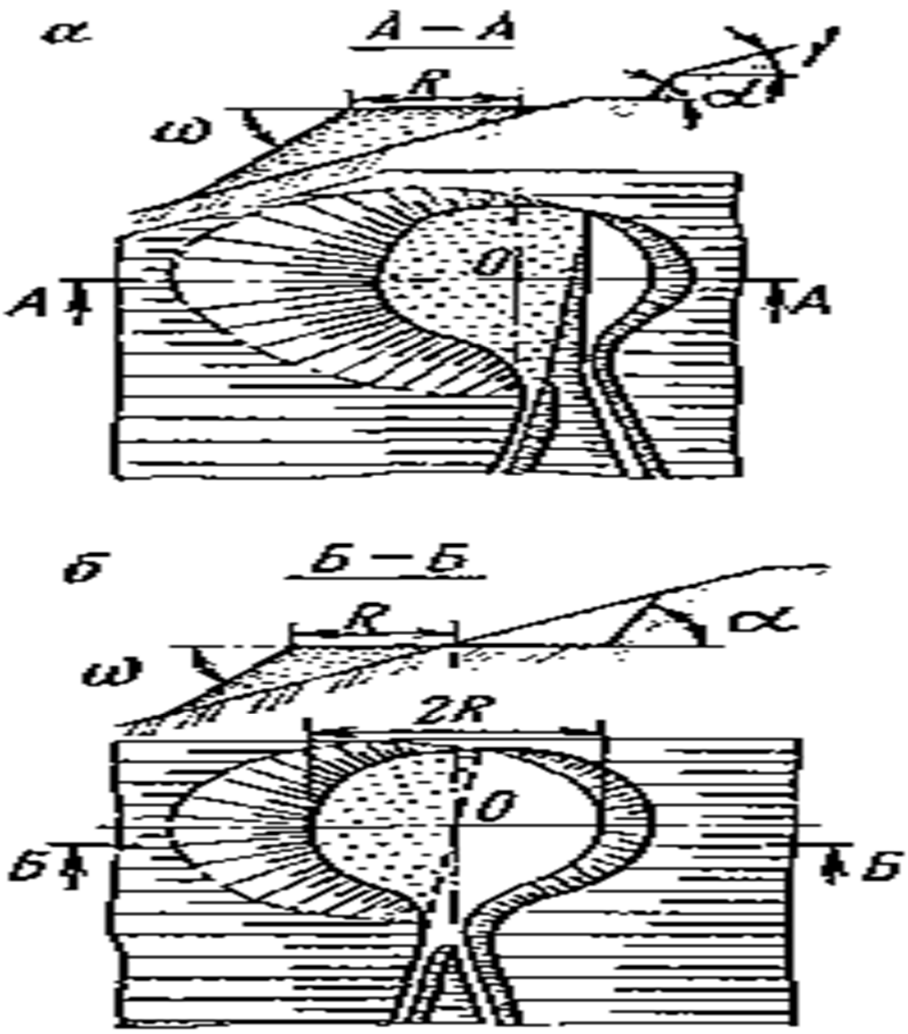


Fig. 20.5. Schemes of loop platforms:
a – on an embankment; b – in a half-recess and on a half-embankment of equal width.

$$V = \frac{2}{3} K_{н} \psi_1 R^3 \lambda.$$

Values ψ *and* ψ_1 defined from expressions.

$$\psi = \frac{\sin\alpha \cdot \sin\gamma}{\sin(\alpha - \gamma)}; \quad \psi_1 = \frac{\sin\omega \cdot \sin\gamma}{\sin(\omega - \gamma)},$$

where α — angle of slope of the side of the recess, degree; γ — angle of inclination of the side of the quarry or slope, degree; ω — slope angle of the embankment, degree.

coefficient λ takes into account the end sections of the half-excavation or half-embankment. Its values when determining the volume of the half - hole are taken as follows:

the angle of the slope of the recess or embankment, degree	90—75	75—60	60—45	45—30	<30
coefficient λ	1,02	1,08	1,13	1,18	1,22

coefficient $K_в$ and $K_н$ take into account the central angle covered by the semi-bulk.

Central corner, covering the semi - recess, degree	180	170	160	150	140	130	120	100	90	60
coefficient $K_в$	0,67	0,53	0,43	0,33	0,25	0,18	0,13	0,07	0,04	0,02

The volume of mining work on the construction of the site for the loop is proportional to the third degree of its radius and can reach several million cubic meters. Therefore, in railway transport, the feasibility of constructing a loop route is checked by comparing the costs of additional volumes of mining operations and possible savings in transportation. A loop connection in railway transport is used if there are exposed sections of the sides on the flanks of the quarry field; such connections can be calculated for the location of paired independent routes.

In case of motor transport, loop routes are generally accepted, since the additional volumes of mining operations in this case are much smaller. In order to allow vehicles to move along the highway at a speed of 20 km / h, it is necessary that the turning radius of the loop connection is at least 25 — 30 m. To do this, the separation of the sides of the quarry or the slope in the areas of the location of the turning platforms increases. Only in cramped conditions (steep slopes, lower horizons of deep quarries) it is allowed to reduce the radius of the loop to 15-20 m.

Reference words: opening, horizon, inner half-trench, angle in plan, adjacent side of the quarry, sections of stationary highway, abutment of stationary exits, slope angle of the side, stationary, semi-stationary, sliding, track development, formulas of E.I.Vasiliev, embankment.

Security questions:

1. What characterizes a simple highway?
2. How are sections of a stationary highway formed?
3. What can be dead-end routes?
4. What makes it possible to increase the capacity of a dead-end highway?
5. What characterizes loop routes?

BRANCH OF THE FEDERAL STATE AUTONOMOUS EDUCATIONAL INSTITUTION OF HIGHER EDUCATION

"National Research Technological University "MISIS" in Almalyk

DEPARTMENT OF "MINING"

METHODICAL INSTRUCTION

to perform practical work

on the discipline

"TECHNOLOGY AND COMPLEX MECHANIZATION OF OPEN-PIT MINING"

Almalyk 2022

PRACTICAL WORK No. 1
Safety precautions during open-pit mining operations

1. General rules

Uniform safety rules for the development of mineral deposits by the open method are mandatory for all organizations (regardless of their organizational and legal forms and forms of ownership) engaged in this type of activity.

The development of mineral deposits by the open method includes activities for the design, construction, operation, expansion, reconstruction, technical re-equipment, conservation and liquidation of open-pit mining facilities.

The objects of open-pit mining are quarries, mines, waste landfills, heap leaching facilities, as well as objects for the development of rock dumps, substandard ores, mines, quarries, hydraulic dumps of processing plants, gold dumps and slag dumps of thermal power plants and metallurgical enterprises.

Workers engaged in jobs that involve combining professions should be trained in occupational safety and instructed in all types of combined work.

Before starting the mechanism and starting the movement of cars, it is necessary to supply sound or light signals. Every incorrectly given or misunderstood signal should be perceived as a ”stop" signal.

The movement of people from ledge to ledge on a dangerous and exploded mass is prohibited.

It is forbidden to work on ledges in the presence of overhanging canopies, boulders and individual large protrusions.

2. Mining operations

When developing a single-bucket excavator of the mechanical shovel type without changing blasting operations, the height of the ledge should not exceed the maximum height of scooping, with the use of blasting operations more than 1.5 times the height of the excavator scooping.

The slope angle of the working ledges is set according to the data of surveying observations, but not more than 80 °.

3. Drilling operations

The drilling rig must be installed on the observation deck and positioned so that the tracks of the machine on the ledge are outside the collapse prism, but not closer than three meters from the edge of the ledge.

When drilling the first row of wells, the drilling rig must be installed so that its longitudinal axis is perpendicular to the edge of the ledge.

It is forbidden to leave drilled wells open.

The drilling rig operating on site must be attached with a safety belt.

The movement of the drilling rig along the ledge should be carried out only on a mounted horizontal platform.

Exploding

The safe distance for people during blasting operations should be established by the project or passport and be such as to exclude accidents.

In order to protect buildings and structures from seismic impacts during blasting and work with explosives, the mass of the explosive charge should be such that when exploding, damage that disrupts their normal functioning is excluded.

When placing several objects with VM on the Earth's surface, distances between them must be observed that exclude the possibility of detonation transmission.

Dangerous zones, as well as the locations of people, the placement of VM during the preparation and conduct of mass explosions is determined by the project.

Mass explosions on the earth's surface that pose a threat to dangerous air traffic can be carried out only after they are agreed to be carried out in accordance with the established procedure.

On impact-hazardous coal seams, before blasting operations in the treatment and preparatory faces, as well as during the working out of the targets, people should be removed from the explosion site at a safe distance, but not less than 200 meters, and be in the fresh air.

Single-bucket excavators

The transportation of excavators by bulldozers is allowed only with the use of a rigid coupling and one carries out specially designed measures to ensure safety.

The presence of unauthorized persons in the cab and on the external platforms of the excavator during its operation is prohibited.

When moving the excavator along a horizontal path or on an ascent, the driving axle should be located behind, and when lowering from a slope - in front.

When loading into vehicles, the excavator driver must signal the start and end of loading.

Road transport

The plan and project of technical roads must comply with the SniP and Protodeakonov.

The width of the carriageway is set based on the size of the cars, taking into account the gaps between oncoming cars of at least 1.5 m. and from the wheel to the edge of the carriageway of at least 0.5 m.

The car must be technically fixed and have a rear-view mirror, a functioning light and sound alarm system and lighting.

On quarry roads, the movement of cars should be carried out without overtaking. The speed is not more than 30 km / h, and at the entrance to the excavator 6 km / h.

In all cases, when reversing, a continuous beep should be produced.

Unilateral or oversized loading, as well as loading exceeding a certain load capacity of the vehicle is not allowed.

Traffic on the roads of the quarry should be carried out according to the technological map.

When vehicles are operating (especially in icy and rainy weather), there is a threat of collision of oncoming dump trucks, their slipping into ditches and falling from ledges. Therefore, the relevant services must maintain the roads in a condition that excludes this danger. In icy conditions, it is necessary to systematically fill roads with slag, sand and other materials that exclude sliding. To exclude the fall of dump trucks from the ledges at the upper edge (from the side of the slope of the ledge), a rock shaft with a height of 0.8 - 1.2 m is poured off.

The longitudinal profile of the highway should be such that the driver has the necessary visibility. To do this, sharp profile fractures should be avoided. When constructing serpentines, the minimum radii of rounding and convergence of serpentine branches must be strictly observed in order to ensure the safety of movement along them.

All vehicles must be carefully checked by competent persons before leaving the garage. Particular attention should be paid to the serviceability of the brake system, steering, clutch, transmission signals, lighting and instrumentation.

The dump truck must be installed for loading so that the bucket of the excavator does not pass over the cab when turning. It is forbidden to be in the cab when loading a dump truck with an excavator (it is allowed only for dump trucks with a reliable protective visor). The movement of dump trucks with a raised body, reversing to the place of loading at a distance of more than 30 m and the transportation of unauthorized persons in the cab is prohibited. It is forbidden to walk on highways and their roadsides because of the danger of injury by falling pieces of rock mass.

Only well-trained drivers who have received appropriate training in safe working methods and who observe production discipline can be allowed to drive vehicles in quarries.

Bulldozers

It is not allowed to leave the bulldozer unattended with the engine running, the blade raised, and when working, direct the cable, stand on the suspension frame and the dump device. It is forbidden to work on a bulldozer across steep slopes.

The maximum slope angles during the operation of the bulldozer should not exceed 25 ° on the rise, and 30 ° on the slope.

PRACTICAL WORK No. 2

Determining the main parameters of a career

The main parameters of the quarry are the volume of rock mass, the final depth, the dimensions on the sole, the angles of the slopes of the sides, mineral reserves, the volume of overburden and the dimensions at the level of the day surface.

The volume of rock mass (m) in the contours of the quarry, characterizing the scale of mining operations, is determined by the formula of Academician V.V. Rzhevsky

$$V_{г.м} = SH_к + \frac{1}{2}\sum_1^n L_n H_к^2 ctg\beta_n + \frac{1}{3}\pi H_к^3 ctg^2 \beta_{cp}\text{, м}^3$$

where S – the area of the sole of the ABCDE quarry (Fig. 1.1), м^2;

$H_к$ – career depth, м;

β_n – angle of slope of the side of the quarry, degrees;

l_n – length of the nth section of the side, м;

β_{cp} – average slope angle of the side, degrees;

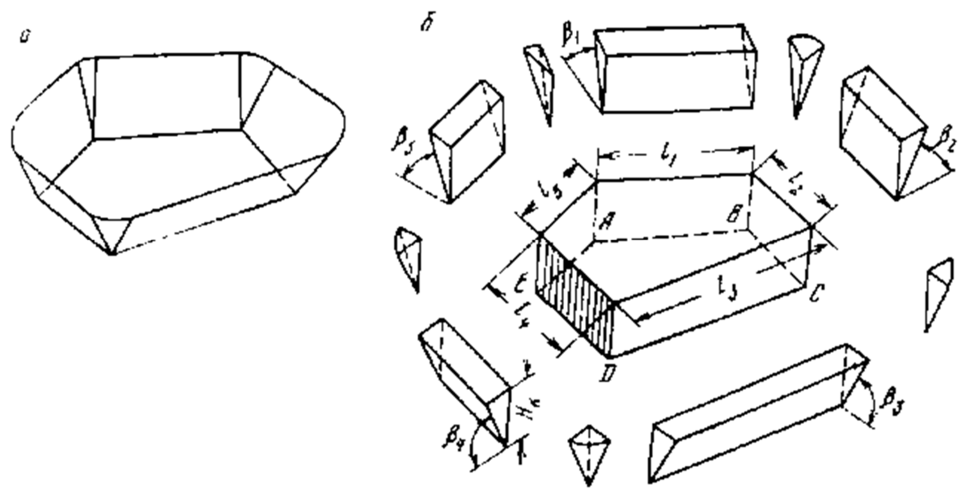


Fig. 1. Scheme for determining the volume of rock mass Vg.m in the contours of the quarry: a – general view; b – geometric bodies that make up the volume of the quarry

If the angles of the slopes of all sides of the quarry are equal or differ slightly from each other, then the formula of Academician V.V. Rzhevsky will take the form

$$V_{г.м} = S * H_к + \frac{1}{2} P * H_к^2 ctg\beta_{cp} + \frac{1}{3}\pi H_к^3 ctg^2 \beta_{cp}\text{, м}^3$$

where P – perimeter of the sole of the quarry, м.

$$P = 2(P_д + B_д)\text{, м}$$

$$S = L_д * B_д\text{, м.}$$

When developing shallow and horizontal deposits, the final depth of the quarry is determined by the mark of the soil of the mineral formation or the sum of the overburden capacity $h_в$ and minerals $h_и$,.

$H_{к.к} = h_в + h_и$, м

An increase in the depth of a quarry developing a steep deposit causes a constant increase in the current stripping coefficient. Upon reaching some intermediate career depth $H_{к.к.}$ (Fig. 2) the value of the current stripping coefficient will become equal to the boundary, $k_т = k_{гр}$.

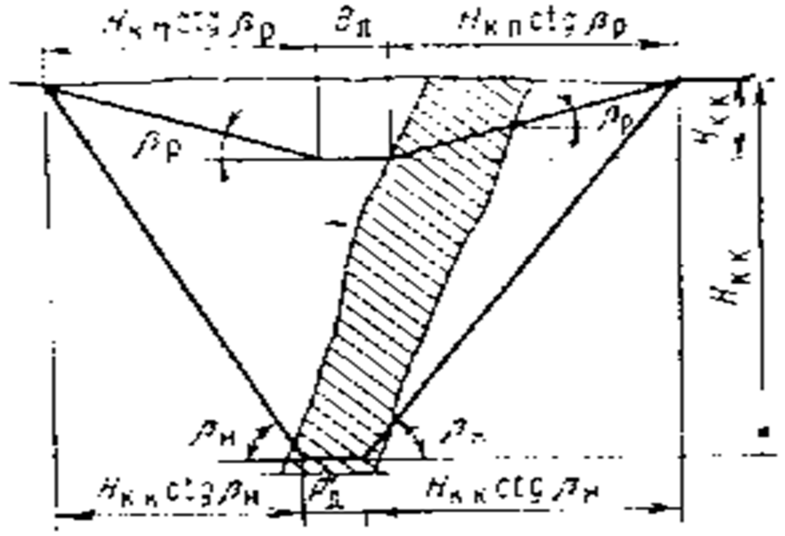


Fig.2. Scheme for determining the final depth of the quarry.

The analytical method for calculating the final depth of quarries is approximate, since it does not take into account all mining-geological, topographic and other features of the deposit. For a more accurate solution of this issue, the methods of graphic, graphoanalytic and the method of variants are used.

The final depth of the quarry is determined by the formula:

$$H_{к.к.} = \frac{-P + \sqrt{P^2 - 4\pi[S - m_г L_д (1 + k_{гр})]}}{2\pi ctg\beta_н}.$$

where $m_г$ – horizontal capacity of the deposit.

$$m_г = \frac{m}{\sin \beta_з}, \text{ м.}$$

The dimensions of the quarry bottom in the final boundaries during the development of horizontal deposits are determined by the contours of the deposit in the plan at the level of the sole. When developing inclined and steep deposits, the minimum width of the quarry bottom is determined by the condition of safe mining operations and is 30-40 m. The length of the bottom of the quarry is assumed to be equal to the length of the deposit along the strike (with its insignificant length). In the case of a large length of the deposit, the length of the bottom of the quarry is assumed to be 3-4 km for technical reasons. The minimum length of the bottom of the quarry should be within 70÷100м.

$$x = \frac{(m_г - B_д)\cdot(tg\beta_з - tg\beta_н)}{2tg\beta_з}.$$

The volume of mineral reserves is determined by the formula:

$$V_и = m_г * L_д\left(H_н - h_н\right) - \left(S_1 + S_2\right) * L_д, \text{ м}^3$$

where S_1, S_2 – accordingly, the area of the mineral formed from the side of the hanging and recumbent sides when the bottom of the quarry is located inside the deposit (Fig.3.)

$$S_1 = \frac{(m_г - x - B_д)^2 tg\beta_з tg\beta_н}{2(tg\beta_з + tg\beta_н)}. \qquad S_2 = \frac{x^2 tg\beta_з tg\beta_н}{2(tg\beta_з - tg\beta_н)}.$$

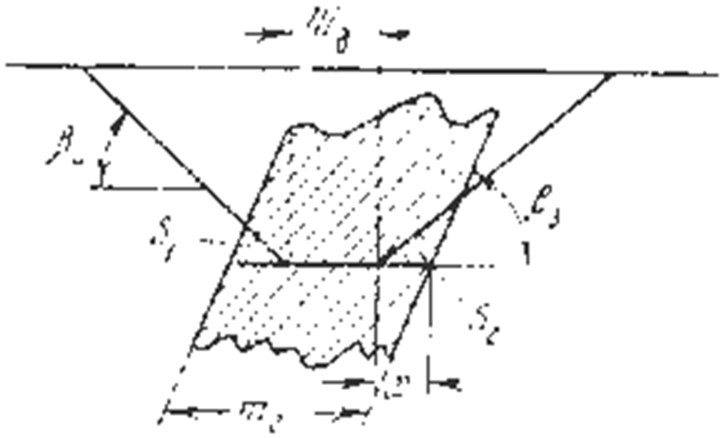


Fig. 3. Scheme for choosing the position of the bottom of the quarry.

Industrial mineral reserves (operational losses are assumed to be equal to 4%) are determined by the formula:

$$З_п = 0{,}96 * З_б, \text{ т.}$$

where $З_б$ – balance reserves of minerals, which we assume to be equal to geological reserves:

$$З_б = З_г = V_и * \rho_и, \text{ т.}$$

The volume of rock mass in the contours of the quarry (we accept $\beta_{ср}=\beta_н$) determined by the formula:

$$V_{г.м} = S * H_к + \frac{1}{2}P * H_к^2 ctg\beta_{ср} + \frac{1}{3}\pi H_к^3 ctg^2\beta_{ср}. \text{ м}^3.$$

The volume of overburden in the final contours of the quarry is determined by the formula:

$$V_в = V_{г.м} - V_и, \text{ м}^3.$$

The average industrial stripping coefficient is determined by the formula:

$$k_{ср} = \frac{V_в}{З_п}, \text{ м}^3/\text{т}.$$

Example. To determine the final depth of the quarry, balance and industrial reserves of minerals, the volume of overburden and the average industrial overburden coefficient during the development of a reservoir.

Given:

deposit capacity m=56 м;

the angle of inclination of the deposit $\beta_з$=70°;

width of the quarry bottom $B_д$=40 м;

quarry bottom length $L_д$=2000 м;

boundary stripping coefficient $k_{гр}$=9,2 $м^3/м^3$;

angle of inclination of the non-working side of the quarry $\beta_н$=39°;

mineral density $\rho_и$=2,8 $т/м^3$;

height of non-working ledge $h_н$= 8 м.

Decision:

1. The horizontal capacity of the deposit is determined by the formula:

$$m_г = \frac{m}{\sin\beta_з} = \frac{56}{\sin 70^0} = 60 м.$$

2. The horizontal capacity of the deposit will be determined by the formula:

$$P = 2(L_д + B_д) = 2(2000 + 40) = 4080 \text{ м}.$$

3. The area of the bottom of the quarry will be determined by the formula:

$$S = L_д * B_д = 2000*40 = 80000 м^2.$$

4. The final depth of the quarry will be determined by the formula:

$$H_{к.к.} = \frac{-P + \sqrt{P^2 - 4\pi[S - m_г L_д (1 + k_{гр})]}}{2\pi ctg\beta_н}.$$

$$H_{к.к.} = \frac{-4080 + \sqrt{4080^2 - 4 \cdot 3{,}14[80000 - 60 \cdot 2000(1 + 9{,}2)]}}{2 \cdot 3{,}14 \cdot ctg 39^0} = 203 м.$$

5. The distance from the bottom of the quarry to the recumbent side of the deposit will be determined by the formula:

$$x = \frac{(m_г - B_д) \cdot (tg\beta_з - tg\beta_н)}{2tg\beta_з}.$$

$$x = \frac{(60 - 40) \cdot (2{,}7 - 0{,}8)}{2 \cdot 2{,}7} = 7 м.$$

6. The volume of mineral reserves will be determined by the formula:

$$V_и = m_г * L_д (H_н - h_н) - (S_1 + S_2) * L_д$$

$$S_1 = \frac{(m_г - x - B_д)^2 tg\beta_з tg\beta_н}{2(tg\beta_з + tg\beta_н)}.$$

$$S_1 = \frac{(60 - 7 - 40)^2 \cdot 2{,}7 \cdot 0{,}8}{2(2{,}7 + 0{,}8)} = 52{,}7 м^2.$$

$$S_2 = \frac{x^2 tg\beta_з tg\beta_н}{2(tg\beta_з - tg\beta_н)}. \qquad S_2 = \frac{7^2 \cdot 2{,}7 \cdot 0{,}8}{2(2{,}7-0{,}8)} = 28{,}1 м^2.$$

$$V_и = 60 * 2000\,(203 - 18) - (52{,}7 + 28{,}1) * 2000 = 22038000 \text{ м}^3.$$

7. The balance reserves of minerals, which we assume to be equal to geological reserves, will be determined by the formula:

$$З_б = З_г = V_и * \rho_и = 22038000 * 2{,}8 = 61708000 \text{ т}.$$

8. Industrial mineral reserves (we assume operational losses equal to 4%) will be determined by the formula:

$$З_п = 0{,}96 * З_б = 61708000 * 0{,}96 = 59239000 \text{т}.$$

9. The volume of rock mass in the contours of quarries (we take βsr = βn) will be determined by the formula:

$$V_{г.м} = S * H_к + \frac{1}{2} P * H_к^2 ctg\beta_{ср} + \frac{1}{3}\pi H_к^3 ctg^2\beta_{ср}.$$

$$V_{г.м} = 80000 * 203 + \frac{1}{2} 4080 * 203^2 * 1{,}24 + \frac{1}{3} 3{,}14 * 203^3 * 1{,}24^2 = 129728000 \text{ м}^3.$$

10. The volume of overburden in the final contours of the quarry will be determined by the formula:

$$V_в = V_{г.м} - V_и = 129728000 - 22038000 = 107690000 \text{ м}^3.$$

11. The average industrial stripping coefficient is determined by the formula:

$$k_{ср} = \frac{V_в}{З_п} = \frac{107690000}{59239000} = 1{,}82 \text{ м}^3/\text{т}.$$

Initial data for solving the problem:

вар.	m,м	βз,град	Bд, м	Lд, м	kгр,м^3/м^3	βн,град	ρи,т/м^3	hн,м
1.	50	65	50	250	8,9	41	2,4	20
2.	40	69	52	1500	8,1	39	2,7	18
3.	60	65	50	250	8,1	39	2,7	20
4.	45	70	40	2000	9,2	41	2,8	17
5.	68	69	45	1800	11,2	38	2,1	15
6.	65	45	50	1250	8,8	42	3,1	10
7.	60	49	52	1000	8,2	39	2,8	18
8.	62	52	50	250	8,1	39	2,7	20
9.	45	62	40	2000	9,2	41	2,8	17
10.	68	59	45	1600	6,2	39	2,3	15

PRACTICAL WORK No. 3

Construction and volume of capital trenches

Trenches open-pit mine workings of considerable length with relatively small transverse dimensions are called. According to their purpose, they are divided into capital and split. Capital trenches provide access from the surface of the earth to the deposit, and split trenches create a front of work for the excavation of minerals or overburden

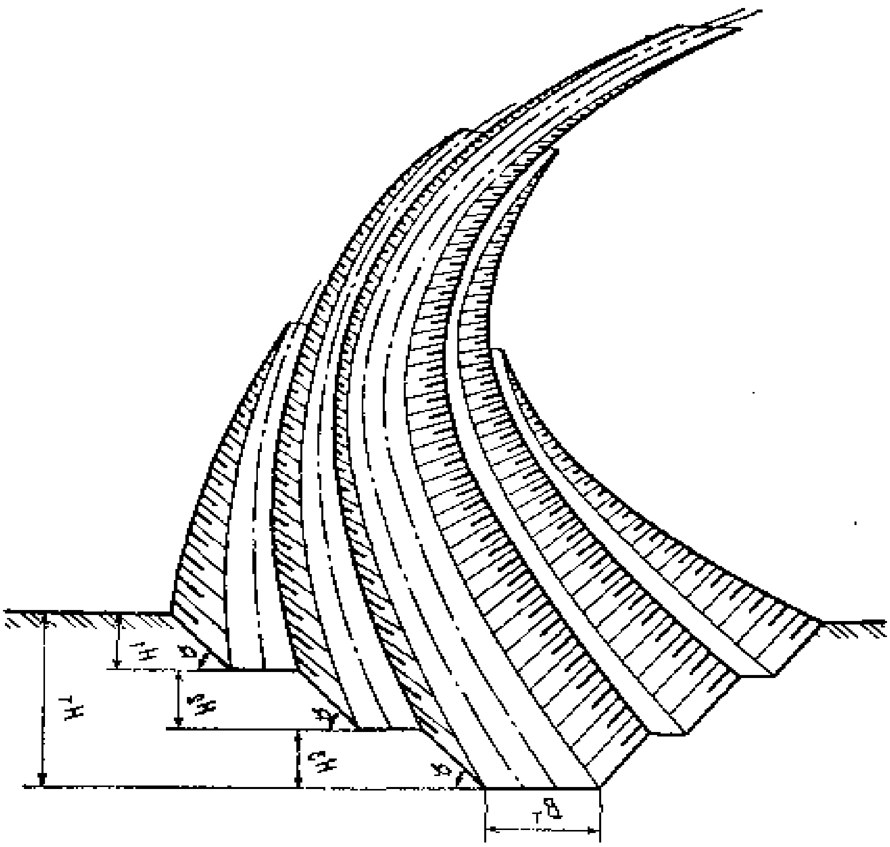


Fig.1. General view of the capital trench: - width of the trench bottom; - the angle of the slope of the side of the trench; – the depth of the trench; , - the height of one ledge

The main parameters of the trench are: the size and shape of the cross-section, the longitudinal slope, depth and length (Fig. 1).

Capital trenches located on a flat surface have a trapezoidal cross-section, and trenches located on a slope often have a cross-section shape approaching a triangle. Capital trenches, opening several ledges, have a complex stepped shape.

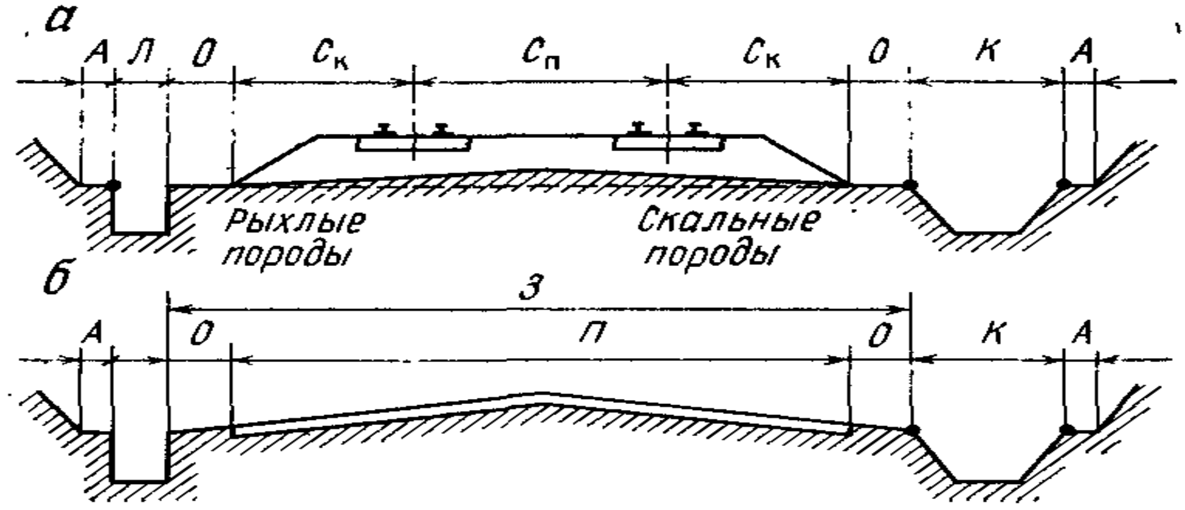


Fig.2. Cross—section of the capital trench:

a - for rail transport; b - for road transport

The angles of the slopes of the sides of the capital trenches depend on the degree of stability of the rocks. In strong rocks they are taken to be equal to 65-75 °, and in fractured rocks — 55-60 °, in sedimentary rocks (sandstones, limestones, mudstones) - from 35 to 55 °, and in loosely connected clay — from 25 to 40 °.

The width of the trench below (Fig.2) is determined by two conditions: the design and size of the transport path and the safe location of the equipment during the trench.

The width of the trench below in railway transport includes the following elements: A — sawn—off, K — ditch, O—platform for installing supports, P — roadway, 3 - roadbed, L — tray.

The width of the cut-off A in loose rocks is 1 m, in rock - 0.5 m; the width of the cuvette on top in loose rocks is 1.65 m, in rock 1 m. The contact support occupies a platform 0.4 m wide and is located at a distance of Sc = 3.7 m from the axis of the nearest path. The distance between the axes of the Sp tracks varies from 4.1 to 5.3 m, depending on their number.and the loading capacity of the dumpcar. For dumpcars with a load capacity of 180 tons or more, the distance between the axes of the tracks in the trench is: for double—track lines — 5 m, for multi-track lines - 5.3 m. The minimum width of the trench bottom for a double-track railway track is 14-15 m.

In road transport between the ditch and the carriageway, roadsides with a width of 0.5—1 m are left. The width of the carriageway P varies from 4.5 to 20 m depending on the number of traffic lanes and the load capacity of dump trucks. With two-lane traffic and a load capacity of more than 15 million tons per year, it is 12.5; 15.5; 17 and 20 m for dump trucks with a load capacity of 40-45, 65-75, 100-120 and 160-180 tons, respectively.

In climatic zones with heavy snowfall, the width of the trenches must be increased by 7-10 m to create backup snowplow lanes and for temporary storage of snow.

The cross-section of the trench should ensure the safe presence of excavators and other equipment used in trench sinking in it, and the possibility of placing an oversized during excavation.

For the productive operation of the excavator, it is necessary that there is a gap of 1 — 1.5 m between its body and the side of the trench.

When carrying out a trench with loading of rock into vehicles, its width is often increased to 25-30 m, which makes it possible to organize the circular movement of dump trucks in the trench and increase the productivity of the excavator by 25-40% compared to the dead-end scheme of maneuvering machines.

The magnitude of the slope of the capital trench depends on the magnitude of the slope of the freight transport tracks, which is limited by the requirements of safe movement of transport vessels, as well as the technical capabilities of locomotives. The maximum permissible slopes of capital trenches, depending on the type of transport, are: with electric traction 0.04; with traction units 0.06; with motor transport 0.9— 0.12; with conveyor transport — 16-18 °.

However, the use of the maximum permissible slope is not in all cases the most economical, since with an increase in the slope, the useful mass of the train decreases and, consequently, the number of wagons in the composition decreases, which leads to an increase in transportation costs. At the same time, with an increase in the slope of the capital trenches, their volume decreases, the length of the tracks decreases and the distance of transportation decreases. The optimal way is such a slope, in which the total capital and operating costs for transporting the rock mass will be minimal.

Length of a simple inclined trench

$$L = \frac{H_т}{i}$$

where $H_т$ — the final depth of the trench, м; i — trench slope.

With a flat surface, the volumes of capital and split trenches are calculated as volumes or the sum of volumes of regular geometric bodies.

The volume of a simple capital trench V (m3) can be represented as the sum of the volume of a semi-prism V and the volumes of two pyramids *V*2 (рис. 3):

$$V_т = V_1 + 2V_2$$

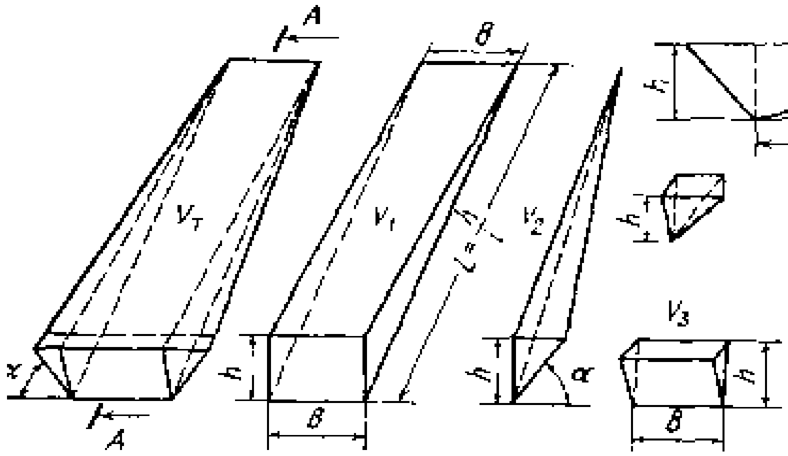


Fig. 3. Elements and parameters of the capital trench

The volume of rock at the end of the trench, enclosed in a V3 prism and two pyramids V_1 insignificant and usually not taken into account in the calculations. Half the volume of a half-prism *Vi,* at the base of which is a rectangle with an area of *bh,* and the height is equal to the length of the trench *L* = *h*/*i*,

$$V_1 = \frac{bh^3}{2l}$$

Pyramid Volume V_2, at the base of which there is a triangle with an area of $\frac{h^2}{2tg\alpha}$

$$V_2 = \frac{h^3}{6tg\alpha}$$

After converting the formula

$$V_1 = V_1 + 2V_2$$

объем траншеи

$$V_1 = \frac{h^2}{i}(\frac{b}{2} + \frac{h}{3tg\alpha})$$

The volume of a steep trench is calculated using the same formula, but instead of i, the natural value of the sine of the slope angle of the trench is entered.

The design of an external complex trench (general or group), revealing several horizons, can be different depending on the type of its adjacency to the side of the quarry, the system of transport communications, the number of transport platforms. The trench can have one- or two-sided abutment, common or separate exits (Fig.4). The height of the ledges and the width of the IT platforms, as well as the magnitude of the slopes on different horizons can be different.

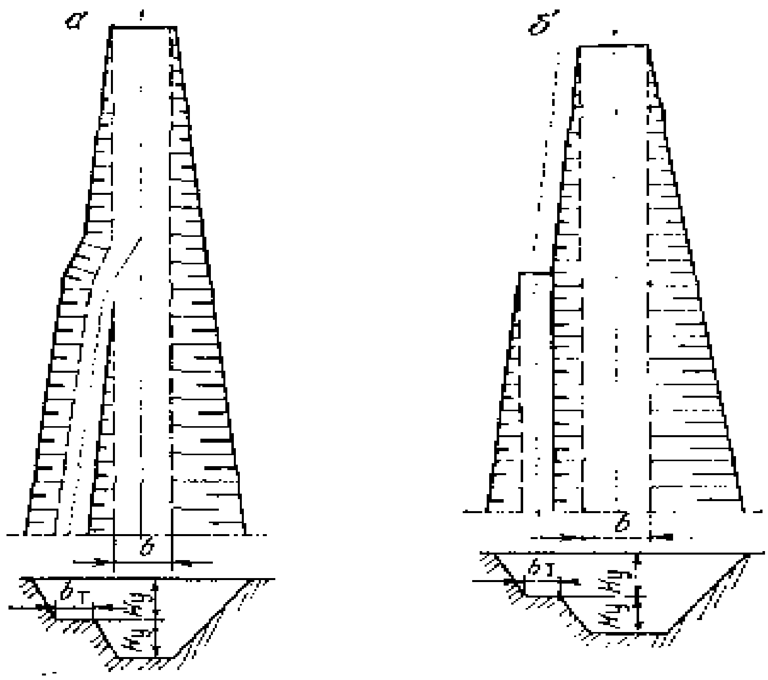


Fig. 4. Schemes of a common trench that reveals two horizons:

a — with one-sided abutment and a common exit; b - with one-sided abutment and separate exits

The volume of a steep trench is calculated using the same formula, but instead of i, the natural value of the sine of the slope angle of the trench is entered.

