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Lecture course on the subject:

Enrichment of useful fossils

for bachelors and specialists

Direction of preparation: 03/22/02-“Metallurgy”
05/21/04-“Mining”

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This course contains 18 lectures on the subject “Beneficiation of mineral resources”.

Basic information is given about the material composition of minerals, methods and processes of their enrichment. The fundamentals of the theory of separation of minerals according to their physical properties in various processing machines and apparatus are outlined. The design and operating principle of the main enrichment equipment and methods for monitoring technological processes are described. Technologies for processing and enrichment of the main types of mineral ores are considered.

For university students studying in the specialties “Mining Processing”, “Technology of Mineral Raw Materials”, the areas of training for certified specialists are “Mining”, “Metallurgy”.

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PREFACE

Modern technology for processing minerals is based on the use of mineral enrichment methods that take advantage of the differences in physical, magnetic, electrical, physicochemical and other properties of the separated minerals. The increasing complexity of the composition of processed minerals, the involvement of new technological types in the exploitation of deposits, while striving for integrated development of subsoil, necessitates the integration of all stages of mining production into a single technology for obtaining mineral products.

Primary processing (enrichment) of minerals is the final link in the overall technology and ensures the production of marketable products that meet the raw material standards for the chemical-metallurgical, fuel-energy and other industries. The rational use of natural resources and environmental protection increasingly depend on the level of technology and technology for primary processing of minerals.

Mineral processing technology has a number of industrial methods for separating minerals according to their physical and physicochemical properties. A targeted change in these properties makes it possible to artificially increase the contrast of the natural properties of minerals, which significantly expands the range of minerals involved in production.

Knowledge of the basic technology of primary processing and enrichment of minerals is necessary for all mining specialists.

The purpose of the course of lectures is to give students of mining and metallurgical specialties the necessary knowledge about the technological properties of minerals, the fundamentals of the theory of enrichment processes and designs, and the most common equipment for their implementation. To familiarize with the modern technology of complex processing and beneficiation of the main types of minerals, technical and economic indicators, processing and beneficiation of various types of mineral raw materials, to create the necessary basis for creative solutions, future specialists of the issues of optimal combination of technological processes of extraction and beneficiation, increasing the complexity of the use of raw materials, technological , economic and environmental indicators of processing and beneficiation of minerals.

Section 1. GENERAL CONCEPTS AND DEFINITIONS

Lecture No. 1. MINERAL RESOURCES AND THEIR CHARACTERISTICS

Introduction. Classification of minerals. Material composition of minerals. Textural and structural characteristics. Physical properties

*Key terms:*minerals, ore, minerals, deposit, enrichment, disseminated, ore condition, material composition, useful component, balance and off-balance ore, useful component, impurities, mineralogical composition, density, magnetic properties

1.1Introduction

*Mining*includes the extraction of minerals from the subsoil and their primary processing (enrichment).

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Primary processing (enrichment) of minerals is the final link in the overall technology and ensures the production of marketable products that meet the raw material standards for the chemical-metallurgical, fuel-energy and other industries. The rational use of natural resources and environmental protection increasingly depend on the level of technology and technology for primary processing of minerals.

Currently, 100% of mined non-ferrous and rare metal ores, more than 90% of ferrous metal ores, all coking and most thermal coal, all mining chemical raw materials and a significant part of raw materials for the production of building materials are subject to enrichment.

Modern enrichment factories are powerful, highly mechanized and automated industrial enterprises with complex technological processes and circuits, full of a variety of machines and devices. The production capacity of individual enterprises reaches 35-40*million tons of ore per year.*

Mineral processing technology has a number of industrial methods for separating minerals according to their physical and physicochemical properties. A targeted change in these properties makes it possible to artificially increase the contrast of the natural properties of minerals, which significantly expands the range of minerals involved in production.

Knowledge of the basic technology of primary processing and enrichment of minerals is necessary for all mining specialists.

1.2 Classification of minerals

Minerals- these are natural mineral formations in the earth's crust of inorganic and organic origin, chemical composition and physical properties, which allow their use in the field of material production. Currently, more than 200 types of mineral raw materials are used in industry and agriculture.

Based on their physical properties, minerals are distinguished between solid (ore, nonmetallic, coal, peat), liquid (oil, mineralized water) and gaseous (natural flammable and inert gases).

*Deposit*mineral deposit is called an accumulation mineral substance in the earth's crust, which is qualitatively and quantitatively suitable for use in the national economy. Deposits, the development of which is economically feasible given the existing level of technology, are called *industrial*; deposits, the development of which under the same conditions is unprofitable, are called*non-industrial.*As technology for the extraction and enrichment of mineral resources develops, non-industrial deposits can become industrial.

The objects of activity of mining and processing enterprises are solid minerals of the group*A: A1*- Coals;*AZ*- Metal minerals;*A4*- Natural materials and stones;*A5*- Non-metallic minerals.

Group A1 - Coals- these are solid flammable substances of organic origin.

Fossil coals have different physical and chemical properties, which is due to differences in the original plant material, the depth of chemical transformations and intramolecular rearrangements of plant residues.

Depending on the stage of metamorphism, they distinguish: brown coal, hard coal and anthracite, which differ in chemical composition, physical properties and quality indicators.

*Brown coals*They are divided into two groups: lignites and brown coals themselves. *Lignites*consist of wood residues and have a fibrous structure. *Actually brown coals*do not have clearly expressed plant leftovers The color of these coals varies - from dark brown to black. Carbon content - 68-80%, hygroscopic moisture - 25-30%, volatile matter yield - more than 45%, density - 800-1250*kg/m3*. Brown coal, when exposed to air, crumbles into fines.

*Coal*has a black color, combustion heat31-37*kJ/kg*, density 1250-1500*kg/m3*; contain 3-4% hygroscopic moisture, 80-92% carbon, 11-45% volatile substances.

*Anthracite*has a black surface with a glassy sheen, sharp edges when broken, calorific value 35-38*kJ/kg*, contains volatile substances up to 6%.

Coal is not a homogeneous substance, but consists of several petrographic varieties:

duren- matte, hard, non-layered coal, found in in the form of powerful packs;

Claren- shiny coal with a pronounced banded texture, occurs in the form of thick packs or even entire layers;

vitren- a lustrous charcoal similar to but different from claren small size of inclusions, absence of inclusions of other varieties and higher density;

fusin- matte coal with a fibrous structure, in appearance resembles crushed charcoal and occurs as small lenses on bedding planes.

Varieties of coal have the following ash content: vitrain and claren - up to 2%; duren - 6-12% and fusain - 15-25%. Claren and Vitrain coke well, Durene weakly, and Fusain does not coke. The most durable variety is duren, and the most fragile is fusin.

Knowledge of the petrographic composition of coals is necessary to determine the optimal limits of crushing, the rational limit of their enrichment and methods of technological processing.

Coals consist of organic (combustible) mass and non-combustible components (mineral impurities and moisture).

The composition of the organic mass includes the following chemical elements: carbon (C), hydrogen (H), oxygen (O), nitrogen (N), sulfur (S), phosphorus (P). The most valuable element in coals is carbon, the content of which increases with increasing stage of metamorphism.

Mineral impurities include: clay shale ($Al_2 O_3 2Si0_2$-$2H_20$), sandy shale (SiO_2), pyrite (FeS_2), sulfates ($CaSO_4$), carbonates ($MgCO_4$, $FeCO_3$, etc.).

Mineral impurities transferred to coal from plant organisms are called*related*, and impurities that entered during the period of accumulation of plant residues -*alluvial*. Mineral impurities that get into coal during its mining are called*free*. During enrichment, only free mineral impurities can be removed.

*Industrial classification*coals involves the division of coals into various brands and groups depending on their physical and chemical properties and the possibility of use for technological or energy purposes.

The coals of each pool are divided into brands and groups, and coals of the same brands and groups of different pools have unequal limits of classification parameters. Therefore, coals from different basins, characterized by the same classification parameters, when used technologically, can produce a product with different physical and mechanical properties.

All coals are conventionally divided into two technological groups: coking and thermal.

Group AZ - Metallic minerals- black ores, non-ferrous, rare and precious metals.

*Ore*is an aggregate minerals, fromwhom It is technologically possible and economically feasible to extract the metal or its compounds. These are, for example, the ores of iron, manganese, lead, zinc, molybdenum, tungsten, etc. Based on the quality of mineral raw materials, rich (high-grade), ordinary (average in quality) and poor (low-grade) ores are distinguished.

Minerals are called natural chemical connections, formed as a result of natural chemical reactions, more or less homogeneous chemically and physically. Depending on their chemical composition, minerals are grouped into classes, of which the most important are:*native*elements;*sulfides*(compounds of metals with sulfur); *oxides*(compounds of metals and some other elements with oxygen); *silicates*(compounds of metals with silicon and oxygen) and *aluminosilicates* (silicates containing aluminum).

Distinguish*bedrock and placer*mineral deposits. In primary deposits, ore occurs in the general rock mass in the place of its original formation. In this case, useful minerals are found in the rock mass in the form of disseminated grains (inclusions) of one size or another, often in close germination with waste rock minerals.

*Placer deposits*are formed as a result of destruction indigenous ores under the influence of water, air oxygen, temperature and other natural factors.

Placers concentrate minerals that are resistant to natural factors. These minerals are usually found in the form of isolated grains, but are often cemented by clay or other material.

According to their material composition they are distinguished*ores of ferrous, non-ferrous, rare, noble and radioactive metals.*

Ores are also divided into*monometallic*, containing only one metal, and complex,*polymetallic*, containing several metals (for example, ores containing copper and zinc, lead and zinc, molybdenum and tungsten).

*By inclusion size*grains of useful minerals distinguish ores very*large inclusions (more than20 mm), coarsely disseminated (more than 2 mm), finely disseminated (2-0.2 mm) and finely disseminated (less than 0.2 mm).*

*According to physical properties*ores are divided according to density, humidity, etc. densities are distinguished*heavy ores - higher density3500 kg/m3, average - 2500-3500 kg/m3, light - below 2500 kg/m3.*

*By humidity*ores are distinguished between highly wet, wet and dry ores.

Depending on the physical properties and chemical composition of ores are divided into*difficult and easy to enrich*.

The industry requirements for ore raw materials are determined by GOSTs and technical specifications, according to which ore raw materials are divided into grades depending on the content of useful components, harmful impurities and the nature of the ore aggregate. There are restrictions on moisture content and particle size distribution.

*Industrial conditions for ore*is a system of indicators in which The minimum acceptable metal content in ore and metal reserves in a given deposit are accepted.

Minimum industrial content counts such the content of a valuable component, the cost of which, when extracted from the subsoil and enriched, ensures the return of all costs for these processes. For example, the minimum industrial copper content in ore must be at least 0.5% (depending on the type of ore and processing method), lead - 1%, zinc - 1.5%, tungsten - 0.15%.

The standards also determine the cut-off content of metal in the ore, which is necessary to delineate the industrial balance reserves of the deposit.

Onboard content- this is the lowest metal content in the edge samples The upper limit of the cut-off grade is the minimum industrial grade, the lower limit of the cut-off grade should slightly exceed the metal content in the tailings of the processing plants.

Group A4 - Natural building materials and stones and Group A5 - Non-metallic minerals- used for chemical industry (sulfur, potassium salts, barite, etc.), agriculture (apatite, phosphorite, etc.), abrasive (diamond, corundum, pumice, etc.) and jewelry industry and precision instrument industry (diamond, ruby, emerald, etc. .). They serve as fillers for paper, rubber, food and other industries (talc, kaolin, chalk, clays, etc.), insulating materials (asbestos, mica, etc.), natural fire- and acid-resistant materials (magnesite, acid-resistant clays, amphiboles etc.), stone building and road materials (limestone, quartzite, gravel, sand), raw materials for cementitious building ceramic and refractory materials (marl, gypsum, kaolin, feldspar, quartz, graphite, etc.).

The quantitative assessment of minerals is expressed by their reserves (balance sheet and off-balance sheet).

*Balance sheet*are mineral reserves, use which are technically possible and economically feasible.

Off-balance sheet- mineral reserves, the use of which at this level of technology it is not economically feasible (low power, deep burial, low content of valuable components, etc.).

1.3 Material composition of minerals

The material composition of minerals is a set of data on the content of useful components and impurities, mineral forms of manifestation and the nature of fusion of grains of the most important elements, their crystal chemical and physical properties.

1.3.1. Chemical composition

The chemical composition of minerals characterizes the content of main and accompanying useful components, as well as useful and harmful impurities. Data on the chemical composition of a mineral are the basis for determining the technology for its processing and enrichment.

Useful component- an integral part of a mineral resource, the extraction of which for the purpose of industrial use is technologically possible and economically feasible. There are main and accompanying useful components.

Main useful component- found in minerals industrial concentrations, determining their main value, purpose and name. For example, iron in iron ores. If there are two or more main useful components, the mineral is characterized as complex. For example, copper-zinc, copper-lead-zinc ores.

Related useful components- components of useful minerals, the extraction of which is economically feasible only together with the main useful components. For example, gold and silver in polymetallic sulfide ores, non-ferrous metals in iron ores, rare elements in coals, etc.

*Useful impurities*name the valuable elements contained in minerals that can be isolated and used together with the main useful component, improving its quality. For example, chromium and tungsten in iron ores, etc.

*Harmful impurities*name the elements present in the useful mineral together with the main useful component and deteriorating its quality. For example, sulfur and phosphorus in iron ores, sulfur in coals.

The chemical composition of minerals is determined by spectral, chemical, assay, nuclear physical, activation and other types of analysis.

1.3.2. Mineralogical composition

The mineralogical composition characterizes the mineral forms of manifestation of the elements that make up the mineral.

In accordance with the mineral forms of manifestation of the main valuable components*non-ferrous metal ores*distinguished as*sulfide, oxidized, mixed.*

Ore gland: magnetite, titanomagnetite, hematitomartite, brown-ironstone, siderite.

Manganese ores: brownite, psilomelanovade, pyrolusite, mixed, complex.

Mining chemical *raw materials*: apatite, apatite-nepheline, phosphorite, sylvinite ores.

*Fossil coals*represented by various lithotypes (vitren, Claren, Duren, Fusain), differing in external structure, chemical composition, physical properties and representing a certain combination of microcomponents.

Mineralogical composition useful fossils provides significant impact on both the choice of methods and technological indicators of enrichment. For example, when enriching non-ferrous metal ores, sulfide minerals are easily extracted by flotation, metal oxides and carbonates are extracted only after their preliminary sulfidization, and silicates of the same metals are not extracted by flotation at all. Similarly, when enriching iron ores, magnetite (Fe_30_4) is easily extracted by magnetic separation at low field strength, hematite (Fe_2ABOUT$_3$) is extracted only in high-gradient fields, and siderite ($FeC0_3$) is practically not extracted using the magnetic method.

To determine the mineralogical composition of minerals, use *macroscopic, microscopic, phase, thermal, luminescent, radiographic, microradiographic methods of analysis.*

1.4 Textural and structural characteristics

Textural and structural features in the structure of the mineral are characterized by*size, shape, spatial distribution of mineral inclusions and aggregates*.

The main forms of mineral grains are euhedral (limited by the crystal faces), allotriomorphic (limited by the shape of the filled space), colloidal, emulsion, lamellar - relict-residual, fragments and fragments.

Depending on the predominant size of mineral deposits, they are distinguished*large (20-2 mm), small (2-0.2 mm), thin (0.2-0.02 mm), very thin or emulsion (0.02-0.002 mm), submicroscopic (0.002-0.0002 mm) and colloidal dispersed (less than 0.0002 mm) dissemination of minerals.*

*The texture of the ore characterizes*relative arrangement of minerals units and can be very diverse. For example, in banded and in layered structures the aggregates are adjacent to each other; in concretionary ones - located one inside the other; in looped ones, they mutually penetrate each other.

The characteristics of mineral discharges are the basis for the development of technology and forecasting of mineral processing performance.

The larger the dissemination of minerals and the more perfect the shape of their secretions, the simpler the technology and the higher the rates of mineral enrichment.

1.5 Physical properties

Each ore mineral has a specific chemical composition and a characteristic structure. This determines the rather constant and individual physical properties of minerals: color; density; electrical conductivity; magnetic susceptibility, etc.

By creating in a certain way the conditions under which certain properties of minerals manifest themselves most contrastingly, it is possible to separate them from each other, including isolating valuable minerals from the total mass.

As*signs of separation*When mineral components are enriched, their physical and chemical properties are used, the most important of which are: mechanical strength; density; magnetic permeability; electrical conductivity and dielectric constant; various types of radiation; wettability; solubility, etc.

*Mechanical strength (strength)*ores and coals are characterized crushability, fragility, hardness, abrasiveness, temporary compressive strength and determines the energy costs during crushing and grinding, as well as the choice of crushing, grinding and processing equipment.

*Rock Density (δ)*determined by the density of their components minerals, which are divided into*heavy*(δ> 4x103 kg/m3),*average*(δ= (from 4.0 to 2.5)x103 kg/m3) and*lungs*(δ<2.5x103 kg/m3).

Nuclear physical properties minerals appear at their interaction with electromagnetic radiation (luminescence, photoelectric effect, Compton effect, fluorescence, etc.). The separation of minerals is based on differences in the intensity of emission or attenuation of radiation by them.

*Magnetic properties*minerals arise and appear in the magnetic field. A measure for assessing the magnetic properties of minerals is their magnetic permeability*μm*and the associated magnetic susceptibility equal to*$1/ \mu m$*. Magnetic properties are determined mainly by the chemical composition and partly by the structure of minerals. Increased magnetic susceptibility characteristic of minerals that include iron, nickel, manganese, chromium, vanadium, titanium.

Based on magnetic susceptibility and the nature of the dependence of magnetic properties on the strength of the external magnetic field, minerals are divided into diamagnetic ($\mu m < 1$), paramagnetic ($\mu m > 1$) and ferromagnetic ($\mu m \gg 1$).

The coal substance is diamagnetic, and the mineral impurities in it are paramagnetic.

Differences in the magnetic properties of minerals are used to separate them using magnetic enrichment methods.