The design of an external complex trench (general or group), revealing several horizons, can be different depending on the type of its adjacency to the side of the quarry, the system of transport communications, the number of transport platforms. The trench can have one- or two-sided abutment, common or separate exits (Fig.4). The height of the ledges and the width of

the platforms, as well as the magnitude of the slopes on different horizons can be different.

The volume of an external deep trench of complex construction can be determined by the formula as the sum of individual geometric bodies. The formulas in this case turn out to be very cumbersome, and the calculation results are not reliable enough.

In reality, trenches are not regular geometric bodies. Therefore, for more accurate calculations, the method of vertical parallel sections is used (Fig.5). At the same time, the volume of the trench.

$$V_0 = \frac{(S_1 + S_2)l_1 + (S_2 + S_3)l_2 + \cdots (S_{n-1} + S_n)l_n}{2}$$

where S_b S_2,... ,$S_{,,}$ — cross-sectional areas м; l_1, 1_2,..., l_n — the distances between the corresponding sections, м.

The accuracy of the volume calculation by this method is greater the smaller the distance between the sections.

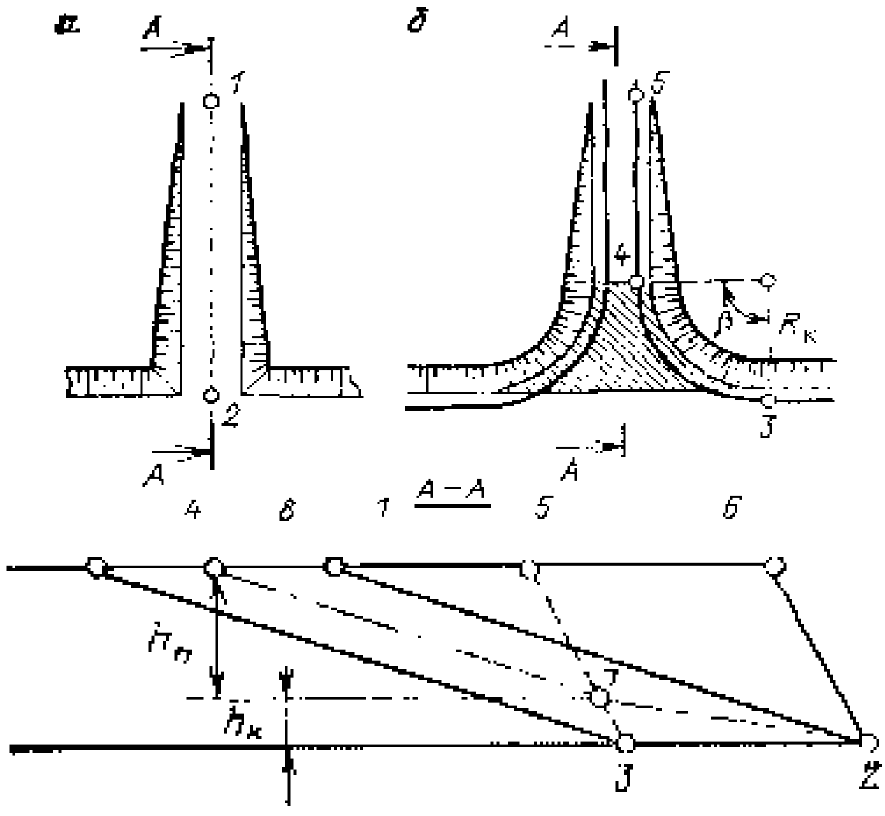


Fig. 5. Schemes of joining the capital trench to the horizon

Approximately the volume of a deep trench of complex construction can be determined by the formula of the volume of a simple trench, replacing its complex transverse contour with straight lines.

With large curve radii and a large trench depth, it is mandatory to take into account the side spacing at the point of the trench's abutment to it.

In the case of joining at right angles (Fig. 5a), the bottom of the trench in the longitudinal section is located on a straight line 1-2. If the abutment is carried out along a curve located at the level of the abutment site (Fig.5 b), then the bottom of the trench in the longitudinal section is indicated by a straight 3-4. In this case, the volume of the trench is much larger due to the additional spacing of the side, which is shaded on the plan, and on the longitudinal section is indicated by the numbers 2, 3, 5, 6.

In the case of joining along a curve located on an inclined section, the bottom of the trench can be represented by a polyline *2*—7—*8*.

Curve height (м)

$$h_k = l_k i_k$$

where i_k — the slope of the paths on the curve, ‰.

Curve length (м)

$$l_k = \frac{\pi R_k \beta}{180}$$

where β — angle of rotation of the curve, degree.

Knowing the height of the curve h_K, you can determine the height of a straight segment h_n and its length $l_п$, that is , the segment *7—8:*

$$h_n = h - h_k$$

$$l_n = \frac{h_n}{i_p}$$

Where h — total height of the ledge, м; i_p — leadership bias, ‰.

Radius of the curve $R_к$ it is accepted depending on the type of transport, and the slope of the curve for railway transport (‰)

$$i_k = i_p - \frac{700}{R_k}$$

Knowing the basic parameters of the trench i_k, $h_к$, $R_к$ и β, you can build a trench plan and profile and then determine the trench volume fairly accurately, taking into account the abutment.

The slope on the section of the curve of the highway is less than by 25—30 % and, as a rule, should not exceed 0,045.

As can be seen from Fig. 5, when the curve is adjacent on the rise, the additional side of the different side (wheelbarrows 2, 7, 5, 6) is less than when adjacent on a horizontal platform.

example. Determine the volume of an inclined trench with a depth of h = 15 m with a bottom width of b = 20 m. Trench slope i = 0.08, slope angles of the sides 40°.

$$v_t = \frac{152}{0.08}\left(\frac{20}{2} + \frac{15}{3 \cdot 0.84}\right) = 45 \text{ тыс. м}^3$$

The dimensions and volumes of deep (up to 100-160 m) trenches of complex construction are very significant and reach: a width of 200-400 m on top, a length of 2-2.5 km, a volume of several million. m3 and even tens of millions of m3. The additional volume of the separation of the side of the quarry in the place where such a deep trench adjoins it is also very large.

Under these conditions, the task of optimizing the design and parameters of the trench becomes of significant economic importance, which, due to the complexity and multivariance of calculations, cannot be solved without using a computer. Thanks to CAD, it becomes possible to choose such an option for the construction or reconstruction of a trench, in which a significant (tens and hundreds of thousands of m3) reduction in the volume of the trench and improvement of the parameters of the route of the tracks, leading to a reduction in operating costs for transport, is achieved.

Initial data for work.

	Глубина траншеи *h, м*	**Ширина дна траншеи *b*, м**	**Уклон траншеи *i*,**	**Углы откосов бортов α, *град***
1	16	22	0,08	50
2	13	18	0,09	48
3	14	20	0,08	49
4	18	24	0,07	45
5	15	20	0,09	48
6	18	20	0,08	46
7	16	24	0,12	40
8	15	25	0,1	45
9	18	20	0,1	49
10	15	22	0,08	40

PRACTICAL WORK NO. 4
CALCULATION OF PARAMETERS AND INDICATORS OF THE SPLIT TRENCH PENETRATION

Let's consider an example of calculating the parameters and indicators of the penetration of a split trench through rocks by a dead-end face with bottom loading into vehicles and plotting the organization of tunneling work.

Sinking conditions: rock strength coefficient f = 12÷13; height of the ledge being prepared for mining H_y = 15 м.

Parameters of the trench cross-section: width of the lower base $B_т$ = 28 м; depth $H_y = H_т = 15$ м; slope angle of the sides α = **70°;** trench cross-sectional area

$$S_т = H_т(B_т + H_т ctg\alpha) = 15(28 + 15ctg70^0) = 502 \text{ м}^2$$

Drilling equipment: drilling rig СБШ-250МНА-32; excavator ЭКГ-8И; dump trucks БелАЗ-549 load capacity 75 т.

Rock breaking — vertical wells, solid core charges. Diameter of blast wells d_c = 250 мм. Specific consumption of explosives (granulotol, grammonite 50/50B) $q_n = 0{,}8$ кг/м^3.

Working hours: three 8-hour shifts per day, continuous working week.

Calculation of the parameters of blasting operations. The distance between the longitudinal rows of wells in the rocks at $H_т$= 15 м, d_c = 250 мм and the adopted BB pa on the basis of practical data is within 6—8 мм. We accept 6 = 7 м. Then with the width of the trench base $B_т$ = 28 м number of longitudinal rows of wells in blast blocks

$$n = \frac{B_T}{b} + 1 = \frac{28}{7} + 1 = 5$$

Blasting is carried out in long blocks using a vertical longitudinal log. As a cut-in, we take the middle row of wells, which explodes in the first place. Then with short slowdowns **(20—35** мс) the blast ores explode in pairs in two steps. The depth of drilling of cut-in wells is 1 m more than the jackholes.

Depth of jackholes with overburden

$$L_c = H_T + l_n = 15 + 5 = 18 \text{ м}$$

vrubovs – 19 м.

Length of the charge column

$$\boldsymbol{l_{зар} = L_c - l_{заб} = 18 - 5 = 13} \text{ м}$$

врубовых – *19-5=14 м 0*

where $\boldsymbol{l_{заб}}$— the length of the face; for these blasting conditions $\boldsymbol{l_{заб}}$= 5 м.

The mass of charges in the bump rows, respectively

$$Q_{з.о.} = p \cdot l_{заб} = 49 \cdot 13 = 638 \text{ кг}$$

$$Q_{з.в.} = p \cdot l_{заб} = 49 \cdot 14 = 686 \text{ кг}$$

where p — specific capacity of the well, кг/м. При d_c = 250 мм and charging density Δ= 0,9 meaning p— 49 кг/м.

Average charge mass

$$Q_{з.с.} = \frac{638 \cdot 4 + 686}{5} = 646 \text{ кг}$$

The volume of rock per charge

$$V_{1зар} = \frac{Q_{з.с}}{q_n} = \frac{646}{0.8} = 807 \text{ м}^3$$

Also, for one transverse row of wells

$$V_{5зар} = 807 \cdot 5 = 4035 \text{ м}^3$$

Distance between wells in longitudinal rows

$$\text{a} = \frac{V_{5зар}}{S_T} = \frac{4035}{502} = 8.03 \text{ м}$$

We accept a=8 м.

Thus, the grid of the location of blast wells was determined:

$$a \cdot b = 8 \cdot 7 \text{ м.}$$

The output of the blasted mass from 1 m of the well

$$v = \frac{V_{5зар}}{L_{5ск}} = \frac{4035}{18 \cdot 4 + 19} = 44{,}4 \text{ м}^3/\text{м}$$

Excavator performance on loading in the face, the speed of trench penetration.

Technical performance

$$П_{э.т.} = \frac{3600}{t_ц} Е \frac{k_н}{k_р} k_{сн} = \frac{3500}{36} \cdot 8 \cdot \frac{0,75}{1,5} \cdot 0,7 = 280 \text{ м}^3/\text{час}$$

where $k_{сн}$— the coefficient of reduction in the productivity of the excavator in a dead-end face due to the deterioration of downhole conditions compared to the performance on side loading when working off ledges; $k_{сн}$ **≈0,7.**

Operational replaceable performance of the excavator

$$П_{э.см.} = П_{э.т.} Т_{см} k_и = 280 \cdot 8 \cdot 0,75 = 1680 \text{ м}^3/\text{смену}$$

where $k_и$— the coefficient of use of the excavator during loading during the shift; when working with vehicles in a dead-end face $k_и$**=0,7÷0,75.**

The daily productivity of the excavator $П_{э.сут.} = \mathbf{5050}\ {}^{м^3}/_{сут}$

Monthly performance $П_{э.мес.} = \mathbf{136400}\ {}^{м^3}/_{мес}$

Trench penetration rate

$$V = \frac{П_{э.мес}}{S_г} = \frac{136400}{502} = 272\ {}^{м}/_{мес}$$

Required fleet of drilling rigs. For timely preparation of blocks for explosion at the rate of trench penetration 272 monthly volume of drilling operations

$$\varepsilon L_{с.мес} = \frac{П_{э.мес}}{\vartheta} = \frac{136400}{44,4} = 3075м$$

Monthly productivity of the drilling machine

$$П_{с.мес} = П_{с.см} n_{см} \cdot 27 = 50 \cdot 3 \cdot 27 = 4050\,{}^{м}/_{мес}$$

where $П_{с.см}$ — replaceable machine performance. The productivity of the adopted machine is 50 м/смену

Required number of working drilling rigs

$$N_{c.p} = \frac{\varepsilon L_{с.мес}}{П_{с.см}} = \frac{3075}{4050} = 0,76$$

Taking into account the reserve coefficient, the list fleet

$$\boldsymbol{N_{c.c} = N_{c.p} \cdot 1.25 = 0.95}$$

We accept one machine.

Thus, one drilling rig is sufficient to perform the required monthly volume of drilling operations.

INITIAL DATA

	Глубина траншеи *h, м*	**Ширина дна траншеи *b*, м**	**Углы откосов бортов α, *град***	**Диаметр скважины, *мм***	**Удельный расход ВВ , кг/м^3**
1	15	28	70	250	0,8
2	12	24	65	240	1
3	14	25	70	250	1,2
4	20	30	72	250	1
5	18	25	65	250	0,8
6	15	20	60	240	1,1
7	12	22	66	250	0,9
8	17	24	70	250	0,8
9	20	30	70	240	1
10	18	22	72	250	0,7

PRACTICAL WORK NO. 5

Determination of the rate of deepening of mining operations at the quarry.

In the case of deep mining systems in the conditions of mining of inclined and steep deposits of deep, upland or mixed type, overburden by means of transport is usually moved to external dumps. the placement of a part of the overburden on internal dumps is possible in special cases (for example, when working out synclinal folds to full depth or an elongated deposit from the flank). on elongated inclined deposits, the development of mining operations is carried out in the direction from the lying side to the hanging side. the route of the capital trench is located permanently in the rocks of the recumbent side. in this case, a longitudinal development system is used. in steep deposits, the development of mining operations begins at the output of minerals under sediments in order to reduce the volume of mining and construction work. with elongated quarry fields, it is necessary to perform a certain amount of mining work on the overlying horizon in order to open and prepare the next horizon in depth $V_{р.т}$ и $V_о$ for carrying out inclined and split trenches and expanding it to form a working platform, the width of which is $Ш_{р.п}$ must be at least minimal

With a large length of the quarry field, calculations can be attributed to 1 m of stretch. Then the specified volumes of work (м^3/м):

With a longitudinal single - board development system

$$V = V_{р.т} + V_0 = H_у \left[b_{р.т} + Б_т + Ш_{р.п} + 1{,}5 H_у \left(ctg\alpha + ctg\alpha_1 \right) \right]$$

With a longitudinal two-board development system

$$V = V_{рТ} + V_0 = H_у \left[b_{рТ} + Б_Т + 2Ш_{р.п} + 2H_у \left(ctg\alpha + ctg\alpha_1 \right) \right]$$

where $b_{р.т}$ – width of the bottom of split trenches, м;

$Б_т$ – width of the transport berm, м;

α и α_1 – slope angles of working and non-working ledges, respectively, degree.

When preparing the horizon along its length simultaneously by several excavators, when the length of the excavator block is equal to $L_б$ (м) and the productivity of the excavator during tunneling is equal to $Q_{эг.}$ (м3/год), total preparation time of the ledge height $H_у$ will be (years)

$$T_п = \frac{V * L_б}{Q_{э.г}}, \text{ лет}$$

and the rate of deepening of mining operations (m/year)

$$V_{уг} = \frac{H_у}{T_п} = \frac{Q_{э.г} * H_у}{L_б * V}, \text{ м/год}$$

Thus, with a single-board system, the development of inclined deposits

$$V_{уг} = \frac{Q_{эг}}{L_б \left[b_{рТ} + Б_Т + Ш_{р.п} + 1{,}5 H_у \left(ctg\,\alpha + ctg\,\alpha_1\right)\right]}$$

and with a two - port system of development of a steep deposit

$$V_{уг} = \frac{Q_{эг}}{L_б \left[b_{рТ} + Б_Т + 2Ш_{р.п} + 2H_у \left(ctg\alpha + ctg\alpha_1\right)\right]}$$

Given: the field development system is longitudinal double-sided; the average length of the work front on the ledge $L_ф$=3000 м; two excavators work on each ledge ($n_э$) with annual productivity $Q_{э.г.}$ – 2,0 млн. м^3; width of the bottom of the split trench $b_{р.т.}$ =30 м; width of the working platform on the ledge $Ш_{р.п.}$=50 м; height of the ledge being developed $H_у$=15 м; the angles of the slopes of the sides of the working ledges from the hanging and lying sides of the deposit are respectively equal $a_{рв}=a_{рл}=75°$.

Decision:

The rate of deepening of mining operations:

$$V_{уг} = \frac{Q_{э.г}}{L_б \cdot \left[b_{р.м.} + 2 \cdot Ш_{р.п.} + 2 \cdot H_у \cdot (ctg\alpha + ctg\alpha_1)\right]} =$$

$$= \frac{2000000}{1500 \cdot \left[30 + 2 \cdot 50 + 2 \cdot 15(ctg75^0 + ctg75^0\right]} = 9{,}1 \text{ м.}$$

where $L_б$ – length of the excavator block, м;

$$L_б = \frac{L_ф}{n}$$

Initial data for solving the problem

№ вар.	$L_ф$, м	$Q_{э.г.}$, млн. м^3	$b_{р.т.}$, м	$H_у$,м	$n_э$, шт	α	α_1
1	3000	2,0	25	15	2	75	60
2	3200	2,5	30	17	3	60	73
3	3500	2,4	28	20	3	75	78
4	3600	1,5	32	21	2	70	65
5	2800	1,6	30	22	2	63	75
6	3200	2,5	30	17	3	60	73
7	3500	2,4	28	20	3	75	78
8	3600	1,5	32	21	2	70	65
9	2800	1,6	30	22	2	63	75
10	3500	2,4	28	20	3	75	78

PRACTICAL WORK NO. 6

Determination of the maximum height of the working area during the development of a steeply falling deposit.

The development of inclined and steep deposits is carried out using technological complexes, which are based on the movement of rock mass by vehicles. Technological complexes with the use of wheeled modes of transport have received the greatest use in quarries.

Given: Width of the quarry bottom $Ш_д$ = 30 м;

career depth $H_к$ = 240 м;

the angle of repayment of the sides of the quarry β = 400 ;

$\gamma_в = \gamma_л$ =180 – the angles of the slopes of the hanging and recumbent working sides of quarries; the height of the ledge.

$H_у$ = 15 м.

Decision:

Height of the working area: $$H_{р.з.} = \frac{B - Ш_д}{ctg\gamma_в + ctg\gamma_л}$$

where B – design width of the quarry on the surface, м;

$$B = Ш_д + 2H_k * ctg\beta = 50 + 550 = 600 \text{ м};$$

$$H_{р.з.} = (600-30)/(3{,}08+3{,}08) \approx 90 \text{ м}$$

that is, the number of working ledges will be equal to 90/15=6.

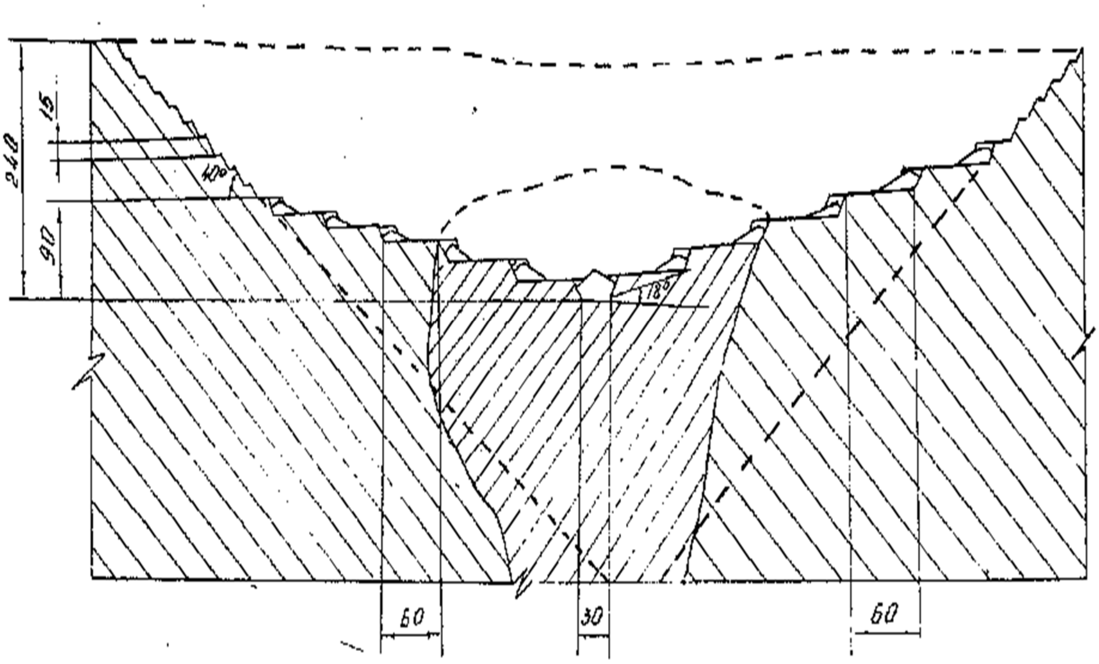


Fig.6.1. Scheme for determining the working area of the Nrz Career

Initial data for solving the problem

№ варианты	$H_к$	$H_у$	β	$Ш_д$	$\gamma_в = \gamma_л$,
1	200	15	40	30	18
2	210	15	47	35	16
3	220	17	45	40	15
4	230	17	50	45	17
5	240	20	55	50	18
6	250	20	60	30	16
7	260	22	65	35	19
8	270	22	40	40	20
9	280	15	47	45	18
10	290	17	45	50	17

PRACTICAL WORK NO. 7
DETERMINATION OF THE SLOPE ANGLE OF THE SIDES OF THE QUARRY

The angles of the slopes of the sides of the quarry at the time of repayment of mining operations are determined by the design of the sides and the conditions of stable equilibrium of the rocks composing it. Structurally, the sides of the quarry may include ledge slopes, the number of ledges being developed, safety and transport berms, the bases of capital trenches (Fig.1).

The slope angle of the non-working side of the quarry (deg.) is determined by the formula:

$$tg\gamma_{н} = \frac{H_y}{\sum H_y ctg\alpha + \sum b_п + \sum b_т + \sum b_{к.т}}$$

where: H_y - ledge height, м; α – slope angle of the ledge, angle.;

$\sum h_y ctg\alpha$, $\sum b_п$, $\sum b_т$, $\sum b_{к.т}$ - accordingly, the total width of the horizontal foundations of the slopes of the ledges of the safety berms, transport berms, the bases of the capital trenches;

$b_т$, $b_п$ – accordingly, the width of the transport and safety berm;

$b_{к.т}$ – width of the base of the capital trench.

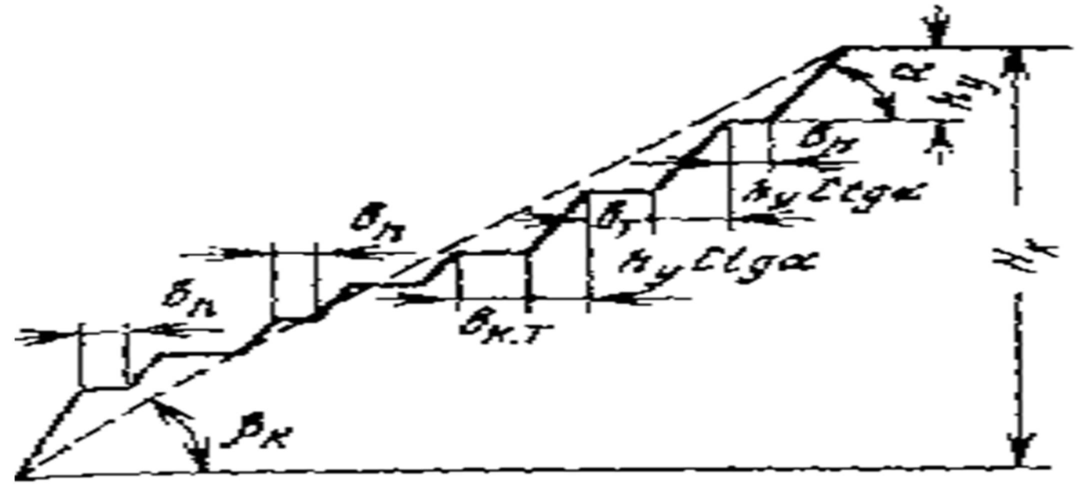


Fig. 1. Scheme for determining the slope angle of the non-working side of the quarry:

Value b_t depends on the type and intensity of the movement of quarry transport. In case of motor transport, it is accepted within 5-10 и 8-20 м and, respectively, for one and two-lane traffic. For railway transport with single-track traffic, it is 8 m, with double-track – 12,14 м.

The width of the base of the capital trenches for single- and double-track traffic is assumed to be 7.6 and 11.5 m, respectively.

The width (m) of the transport berm (Fig.2) is determined by the formula

$b_t = z + T + k$

$z = h_{y*}(ctg\alpha_e - ctg\alpha_p)$

where: z – the width of the base of the prism of possible collapse, м;

α_e – the angle of the natural slope of the ledge, м;

α_p – slope angle of the working side of the ledge, м;

T = 4÷7,5 – width of the transport lane, м;

k= 0,5÷0,7 – width of the cuvette, м.

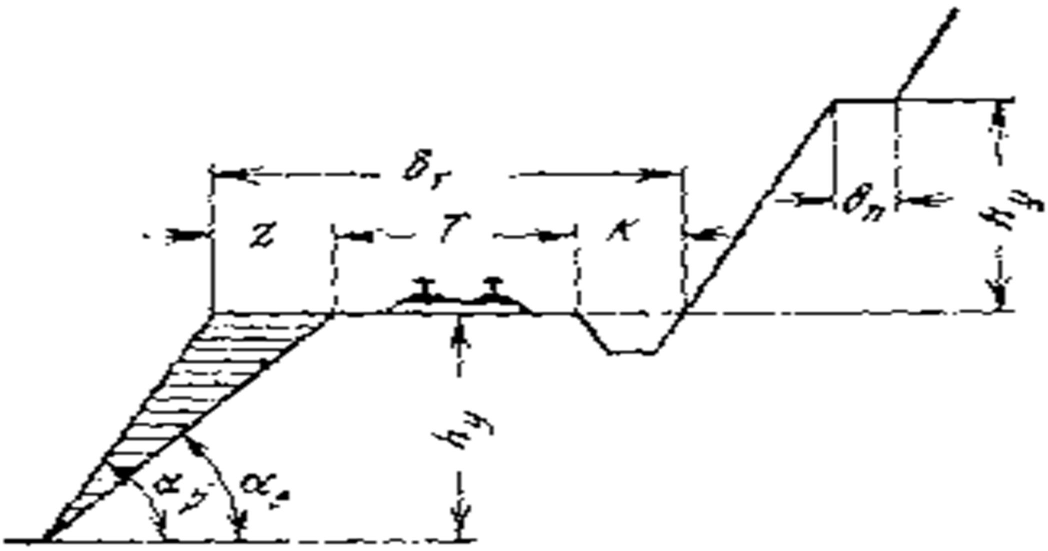


Fig.2. Scheme for determining the width b t of the transport berm.

The slope angle of the working side of the quarry (deg.) is determined by the formula (Fig. 3):

$$tg\gamma_p = \frac{H_к}{\sum H_y ctg\alpha + \sum Ш_{р.п}}$$

where: $\sum Ш_{р.п}$ - accordingly, the total width of the working platform of the ledge, м.

The maximum possible slope angle of the side of the quarry by the safety factor depends mainly on the physical and technical characteristics and the degree of uniformity of the rocks composing the side, the direction of the planes of stratification relative to the side, the depth of the quarry and the shape of the side in plan. With increasing depth, the stability of the side changes. A concave side is more stable than a flat or convex one. The slope angle of the side of the quarry also depends on the water content and filtration properties of the rocks composing the side, the time of the condition of the side of the quarry or climatic conditions. The slope angle of the side of the quarry, determined by calculation, is indicative and is specified in the course of mining operations. For approximate calculations, you can use the Giproruda data (Table 1.).

From the values of the slope angle determined by the design of the side and the conditions of stable equilibrium of the rocks composing it, the minimum value is taken, which provides the necessary degree of safety and the minimum amount of stripping work in the final boundaries of the quarry. It should be noted that a decrease in the slope angle of the side by only one degree at a pit depth of 200 m and at a slope angle of 35-450, causes an increase in the volume of stripping by 0.8-1 million m3 per 1 km of the side.

Table 1

Породы	Коэффициент крепости по шкале проф. М.М. Протодьяконова	Угол откоса борта (градусы) при глубине карьера, м				
		≤ 90	≤180	≤240	≤300	>300
В высшей степени крепкие и очень крепкие.	15-20	60-68	57-65	53-60	48-54	43-49
Крепкие и довольно крепкие	8-14	50-60	48-57	45-53	42-48	37-43
Средней крепости	3-7	45-50	41-48	39-45	36-43	32-37
Довольно мягкие и мягкие	1-2	30-43	28-41	26-39	26-36	-
Мягкие и землистые	0,6-0,8	21-30	20-28	-	-	-

Example. Determine the slope angle of the side of the quarry at the formation-like deposits (where: the depth of the quarry $H_к$ =105м; ledge height $h_у$ =15 м; α =75^0 – slope angle of the ledge; width of the transport lane T = 4÷7,5 м; width of the base of capital trenches $b_{к.т}$=7.6 м; width of safety berm ledges $b_п$=5 м; width of the cuvette k= 0,5÷0,7 м.– the angle of the natural slope of the ledge α_e= 60^0; slope angle of the working side of the ledge α_p =75^0)

Decision. Number of ledges being developed

$$n = \frac{H_k}{H_y} = \frac{105}{15} = 7;$$

Angle of slope of the side of the quarry (degrees) are determined by the formula:

$$tg\gamma_н = \frac{H_к}{\sum H_у ctg\alpha + \sum b_п + \sum b_т + \sum b_{к.т}}$$

The total width of the horizontal foundations of the slopes of the ledges

$$\sum h_у ctg\alpha = \sum 15 * ctg75^0 = \sum 15 * 0,23 * 2 = 6,9,$$

Total width of the safety berm

$$\sum b_п = \sum 5 * 3 = 15 м$$

Total width of transport berms

$$\sum b_т = \sum (5.25 + 4.25 + 0.5) * 3 = 30 м,$$

Total width of the capital trench

$$\sum b_{к.т} = \sum 7.6 * 1 = 7.6 м.$$

$$b_t = z + T + k$$

$$b_t = 5,25 + 4,25 + 0,5 = 10 м,$$

$$z = h_у * (ctg\alpha_e - ctg\alpha_р) = 15(ctg75^0 - ctg60^0) = 15(0,58-0,23) = 5,25 м$$

$$tg\gamma_н = \frac{105}{6,9 + 15 + 30 + 7,6} = 1,76, \ \gamma_н = 60^0$$

The slope angle of the working side of the quarry (deg.) is determined by the formula

$$tg\gamma_р = \frac{H_к}{\sum H_у ctg\alpha + \sum Ш_{р.п}}$$

Initial data for solving the problem

№ вар.	$H_к$, м	$\alpha_р$, град	$H_у$, м	$в_п$, м	k,	$\alpha_е$, град	$в_{к.т}$, м	$Ш_{р.п}$, м
1.	90	65	15	5	0,5	41	7,6	18
2.	100	69	15	5	0,6	39	7,6	22
3.	120	65	20	6	0,7	39	11,6	30
4.	150	70	15	6	0,7	41	11,6	35
5.	180	69	20	6	0,6	38	7,6	40
6.	200	45	22	5	0,7	42	7,6	45
7.	210	49	21	6	0,5	39	11,6	45
8.	240	52	22	7	0,7	39	11,6	40
9.	180	62	15	5	0,5	41	7,6	36
10.	170	59	15	6	0,6	39	11,6	34

PRACTICAL WORK NO. 8
DETERMINATION OF PARAMETERS OF THE TECHNOLOGICAL SCHEME OF DREDGING ROCKS BY DRAGLINES

The excavation of soft and finely torn rocks by draglines is carried out mainly in the end and trench faces (Fig. 1).

The calculation of the technological scheme for dredging rocks by draglines consists in determining the width of the entry, the height of the overburden ledge, the slope angles and the installation location of the excavator.

<u>1. The scheme of operation of the dragline in the end face when installing it on the roof of the ledge</u>

This scheme ensures maximum dragline performance. This is achieved due to the fact that the angle of rotation of the excavator (Fig. 1, a) from the place of scooping the rock to the place of its unloading is minimal.

The height of the ledge is set according to the depth of digging, taking into account the location of the dragline on the roof outside the collapse prism at the angle of inclination of the face plane to the horizon $\beta = 30 - 60^0$:

$$h \le H_{\div} \qquad (1)$$

The width of the excavator entry:

$$\grave{A}_{\acute{y}} = R_{\div}(\sin\varphi_1 + \sin\varphi_2) \qquad (2)$$

where φ_1, φ_2 - the angles of rotation of the dragline from the axis of its course, respectively, in the direction of the array and the developed space ($\varphi_1 \approx \varphi_2 = 30 - 45$), град.

Usually when working in the dump $\varphi_1 = 0$ и тогда

$$\grave{A}_{\acute{y}} = R_{\div} \sin\varphi_2 \qquad (3)$$

Slope angles of the ledge, deg:

- by deposits - working $\alpha_i = 60^0$, stable $\alpha_{i\acute{o}} = 40^0$ (4)
- according to the blasted rock - worker $\alpha_i = 50^0$,
- steady $\alpha_{i\acute{o}} = 35^0 - 40^0$, (5)
- by bedrock - worker $\alpha = 75^0$,
- steady $\alpha_{\acute{o}} = 60^0$. (6)

2. The scheme of operation of the dragline in the end face with its location on the intermediate site

They are used for the purpose of more complete use of the excavator parameters and increasing the height of the ledge (fig. 1, b).

Ledge height, м:

$$h \le h_i + h_{\hat{a}}, \tag{7}$$

Where $h_i, h_{\hat{a}}$ - accordingly, the height of the lower and upper approaches, м.

$$h_i = (0{,}7 - 0{,}8)H_k \qquad h_{\hat{a}} = (0{,}4 - 0{,}8)H_{\eth} \tag{8}$$

The width of the approach is determined by the formula (1), and the angles of the slope of the ledge according to the formulas (3) - (5). In this case, the angle of the slope of the face of the upper approach to prevent slipping of the bucket should not exceed 25 degrees.

When calculating the performance of a dragline operating according to this scheme, it should be remembered that with the upper scooping, the productivity of the excavator is 10-15% lower than with the lower scooping.

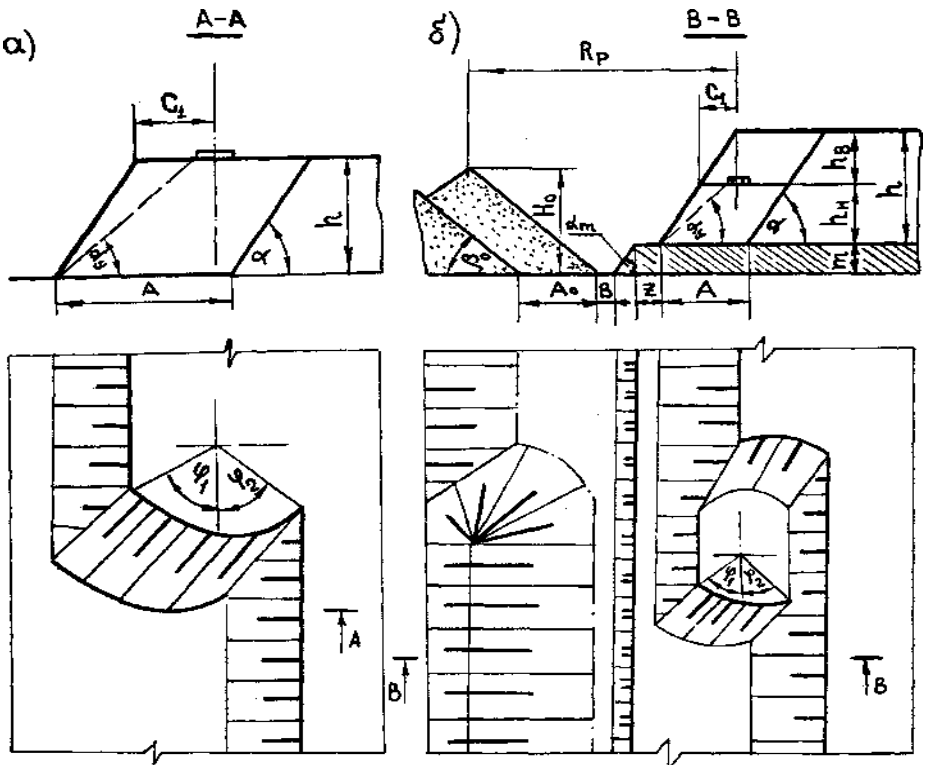
3. The scheme of operation of the dragline in the end face when installing it on the soil of the ledge

Due to the low dragline performance, this scheme is rarely used, mainly in the development of unstable rocks (1, в).

Ledge height, м:

$$h = (0{,}7 - 0{,}8)H_{\eth} \tag{9}$$

The width of the approach is determined by the formula (1), and the angles of the slope of the ledge are determined by the formulas (3) - (5).



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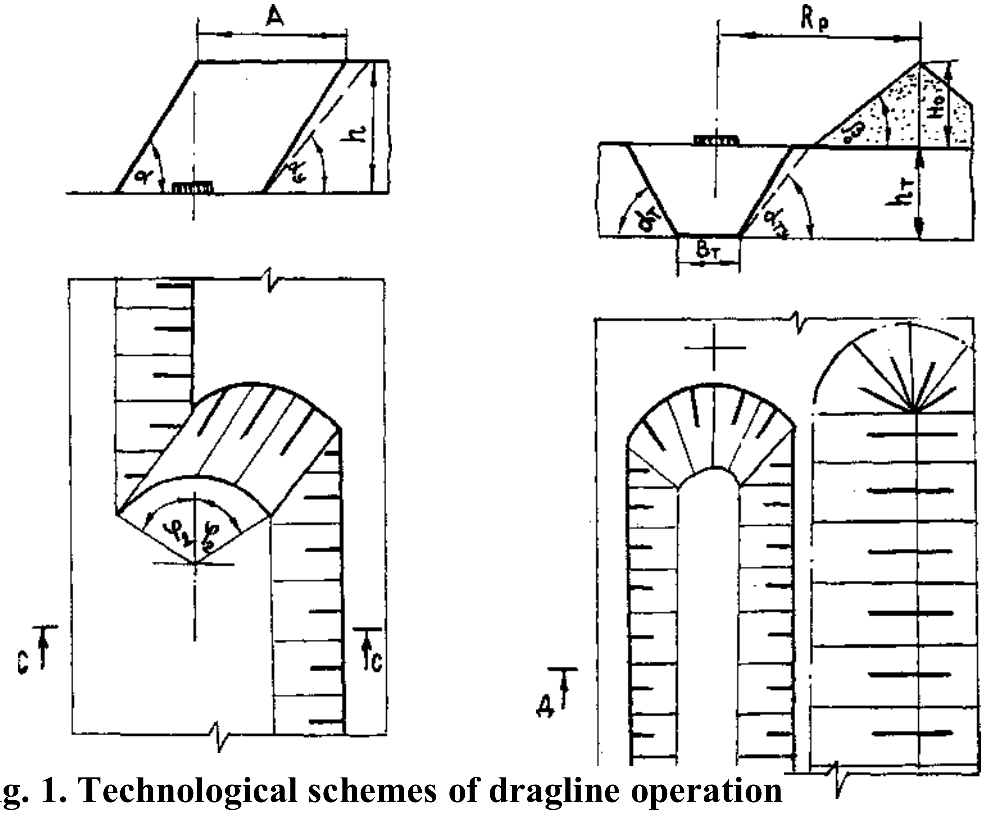

Fig. 1. Technological schemes of dragline operation

4. The scheme of operation of the dragline in the trench face when installing it on the roof of the ledge

When conducting trenches, the dragline, depending on its parameters and the parameters of the trench, is positioned either along the axis of the trench, or closer to one of its sides. In this case, the dumps can be located on either one or two sides of the trench (Fig. 1, d).

Trench depth, м:

$$h_{ò} \leq H_{ê.\max} \tag{10}$$

The width of the trench at the bottom is determined by the formula (1) or (2), and the slope angles of the trench are determined by the formulas (3) и (5).

PRACTICAL WORK NO. 9
DETERMINATION OF DUMP FORMATION PARAMETERS IN ROAD TRANSPORT

When transporting overburden rock to the dump by road, bulldozer dumping is used, which includes unloading a dump truck on the top of the dump, moving the rock under the slope of the dump, repairing and constructing highways.

There are two methods of bulldozer dumping - areal and peripheral (Fig. 1).

With the areal method of dump formation, dump trucks are unloaded throughout the dump site, then the dump area is planned and compacted with rollers. Similarly, the subsequent overlying layers are poured off. In this case, the bulldozer blade develops vertically. Due to the large amount of planning work, this method is more expensive than the peripheral one, so it is rarely used, mainly when laying soft, low-resistant rocks (see Fig. 1, b).

With the peripheral method, dump trucks with a lifting capacity of up to 75 tons are unloaded directly under the slope on stable dumps, and with a larger load capacity - at a distance of 3-5 m from the upper edge of the slope of the dump. Then this rock is bulldozed downhill, i.e. in this case, the dump develops in the plan. For safety reasons, in order to exclude the possibility of a dump truck falling from the dump during direct unloading under the slope, metal stops for the rear wheels of the dump truck are installed at the upper edge of the dump or a rock shaft with a height of at least $0,5Б_к$ ($Б_к$ - the diameter of the wheel of the largest dump truck unloading on the dump) and a width of 2-3.5 m. In addition, for the same purposes, the surface of the bulldozer dump should have a slope of 3-5o towards the center of the dump (see Fig. 1, a).

The parameters of the blade are determined in the following order.

The height of the dump on a flat surface is up to 30-60 m, in mountainous terrain - up to 150 m or more.

Dump area:

$$S_0 = \frac{V_{\hat{A}} \hat{E}_{\eth\hat{\imath}}}{H_0 K_{\grave{e}\hat{\imath}}} \quad (1)$$

where $V_в$ - the volume of rocks to be placed in the dump during its existence, м^3; K_{uo} - a coefficient that takes into account the use of the dump area (with one tier K_{uo}=0,8-0,9; with two tiers K_{uo}=0,6-0,7).

With a known area, an accepted shape and a given one of the sides of the blade, the dimensions of the blade in the plan are determined.

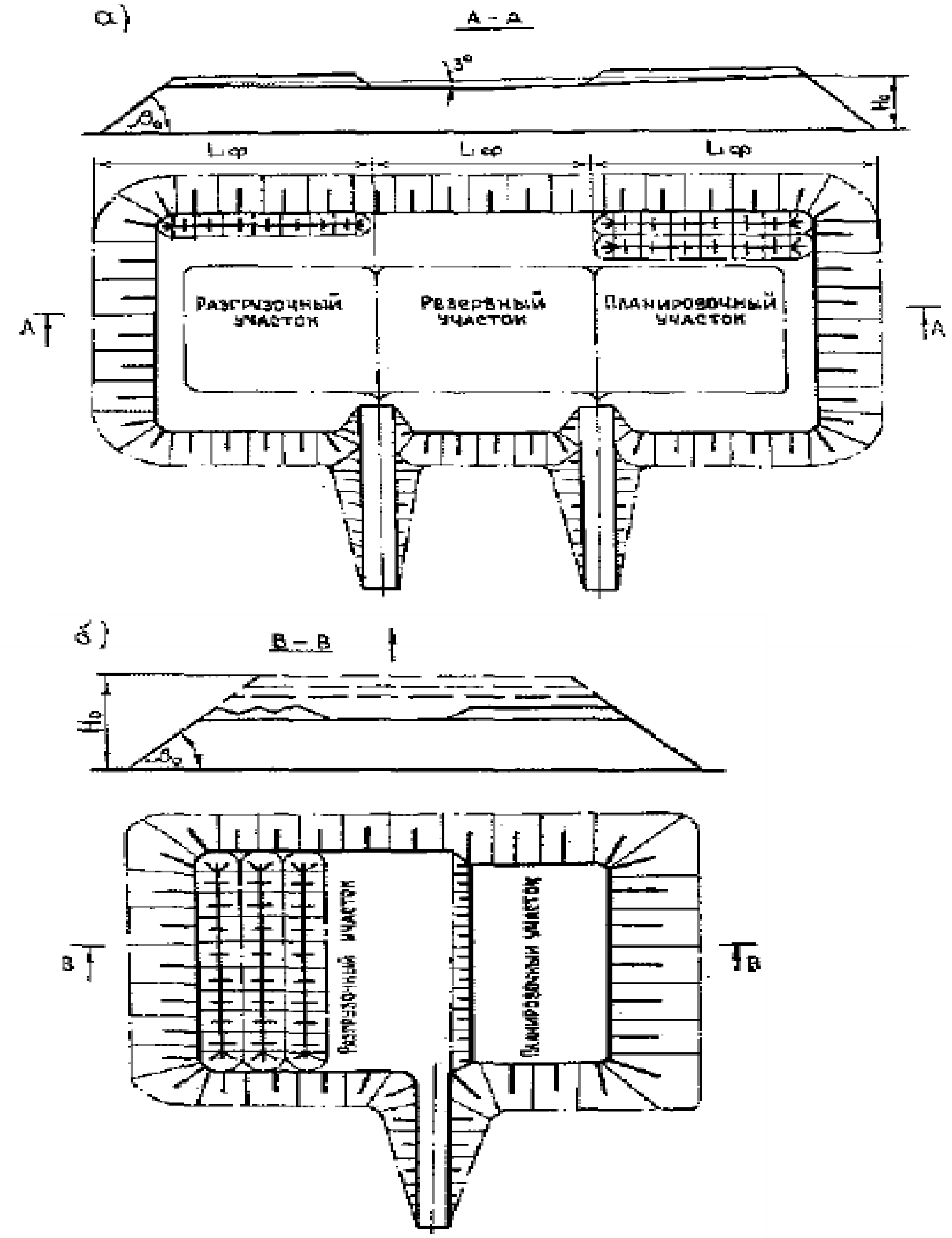

Рис. 1 . Способы бульдозерного отвалообразования:
а – периферийный; б - площадной

The average number of dump trucks unloading at the dump within one hour: $N_{ч} = \frac{V_{вч} К_{нер}}{Q_{вч}}$ (2)

where $V_{вч}$ - career performance by stripping, м^3/ч; $К_{нер}$ - the coefficient of unevenness of the work of the quarry overburden ($К_{нер}$=1,25-1,5); $Q_{ач}$- dump truck performance.

The number of dump trucks unloading at the dump at the same time:

$$N_а = \frac{N_ч t_p}{60} \qquad (3)$$

where N_a и $N_ч$ - the number of dump trucks unloading at the dump, respectively, simultaneously and within an hour; t_p - duration of unloading and maneuvering of the dump truck (t_p=l,5-2), мин.

Length of the unloading front on the dump:

$$L_ф = N_a\, l_n, \text{ м} \qquad (4)$$

where $L_ф$ - length of the dump truck unloading front on the dump, м; 1_n - the width of the strip along the front of the blade occupied by one dump truck when maneuvering ($l_п$ = 18-20), м.

The number of unloading areas of the dump that are in simultaneous operation:

$$N_{ур} = \frac{L_ф}{L_i} \qquad (5)$$

where L_i - length of one section (L_i = 60-80), м.

Number of planning plots:

$$N_{уп} = N_{ур} \qquad (6)$$

Number of reserve sites:

$$N_{у.рез} = (1{,}0 - 0{,}5) N_{ур} \qquad (7)$$

Total number of plots:

$$N_у = N_{ур} + N_{уп} + N_{у.рез} \qquad (8)$$

Total length of the dump work front:

$$L_{фо} = (60 - 80) N_у \qquad (9)$$

Annual productivity of the bulldozer:

$$Q_{б.год} = Q_{бч} К_и Т_{см}\, n_{см}\, n_{год} \qquad (10)$$

where $К_и$ - the utilization rate of the bulldozer during the shift *($К_и$=0,8-0,9);* $Т_{см}$ - shift duration ($Т_{см}$=*8*), ч; $n_{см}$ - number of shifts per day *($n_{см}$=3* or according to the mode of operation of the quarry); $n_{год}$ - the number of working days of the bulldozer per year *($n_{год}$=252); $Q_{бч}$* - hourly productivity of the bulldozer *($Q_{бч}$=300-400)*.

PRACTICAL WORK NO. 10
CALCULATION OF DUMP FORMATION PARAMETERS IN RAILWAY TRANSPORT

In railway transport, dump plows, abzetzer, bulldozers, etc. are used for storing waste rocks, but 85-90% of all volumes of stored rocks are stacked with mechlopats and draglines.

Depending on the technological properties of the stored rocks, the following schemes of operation of single-bucket excavators on the dump are used.

If the stored rocks have weak stability, a scheme of sequential dumping of dump tiers is used.

The excavator in a straight course, being located below the level of the railway tracks, carries out the filling of only the lower approach. To receive the overburden from dumpcars, the excavator constructs a receiving hopper, which moves as the podustup is poured. The length of the receiving hopper is one and a half or double the length of the dumpcar, the depth is 0.81.0 m. The train is fed to the dump by wagons forward, and dumpcars are unloaded alternately. During the reverse course, the excavator lays the rock in the upper step of the dump tier.

If the stored rocks are stable, then a scheme is used with simultaneous laying of empty rocks in the lower and upper approaches.

In this scheme, the overburden is first placed in the lower riser by the amount of the unloading radius, and then in the upper one. The technology of receiving waste rock from dumpcars and feeding wagons is similar to the above scheme. Taking into account the shrinkage of rocks in the dump, the upper approach is poured off with a height slightly exceeding the level of the railway track. After filling the dump entry, the excavator is returned to its original position and the filling of a new entry begins.

<u>The parameters of the blade are determined in the following order.</u>

The total height of the dump on a flat surface should not exceed 30-60 m, on a mountainous one - 100 m or more.

The height of the blade tier when filling it with two approaches:

$$H_0 = h_1 + h_2, \text{ м}$$

(1)

where h_1, h_2 - accordingly, the height of the lower and upper approaches, м.

$$h_5 + h_2 \le H_{p.\max} - e_0 \qquad (2)$$

where e_0 - minimum gap between the bottom of the open bucket and the dump blade $e_0 = 0.7 - 1.0$, м.

$$h_1 = (H_0 + h_5) - h_2 \tag{4}$$

$$h_5 = H_0 (K_{p1} - K_{p0}) \tag{5}$$

where h_5 - excess of the newly formed dump loading over the old one, м; K_{p1} - the coefficient of loosening of the rock poured into the dump; K_{p0} - the coefficient of residual loosening of the rock in the dump.

$$K_{p1} = (1.1 - 1.0.5) K_{ðê} \qquad K_{p0} = (1.15 - 1.0.6) K_{ð1} \tag{6}$$

The step of moving the dump tracks (width of the dump approach), м:

$$\grave{A}_0 = R_{÷} + R_p \tag{7}$$

where $R_{ð}, R_{÷}$ - accordingly, the actual unloading radius and the scooping radius, м.

The receiving capacity of the dump dead end in terms of volume in the whole between the two re-laying of the path:

$$V_0 = \frac{L_0 A_0 H_0}{K_{po}} \tag{8}$$

where V_0 - receiving capacity of the dump dead end, м^3; L_o - the length of the dump dead end (accepted according to the task), м.

The duration of operation of the dump dead end between two re-laying of the path:

$$t_T = \frac{V_0}{V_{ñóò}}, \textit{ сутки} \tag{9}$$

where $V_{сут}$ - daily intake capacity (by volume in the whole) of the dump dead end, м^3/сутки.

$$V_{ñóò} = n_c V_{ãð} \tag{10}$$

where n_c - the number of trains that can be unloaded at the dump cul-de-sac per day;

V_{zp} - the volume of rock transported by locomotives in one trip).

$$n_c = K_{ið} \grave{O}_{ñóò} (t_0 + t_{ðàçã}) \tag{11}$$

where $K_{нр}$ - a coefficient that takes into account the unevenness of the deadlock operation *($K_{нр}$= 0,85-0,95); $T_{сут}$* - the number of hours of operation of the dump dead end per day ($T_{сут}$=21), ч; t_o - train exchange time at the dump, ч; $\boldsymbol{t_{раз}}$ - train unloading time, ч.

$$t_0 = \frac{2L_{\text{îá}}}{V_0 + \tau_{\text{â}}} \tag{12}$$

where $L_{об}$ - average distance from the exchange office to the dump excavator (according to the task), км; V_o - average speed of trains on dump dead ends (*V_o = 15-20),* км/ч; $\tau_в$ - communication time (with automatic communication $\tau_в$ = 0, with telephone $\tau_в$ = 0,05-0,1), ч.

$$L_{\text{îá}} = 0{,}5L_0 \tag{13}$$

$$t_{\text{ðàçã}} = n_{\text{ä}} \tau_n \tag{14}$$

where τ_n - duration of unloading of one car (τ_n = 0,025-0,033- in the summer, τ_n = 0,05-0,07 - in winter), ч.

The number of dump dead ends in operation:

$$N_{\text{òð}} = \frac{V_{\text{âñ}}}{V_{\text{ñóò}}} \tag{15}$$

where $N_{\text{òð}}$ - number of dump dead ends in operation, pcs.; $V_{вс}$ - average daily volume of overburden entering the dump, $м^3$.

Total number of railway dead ends on the dump:

$$N_{\text{òî}} = N_{\text{òð}} (1 + \frac{t_{nt}}{t_T}) \tag{16}$$

where $N_{\text{òî}}$ - number of dead ends on the dump, pcs.; $\boldsymbol{t_{nT}}$ - the duration of the re-laying of the track on the dump dead end, day.

The productivity of excavators on the dump is 25% more than when excavating blasted rocks.

PRACTICAL WORK NO. 11

CALCULATION OF PARAMETERS OF TECHNOLOGICAL PROCESSES OF MINING OPERATIONS AT THE QUARRY

Calculation of the parameters of the excavation and loading process

Production processes of mining operations at the quarry: Opening, preparation of rocks for excavation, excavation and loading operations, transportation of rock mass, dumping. Consider the general calculation of the listed processes.

I. <u>Initial data:</u>

- name of the process flow — вскрышной ;
- name and properties of the rock — песчаник;
- compressive strength — $\sigma_{сж}$=495*10^5 Па;
- rock density — р=2660 кг/м^3;
- fracturing of rock — $d_{о.м.}$=0,4м;
- modulus of elasticity — Е=1,85*10^{10} Па;
- type and standard size of the dredging and loading equipment — ЭКГ-8;
- type of transport — автомобильный
- cargo flow length — Lтр=2 км.

<u>II. Calculation of the parameters of the excavation and loading process and drawing up a passport of the face.</u>

1 Parameters ЭКГ-8

Bucket capacity — Ек=8м^3;

Scooping radius at standing level — Rч.у.=11,9м;

Maximum scooping radius — Rч.max=18,2 м;

Maximum scooping height — Hч.max.=12,5м;

Maximum unloading radius — Rр.max.=16,3 м;

Working cycle duration — tц=28 с.

2 Performance of a single-bucket excavator ЭКГ-8

1) Theoretical performance (м^3/ч)

$$Q_{tt} = E_k \cdot v$$

$$v = \frac{3600}{t_ц} = \frac{3600}{28} = 128{,}5ч^{-1};$$

where v- number of working cycles per hour (1/ч)

2) Technical performance (м^{3}ч)

$$Q_{tt} = 8 \cdot 128{,}5 = 1028м^3/ч.$$

$$Q_t = Q_и \cdot \frac{k_и}{k_р}$$

where $k_н$ - the coefficient of filling the bucket with a mechanical shovel ($k_н$=1)
$k_р$ - the coefficient of loosening of the rock in the bucket ($k_р$ =1,4)

$$Q_t = 1028 \cdot \left(\frac{1}{1,4}\right) = 734,2 \approx 734\, м^3 / ч.$$

3) Operational productivity per shift (м^3/смену)

$$Q_{см} = Q_t \cdot T \cdot k_и;$$

where T - shift duration (T =8 часов);
$k_и$ - the utilization rate of the excavator during the shift *($k_и$ =0,7);*

$$Q_{см} 734 \cdot 8 \cdot 0,7 = 4110,4 \approx 4110\, м^3 / смену.$$

4) Operational productivity per day (м^3/сутки)

$$Q_{сут} = Q_{см} \cdot n;$$

where n- number of working shifts per day (n=3);

$$Q_{сут} = 4110 \cdot 3 = 12330\, м^3 / сут.$$

5) Operational performance per year (м^3/год)

$$Q_г = Q_{сут} \cdot N;$$

where ***N*** - the number of working days of the excavator per year, taking into account planned downtime for repairs (***N*** =305 days);

$$Q_г = 12330 \cdot 305 = 3760650 \approx 3,76 млн. м^3 / год.$$

3. Technical parameters of the face:

Ledge height h=15м;
Slope angle of the ledge α=80°;
Safe distance from the upper edge C=5м**;**
The coefficient of loosening of rock in the collapse $k_р$=l,3;
The height of the collapse from the first row of wells $h_р$=l * $H_{ч.\mathbf{max}}$.=12,5м;
The number of visits of the excavator in the collapse n^1=4;
The distance from the mass explosion to the guarded object **L**=1000м.

The results of calculations of the parameters of the dredging and loading process are shown in Fig. 1

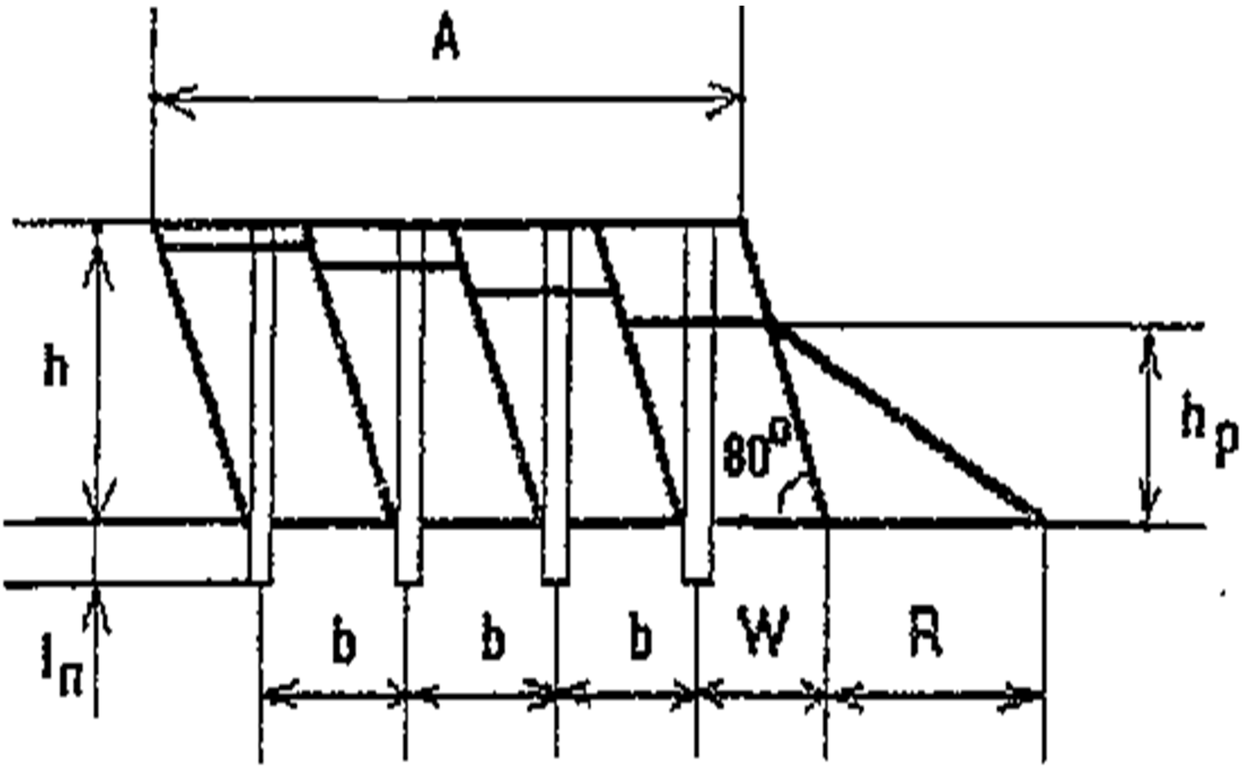

Fig. 1 Passport of the mechanical shovel face.

PRACTICAL WORK NO. 12
CALCULATION OF BVR PARAMETERS

$$B=1{,}2\sqrt[3]{E_k}=1{,}2\sqrt[3]{8}=2{,}4м.$$

1) The required composition of the rock mass by size for the excavator

$$d_{ср}=\frac{B}{6{,}5}=\frac{2{,}4}{6{,}5}0{,}37\ м\ .$$

2*) The required degree of crushing of the array*

$$n=\frac{d_{о.м.}}{d_{ср}}=\frac{0{,}4}{0{,}37}=1{,}08\ .$$

3) Dynamic coefficient for the breed $k_Д$ =1,04;

4) Specific crushing energy to the required degree

$$F_{ДР}=\frac{0{,}12\cdot\sigma^2_{сж}\cdot k^2_Д}{2\cdot E}\cdot\lg n=\frac{0{,}12\cdot(495\cdot10^5)^2\cdot1{,}04^2}{2\cdot1{,}85\cdot10^{10}}\cdot\lg1{,}08=287{,}28\approx287Дж/м^3$$

5) Specific energy of the collapse formation required by the technology

$$F_p=\left(\frac{v_0^2\cdot p}{2}\right)\cdot\left[\lg k_p+\lg\frac{(c+h\cdot ctg\alpha)\cdot(h\cdot k_p-h_p)}{2\cdot h_p}\right];$$

where *V*o - the initial velocity of the rock mass during the explosion (*V*o=10м/с);

$$F_p=\left(\frac{10^2\cdot2660}{2}\right)\cdot\left[\lg1{,}3+\lg\frac{\left(5+15\cdot ctg80^0\right)\cdot\left(15\cdot1{,}3-11\right)}{2\cdot12{,}5}\right]\approx58520Дж/м^3.$$

6) The calculated specific consumption of explosives for the fulfillment of technological conditions (кг/м³)

$$q=\frac{F_{ДР}+F_p}{\eta\cdot F_{ВВ}};$$

where η - energy efficiency factor BB (η=0,05) Type BB - grammonite 50/50-B;

$F_{ВВ}$- complete perfect blast job ($F_{ВВ}$ - 3524000 Дж/кг);

$$q=\frac{287+58520}{0{,}05\cdot3524000}\approx0{,}03кг/м^3.$$

W - the resistance line along the sole (м)

$$W=C+h\cdot ctga=-5+15\cdot ctg80^{\circ}=7,6\sim 8м$$

α - distance between wells (м)

$$\alpha=W0,85=80,85=6,8\sim 7м$$

b- distance between rows (м)

$$b=W=8м$$

8) Bust length

$$l_n = 0,5\cdot q\cdot W = 0,5\cdot 0,03\cdot 8 = 0,12м$$

9) Borehole length

$$l_{скв} = h + l_n = 15 + 0,12 = 15,12м$$

10) Minimum size of the cull

$$l_з = l_n = 0,12м$$

11) Maximum charge length

$$l_{зар} = l_{скв} - l_з = 15,12 + 0,12 = 15м$$

12) The mass of the charge in the well

$$P = a\cdot W\cdot h\cdot q = 7\cdot 8\cdot 15\cdot 0,03 = 25,2 \approx 25кг.$$

13) Charge diameter (м)

$$d_з = 2\cdot\sqrt{\frac{P}{\pi\cdot l_{зар}\cdot\Delta}};$$

where Δ- charging density (Δ=930 кг/м^3);

$$d_з = 2\cdot\sqrt{\frac{25}{3,14\cdot 15\cdot 930}} \approx 0,024\,м.$$

14) Diameter of wells

$$d_{скв} \geq d_з \qquad d_{скв} = 0,2м$$

Choosing a drilling rig СБШ-200

15) Выбор буровой установки (м)

$$l'_{зар} = \frac{P}{e};$$

where e- capacity of 1m wells (кг/м)

$$e = \frac{\pi\cdot d^2_{скв}\cdot\Delta\cdot\alpha}{4};$$

where α - fill factor (α =1) ;

$$e = \frac{3,14\cdot 0,2^2\cdot 930\cdot 1}{4} = 29,2\ кг/м$$

$$l'_{зар} = \frac{25}{29,2} \approx 0,86м.$$

$$l_{зар}^{/} \le l_{зар} \qquad 0{,}86м < 15м$$

16) Length of the lower (main) charge

$$l_{н.з.} = 1{,}2 \cdot W = 1{,}2 \cdot 8 = 9{,}6м.$$

17) Air gap length

$$l_{в.пр.} = l_{зар} - \frac{4 \cdot P}{\pi \cdot d_{скв}^2 \cdot \Delta \cdot \alpha} = 0{,}86 - \frac{4 \cdot 25}{3{,}14 \cdot 0{,}2^2 \cdot 930 \cdot 1} = 0{,}01м.$$

Since $d_{скв} = d_з$, then the charge will be solid.

18) the volume of the exploding block

$$V_{бл} = 15 \cdot Q_{сут} = 15 \cdot 12330 = 184950 \; м^3.$$

19) The width of the collapse during single-row explosion

$$R = \frac{2 \cdot (c + h \cdot ctg\alpha) \cdot (h \cdot k_p - h_p)}{h_p} = \frac{2 \cdot (5 + 15 \cdot ctg 80^0) \cdot (15 \cdot 1{,}3 - 12{,}5)}{12{,}5} = 8{,}6 \approx 9\,м.$$

20) The width of the exploding block on the whole, based on the number of visits of the excavator in the collapse

$$A = 1{,}5 \cdot R_{ч.у.} \cdot n^{/} - R = 1{,}5 \cdot 11{,}9 \cdot 4 - 9 = 62{,}4 \approx 62 \; м.$$

21) Number of rows of wells

a) from the technological working conditions of excavators in the face

$$n = \frac{1{,}5 \cdot R_{ч.у.} \cdot n^{/} - W - R}{b} + 1 = \frac{1{,}5 \cdot 11{,}9 \cdot 4 - 8 - 9}{8} + 1 = 7{,}8 \approx 8 \; рядов.$$

б) from the safety conditions, the maximum height of the camber should not exceed the maximum height of the scooping, i.e.

$h_{p.max.} = 1{,}5 \cdot H_{ч.max.}$, at the height of the camber from the first row $h_p = H_{ч.max}$.

$$n = \frac{0{,}5 \cdot H_{ч.\max} \cdot b}{h \cdot (k_p - 1)} - \frac{b^2}{b + W} - 2 = \frac{0{,}5 \cdot 12{,}5 \cdot 8}{1{,}5 \cdot (1{,}3 - 1)} - \frac{8^2}{8 + 8} - 2 = 5{,}1 \approx 5 \; рядов$$

We take the minimum value of n=5 rows

Therefore, it is necessary to adjust the parameters of the excavator face:

The width of the face on the whole

$$A = b \cdot (n-1) + W = 8 \cdot (5 - 1) + 8 = 40 \; м.$$

The number of visits of the excavator in the collapse

$$n^{/} = \frac{A + R}{1{,}5 \cdot Rч.у.} = \frac{40 + 9}{1{,}5 \cdot 11{,}9} = 2{,}7$$

We accept $n^{/} = 3$.

22) The length of the exploding block

$$L_{бл} = \frac{V_{бл}}{h \cdot A} = \frac{184950}{15 \cdot 40} = 308\ м$$

23) Number of wells in the exploding block

$$n_{скв} = \frac{A \cdot L_{бл}}{a \cdot b} = \frac{40 \cdot 308}{7 \cdot 8} = 220$$

24) total length of drilling wells in the block

$$L_{скв} = l_{скв} \cdot n_{скв} = 15{,}12 \cdot 220 = 3326 м$$

25) Block drilling time(day)

$$t_{бур} = \frac{L_{скв}}{П_{б.ст.}};$$

where $П_{б.ст.}$ - technical performance of the drilling machine ЗСБШ-200 (**$П_{б.ст}$** =23м/ч);

$$t_{бур} = \frac{3326}{23} \approx 144{,}6 часа \approx 6 суток.$$

26) Number of explosives to destroy the block

$$P_{ВВ} = q \cdot V_{бл} = 0{,}03 \cdot 184950 = 5548{,}5 кг \approx 5{,}5\ т.$$

27) The number of series in a mass explosion, safe for seismic impact on a protected object

$$N = \frac{29^3 \cdot P_{ВВ}}{L^3} = \frac{29^3 \cdot 5548}{1000^3} = 0{,}14$$

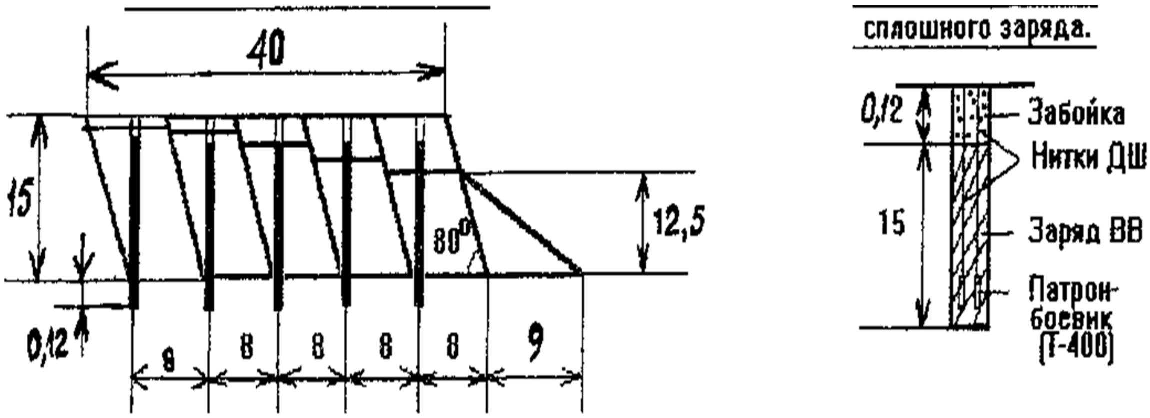

Fig.2. The results of calculations of the parameters of drilling and blasting operations are shown in

PRACTICAL WORK NO. 13
CALCULATION OF PARAMETERS OF TRANSPORT COMMUNICATIONS OF CARGO FLOW

1) *Bulk density of the transported rock* (т/*м*^{3})

$$\gamma = \frac{\rho}{k_p} = \frac{2,66}{1,4} \approx 1,9\, m / м^3.$$

2) We accept БелАЗ-540А.

3) Parameters БелАЗа-540А:

Load capacity q=27т;

Body capacity Vк=15м^3;

Basic dimensions:

length Lм=7350мм;

width b=3480мм;

height Hк=4500мм.

4) Average vehicle speed

$$V_{ср}\, 25 км / ч.$$

5) Duration of the flight (ч)

$$T = t_n + t_p + \frac{2 \cdot L_{тр}}{V_{ср}} + t_м;$$

where t_n- time of loading by excavator of one dump truck (ч)

$$t_n = \frac{V_k}{Q_t} = \frac{15}{734} \approx 0,02 ч.$$

t_p- dump truck unloading time (t_p=0,017ч);

$t_м$ - time spent on maneuvers in the face and unloading point ($t_м$=0,017);

$$T = 0,02 + 0,017 + \frac{2 \cdot 2}{25} + 0,017 \approx 0,21\, ч.$$

6) Technical performance of the dump truck (т/ч)

$$Q_t = q \cdot n_p \cdot k_г;$$

where n_p- number of flights per hour

$$n_p = \frac{1}{T} = \frac{1}{0,21} \approx 4,8.$$

$k_г$- load capacity utilization factor ($k_г$ - 0,95);

$$Q_t = 27 \cdot 4,8 \cdot 0,95 = 123,12 \approx 123 m / ч.$$

7) Operational performance of the dump truck (m/см)

$$Q_{см} = Q_t \cdot T \cdot k_и;$$

where $K_и$ - dump truck utilization rate per shift ($K_и$ - 0,7);

$$Q_{см} = 123 \cdot 8 \cdot 0,7 = 688,8 \approx 689 m / см;$$

8) Annual dump truck performance (т/год)

$$Q_{год} = Q_{см} \cdot N \cdot k_{т.г.};$$

where $k_{т.г.}$ - the coefficient of technical readiness for the daily mode of operation ($k_{т.г}$ =0,9)

$$Q_{год} = 689 \cdot 305 \cdot 0{,}9 = 189130 т/год.$$

9) The number of dump trucks required to service one excavator

$$N = \frac{T}{t_П} = \frac{0{,}21}{0{,}02} \approx 10{,}5$$

We accept N=11

10) Working fleet of dump trucks

$$N = \frac{W_{г.о.} \cdot k_н}{Q_{см} \cdot n};$$

where $W_{г.о.}$ - cargo turnover of the quarry per day (т/сут)

$$W_{г.о.} = Q_{см} \cdot n \cdot \gamma \cdot n_{экс};$$

where $n_{экс}$ - number of excavators at the quarry ($n_{экс}$=1);

$$W_{г.о.} = 4110 \cdot 3 \cdot 1{,}9 \cdot 1 = 22194 т/сут.$$

$k_н$ - transport operation coefficient ($\boldsymbol{k_н}$=1,1);

$$N_р = \frac{22194 \cdot 1{,}1}{689 \cdot 3} \approx 12$$

We accept $N_р$=12

11) Inventory fleet of dump trucks

$$N_и = \frac{N_р}{k_Т};$$

where $k_Т$ - the coefficient of technical readiness of the fleet ($k_Т$ =0,9);

$$N_и = \frac{12}{0{,}9} = 13{,}3$$

We accept $N_и=13$

12) The capacity of a highway lane in one direction (flights)

$$П = \frac{1000 \cdot V_{ср}}{k_Д \cdot S};$$

where $k_Д$- coefficient of unevenness of movement *($k_Д$ =1,5)* ;

S - interval between dump trucks (м)

$$S = 0{,}278 \cdot V_{ср} \cdot t_{реак} + \frac{3{,}9 \cdot (1-\gamma) \cdot V_{ср}^2}{\left(1000 \cdot \Psi_т + \omega_о - i\right)} + L_м;$$

where $t_{реак}$ - driver reaction time and brake actuation time ($t_{реак}$ = 1,5 c);

γ - a coefficient that takes into account the inertia of the rotating masses of the car (for cars with a hydromechanical transmission γ =0,02);

ω_o - specific basic resistance to vehicle movement (ω_o =60 Н/кН);
$\psi_т$ - coefficient of adhesion of wheels to the road during braking ($\psi_т$ = 0,25);
i- road slope (i=60 $^0/_{00}$);

$$S = 0{,}278 \cdot 25 \cdot 1{,}5 + \frac{3{,}9 \cdot (1 - 0{,}02) \cdot 25^2}{(1000 \cdot 0{,}25 + 60 - 60)} + 7{,}3 \approx 27\, м;$$

$$П = \frac{1000 \cdot 25}{1{,}5 \cdot 27} \approx 618\, рейсов.$$

13) Carrying capacity of the road (т/сут)

$$M_{пр} = \frac{П_{о.у.} \cdot q}{f_p};$$

where f_p - bandwidth reserve ratio (f_p =2) ;
$П_{о.у.}$ - the capacity of the limiting section of the road (cars / day)

$$П_{о.у.} = П \cdot T \cdot n = 618 \cdot 8 \cdot 3 = 14832 \ автомобилей / сут.$$

$$M_{пр} = \frac{14832 \cdot 27}{2} = 200232\ т / сут.$$

14) The width of the carriageway in two-lane traffic (м) (см. рис.3)

$$B = 2 \cdot b \cdot k \cdot v + G;$$

where $k{\cdot}v$- a coefficient that takes into account the total speed of oncoming cars ($k{\cdot}v$ =1,75);

G - a value that takes into account the dimensions of the car (G=1)

$$B=2 \cdot 3{,}48 \cdot 1{,}75+1 \approx 14 м.$$

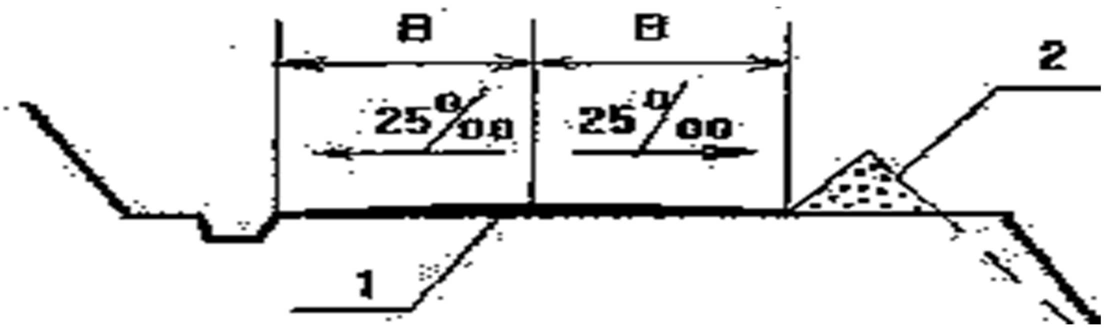

Fig.3 The profile of the highway on the ledge.