Electrical *properties* minerals are determined electrical conductivity and dielectric constant.

A measure of electrical conductivity is electrical resistivity ρ and electrical conductivity *1/ ρ.*By electrical conductivity and type electronic structure, all minerals are divided into conductors (ρ=10-6 ÷10-3*Ohm•m*), semiconductors (*ρ = 10-3 ÷1010 Ohm•m*) and dielectrics (*ρ= 1010 ÷1017 Ohm•m*). Semiconductor minerals include most sulfides, some oxides and fossil coals. Dielectric minerals include minerals with typically ionic or covalent bonds: halides, silicates, and some salts of oxygen acids.

The dielectric constant*εm*for most silicates (quartz, mica) is 4-5, for salts of oxygen acids (calcite, apatite) - 6-8, for some oxides - up to 80 or more (for rutile -150).

Differences in the electrical properties of minerals are used to separate them using methods*electrical enrichment*.

Wetting- manifestation of intermolecular interaction on the boundary of contact of three phases - solid, liquid and gas, expressed in the spreading of liquid over the surface of the solid. A measure of wettability is the contact angle:

$$\cos \theta = (\sigma t\text{-}g - \sigma t\text{-}l)\, \sigma l\text{-}g \qquad (1.1)$$

Where*σ*- surface tension at the boundary of the corresponding phases. In extreme cases =0° - complete wetting (hydrophilic body), =180° - complete non-wetting (hydrophobic body).

Differences in surface wettability finely ground mineral particles are used for their separation by flotation enrichment methods.

Mineral solubility- the ability of minerals to dissolve in inorganic and organic solvents. The transition of the solid phase to the liquid state can be carried out by dissolution as a result of diffusion and intermolecular interaction or through chemical reactions. The ideal solubility of solids in liquid can be calculated using the equation of I.F. Schroeder:

$$\log = \frac{(- T_{pl})}{pl} \qquad (1.2)$$

Where*N*- mole fraction of the dissolved substance in the solution; - heat melting a mole of a solid,*kJ/mol; T_{pl}*- melting temperature;*T* - dissolution temperature,*TO;R*- universal gas constant. The actual solubility of solids is determined empirically. Differences in the solubility of mineral components are used in chemical methods of ore beneficiation.

Control questions

1. What is the subject "Mining"?
2. What are modern enrichment plants like?
3. What is a mineral?
4. According to physical properties, what types of minerals are shares?
5. What do you mean by mineral deposit?
6. Depending on the stage of metamorphism, what types of varieties coal do you know?
7. Define the concepts of ore and mineral.
8. What is the difference between a primary deposit and a placer deposit?
9. Based on their material composition, what types of ores are divided into?
10. What is the difference between monometallic ores and polymetallic ores?
11. What is the minimum industrial useful content? component in the ore?
12. Define the concepts of the main useful component, accompanying beneficial components, beneficial and harmful impurities.
13. How are the textural and structural features of ores characterized?
14. What physical properties are minerals characterized by?
15. What physical properties of minerals are used in flotation enrichment method?

Lecture No. 2. GENERAL INFORMATION ABOUT ENRICHMENT

MINERAL

The purpose and objectives of mineral processing. Classification of enrichment methods and processes. Technological enrichment schemes. Technological indicators of enrichment

*Key terms:*mineral processing, concentrate, tailings, concentrate quality, preparatory processes, main enrichment processes, enrichment methods, gravity enrichment, magnetic enrichment, flotation, auxiliary processes, process flow diagram, product yield, recovery, degree of concentration, degree of reduction, enrichment efficiency

2.1 Goals and objectives of mineral processing

Natural mineral raw materials extracted from the depths of the earth, in most cases, cannot be used in their natural form in the national economy, since they do not meet quality requirements. Direct metallurgical or chemical processing of mined ores is not economically viable due to the low content of useful components. Therefore, there is a need to first improve their quality.

In addition, extracted minerals often contain harmful components. For example, silica, sulfur and phosphorus - in iron ores, phosphorus - in titanium and niobium ores, iron - in zirconium ores, sulfur - in coals, etc. Harmful impurities must be removed from the ore as much as possible before metallurgical processing, as they deteriorate the quality of the resulting metal.

In connection with the noted circumstances, more than 80% of mined minerals are enriched.

*Mineral beneficiation*is a set of methods and processes of primary processing of mineral raw materials in order to concentrate valuable components in quality products by removing waste rock and separating minerals.

When enriching mineral resources, the following main tasks are solved:

- the content of useful components in raw materials increases;
- most of the harmful impurities are removed from the raw materials;
- homogeneity of raw materials in terms of size and material content is achieved composition.

The beneficiation of minerals is carried out at processing plants, which are independent structures or are part of mining and processing or mining and metallurgical plants.

As a result of the enrichment of natural mineral raw materials, one or more concentrates and waste (tailings) are obtained.

*Concentrate*called a enrichment product that has more high content of useful components compared to ore and suitable for further processing or direct use in the national economy. In terms of the content of the main useful component, impurities, moisture and granulometric composition, concentrates must meet the requirements of the relevant GOSTs, OSTs or technical specifications. Concentrates get their name from the base metal or mineral (copper, lead, rutile, etc.) that is concentrated in them during the enrichment process.

*Dump tailings*are called enrichment wastes consisting of mainly from waste rock with an insignificant content of useful components, the extraction of which is technologically impossible or economically unprofitable.

Product quality enrichment (concentrates) determined their content of valuable components (useful minerals), impurities and particle size distribution.

At enrichment plants factories, processing some non-metallic minerals often produce concentrates that are final marketable products (limestone, asbestos, graphite, etc.), but in most cases the beneficiation process is an intermediate link between the extraction of raw materials and the metallurgical smelting (or chemical processing) of the concentrates.

As a result of enrichment, a significant increase in the content of useful components in concentrates is achieved compared to ore.

At a number of non-ferrous metallurgy factories, more than 93% of the copper contained in the original ore, 82-90% of lead and zinc, 70-85% of nickel, tungsten, molybdenum, tin and other metals are extracted into concentrates.

When beneficiating minerals, it is important to correctly establish the depth of enrichment, which determines the content of valuable components in tailings and enrichment products. For each type of raw material, the optimal enrichment depth is determined through a feasibility study, taking into account technological, economic and environmental factors.

2.2 Classification of enrichment methods and processes

At processing plants, minerals are subjected to a series of sequential processing processes, which, according to their purpose, are divided into*preparatory, main enrichment, auxiliary and production service processes.*

Preparatory processes. Preparatory processes include crushing and grinding processes, in which the opening of minerals is achieved as a result of the destruction of intergrowths of useful minerals with waste rock (or intergrowths of some useful minerals with others) with the formation of a mechanical mixture of particles and pieces of different mineral composition, as well as screening and classification processes used for separation by size of mechanical mixtures obtained during crushing and grinding. The task of the preparatory processes is to bring mineral raw materials to the size necessary for subsequent enrichment, and in some cases, to obtain the final product of a given particle size distribution for direct use in the national economy (sorting of ores and coals).

*Basic enrichment processes.*To the main enrichment processes include those physical and physicochemical processes of mineral separation, in which useful minerals are separated into concentrates, and waste rock into tailings.

The processes of separation of minerals during the beneficiation of minerals are very numerous and are classified according to their affiliation with one or another beneficiation method, the separating feature, the nature of the separating forces and the design of the apparatus.

Enrichment methods are classified depending on what property of minerals is used as a separating feature and what the main separating forces are. The following enrichment methods are distinguished (Figure 2.1).

1. Gravity enrichment method (gravity enrichment), based on the difference in density of separated mineral grains, carried out in the field of gravitational forces.

2. Magnetic enrichment method (magnetic enrichment) based on the difference in the magnetic susceptibility of the separated minerals, carried out in a field of magnetic forces.

3. Electrical enrichment method (electrical enrichment), based on the difference in electrical conductivity of the separated minerals, carried out in a field of electrical forces.

4. Flotation enrichment method (flotation enrichment, or flotation), based on the difference in physicochemical properties (wettability) of the separated minerals.

5. Special enrichment methods based on difference combinations of properties of separated minerals. The latter include separation based on differences in radiospectroscopic properties, solubility, mechanical strength, decripitation, shape and friction, rebound elasticity, etc. The methods of radiometric and chemical enrichment are of greatest importance.

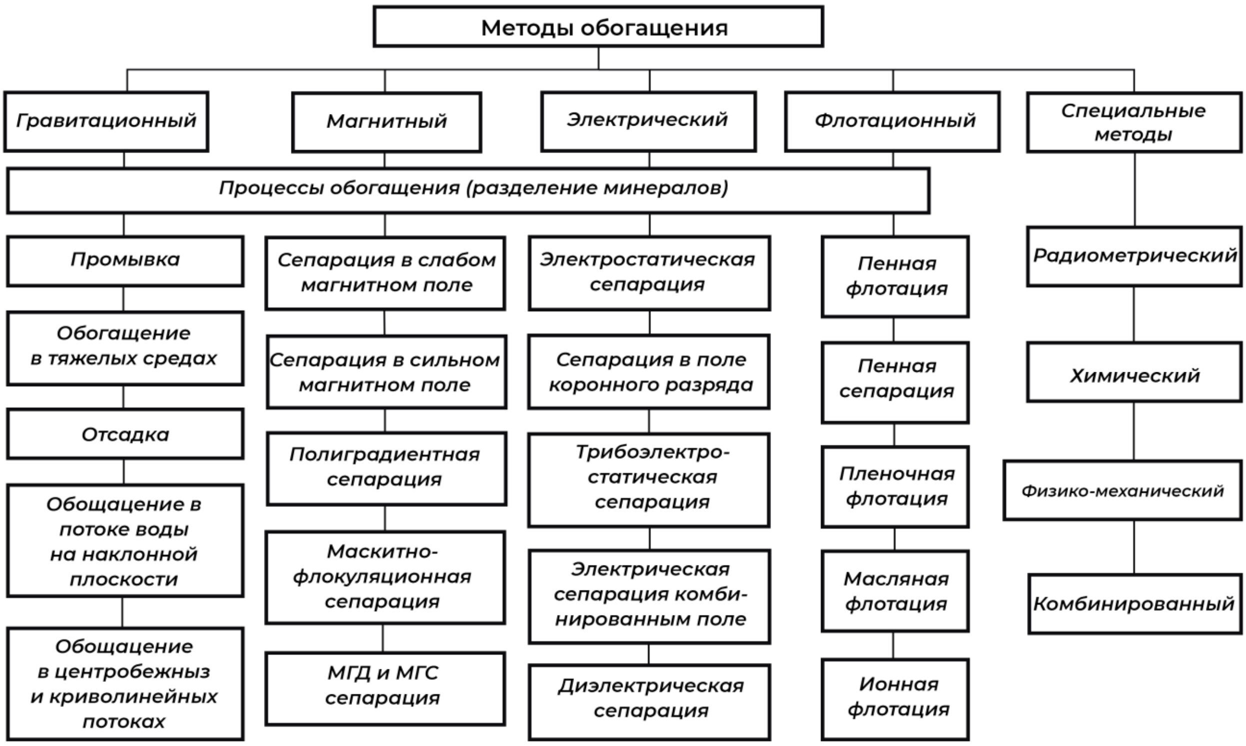

Figure 2.1 - Classification of methods and processes for enriching useful fossils

5.1. Radiometric enrichment method (radiometric enrichment), based on the difference in radiospectroscopic properties of the separated minerals, carried out using mechanical separating forces.

5.2. Chemical enrichment method (chemical enrichment), based on the difference in chemical properties (solubility) of separated minerals or harmful impurities.

5.3. Mechanical enrichment method (mechanical enrichment), based on the difference in physical and mechanical properties of minerals (mechanical strength, shape and friction, rebound elasticity, etc.).

Enrichment processes related to one or another enrichment method are distinguished by the variety of additional separating forces used, as well as the design of machines and apparatus.

Helper Processes. Auxiliary processes include dehydration of enrichment products (by thickening, filtering and drying) to bring their moisture content to the established standard or to obtain recycled water; processes of refining products and preparing them for metallurgical or chemical processing (agglomeration, pelletizing, briquetting, etc.); dust removal; sampling and control of technological processes; storage of enrichment waste.

Processes production service. TO processes production services include operations that ensure the continuity and stability of technological processes: intra-factory transport of raw materials and enrichment products, water supply, electricity supply, compressed air supply, mechanization and automation, technical control, etc.

2.3 Technological enrichment schemes

*Technological diagram*mineral enrichment is called graphical representation of the totality of all sequential technological operations of processing mineral raw materials at processing plants.

The technology of mineral enrichment, starting from the acceptance of raw materials at the factory and ending with the issuance of finished products, consists of separate techniques or operations. For example, operations of large, medium and fine crushing, concentrate cleaning, etc.

In very rare cases, the beneficiation of mineral raw materials can be completed in one step, isolating the final products immediately. Usually, after the first beneficiation of raw materials, the concentrate is not yet rich enough, and the tailings are not yet lean enough. In these cases, enrichment operations are repeated and are called cleaning operations, if applied to the resulting crude concentrates and middling products (recleaning of concentrates and middling products), and control operations, if applied to the tails of previous enrichment operations (for example, control flotation of tailings).

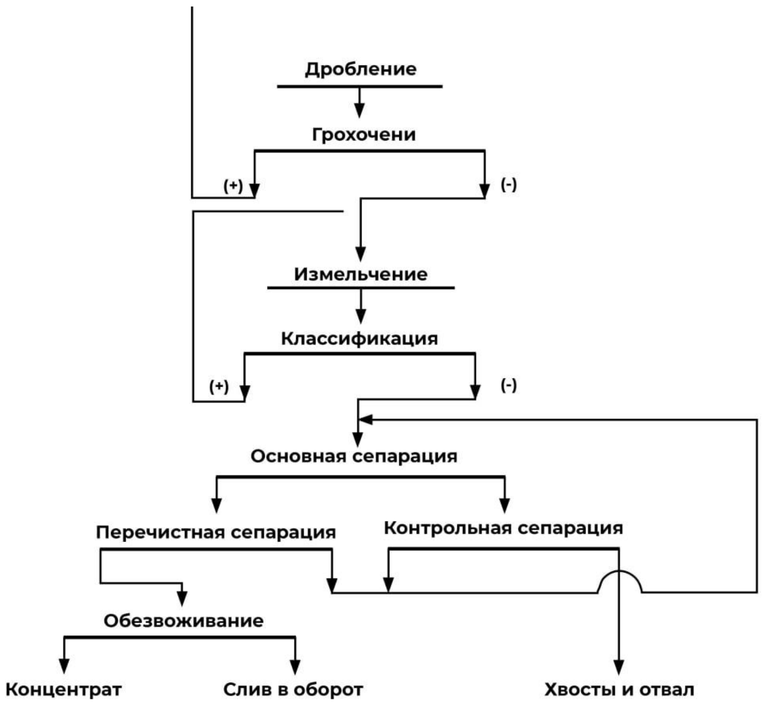

Figure 2.2 - Schematic diagram of mineral processing

Enrichment schemes are depicted in a certain established order. *Technological operations depict*thick horizontal line 1-2 thick*mm*, above which the name of the operation is written. The movement of products is indicated by lines with an arrow. When vertical and horizontal product movement lines intersect, the outline is shown on the horizontal line. When constructing a circuit, they strive to ensure a minimum of flows directed vertically and the output of all enrichment products to a horizontal straight line at the bottom of the circuit. Figure 2.2 shows a simplified ore beneficiation diagram.

The same enrichment scheme can be implemented in different ways. Thus, individual operations can be carried out in different machines, and the same operation can be performed in one or more machines.

2.4 Technological indicators of enrichment

The main technological indicators of mineral processing processes are*quality and yield of products, extraction valuable components.*

The quality of enrichment products is determined by the content of valuable components, harmful impurities, particle size distribution and must meet the requirements for them by consumers. The quality requirements for concentrates are called*condition*, they are regulated by GOSTs, technical conditions (TU) and temporary standards. The standards establish the average and minimum or maximum permissible content of various components in the final enrichment products and, if necessary, the content of classes of a certain size in the resulting products or their particle size distribution.

*Contents of components*in the original mineral ( ), obtained concentrates ( ), and in tails ( ) is usually given as a percentage, and precious metals - in grams per ton of product (*g/t*).

Yield of enrichment product ( ) - amount of product received (concentrate, tailings), expressed as a percentage or fraction of a unit to the original. The total yield of all enrichment products must correspond to the yield of the starting material, taken as 100%. When dividing the enriched raw material into two final products - concentrate (with the yield $_{To}$*)*and tails (with exit $_{xv}$) - this condition is written in the form of the following equality, which is called the product balance equation:

$$\gamma_{To}+\gamma_{xv}=100\ \%, \qquad (2.1)$$

Considering that the amount of a valuable component in the original (100) is equal to its total amount in the concentrate ($_{To}$) and tails ($_{xv}$), you can

Taking into account equality (2.1), draw up a component balance equation for the starting material and enrichment products:

$$100 \;\; = \gamma_{To}\beta + \gamma_{xv}\theta, \qquad (2.2)$$

Solving equation (2.2) for $_{To}$or γ_{xv} *(%)*, we obtain dependencies for calculating the yield of concentrate and tailings:

$$100 \;\; = \gamma_{To}\beta + (100 - \gamma_{To})\, \theta \text{ or } 100 \;\; = (100 - \gamma_{xv})\beta + \gamma_{xv}\theta,$$

$$\gamma_{To} = 100\,(\;\; - \theta\,)/(\,\beta - \theta); \text{ or } \gamma_{xv} = 100\,(\beta - \;)/(\,\beta - \theta)\,, \% \qquad (2.3)$$

Extraction(휀) - an indicator indicating what part of the extracted component contained in the source material has passed into a concentrate or other enrichment product. Extraction is expressed as a percentage, less often - in fractions of a unit and is defined as the ratio of the mass of a component in a given product (,) to its mass in the starting material (100).