1-transport lane;

2- safety shaft.

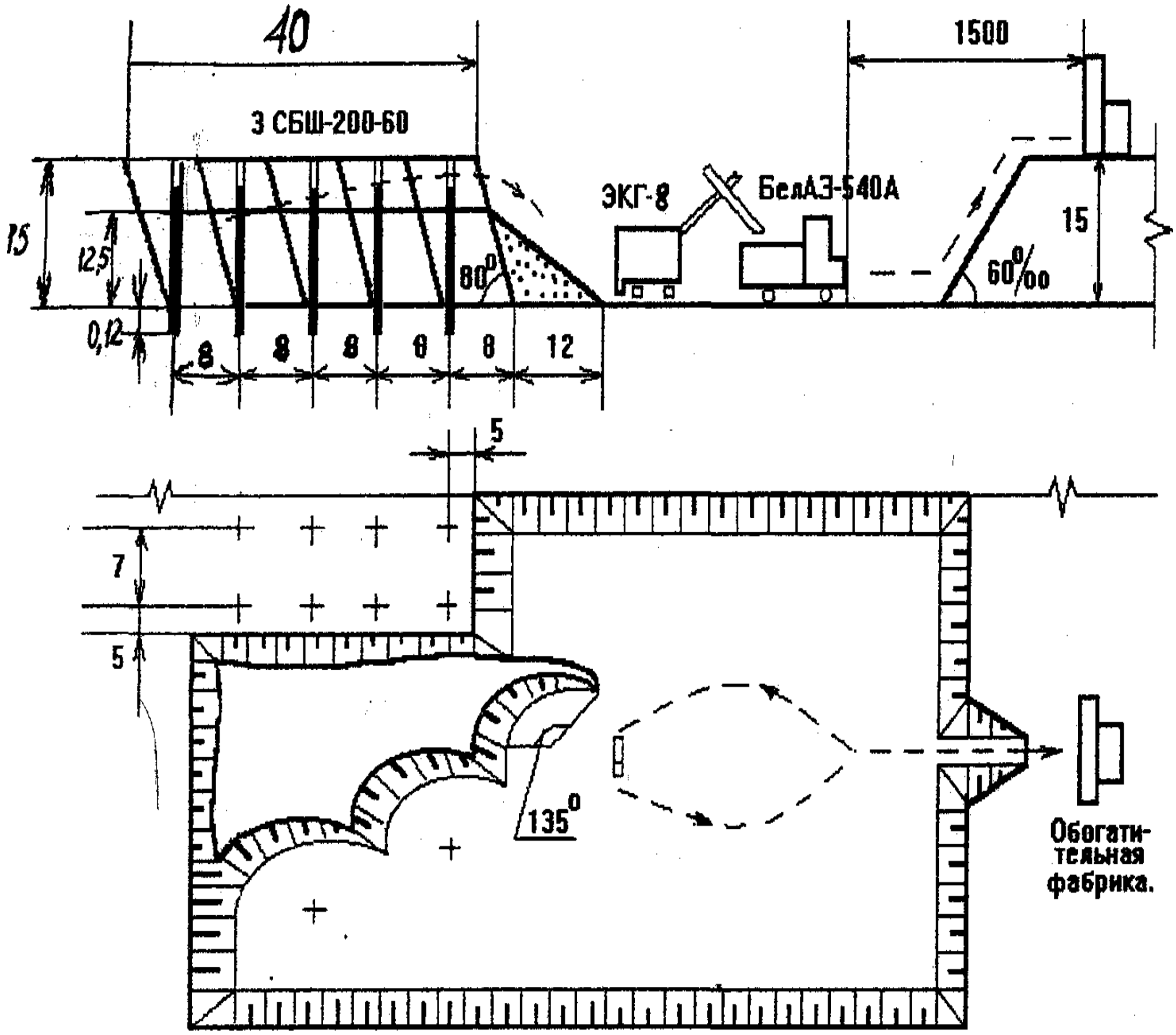

Fig.4 Diagram of the mining process flow.

PRACTICAL WORK NO. 14
DETERMINATION OF THE PARAMETERS OF THE TECHNOLOGY FOR THE DEVELOPMENT OF SHALLOW DEPOSITS
determination of the parameters of the technology of transshipment of overburden into the developed space of the overburden machine

In shallow deposits, the largest costs are incurred for the development and cleaning of overburden rocks, so here the development systems are distinguished by the equipment used in overburden operations.

Continuous development systems are most often used:

- with transshipment of (disposable) rock by excavators to the internal dump;
- with multiple transshipment of rock by excavators to the internal dump;
- with the movement of the rock into the inner dump by dumpers;
- with the transportation of rock by transport to the internal dump;
- with the transportation of rock by transport to the external dump;
- with partial transportation by transport of the rock to the external dump and with partial transshipment - to the internal dumps;

combined systems.

Technology of transshipment of overburden into the developed space with an overburden mehlopat

An overburden excavator is installed on the roof of the ore formation and the entire thickness of the overburden rocks is extracted with one ledge. Following the movement of this face, ore mining is carried out.

The calculation of this technological scheme consists in choosing the necessary operating parameters of overburden excavators, depending on the thickness of the formation of empty rocks and from the condition of equality of rock volumes in the excavator approach (V_1) and in the dump ($У_2$).

$$V_1 = B \cdot H \cdot K_p$$

$$V_2 = B \cdot H_0 - 0.25 \cdot B^2 \cdot tg\beta$$

where B - entry width, м;

H - overburden formation capacity, м;

$К_р$ - loosening coefficient;

H_0 - blade height, м;

β - slope angle of the blade, град.

Т.к. $V_1 = V_2$, then the maximum capacity of the overburden formation is equal to:

$$H_{max} = \frac{(H_0 - 0.25 \cdot B \cdot tg\beta)}{k_p}$$

Excavator unloading radius:

$$\boldsymbol{R_p = c + d + h \cdot ctg\alpha + H_0 \cdot ctg\alpha}$$

where c - the distance from the excavator axis to the upper edge of the ore ledge, м;

d - the width of the free developed space, м;

h - ore formation capacity, м;

α- the angle of the slope of the ore ledge, град.

Hence the maximum height of the blade:

$$H_0 = (R_p - c - d - h \cdot ctg\alpha) \cdot tg\beta$$

and the maximum capacity of the extracted formation of empty rocks:

$$H_{max} = \frac{R_p - (c - d - h \cdot ctg\alpha + 0.25 \cdot B)}{K_p \cdot ctg\alpha}$$

Technology of overburden transshipment using draglines

Draglines are placed on the roof of the overburden formation or on the roof of the ore formation or on the intermediate dump.

Dragline unloading radius

$$\boldsymbol{R_p = a + c + d + h \cdot ctg\alpha + H \cdot ctg\gamma + H_0 ctg\beta}$$

where $\boldsymbol{a}$ – width of the safety berm, м;

$\boldsymbol{c}$ – the distance from the dragline axis to the upper edge of the overburden ledge, м;

$\boldsymbol{d}$ - the width of the free developed space, м;

$\boldsymbol{\gamma}$ – the width of the free developed space.

Maximum blade height

$$H_0 = \frac{R_p - (c + a + d + H \cdot ctg\gamma + hctg\alpha)}{ctg\beta}$$

Maximum height of overburden ledge:

$$H_{max} = \frac{R_p - (c + a + d + hctg\alpha + 0.25B)}{K_p \cdot ctg\beta + ctg\gamma}$$

PRACTICAL WORK NO. 15
DEVELOPMENT TECHNOLOGY WITH MULTIPLE TRANSSHIPMENT OF ROCK TO THE DUMP

the distance between excavators must be at least the sum of their maximum radii of scooping and unloading, for example, *R > 2R*$_P$.

Scooping radius of the second excavator:

$$\boldsymbol{R_ч = b_1 + c_1 + d + H_0^1 \cdot ctg\beta}, \ \ \text{м}$$

Unloading radius of the second excavator:

$$R_p = b_1 + c_1^1 \ \ + H_0^{11} \cdot ctg\beta$$

Scooping depth: $H_ч > H_o^1$

Unloading height: $H_p > H_o^1$

The performance of the second dragline depends on the actual performance of the first excavator and the coefficient of overexcavation:

$$Q_2 = Q_1 \cdot K_{пер}, \ \text{м}^3/\text{час}$$

where $K_{пер}$ - the coefficient of overexcavation:

$$K_{пер} = \frac{V_{пер}}{V}$$

$V_{пер}$- overexcavable volume of overburden:

$$V_{пер} = L_1 \cdot (H_0 - 0.25 \cdot L_1 \cdot tg\beta), \text{м}$$

V - the volume of overburden from 1 pm of the length of the excavator approach:

$$\boldsymbol{V = A \cdot H \cdot K_p}, \ \text{м}^3$$

L_1- the width of the excavator entry by ore

$$\boldsymbol{L_1 = d + h \cdot (ctg\alpha + ctg\beta)}, \text{м}$$

3. Technology of development with the movement of overburden by dumpers

Self-propelled cantilever dumpers are used in quarries with rotary or chain multi-pack excavators, where the rock is stored in internal dumps. The parameters of the development system depend on the size of the dumper - the lifting height and the radius of its unloading. Transport dump bridges are used on shallow deposits with a reservoir capacity of up to 20-25 m. The development technology is similar to the technology with cantilever dumpers. The range of movement of the rock can reach 500 m with a single conveyor drive.

Technology of development with transportation by overburden transport to the internal dump

The transportation of rock to the internal dump is carried out in deep quarries with the excavation of a shallow layer of ore at once at full capacity. In this case, multi-pack or rotary excavators are used on overburden, and the rock is transported to the internal dump by rail or conveyor transport. In the initial period, when there is no developed space in the quarry (ore has not been extracted on some area), the waste rocks are taken to an external dump.

Development technology with transportation of rock by transport to external dumps

This technology is used when it is impossible to place rock dumps in the worked-out space of the quarry, i.e. with a large capacity of the ore formation or with a steep and inclined fall of the deposit. This system is also used on shallow deposits of building materials with a small capacity of covering rocks (after reclamation, this quarry is filled with water and serves as a recreation area for local residents).

Development technology with transportation of rock by transport partly to external, partly to internal dumps

The technology is used in the development of deep, with a powerful thickness of ore, and extended quarries, when it is physically impossible to place the entire overburden in internal dumps. Or when the deposit is represented by two layers, then the rock of the external overburden is taken to the external dumps, and the rock from the proplast is taken to the internal ones.

Development technology with transshipment and transportation of rock to internal dumps

This combined technology is used in quarries with a significant capacity of covering rocks. Then one part of the overburden volume is moved to the first tier of the internal dump by a transport-free method (using dumpers, conveyors), and the other part of the overburden volume from the first tier is re-exported to the second tier of the internal dump.

PRACTICAL WORK NO. 16
CALCULATION OF THE REQUIRED AMOUNT OF MINING EQUIPMENT

- The calculation of the required amount of mining equipment is carried out based on its performance, the specified annual volumes of overburden and minerals.
- Required number of drilling rigs in operation (working park):
- after stripping:

$$N_{брв} = V_{бв} / P_{год.в} \qquad (1)$$

- by mineral resource:

$$N_{брп} = \frac{V_{бп}}{P_{год.п}} \qquad (2)$$

where $P_{год.в}$, $P_{год.п}$ - accordingly, the annual productivity of the drilling rig for rocks and minerals, m/ year;

$V_{бв}$, $V_{бп}$ - accordingly, the annual volume of drilling for bedrock and minerals, m/year.

$$V_{бв} = \frac{V_к}{\eta_к}$$

$$V_{бп} = \frac{A\rho_{пи}}{\eta_{пи}} \qquad (3)$$

where V_K, A- accordingly, the annual volume of the bedrock (м^3/year) and minerals (tons/year) (according to the task); $\rho_{пи}$ – mineral density ($\rho_{пи}$ =(1,35-1,5) т/м^3); $\eta_к$, $\eta_{пи}$ - accordingly, the output of rock mass from the 1m well for bedrock and minerals, м^3/м.

$$\eta_к = \frac{a_к \cdot b_к \cdot h_к}{l_{скв.к}} \qquad \eta_{пи} = \frac{a_{пи} \cdot b_{пи} \cdot h_{пи}}{l_{скв.пи}}$$

where $a_к, b_к, h_к$ и, $l_{скв.к}$ $a_{пи}, b_{пи}, h_{пи}$, $l_{скв.пи}$ – accordingly, the distance between wells and between rows of wells, the height of the ledge and the length of wells for bedrock and minerals, м.

Inventory park of drilling rigs for stripping (Upholstery) and minerals:

$$N_{бив} = N_{брв} f_б \qquad (5)$$

where $f_б$ - drilling rigs reserve ratio ($f_б$ = 1,2 -1,25).

Here and further, the size of the working fleet of machines can be fractional, inventory - whole.

Required number of excavators in operation:

by deposits:

$$N_{эри} = \frac{V_н}{Q_{э.год.н}} \quad (6)$$

by bedrock:

$$N_{эрк} = \frac{V_к}{Q_{э.год.к}} О \quad (7)$$

by mineral resource:

$$N_{эрп} = \frac{A\rho_{пи}}{Q_{э.год.п}} \quad (8)$$

where V_H - by mineral resource, м^3/year; $Q_{э.год.н}$, $Q_{э.год.к}$, $Q_{э.год.п}$ accordingly, the annual productivity of the excavator for sediments, bedrock and minerals, м^3/year.

Inventory fleet of excavators by sediment $N_{эин}$, by bedrock $N_{эик}$ and minerals $N_{эип}$:

$$N_{эин} = N_{эрн} f_э$$
$$N_{эик} = N_{эрк} f_э$$
$$N_{эип} = N_{эрп} f_э \quad (9)$$

where $f_э$- excavator reserve ratio ($f_э$= 1,2 -1,4).

The required number of locomotives in operation:

by deposits:

$$N_{лрн} = \frac{К_{нер} V_н}{Q_{л.год.н}} \quad (10)$$

by bedrock:

$$N_{лрк} = \frac{К_{нер} V_к}{Q_{л.год.к}} \quad (11)$$

by mineral resource:

$$N_{лрп} = \frac{К_{нер} A\rho_{пи}}{Q_{л.год.п}} \quad (12)$$

where $Q_{л.год.н}$, $Q_{л.год.к}$, $Q_{л.год.п}$ – accordingly, the annual productivity of the locomotive composition for sediments, bedrock and minerals, м^3/year; $K_{нер}$ - the coefficient of uneven operation of transport ($K_{нер}$ =1,1-1,2).

Working fleet of dumpcars (wagons) by sediment $N_{дн}$, indigenous breeds N_{dK} and minerals $N_{дп}$:

$$N_{дн} = N_{лрн} n_{дн}; \quad N_{дк} = N_{лрк} n_{дк}; \quad N_{дп} = N_{лрп} n_{дп} \quad (13)$$

where $n_{дн}$, $n_{дк}$, $n_{дп}$ - accordingly, the number of dumpcars (wagons) in the train during the transportation of sediments, bedrock and minerals, pcs.

The inventory fleet of locomotives and dumpcars (wagons) take 20-25% more than the working fleet.

The required number of dump truck workers to ensure the efficient operation of excavators by type of work: Tarn deposits, bedrock $N_{арк}$; mineral resources $N_{арп}$:

$$N_{арн} = \sum_{i=1}^{n} N_{ан}; \quad N_{арк} = \sum_{i=1}^{n} N_{ак}; \quad N_{арп} = \sum_{i=1}^{n} N_{ап}; \quad (14)$$

where $N_{ан}, N_{ак}, N_{ап}$- accordingly, the number of dump trucks required for use complete with one excavator working on the excavation of sediments, bedrock and minerals, pcs.

The inventory fleet of dump trucks, taking into account those under repair and maintenance, take 20-30% more workers. Number of working excavators on the dump:

$$N_{зор} = \frac{(V_к + V_н)}{1{,}25 Q_{з.год.к}} \quad (15)$$

Total number of excavators on the dump (inventory park):

$$N_{зои} = f_о N_{зор} \quad (16)$$

where $f_о$ - the coefficient of the reserve of excavators on the dump ($f_о$= 1,1 -1,3). Number of working bulldozers on the dump:

$$N_{бор} = \frac{(V_к + V_н) К_{зв}}{Q_{б.год}} \quad (17)$$

where $К_{зв}$- the coefficient of blockage of the dump with rock ($К_{зв}$ =0,6-0,7); $Q_{б.год}$ – annual productivity of the bulldozer on the dump ($Q_{б.год}$ = 1500000-2100000), м^3/year.

Inventory park of bulldozers:

$$N_{био} = f_{бо} N_{бор} \quad (18)$$

where $f_{бо}$ - the ratio of the reserve of bulldozers on the dump ($f_{бо}$ = 1,3 - 1,4)

PRACTICAL WORK NO. 17
DETERMINATION OF PARAMETERS OF A TRANSPORTLESS DEVELOPMENT SYSTEM

Example.1. Determine the maximum width of the entry A and the width of the working area of the Sr.p. when working with the excavator EVG-15 with the transshipment of the overburden into the developed space.

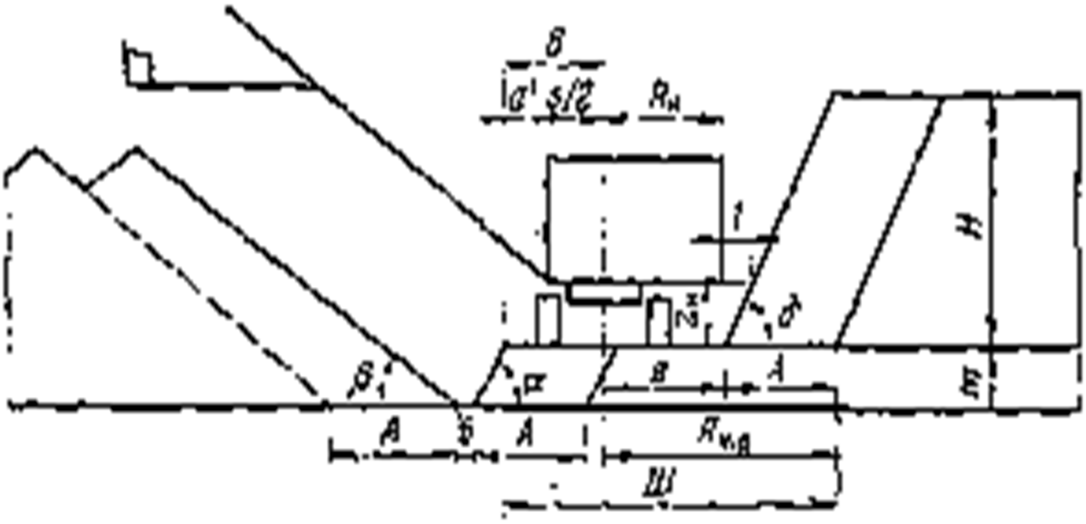

Fig. 1. Scheme of transshipment of rocks with a mehlopatoy at idle transitions of the excavator.

Coal mining and stripping are carried out in one block. For the reverse idle passage of the excavator, a platform is left on the roof of the formation (Fig.1), Coal is transported along the roof of the formation by dump trucks; the angle of the stable slope of the overburden ledge δ = 600.

Decision. The maximum width of the entry (m) is determined by the formula

$$\text{A} = R_{\text{ч.у}} - e_{min}$$

where $R_{\text{ч.у}}$– the maximum scooping radius of the overburden excavator on the horizon of its installation, м; $e_{min} = R_k + 1 - z_k ctg\delta$=$e_{\min}$ – minimum distance from the excavator axis to the bottom edge of the overburden ledge, м; R_k – the radius of rotation of the excavator body, m; 1 m – the minimum safety clearance between the slope of the ledge and the body; z_k – clearance under the rotary platform of the excavator, м.

At the excavator ЭВГ-15 $R_{\text{ч.у}} = 20.5$ м;

$R_k = 12$ м; $z_k = 6$ м.

$$e_{min} = 12 + 1 - 6 \cdot ctg60^0 = 9.5 \text{ м};$$

$$A = 20{,}5 - 9{,}5 = 11 \text{ м.}$$

Working area width

$$Ш = a^{/} + \frac{S}{2} + e + A$$

where $a^{/}$ – the minimum distance from the excavator running gear to the upper edge of the mining ledge, assumed to be at least 2÷3 м;

S – excavator stroke width (ЭВГ-15, s=13,5 м).

$$Ш = 2 + \frac{13.5}{2} - 9.5 + 11 = 29 \text{ м}$$

Example.2. Determine the rate of advance of the work front and the possible productivity of the quarry for minerals.

The overburden ledge with a height of H = 20 m is worked out according to a simple transportless scheme with an ASH-15/90 excavator; the annual productivity of the excavator Qa = 3.5 million m3, the length of the overburden front Lf.v = 2000 m, for extraction Lf.d = 1950 m; the average thickness of the mineral reservoir t = 3 m; density γ =1.2 t/m3: extraction coefficient $к_{изв}$=0,95.

Decision. The speed of moving the work front

$$v_ф = \frac{Q_э}{L_{ф.в} \cdot H} = \frac{3500000}{2000 \cdot 20} = 87{,}5 \text{ м/год}$$

Productivity of the quarry by the started fossil

$$Q_{пи} = L_{ф.д.} m K_{изв} \gamma = 3 \cdot 1950 \cdot 87{,}5 \cdot 0{,}95 \cdot 1{,}2 = 583{,}5 \text{ тыс.т/год}$$

Example.3. Determine the width of the working platform W, the width of the entry and the maximum height of the overburden ledge when working with the EVG-15 excavator with overburden transshipment into the developed space.

The working stroke of the excavator is direct and reverse; minerals are delivered by rail along the roof of the formation (Fig. 2); the thickness of the horizontally lying mineral reservoir t = 3 m; the angles of stable slopes of the mining ledge α = 60 °, the overburden ledge δ = 60, the dump β = 35°; the coefficient of loosening of rocks in the dump Kr = 1.22; the site on the soil of the formation is not left (B = 0).

Decision. Working area width.

$$Ш = a^{/} + \frac{S}{2} + e$$

where e – distance from the excavator axis to the bottom edge of the overburden ledge, м; s – excavator stroke width, м; a – the distance from the upper edge of the mineral ledge to the excavator chassis, м.

$$e_{min} = R_k + 1 - z_k ctg\delta = 9{,}5 \text{ м;}$$

$$e_{max} = R_{ч.у}=20,5 \text{ м};$$

$$a' = C_з + b' = 2,5 + 1,5 = 4 \text{ м},$$

where C_3 – the minimum distance from the axis of the path to the edge of the mining ledge, which is according to the current safety standards 2,5 м; b' – minimum distance from the track axis to the excavator chassis b'= 1,5 м.

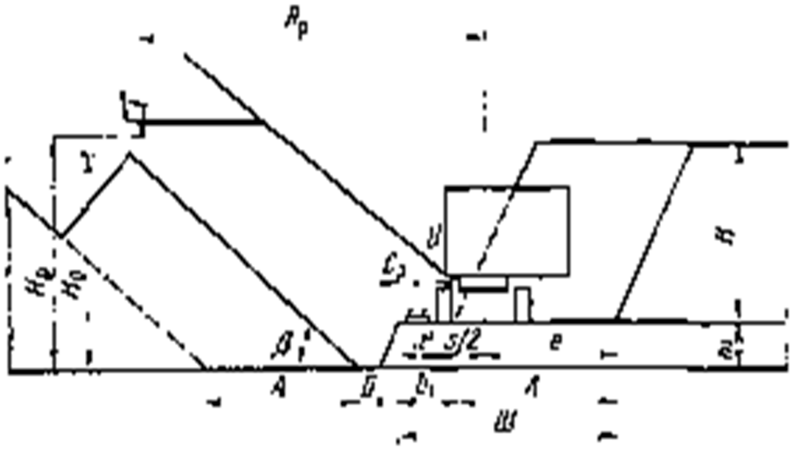

Fig. 2. Scheme of transshipment of rocks by the mehlopata.

Minimum width of the working platform $Ш_{min}=4+13/2+9,5=20$ м; maximum width of the working area $Ш_{max}= 4+ 13/2 + 20,5=31$ м.

The width of the excavator entry

$$A = Ш - b_т,$$

where $b_т = C_з + b'' = 2,5 + 2,5 = 5$ м – width of the transport berm;

b'' – the distance from the axis of the path to the lower edge of the overburden ledge (in the absence of a contact network b" = 2÷3 м).

$$A_{max} = 31-5=26 \text{ м}; \quad A_{min} = 20-5= 15 \text{ м}.$$

Possible height of the blade along the radius of unloading of the excavator

$$H_0 \leq (R_{pmax} - B - m*ctg\alpha - Б)/ ctg\beta,$$

where R_{pmax} – maximum unloading radius of the excavator. м,

B=a'+s/2=4+13/2=10,5 м – the distance from the axis of movement of the excavator to the upper edge of the mining ledge.

$$H_o \leq (37,8 - 10,5 - 3 \text{ ctg}60° - 0)/ctg35^0; \; H_0 \leq 17,7 \text{ м}.$$

The possible height of the blade according to the height of unloading of the excavator Hp, corresponding to the maximum radius of unloading,

$$H_0 \leq H_p + m; \; H_o \leq 15 + 3; \; H_0 \leq 18 \text{ м}.$$

We accept a smaller value: Ho=17,7 м.

Maximum capacity of the stripping ledge

$$H=(H_0 - 0,25A*tg\beta)/K_p.$$

When entering the maximum width $A = 26$ м.

$$H = (17,7-0,25-26*tg35^0)/1,22 = 10,9 \text{ м}.$$

When entering the minimum width A=15 м.

$$H = (17,7 - 0,25*15*tg35^0)/1,22 = 12,3 \text{ м}.$$

Example.4. Determine the width of the entry A, the maximum height of the overburden ledge H and the loss of coal in the tseliki during the operation of the excavator EVG-35/65 according to the scheme shown in Fig.3.

The power of the coal seam m = 5 m; the angles of the stable slope of the mining ledge α = 60 °, the overburden ledge δ = 60 °, the dump β1 = 37 °; the coefficient of loosening of rocks in the dump Kr = 1.33; the working stroke of the excavator is direct and reverse; coal transportation through the soil of the formation is carried out by dump trucks, a berm on the roof the formation is not left.

Decision. The distance from the excavator axis to the upper edge of the mining ledge

$$B=s/2+a'=20{,}8/2+3=I3{,}4 \text{ м}.$$

With the width of the entry A =29 м, $П_у=100*5^2ctg60^0/5*29 = 10$ %; with the width of the entry A = 50,4 м, $П_у=100*5^2ctg60^0/5*50{,}4 = 5{,}75$ %;

Example.5. Determine the maximum height of the overburden ledge H, which is worked out according to a simple transport-free scheme (Fig. 4) by the ESH-40/85 dragline.

The capacity of the horizontal mineral reservoir m = 6 m; the angles of the stable slope of the mining ledge α = 60 °, the overburden ledge δ = 50 °, the dump β1 = 34°; the coefficient of loosening of rocks Kr = 1.2; the transport of minerals through the soil of the reservoir by automobile.

Decision**.**

$$H = \frac{\{Rp\ rmax\ -\ (B + b + m * ctg\,\alpha + Б + 0{,}25A)\ + H_2 ctg\,\delta\}}{(K_p ctg\,\beta + ctg\,\delta)}$$

where R_{pmax} – maximum unloading radius of the stripping excavator, м;

B = s/2 + a' – the distance from the axis of the excavator to the upper edge of the overburden ledge, m; s – the width of the excavator stroke, m; a – the minimum distance from the excavator running gear to the upper edge of the overburden ledge, which is usually taken equal to 0.2 of the height of the overburden ledge, but not less than 3 m; b – the width of the berm on the roof of the formation, m (when transporting minerals on the soil, it can be equal to 0); B = bK + T + X'—the width of the platform on the soil of the formation, m; bk – the width of the drainage groove on top, m (under favorable hydrogeological conditions or with a closed drainage device, bk = 0 can be taken);T is the width of the transport lane, m (when the road is installed behind the mining excavator, T=0); X' is the width of the collapse when blasting a mineral, m (when blasting for shaking, usually X/= 0.5 m, in the absence of blasting, X' = 0); A is the width of the approach, m (usually it is taken within 0.4—0.6 of the maximum scooping radius of the overburden dragline); H2 is the height of the upper overburden ledge [usually H2 =(0.4—0.6)Hpmax, when the dragline performance decreases slightly during upper scooping; the maximum value of H2= (0.74-0.8)Hpmax];Hp Max is the maximum unloading height of the overburden dragline, м.

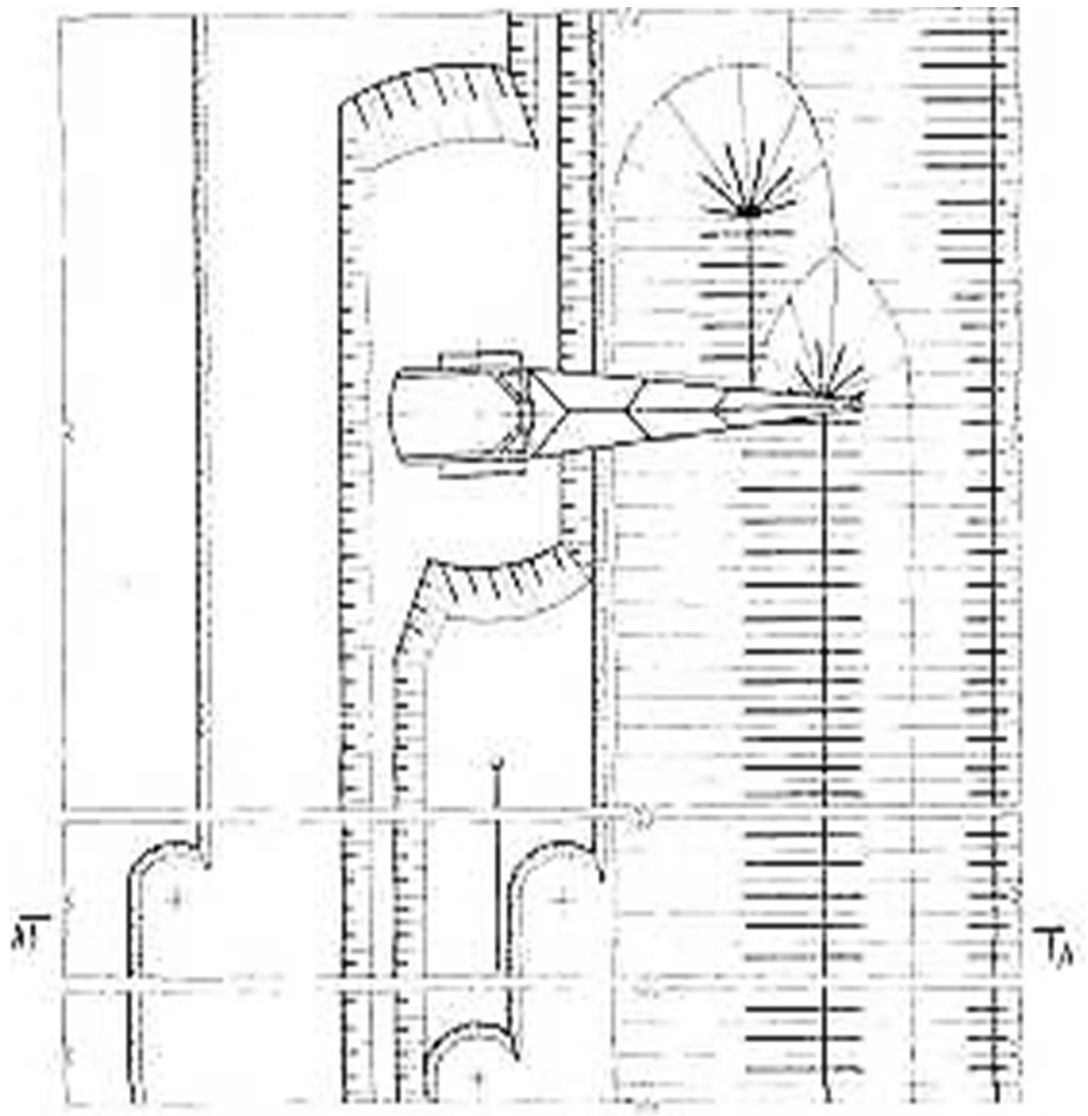

Fig.4. Scheme of transshipment of rocks by dragline.

$$H = (82-(19+0+6ctg60^0+0+0,25*40) + 18\ ctg50^0) / (1,2\ ctg34^0 + ctg50^0) = 32\ м.$$

A task. 1. Determine the maximum width of the entry A and the width of the working area of the Sr.p. when working with the EKG-15 excavator with overburden transshipment into the developed space.

Coal mining and stripping are carried out in one block. For the reverse idle passage of the excavator, a platform is left on the roof of the formation (Fig.1), Coal is transported along the roof of the formation by dump trucks; the angle of the stable slope of the overburden ledge δ = 600.

A task. 2. Determine the maximum width of the entry A and the width of the working area of the Shr.p. when operating the EKG-12.5 excavator with overburden transshipment into the developed space.

Coal mining and stripping are carried out in one block. For the reverse idling passage of the excavator, a platform is left on the roof of the formation (Fig.1), Coal is transported along the roof of the formation by rail; the angle of the stable slope of the overburden ledge δ = 600.

A task. 3. Determine the rate of advance of the work front and the possible productivity of the quarry for minerals.

The overburden ledge with a height of H = 20 m is worked out according to a simple transportless scheme with an EKG-15 excavator; the annual productivity of the excavator Qa = 4.5 million m3, the length of the overburden front Lf.v = 2000 m, for extraction Lf.d = 1950 m; the average thickness of the mineral reservoir t = 3 m; density γ =1,2 t/m3: the extraction coefficient of kzv = 0,95.

A task. 4. Determine the width of the working platform W, the width of the entry and the maximum height of the overburden ledge when operating the EKG-12.5 excavator with overburden transshipment into the developed space. The working stroke of the excavator is direct and reverse; the mineral is delivered by rail along the roof of the formation (Fig. 11.2); the power of the horizontally lying mineral reservoir t = 3 m; the angles of stable slopes of the mining ledge α = 70 °, the overburden ledge δ = 50, the dump β = 47 °; the coefficient of loosening of rocks in the dump Kr= 1.22; the site on the soil of the formation is not left (B = 0).

A task. 5. Determine the width of the entry A, the maximum height of the overburden ledge H and the loss of coal in the tseliki during the operation of the excavator EVG-35/65 according to the scheme shown in Fig. 11.3.

The power of the coal seam m = 5 m; the angles of the stable slope of the mining ledge α = 65 °, the overburden ledge δ = 70 °, the blade β1 =47 °; the coefficient of loosening of rocks in the dump Kr = 1.33; the working stroke of the excavator is direct and reverse; coal transportation through the soil of the formation is carried out by dump trucks, the berm is not left on the roof of the formation.

PRACTICAL WORK NO. 18

DETERMINATION OF THE PARAMETERS OF THE TRANSPORT SYSTEM OF DEVELOPMENTПРИМЕР

1. Determine the minimum width of the working platform when excavating soft overburden rocks with an EKG-8I excavator using railway transport (Fig. 1).

Решение.

$$Ш = A + C_2 + E + C_1 + П_э + П ,$$

where $A = a_1 + a_2$ - the width of the excavator approach;

C_2 - distance from the axis of the path to the bottom edge of the ledge, м;

E - the distance between the axes of railway tracks with diesel and diesel-electric traction E=4,5 м^3,

when using contact electric locomotives E=7÷8.5 m3 (smaller figure - when loading with an excavator ECG-4.6, large - ЭКГ-12,5), with a single-track track E = 0; C1 - the distance from the axis of the track to the power supply lane (with diesel and diesel-electric traction C1 = 2.5 m, with contact electric locomotives C_1=5÷6 м);

Pe and P - the width of the strips, respectively, for the placement of power supply devices and additional equipment, is taken in total within 6÷12 м.

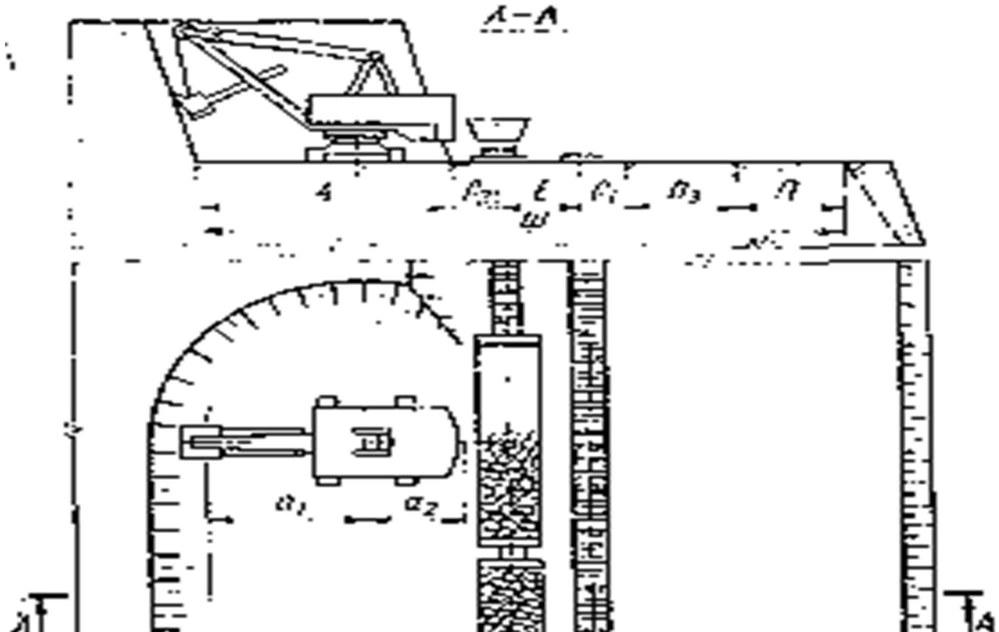

Fig.1. Scheme of development of soft rocks by a mehlopata in railway transport.

The width of the inner part of the entry is limited by the scooping conditions

$\alpha_1 \leq R_{ч.у}$; $\alpha_1 \leq 12{,}2$ м

and the conditions of safe rotation of the excavator

$\alpha_1 \geq 7$м, $\alpha_1 \geq R_{ч.у}+1-z \cdot ctg\alpha$; $\alpha_1 \geq 7{,}6+1-2{,}8ctg60^0$;

where $R_к$ - radius of rotation of the excavator body, м; 1 м - minimum clearance according to safety standards between the body and the slope of the ledge or transport vessel;

$R_{ч.у}$ - the maximum radius of the excavator scooping on the horizon of its installation; м;

z_k- clearance of the rotary platform of the excavator, м;

α - slope angle of the ledge, degree.

The width of the outer part of the entry is limited by the conditions of normal scooping without pushing out the rock: $\alpha_2 \leq 0{,}7R_{ч.у}$; $\alpha_2 \leq 0{,}7*12{,}5$; $\alpha_2 \leq 8{,}5$ м.

We accept the maximum width of the excavator entry in order to reduce the cleanliness of the movement of the downhole path:

$$A=12{,}2+8{,}5=20{,}7 \text{ м}.$$

Minimum width of the working platform;

with a single - track railway

$$Ш = 20{,}7+3+2{,}5+6+6 =38{,}2 \text{ м}.$$

with a two - track

$$Ш = 20{,}7+3+4{,}5+2{,}5+6+6 = 42{,}7 \text{ м}.$$

EXAMPLE 2. Determine the maximum height of the ledge and the width of the working platform when loading soft rocks into dumpcars 2VS-105 with an EKG-6 excavator, Memory (Fig. 2).

The slope angle of the developed ledge α = 60 °, the angle of the stable slope of the ledge α_0 = 450.

Decision. 1. The height of the ledge is limited by the radius of unloading Rp and the height of unloading Hp of the excavator:

$$h \leq H_p - \left(h_д + c_1 + h_p\right)$$

$$h \leq \left(R_p - \alpha_1 - C_3\right)tg\alpha_0$$

$$h \leq \left(R_p - 9 - 2{,}5\right)tg45^0;\ h \leq R_p - 11{,}5; \quad (1)$$

$$h \leq H_p - 3{,}4 - 0{,}4 - 0{,}5;\quad h \leq H_p - 4{,}3; \quad (2)$$

where H_p and R_p unloading height and the corresponding maximum unloading radius given for the excavator ЭКГ – 6,3у in fig. 3.;

$h_д$ – dumpcar height, м;

c_1 – minimum gap between bucket and transport vessel, м;

h_p - height of the upper structure of the railway track (rails, sleepers and ballast), м;

α_1 - width of the inner part of the entrance, м

$\alpha_1 \geq 9$ м, $\alpha_1 \geq R_к + 1 - z_к \cdot ctg\alpha$; $\alpha_1 \geq 10+1-3,3ctg60^0$;

where $R_к$ - radius of rotation of the excavator body, м;

z_k – clearance under the turntable, м);

$C_3 = 2,5$ *м* - the minimum distance according to safety standards from the axis of the path to the upper edge of the ledge or the lines of possible collapse.

The maximum height of the ledge is determined by the joint solution of dependencies (1) and (2) and the dependencies between H_p and R_p shown in Fig. 3. From (1) and (2) we find;

$$H_p = R_p - 7,2. \quad (3)$$

The point of intersection of the line (3) with the dependency $H_p(R_p)$ for the excavator ЭКГ-6,3у (fig. 3) corresponds to the values $H_p = 21,8$ and $R_p = 29$ м. At the same time, the maximum possible height of the ledge h = 21,8–4,3 =17,5 м

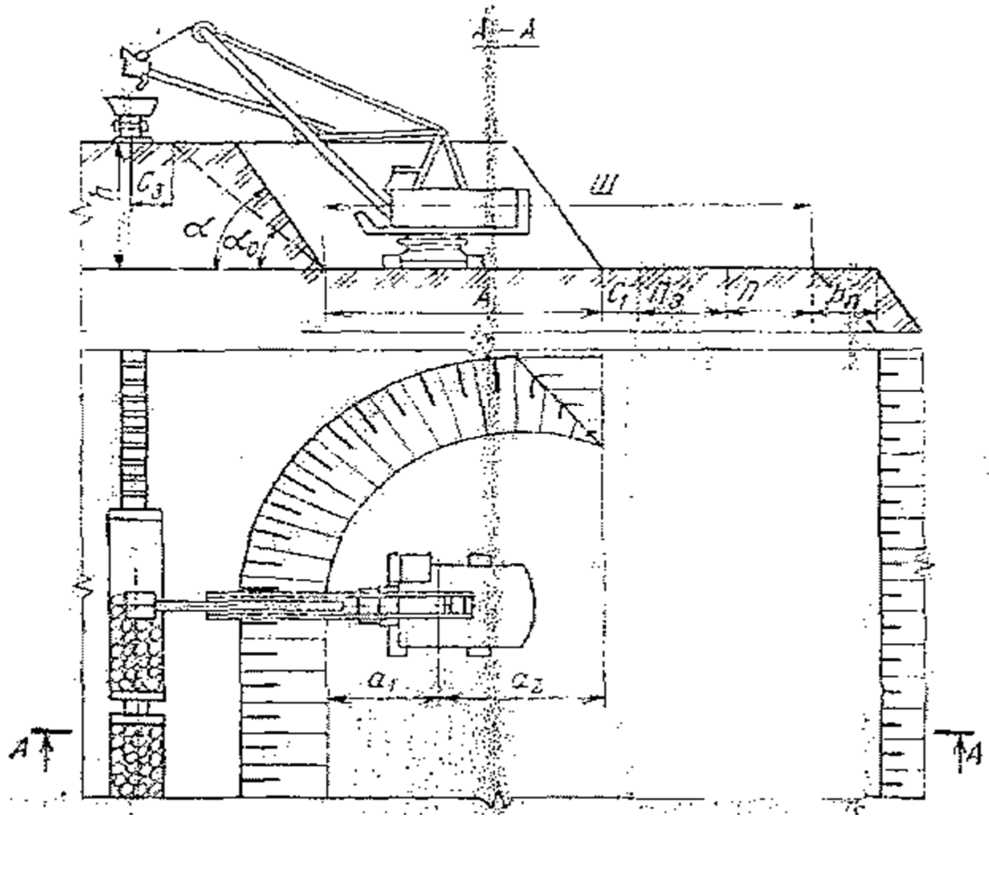

Fig. 2. Scheme of development of soft rocks by excavator top loading.

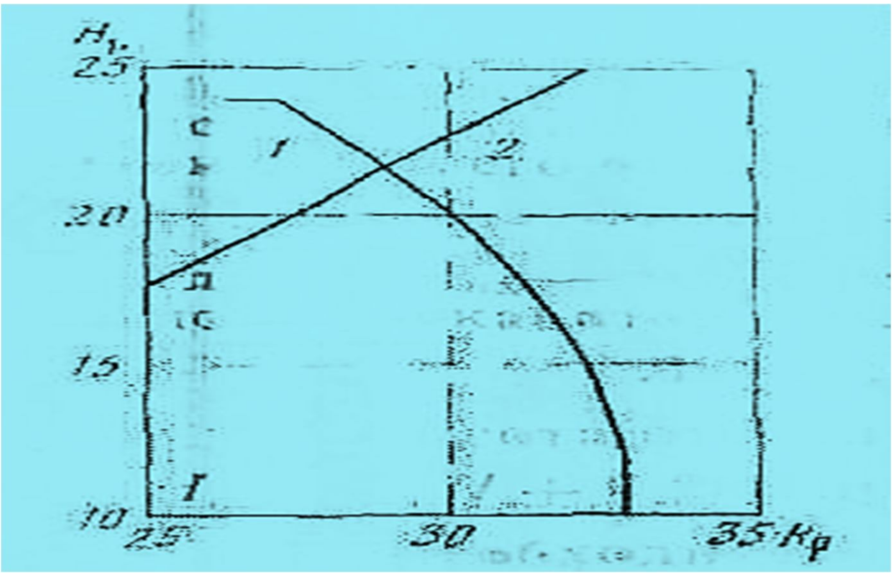

Fig. 3. The graph for determining the height of the ledge developed by the excavator with top loading:

1 – the relationship between H_p and R_p for the excavator ЭКГ – 6,3У; **2** – graph of the equation H_p- R_p–7,2.

Working area width

$$Ш = A+C_1+ П_э+П,$$

where A= α_1+ α_2 = 9+14=23 м excavator entry width, м;

α_2 - the width of the outer part of the entrance, accepted under the conditions of scooping the rock without pushing it out

$$\alpha_2=0,7 \cdot R_{ч.у}=0,7 \cdot 20 = 14 \text{ м},$$

where $R_{ч.у}$ - the maximum radius of the excavator scooping on the horizon of the ugo installation;

C_1=2,5 м - distance from the lower edge of the ledge to the power supply strip;

$П_э$, П - the width of the strips for placing power supply devices and additional equipment, respectively, *м.*

$$Ш=23+2,5+6 + 6 = 37,5 \text{ м}.$$

EXAMPLE 3. To determine the minimum width of the working platform when developing a ledge with a height h = 20 m, with an EKG-12.5 excavator using railway transport and drilling and blasting operations (Fig. 4.).

The annual productivity of the excavator Q = 2.4 million m3, the length of the work front Lb = 1000 m; specific consumption BB qr = 0.6 kg / m3; line resistance along the sole W = 8.5 m; frequency of production of mass explosions of TV - once a month.

Decision: 1. The width of the entry, but the whole (width of the exploding block), providing a given frequency of mass explosions,

$$A \geq \frac{Q \cdot T_{â}}{12 \cdot L_{á} \cdot h};\ A \geq \frac{2{,}4 \cdot 10^{6} \cdot 1}{12 \cdot 1000 \cdot 20};\ A \geq 10 \text{ м.}$$

We accept a two-row arrangement of wells (n_p= 2) и

A=2·W=2·8,5=17 м.

2. The width of the collapse of the blasted rock

$$X = 5 \cdot q_p \sqrt{W \cdot h} + (n_p - 1) \cdot W = 5 \cdot 0{,}6 \sqrt{8{,}5 \cdot 20} + (2 - 1) \cdot 8{,}5 = 47{,}6$$

3. Working area width

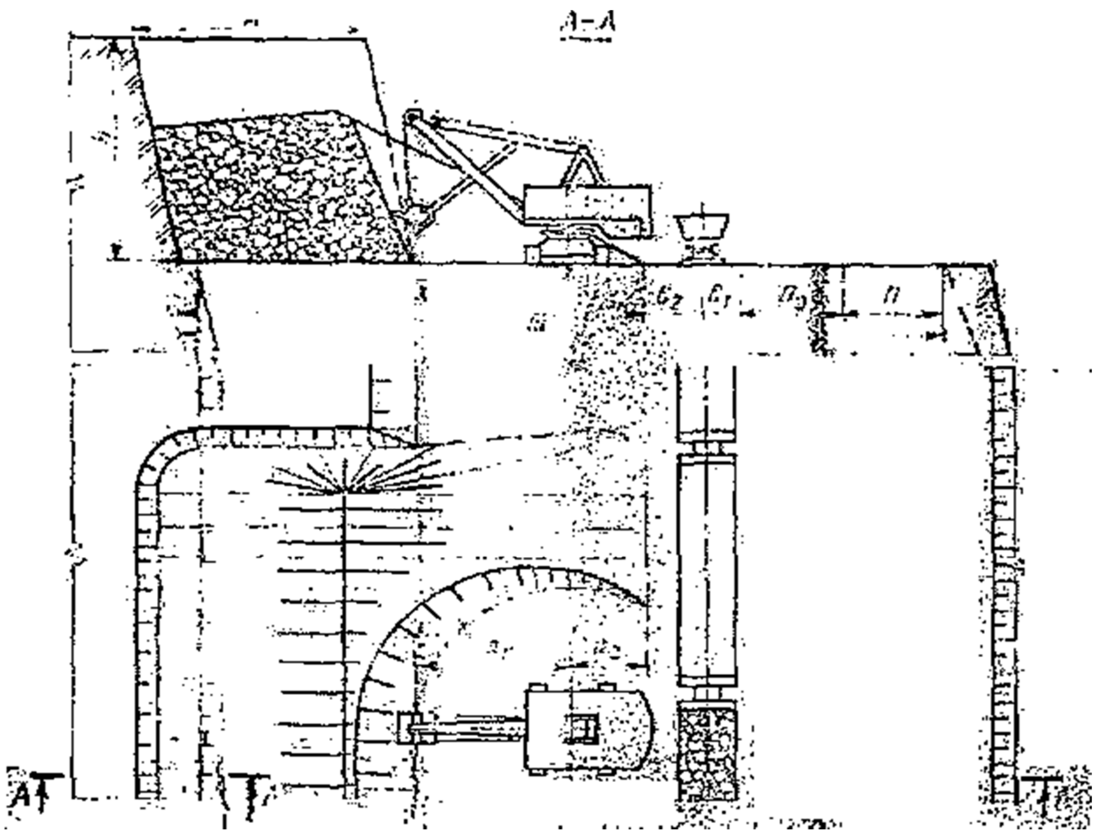

Fig.4. The scheme of the development of rocks by a mechanical shovel in railway transport.

$$Ш = X + C_2 + C_1 + П_э + П = 47{,}6 + 2{,}5 + 2{,}5 + 6 + 6 = 64{,}6 \text{ м.}$$

where C_2 and C_1 - distances from the axis of the path, respectively, to the lower edge of the camber and the power supply strip, м;

$П_э$ и $П$ - ширина полос для размещения соответственно устройств электроснабжения и дополнительного оборудования, м.

EXAMPLE 4. Determine the maximum height of the ledge, the width of the approach along the whole and the minimum width of the working platform when developing the ledge with an EKG-4y excavator with top loading into VS-85 dumpcars (Fig. 5).

The rocks being developed are easily explosive, with a strength coefficient of M. M. Protodiakonov f = 6; the slope angle of the ledge α = 80 °, the angle of stable slope α0 = 70°.

Decision. With a steep angle of a stable slope of the ledge, its height h is limited by the maximum height of unloading the excavator $H_{p.max}$:

$$h_{\max} = H_{p.\max} - h_{\partial} - c_1 - h_p ,$$

where h_{∂}- dumpcar height, м;

c_1 - minimum gap between bucket and transport vessel, м;

h_p - height of the upper structure of the railway track, м.

$$h_{\max} = 17{,}5 - 3{,}3 - 0{,}4 - 0{,}5 = 13{,}3 \text{ м.}$$

We accept h_{max} =13

Width of the inner part of the excavator approach:

according to loading conditions

$$\grave{a}_1 \le R_p - C_3 - h \cdot ctg\alpha_0 ; \;\; \grave{a}_1 \le 18{,}7 - 2{,}5 - 13 \cdot ctg70^0 ;$$

$$a_1 \le 11{,}5 \text{ м;}$$

by scooping conditions

$$a_1 \le R_{ч.у} ; \;\; a_1 \le 16{,}5 \text{ м;}$$

according to the conditions of safe rotation of the excavator

$$\grave{a}_1 \ge R_k + 1 - z_k \cdot ctg\alpha ; \;\; \grave{a}_1 \ge 7 + 1 - 2{,}8 \cdot ctg\ 80^0 ; \;\; \grave{a}_1 \ge 7{,}5 \text{ м;}$$

when moving the front from the hanging side to the recumbent

$$L_{\acute{a}} \le \frac{12 \cdot 80000}{15 \cdot 57{,}1} ; \;\; L_б \le 1120 \text{ м.}$$

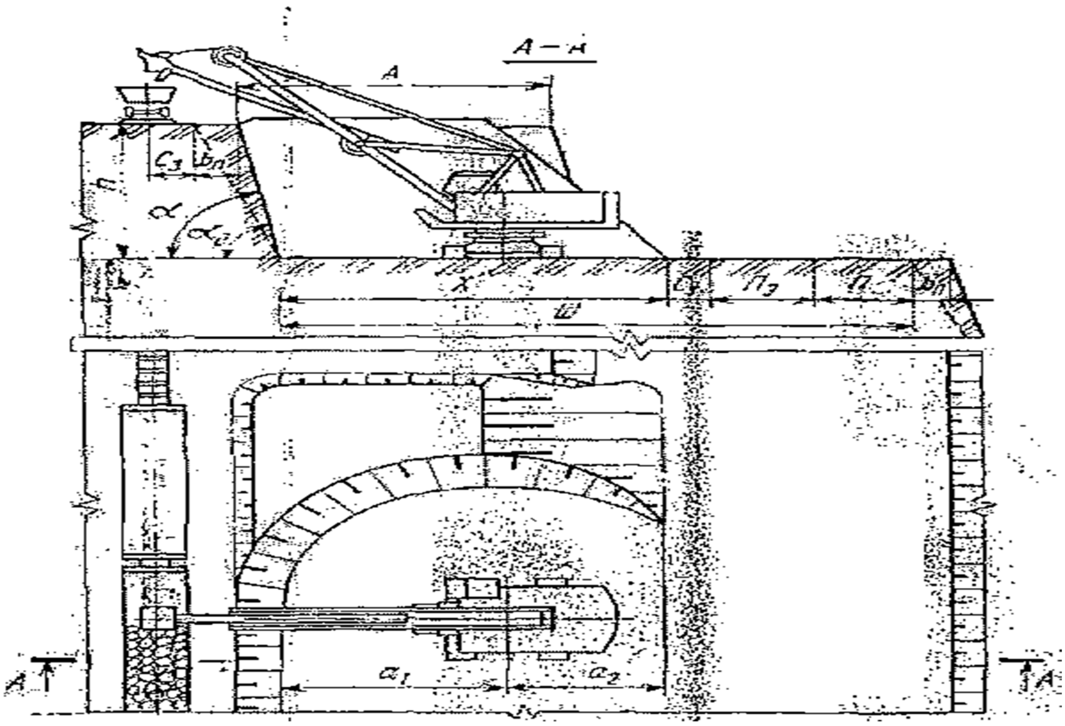

Fig. 5. The scheme of the development of rocks by a mehlopatoy with top loading.

The minimum length of the excavator block under the conditions of providing the excavator with an exploded rock mass

$$L_{\acute{a}} \geq \frac{Q \cdot T_{\hat{a}}}{A \cdot h}; \;\; L_{\acute{a}} \geq \frac{80000 \cdot 2}{16 \cdot 15}$$

$$L_{б} \geq 667 \text{ м.}$$

Thus, the length of the excavator block from the hanging side of the deposit should be within 667 – 912 m, and from the lying side - 667-1120m.

Task 1. Determine the minimum width of the working platform when excavating soft overburden rocks with an EKG-12.5 excavator using railway transport (Fig. 1).

Task 2.Determine the maximum height of the ledge and the width of the working platform when loading soft rocks into dumpcars 2VS-105 with an EKG-10 excavator (Fig. 2).

The slope angle of the developed ledge α = 63 °, the angle of the stable slope of the ledge α0 = 470.

Task 3. Determine the minimum width of the work site when developing a ledge with a height of h = 25 m, with an EKG-10 excavator using road transport and drilling and blasting operations (Fig. 4).

The annual productivity of the excavator Q = 2.7 million m3, the length of the work front Lb = 1100 m; the specific consumption of explosives qr = 0.7 kg / m3; the resistance line along the sole W = 8.7 m; the frequency of production of mass explosions Tv - once a month.

Task 4.To determine the maximum height of the ledge, the width of the approach along the whole and the minimum width of the working platform when developing the ledge with an EKG-12.5 excavator with top loading into VS-85 dumpcars (Fig. 5).

The rocks being developed are easily explosive, with a strength coefficient of M. M. Protodiakonov f = 8; the slope angle of the ledge α = 82 °, the angle of stable slope $\alpha 0$=71°.

PRACTICAL WORK NO. 19
DETERMINATION OF PARAMETERS OF THE TRANSPORT DUMP SYSTEM OF DEVELOPMENT

Example 1. Determine the required width of the strip of uncovered reserves for the winter period of Al during the development of the deposit according to the transport-dump scheme.

The productivity of the quarry for minerals Wi=5 million tons/year; the duration of the winter stop of the stripping complex Nz = 3 months; the thickness of the mineral reservoir t = 5 m; the length of the front of mining operations Lf.i = 2500 m; the density of the mineral γ=1.5 t/m3; the coefficient of extraction of the mineral Qizv.=0.95; the minimum allowable advance of the dumper by the mining face l1 = 150 m; the minimum allowable advance of the mining face by the dumper l2 = 250 m; the width of the approach A=70 м.

Решение.

$$\grave{A}_{\text{ç}} = \frac{W_{\text{è}} \cdot N_{\text{ç}}}{12 \cdot m \cdot L_{\text{ô.è}} \cdot \gamma \cdot K_{\text{èçâ}}} - \frac{A\left(L_{\text{ô.è}} - l_1 - l_2\right)}{L_{\text{ô.è}}} =$$

$$= \frac{5000000 \cdot 3}{12 \cdot 5 \cdot 2500 \cdot 1{,}5 \cdot 0{,}95} - \frac{70 \cdot (2500 - 150 - 250)}{2500} = 11{,}4 \text{ м.}$$

Example 2. Determine the performance of the stripping complex Qb.k, the required unloading height Nr.o and the unloading radius Rr.o of the cantilever dumper.

The capacity of the horizontal mineral reservoir t = 1 m; the height of the stripping ledge, worked out according to the transport-dump scheme, H = 35 m; the coefficient of loosening of rocks in the dump Kr = 1.15; the length of the front of mining operations Lf.i = 2000 m; the length of the front of stripping operations Lf.v = 2100 m; the length of the front of the dump Lf.o=1970 m; the width of the overburden A = 50 m; the angle of the natural slope of rocks in the dump βe = 38 °; the stable angle of the slope of the dump β = 22°; the angle of the slope of the mining ledge α = 45 °; the productivity of the quarry Wu = 2 million tons; the density of the mineral γ =2 t/m3; the coefficient of extraction of the mineral Kizv. = 0.97; the duration of the winter stop of the stripping complex Nz = 3 months; the dumper is located on the roof of the mineral formation.

Consider the work without changing the distance between the overburden excavator and the dumper and with a change in distance by 50 m (the value of telescopicity = 50 m).

Decision. 1. The productivity of the stripping complex, which provides the specified productivity of the quarry for minerals

$$Q_{\hat{a}.\hat{e}} = \frac{H \cdot L_{\hat{o}.\hat{a}} \cdot W_{\grave{e}}}{m \cdot L_{\hat{o}.\grave{e}} \cdot K_{\grave{e}\c{c}\hat{a}} \cdot \gamma} = \frac{35 \cdot 2100 \cdot 2 \cdot 10^{6}}{2 \cdot 2000 \cdot 0{,}97 \cdot 2} = 19 \text{ млн.м}^3\text{/год.}$$

2. *The width of the strip of prepared stocks for mining in winter*

$$A_{\c{c}} = \frac{W_{\grave{e}} \cdot N_{\c{c}}}{12 \cdot m \cdot L_{\hat{o}.\grave{e}} \cdot \gamma \cdot K_{\grave{e}\c{c}\hat{a}.}} - \frac{A\left(L_{\hat{o}.\grave{e}} - l_1 - l_2\right)}{L_{\hat{o}.\grave{e}}},$$

where l_1 – minimum advance of the dumper by the mining face, м;

l_2 – minimum advance of the mining face by the dumper, m; we accept l_1=100 м и l_2=150 м according to the conditions of maneuvering of the dumper.

$$\grave{A}_{\c{c}} = \frac{2 \cdot 10^6 \cdot 3}{12 \cdot 2 \cdot 2000 \cdot 2 \cdot 0{,}97} - \frac{50 \cdot (2000 - 100 - 150)}{2000} = 20{,}7 \text{ м.}$$

3. *Height of the inner blade*

$$H_0 = K_{\hat{o}} \cdot K_{\eth} \cdot K_{\c{c}\grave{a}\tilde{o}} \cdot H + \frac{0{,}25 \cdot A \cdot tg\beta_1}{K_{\c{c}\grave{a}\tilde{o}}},$$

where $К_ф = L_{ф.и}/L_{ф.о}$=2100/1970=1,07 – a coefficient that takes into account the reduction of the front of dump operations compared to the front of stripping operations;

$К_{зах}$ – the coefficient of reduction of the entry width, equal to the ratio of the width of the stripping entry to the width of the dump entry. When working without changing the distance between the overburden excavator and the dumper *(T*= 0) $К_{зах}$=1; when creating reserves for mining in winter due to the telescopicity of the complex

$$\hat{E}_{\c{c}\grave{a}\tilde{o}} = \frac{Q_{\hat{a}.\hat{e}}}{Q_{\hat{a}.\hat{e}} - T \cdot L_{\hat{o}.\hat{a}} \cdot H} = \frac{18 \cdot 10^6}{18 \cdot 10^6 - 50 \cdot 2100 \cdot 35} = 1.257.$$

By T=0; H_o=1,07·1,15·1·35 + 0,25·50tg38^0=53 м;

By T = 50 м; H_0= 1,07·1,15·1,257·35 + 0,25·50tg38°/1,257 = 62 м.

4. *Required height of unloading of the dumper*

$$H_{p.o} = H_o - m + p,$$

where *p* – minimum distance between the blade ridge and the dumper console (according to safety standards p=1,5 м when using cantilever dumpers p = 2 m when using transport dump bridges).

by T= 0, $H_{р.о}$ = 53–2 +1,5=52,5 м;

by T =50 м, $H_{р.о}$= 62–2+1,5=61,5 м.

5. *The required radius of unloading of the dumper*

$$R_p = A_{\c{c}} + H_0 \cdot ctg\beta_1 + B + m \cdot ctg\alpha + \acute{A} - \grave{O} - l_{\tilde{n}.\ddot{\i}},$$

where B – the minimum distance from the axis of movement of the dumper to the upper edge of the mining ledge, equal to half the width of the running device of the dumper and the gap between the running device and the brow of the ledge, we accept B=25 м;

Б – the width of the free lane between the dump and the mining ledge, including the width of the drainage ditch and the transport lane, we accept Б=5 м:

$l_{c.n}$– horizontal distance of free movement of rock to the ridge of the dump ($l_{c.n}$ =2÷3 м).

by T=0, R_p=20,7+53ctg22^0+25+2ctg45°-0-3=181 м;

by T = 50 м, R_p =20,7+62ctg22°+25+2ctg45°+5-50-3=153 м,

Example 3. Determine the maximum height of the overburden ledge, worked out according to the transport-dump scheme using the OSH-1600/110 dumper.

The maximum unloading height of the dumper is 34 m, the maximum unloading radius is Rp = 110 m; the dumper is located on the roof of the mineral reservoir with a capacity of m = 3 m; the required width of the strip of uncovered reserves for the winter period, located under the console, Ap = 20 m; the minimum distance from the axis of the dumper to the upper edge of the mining ledge B=15 m; width of the free strip between the dump and the mining ledge B = 5 m; angles of stable slope of the mining ledge (α =45°; blade β = 25°; angle of natural slope of rocks in the dump βe = 35°; length of the overburden front Lf.v = 1100 m; the length of the front of the dump works Lf.o = 1050 m; the width of the overburden and dump approaches A = 24 m.

Decision. We determine the maximum height of the dump, which the dumper can pour out:

$$H_o \le H_p + m - p\,;$$

$$H_o \le 34 + 3 - 1{,}5 = 35{,}5 \text{ м.}$$

$$H_o \le \left(R_ð - A_ç - B - m \cdot ctg\,\alpha - Á + Ò + l_{ñ.ï}\right) \cdot tg\,\beta$$

$$H_o \le \left(110 - 20 - 15 - 3 \cdot ctg\,45^0 - 5 + 0 + 2\right) \cdot tg\,25^0 = 32 \text{ м.}$$

We accept a smaller value H_o=32 м.

We find the maximum height of the overburden ledge:

$$Í = \frac{\left(Í_í - 0{,}25 \cdot À_í \cdot tg\beta_e\right) \cdot L_{ô.î}}{K_ð L_{ô.â}} = \frac{\left(32 - 0{,}25 \cdot 24\,tg\,35^0\right) \cdot 1050}{1{,}2 * 1100} = 22{,}1 \text{ м,}$$

Example 4. Determine the required length of the reloader (connecting bridge) L.m. for reloading the rock from a rotary excavator located on the roof of the mineral reservoir to a cantilever dumper located on the rock.

The thickness of the mineral reservoir m = 20 m; the slope angle of the working side of the mineral φ = 20 °; the width of the free strip between the dump and the mining ledge B = 7 m; the height of the slope = 20 m, the angle of its slope βp = 35

°; the minimum safe distance from the axes of the bridge supports to the upper edge of the mining ledge a1=20 m, to the upper edge of the cliff a_2=25 м.

Decision.

$$L_{\tilde{n}.\grave{i}} = \grave{a}_1 + \grave{a}_2 + m \cdot ctg\,\varphi + H_{\ddot{i}} \cdot ctg\,\beta_{\ddot{i}} =$$
$$= 20 + 25 + 20 \cdot ctg\,20^0 + 20 \cdot ctg\,35^0 = 128 \text{ м.}$$

Example 5. Determine the required length of the superstructure and the departure of the transport-dump bridge console.

The covering rocks with a capacity of H = 55 m are worked out by two ledges with a height of H1 = 30 m (lower) and H2 = 25 m (upper); the bridge supports are located on the roof of the lower overburden ledge and the rock, whose height is Np = 25 m; the power of the coal seam being developed is m = 15 m; coal density γ = 1.2 t/m3; extraction coefficient Kzv.=0.98; annual productivity of the quarry for coal Wi =7 million tons; the movement of the work front is parallel; the length of the work front for extraction Lf.i = 2000 m, for overburden Lf.v =2100 m, dump work Lf.o =1950 m; slope angles of overburden ledges δ=45°, mining ledge α= 60°, dump tiers β=35°; safe distances from the upper edge of the lower overburden ledge and the upper edge of the dump to the axes of the supports of the transport-dump bridge, respectively a1 = 20 m and a2 = 30 m; the distance between the lower brow of the second tier of the dump and the axis of the bridge support a3 = 15 m; the width of the safety berm on the roof of the formation bp = 10 m; the width of the free space between the dump and the mining ledge B = 5 m.

Decision.1. The required width of the strip of winter reserves of exposed coal for the duration of the winter stop of the stripping complex N3 = 3 months.

$$A_{\c{c}} = \frac{W_{\grave{e}} \cdot N_{\c{c}}}{12 \cdot m \cdot L_{\hat{o}.\grave{e}} \cdot \gamma \cdot K_{\grave{e}\c{c}\hat{a}}} = \frac{7 \cdot 10^6 \cdot 3}{12 \cdot 15 \cdot 2000 \cdot 1{,}2 \cdot 0{,}98} = 50 \text{ м.}$$

2. Required bridge span length

$$L_{\ddot{i}.\tilde{n}} = a_1 + a_2 + H_1 \cdot ctg\delta + b_{\ddot{i}} + A_{\c{c}} + m \cdot ctg\alpha + \acute{A} + \acute{I}_{\ddot{i}} \cdot ctg\beta =$$
$$= 20 + 30 + 30 \cdot ctg\,45^0 + 10 + 50 + 15 \cdot ctg\,60^0 + 5 + 25 \cdot ctg\,35^0 = 190$$

м.

3. Blade height

$$H_o = \frac{K_p \cdot L_{\hat{o}.\hat{a}} \cdot H}{L_{\hat{o}.\hat{i}}} = \frac{1{,}2 \cdot 2100 \cdot 55}{1950} = 71 \text{ м.}$$

4. Required length of the dump console

$$L_{o.k} = (H_o - H_n) ctg\,\beta + a_3 - l_{c.n},$$

where $l_{c.n}$- horizontal distance of free flight of the rock to the dump, м.

$$L_{o.k} = (71 - 25)ctg35^0 + 15 - 3 = 78 \text{ м.}$$

Task 1. To determine the required width of the strip of uncovered reserves for the winter period of Al during the development of the deposit according to the transport-dump scheme.

The productivity of the quarry for minerals Wi = 7 million tons/year; the duration of the winter stop of the stripping complex Nz = 4 months; the thickness of the mineral reservoir t =6 m; the length of the front of mining operations Lf.i = 2700 m; the density of the mineral γ =1.4 t/m3; the coefficient of extraction of the mineral Qizv.=0.93; the minimum allowable advance of the dumper by the mining face l1 = 160 m; the minimum allowable advance of the mining face by the dumper l2 = 270 m; the width of the entry A = 68 m.

Task 2. Determine the performance of the stripping complex Qb.k, the required unloading height Nr.o and the unloading radius Rr.o of the cantilever dumper.

The capacity of the horizontal mineral reservoir t = 1.5 m; the height of the stripping ledge, worked out according to the transport-dump scheme, H = 37 m; the coefficient of loosening of rocks in the dump Kr = 1.15; the length of the front of mining operations Lf.i = 2100 m; the length of the front of stripping operations Lf.v = 2300 m; the length of the front of the dump works Lf.o=1987 m; the width of the overburden A = 55 m; the angle of natural slope of rocks in the dump βe = 37°; stable slope angle of the dump β = 23°; slope angle of the mining ledge α= 45°; quarry productivity Wu = 3 million tons; mineral density γ =2 t/m3; the coefficient of extraction of the mineral Kizv. = 0.97; the duration of the winter stop of the stripping complex Nz = 3 months; the dumper is located on the roof of the mineral formation.

Consider the operation without changing the distance between the overburden excavator and the dumper and with a change in distance by 60 m (the value of telescopicity T = 50 m).

Task 3. Determine the maximum height of the overburden ledge, worked out according to the transport-dump scheme using the OSH-1600/110 dumper.

The maximum unloading height of the dumper is 34 m, the maximum unloading radius is Rp = 110 m; the dumper is located on the roof of the mineral reservoir with a capacity of m = 5 m; the required width of the strip of uncovered reserves for the winter period, located under the console, Ap = 25 m; the minimum distance from the axis of the dumper to the upper edge of the mining ledge B=17 m; width of the free strip between the dump and the mining ledge B = 6 m; angles of stable slope of the mining ledge (α = 45°; blade β = 27°; angle of natural slope of rocks in the dump βe = 36°; length of the overburden front Lf.v = 1200 m; the length of the front of the dump works Lf.o = 1070 m; the width of the overburden and dump approaches A = 25 m.

Task 4. Determine the required length of the loader (connecting bridge) Ls.m to overload the rock from a rotary excavator located on the roof of the mineral reservoir to a cantilever dumper located on the rock.

The thickness of the mineral reservoir m = 22 m; the angle of the slope of the working side of the mineral $\varphi = 23$ °; the width of the free strip between the dump and the mining ledge B = 7 m; the height of the incline = 22 m, the angle of its slope $\beta p = 37$ °; the minimum safe distance from the axes of the bridge supports to the upper edge of the mining ledge a1 = 23 m, up to the upper edge of the wall a2 = 27 m.

Task 5. Determine the required length of the superstructure and the departure of the transport dump bridge console.

The covering rocks with a capacity of H = 57 m are worked out by two ledges with a height of H1 = 32 m (lower) and H2 = 27 m (upper); the bridge supports are located on the roof of the lower overburden ledge and the rock, whose height is Np = 23 m; the power of the coal seam being developed is m = 17 m; coal density $\gamma = 1.2$ t/m3; extraction coefficient of Kzv.=0.98; annual productivity of the quarry for coal Wi =8 million tons; the movement of the work front is parallel; the length of the work front for extraction Lf.i =2100 m, for stripping Lf.b=2300 m, dump work Lf.o =1987 m; slope angles of overburden ledges δ=45°, mining ledge α= 61°, dump tiers β = 37°; safe distances from the upper edge of the lower overburden ledge and the upper edge of the dump to the axes of the supports of the transport-dump bridge, respectively a1 = 25 m and a2 = 35 m; the distance between the lower brow of the second tier of the dump and the axis of the bridge support a3 = 16 m; the width of the safety berm on the roof of the formation bp = 10 m; the width of the free space between the dump and the mining ledge B = 7 m.