The extraction of the component into the concentrate is:

$$휀_{To} = \frac{\gamma_{To}\beta}{100}\, 100 = \gamma_{To}\beta, \% \qquad (2.4)$$

If the yield of the concentrate is unknown, then the recovery of the component into the concentrate can be calculated using the equation:

$$휀_{To} = \frac{\beta(\; - \theta)}{(\beta - \theta)}\, 100, \% \qquad (2.5)$$

obtained by substituting into equation (2.4) the expression for $\gamma_{To}$from equation (2.3).

The total recovery of each component in all final enrichment products obtained is 100%.

The extraction of valuable components into concentrate during mineral processing ranges from 60 to 95% and higher.

Degree of concentration(K) - an indicator indicating how many times the content of the useful component in the concentrate increased compared to its content in the source material. Defined as the ratio of the content of a useful component in a concentrate*(* $_{To}$) to its content in the source material ( ):

$$= \frac{_{To}}{} \qquad (2.6)$$

The degree of concentration during mineral processing can be from 2 to 100.

Degree of reduction(R) - an indicator indicating how many times the mass of the resulting concentrate ($_{To}$) less than the mass of processed mineral. The degree of reduction during mineral processing can range from 2 to 50 or more.

Enrichment efficiency( ) mineral upon separation it into two products is usually determined by the Hancock-Luyken formula:

$$= \frac{\text{휀} - \gamma_{To} 100,}{100 - \alpha} \qquad (2.7)$$

Process:
- very effective if η > 75%,
- effective at η > 50% and
- ineffective - η < 25%

Control questions

1. What are the goals and objectives of mineral processing?
2. What is mineral processing?
3. What problems are solved during mineral processing?
4. What is a concentrate and what are the requirements for it?
5. What is dump tailings and what are the requirements for them?
6. What are the main types of mineral processing processes? fossils?
7. What processes are preparatory?
8. What processes are the main enrichment processes?
9. What methods of mineral processing exist?
10. What processes are auxiliary processes?
11. What processes are related to production processes service?
12. Define the concept of “process flow diagram”
13. What parameters are the main technological ones indicators of enrichment processes?
14. How is the yield of enrichment products determined? (Write the formulas)
15. What is recovery and how is it determined?
16. What does the degree of concentration of a useful component mean?
17. What formula is used to determine the efficiency of enrichment mineral?

Section 2. PREPARATORY PROCESSES

Lecture No. 3. SPLITTING UP

Purpose and classification of processes. Theoretical foundations of crushing. Classification of crushing machines. Jaw crushers. Cone crushers. Roll crushers and impact crushers

*Key terms:*crushing, degree of crushing, crushing stage, crushing efficiency, crushing methods, jaw crusher, grip angle, lining plate, spacer plate, cone crusher, moving cone, exit slot width, eccentric shaft, roll crusher, hammer crusher

3.1 Purpose and classification of the crushing process

Splitting up- this is the process of reducing the size of useful pieces fossils by destroying them under the action of external forces that overcome the internal cohesive forces that bind together the particles of solid matter.

According to their technological purpose, three types of crushing processes are distinguished:

independent- crushing products are final (commodity) and are not subject to further processing. For example, crushing coals, rocks to produce crushed stone, etc.;

preparatory- crushed products are given a given size and subjected to subsequent processing. For example, crushing ores for subsequent enrichment;

selective- one of the material components is different insignificant strength and is destroyed more efficiently than the other, with their subsequent separation by size.

The classification of processes and crushing and grinding machines is carried out according to the method of destruction of the material, which is determined by the type of energy used for destruction. The following methods are distinguished:

- *mechanical*carried out through the use of mechanical strength;
- *pneumatic (explosive)*- use of steam or compressed energy air;
- *electrohydraulic, electric pulse, electrothermal*- electricity use;
- *aerodynamic (jet)*- use of gas jet energy, accelerating pieces of material before their collision;

• *ultrasonic* carried out through the use of energy ultrasound, which causes resonant vibrations in pieces of material and their destruction.

Mining and processing enterprises mainly use mechanical methods of crushing and grinding.

Mechanical crushing called the process of particle destruction rocks under the influence of external mechanical forces to obtain a product of a given size.

The intensity of the crushing process is characterized by the degree of crushing.

Degree of crushing - an indicator indicating how many times per As a result of crushing, the size of the largest pieces of minerals decreased. Degree of crushing *i* defined as the ratio of size maximum pieces *Dmax* contained in the material supplied for crushing, to the maximum size of pieces d_{max} contained in the crushed product:

$$I= \quad / \tag{3.1}$$

Sometimes the degree of crushing is determined as the ratio of the average diameters of the particles of the initial feed D_{Wed} and crushed product d_{Wed}:

$$= \ _{Wed} / \ _{Wed} \tag{3.2}$$

Crushing stage called part of the overall crushing process, carried out in one crushing machine. Depending on the upper limit of the size of the crushed product, three stages of crushing are conventionally distinguished:

- *large* - from1200 to 300 mm;
- *average* - from300 to 75 mm;
- *small* - from75 to 10-15 mm.

Then for coarse crushing the degree of crushing is I_1= 1200/300 = 4; for average I_2= 300/75 = 4; for small I_3= 75/15 = 5.

When crushing in several successive stages, the total degree of crushing is equal to the product of the degrees of crushing of the individual stages:

$$I=I_1I_2I_3= 4•4•5 = 80$$

Final size crushing (grinding) of minerals before enrichment is determined by the size of the inclusion of useful minerals and the enrichment method used.

3.2 Theoretical basis of crushing

The opening of minerals during crushing and grinding occurs due to the destruction of pieces of rock under the influence of external loads. To break pieces of ore, it is necessary to overcome the adhesion forces between individual crystals and within the crystals. These forces determine the strength of rocks, which depends on defects in the internal structure, for example, the presence of internal weakened zones (cracks, inclusions)

Figure 3.1 shows the destruction methods used or their combinations.

*Crushing methods*differ in the type of basic irreversible deformations:

- *crushing*(drawing3.1,*A*) - destruction as a result of compression of a piece between crushing surfaces, occurring after the stress passes beyond the compressive strength;

- *splitting*(drawing3.1,*b*) - destruction as a result wedging a piece between the edges of crushing surfaces and its subsequent rupture;

- *kink*(drawing3.1,*V*) - destruction of a piece as a result of its bending ribbed crushing surfaces;

- *cutting*(drawing3.1,*G*) - destruction in which the material undergoes shear deformation;

- *abrasion*(drawing3.1,*d*) - destruction of pieces of sliding working the surface of the machine, in which the outer layers of the piece are subjected to shear deformation and are gradually cut off due to the transition of tangential stresses beyond the strength limits;

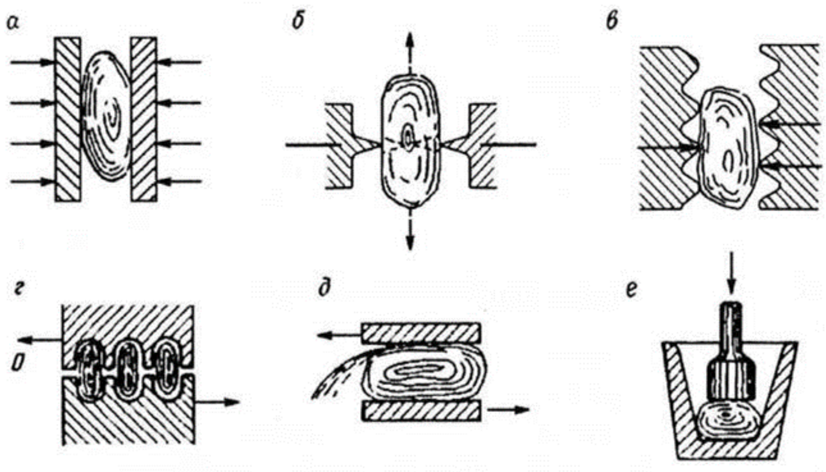

Figure 3.1 - Methods of crushing minerals

- impact (Figure 3.1,*e*) - destruction of a piece as a result of impact dynamic short-term loads. Impact crushing leads to destruction primarily along cracks and contact surfaces of grains of individual components.

Depending on the properties of rocks (strength, fragility, viscosity, etc.), the most effective method of external force is selected impact on pieces of rock for the purpose of crushing them. For example, for rocks that are strong and not brittle, the best method of destruction may be crushing or impact. If there are a large number of cracks in brittle rock, destruction by impact is preferable, but when the viscosity of the rock is high, the effectiveness of the impact is sharply reduced.

The choice of crushing method is also influenced by the value of the mineral and the quality requirements of the crushed product. If, for example, the ore contains fragile and valuable useful minerals, then when crushing it it is necessary to eliminate as much as possible the effect of abrasion, which leads to over-grinding of the ore and the formation of difficult-to-process sludge.

The energy used for crushing (grinding) is spent on elastic deformation of destroyed grains and on the formation of a new surface, dissipates into the surrounding space in the form of heat and turns into free surface energy of crushed grains.

In the general case, the elementary work A spent on the destruction of a piece of material is the sum of the work spent on its deformation and on the formation of a new surface, determined by the equation P.A. Rebindera:

$$A=Ad + As=k\Delta V+A0\Delta S, \qquad (3.3)$$

Where*Hell*- work of elastic deformations;*As*- work spent on formation of a new surface;*k*- proportionality coefficient, representing the work of deformation per unit of deformed volume of grain;Δ*V*- change in the volume of deformed grain;*A0*- proportionality coefficient, which represents the cost of work to form a unit of new surface;Δ*S*- newly formed surface during grinding.

During coarse crushing of ore with a small degree of crushing, the area of the newly formed surface is relatively small and the work spent on the formation of this surface can be neglected. In this case, the entire crushing work will be proportional to the deformed volume of the pieces (Kirpichev-Kick hypothesis):

$$A=k_1\Delta V, \qquad (3.4)$$

Wherek_1- empirical coefficient.

With fine crushing and grinding, the work required to deform the grains is significantly less than the work required to form new surfaces and can also be neglected. Then the work spent on the destruction of grains will be proportional only to the area of the newly formed surface (Rittinger's hypothesis):

$$A = k_2 \Delta S, \qquad (3.5)$$

Where k_2 - empirical coefficient.

According to Bond's hypothesis, the work spent on crushing is proportional to the geometric mean of the volume and surface of the grain being destroyed and is:

$$A = A_d + A_s = k\sqrt{\ \ } = \sqrt{\ 1\ 3\sqrt{\ 2\ 3}} = 0\ \overline{2.5}, \qquad (3.6)$$

Where 0 - empirical coefficient.

The considered crushing laws characterize the dependence of the work spent on crushing (grinding) on the results of crushing (grinding), i.e. on the size of the final product:

$$A = K\ , \qquad (3.7)$$

Where *TO* - proportionality coefficient, *N m/*m2*; D* - characteristic piece size*, m*.

Exponent *m* when determining *A* according to the Kirpichev-Kick law is equal to 3, according to Rittinger's law - 2, according to Bond's law - 2.5.

In the general case, the proposed laws of crushing practically describe different parts of the curve of specific energy consumption for crushing and grinding *E =f(S)*, Where *S* - area of the newly formed surfaces.

3.3 Classification of crushing machines

The classification of crushing and grinding machines and devices is based on the principle of their operation, i.e. method of destruction, which is determined by the type of energy directly used to destroy the material.

Based on their operating principle, crushers are divided into the following types:

- *cheek* - with simple and complex movements of the cheek;
- *conical* - with a suspended shaft (KKD, KRD), with a cantilever shaft (KSD, KMD);
- *roller* - two-roll with smooth rolls, two- and four-roll with toothed rolls and single-roll with toothed or grooved rolls;
- *drums* - hammer, rotary and rod;
- *with shockless rotor* - centrifugal single-disk and centrifugal multi-disc. :

At high-capacity processing plants for coarse, medium and fine crushing of ores, cone crushers are most widely used, characterized by high productivity and operating mainly on the principle of crushing, partially on the principle of abrasion and bending. In medium-capacity factories, jaw crushers are used instead of cone crushers, which are characterized by lower productivity. When a high degree of crushing is required at low productivity, roller crushers with smooth rolls are used. For soft ores, if it is necessary to obtain the lowest possible yield of fines, roller gear crushers are used. Impact crushers, which provide the highest degree of crushing, are becoming increasingly widespread for crushing minerals of various strengths.

When choosing a crusher, the size of the source material, the required particle size distribution of the crushed product and the required productivity are taken into account.

3.4. Jaw crushers

In a jaw crusher (Figure 3.2), the material is destroyed by crushing combined with splitting and bending between the fixed 1 and movable 2 jaws. The movable jaw 2 approaches (during the working stroke) or moves away (during idle stroke) from the fixed jaw 1 when the eccentric shaft 3 rotates. During the working stroke, crushing occurs, and during the idle stroke, the crushed material is discharged downwards under the influence of its own weight. The cheek 2 is driven by a connecting rod 4, movably connected to the eccentric shaft 3, and two hinged spacer plates - front 5 and rear 6. Rod 7 and spring 8 create tension in the moving system and promote idling of the movable cheek. The width of the outlet and, consequently, the degree of crushing is adjusted by mutual movement of the wedges 9.

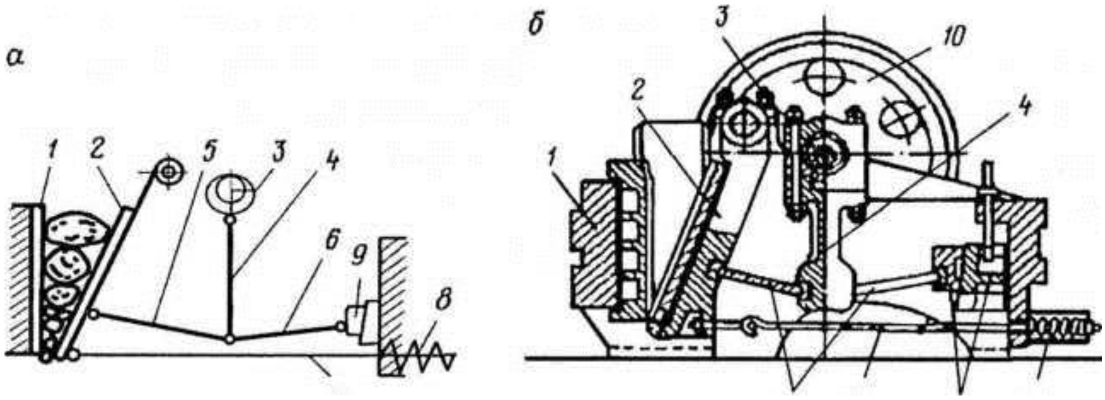

Figure 3.2 - Operating principle (*A*) and general view of the jaw crusher (*b*):

1 - fixed cheek; 2 - movable cheek; 3 - eccentric shaft; 4 - connecting rod; 5 - front spacer plate; 6 - rear spacer plate; 7 - traction; 8 - spring; 9 - adjusting wedge; 10 – flywheel

Depending on the location of the axis of the movable cheek, jaw crushers are distinguished with an upper and lower suspension of this cheek.

Figure 3.2 shows a general view of a jaw crusher with an upper suspension of a movable jaw. Crushers of this type are the most widely used in industry. The crusher body 1, the front wall of which is a fixed jaw, is usually made of steel casting, and the jaws are lined with steel plates 2 with a corrugated working surface. These plates wear out the most, as a result of which they are removable and made of wear-resistant material (cast manganese or chromium steel).

The crushing force in the crusher is transmitted through spacer plates 5 and 6. Therefore, the liners 4, into which the ends of the plates fit, are made replaceable from a material of great hardness. The rear spacer plate is used to protect the crusher from damage when non-crushable objects enter the working space. This plate is made with reduced strength and breaks when random metal objects enter the crusher.

Depending on the kinematic drive diagram of the crusher with upper and lower jaw suspension, there can be two types: with simple and complex jaw movement.

In addition to the simplest kinematic scheme (see Figure 3.2), more complex ones are also used: both cheeks are driven, or one cheek is driven by two eccentric shafts and the movement of the cheeks is carried out using unbalanced vibrators, etc.

The main parameters characterizing a jaw crusher are the dimensions of the receiving opening (B - width, L - length).

The maximum size of the largest piece of material loaded into the crusher should be 15-20% less than the width of the receiving opening. Attitude*L/B*crushers is assumed to be equal1.3-1.5

The size of the crushed product depends on the width of the discharge opening*b*, measured by the distance from the extreme point of the protrusion at the lower end of the lining plate of one cheek to the most distant point in the cavity of the lining plate of the other cheek when they are open:

The width of the crusher discharge opening is adjusted using a special mechanism.

The main technological parameters of the mechanical operating mode of jaw crushers are: grip angle - a; movement of the movable cheek - *S*; working shaft rotation speed; performance; power consumption of the electric motor.

The maximum crushing ratio that can be achieved in jaw crushers is 8. Typically, crushers operate at crushing ratios of 3 to 4.

Characteristic size product crushing determined properties of the crushed material, and above all its strength

3.5 Cone crushers

Crushing of material in cone crushers is carried out in the annular space between the fixed body 1 and the movable (crushing) cone 2 located inside it (Figure 3.3). Axis of rotation*K.O.* the movable cone is slightly inclined to the axis 1 of a stationary cone, which provides a certain eccentricity value (e). Therefore, when moving along eccentricity, the movable cone performs a gyrational movement inside the stationary cone, approaching or moving away every half turn to one or another side of the stationary cone located opposite. The movable cone, as if rolling the inner surface of the stationary cone, crushes large pieces as a result of their crushing, as well as partial abrasion and breaking due to the curvilinear shape of the crushing surfaces. The source material is loaded from above into the space between the movable 2 and stationary 1 cones, and the crushed product is unloaded downwards under the crusher through the hole formed when the movable cone moves away from the stationary one.