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GLOSSARY

by discipline:
“TECHNOLOGY AND COMPLEX MECHANIZATION OF OPEN-PIT MINING”

A face is the end of a pit, mine, well or other mining work moving in space.

Mining operations - a complex of works related to the excavation of rocks, sinking, conducting and maintaining mining operations. There are underground and open-pit mining operations.

Blasting - works performed by the impact of an explosion on natural rocks in order to control their destruction and displacement or change the structure and shape.

Explosion safety is the state of the production process in which the possibility of an explosion is excluded, or, if it occurs, the harmful effects of dangerous and harmful factors caused by an explosion are prevented on people; and the preservation of material values is ensured.

Sudden release - spontaneous instantaneous destruction of a part of the massif near the bottom of the mine, accompanied by the rejection of rock, ore, coal, enhanced gas emission.

Overburden rocks - rocks:

-covering the extracted mineral;

-extracted from the depths together with minerals.

The developed space is the space formed in the bowels after the extraction of a mineral.

Mining is an artificial cavity under the ground or a depression on its surface created as a result of mining operations.

Mining and capital works - works on carrying out capital open and underground mining.

Mining and technological properties of a rock are properties that characterize the interaction between a rock and a tool, mechanism or technological process during mining operations.

A mountain blow is an instantaneous brittle destruction of a whole or a marginal part of an array, manifested in the form of an ore or rock ejection into underground workings.

Boundary angles are angles external to the worked-out space formed on vertical sections along the main sections of the slide by horizontal lines and lines sequentially drawn in bedrock, later sediments and sediments connecting the boundary of the treatment work with the boundary of the zone of influence of underground development on the Earth's surface.

Rock collapse is the displacement of rocks with the destruction of layers and the separation of individual pieces and blocks from the array.

Open-pit mining - Open-pit mining - mining operations carried out directly from the earth's surface in open-pit mines.

Treatment mine workings are mine workings formed directly during the extraction of minerals from the deposit.

Cleaning works - works on extraction of minerals from the deposit by underground method. During cleaning operations, there are: joint (gross) extraction of minerals and- separate (selective) extraction of minerals, in which individual grades of ores, coal, layers of rocks, etc. are removed separately.

A rock mass is a part of the earth's crust that is affected by mining operations.

The collapse prism is an unstable part of the ledge array from the side of its slope, enclosed between the working and stable angles of the ledge slope.

Rock displacement is the displacement and deformation of rocks as a result of their imbalance under the influence of mining, changes in the physical and mechanical properties of rocks. The displacement of rocks can extend over the entire thickness to the earth's surface.

Concussive blasting - explosive work performed by

- in compliance with special requirements in arrays prone to sudden emissions;
- to reduce rock pressure and prevent sudden emissions.

The angles of displacement are the angles external to the worked-out space formed on vertical sections along the main sections of the mulda of displacement with full moonlighting by horizontal lines and lines sequentially drawn in bedrock, later sediments and sediments connecting the boundary of the development with the boundary of the zone of dangerous influence on the earth's surface.

Rock pressure management is a set of measures to regulate the manifestations of rock pressure in the working space of treatment faces and mine workings to create the necessary production conditions.

Management of the stress-strain state of a mountain massif is a set of measures to prevent the mass collapse of lateral and overlying rocks of treatment faces to create safe production conditions.

The front of the cleaning works is the spatial location of the line of the cleaning faces on the wing of the mine (mine), the shaft layer, floor, tier.

Cementation of rocks is a method of artificially fixing rocks and increasing their water resistance by pumping cement mortar into them.

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Methodological instructions for the implementation of the course project

by discipline:
“TECHNOLOGY AND COMPLEX MECHANIZATION OF OPEN-PIT MINING”

CONTENT

INTRODUCTION

The economic development of our country is inextricably linked with the further development of the mining industry. Our Republic has large reserves of minerals. Currently, the largest quarries for the extraction of gold, silver, copper and coal are operating. The development of open-pit deposits, in comparison with underground, provides significantly better technical and economic indicators. At the same time, it is associated with a number of negative consequences: land disturbance, changes in microclimate and water balance, etc.

There are more than 440 mines, mines, mineral processing plants operating in the Republic of Uzbekistan, including such large enterprises as the Navoi Mining and Metallurgical Combine, Almalyk Mining and Metallurgical Combine. There are more than 2,700 deposits and manifestations of minerals in the Republic of Uzbekistan, of which 940 have already been explored, including 155 deposits of oil and gas and condensate, more than 40 precious metals, 42 rare, non-ferrous and radioactive metals, 463 building materials. The Republic of Uzbekistan occupies a leading place not only in the CIS countries in terms of confirmed reserves and their prospective increase. So, in terms of gold reserves, it occupies the 4th place, and in terms of its production - the 7th place. Less than 20% of the proven gold reserves have been worked out, which suggests the possibility of a significant expansion of its production.

The term "Technology" generally means a set of knowledge about the methods, means and organization of performing any production and technical work.

The technology of field development is a set of interrelated processes, methods and techniques of mechanized mining, based on fundamental knowledge of the laws of development and capabilities of technical means.

The technology of the open-pit mining method includes two aspects: the technology of production processes (excavation, movement and storage of rocks) and the technology of open-pit mining (construction and development as the field is developed in time and space of the quarry as a mining complex).

THE ORDER OF THE WORK

The course project is carried out according to individual tasks, the variants of which are presented in appendix 8.

Initial data for work:

The nature of the daytime surface (horizontal, inclined).

The shape of the ore body is plast –shaped.

Horizontal capacity of the ore body – $m_г$, м.

The length of the deposit along the strike – $L_р$, м.

The angle of incidence of the deposit – $\alpha_{р.т}$, град.

The slope of the upper contact of the ore body with sediments, deg.

Vertical sediment capacity – $h_м$, м.

Characteristics of overburden, ore, sediment rocks (set according to Appendix 1):

– ore density – $\delta_р$, т/м^3.

– rock density – $\delta_в$, т/м^3.

– sediment density – $\delta_м$ = 2 т/м^3.

Basic career depth – $H_к$, м.

The development system is with the deepening of the quarry and the removal of overburden rocks into external dumps.

Preparation of excavator blocks is longitudinal.

The direction of deepening of the quarry is by contact of the recumbent side of the deposit.

The height of the ledge along the rocky zone – $h_у$ = 15 м.

Height of the ledge along the moraine – $h_{у.м}$, м, selected according to $h_м$.

Angle of inclination of the final side of the quarry $\beta_{к.в.}$, град.:

1) on the rock:

– for hanging side $\beta_{к.в.} = 41°$;

– for the recumbent side $\beta_{к.л} = \beta_{к.в}$ (by $\beta_{к.в} < \alpha_{р.т}$); $\beta_{к.л} = \alpha_{р.т}$ (by $\beta_{к.в} > \alpha_{р.т}$);

2) by deposits:

$\beta_{к.м.} = 16°$.

Width of the quarry bottom $b_д$ = 40 м.

Excavation of rock mass – with an excavator of the type ЭКГ.

Bucket capacity E, м^3.

Technological transport – automobile.

Highways are two–lane.

The initial data is given on the first page after the title.

Identify:

1. Parameters of the quarry at the end of mining and build the contours of the quarry on the cross section and in the plan.

2. The volume of balance and industrial ore reserves in the contour of the quarry.

3. Volumes of rock mass in the quarry and overburden rocks exported to the dump.

4. Overburden coefficients (average and operational of the main working period of the quarry).

5. Excavator performance.

6. The loading capacity of the dump truck.

7. Parameters and indicators of the development system:

7.1. The angle of the slope of the working ledges.

7.2. The width of the excavator approach and the working area.

7.3. The angle of inclination of the working side.

7.4. The length of the active front of the excavator.

7.5. The speed of moving the working ledges.

7.6. The rate of annual lowering of the bottom of the quarry.

7.7. Adjustment of the parameters of the development system (carried out as necessary in accordance with Appendix 7).

8. The annual productivity of the quarry for ore, stripping, rock mass and the inventory of excavators.

The task for the above points is given after the initial data.

The report on the completion of the work consists of an explanatory note, drawn up in accordance with the requirements for writing a course project, and drawings.

After the initial data and a brief description of the task, sections with calculations in accordance with the task should follow.

The final results of calculations should be rounded:

– for quarry parameters – up to tens of meters;

– for volumes – up to tens thous. $м^3$;

– for stocks – up to tens of thousands. $м^3$ or tens of thousands. т;

– for the parameters of the development system – up to meters;

– for the lengths of the excavator work fronts – up to hundreds of meters.

After the literal spelling of the formula, its numerical form should follow, and then the result.

Drawings are made on an A4 sheet in scale, the signature is placed at the bottom, a frame is required, a stamp is not required.

1. THE PARAMETERS OF THE QUARRY AT THE END OF WORKING OUT AND THE CONSTRUCTION OF THE CONTOURS OF THE QUARRY

1.1. THE CONTOUR OF THE QUARRY ON THE CROSS SECTION

On the horizon of the final bottom of the quarry $H_к$ a horizontal segment equal to the width of the bottom is built from the lying contact of the ore body ($b_д$ = 40 м).

From the contours of the bottom , the lines of the end sides of the quarry are drawn along the rocky zone at angles $\beta_{к.в}$ and $\beta_{к.л}$. From the points of intersection of the lines drawn with the sediment horizon at a depth of rm, the lines of the sides of the quarry are drawn along the sediments at an angle $\beta_{к.м}$ before the intersection with the daytime surface (Fig. 1).

The width of the quarry along the rocky zone $B_{к.с.}$ (м) determined by the formula:

$$B_{к.с.} = (H_к – h_м)\cdot(ctg\ \beta_{к.в} + ctg\ \beta_{к.л}) + b_д. \qquad (1)$$

The width of the quarry on the surface of the $B_к$ (m) is determined by the formula

$$B_к = B_{к.с} + 2\ h_м\ ctg\ 16^\circ \qquad (2)$$

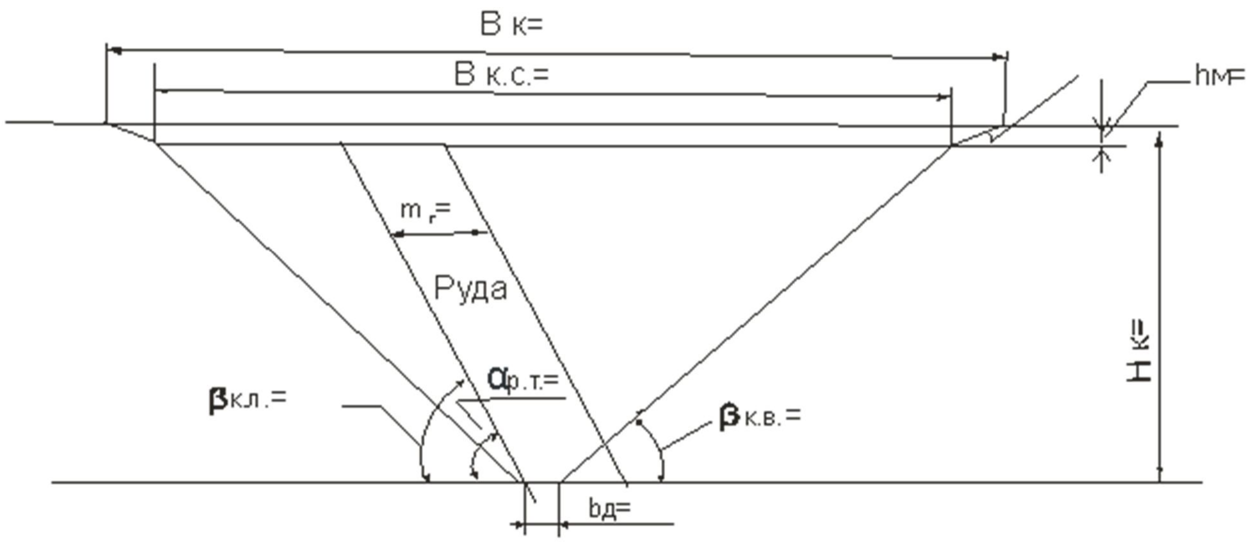


Рис. 1. Контур карьера на поперечном разрезе.
М 1:10000

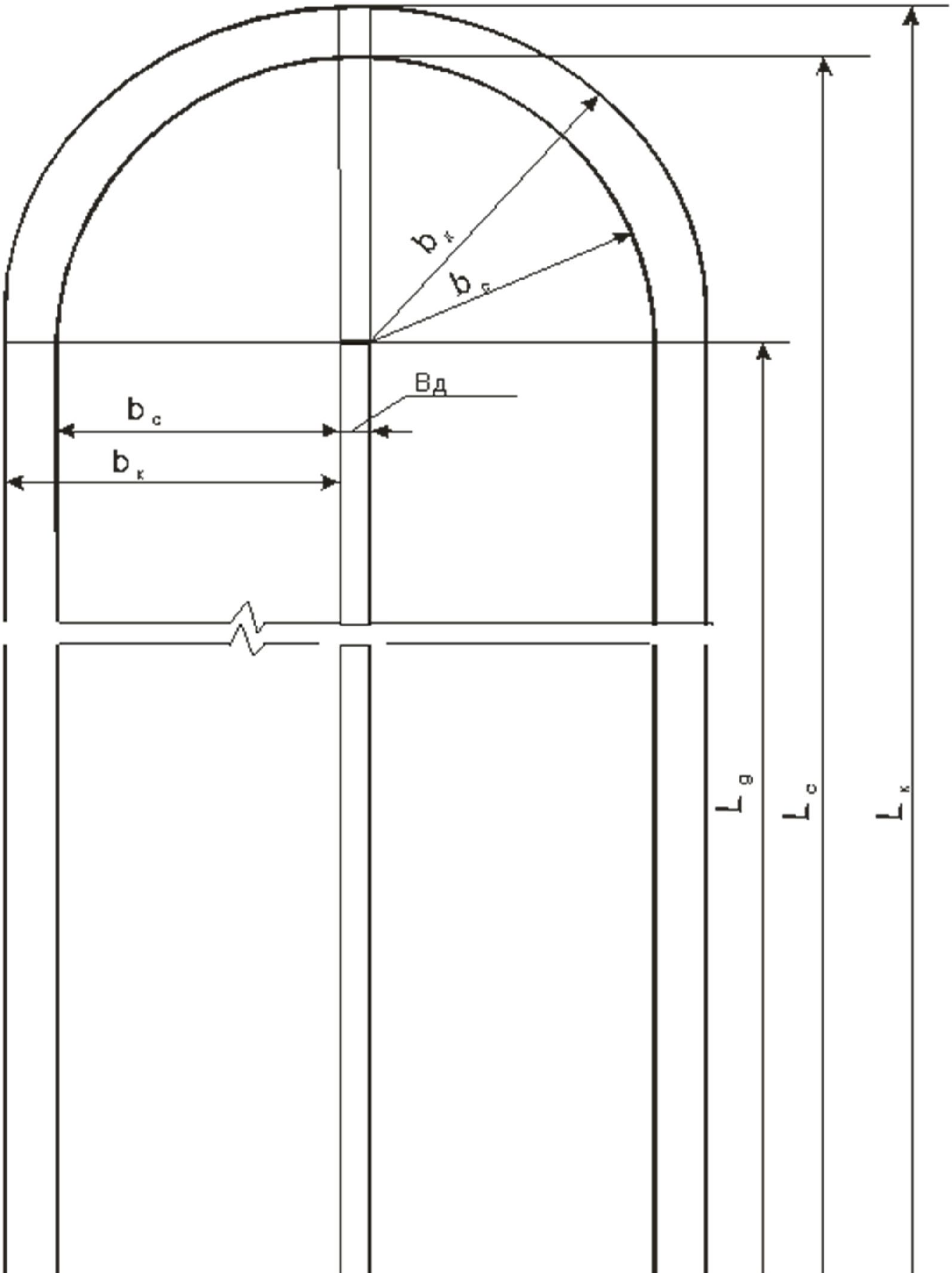


Рис.2. План границ карьера.
М 1:10000

1.2. OUTLINE PLAN OF THE QUARRY

The length of the bottom of the quarry is equal to the length of the ore body $L_д = L_р$.

To build the end sides of the quarry in the plan, the radii of rounding along the rocky zone and on the surface are determined (Fig. 2).

The radii of rounding at the ends of the quarry along the rocky zone are determined:

– from the hanging side $b_с$ (м) – according to the formula

$$b_с = (H_к – h_м) \operatorname{ctg} \beta_{к.в} \tag{3}$$

– from the recumbent side $b'_с$ (м) – according to the formula

$$b'_с = (H_к – h_м) \operatorname{ctg} \beta_{к.л} \tag{4}$$

The radii of the curves at the ends of the quarry on the surface are determined by – from the hanging side $b_к$ (м) – according to the formula

$$b_к = b_с + h_м \operatorname{ctg} \beta_{к.м} \tag{5}$$

– from the recumbent side $b'_к$ (м) – according to the formula

$$b'_к = b'_с + h_м \operatorname{ctg} \beta_{к.м} \tag{6}$$

The length of the quarry along the rocky zone $L_{ск}$ (м) determined by the formula

$$L_{ск} = L_д + 2\, b_с \tag{7}$$

The length of the quarry on the surface $L_к$ (м) calculated by the formula

$$L_к = L_д + 2\, b_к \tag{8}$$

2. BALANCE SHEET AND INDUSTRIAL ORE RESERVES

2.1. BALANCE RESERVES OF ORE IN THE CONTOURS OF THE QUARRY

According to the condition of the task, the reserves are limited in length by the extension of the ore body, in depth – by the end sides of the quarry and its bottom. Since the ore body is represented by a regular-shaped formation, its volume $P_б$ (thous. $м^3$) in the contour of the quarry, it is calculated as the product of the area of the ore body on the cross section and the length $L_р$. The area of the quarry in the cross section is calculated as the area of a rectangle with a height of $(H_к – h_м)$ and the width of mg minus the area of the triangle formed by the hanging side of the quarry, the hanging contact of the ore body and the horizon line at the mark $H_к$.

Formula for calculating the balance reserves of ore in the contours of the quarry $P_б$ (thous. $м^3$) has the form:

$$P_б = \left[m_т (H_к - h_м) - \frac{(m_т - b_д)^2}{(\text{ctg}\,\alpha_{рт} + \text{ctg}\,\beta_{кв})2} \right] L_р \tag{9}$$

2.2. INDUSTRIAL ORE RESERVES

Industrial ore reserves P (thous. $м^3$) calculated taking into account losses and dilution:

$$P = P_б \frac{1-\eta}{1-\rho}, \tag{10}$$

where h and ρ – coefficients of losses and dilution, respectively, fractions of units. (appendix 4).

Meaning P calculated in thousands of tons, rounded down to tens of thousands of tons and converted to thousands of m3 without rounding.

3. VOLUMES OF ROCK MASS AND OVERBURDEN IN THE QUARRY

3.1. VOLUME OF ROCK MASS

The volume of the rock mass of a regular-shaped quarry is calculated as the sum of the volumes of a central straight prism of trapezoidal cross-section, four quarters of cones at the ends of the quarry and two triangular prisms with a thickness equal to the width of the bottom (40 m) at the ends of the quarry.

Since there is a sediment zone, the figure of the quarry is further divided into rocky and alluvial parts.

The calculation of the volume is carried out piecemeal using pre-calculated radii of rounding.

The volume of rock mass is calculated

– for a straight section $V_{г.м.п.}$ (thous. $м^3$) – according to the formulas:

(by $\beta_{к.л.} = \beta_{к.в.}$); (11)

$$V_{гмп.} = \left[(b_с + b_д)(H_п - h_м) + (b_с + b_к + b_д) h_м \right] L_р$$

(by $\beta_{к.л.} = \alpha_{р.т}$); (12)

$$V_{гмп.} = \left[\frac{b_с + b_с' + 2b_д}{2}(H_к - h_м) + \frac{b_с + b_с' + 2b_д + b_к + b_к'}{2} h_м \right] L_р$$

– for end sections $V_{г.м.т.}$ (тыс. $м^3$) – according to the formulas:

(by $\beta_{к.л.} = \beta_{к.в.}$);

$$V_{\text{г.м.т.}} = \frac{\pi b_c^2}{3}(H_к - h_м) + b_c(H_к - h_м)b_д + \frac{\pi}{4}(b_c + b_к)^2 h_м + (b_c + b_к)h_м b_д \tag{13}$$

(by $\beta_{\text{к.л.}} = \alpha_{\text{р.т}}$).

$$V_{\text{г.м.т.}} = \frac{\pi b_c^2}{6}(H_к - h_м) + \frac{\pi b_c b_c'}{6}(H_к - h_м) + b_c(H_к - h_м)b_д + \frac{\pi}{8}(b_c + b_к)^2 \times$$
$$\times h_м + \frac{\pi}{8}(b_c + b_к)(b_c' + b_к')h_м + (b_c + b_к)h_м b_д. \tag{14}$$

Total volume of rock mass $V_{\text{г.м.}}$ (thous. м^3) equal to

$$V_{\text{г.м.}} = V_{\text{г.м.п}} + V_{\text{г.м.т}} \tag{15}$$

3.2. OVERBURDEN VOLUME

Overburden volume $V_в$ (thous. м^3) determined by the formula

$$V_в = V_{\text{г.м.}} - P. \tag{16}$$

4. STRIPPING COEFFICIENTS

4.1. AVERAGE STRIPPING RATIO

Average stripping ratio $n_{\text{ср.}}$ (м^3/м^3) determined by the formula

$$n_{\text{ср}} = \frac{V_в}{P}. \tag{17}$$

4.2. OPERATIONAL STRIPPING COEFFICIENT OF THE MAIN PERIOD OF OPERATION OF THE QUARRY

Operational stripping coefficient n_1 (м^3/м^3) determined by the formula

$$n_1 = \lambda(1 - \mu)n_{\text{ср}} \tag{18}$$

where λ_1 – the coefficient of unevenness of stripping operations of the main (first) period of operation of the quarry is usually 1.15–1.65 (for reservoir deposits of the correct shape – 1,2–1,35); μ – the share of the initial value of the stripping coefficient n0 from the average $n_{\text{ср}}$.

The fraction of the initial value of the stripping coefficient μ (fractions of units) is determined by the formula

$$\mu = \frac{n_0}{n_{\text{ср}}} \tag{19}$$

for overlaying rocks of low power $\mu = 0{,}05 - 0{,}2$.

5. THE ANNUAL PRODUCTIVITY OF THE EXCAVATOR

For simplified calculations, the annual productivity of the excavator Q (thous. м^3) it can be determined using the specific annual productivity of the excavator:

$$Q = q\,E, \tag{20}$$

where q – the specific annual productivity of the excavator per 1 m3 bucket capacity is usually in the range 120–170 thous. м^3/м^3. For this work, it is advisable to choose the value of q from the range 120–135 тыс. м^3/м^3.

6. DUMP TRUCK LOAD CAPACITY

The loading capacity of the dump truck is selected from the condition of the rational ratio of the bucket capacity of the excavator E and the dump truck body $V_к$. This ratio depends on the distance of cargo transportation: the longer the running-in shoulder, the larger the capacity of the body should be compared to E.

For the average values of the transportation distance (2-4 km), the rational ratio is in the range

$$E{:}V_к = 1{:}4 \div 1{:}6 \tag{21}$$

The ratio of 1:7 is used according to the practice of quarries in which heavy-duty dump trucks and excavators with a reduced bucket capacity are operated, since the typical parameters of the excavator bucket E are somewhat behind the values of the typical load capacities of dump trucks.

The weight of the rock in the excavator bucket g (t) is determined by the formula

$$g = E\,\delta_р\,K_э \tag{22}$$

where $K_э$ – the coefficient of excavation depends on the density of rocks (selected according to the application 5).

The range of the rational load capacity of the dump truck G (t) is

$$G = g\,4 \div g \cdot 7 \tag{23}$$

It is more expedient to choose a car with a larger load capacity, since it is more economical and environmentally friendly in operation. The choice is made from a typical range of dump trucks with a lifting capacity: 42, 75, 90, 105, 110, 120, 130, 150, 180 т.

7. DEVELOPMENT SYSTEM PARAMETERS

7.1. SLOPE ANGLE OF THE WORKING LEDGE

The highest value is selected for this type of breed according to the norms of VNIMI (appendix 1).

7.2. EXCAVATOR APPROACH WIDTH

The width of the excavator approach a (m) is determined from the expression

$$a = R_{ч.у}\ (1,5 \div 1,7) \tag{24}$$

where $R_{ч.у}$ – the scooping radius at the excavator standing level (selected by the application 6).

For rocks, it is more expedient to choose the largest integer value of a, since a large collapse of crushed rock is formed during multi-row blasting.

7.3. LENGTH OF THE ACTIVE FRONT PART OF THE EXCAVATOR

It is determined based on the minimum standard front for one excavator $L_{ф.min}$ (application 6).

The optimal length of the active front is within

$$L_ф = (1,5 \div 2,0)\ L_{ф.min},\ м. \tag{25}$$

For preliminary calculations , first set

$$L_ф = 1,5\ L_{ф.min},\ м. \tag{26}$$

7.4. THE SPEED OF MOVING THE WORKING LEDGES

The rate of movement of the working ledges Vu (m / year) is determined by the formula

$$V_у = \frac{Q}{h_у L_ф}\ . \tag{27}$$

7.5. WORKING AREA WIDTH

Working area width $B_{р.п.}$ (м) determined by two methods:

1) taking into account the standards of stocks ready for extraction ($\mu_г$) according to the formula:

$$B_{р.п.} = B_{тр.б.} + \mu_г \cdot V_у, \tag{28}$$

where $B_{тр.б.}$ – the width of the transport berm (according to the standards value $B_{тр.б}$ it is determined depending on the load capacity of the dump truck Q for a quarry with a depth of more than 50 m, appendix 2). The result is rounded up to 0.5 m up.

The minimum value of reserves ready for extraction according to the standards of the Giprorud Institute is 2.5 months, that is, their share of the year µg is 0.21 years.

Composition $\mu_г\ V_у$ denote as $\Delta B_{р.п.}$ – the required additional width of the working platform to account for the normative reserve of excavator stocks.

2) taking into account equality $\Delta B_{р.п.}$ the width of the selected excavator entry a.

Finally, the width of the working area $B_{р.п}$ (м) calculated as the sum of:

$$B_{р.п} = B_{тр.б} + \Delta B_{р.п\,max.}, \tag{29}$$

where $\Delta B_{р.п.max}$ – the highest value $\Delta B_{р.п.}$ from those defined by the above methods.

7.6. ANGLE OF INCLINATION OF THE WORKING SIDE

Angle of inclination of the working side $\alpha_{р.б.}$ (angle.) calculated by the formula

$$\alpha_{р.б.} = \operatorname{arcctg} \frac{B_{р.п.} + h_у \operatorname{ctg} \alpha_у}{h_у}, \tag{30}$$

the result is rounded to whole degree values.

7.7. THE RATE OF ANNUAL LOWERING OF THE BOTTOM OF THE QUARRY

The rate of annual lowering of the bottom of the quarry $h_г$ (м) is determined by the formula

$$h_г = \frac{V_у}{\operatorname{ctg} \alpha_{р.б.} + \operatorname{ctg} \beta_н} \tag{31}$$

where $\beta_н$ – angle of the recess direction.

Since, in accordance with the task, the direction of deepening should be chosen along the lying contact of the ore body with the host rocks, then $\beta_н = \alpha_{р.т}$.

The resulting value $h_г$ compared with the reference value (see appendix 7).

If the conditions of occurrence of the ore body do not match with the table value, the table values of hg are interpolated in accordance

with the task condition. For example, if, according to the assignment, the angle of the direction of the recess is 50 or 55 degrees, then for an excavator with a bucket capacity of 8 m3, the value of hg is chosen between 18 and 20 m (in integers): $h_г$ = 19 м.

If the value hg initially calculated by the formula exceeds the control value, then the control value h'g is taken and the parameters of the development system are recalculated.

7.8. ADJUSTING THE PARAMETERS OF THE DEVELOPMENT SYSTEM

Most often, the length of the active front of work per excavator is subject to change. To do this, you need to do the following:

Using the reference value of the rate of annual lowering of the bottom of the quarry $h'_г$, calculate a new value of the speed of movement of the working ledges $V'_у$ (м/год):

$$V'_у = h'_г \left(\operatorname{ctg}\alpha_{р.б} + \operatorname{ctg}\beta_н\right) \tag{32}$$

Next, check the value $\Delta B_{р.п}$ (м):

$$\Delta B_{рп} = \mu_т V'_у \tag{33}$$

If the value $\Delta B_{р.п}$ it turns out to be less than previously accepted, then the value of the angle of the working side will not need to be adjusted.

Then adjust the length of the active work front for one excavator $L'_ф$ (м):

$$L'_ф = \frac{Q}{h_у V'_у} \tag{34}$$

the value is rounded up to tens of meters up.

8. ANNUAL PRODUCTIVITY OF THE QUARRY FOR ORE, STRIPPING, ROCK MASS

8.1. ANNUAL PRODUCTIVITY OF THE ORE QUARRY

Annual productivity of the ore quarry $A_р$ thous. $м^3$ it is calculated based on the accepted value of the speed of deepening of the quarry, the average horizontal area of the ore body $S_р$ and previously selected values of loss and dilution coefficients

$$A_р = h_г S_р \frac{1-\eta}{1-\rho} \tag{35}$$

The value expressed in thousand m3 is converted to thousand tons and rounded up to hundreds of thousand tons.

Since the ore body is of the correct shape, the average horizontal area of the ore body is $S_р$ (thouse. $м^2$) determined by the formula

$$S_р = m_г L_р \,. \tag{36}$$

8.2. ANNUAL STRIPPING PERFORMANCE

Annual productivity of the overburden quarry $A_в$ (thous. $м^3$) determined by the formula

$$A_в = A_р n_1 \,. \tag{37}$$

Since the value $A_р$ expressed in thousands. $м^3$, that accepted in section 8.1. the value in thousand tons is translated into thousand. $м^3$. The resulting value $A_в$ rounded up to tens of thousands. $м^3$.

8.3. ANNUAL PRODUCTIVITY OF THE QUARRY BY ROCK MASS

Annual productivity of the quarry by rock mass $A_{г.м}$ (thous. $м^3$) determined by the formula

$$A_{г.м} = A_р + A_в, \tag{38}$$

rounded up to tens of thousands. $м^3$.

8.4. EXCAVATOR FLEET

Calculation of the excavator fleet $N_э$ (ед.) it is made according to the following formula:

$$N_э = \frac{A_{гм}}{Q} \tag{39}$$

the result is rounded up to an integer.

Similarly, the number of machines engaged in ore mining is calculated.

BRANCH OF THE FEDERAL STATE AUTONOMOUS EDUCATIONAL INSTITUTION OF HIGHER EDUCATION
"National Research Technological University "MISIS" in Almalyk

application

by discipline
"TECHNOLOGY AND COMPLEX MECHANIZATION OF OPEN-PIT MINING"

BRANCH OF THE FEDERAL STATE AUTONOMOUS EDUCATIONAL INSTITUTION OF HIGHER EDUCATION "National Research Technological University "MISIS" in Almalyk

DEPARTMENT OF "MINING"

Регистрирован	“УТВЕРЖДАЮ”
	Проректор по учебной работе
№________________________	_______________С.Худояров
«____»_______________2022 г.	«_____»____________2022 г.

WORKING CURRICULUM
In the course: Technology and complex mechanization of open-pit mining
For specialists

Area of expertise	**300 000**	Production and technical sphere
Field of education	**2140070**	Engineering
The direction of education	**21.05.04.**	Mining
Specialization	**СГД-16-9**	Open-pit mining

Term	**8 (4.2)**		**9(5.1)**		**Total**	
Weeks	18		18			
Type of occupation	УП	РП	УП	РП	УП	РП
Lecture	36	36	54	54	90	90
Practical exercises	54	54	36	36	90	90
Total aud.	90	90	90	90	180	180
Contact work	90	90	90	90	180	180
Independent work	54	54	63	63	117	117
Watch for control			27	27	27	27
Total	144	144	180	180	324	324

Алмалык-2022г.

The program was compiled (and):

PhD. Sh.V. Karimov, Associate Professor of the Department of "Mining" at NUST MISIS

Work program

Technology and complex mechanization of open-pit mining

Developed in accordance with the OS in:
Independently established educational standard of higher education Federal State Autonomous Educational Institution of Higher Education "National Research Technological University "MISIS" in the specialty 21.05.04 MINING (Order No. 602 of 02.12.2015 O.V.)

Compiled on the basis of the curriculum:
Direction 21.05.04 MINING, Open-pit mining, approved by the Scientific Council of the Federal State Budgetary Educational Institution of NUST MISIS on 21.05.2020, protocol № 10/зг

The working program was approved at the meeting of the department

Department of Mining

Protocol from 09.06.2021 г., №10

Head of the department Kakharov S.K.

COURSE PROGRAM

"TECHNOLOGY AND COMPLEX MECHANIZATION OF OPEN-PIT MINING"

Section one.

FUNDAMENTALS OF OPEN MINING OF MINERAL DEPOSITS.

Lecture 1

INTRODUCTION. GENERAL INFORMATION ABOUT THE TECHNOLOGY OF OPEN-PIT MINING.

The subject and objectives of the discipline and its relationship with related disciplines. The essence and elements of open-pit mining. Basic concepts. Terminology. The basic principles of complex mechanization of open development. The concept of a complex of mining equipment.

Teaching methods and techniques	«З-С-У» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Lecture 2

TYPES AND PERIODS OF MINING OPERATIONS. THE PROCEDURE FOR THE DEVELOPMENT OF OPEN-PIT MINING

Career field. The main parameters of the career. Extensive career fields. Elongated career fields. Rounded quarry fields. The scheme of career fields. Surface preparation. Drainage of the rock mass. Mining and capital works. Operational mining operations. Reconstruction of the quarry economy. Stripping works. Mining operations

Teaching methods and techniques	«Кластер» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Lecture 3

THE CONCEPT OF THE MODE AND STAGES OF MINING OPERATIONS. PREPARING A CAREER FIELD FOR DEVELOPMENT

The ratio of the volumes of stripping and mining operations. The average stripping ratio. Average operational stripping coefficient. The current stripping ratio. The boundary stripping coefficient. Planned stripping coefficient. Mining mode. The development stage. Uniform. Uneven. Natural obstacles. Artificial obstacles. Field drainage system. Surface dehumidification method. Underground drainage method. Combined dehumidification method.

Teaching methods and techniques	«Мозговой штурм» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Section two.

CARGO FLOWS AND THE SYSTEM OF OPENING WORKING HORIZONS.

Lecture 4

THE ORDER OF FORMATION OF CARGO FLOWS. TYPES OF CARGO FLOWS

Variety of deposit forms. Conditions of occurrence. Step-by-step schedule of mining operations. Summary table. Schedule for the formation of cargo flows. Cargo flow. Elementary cargo traffic. The excavation layer. Cargo flow from the ledge. Divergent cargo flow. Heterogeneous cargo traffic. Focused. Dispersed.

Teaching methods and techniques	«Кластер» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Lecture 5

OPENING UP THE WORKING HORIZONS OF A CAREER

The initial stages of mining development. Construction of special workings. The initial front. Trench. Horizon. Choosing the location of the trenches. The speed of moving the work front. Opening mine workings. External trenches. Internal trenches. Capital trenches. Split trenches. Stationary. Cross section. Methods of autopsy.

Teaching methods and techniques	«З-С-У» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Lecture 6

ROUTES OF OPENING WORKINGS

The route of the trench. The path plan. The longitudinal profile of the path. Tracing. The position of the highway. Service life. The basis for tracing. The theoretical length of the route. The actual length of the route. The shape of the route. Mixed track. The scheme of opening routes. The system of opening routes.

Teaching methods and techniques	«Кластер» презентация, раздаточные материалы.

Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Section three.

FIELD DEVELOPMENT SYSTEM

Lecture 7

DIVISION OF THE QUARRY FIELD INTO EXCAVATION LAYERS. HEIGHT AND STABILITY OF LEDGES.

A certain order. Excavation layers. Horizontal. Inclined. Cool. The number of ledges. The most important element of open development. The height of the ledge. Rational height of ledges. Analytical method. Development of horizontal and shallow deposits. Development of inclined and steeply falling deposits. Stability of slopes. Visors. Hang on. A group of geological factors. A group of hydrogeological factors. Group of technological factors.

Teaching methods and techniques	«Мозговой штурм» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Lecture 8

BASIC CONCEPTS OF THE MINING FRONT

The direction of development. Location. Along the long axis. Along the short axis. Concentrically. Structure. Homogeneous front. Heterogeneous. Difficult heterogeneous. The direction of movement of the rock mass. Loading of rock mass. The number of transport cargo exits. The position of the transport exit.

Teaching methods and techniques	«Кластер» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Lecture 9

DIRECTIONS OF MOVEMENT OF THE WORK FRONT. THE LENGTH AND SPEED OF THE ADVANCE OF THE WORK FRONT.

Panel along the work front. Panel block. Part of the panel. Working blocks. Excavation workings. Longitudinal. Transverse. Diagonal. Normal. Layer boundary. The length of the front. Length. The initial front of the ledge. Intensity of development. The speed of movement. The annual productivity of the excavator.

Teaching methods and techniques	«Мозговой штурм» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Lecture 10

WORKING AREA OF THE QUARRY. PREPARED, OPENED AND READY FOR EXCAVATION STOCKS

Development of several ledges. Working and non-working front. Safety and transport berms. Working area. Examples of work zones. Coverage of the sides of the quarry. Intensive mining operations. Non-intensive mining operations. Solid zone. Deepening work area. Prepared stocks. Opened. Ready. Current. Planned.

Teaching methods and techniques	«Кластер» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Lecture 11

CLASSIFICATION OF OPEN-PIT MINING SYSTEMS

Order. Sequence. The established volume and order. Dependent. Semi-dependent. Independent. Solid. Deepening. Deep-solid. Longitudinal. Transverse. Fan-shaped. Ring. One on-board. Double-sided. Central. Peripheral. Dispersed.

Teaching methods and techniques	«З-С-У» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Section four.

THEORY OF COMPLEX MECHANIZATION OF OPEN-PIT MINING

Lecture 12

GENERAL INFORMATION ABOUT THE COMPLEX MECHANIZATION OF OPEN-PIT MINING. PRINCIPLES OF COMPLEX MECHANIZATION

The essence of the main processes. Drilling. Exploding. Recess. Transportation. Warehousing. Initial and final warehouses. Cargo flow. Elementary cargo traffic. Complex of mining and transport equipment. Completeness of mechanization. High-quality. Quantitative. Complex mechanization. Automation. Development of rocks. A set of equipment. In-line technology. The main requirements for equipment complexes. The number of operating machines and mechanisms.

Teaching methods and techniques	«З-С-У» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Lecture 13

TECHNOLOGICAL CLASSIFICATION OF EQUIPMENT COMPLEXES

Technology class. Dredging. Excavators. Dredging-dump. Excavator-dump. Dredging-transport-dump. Excavation, transport and unloading.

Teaching methods and techniques	«Кластер» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Section five

TECHNOLOGY AND COMPLEX MECHANIZATION WITH CONTINUOUS DEVELOPMENT SYSTEMS.

Development system and methods of opening

Lecture 14

LONGITUDINAL, TRANSVERSE, FAN AND RING DEVELOPMENT SYSTEMS

Longitudinal single-sided. Longitudinal double-sided. Transverse single-sided. Central fan. Solid ring. Ring central. Complexes IN and THIS. WTO complexes and THIS. Transport communications. Parallel movement.

Teaching methods and techniques	«Мозговой штурм» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Lecture 15

OPENING OF WORKING HORIZONS WITH CONTINUOUS DEVELOPMENT SYSTEMS.

Trenchless autopsy. External separate trenches. External group trenches. External common trenches. Internal trenches. Trenches of mixed laying. Schemes of opening routes for horizontal and shallow deposits. Schemes of opening routes of internal laying. Routes of mixed laying. Schemes with parallel use of opening routes. Systems of opening routes.

Teaching methods and techniques	«Кластер» презентация, раздаточные материалы.

Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Lecture 16

THE ORDER OF EXCAVATION BY EXCAVATOR-DUMP TECHNOLOGICAL COMPLEXES.

Application of technological complexes. Mutual arrangement of equipment. Simple transshipment. Multiple transshipment. The coefficient of multiplicity of transshipment. Economically acceptable coefficient of overexcavation. Elements of the development system on the downhole side. Elements of the development system on the dump side. The width of the entry. Length of the work front.

Teaching methods and techniques	«Мозговой штурм» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Lecture 17

METHODS OF OPENING AND CARRYING OUT TRENCHES AT THE EXCAVATOR-DUMP TECHNOLOGICAL COMPLEX.

Opening of one flank capital trench. Opening with two flanking capital trenches. Opening of one central capital trench. Opening with two flanking capital trenches. Opening with three capital trenches. Transportless carrying out of trenches. The recess is a wide entry. Excavation in two or three passes. Layering of trenches.

Teaching methods and techniques	«З-С-У» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Lecture 18

TECHNOLOGICAL COMPLEXES WITH CANTILEVER DUMPERS.

Movement of rock by dumpers. Development system. Advantages of technological complexes. Continuity of production. Excavation schemes. Installation of a dumper on the roof of the deposit. Installation of a dumper on different horizons with an excavator. Installation of the dumper on the dump. Changing the standing position.

Teaching methods and techniques	«Кластер» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Lecture 19

TRANSPORT TECHNOLOGICAL COMPLEXES. TECHNOLOGICAL COMPLEXES WITH CONVEYOR MOVEMENT OF ROCK MASS WITH CONTINUOUS DEVELOPMENT SYSTEMS.

Horizontal and shallow deposit. Along the front. The use of complexes THIS and THAT. Rational travel distance. The conveyor. The length of the conveyor lines. The loader. Downhole conveyor. Transfer conveyor. Dump conveyor. Connecting conveyor. The main conveyor. Cantilever dumper.

Teaching methods and techniques	«Мозговой штурм» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Lecture 20

TECHNOLOGICAL COMPLEXES WHEN MOVING ROCK MASS BY MOTOR TRANSPORT WITH CONTINUOUS DEVELOPMENT SYSTEMS.

Autodumper. Load capacity. Development system. Transverse. Longitudinal. Transversely-longitudinally. Radial. The movement of individual sections. A powerful horizontal deposit. Autopsy diagram. Working horizons. Minimum width of the stripping panel. Layer-by-layer mining of a steep deposit. The advance of the work front. Full depth.

Teaching methods and techniques	«Кластер» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Lecture 21

COMBINED TECHNOLOGICAL COMPLEXES WITH CONTINUOUS DEVELOPMENT SYSTEMS.

Parallel work. A combination of complexes IN and IT. The combination of WTO complexes IS. Complexes with various types of transport. Complexes with hydro-mechanized and mechanical equipment. Scraper and bulldozer units. The main combined technological complexes.

Teaching methods and techniques	«Кластер» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Section six

TECHNOLOGY AND COMPLEX MECHANIZATION FOR IN-DEPTH DEVELOPMENT SYSTEMS.

Development system and autopsy methods

Lecture 22

IN-DEPTH DEVELOPMENT SYSTEM. CONDITIONS FOR THE USE OF IN-DEPTH DEVELOPMENT SYSTEMS.

The shape and structure of deposits. The prevailing types. The power of rocks. Simultaneous development. Increasing difficulty. The power of the covering rocks. Water cut. Temperature regime. The relief of the surface. The shape and size of quarries. Production conditions. Volumes of mining operations. Ensuring planned volumes.

Teaching methods and techniques	«Мозговой штурм» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Lecture 23

OPTIONS FOR THE DEVELOPMENT OF MINING OPERATIONS, DESIGNS AND PARAMETERS OF BERM IN DEEP-HOLE DEVELOPMENT SYSTEMS.

Development of an inclined and steep deposit. Parallel advance of the front. The initial position. The angle of incidence of the deposit. Longitudinal double-sided. Transverse single-sided. Muldoobraznaya deposit. Fan development of mining operations. Transport berms. Safety berms. Elements of connecting berms.

Teaching methods and techniques	«Кластер» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(4 часа)

Lecture 24

OPENING BY EXTERNAL CAPITAL TRENCHES.

The required throughput of the route of the opening workings. Opening of working horizons. The depth of the external trenches. Scope of work. Opening of several horizons. Deep laying. The height of the ledges. The output of deposits under sediments. Prostration. The size of the quarry. Thickness of the covering rocks.

Teaching methods and techniques	«З-С-У» презентация, раздаточные материалы.

Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,

(2 часа)

Lecture 25

SIMPLE, DEAD-END AND LOOP ROUTES.

Types of internal semi-tranches. Congress. The angle in the plan. A simple route. Stationary. Semi-stationary. Sliding. The movement of vehicles. Sections of the highway. The junction of the exits. The scheme of congresses. Dead-end roads. The length of dead-end platforms. Single-stage. Multi-stage. Track development. Loop routes. Semi-recess. Semi-bulk. The central corner.

Teaching methods and techniques	«Кластер» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(2 часа)

Lecture 26

FEATURES OF TECHNOLOGY AND COMPLEX MECHANIZATION IN THE COMBINATION OF AUTOMOBILE AND CONVEYOR TRANSPORT.

Overload of rock mass through crushing units. The use of screening units. Special conveyors. Complex with crushing. A complex with a screen. Lamellar. Wheel-belt. Lifting. Trunk roads. Crushing and processing. Dump. Secondary overload. Placement of PP. Design schemes. Mobile crushing plants. Links of mechanization.

Teaching methods and techniques	«Кластер» презентация, раздаточные материалы.
Learning tools	Видеопроектор, визуальные материалы информационное обеспечение,.

(4 часа)

Thematic plan of lectures on the discipline "Technology and complex mechanization of open-pit mining" 8TH SEMESTER

№	View	The name of the topic and its summary	Withdrawn
1	2	3	4
I	**Fundamentals of open mining of mineral deposits**		
1.1	Лекция	Introduction. General information about the technology of open-pit mining .	
1.2	Лекция	Types and periods of mining operations. Procedure for the development of open-pit mining .	
1.3	Лекция	The concept of the mode and stages of mining operations. Preparing a career field for development .	
II	**Cargo flows and the system of opening working horizons .**		
2.1	Лекция	The order of formation of cargo flows. Types of cargo flows .	
2.2	Лекция	Opening up the working horizons of a career .	
2.3	Лекция	Routes of opening workings .	
III	**Field development systems**		
3.1	Лекция	Division of the quarry field into excavation layers. Height and stability of the ledge .	
3.2	Лекция	Basic concepts of the mining front .	
3.3	Лекция	Directions of movement of the work front. The length and speed of the advance of the work front .	
3.4	Лекция	The working area of the quarry. Prepared, opened and ready-to-be-dredged stocks .	
3.5	Лекция	Classification of development systems	
IV	**Theory of complex mechanization of open-pit mining**		
4.1	Лекция	General information about the complex mechanization of open-pit mining. Principles of complex mechanization.	
4.2	Лекция	Technological classification of equipment complexes .	
V	**Technology and complex mechanization with continuous development systems .**		
5.1	Лекция	Longitudinal, transverse, fan and ring development systems .	
5.2	Лекция	Opening of working horizons with continuous development systems .	
1	2	3	
5.3	Лекция	The order of excavation by excavator-dump technological complexes .	
5.4	Лекция	Methods of opening and carrying out trenches at the excavator-dump technological complex	

5.6	Лекция	Technological complexes with cantilever dumpers .	
		Total :	

9- term

№	View	The name of the topic and its summary	Withdrawn
V	**Technology and complex mechanization with continuous development systems.**		
5.7	Лекция	Transport technological complexes. technological complexes with conveyor movement of rock mass with continuous development systems.	
5.8	Лекция	Technological complexes when moving rock mass by motor transport with continuous development systems.	
5.9	Лекция	Combined technological complexes with continuous development systems.	
VI	**Technology and complex mechanization for in-depth development systems.**		
6.1	Лекция	In-depth development system. conditions for the use of in-depth development systems	
6.2	Лекция	Options for the development of mining operations, designs and parameters of berm in deep development systems.	
6.3	Лекция	Opening by external capital trenches.	
6.4	Лекция	Simple, dead-end and loop routes. spiral routes.	
6.7	Лекция	Features of technology and complex mechanization in the combination of automobile and conveyor transport.	
		Всего:	

TOPICS OF PRACTICAL CLASSES

Practical work No.1. Safety precautions for OGR (2 hours). The purpose of the work: To familiarize students with the safety techniques at OGR. Lesson plan: Basic concepts and terms. .TB in the management of BVR. TB during excavation work. TB during transportation of G.P. TB during dumping.

Teaching methods and techniques	Brainstorming technology, presentation, handouts.
Learning tools	Video projector, visual materials information support,

Practical work No.2. Determination of the main parameters of the career (2 hours). The purpose of the work: To master the methodology for determining the main parameters of a career. Lesson plan: Basic concepts and terms. Familiarization with the initial data for the work. Calculation of career parameters

Teaching methods and techniques	Technology "Cube" , presentation, handouts.
Learning tools	Video projector, visual materials information support,.

Practical work No.3. The design and volume of capital trenches. (2 hours). The purpose of the work: To master the methodology for calculating the volume of capital tranches. The plan of the training session: Familiarization with the construction of trenches. Calculation of the parameters of the capital trench. Calculation of the volume of the capital trench

Teaching methods and techniques	Cluster technology, presentation, handouts.
Learning tools	Video projector, visual materials information support,.

Practical work No.4. Calculation of parameters and indicators of the split trench penetration. (2 hours). The purpose of the work: Mastering the methodology for calculating the parameters and indicators of the penetration of a split trench. Lesson plan: Basic concepts and terms. Determination of the main parameters of the split trench. Calculation of the penetration indicators of the split trench. Plotting the trench sinking.

Методы и техники обучения	Cluster technology, presentation, handouts.
Learning tools	Video projector, visual materials, information support,.

Practical work No.5. Determination of the rate of deepening of mining operations at the quarry. (2 hours). The purpose of the work: Mastering the skills of calculating the speed of deepening mining operations at the quarry. Lesson plan: Basic concepts and terms. Calculation method.

Методы и техники обучения	Brainstorming technology, presentation, handouts.
Learning tools	Video projector, visual materials information support,.

Practical work No. 6. Determination of the maximum height of the working area when processing a steeply falling deposit. (2 hours). The purpose of the work: Mastering the skills of calculating the maximum height of the working area when processing a steeply falling deposit. Lesson plan: Basic concepts and terms. Calculation method

Методы и техники обучения	Brainstorming technology, presentation, handouts.
Learning tools	Video projector, visual materials, information support,.

Practical work No. 7. Determination of the slope angle of the sides of the quarry. (2 hours). The purpose of the work: Mastering the methodology for calculating the slope angle of the sides of the quarry. Lesson plan: Basic concepts and terms. Determination of the slope angle of the working side.Determination of the slope of the non-working side

Методы и техники обучения	Boomerang technology, presentation, handouts.
Learning tools	Video projector, visual materials, information support,.

Practical work No. 8. Determination of parameters of technological excavation of rocks by draglines. (2 hours). The purpose of the work: To master the methodology for calculating the parameters of the technological scheme of dredging rocks by draglines. Lesson plan: Basic concepts and terms. Calculation method

Методы и техники обучения	Cluster technology, presentation, handouts.
Learning tools	Video projector, visual materials, information support,.

Practical work No. 9. Determination of dump formation parameters in road transport. (2 hours). The purpose of the work: To master the methodology for calculating the parameters of dump formation in road transport. The plan of the training session: The technology of dumping. Dump formation in road transport. Calculation method.

Методы и техники обучения	Brainstorming technology, presentation, handouts.
Learning tools	Video projector, visual materials, information support,.

Practical work No. 10. Calculation of dump formation parameters in railway transport. (4 hours). The purpose of the work: To master the methodology for calculating the parameters of dump formation in railway transport. Lesson plan: Basic concepts and terms. Calculation method.

Методы и техники обучения	Cluster technology, presentation, handouts.
Learning tools	Video projector, visual materials, information support,.

Practical work No. 11. Calculation of parameters of technological processes of mining operations at the quarry. (4 hours). The purpose of the lessons: To master the methodology for calculating the parameters of technological processes of mining operations at the quarry. Lesson plan: Basic technological processes. Calculation of the dredging and loading equipment. Calculation of BVR parameters. Calculation of transportation parameters.

Методы и техники обучения	Boomerang technology, presentation, handouts.
Learning tools	Video projector, visual materials, information support,.

Practical work No. 12. Determination of the parameters of the technology for the development of shallow deposits. (2 hours). The purpose of the work: To master the methodology for calculating the parameters of the technology for the development of shallow deposits. Lesson plan: Basic concepts and terms. Calculation method.

Методы и техники обучения	Brainstorming technology, presentation, handouts.
Learning tools	Video projector, visual materials, information support,.

Practical work No. 12. Determination of the parameters of the technology for the development of shallow deposits. (2 hours). The purpose of the work: To master the methodology for calculating the parameters of the technology for the development of shallow deposits. Lesson plan: Basic concepts and terms. Calculation method.

Методы и техники обучения	Brainstorming technology, presentation, handouts.
Learning tools	Video projector, visual materials, information support,.

Practical work No. 14. Determination of parameters of a transportless development system. (2 hours). The purpose of the work: To master the methodology for calculating the parameters of a transport-free development system. Lesson plan: Basic concepts and terms. Calculation method.

Методы и техники обучения	Cluster technology, presentation, handouts.
Learning tools	Video projector, visual materials, information support,.

Practical work No. 15. Determination of the parameters of the transport system of the development (2 hours). The purpose of the work: To master the methodology for calculating the parameters of the transport system development. Lesson plan: Basic concepts and terms. Calculation method.

Методы и техники обучения	Cluster technology, presentation, handouts.
Learning tools	Video projector, visual materials, information support,.

Practical work No. 16. Determination of parameters of the transport dump system of development. (2 hours). The purpose of the work: To master the methodology for calculating the parameters of the transport dump system of development. Lesson plan: Basic concepts and terms. Calculation method.

Методы и техники обучения	Brainstorming technology, presentation, handouts.
Learning tools	Video projector, visual materials, information support,.

Practical work No. 17. Determination of loss and dilution coefficients. (2 hours). The purpose of the work: To master the methodology for calculating the coefficient of loss and dilution. Lesson plan: Basic concepts and terms. Calculation of losses . Calculation of dilution.

Методы и техники обучения	Cluster technology, presentation, handouts.
Learning tools	Video projector, visual materials, information support,.

Recommended topics for independent studies

№	**1- independent work**	**2- independent work**
	Fundamentals of open mining of mineral deposits	
1	Abstract and presentation on the topic: "Principles of open mining of mineral deposits".	Compilation of a crossword puzzle, tests and glossary on the topic: “Principles of open mining of mineral deposits».
2	Summary and presentation on the topic: “Quality of mining enterprises' products”.	Compilation of crosswords, tests and glossary on the topic:
3	Abstract and presentation on the topic: “Excavation and loading works”.	Making a crossword puzzle, tests and glossary on the topic: “Excavation and loading works".
4	Abstract and presentation on the topic: “Preparation of rocks for excavation”.	Making a crossword puzzle, tests and glossary on the topic: “Preparation of rocks for excavation”.
5	Abstract and presentation on the topic: “Technological fundamentals of drilling operations".	Compilation of a crossword puzzle, tests and glossary on the topic: “Technological fundamentals of drilling operations”.
6	Abstract and presentation on the topic: “Technological foundations of blasting”.	Compilation of a crossword puzzle, tests and glossary on the topic: “Technological foundations of blasting”.
7	Abstract and presentation on the topic: “Elements of a quarry and basic mining concepts".	Compilation of a crossword puzzle, tests and glossary on the topic: Elements of a quarry and basic mining concepts".

8	Abstract and presentation on the topic: “Mechanization of production processes".	Compilation of a crossword puzzle, tests and glossary on the topic: “Mechanization of production processes".
9	Summary and presentation on the topic: “General information about the mineral deposit and the technology of their development".	Compilation of a crossword puzzle, tests and glossary on the topic: “General information about the mineral deposit and the technology of their development”.
10	Abstract and presentation on the topic: “Elements of a quarry and basic mining concepts".	Compilation of a crossword puzzle, tests and glossary on the topic: Elements of a quarry and basic mining concepts".
	Cargo flows and the system of opening working horizons	
1	Abstract and presentation on the topic: “The opening ceremony of working horizons.”	Making a crossword puzzle, tests and glossary on the topic: “The opening ceremony of working horizons.”
2	Summary and presentation on the topic: “Volumes of capital tranches and half-tranches”.	Compilation of a crossword puzzle, tests and glossary on the topic: “Volumes of capital tranches and semi-tranches”.
3	Abstract and presentation on the topic: “Basic concepts of the mining front".	Compilation of a crossword puzzle, tests and glossary on the topic: “Basic concepts of the mining front”.
4	Abstract and presentation on the topic: “Development systems and methods of autopsy”.	Compilation of crosswords, tests and glossary on the topic:
5	Abstract and presentation on the topic: “Opening of working horizons in the development of in-depth systems".	Compilation of a crossword puzzle, tests and glossary on the topic: “Opening working horizons with in-depth development systems".
6	Abstract and presentation on the topic: “Parameters of tranches and methods of their implementation".	Compilation of a crossword puzzle, tests and glossary on the topic: “Parameters of tranches and methods of their implementation”.
7	Abstract and presentation on the topic: “Parameters of tranches and methods of their implementation".	Compilation of a crossword puzzle, tests and glossary on the topic: “Parameters of tranches and methods of their implementation”.
8	Abstract and presentation on the topic: “Technological flows in quarries".	Compilation of a crossword puzzle, tests and glossary on the topic: “Technological flows in quarries”.
	Field development systems	

1	Abstract and presentation on the topic: “Theory of development systems".	Making a crossword puzzle, tests and glossary on the topic: “Theory of development systems".
2	Abstract and presentation on the topic: “Transportation of quarry cargo".	Compilation of crosswords, tests and glossary on the topic:
3	Abstract and presentation on the topic: “The main parameters of a career".	Compilation of a crossword puzzle, tests and glossary on the topic: “The main parameters of a career".
4	Abstract and presentation on the topic: “Mutual connection of production processes in a career”.	Compilation of a crossword puzzle, tests and glossary on the topic: “Mutual connection of production processes in a career".
5	Abstract and presentation on the topic: “Transport development systems".	Compilation of a crossword puzzle, tests and glossary on the topic: “Transport development systems".
6	Abstract and presentation on the topic: “Combined development systems".	Making a crossword puzzle, tests and glossary on the topic: “Combined development systems".
6	Abstract and presentation on the topic: “Excavation and loading works".	Making a crossword puzzle, tests and glossary on the topic: “Dredging and loading operations".
7	Abstract and presentation on the topic: “Classification of development systems".	Compilation of a crossword puzzle, tests and glossary on the topic: “Classification of development systems".
	Theory of complex mechanization of open-pit mining	
1	Abstract and presentation on the topic: “Theory of complex mechanization of open-pit mining".	Making a crossword puzzle, tests and glossary on the topic: “Theory of complex mechanization of open-pit mining".
2	Abstract and presentation on the topic: “Technological complexes with a combination of means of transport".	Compilation of a crossword puzzle, tests and glossary on the topic: “Technological complexes with a combination of means of transport".
3	Abstract and presentation on the topic: “Cyclic flow technology in deep quarries".	Compilation of a crossword puzzle, tests and glossary on the topic: “Cyclic flow technology in deep quarries".
4	Abstract and presentation on the topic: “Types of slaughtering and entering. Methods of rock excavation".	Compilation of a crossword puzzle, tests and glossary on the topic: “Types of slaughters and visits. Methods of rock excavation".

5	Abstract and presentation on the topic: “General observations on the excavability of rocks and the productivity of excavating machines".	Compilation of a crossword puzzle, tests and glossary on the topic: “General observations on the excavability of rocks and the performance of excavation machines.”
6	Abstract and presentation on the topic: “Dumping of waste rocks and storage of substandard minerals".	Compilation of a crossword puzzle, tests and glossary on the topic: “Waste rock formation and storage of substandard minerals.”
	Technology and complex mechanization with continuous development systems	
1	Abstract and presentation on the topic: “Technology and complex mechanization with continuous development systems".	Compilation of a crossword puzzle, tests and glossary on the topic: “Technology and complex mechanization with continuous development systems".
2	Abstract and presentation on the topic: “Classification of development systems".	Compilation of a crossword puzzle, tests and glossary on the topic: “Classification of development systems".
3	Abstract and presentation on the topic: “Technological complexes when moving rock mass by motor transport".	Compilation of a crossword puzzle, tests and glossary on the topic: “Technological complexes when moving rock mass by motor transport".
4	Abstract and presentation on the topic: “Technological complexes of mining of construction rocks".	Compilation of a crossword puzzle, tests and glossary on the topic: “Technological complexes for mining construction rocks".
5	Abstract and presentation on the topic: “Technological complexes with conveyor movement of rock mass".	Compilation of a crossword puzzle, tests and glossary on the topic: “Development systems and methods of opening”.
6	Abstract and presentation on the topic: “Technological complexes with conveyor with the movement of rock by rail to internal dumps".	Compilation of a crossword puzzle, tests and glossary on the topic: “Technological complexes with conveyor with the movement of rock by rail to internal dumps.”
	Technology and complex mechanization for in-depth development systems	
1	Abstract and presentation on the topic: “Technology and complex mechanization in the development of in-depth systems”.	Compilation of a crossword puzzle, tests and glossary on the topic: “Technology and complex mechanization in advanced development systems".
2	Abstract and presentation on the topic: “Technological complexes in railway transport (with in-depth development systems)".	Compilation of a crossword puzzle, tests and glossary on the topic: “Technological complexes in railway transport (with in-depth development systems)".

3	Abstract and presentation on the topic: “Technological complexes in road transport (with in-depth development systems)".	Compilation of a crossword puzzle, tests and glossary on the topic: “Technological complexes in road transport (with in-depth development systems)".
4	Abstract and presentation on the topic: “Technological complexes for conveyor transport (with deep-development systems)".	Compilation of a crossword puzzle, tests and glossary on the topic: “Technological complexes for conveyor transport (with in-depth development systems)".

Recommended literature for independent work.

1. Ржевский В.В. Открытые горные работы: Технология и комплексная механизация: Учебник. Изд. 5-е – М.; Книжный дом «ЛИБРОКОМ», 2010. -552 с.
2. Хохряков В.С. Открытая разработка месторождений полезных ископаемых: Учебник. Изд. 5-е, перераб. и доп. – М.; Недра, 1991. – 336 с.
3. Кучерский Н.И. Современные технологии при освоении коренных месторождений золота. – М.; Издательский дом «Руда и Металлы», 2007. -696 с.
4. Виницкий К.К. Параметры систем открытой разработки. – М.; Из. – Недра, 1966. -263с
5. Анистратов Ю.А., Анистратов К.Ю., Щадов М.И. Справочник по открытым горным работам: М.: НТЦ «Горное дело», 2010.700с.,453 илл.
6. Н.Я.Репин., Л.Н. Репин. Выемочно-погрузочные работы. М.: Изд МГГУ, 2010.267с.
7. Н.Я. Репин. Подготовка горных пород к выемке. Ч. 1: Учебное пособие. – М.: «Мир горной книги», Изд. МГГУ,2009. -188 с.:ил.

EVALUATION CRITERIA

students' knowledge based on the subject rating system

"Technology and complex mechanization of open-pit mining"

The discipline is considered mastered if the following conditions are met:

- the current lecture control has positive ratings ("satisfactory", "good", "excellent"),

- all practical work has been completed and protected,

the results of lecture control and protection of practical work:

from 56 and less than 70% "satisfactory"

from 71 and less than 85% - "good"

from 86 to 100% - "excellent",

- completed and protected for a positive assessment ("satisfactory", "good", "excellent") course project.

The assignment for course design is issued by the teacher - head of course design at the beginning of 9

semester individually for each student in accordance with the characteristics of the object. The student receives an individual

task to substantiate and calculate the parameters of production processes in the open-pit mining

of mineral deposits.

The course project consists of a graphic part and an explanatory note. The graphic part is performed on 1 sheet

of A-1 format, which shows the accepted version of the decision on the topic of the course project. Explanatory note

it must contain the necessary calculations and justification of the decisions taken. The volume of the explanatory note is 15-20 pages

of printed text.

The explanatory note includes the following mandatory parts:

Title page

Assignment for course design

Table of contents

Introduction

1. Justification of the boundaries and volumes of mining operations

2. Justification of the development system and parameters of its elements

3. Justification of the opening scheme and parameters of the opening workings

4. Calculation of parameters of technological schemes for conducting stripping and mining operations

Conclusion

List of literature

Applications (if available)

The graphic part includes:

Sheet 1: Plan and section of the current mining situation

The course project is evaluated according to the following criteria:

- design of the course project,
- the structure of the course project,
- content of the course project.

The rating "excellent" is set:

- according to the criterion "Design of the course project":

The explanatory note of the course work is designed in accordance with the requirements (in terms of field sizes, font

of the main text, paragraph margins, line spacing, rubrication, numbering, writing formulas,

design of tables, illustrations, list of references). There are references to the sources used in the text of the workl.

- according to the criterion "Content of the course project":

The explanatory note includes all the mandatory parts

- according to the criterion "Content of the course project":

All sections of the course project are completed in full, contain sound engineering solutions and correct

calculations.

The rating "Good" is set:

- according to the criterion "Design of the course project":

in the design of the course project, small deviations from the requirements are allowed (for example, incorrect ones are installed

field sizes, paragraph margins, etc.), provided that all other requirements are met.

- according to the criterion "Structure of the course project":

The explanatory note does not contain one of the mandatory sections.

- according to the criterion "Content of the course project":

One of the sections of the course project contains minor errors.

The rating "Satisfactory" is set:

- according to the criterion "Design of the course project":

The design of the explanatory note partially meets the established requirements

Handouts on the discipline:
"TECHNOLOGY AND COMPLEX MECHANIZATION OF OPEN-PIT MINING"

Career elements and parameters

1 – the developed space; 2 – the non–working side; 3 – the working side; 4 – the final contour of the quarry; 5 – berm; α – the angle of the slope of the ledge; φ – the angle of the slope of the working side; uv - the angle of the slope of the non-working side from the hanging side of the deposit; ul – the angle of the slope of the non-working side from the lying side deposits; mg – horizontal capacity of the deposit; P – width of the working platform; h – height of the ledge; NC – depth of the quarry; BB – width of the quarry along the upper contour; δT - width of the transport berm; δP - width of the safety berm

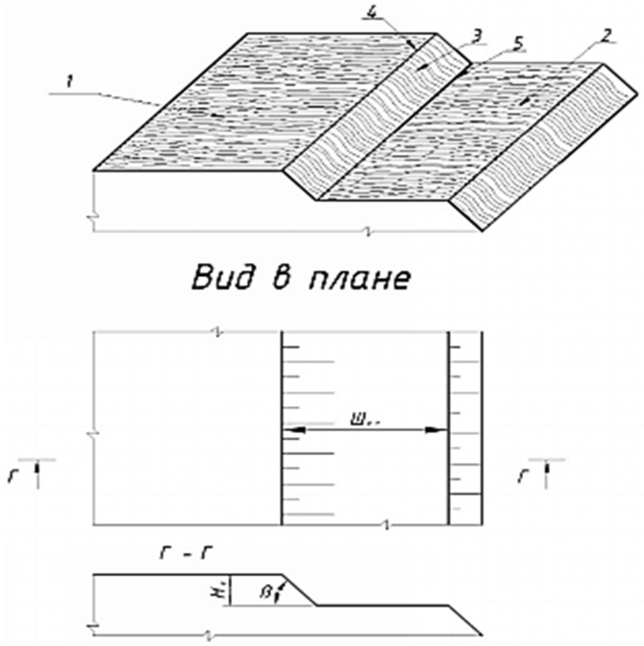


Fig.1. Overburden ledge

Symbols:

1 – the upper platform of the ledge;

2 – the lower platform of the ledge;

3 – slope of the ledge;

4 – upper edge of the ledge;

5 – lower edge of the ledge;

– height of the ledge;

β – slope angle of the ledge;

– width of the working area.

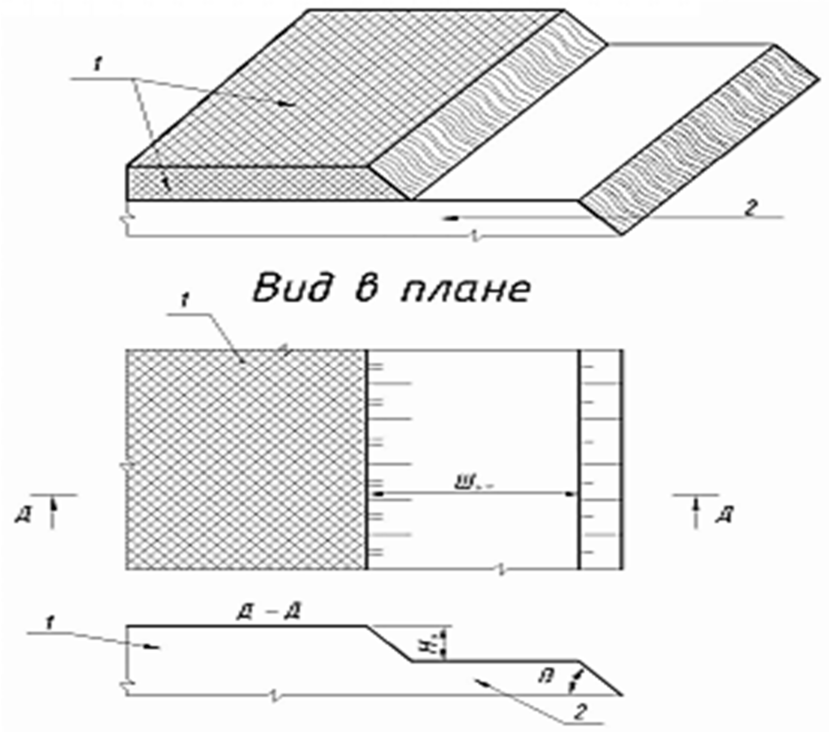


Fig.2 Mining ledge

1 – mineral resources;

2 – overburden rocks;

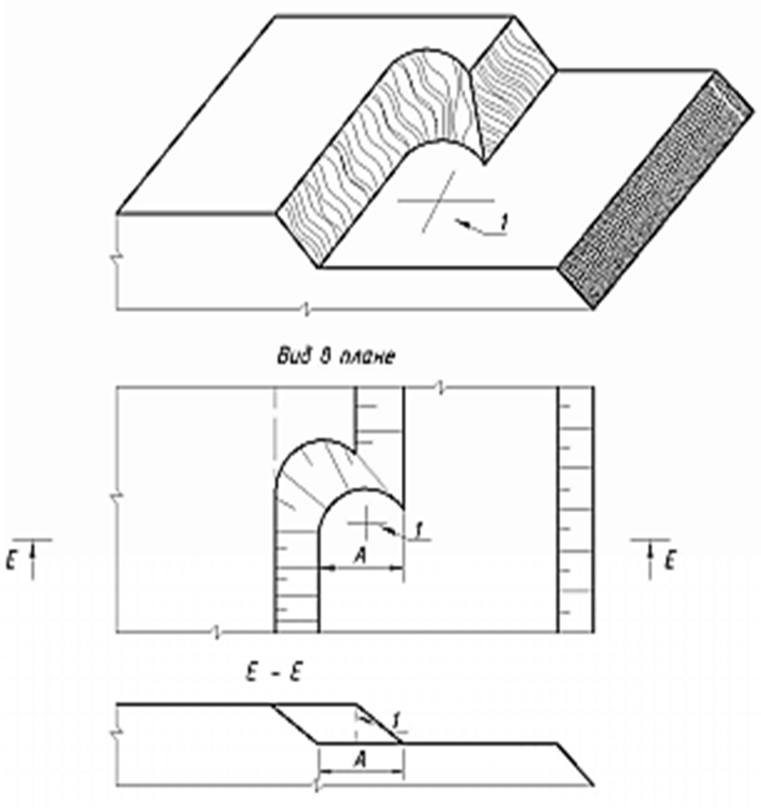


Fig.3. Excavator face in soft rocks:

1 – excavator axle;

A – the width of the excavator entry.

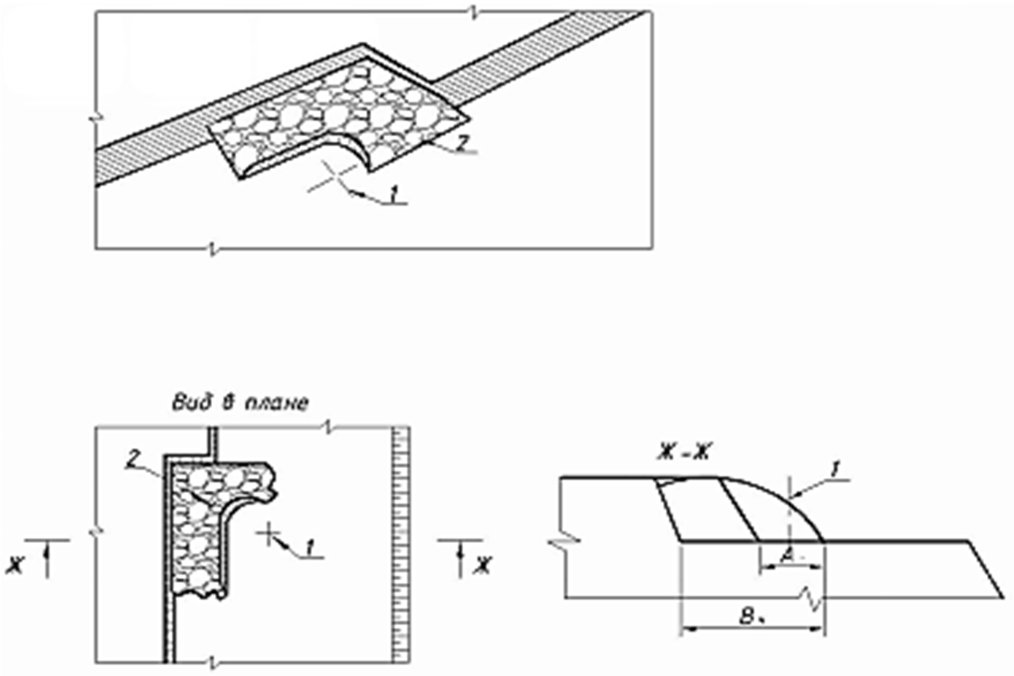


Fig.4. Excavator face in overburden destroyed (exploded) rocks.

1 – excavator axle;

2 – collapse of exploded rocks;

– width of the blasted rocks;

– width of the first entry of the excavator.