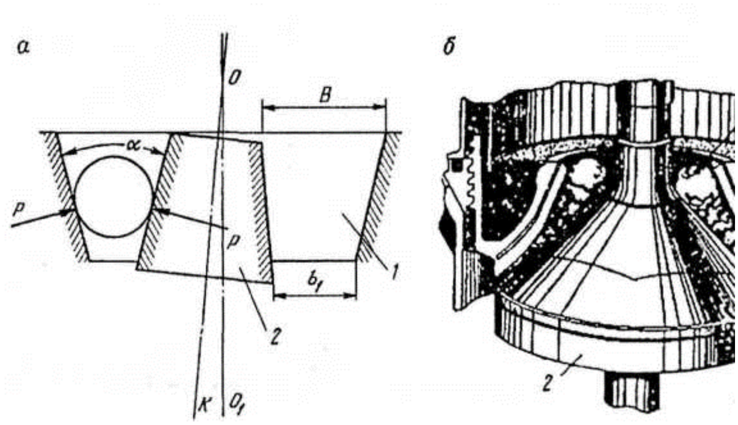

Figure 3.3 - Operating principle (*A*) and general view (*b*) cone crusher:
1 - fixed cone (body); 2 - movable cone; B is the width of the receiving hole; b - width of the discharge opening; a - grip angle; R - crushing force

Exit slit width*b*for modern crushers is (0.1- 0.2)*IN*, and the maximum diameter of the crushing cone is approximately 1.*5V* (Here*IN*- width of the crusher receiving opening). Width*b*unloading The holes are adjusted by raising or lowering the crushing cone.

Cone crushers are used for coarse (KKD), medium (KSD) and fine (KMD) crushing of ores, mining chemical raw materials and construction rocks.

The main differences between cone crushers for coarse, medium and fine crushing are: the profile of their working space; kinematics of movement of the working cone and the method of supporting it; drive mechanism cars; a method for unloading crushed material and a method for generating crushing force.

In a coarse crusher (Figure 3.4,*a B C*) a steep movable cone 1 is driven around a fixed axis by an eccentric shaft using a bevel gear. The fixed cone (bowl) 2 faces upward with its large base.

In a medium and fine crusher (Figure 3.4,*G*) a flat movable cone 1, mounted on a shaft rotating with the help of an eccentric glass, is located inside a stationary cone 2 (facing the large base down). At the moment of maximum approach of the crushing cone to the bowl of such a crusher, a “parallel zone” of length*l*. The width of this zone determines the size of the crushed product pieces.

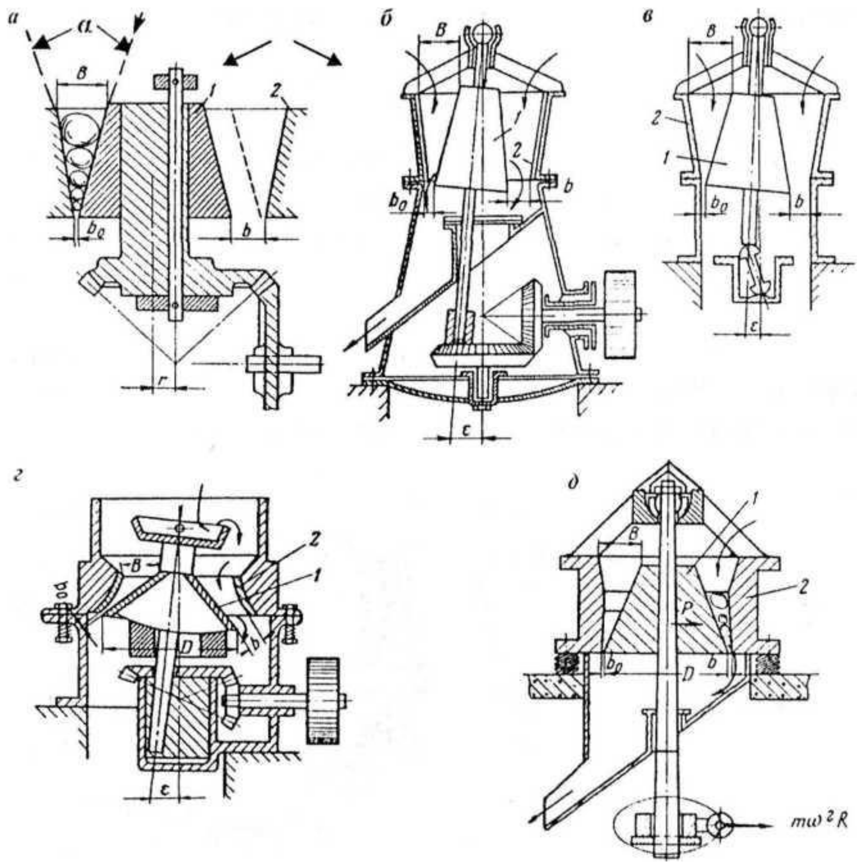

Figure 3.4 - Schemes of cone crushers:

A- cone crusher with a fixed axis;*b*- Same. with suspended shaft (KKD “gyrational”);*V*- the same, with a shaft having a support (main switchboard);*G*- the same, with a cantilever shaft resting on a ball bearing (KSD and KMD);*d*- cone inertial crusher (vibration without eccentric KID)

Coarse and fine crushers can have a drive either in the form of an eccentric shaft or in the form of an eccentric cup. Coarse crushers are widely used, in which the movable cone is also driven by an eccentric glass.

The crushing process in cone crushers occurs continuously with sequential movement of the crushing zone around the circumference of the cones.

The crushed material, under the influence of its own gravity, is unloaded through the unloading opening, which in the open position has a width*b*. When the crushing organs are closed, the hole is reduced to the size $_0$, and $_0 = b - s$, Where*s*- cone stroke (double amplitude) at the lowest point.

Nominal size $_n$the largest pieces of material that can be loaded into the crusher is determined by the radial width B of the receiving opening. Usually taken $_n$-0.8*IN*.

The size of the crushed product and the productivity of a crusher of a given size depend mainly on the width of the discharge opening

*Main parameters*mechanical mode of cone crushers are: grip angle; diameter, eccentricity and stroke of the crushing cone; frequency of its swings; crushing force and power consumption.

3.6. Roll crushers and impact crushers

In roller crushers, the material is crushed between two rolls rotating towards each other (Figure 3.5). One of them rotates in fixed bearings 1, the other in movable bearings 2, held in guides by spiral shock-absorbing springs 4, which protect the crusher from damage when non-crushable objects enter it. With a smooth surface of the rolls, crushing of the material is carried out by crushing and partially abrasion, with a serrated and corrugated surface - by splitting, breaking and crushing. The crushed material enters the space between the rolls from above, is drawn in by them and is crushed. The crushed product falls out of the crusher under the influence of gravity.

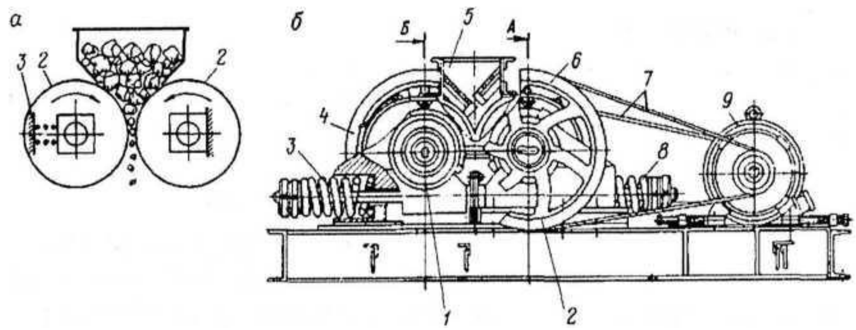

Figure 3.5 - Operating principle (a) and general view (b) of a two-roll crusher:

1 - bed; 2 - rollers; 3.8 - clamping springs; 4 - stationary roll pulley; 5 - loading funnel; 6 - movable roll pulley; 7 - V-belt drive; 9 – electric motor.

The rolls are made of cast iron and lined on the outer surface with tires made of carbon or wear-resistant manganese steel. The peripheral speed of the rolls ranges from 2 to 7*m/s.*The drive mechanism of the roll crusher consists of two belt drives - on the pulley of each roll from a separate engine

Depending on the design features and purpose, the following types of roller crushers are used: single-roller - for sinter and coal; two-roll (with smooth and grooved rolls) - for rocks and ores; twin-roll with toothed rolls - for coal and soft rocks.

In impact crushers, the crushed material is destroyed by impact due to the kinetic energy of moving bodies. Impact crushers, based on the design of the main crushing body, are divided into hammer, rotary and disintegrators.

The crushing elements of hammer and rotary crushers are rotating beaters (hammers), which are hinged on the disks of the rotating rotor, and rigidly in rotary crushers.

These types of crushers are used for crushing low-strength materials (f< 10), as well as sticky material

Control questions

1. What is crushing and what types is it divided into?
2. What is the degree of crushing and how is it determined?
3. What stages of crushing do you know?
4. How do you understand the process of destruction (crushing) of pieces of ore?
5. What methods of crushing ore are there?
6. Based on the principle of operation, what types of crushers are divided into?
7. What are the main parts of a jaw crusher?
8. Explain the operating principle of a jaw crusher
9. On what parameters does the size of the crushed product depend? jaw crushers?
10. What are the main parts of a cone crusher?
11. What types of cone crushers do you know?
12. How does the crushing process occur in cone crushers?
13. What do you understand the term “parallel zone” in conical crushers?
14. Explain the operating principle of a cone crusher
15. What are the main parts of a roll crusher?
16. What types of impact crushers do you know and where are they? are used?

Lecture No. 4. CRUSHING SCHEME. EXPLOITATION CRUSHERS

Crushing schemes. Operation of jaw crushers. Operation of cone crushers. Operation of roller crushers. Operation of hammer and impact crushers

Basic terms: crushing operations, open crushing cycle, closed crushing cycle, crushing stage, crushing scheme, crusher discharge hole, coarse crushing, medium crushing, fine crushing, receiving hopper, apron feeder, grate screen, belt conveyor, lining plates, spacer plates, eccentric shaft, moving jaw, coarse ore warehouse , eccentric cup, support ring, bevel gear.

4.1 Crushing schemes

Crushing operations are used, as a rule, to prepare the mineral for crushing and only in some cases, when the mineral is characterized by large inclusions of valuable minerals, directly for enrichment operations. In crushing and screening factories, crushing has its own significance.

Splitting up is a very energy-intensive process for which About half of the energy consumed by the enrichment plant is consumed. Therefore, they strive to reduce as much as possible the volume of material sent to these operations, guided by the principle *"Not crush nothing superfluous."*

Materials subject to crushing always contain pieces (grains) smaller than the size to which crushing occurs at this stage. It is advisable to separate such pieces from the source material before crushing machines on screens.

Crushers can operate in open and closed cycles with screens (Figure 4.1).

Open crushing circuit - this is the crushing of material without subsequent screening or without returning the coarse product to the crushing equipment.

Closed crushing cycle - this is the crushing of material carried out followed by screening, after which the coarse product is continuously returned back to the crushing equipment.

In an open circuit, the material passes through the crusher once and there is always some oversized pieces in the crushed product.

In a closed cycle, the material passes through the crusher repeatedly. The crushed product is fed to a screen, which separates out oversized pieces, which are returned to the same crusher for additional crushing.

*Crushing stage*called a set of crushing operations and screening. In this case, the screening operation is usually referred to as the crushing operation into which the upper product of the screen enters.

If screening precedes crushing, then it is called *preliminary*, and if after crushing, then*verification (control*). When the crushed product is returned to the previous screening operation, the latter is called*combined* operation of preliminary and calibration screening.

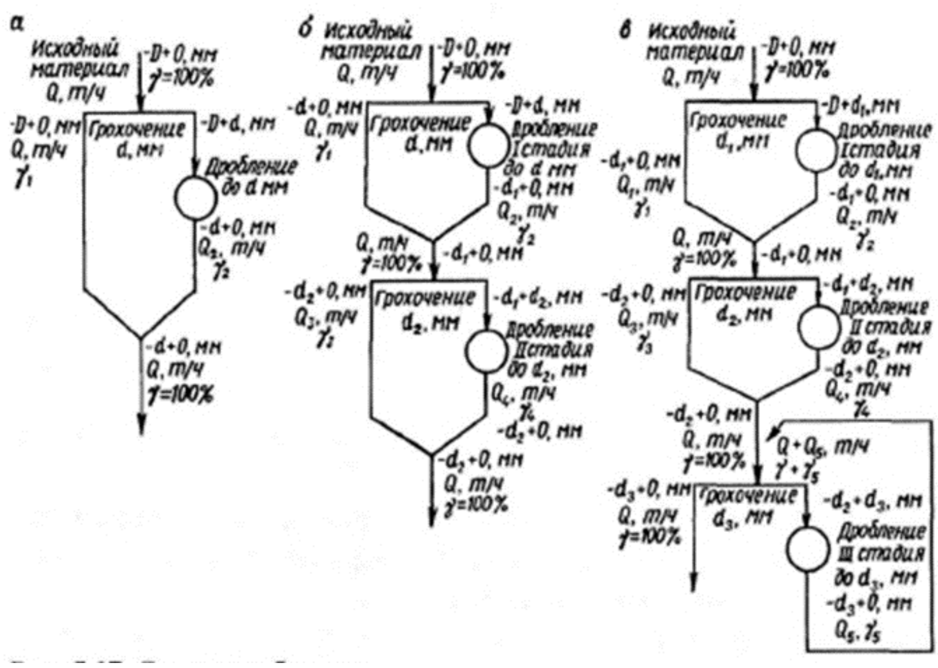

Figure 4.1 - Crushing schemes:

A- one-stage; *b*- two-stage; *V*- three-stage; *Q*- Product mass productivity; – product yield;*D*And*d*-sizes of pieces in products

*Crushing scheme*is a graphical representation of the sequence crushing operations. Crushing schemes consist of one or several crushing stages, including preliminary and calibration screening operations. Such diagrams are explained with a graph on which, in addition to the names of the stages, they indicate the mass, yield and size of crushed products, as well as the dimensions of the discharge openings of the crushers in each stage (Figure 4.1).

The diagram in Figure 4.1,*A*is single-stage, with an open cycle, with preliminary screening. The diagram in Figure 4.1,*b*- two-stage with an open cycle, with preliminary screening in the second stage, with preliminary and calibration screening in the first stage of crushing.

The diagram in Figure 4.1, *is* three-stage, with an open cycle in the first and second stages and a closed cycle in the third stage, with preliminary and calibration screening in all three stages, while preliminary and calibration screening in the third stage is combined.

Depending on the crushability, mineralogical fracturing, size and composition, other properties of the rock and on the required granulometric composition of the crushed product, the type of crusher is selected.

It is advisable to carry out large, medium and fine crushing of hard rocks in crushers operating primarily on the principle of crushing (jaw, cone and roll crushers with smooth rolls); coarse crushing of soft and brittle rocks - in crushers operating primarily on the splitting principle (toothed roll crushers, needle crushers, peak crushers), and medium and fine crushing - in impact crushers (hammer crushers, selective crushers); medium and fine crushing of hard and viscous rocks - in crushers operating on the principle of crushing with the participation of abrasion (roll crushers with smooth rolls, etc.)

4.2 Operation of jaw crushers

The simplicity of design, maintenance and repair of jaw crushers determines their widespread use in processing plants. However, these crushers also have significant disadvantages: they vibrate strongly during operation (the crusher must be installed on a very solid foundation and only on the lower floors of buildings), become clogged with ore when loaded unevenly, and produce a product of uneven size.

In processing plants, jaw crushers are used for coarse crushing. These crushers cannot operate under rubble and therefore, to receive ore arriving at the factory, receiving funnels or bunkers with a small capacity are built (Figure 4.2). From the receiving device to the crusher, ore is fed evenly by a plate feeder. Often a grate screen is installed in front of the crusher. The apron feeder feeds the ore to the screen and only the over-size product enters the crusher. The crushed product is usually discharged from the crusher onto a conveyor belt, which transports it to the next crushing stage.

Jaw crushers are installed on foundations that are not connected to the foundations of the building, so that vibrations and shocks of the crusher, which are inevitable during its operation, are not transmitted to the foundations.

Wearing parts that are subject to periodic replacement or restoration for jaw crushers are the following: lining plates, spacer plates, liners in the sockets for spacer plates, bearing liners for the eccentric shaft and the axis of the moving jaw, liners or filling for the connecting rod head.

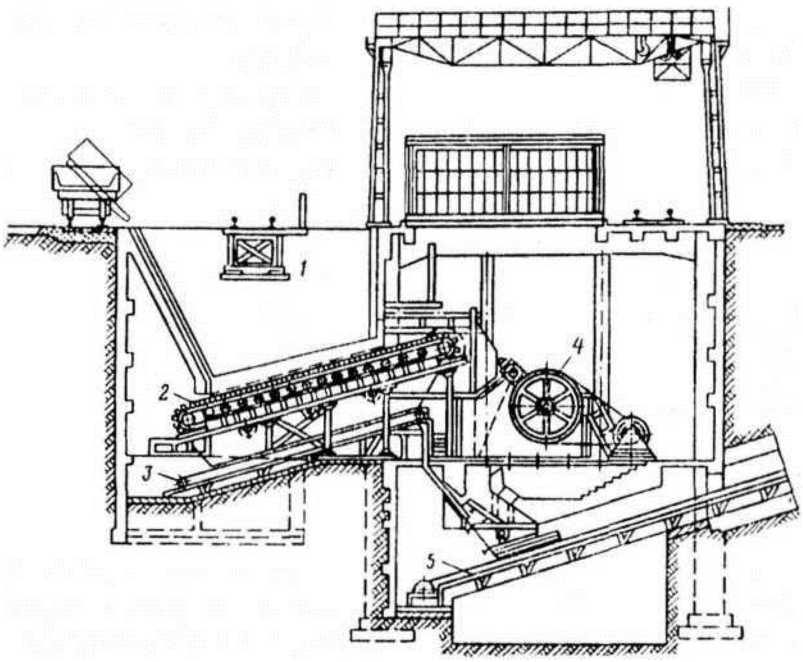

Figure 4.2 - Jaw crusher installation diagram:
1 - receiving funnel; 2 - plate feeder; 3 - belt conveyor for spilling; 4 - jaw crusher; 5 - belt conveyor for crushed product.

The average service life of these parts (*in months*): lining plates - 6; replaceable tips of spacer plates - 5; crackers in the nests of the spacer plates - 12; bearing shells for the crankshaft and movable cheek axis - 12; liners and filling of the connecting rod head - 12.

The consumption of steel during crushing in jaw crushers is determined by the abrasion of the lining plates; it depends on the durability of the material from which the plates are made, and on the strength of the crushed material. When using manganese steel plates, its consumption ranges from 0.02 to 0.08 *kg*, and from hardened cast iron - from 0.3 to 0.1*kg*by 1*T*crushed product.

Scheme automatic management crusher founded on controlling the level of material in the crushing zone.

The jaw crusher is started only empty (without ore). If there is no abnormal noise during idle operation (knocking, rattling, creaking, etc.), the crusher is loaded with ore. The jaw crusher can be stopped only after releasing all the material remaining in the working area.

4.3 Operation of cone crushers

Coarse cone crushers (*IN*>900*mm*) provided that the car (dump truck) supplying ore matches the productivity of the crushers, they can operate under the blockage, which allows loading crushed material directly from tipping cars (Figure 4.3,*A*). Smaller crushers cannot operate under the rubble, and therefore receiving devices for the source material are built for them. In this case, the material is fed from the receiving device to the crusher by an apron feeder.

The crushed material is unloaded onto a conveyor belt, which transfers it to the next crushing stage. Typically, the material is transferred to secondary crushing by a belt conveyor, and a screen is installed in front of the crusher to screen out fines that cannot be crushed. The over-size product from the screen enters the crusher, and the under-size product, bypassing the crusher, is sent to the crushed product conveyor (Figure 4.3,*b*).

In high-capacity factories, the operating mode of the coarse crushing department often does not coincide with the operating mode of the medium and fine crushing department. Therefore, a warehouse for coarse crushed ore is built between these departments, which is also used at the same time for distributing ore to medium-crushing crushers, since in terms of productivity it is necessary to install several crushers operating in parallel. From the warehouse, ore is supplied by separate conveyors to one crusher (Figure 4.3,*b*). In modern processing plants, medium and fine crushers are located on the same level and in the same building. To distribute ore among crushers, small-capacity bunkers were built, under which screens were placed. The large grade is fed to the crushers by a short conveyor.

With this layout solution, all crushers are placed above one collection conveyor, which removes the material unloaded from the crushers from the housing.

For coarse cone crushers operating at a crushing degree of 6, energy consumption for crushing ranges from 0.1 to 0.8*kW•h/t* crushed product.

Wearing parts that are subject to periodic replacement or restoration: lining of the fixed outer bowl, crushing cone and traverse; contact surfaces in the place where the crushing cone shaft is suspended and the surface of the eccentric glass; eccentric cup support ring, drive shaft bushings and bevel gears. Service life for lining the outer fixed part - from 6*months*up to 2 years (usually about 1 year).

The consumption of steel during crushing in cone crushers is determined by the abrasion of the lining plates. When using a manganese steel lining, its consumption ranges from 0.005 to 0.03*kg/t*crushed product.

Cone crushers are put into operation when there is no crushed material in the crushing chamber. Before starting, check the amount of lubricant in the liquid lubricant tank and in the grease tank. First turn on the oil pump and oil cooling system

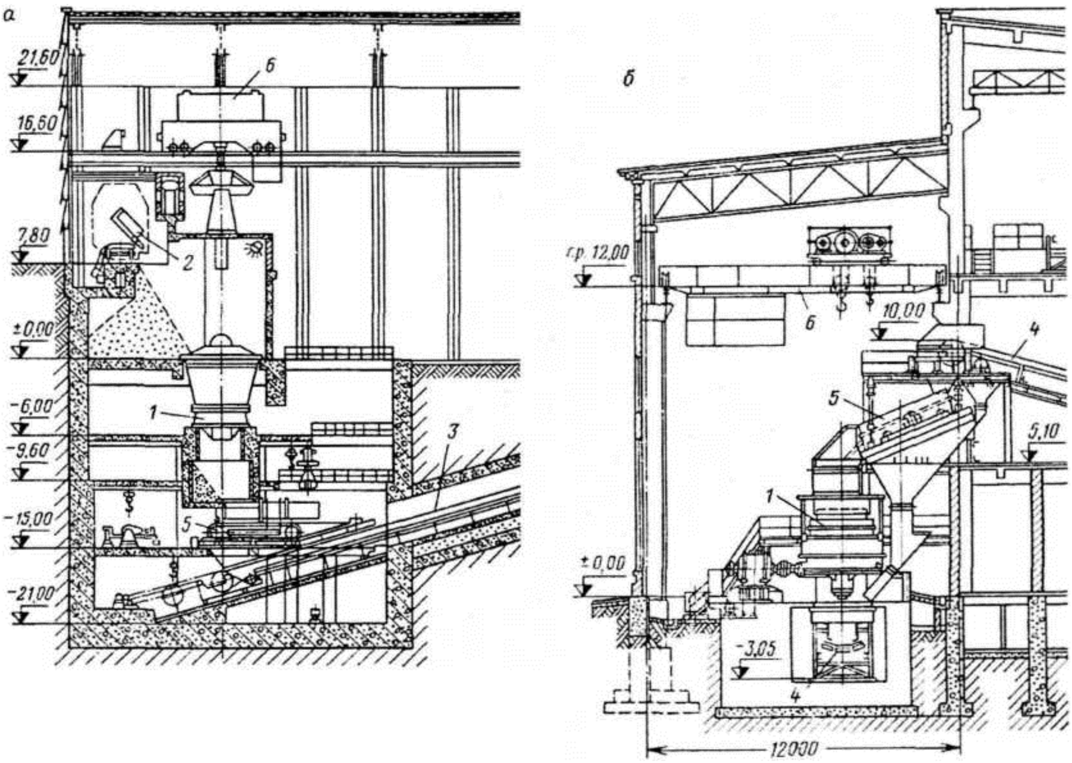

Figure 4.3 - Installation diagrams for large (a) and medium cone crushers (b) crushing:

1 - cone crusher; 2 - railway tipping car; 3 - belt conveyor for crushed product; 4 - conveyor for feeding material into the medium crusher; 5 - roar; 6 - overhead crane

Metal objects getting into medium and fine crushers along with ore can cause breakage of the crushers. A modern scheme for collecting metal objects from ore fed by a conveyor to a crusher involves installing two metal detectors along the conveyor and a powerful suspended electromagnet between them.

Cone crushers are widely used in the mining industry due to their high productivity, relatively low specific energy consumption and the ability to produce a crushed product that is fairly uniform in size. The disadvantages of cone crushers (compared to jaw crushers) are: more complex and expensive design, greater height, more complex maintenance

4.4 - Operation of roller crushers

Roller crushers with smooth rollers are used for fine crushing of ores when overgrinding of a valuable brittle mineral (cassiterite, wolframite ores) is unacceptable. The crushed material is fed to the crusher rolls in a stream of one piece or rolls thick working under the rubble. With the first loading method, the productivity of the crushers is less than with the second, but the overgrinding of the material is also less.

Steel consumption during crushing is determined by the wear of the lining bands and ranges from 0.02 to 0.06*kg/t*with bandages made of high carbon steel.

Crushers with smooth rollers in an open circuit most often operate at a crushing degree of 3-4. In a closed cycle they can work with large degrees of crushing.

Electricity consumption for crushing depends on the method of feeding the crushed material to the rolls. When feeding crushers in-line, the flow rate ranges from 0.3 to 1.5*kW·h/t*; when working under rubble - on average 0.13 *kW·h/t*.

Roller crushers with toothed rolls are most often used for coarse crushing of coal.

Before crushing, raw coal is usually screened and only the over-size screen product is sent to the crusher. The material is usually fed to the screen by a conveyor belt. When loading material into the crusher, it is necessary to ensure that it flows along the entire length of the rolls so that their entire surface is worked. The crushed product is usually sent to the next operation by conveyor or gravity feed through a chute. In coal preparation plants, roller gear crushers are often installed on floors.

The fastest wearing parts of these crushers are the gear segments and bearing shells. The segments are made of manganese steel, and the teeth along the cutting edges are surfaced with a hard alloy.

4.5 - Operation of hammer and impact crushers

Hammer and impact crushers (Figure 4.4) are installed on high and hollow foundations, the dimensions of which are selected taking into account the placement of vehicles under the crusher. The mass of the foundation must be sufficient to compensate for the vibrations that occur during operation of the crusher.

It is necessary to load material into the crusher evenly across the width of the receiving opening and over time, which ensures maximum crusher performance and a more uniform product size. Therefore, feeders are usually installed in front of hammer and impact crushers. To prevent pieces from flying out of the receiving hole, there are boxes closed at the top, the material entrance into which has a curtain made of conveyor belts or chains. These same boxes are also used as a shelter for suction of dust ejected from the crusher through the receiving hole.

The noise level in the immediate vicinity of a working hammer or impact crusher is above sanitary standards. For example, when crushing coal it reaches 102-104*dB*. Therefore, it is necessary to install crushers so that there are no permanent workplaces in the immediate vicinity.

Steel consumption in hammer and rotary crushers depends on the properties of the crushed material and is, for example, for average crushing of soft limestones about 0.0015*kg/t*, coal - 0.001*kg/t*, oil shale - 0.015*kg/t*.

Energy consumption in hammer crushers when crushing coal with a degree of crushing from 6 to 12 ranges from 0.6 to 1.5*kW·h/t*. Catchers are installed in front of the crushers to remove foreign, non-crushable metal objects from the loaded material.

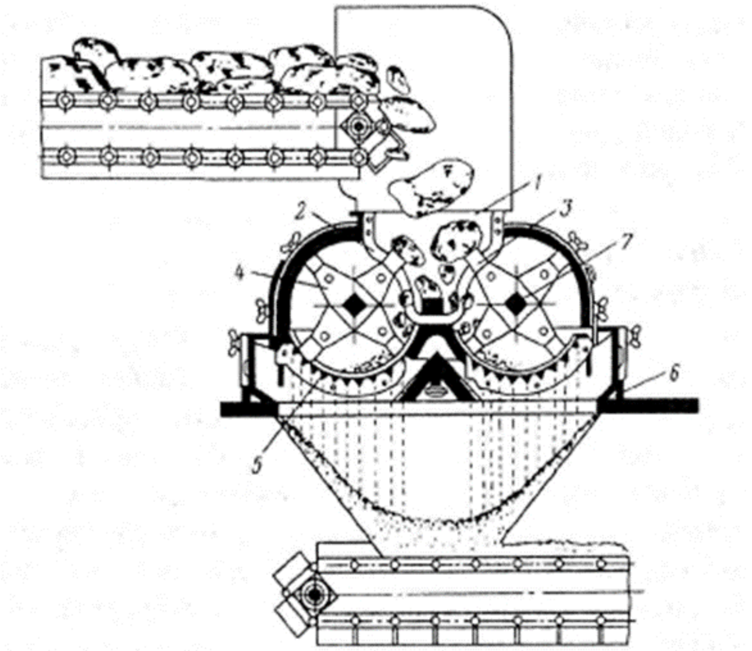

Figure 4.4 - Double-rotor hammer crusher with parallel rotor arrangement:

1 - loading funnel; 2 - figured grate bars; 3 - hammers; 4 - hammer holders; 5 - grates; 6 - crusher body; 7 - rotor shafts

Control questions

1. For what purpose is the crushing process carried out?
2. How to state the principle of crushing and tell the essence of this principle.
3. Define the concepts of open and closed crushing cycle?
4. How do you understand the term “crushing stage”?
5. What types of crushing schemes are there?
6. Explain the advantages and disadvantages of jaw crushers
7. Which jaw crusher parts are considered wear parts?
8. Explain the procedure for starting a jaw crusher

9. Tell us about the place of cone crushers in the crushing scheme
10. How is a cone crusher started?
11. What protective measures are taken against the ingress of unbreakable materials into the crusher?
12. Where and for what purposes are roller crushers used?
13. Tell us about the operating procedure for impact crushers

Lecture No. 5. SCREENING

Purpose And classification processes screening. granulometric composition of minerals. Theoretical foundations of screening. Classification and designs of screens. Technological parameters of the screening process.

*Key terms:*screening, size class, screening surfaces, screening scale, classification scale module, under-screen product, over-screen product, independent screening, preparatory screening, auxiliary screening, particle size distribution, sieve analysis, “light grains”, “difficult grains”, screening surfaces, grate sieves, open section.

5.1 - Purpose and classification of screening processes

*Screening*is the process of dividing materials into size classes, carried out on screening surfaces.

Screening surfaces are made of various materials and have through holes of various shapes and sizes.

The essence of the screening process is that particles of the initial feed, smaller in size than the sieve holes, pass through these holes under the influence of gravity and vibrations of the screen. Particles larger than the sieve openings remain on it and are removed from the screen (Figure 5.1).

Figure 5.1 - Screening diagram

The material entering the screening process is called*original* remaining on the sieve -*oversize (*top) product falling through the sieve holes*- under the slat*(bottom) product.

When sequentially sifting the material into *n* sitah get *n+1* products. In this case, one of the products from the previous sifting serves as the starting material for subsequent sifting.

A sequential series of absolute values of screen openings (from large to small) used during screening is called *scale screening or classification*.

Classification scale module - constant size ratio openings of previous sieves to the size of openings of subsequent ones. For example, for a classification scale of 100; 50; 25; 12.5; 6.25 *mm* the modulus is 2 (100/50 = 50/25 = 25/12.5 = 12.5 / 6.25).

Size *d* the largest grains (pieces) of the under-sieve product are the same, as well as the size of the sieve holes through which the material is sifted

Respectively designate: under-slat product -*d* (minus *d*); oversize product +*d* (plus *d*)

Size class called material that has passed through a sieve with holes $d1$ and remaining on a sieve with holes $d2$, and $d2 < d1$. The size of the class is indicated by: $-d1+d2$, or $d1-d2$, or $d2-d1$. For example, class - 25 + 10 *mm*, class 25 - 10 *mm*, class 10 - 25 *mm*.

Screening operations are widely used in processing plants and screening plants in the production of building materials, as well as in the chemical, abrasive and many other industries.

Depending on the purpose, screening can be independent, preparatory, auxiliary, or for the purpose of dewatering.

Self screening - the process of separating material into products of a given size, which are final commercial products intended for shipment to consumers.

Preparatory screening - the process of dividing material into two or several classes subjected to separate processing at a given factory, for example, before separate enrichment of size classes on different devices.

Auxiliary screening provided for in crushing schemes and grinding in order to isolate small classes that are not subject to crushing (grinding).

Screening for dewatering - dehydration operation on screens of enrichment products or desliming of material before further enrichment.

In some cases, screening is aimed at enriching the mineral and is often called selective screening. As a result of such screening, products are obtained that differ not only in size, but also in the content of a valuable component in them. Selective screening does not take advantage of differences in the physical properties of the individual components that make up the fossil raw material, for example, differences in hardness and strength or in the shape of pieces of a valuable component and waste rock. When extracting, transporting and crushing such raw materials, products of different sizes will have different contents of useful minerals.

According to the method of identifying machine classes, the following types of screening are distinguished:

- dry - without the use of a processing medium or with use in the quality of the specially supplied air;
- wet or hydro-screening - using as processing medium specially supplied water;
- combined - sequential combination of dry and wet screening.

5.2 Particle size distribution of minerals

Particle size distribution is the composition of a material, expressed through the content of particles of different size classes in it as a percentage of the whole. The particle size is usually characterized by the average diameter *dav*, depending on the length *l*, width *b* and heights *L* particles:

$$d_{Wed} = (l+b+L)/3, \text{ or } d_{Wed} = \sqrt{\ .}\qquad (5.1)$$

To determine the granulometric composition of a mineral, sieve sedimentation and microscopic analysis is carried out, divided into size classes limited by the sizes of the maximum and minimum grains in them

Size class called a collection of pieces with dimensions determined by the size of the openings of the sieves, which are used to separate these pieces.

Accepted designation of size class ($-A+b$) means that all grains in this class are smaller in size than A, but more than b.

Average class size determined by the arithmetic mean diameter:

$$d_{Wed} = (d_1 + d_2)/2 \qquad (5.2)$$

Where d_1 And d_2 - the minimum and maximum particle size of the class, respectively, *mm*.

Sieve analysis is called a method for determining granulometric composition by sieving a sample of the material on sieves.

The weight of the sample for sieve analysis is taken depending on the size of the largest piece in the sample.

Size of the largest piece, mm	*0.1*	*0.3*	*0.5*	*1*	*3*	*5*	*10*
Minimum sample weight, kg	*0.025*	*0.05*	*0.1*	*0.2*	*0.3*	*2.25*	*18*

Samples are sieved dry or wet depending on the size of the material and the required accuracy of the sieve analysis.

Material larger than 25*mm*dispersed on oscillating horizontal screens and hand sieves, and finer than 25*mm*- on laboratory sieves.

For sieving, a set of wire sieves with square holes corresponding to the standard scale is used.

To carry out sieve analysis, a standard set of sieves is used. The yield of classes is determined by dividing their mass by the mass of the sample. The total yield of the oversize product is determined by sequentially summing the yields of the classes from above. The results of the sieve analysis are recorded in a table.

5.3. Theoretical foundations of screening

Sifting grains of the lower class of bulk material through a sieve can be considered as an operation consisting of two stages: the grains of the lower class must pass through a layer of grains of the upper class to reach the surface of the sieve; The lower class grains must pass through the sieve holes. The implementation of both stages is facilitated by the appropriate movement of the screen box, which brings the layer of grains on the sieve into a loosened state and frees the sieve from grains stuck in its holes.

When the box is shaken, in the layer of grains lying on the sieve, their segregation (separation by size) occurs, with the largest grains ending up in the upper layer, and the smallest ones on the surface of the sieve. The latter easily reach the surface of the sieve and pass through its holes.

The grains pass through the holes unhindered if they do not touch the wire, i.e. when the center of the grain when falling is projected onto the shaded area$(l\text{-}d)^2$(Figure 5.2).