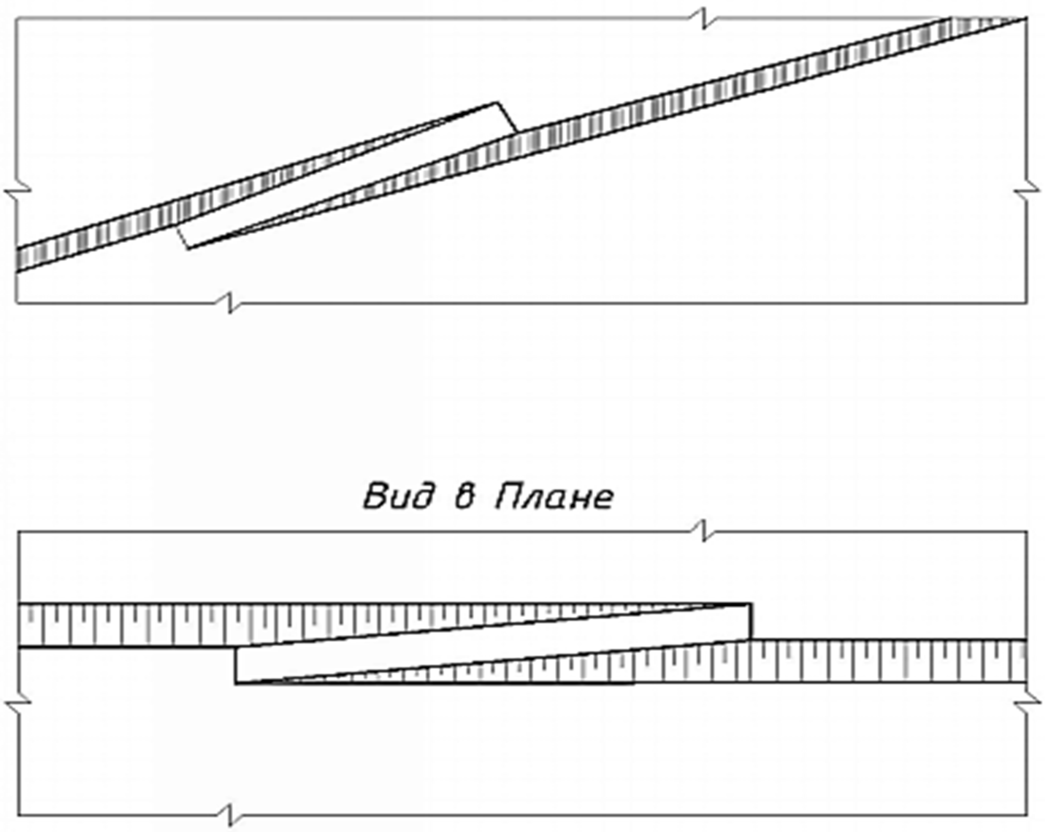


Fig.5. Inner trench - exit.

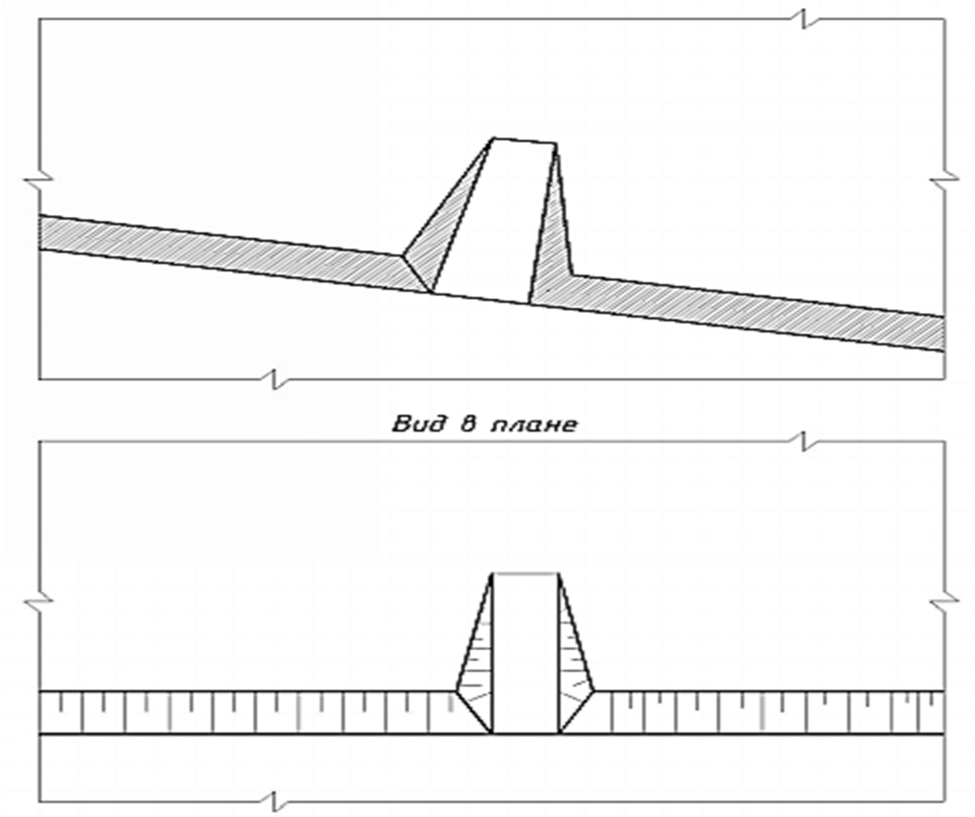


Fig. 6. The outer trench.

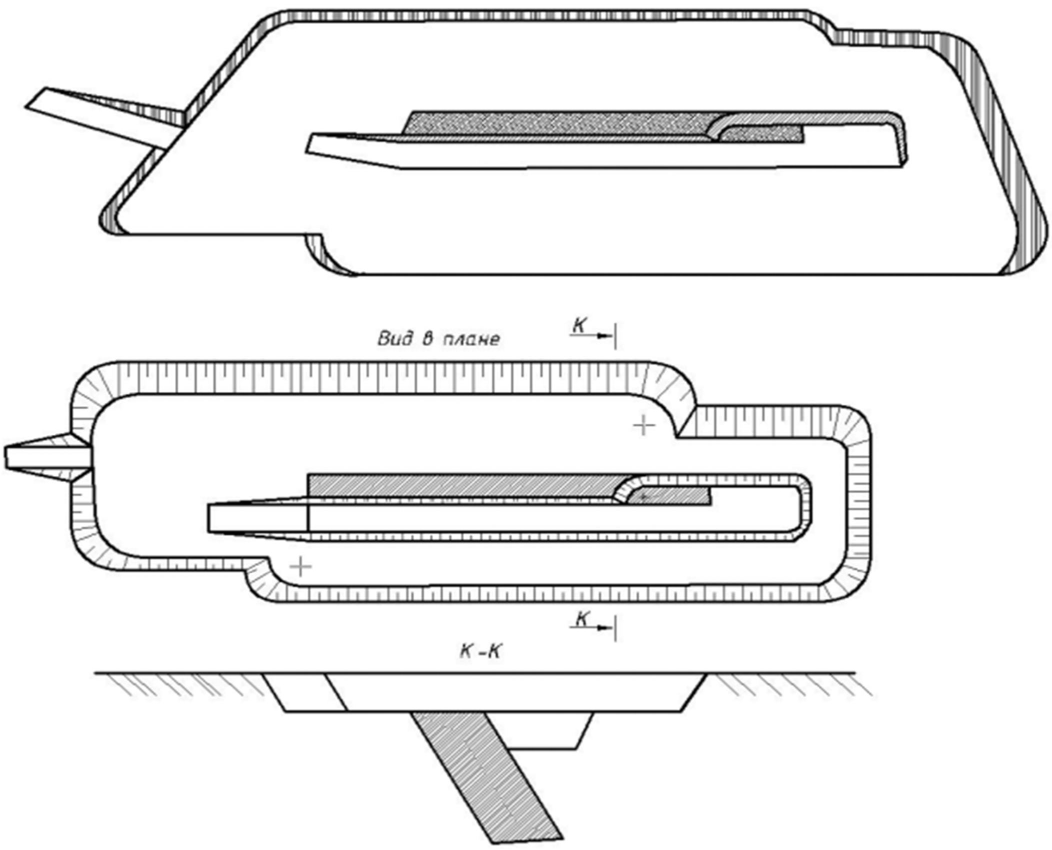


Fig.7. Quarry.

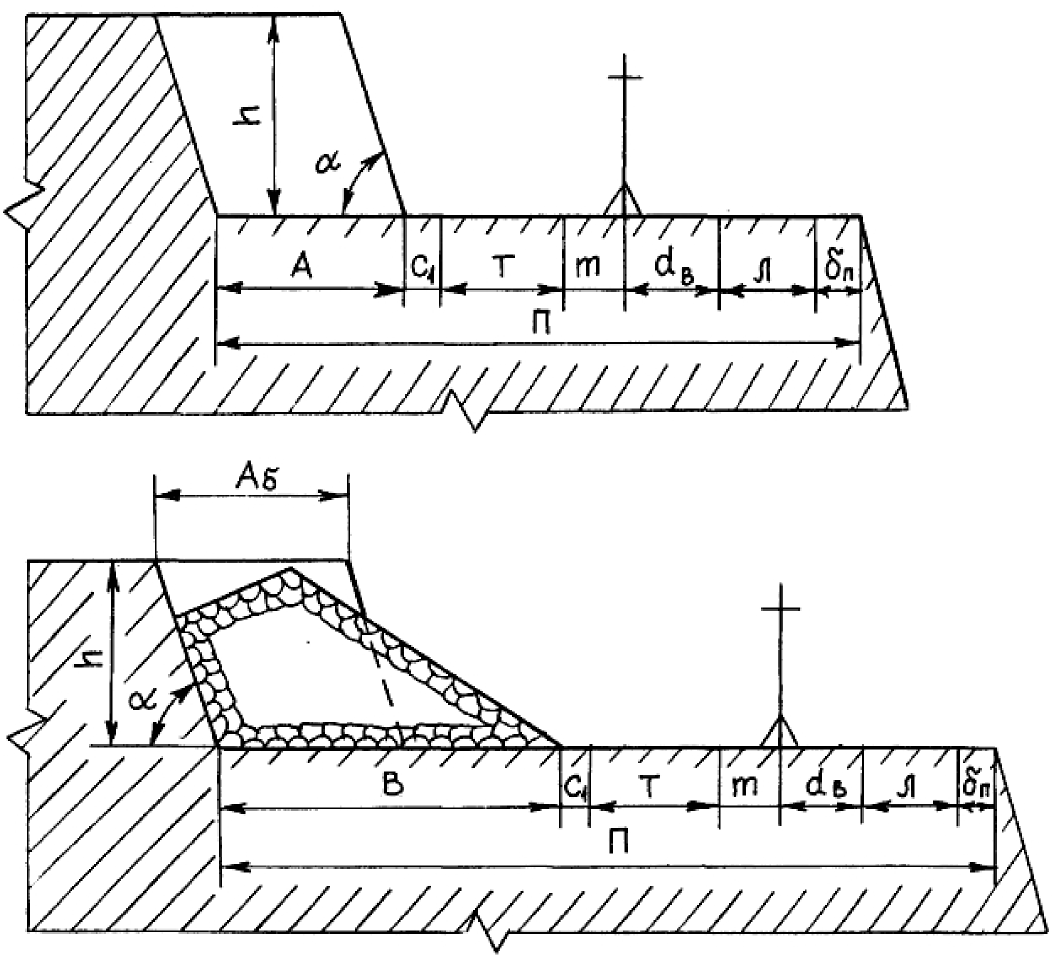
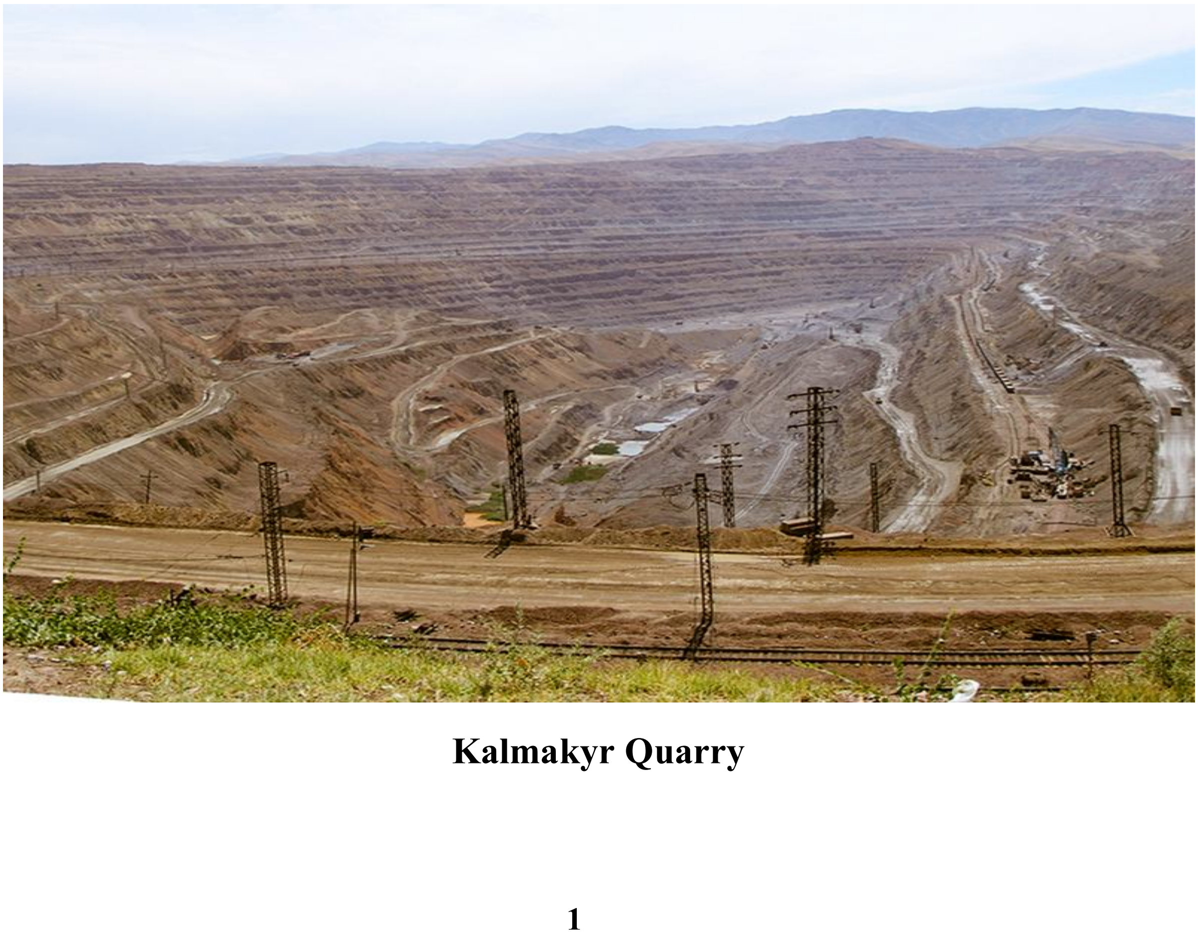


Fig. 8. Diagram for calculating the width of the working platform in soft (a) and rocky (b) rocks



Kalmakyr Quarry

View of the Muruntau quarry

Drilling operations (Kalmakyr)

Blasting operations (Muruntau).

Blasting operations (Kalmakyr).

Excavation and loading works (Muruntau)

Excavation and loading works (Kalmakyr)

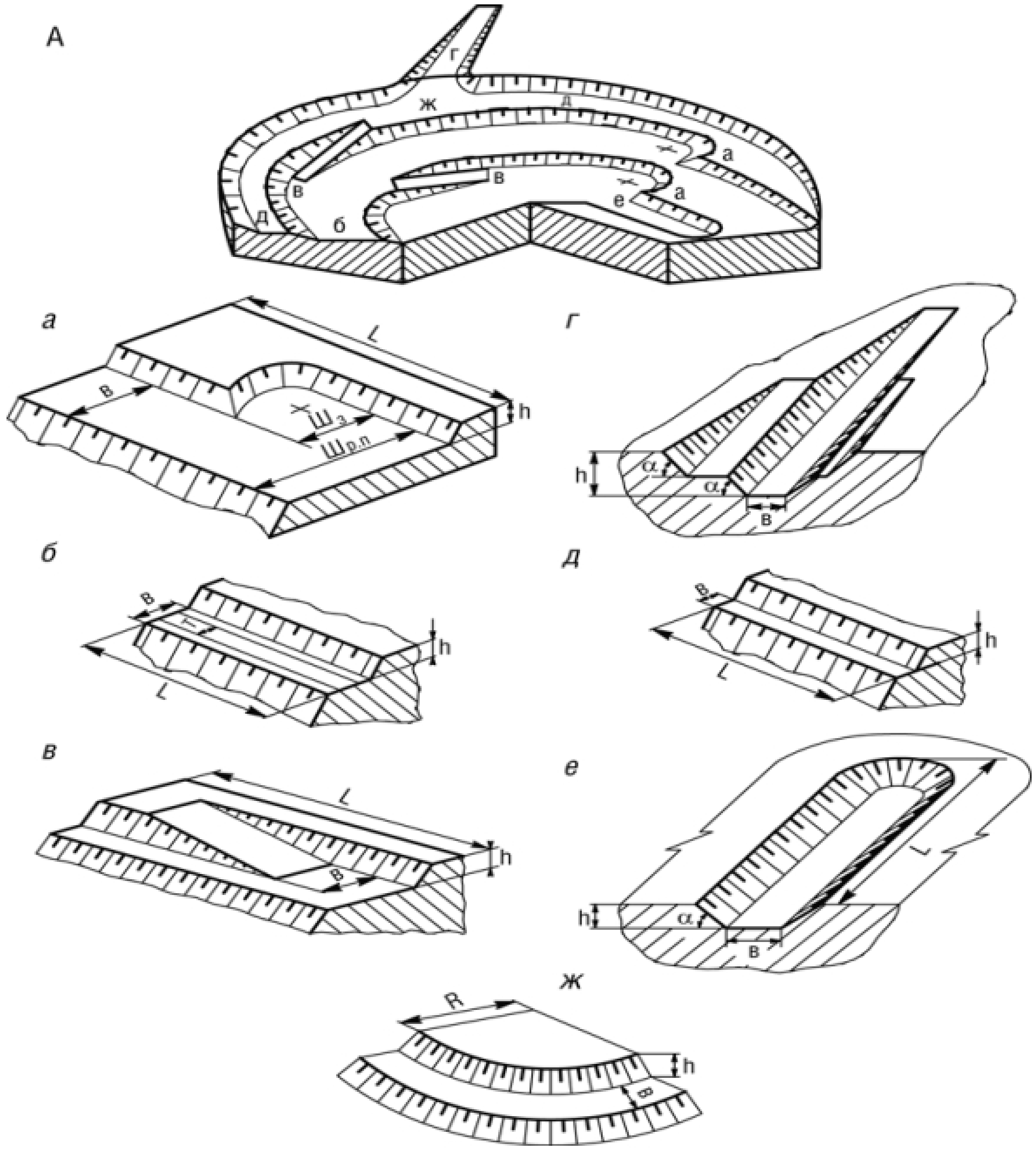
Transportation of rock mass (Kalmakyr)

Transportation of rock mass (Muruntau)

KNK-270 complex (Muruntau steep-slope conveyor)

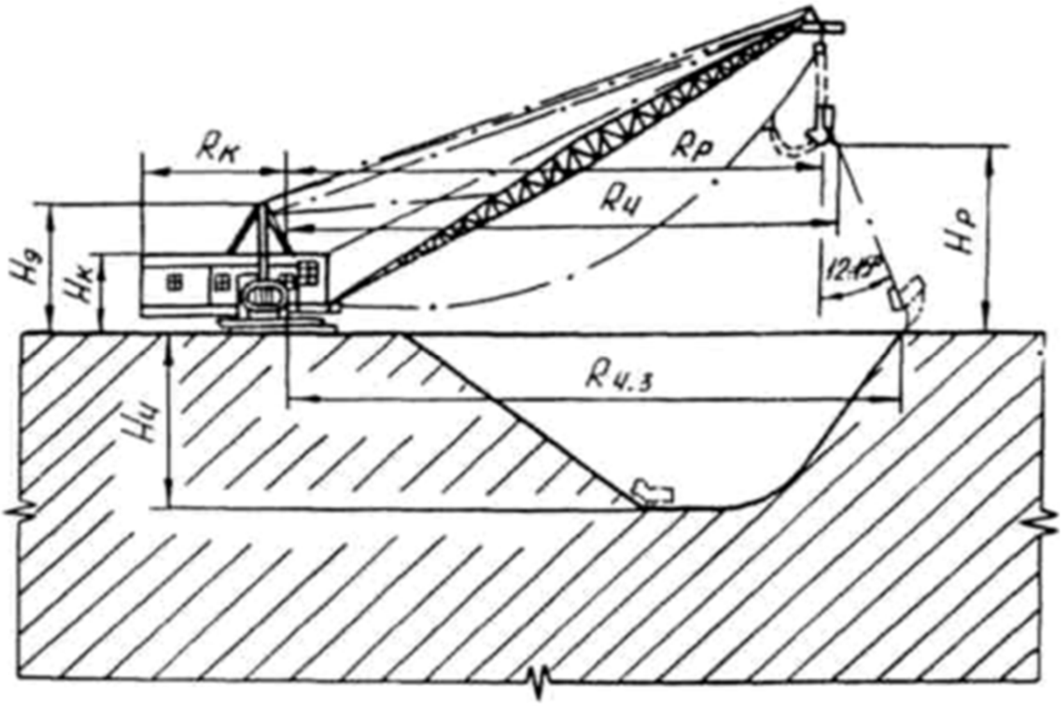
Dumps of the Muruntau quarry

Dumps of the Kalmakyr quarry



Elements of the development systems:

a – excavation (excavation) block, b – horizontal transport berm, c – inclined transport berm, d – inclined trench, d – safety and periodic cleaning berm, e – split trench, w – rounding



Dragline operation scheme

The design scheme of the chain excavator

1 – buckets; 2 – bucket frame; 3 – chain drive sprocket;

4 – receiving chute; 5 – lower frame; 6 – dump console;

7 – central column; 8 – boom suspension of the dump console;

9 – rotary wheel for rock selection; 10 – bucket frame suspension

Layout schemes of parametric series chain excavators

a - full-turn on the rail course of the upper and lower scooping with unloading through a hopper located in the lower frame; b - through a cantilever non-rotating conveyor; c, d - on a crawler with full-turn balanced cantilever conveyors; d - cantilever dump-forming arrows; e - with an unloading bridge

General view of the rotary excavator

Mobile removal and loading equipment of continuous operation

SECURITY QUESTIONS.

1. What is called the technology of field development?

2. What is called a quarry?

3. What layers are used to develop an array of rocks?

4. What form does the side surface of the quarry take when conducting open-pit mining?

5. What are the main production processes of open-pit mining.

6. What are the deposits classified by shape?

7. What determines the shape of the deposits?

8. According to the angle of incidence, deposits are distinguished…

9. Name the main types of open-source development.

10. What is the essence of surface view development?

11. What is the surface preparation?

12. What works are related to mining and capital?

13. What works are operational mining operations divided into?

14. Name the final stage of open development.

15. What is the main purpose of open-pit mining?

16. Name the main types of stripping coefficients.

17. What determines the boundary stripping coefficient?

18. What do you mean by the mining regime?

19. What is called the development stage?

20. What are the ways of draining the quarry.

21. What determines the need for the formation of cargo flows?

22. For which fields do they build schedules for the formation of cargo flows?

23. What is called an elementary cargo flow?

24. Which cargo flows are called heterogeneous?

25. Cargo flows in the quarry can be…

26. How is the opening of working horizons carried out?

27. What determines the rate of advance of the front of work?

28. What are the signs of separation of capital trenches.

29. Which method of opening is called trenchless?

30. Name the methods of opening.

31. What is called a trench route?

32. What is path tracing?

33. Which route is called simple?

34. Which route is called difficult?

35. In what cases are mixed routes used?

36. Why is the quarry field divided into excavation layers?

37. Which layers are called horizontal?

38. Which layers are called inclined?

39. What is the angle of inclination of the steep layers?

40. How many ledges work out deposits of very low capacity?

41. What is the height of the ledge considered rational?

42. What should be the height of the ledge in the development of soft rocks?

43. What should be the height of the ledge when developing rocky and semi-horizontal rocks?

44. On what factors does the stability of the slopes of the ledge depend?

45. By what signs does the front of mining operations differ?

46. What can be the front of work on the location?

47. What can be the front of work on the structure?

48. What can be the front of work in the direction of movement of the rock mass?

49. What can be the front of work on the position of the transport exit?

50. What is the working area of the quarry?

51. What mining operations are distinguished on each ledge of the working area?

52. What stocks are called prepared?

53. What stocks are called opened?

54. What do you mean by ready-to-excavate reserves of rock mass?

55. What do you mean by the completeness of mechanization?

56. What is the essence of the technological processes of mining?

57. What factors explain the resistance of rocks for each subsequent process?

58. What are the distinctive features of rocks of each class, divided by relative difficulty of development?

59. What are the main requirements for equipment complexes?

60. What requirements should equipment complexes meet?

61. What should be provided by the complete set of means of mechanization of auxiliary works and processes?

62. What classes are divided into equipment complexes used and implemented in quarries?

63. What equipment complexes are called excavation?

64. Which sets of equipment are called excavator?

65. What machines and mechanisms include dredging-dump complexes?

66. What is the difference between the dredging-transport-unloading complexes of equipment from other complexes?

67. Describe the longitudinal development system.

68. In what cases are transverse single-board development systems used?

69. What equipment complexes are used for longitudinal development systems?

70. In what cases are fan development systems used?

71. In what cases are ring development systems used?

72. In what cases are schemes with parallel use of opening routes used?

73. In the development of which deposits are used transport technological complexes?

74. What development systems are used to reduce the distance of intra-barrier transportation?

75. In what cases is the length of conveyor lines minimal?

76. Describe the schemes of transportation of overburden by conveyors.

77. How is the grouping of cargo flows of the same rocks carried out during internal dumping?

78. When developing which deposits, a technological complex of layer-by-layer mining is used?

79. How is the minimum width of the overburden panel determined on the lower overburden horizon?

80. What technological complex is used for a transverse single-board system for the development of elongated steep deposits?

GENERAL QUESTIONS

The subject and objectives of the discipline

(Technology and complex mechanization of open-pit mining. The essence and elements of open-pit mining. Basic concepts. Terminology. The basic principles of complex mechanization of open development. The concept of a complex of mining equipment).

Types of career fields

(Career field. The final depth. The dimensions of the bottom of the quarry. The angles of the slopes of the sides of the quarry. The dimensions of the quarry along the strike and across the strike of the deposit on the surface. The total volume of rock mass in the contours of the quarry field. Extensive career fields. Elongated career fields. Rounded quarry fields. Scheme of career fields).

Stages of mining operations

(The ratio of the volumes of stripping and mining operations. Stripping coefficients. Mining mode. The development stage. Uniform. Uneven. Natural obstacles. Artificial obstacles. Field drainage system. Surface dehumidification method. Underground drainage method. Combined dehumidification method).

The essence of open-pit mining

(The essence and elements of open-pit mining. Basic concepts. Terminology. The basic principles of complex mechanization of open development. The concept of a complex of mining equipment).

Types and periods of mining operations

(Surface preparation. Drainage of the rock mass. Mining and capital works. Operational mining operations. Reconstruction of the quarry economy. Stripping works. Mining operations. The attenuation period. The choice of the type of open-pit mining. Mining development scheme).

Types of stripping coefficients

(Average stripping ratio. Planned stripping coefficient. The current stripping ratio. The boundary stripping coefficient. Average operational stripping coefficient).

Types of fields being developed

(Open-pit mining facilities. The deposit. The shape of the deposit. The relief of the surface. Depth of occurrence. Angle of incidence. Power. The quality of the mineral).

Methods of drainage of deposits

(waterlogging. Field drainage system. The method of draining the quarry. Surface, underground and combined drainage methods).

Formation of cargo flows

(Variety of deposit forms. Conditions of occurrence. Step-by-step schedule of mining operations. Summary table. Schedule for the formation of cargo flows. Cargo flow. Elementary cargo traffic. The excavation layer. Cargo flow from the ledge. Divergent cargo flow. Heterogeneous cargo traffic. Focused. Dispersed).

Opening up the working horizons of a career

(Initial stages of mining development. Construction of special workings. The initial front. Trench. Horizon. Choosing the location of the trenches. The speed of moving the work front. Opening mine workings. External trenches. Internal trenches. Capital trenches. Split trenches. Stationary. Cross section. Methods of autopsy).

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Opening mine workings

(Opening mine workings. External trenches. Internal trenches. Capital trenches. Split trenches. Stationary. Cross section. Methods of autopsy).

Initial stages of mining development

(The rate of movement of the work front. Front of work. The direction of mining development. Choosing the location of the trenches. Mining operations on the horizon. Schemes of the initial period of mining development on the horizon. Opening of working horizons).

Types of open-pit mining

(Type of rocks. Surface-type developments. Deep-view developments. Developments of the upland view. Developments of the upland-deep view. Underwater view developments).

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The procedure for the development of open-pit mining

(Stripping. Mining operations. The attenuation period. The choice of the type of open-pit mining. Mining development scheme).

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List of questions for the final control of the discipline
"Technology and complex mechanization of open-pit mining"

General information about the technology of open-pit mining.

The subject and objectives of the discipline and its relationship with related disciplines. The essence and elements of open-pit mining. Basic concepts. Terminology. The basic principles of complex mechanization of open development. The concept of a complex of mining equipment.

Types and periods of mining operations. Procedure for the development of open-pit mining

Career field. The basic parameters of the quarry. Extensive career fields. Elongated career fields. Rounded quarry fields. The scheme of career fields. Surface preparation. Drainage of the rock mass. Mining and capital works. Operational mining operations. Reconstruction of the quarry economy. Stripping works. Mining operations

The concept of the mode and stages of mining operations. Preparing a career field for development

The ratio of the volumes of stripping and mining operations. The average stripping ratio. Average operational stripping coefficient. The current stripping ratio. The boundary stripping coefficient. Planned stripping coefficient. Mining mode. The development stage. Uniform. Uneven. Natural obstacles. Artificial obstacles. Field drainage system. Surface dehumidification method. Underground drainage method. Combined dehumidification method.

The order of formation of cargo flows. Types of cargo flows

Variety of deposit forms. Conditions of occurrence. Step-by-step schedule of mining operations. Summary table. Schedule for the formation of cargo flows. Cargo flow. Elementary cargo traffic. The excavation layer. Cargo flow from the ledge. Divergent cargo flow. Heterogeneous cargo traffic. Focused. Dispersed.

Opening up the working horizons of a career

The initial stages of mining development. Construction of special workings. The initial front. Trench. Horizon. Choosing the location of the trenches. The speed of moving the work front. Opening mine workings. External trenches. Internal trenches. Capital trenches. Split trenches. Stationary. Cross section. Methods of autopsy.

Routes of opening workings

The route of the trench. The path plan. The longitudinal profile of the path. Tracing. The position of the highway. Service life. The basis for tracing. The theoretical length of the route. The actual length of the route. The shape of the route. Mixed track. The scheme of opening routes. The system of opening routes.

Division of the quarry field into excavation layers. Height and stability of ledges.

A certain order. Excavation layers. Horizontal. Inclined. Cool. The number of ledges. The most important element of open development. The height of the ledge. Rational height of ledges. Analytical method. Development of horizontal and shallow deposits. Development of inclined and steeply falling deposits. Stability of slopes. Visors. Hang on. A group of geological factors. A group of hydrogeological factors. A group of technological factors.

Basic concepts of the mining front

The direction of development. Location. Along the long axis. Along the short axis. Concentrically. Structure. Homogeneous front. Heterogeneous. Composite. The direction of movement of the rock mass. Loading of rock mass. The number of transport cargo exits. The position of the transport exit.

Directions of movement of the work front. The length and speed of the advance of the work front.

Panel along the work front. Panel block. Part of the panel. Working blocks. Excavation workings. Longitudinal. Transverse. Diagonal. Normal. Layer boundary. The length of the front. Length. The initial front of the ledge. Intensity of development. The speed of movement. The annual productivity of the excavator.

The working area of the quarry. Prepared, opened and ready-to-be-dredged stocks

Development of several ledges. Working and non-working front. Safety and transport berms. Working area. Examples of work zones. Coverage of the sides of the quarry. Intensive mining operations. Non-intensive mining operations. Solid zone. Deepening work area. Prepared stocks. Opened. Ready. Current. Planned.

Classification of open-pit mining systems

Order. Sequence. The established volume and order. Dependent. Semi-dependent. Independent. Solid. Deepening. Deep-solid. Longitudinal. Transverse. Fan-shaped. Ring. Single-board. Double-sided. Central. Peripheral. Dispersed.

General information about the complex mechanization of open-pit mining. Principles of complex mechanization

The essence of the main processes. Drilling. Exploding. Recess. Transportation. Warehousing. Initial and final warehouses. Cargo flow. Elementary cargo traffic. Complex of mining and transport equipment. Completeness of mechanization. High-quality. Quantitative. Complex mechanization. Automation. Development of rocks. A set of equipment. In-line technology. The main requirements for equipment complexes. The number of operating machines and mechanisms.

Technological classification of equipment complexes

Technology class. Dredging. Excavators. Dredging-dump. Excavator-dump. Dredging-transport-dump. Dredging, transport and unloading.

Longitudinal, transverse, fan and ring development systems

Longitudinal single-sided. Longitudinal double-sided. Transverse single-sided. Central fan. Solid ring. Ring central. VO and EO complexes. WTO complexes and THIS. Transport communications. Parallel movement.

Opening of working horizons with continuous development systems.

Trenchless autopsy. External separate trenches. External group trenches. External common trenches. Internal trenches. Trenches of mixed laying. Schemes of opening routes for horizontal and shallow deposits. Schemes of opening routes of internal laying. Routes of mixed laying. Schemes with parallel use of opening routes. Systems of opening routes.

The order of excavation by excavator-dump technological complexes.

Application of technological complexes. Mutual arrangement of equipment. Simple transshipment. Multiple transshipment. The coefficient of multiplicity of transshipment. Economically acceptable coefficient of overexcavation. Elements of the development system on the downhole side. Elements of the development system on the dump side. The width of the entry. The length of the work front.

Methods of opening and carrying out trenches at the excavator-dump technological complex.

Opening of one flank capital trench. Opening with two flanking capital trenches. Opening of one central capital trench. Opening with two flanking capital trenches. Opening with three capital trenches. Transportless carrying out of trenches. The recess is a wide entry. Recess with two or three strokes. Layering of trenches.

Technological complexes with cantilever dumpers.

Movement of rock by dumpers. Development system. Advantages of technological complexes. Continuity of production. Excavation schemes. Installation of a dumper on the roof of the deposit. Installation of a dumper on different horizons with an excavator. Installation of the dumper on the dump. Changing the standing position.

Transport technological complexes. Technological complexes with conveyor movement of rock mass with continuous development systems.

Horizontal and shallow deposit. Along the front. The use of complexes is the WTO. Rational travel distance. The conveyor. The length of the conveyor lines. The loader. Downhole conveyor. Transfer conveyor. Dump conveyor. Connecting conveyor. The main conveyor. Cantilever dumper.

Technological complexes when moving rock mass by motor transport with continuous development systems.

Autodumper. Load capacity. Development system. Transverse. Longitudinal. Transversely-longitudinally. Radial. The movement of individual sections. A powerful horizontal deposit. Autopsy diagram. Working horizons. Minimum width of the stripping panel. Layer-by-layer mining of a steep deposit. The advance of the work front. Full depth.

Combined technological complexes with continuous development systems.

Parallel work. A combination of HE and EO complexes. The combination of WTO complexes IS. Complexes with various types of transport. Complexes with hydro-mechanized and mechanical equipment. Scraper and bulldozer units. The main combined technological complexes.

In-depth development system. Conditions for the use of in-depth development systems.

The shape and structure of deposits. The prevailing types. The power of rocks. Simultaneous development. Increasing difficulty. The power of the covering rocks. Water cut. Temperature regime. The relief of the surface. The shape and size of quarries. Production conditions. Volumes of mining operations. Provision of planned volumes.

Options for the development of mining operations, designs and parameters of berm in deep-hole development systems.

Development of an inclined and steep deposit. Parallel advance of the front. The initial position. The angle of incidence of the deposit. Longitudinal double-sided. Transverse single-sided. Muldoobraznaya deposit. Fan development of mining operations. Transport berms. Safety berms. Elements of connecting berms.

Opening by external capital trenches.

The required throughput of the route of the opening workings. Opening of working horizons. The depth of the external trenches. Scope of work. Opening of several horizons. Deep laying. The height of the ledges. The output of deposits under sediments. Prostration. The size of the quarry. The thickness of the covering rocks.

Simple, dead-end and loop routes.

Types of internal semi-tranches. Congress. The angle in the plan. A simple route. Stationary. Semi-stationary. Sliding. The movement of vehicles. Sections of the highway. The junction of the exits. The scheme of congresses. Dead-end roads. The length of dead-end platforms. Single-stage. Multi-stage. Track development. Loop routes. Semi-recess. Semi-bulk. The central corner.

Features of technology and complex mechanization in the combination of automobile and conveyor transport.

Overload of rock mass through crushing units. The use of screening units. Special conveyors. Complex with crushing. Complete with a rumble. Lamellar. Wheel-belt. Lifting. Trunk roads. Crushing and processing. Dump. Secondary overload. Placement of PP. Design schemes. Mobile crushing plants. Links of mechanization.

The procedure for the final control

The final control in this discipline is carried out at the end of the 6th semester in accordance with the approved schedule.

The final control work is accepted in writing. Each option consists of three questions and reference words. The questions should meet the requirements of the material passed.

At the beginning of the academic year, the list of questions and tickets are updated by the teacher and approved at a meeting of the department.

After the final control, the teacher is obliged to check and evaluate the student's work within two days, and also inform them about the score received.

Main references

1. Ржевский В.В. Открытые горные работы: Технология и комплексная механизация: Учебник. Изд. 5-е – М.; Книжный дом «ЛИБРОКОМ», 2010. -552 с.
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Additional references

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6. Н.Я.Репин., Л.Н. Репин. Выемочно-погрузочные работы. М.: Изд МГГУ, 2010.267с.
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