We can assume that the number of cases favorable for the passage of grain through the hole is proportional to the shaded area$(l\text{-}d)^2$, and the number of all possible cases of grain falling onto a hole is proportional to its areal^2. The probability of grain passing through the hole is determined by the area ratio:

$$= \frac{(\ -\ )^2}{^2} = (1 - \ \frac{}{}\ )^2 \qquad (5.3)$$

Taking into account the thickness of the sieve wires, the following expression was obtained for the probability of grain passing through the sieve:

$$= \frac{(\ -\ )^2}{(\ +\ )^2} = \frac{^2}{(\ +\ )^2}(1 - \ \frac{}{}\ )^2 \qquad (5.4)$$

The first term of this expression $\frac{\quad}{(\ +\ )}$ is *coefficient live section of the sieve*. Consequently, the probability of grain passing is directly proportional to the open section of the sieve.

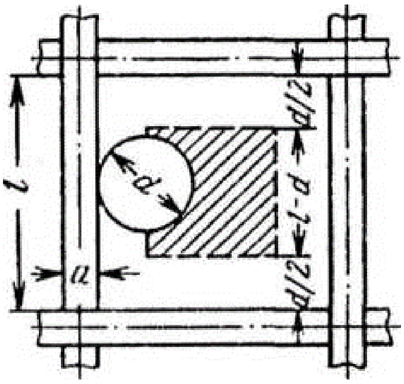

Figure 5.2 - Scheme of grain passage through a square sieve hole

Grains up to 0.75 in size l are more likely to pass through the holes of the sieve and are called *easy to screen*. Slight increase in grain size beyond 0.75l predetermines a sharp decline probability of passage. Therefore, grain sizes ranging from 0.75l before l are called *difficult to screen*. Grains with a diameter of l before 1.5l are called *"obstructive"* because they make sifting difficult *"difficult"* grains Grains larger than 1.5l do not significantly affect the movement of the “lungs” and “difficult” grains on the surface of the sieve (Figure 5.3)

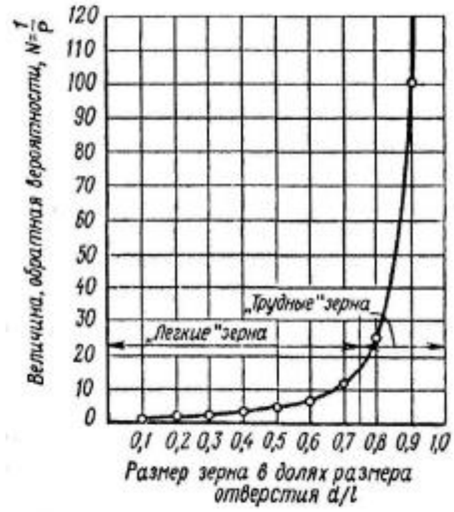

Figure 5.3 - Probability of grains passing through the sieve depending on their relative size

The higher the speed of movement of the material along the screen, the lower the probability of sifting, all other things being equal, and the greater the productivity of the screen for the source material

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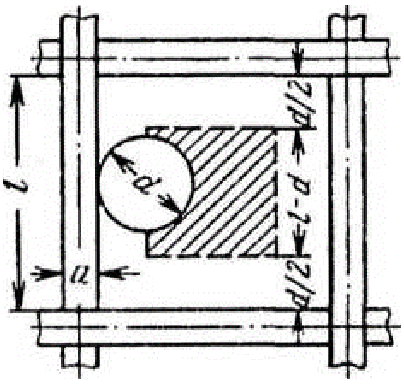

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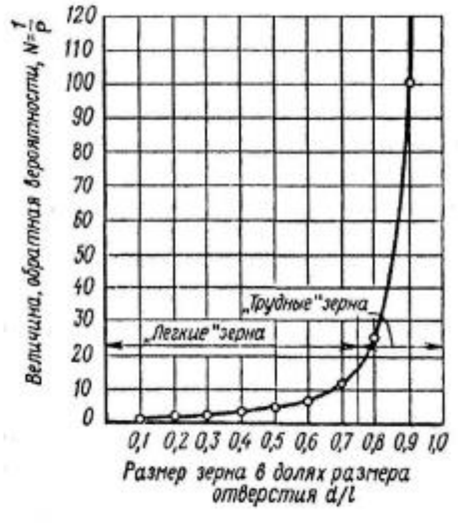


Figure 5.3 - Probability of grains passing through the sieve depending on their relative size

The higher the speed of movement of the material along the screen, the lower the probability of sifting, all other things being equal, and the greater the productivity of the screen for the source material

5.4. Classification and designs of screens

Bolt- a machine for separating the starting material into two or more size class, for washing and dewatering on a screening surface.

Known big number various designs screens, designed to separate minerals into size classes. However, the principle of operation is the same for all - separation by size occurs by sifting out fines from the material entering the screening when it moves in a loosened state along the sifting surface. The difference between individual types of screens lies in the method of loosening the material on the screening surface of the screen. Based on their design, screens are divided into:*into stationary, flat and curvilinear, roller, drum, flat swinging, gyratory and inertial with circular movements, vibration with rectilinear reciprocating movements (resonance).*

Conventionally, different types of screens are designated by letters and numbers. First letter*G*stands for rumble, second letter*G*stands for hydraulic screen,*AND* - inertial,*WITH*- self-balanced,*R*- resonant,*C*- cylindrical,*D*- two-box,*L*- light type, *WITH*- average,*T*- heavy type. The first digit of the numerical designation characterizes the width of the screen box (3 -1250, 4-1500, 5-1750, 6-2000, 7-2500*mm*), the second digit is the number of sieves. For example, GIL-43 is a lightweight inertial screen with a box width of 1500*mm*, three-sieve.

Screens are produced with a cover (for the dry screening method) and without cover (for the wet screening method).

*Screening surfaces.*As working sifters surfaces, grates, stamped cast or welded sieves, wire and rubber sieves are used (Figure 5.3).

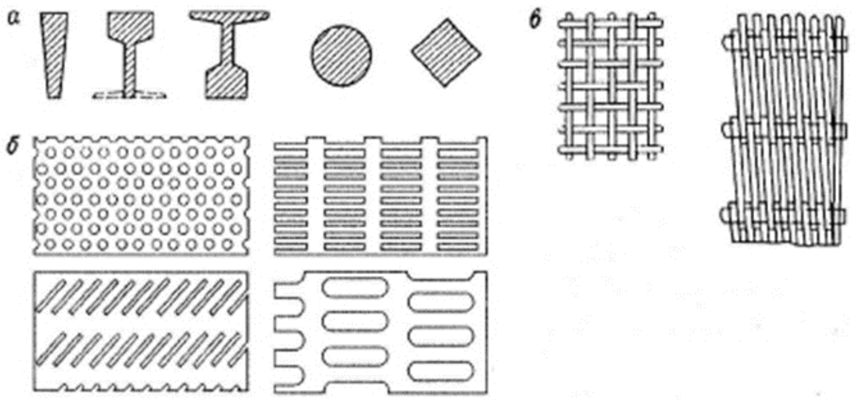


Figure 5.3 - Screening surfaces of screens:
a - cross sections of the grate; b - sheet sieves; c - wire sieves

Grate sieves consist of a series of parallel located grates of various sections. The size of the grate openings is determined by the width of the gap between the grates. Slot width 40-50*mm*. Grids are used for preliminary screening (usually before coarse crushers).

Sieve (stamped, drilled and cast) are used for screening by size from 10 to 100*mm*. The sieves are made of carbon and stainless steel with round, square and rectangular holes measuring 5-150*mm*.

To ensure the strength of the sheet thickness *b* take depending on size of screening holes. For holes with diameter *d* more 20*mm* And *b* = (0.1 -÷- 0.25)*d*, diameter less than 20*mm* *b* = (0.25 -÷- 0.6)*d*. Service life of stamped sieves 4-6*months*. To increase durability, sieves are made of cast rubber. Their service life increases by 10-20 times compared to metal ones.

Sita There are woven, wicker, string and spalt. Woven and woven sieves are made mainly with square and rectangular holes ranging in size from 100 to 0.04*mm* made of steel, brass, bronze, copper or nickel wire. Recently, rubber, nylon and caprosteel sieves have been produced. Spalt sieves are slot-shaped sieves made from round wire, usually from trapezoidal cross-section rods. Spalt sieves are intended for removing small grades. The width of the slot-shaped holes in the light can be from 0.25 to 16*mm*. Slot-shaped sieves are made of stainless steel, their service life is 2-3*months*.

Live section of the sieving surface (live coefficient sections) L_0 represents the ratio of the area occupied by holes to of the entire area, expressed as a percentage.

For wire mesh with square holes:

$$L_0 = 100a^2/(a+b)^2, \tag{5.5}$$

Where *A* - side size of a square hole, mm; *b* - thickness (diameter) wires, *mm*.

For screens made of perforated sheets with square holes:

$$L_0 = 100 \quad ^2 \tag{5.6}$$

Where - number of holes per 1 m^3 sieves; – side of the cell, *m.*

For sieves with round holes:

$$L_0 = 100 \quad ^2/4 \tag{5.7}$$

Where *d* - hole diameter, *m*.

For grates and sheet sieves, the open section coefficient does not exceed 40-50%, for wire sieves it reaches 70%. For the smallest (control) meshes, the wire for which is made from non-ferrous metal alloys (brass, bronze), the open section coefficient varies from 32.5% for a mesh of 0.04 *mm* up to 70% for mesh 2.5 *mm*.

With a decrease in the live cross-section, but with the same cell sizes, the productivity of the screen decreases, but the service life of the meshes increases, since larger diameter wire is used for denser meshes

Grizzly screens. Processing plants use both fixed and moving grate screens.

A stationary grate screen is a grid assembled from grate bars fastened with bolts. Spacers are installed on the bolts between the grates, which determine the width of the gap. The slot width is usually at least 50 *mm*. The screen is installed obliquely at an angle of 40-50° to the horizontal.

The ore moves along the grate under the influence of gravity. Small pieces fall through the cracks, and the over-sieve product comes off the bottom end.

Self-balanced screens. Screen with self-balanced vibration exciter (Figure 5.4) has a horizontally located box, which, with the help of shock absorbers *4* installed on supports *5*. A vibration exciter is mounted on the box *2* in such a way that the direction of action of the inertial force vibration exciter is approximately *50°* to the sieve plane *3*.

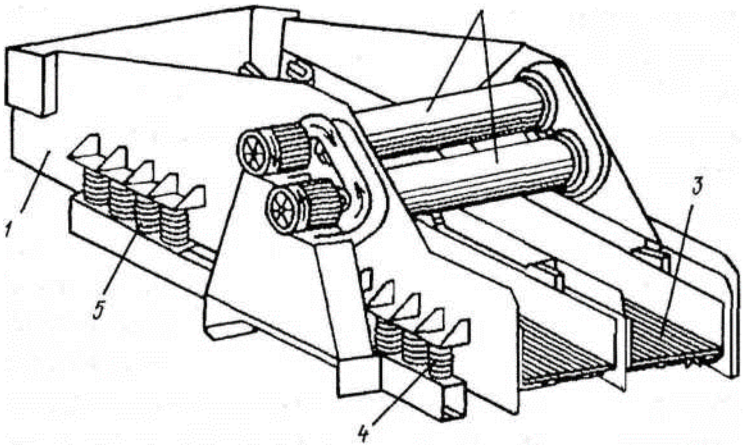


Figure 5.4 - Self-balanced screen (self-synchronizing)

A self-balanced vibration exciter consists of two identical unbalances rotating on parallel shafts at the same speed in opposite directions.

The vibration exciters are located in such a way that when the shafts rotate, the resulting centrifugal forces of the unbalances are directed along the axis passing through the center of gravity of the box. Changing according to a sinusoidal law, this force acts on the box and causes it to oscillate. Since the rigidity of the support springs is relatively small, all points of the box oscillate in vertical planes along straight trajectories at an angle to the screening surface. In this case, the material is thrown up, moves forward and makes its way through the holes of the sieve.

Self-balanced screens are used for screening ores, dewatering coal, oil shale, wet classification operations, and separating suspensions from separation products in heavy environments.

Self-balanced screens have small dimensions in height, relatively low metal consumption (600-900 *kg* by 1 m^2 working surfaces), they are simple and reliable in operation, and are characterized by an effective screening mode.

The disadvantage of a self-balanced vibrator is its complexity - four bearings for two shafts, a pair of gears, a sealed housing with an oil bath.

Curved and flat screens. Arc sieves are used for wet screening and dewatering with screen opening sizes of 0.3-3 *mm*. The principle of operation of the arc sieve is clear from Figure 5.5. The working sieve is a part of a cylindrical surface with a radius of 500-1200 *mm* with a central angle of 90-270°. The slotted sieve is made from stainless steel wires of trapezoidal cross-section. For classification arc sieves, the slots between the wires are located across the pulp flow, and for dewatering sieves - along the flow. The pulp is fed tangentially to the screen at the upper end and is subjected to the centrifugal force of inertia and gravity.

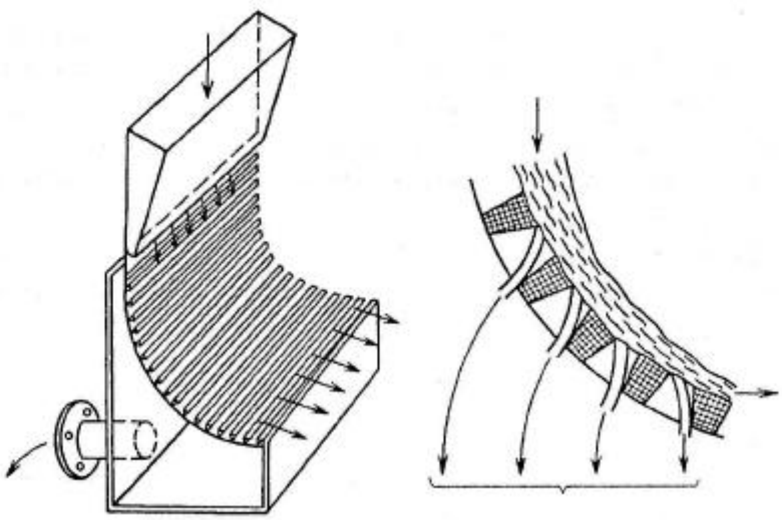


Figure 5.5 - Operating principle of the arc sieve

As a result of the pressure caused by these forces, part of the flow passes through the openings of the sieve, carrying with it small grains of material

Pulp with different solid contents (from 7 to 70% solid by weight) can be fed to arc sieves. The efficiency of screening depends on the characteristics of the initial size, the size of the sieve sieve, pulp liquefaction, etc. For a sieve with a slit size of 1*mm*efficiency is about 90%; 0.7-0.3*mm*- 70% and 0.3*mm*- 33%.

Arc performance sieves By original pulp is proportional to the open cross-sectional area of the sieve and the pulp feed rate.

Arc sieves are widely used for coal desliming and sludge classification, also in ore processing plants for classification in the grinding cycle

5.5 Process parameters affecting the screening process

The main technological parameters influencing the screening process are*humidity, granulometric composition of the original material, presence of clumping impurities, screening method (wet or dry), uniformity of feed material supply to the screen, condition of the screening surface.*Influence of physical properties of the original material impact on screening results is taken into account using appropriate experimental coefficients when calculating the productivity of screens.

Grading. If the size of the bottom (under-sieve) product is much smaller than the size of the sieve openings, then screening occurs effectively. However, if the content of grains of any particular size, close to the size of the sieve openings, becomes significant, they prevent the fine material from falling down to the surface of the sieve. In this case, sieves with a hole size 20-30% larger than the required size of the under-sieve product are used.

The screening efficiency is maximum when the content of the under-screen product in the initial coal is from 60 to 80%, and then decreases. When the content of the under-screen product in the initial coal is up to 40%, the screening efficiency does not exceed 60-70%, and at a content above 75% it is 90-95%.

*Shape of grains.*A material consisting of grains of lamellar and oblong in shape, less favorable for screening than material made from round and cube-shaped grains.

Live section. As the open cross-section of the sieve increases, its sifting capacity increases. However, this reduces the strength of the sieve.

*Humidity of the initial feed.*Increased humidity of the material leads to small particles sticking together, sticking to small pieces and covering the holes of the whitefish. With an increase in the moisture content of hard coals and anthracites up to 6%*Efficiency*the noise decreases slightly. A sharp decline*Efficiency*and specific productivity occurs at a moisture content of more than 7%. With humidity more than 12% and wet screening*Efficiency* increases to 95%.

Sieve length. To obtain sieving with efficiency up to 95%, the optimal sieve length is in the range of 5.5-6.5*m*.

*Shape of screening surface holes***.**Round holes Compared to other forms of the same nominal size, the under-slat product is smaller. Sieves and sieves with rectangular holes have significant advantages over work surfaces with square and round holes.

- they have a larger open section coefficient, their weight and cost are lower, they have greater productivity, and are less susceptible to clogging when the source material is wet. The possibility of using sieves with rectangular holes is limited by the fact that they cannot be used to obtain classes (grades) of material that are accurate in terms of grain size.

The speed of movement of the material along the screen sieve determines its performance as a transporting device. High speeds should adversely affect the efficiency of the screening process. Due to the complexity of the phenomena occurring on the screen, the optimal speed of movement of the material through the screen is established experimentally when adjusting the screen. In many cases, the speed of material movement is controlled by changing the angle of the screen box.

Amplitude and frequency of oscillations of the vibrating screen box. The amplitude (radius of circular oscillations) and frequency of oscillations affect screening performance and efficiency. As the amplitude and frequency of oscillations increase, the productivity and efficiency of screening increase. When screening large material, larger amplitudes and lower frequencies are assigned, and when screening small material
- smaller amplitudes and higher frequencies.

*Sequence of class identification during screening.*At When screening bulk material with the separation of more than two size classes, the sequence of their separation is determined by the location of the sieves.

The following schemes for classifying classes are distinguished: from large class to small class; from small class to large class; mixed or combined

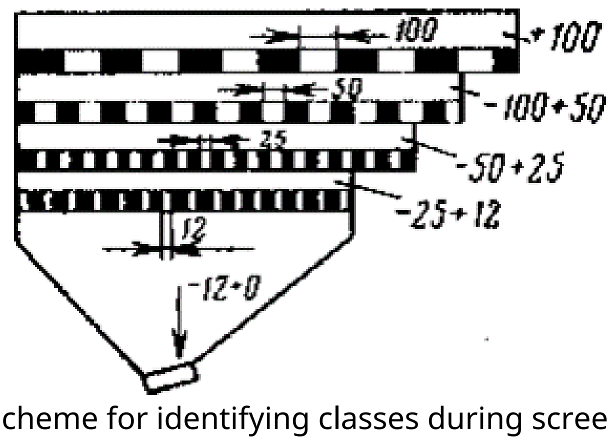


Figure 5.6 - Scheme for identifying classes during screening

When screening from large class to small class, the sieves are placed one below the other (Figure 5.6). The top sieve has the largest holes, and downwards the sizes of the sieve holes decrease.

This sequence of class allocation has the following advantages:

• less wear on sieves, since the entire mass of material and the largest pieces arrive on a work surface with large holes, which is usually assembled from steel sieves and protect surfaces with small holes from wire mesh; higher efficiency of screening of small classes, since less material is supplied to sieves with small holes;

• less crumbling of large pieces during screening, since they are removed from the process faster (especially important for coal);

• compactness of the screening installation in terms of space required due to the multi-tier arrangement of sieves

Control questions

1. What is screening?
2. What is the essence of the screening process?
3. What do you understand by classification scale module?
4. What is the size class?
5. Depending on the purpose, what types of screening are there?
6. How are the average diameter and average size of ore determined?
7. How is sieve analysis performed?
8. Give an explanation of the concepts of “light grains”, “difficult grains”
9. Based on their design, what types of screens are divided into?
10. What types of screening surfaces are used on rattles?
11. What types of sieves do you know?
12. What is the live section of the sieving surface?
13. Explain the operating procedure of the grate screen
14. What are the main parts of a self-balanced screen?
15. What are the main technological parameters What characterizes the screening process?

Lecture No. 6 GRINDING

Purpose and classification of grinding processes. Drum mills. Ball mill with grate. Mills of other designs

Basic terms: grinding, degree of grinding, grinding bodies, mechanical mills, air mills, drum mill, balls, rods, pebbles, end caps, axle, autogenous mill, pebble mills, armor plate, scoop scoop, lifter, axle, rod mills, autogenous mills , vibration mills

6.1 Purpose and classification of grinding processes

Grinding- this is the process of destruction of pieces of minerals under the influence of external forces in a closed volume in order to achieve the required size (less than 5*mm*) or the degree of mineral disclosure. As a rule, material is supplied for grinding after crushing with a particle size of less than 10-25*mm*.

As a result of grinding, a product is obtained that is suitable in size for subsequent enrichment and contains useful minerals in the form of particles, maximally freed from waste rock (Figure 6.1).

The particle size of the crushed product usually does not exceed 1 mm, and often, in order to fully open the aggregates, the ore is crushed to a particle size of less than 0.1*mm*.

*Grinding degree*determined by the ratio of the maximum size grains of the original product$D_{max}$to the maximum grain size of the crushed d_{max}:

$$I_{change} = \underline{\quad ,\quad} \qquad (6.1)$$

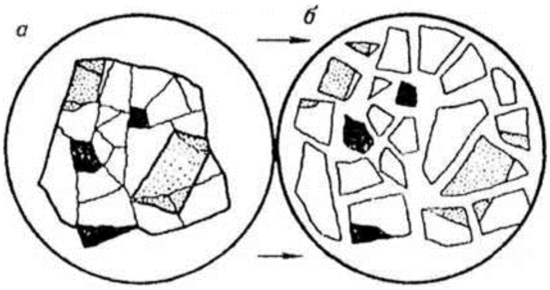


Figure 6.1. Scheme of destruction of a piece of ore during grinding:*A*- before grinding;*b*- after grinding

The grinding degree can be up to 250.

Achieving such a degree of grinding in one step is difficult, therefore grinding, like crushing, is most often carried out in several steps (stages).

Based on the type of implementation of destruction methods, a distinction is made between mechanical grinding with grinding bodies, pneumatic and aerodynamic grinding without grinding bodies.

All grinding machines can be divided into two main groups according to their operating principle: mechanical mills (with grinding media) and aerodynamic mills (jet grinding machines without grinding media).

In turn, mechanical mills, depending on the geometric shape of the working body, are divided into drum, ring, bowl and disk (Figure 6.2).

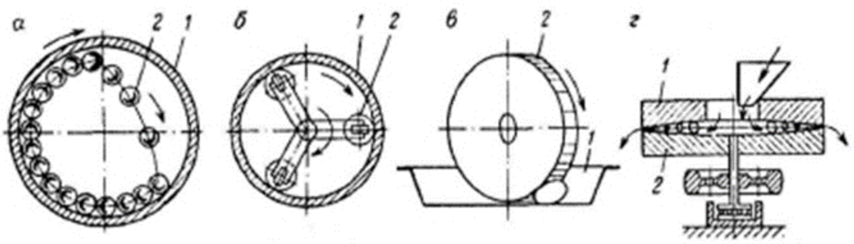


Figure 6.2 - Mechanical mills

The working body of drum mills is an internally lined drum *1*. Grinding media*2*steel balls, rods, pebbles or large pieces of ore (Figure 6.2,*A).*

Ring mills (Figure 6.2,*b*) are sometimes used for dry grinding of easily crushed materials - coal, phosphorites, etc. The working surface here is the inner lining of the ring*1*, and the grinding bodies are rollers*2* cylindrical or other molds that roll the inner surface of the ring.

In a bowl mill (Figure 6.2,*V*) working body - bowl*1*, and the grinding bodies are runners*2*, rolling along the bottom of the bowl.

In a disk mill (Figure 6.2,*G)*main working parts stationary*1*and mobile*2*disks.

Aerodynamic (jet) mills V mining industries are used relatively rarely and exclusively for fine and ultrafine grinding of material. The principle of their operation is that particles of the material are accelerated by counter jets of compressed air, superheated steam or gas and are crushed as a result of collision with each other.

At processing plants and in the ore preparation departments of metallurgical enterprises, rotating drum mills are mainly used.

6.2 - Drum mills

Drum mills are classified into rotary drum, vibratory and centrifugal mills.

The drum mill (Figure 6.2) is a hollow drum*7*, closed with end caps*2* And*3*, in the center of which there are hollow pins*4*And*5*. Trunnions rest on bearings*6*And*7*, and the drum rotates around a horizontal axis.

The drum is filled to approximately half the volume with crushing medium (crushing bodies). As it rotates, the crushing bodies, due to friction, are carried away by its inner surface, rise to a certain height and fall down freely or rolling. Through one hollow pin*4*inside The drum continuously feeds crushed material, which passes along it and, exposed to crushing bodies, is crushed by impact, abrasion and crushing. The crushed product is continuously discharged through another hollow axle*5* . When the drum rotates, the material moves along its axis due to the difference in loading and unloading levels and the pressure of the continuous supply of material; if the grinding is wet, then the material is carried away by the drain flow of water, and if it is dry, by the air flow that occurs when air is sucked out of the drum.

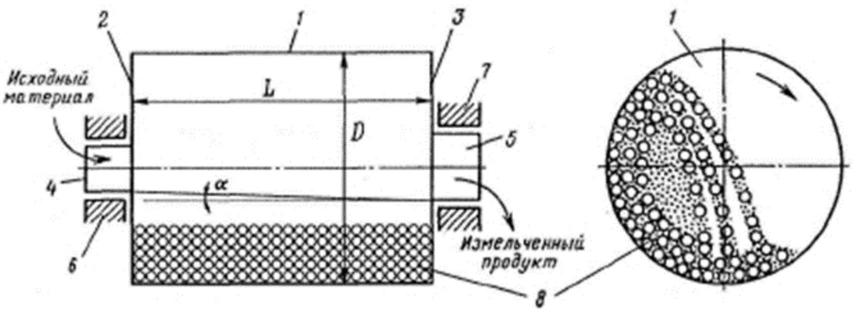


Figure 6.2 - Diagram of the device and principle of operation of the drum (ball) mills:

1- cylindrical drum; 2, 3 - end caps; 4, 5 - loading and unloading pins; 6,7 - support bearings; 8 - grinding medium (balls); a is the angle of inclination of the pulp flow in the mill

Drum mills used for grinding various materials differ in the shape of the drum, the nature of the medium and grinding bodies, the method of unloading the crushed product and the principle of operation.

Depending on the shape of the drum, mills are divided into cylindrical and cylindrical-conical. The first, in turn, are classified into three types: short, long and pipe.*To short* Mills include those whose drum length is less than or equal to its diameter.*To long*- which have a drum length greater than one, but less its three diameters.*To pipe*- mills with a drum length greater than three diameters.

Depending on the type of grinding medium, drum mills are divided into*ball (MS), core (MS), pebble (MG) and ore-pebble (MPG), autogenous (MS) and semi-autogenous (SAG)*.

In ball mills, the crushing medium is represented by steel or cast iron balls, in rod mills - steel rods, in pebble mills - rounded flint pebbles, in autogenous mills - large pieces of crushed ore.

Depending on the method of unloading the crushed product, mills are distinguished*with central unloading (MShTs) and unloading through the grate (MSHR).*In mills with central discharge, removal of crushed material The product flows freely through a hollow discharge pin. Mills with grate unloading have a lifting device that forcibly unloads the crushed product.

The standard sizes of drum mills are determined by the diameter of the internal drum*D*and its working length*L*(drawing6.3). For example, the designation МШЦ-3200x4500 means: ball mill with central discharge with a drum with a diameter of 3200*mm*and length 4500*mm*.

Cylindrical ball and rod mills are used in processing plants to grind wide ores. Pebble mills are used in cases where it is impossible to allow even insignificant iron impurities into the crushed material. Autogenous grinding mills have been introduced into ore grinding practice in recent decades.

6.3 - Ball mill with grate (MSHR)

A ball mill with a grate (BMR) (Figure 6.3) consists of a drum *1*with end caps*2*And*3*, boot*4*and unloading*5*trunnions, resting on bearings*6*And *7*. The rotation of the mill drum is carried out by an electric motor through a small gear*9*mounted on the drive shaft*10*, and ring gear*8*, fixed on the outer surface of the drum.

The mill drum is made of welded or riveted sheet steel, and the end covers are cast from cast iron or steel. They are connected to each other using bolts. The drum and end covers are lined with armor plates to prevent wear.*10*, which are secured with bolts, and the inside of the hollow journals with removable funnels.

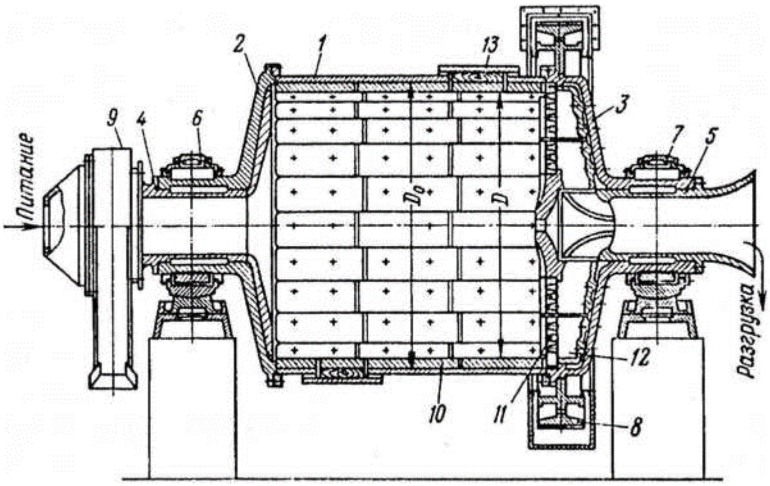


Figure 6.3 - Ball mill with unloading through a grate:
1 - drum; 2 - end cover of the loading end; 3 - end cover of the unloading end; 4 - axle; 5 - unloading funnel; 6, 7 - bearings; 8 - ring gear casing; 9 - combined feeder; 10
- drum lining; 11 - unloading grid; 12 - lifters; 13 – hatch

Steel or cast iron balls of different sizes (from 40 to 150) are loaded into the mill.*mm*) by about half its volume.

The feed material is loaded into the mill by the feeder*9,*and the sands, returning for regrinding from the classifier using a snail scoop. As the mill drum rotates, the balls, rolling, sliding and falling, crush the grains of the mineral. The neck of the unloading axle has a slightly larger diameter, due to which the pulp moves towards unloading.

A grate is installed at the unloading end of the mill*eleven*. The space between this grille and the end cap*3*divided radial partition-lifters*12*to sector cameras opened in axle. The presence of a grid and chamber sectors allows for forced unloading of the crushed product from the mill and maintaining a low pulp level in the mill. When the mill rotates, the lifters*12*raise the pulp to the level of the unloading journal*5*, through which it is removed from the mill.

Ball mills with a grate type MShR are produced with a diameter of 0.9-6 *m*, length 0.9-8*m*and drum working volume 0.45-208m^3. They are usually used in stage I for grinding finely crushed (up to 30-5*mm*) materials in order to obtain a uniform product size of less than 0.15*mm* with a small amount of sludge, and ShBM type mills of size (*DxL*) from 2070x2650 to 3700x8500*mm*They are mainly used for grinding coal into pulverized fuel in a closed cycle with air separators.

The disadvantages of mills with grates include: the complexity of the design of the unloading unit, the high cost of the mill, the possibility of clogging the grate holes with wood chips, worn balls and ore.

6.4 - Mills of other designs

Center Feed Ball Mills(MShTs). By design similar to MShR mills and differ in that they do not have gratings. Pulp unloading occurs by free drainage through a hole in the unloading axle, the diameter of which is slightly larger than the loading axle, in order to create the difference in pulp levels between the axles necessary for unloading. Mills are produced with a diameter of 0.9-6*m*, length 1.8-3.5*m*and drum working volume 0.9-221$m^3$and used for grinding crushed ore (up to30-5 *mm*) and enrichment products up to 0.05*mm*.

*Rod mills (MSM).*The design is similar to a ball mill with central discharge. To more sharply reduce the pulp level in the direction of its movement and increase the speed of material passage, the diameter of the unloading journal of a rod mill is significantly larger than that of a ball mill of the same size. The discharge necks of such mills have a diameter of 1200*mm*and more, which allows them to be used to access the inside of the drum for inspection and repair.

The crushing bodies for mills are high-carbon steel rods with a from diameter of 40 to 100*mm*. The length of the rods is usually 25-50*mm*less than the internal length of the drum. The ratio of drum length to diameter for rod mills is usually 1.4-2.

They are used for relatively coarse grinding of ore (up to 1-2*mm*) before gravitational and magnetic enrichment methods, and are also installed in the first stage before ball mills when grinding ore in several stages.

Ore pebble mills(IWG). For ore pebble grinding Conventional drum mills with discharge grates are used. The grinding medium for such grinding is pieces of ore (galya) with a particle size of 25 to 120*mm*, loaded into the mill instead of balls or rods. The crushed product after medium crushing, from which small pieces are previously removed (by screening), can be used as ore galley. Mills with unloading through a grate type MGR or MSHRGU with a diameter of 4-6*m*, length 6-12.5*m*and working volume 83-320m^3
used for fine grinding of gold-containing, polymetallic, iron ores and enrichment products of other minerals with a particle size of 3-1*mm*, especially in cases where it is necessary to prevent contamination of the crushed material with iron formed as a result of wear of the balls and lining.

*Autogenous mills (AGM).*The essence of the ore process Self-grinding is that the large pieces contained in the ore grind smaller grains of the ore and at the same time grind themselves. During self-grinding, ore with a particle size of up to 500 is loaded into the mill.*mm*, thereby eliminating the need for fine, medium, and sometimes coarse crushing.

In their design, the mills are similar to conventional drum mills: their fundamental difference is only in their large diameter (up to 11-13*m*) with a short length (0.3-0.5 diameter). To lift the crushing medium to a greater height, special plates - lifters - are welded on the inner surface of the drum. The large diameter provides the necessary impact force on the pieces and increases the specific productivity of the mill. Autogenous grinding is carried out dry in Aerofol mills (MCS) and wet in Cascade mills (MMC).

Autogenous grinding mills are designed for fine grinding (up to 0.3-0.07*mm*) large lump material during the processing of copper-molybdenum, iron, gold-containing and other types of ores. Ore semiautogenous grinding differs from ore autogenous grinding in that steel balls of large diameter (100-125 mm) are additionally loaded into the mill.*mm*) in an amount of 6-10% of the mill volume. Balls are added when there is a lack of large pieces in the crushed ore, as well as to increase the productivity of the mill.

Compared to grinding in ball and rod mills, autogenous grinding can dramatically reduce the cost of crushing and grinding operations, increase the degree of disclosure of useful minerals due to preferential destruction along cleavage planes, reduce overgrinding of ore and increase labor productivity.

Vibratory Mills. In vibration mills, the rotational movement of thetype drum is replaced by an oscillatory one. The mill body is equipped with an unbalanced vibration exciter and mounted on springs. Balls or rods made of steel, high-chromium alloy or tungsten carbide are usually used as grinding media.

When the drum vibrates, created by a special vibration exciter, each of its points describes a trajectory close to circular. In this case, the lower layers of the grinding medium adjacent to the drum move upward, and the layers located near the free surface move downward. As a result of such shockless circulation of the grinding bodies, the ore grains located in the spaces between them are abraded. Vibratory mills are used primarily for fine and ultrafine grinding.

Control questions

1. What is grinding?
2. What do you mean by degree of grinding?
3. Tell us about the classification of drum mills
4. What types of grinding media are used in drums?
mills?
5. What are the main parts of a ball mill with grate?
(MSHR)?

6. Explain the working principle of a grate ball mill
(MSHR)
7. What does MShTs-3200x4500 mean? (decipher)
8. In what industries are ore pebbles used?
mills?
9. Why is there a discharge journal neck in ball mills?
has a slightly larger diameter than the loading one?
10. What are the disadvantages of ball mills with grates?
11. What is the fundamental difference between the design of the MSR and the MSC?
12. Tell us the features in the design of a rod mill
13. What is the fundamental difference between the mill design
autogenous grinding from conventional mills?
14. What are the advantages of autogenous mills?
comparison with other species?
15. How does a vibration mill work?

Lecture No. 7 FACTORS AFFECTING THE GRINDING PROCESS. MILL OPERATION

Factors influencing the grinding process. Operation of drum mills. Grinding schemes.

*Key terms:*drum rotation speed, pulp density, degree of grinding, grinding medium, mill lining, cascade mode, waterfall mode, mixed mode, drum speed, ball wear, pulp density, optimal ore size, ball fill factor, circulating load of mills, grinding schemes, open grinding cycle

7.1 Factors affecting the grinding process

The technological and economic efficiency of drum mills is determined by the following main parameters: drum rotation speed; characteristics of the grinding medium and the degree of filling of the drum with it; pulp density during wet grinding; size of the loaded ore and degree of grinding.

*Drum rotation speed.*When the drum mill rotates the grinding medium moves along circular trajectories under the influence of friction forces arising between the mill lining and the surrounding medium bodies, as well as between individual bodies. In order for the ball load not to slide along the lining, the moment of frictional forces relative to the mill axis must balance the moment of the tangential components of gravity forces relative to the same axis (Figure 7.1).

At a certain speed of motion of the ball along a circular path at the point *A* radial force*N*may become equal to the centrifugal force*WITH*(drawing7.1, *A*)**,**the ball becomes free from the point*A*will move under the influence gravity as a body thrown with speed at some angle ∝ to horizon, i.e. along a parabolic trajectory. The angle of separation of the ball from the circular trajectory will be equal to∝ , and at the point*A*the following ratio will be valid:

$$C = N \text{ or } \quad \frac{^{2}}{} = \cos\propto \qquad (7.1)$$

Where*N*= cos ∝;

G- gravity of the body;∝ - ball separation angle;

C= $\frac{2}{g}$; = $-\frac{}{\text{thirty}}$ peripheral speed of the ball in a circular motion trajectories;*g*- acceleration of gravity;*R*- radius of the circular path bodies, *m*;*n*- rotation speed of the mill drum, rpm.

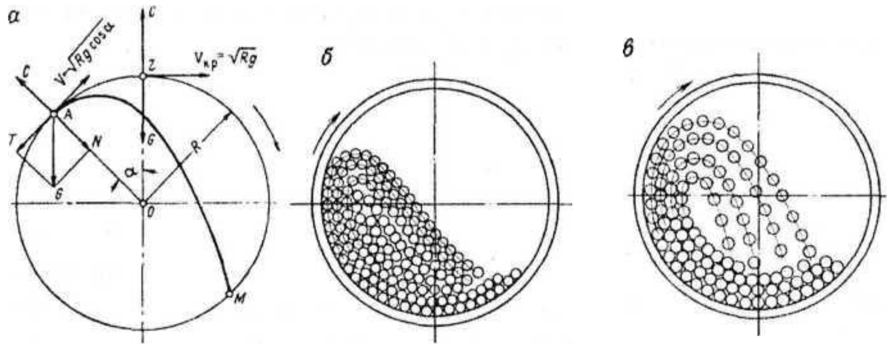


Figure 7.1 - Scheme of the movement of the grinding medium in a drum mill: *A*- diagram of the forces acting on the ball;*b*- cascade mode;*V*- waterfall mode

Substituting the values*N, C*And into expression (7.1) and solving it relative to the drum rotation speed*n*, we get:

$$n = \frac{thirty}{\sqrt{\ }} \sqrt{\cos \propto}. \qquad (7.2)$$

Formula (7.2) determines the drum rotation frequency at which the ball moves from a circular path with a radius*R*to parabolic at angle separation∝.

From formula (7.2) at∝ =0 andcos ∝. =1 we obtain the rotation frequency of the mill drum, at which the ball will not be able to come off its surface and will move along a circular path, i.e. will centrifuge. In this case, the material will not be crushed.

$$n_{cr} = \frac{thirty}{\sqrt{cr}} \qquad (7.3)$$

Speed*ncr*at which they begin to centrifuge bodies, located on the inner surface of the drum are called *critical*.

If we introduce into formula (7.2)n_{cr}then the drum rotation speed will be:

$$n = {}_{cr}\sqrt{\cos \propto_1}. \qquad (7.4)$$

Where$\propto_1$- angle of separation of the body of the outer layer

Depending on the rotation speed of the mill and the filling factor of the drum with the grinding medium, the following operating modes are distinguished:*cascade, waterfall and mixed*.

Cascade mode(drawing7.1,*b*) is carried out at a drum rotation frequency of 50-60% of the critical one. In the cascade mode, the entire load of the mill is turned in the direction of rotation at a certain angle along circular paths, and then rolls or slides down in parallel layers (cascade). Ore grinding occurs mainly crushing and abrasion during cascade movement of the entire load. In the center of the load there is a zone where there is no rolling of bodies.

The cascade mode is used in rod and ball mills for dry grinding, wet autogenous grinding and ore-pebble grinding in the second and subsequent stages.

Waterfall mode(drawing7.1,*V*) is carried out at a drum rotation frequency of 78-85% of the critical one. In the waterfall mode, the load rises along circular paths to a great height and falls like a waterfall along parabolic paths, striking the ore located on the circular paths. Ore grinding occurs mainly by impact of the grinding medium and partly by attrition. The waterfall mode is widely used in wet ball and primary autogenous grinding.

*Mixed mode*characterized by a gradual transition from cascade to waterfall mode at a drum rotation frequency of 60-75% of the critical one. In this case, the outer layers of crushed bodies fall onto the inner layers of the load, sliding or rolling down the slope.

In practice, mills operate at a rotation speed equal to 75-85% of the critical one, i.e. for production conditions, the optimal drum rotation speed is:

$$= (32 \div 36)/\sqrt{\ } \tag{7.5}$$

At this drum rotation speed, the productivity and efficiency of the mill will be maximum.

*Ball mill fill factor*is one of the most important factors affecting the operation of mills, and is determined by the formula:

$$= {}_{w}/ \, , \tag{7.6}$$

Where $_{w}$- the volume of the entire mass of the balls; *V*- useful volume of the mill.

Because:

$$_{w} = \frac{_{w}}{\ } \text{ and } = {}^{2}/4 \; _{w} \tag{7.7}$$

Where $_{w}$| - mass of all balls; $_{w}$- bulk density of balls; *D*- drum inner diameter; *L*- useful length of the drum, then

$$= {}_{w}/ \; = 4 \; _{w}/(\; _{w} \; ^{2} \;) \tag{7.8}$$

For an approximate calculation, take:

for balls

$$_w = 4.8, t/m^3 \quad (7.9)$$

for rods

$$= 6.5, t/m^3 \quad (7.10)$$

The relative rotation speed of the drum and the relative filling of the mill drum with grinding (crushing) bodies are the main parameters of the mechanical mode of a drum mill (ball, rod, ore and ore-pebble).

It has been established that the optimal drum rotation speed depends on the degree of its filling with grinding bodies, which is characterized by the filling factor . For rod mills, coefficient usually equal to 0.3-0.4, for ball mills - 0.4-0.5, autogenous mills 0.35-0.45.

The higher the fill factor , the lower the frequency should be drum rotation.

The relationship between the rotation speed of the drum, its diameter and the filling factor of the drum with grinding media is expressed by the empirical formula:

$$n=8(5 \ + 2)/\sqrt{\ } \quad (7.11)$$

Characteristics of the grinding medium. The maximum size of grinding media loaded into the mill depends on the size of the ore. To grind large and hard materials, grinding bodies (balls, rods) of large sizes are used, and for small and soft materials, smaller sizes are used.

The diameter of the ball D is related to the size d of the ore supplied to grinding by the dependence

$$D=kd \quad (7.12)$$

Where k- coefficient depending on the hardness of the ore (k=13÷32.5).

In practice, a ball load is made up of balls of various diameters, i.e. carry out its rationing. Along with large-diameter balls that crush large pieces of ore, smaller diameter balls are loaded into the mill, which grind smaller pieces of ore. The filling factor of mills with balls should be 0.4-0.5, with rods - 0.3-0.4.

Optimal ball diameter for grinding

Source size material, *mm*	0.2	0.3-0.42	0.6-0.8	1.2-1.7	2.4-3.3	4.7-6.7

Ball diameter, *mm*.	15	20	25	thirty	40	50
Source size material, *mm*	6.8-9.5	13-19	27-38	38-53	53-60	
Ball diameter, *mm*	60	70-80	90-100	100-110	125	

Wear on balls and rods depends on the material from which they are made, hardness, size of the ore being fed and the final product, and other factors. Consumption of balls and rods per 1 *T* ore ranges from: balls - from 0.5 to 2.5 *kg*, rods - from 0.2 to 0.5 *kg*. Resorting of balls, removal of worn ones and reloading of new balls is carried out once every 2-3 *months*. The service life of mill linings ranges from 6 to 15 *months* and its consumption for 1 *T* the original ore is 0.1-0.4 *kg*.

Pulp Density expressed as the ratio of solid to liquid mass (T:L) or the percentage of solid mass in the pulp. The solid content in the pulp when crushed in ball mills is usually 65-75% and in rod mills - 50-60%. With high liquefaction, the pulp passes through the mill faster and a larger grinding product is obtained. High pulp density leads to overgrinding of ore and can cause mill clogging.

Optimal ore size, sent after crushing for grinding, is determined based on the fact that the total costs of crushing and grinding operations 1 *T* ores were minimal. For rod mills this fineness is 15-25 *mm,* for balls - 8-15 *mm*, for autogenous mills - 1/3 of the diameter of the loading pin

7.2 - Operation of drum mills

Both with ball and ballless grinding in grinding shops (Figure 7.2), the initial ore from the hopper is fed by a feeder onto a collection belt conveyor. It is then loaded by an inclined conveyor into a mill operating in a closed cycle with a classifier, hydrocyclone or screen. Classifier sands are usually fed into the mill by gravity. If this is not possible, then sand pumps, screw conveyors, etc. are used.

There are special areas for equipment maintenance. The grinding shop equipment starts up starting from the end of the technological chain, and stops starting from the feeder of the original ore.

The operation of grinding plants is regulated by uniform supply of the original ore to the mill, maintaining the water regime of grinding and the weight of the crushing medium.

The ore size of the initial feed for ball mills should not exceed 25 *mm*, for rod mills - 40 *mm*; for autogenous and ore pebble mills - 1/3 of the diameter of the loading pin.

To obtain maximum mill productivity, the solid content in the crushed product when the initial ore is larger than 12*mm*should be 65-75%, finer than 12*mm*- 50-60%.

The most efficient operation of the mill is ensured when it is loaded with grinding media having a bulk densityt/m^3: ball - 4.5-4.6; rod - 6.6.

The ball filling coefficient should be: for mills with unloading through a grate - 0.45--0.5; mills with central unloading - 0.35-0.4; autogenous mills - 0.35-0.45.

The fill factor of mills with rods is 0.3-0.4.

To ensure stable, high-performance operation of autogenous mills, it is necessary that the source material contains class +350 *mm*within 4-6%; class + 100*mm*- at least 40%.

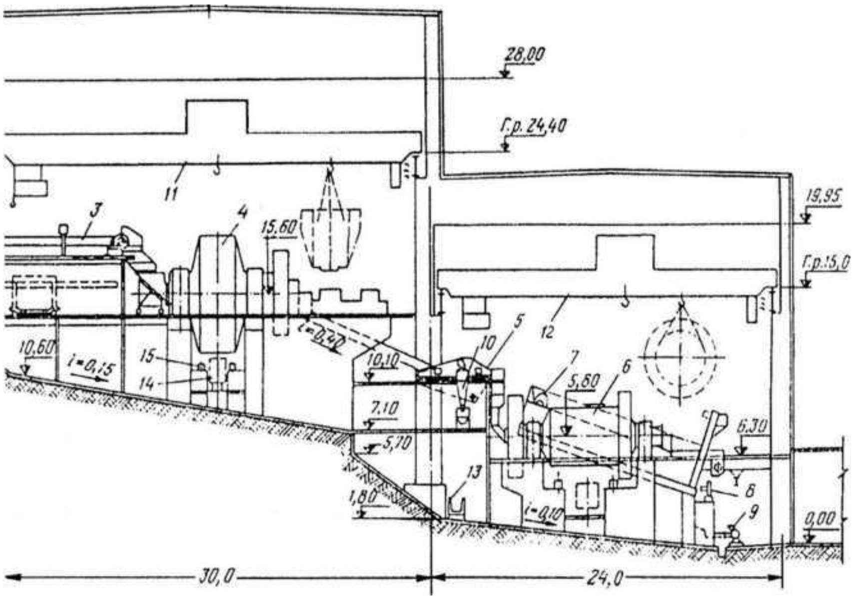


Figure 7.2 - Grinding department with wet autogenous mills with a diameter of 7000 mm of a flotation plant for ores non-ferrous metals:

1- belt conveyor*IN*= 1400*mm*with drum unloading cart; 2 - belt feeder;*3*- belt conveyor*B =*1400*mm*with automatic scales;*4*- autogenous mill7000x2300*mm*,*V*=80 m^3;*5*- self-balanced screen1750x4500*mm*;*6*- ball mill with grate 3600x5000*mm*; 7 - single-spiral classifier*D*=3000*mm*,*L*= 12,500 *mm*;*8*- bottom gate;9 - sand pump;*10*- hydrocyclone*D*=750*mm*; *eleven*- electric overhead crane*Q*=50/10*T*;*12*- electric overhead crane *Q = 5 t*;*13*- pulp hydrotransport tray;*14*- trolley for unloading mills;*15*- jacks for lifting mills

The rotation speed of the mill drum should be, as a percentage of the critical one: rod mills - 60-72; ball with central discharge - 60-86;

ball with grid - 75-86; ore-pebble - 80-85; wet autogenous mills - 70-80.

Optimal specific loading of mills with grinding media, i.e. mass of grinding media per unit volume of the drum, *t/m^3*: balls (with a fill factor of 0.45) - 2-2.07; rods (with a fill factor of 0.35) -2.3-2.35.

Typically, grinding bodies should be located below the mill axis at a distance, mm: balls - 200-250; rods - 100-200.

Balloon consumption per *1 kW·h* energy spent on grinding, is 0.09 *kg*.

Resorting of balls and removal of scrap should be performed at least every 2-3 *months*.

Optimal circulating load of mills, %: ball mills - 300-500; rod - 50-75

The specific circulating load should not exceed 12 *t/(m·h).* The scope of application of rod mills for wet grinding is limited to grinding to a particle size of at least 0.8-1 *mm*; calculation class content -0.074 *mm* in the finished product should not exceed 25-30%.

7.3 Grinding schemes

In the practice of concentrating factories, drum mills operate in an open, closed and partially closed cycle with a classifying apparatus (Figure 7.3).

Open grinding cycle - this is the grinding of material without subsequent application of classification or without returning the large classification product to the grinding equipment (Figure 7.3, *A*). In this cycle, the ground material passes through the mill once and the finished product is obtained directly from the mill.

In modern short mills with an open grinding cycle, the finished product is relatively large, reaching (in largest dimension) 2-3 *mm*. A product of this size can be sent for enrichment using gravitational, electromagnetic and other methods that do not require significant grain fineness.

Closed grinding cycle - this is the grinding of material, carried out with its subsequent classification and return of the large product to the grinding equipment (Figure 7.3, *b*). A closed grinding cycle is used to obtain finely ground (to a particle size of less than 1 *mm*) product before flotation and other enrichment processes. In this case, relatively large sands, after classification, are returned to the mill for additional grinding to the established standard.

A semi-closed or partially closed grinding cycle is used for two-stage grinding.

*Circulating load*that steady-state quantity is called circulating sands, which can be expressed either in absolute value - mass*S*, or relative value *WITH*- the ratio of the mass of sand to the mass of the source material (fresh load*Q*) or to the mass of the finished product (solid in the classifier drain), equal to the mass of the starting material, i.e.

C=/, or = . (7.13)

The relative magnitude of the circulating load is expressed in fractions of a unit or as a percentage. The magnitude of the circulating load depends on the properties of the ore, grinding conditions and classification efficiency. The optimal circulating load of ball mills is 300-500%, for rod mills - 50-75%.

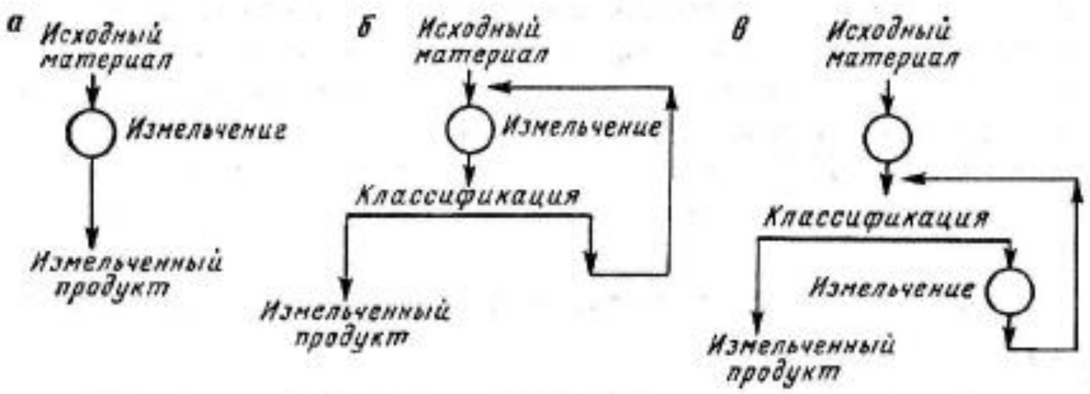


Figure 7.3 - Grinding schemes:

A- single-stage in an open cycle;*b*- single-stage in closed cycle with verification classification;*V*- two-stage with a closed cycle in the second stage

Depending on the number of grinding stages, there are different schemes *single-stage, two-stage and three-stage.*

*Single-stage schemes*grinding is mainly used at processing plants with a capacity of up to 200*t/day*, as well as in factories with a relatively large final grinding product (< 0.2 *mm*). At*two-stage schemes*grinding mills are installed sequentially - one mill (rod) in stage I for larger grinding of the material (in an open cycle) and one or more mills (ball) in stage II for additional grinding, always in a closed cycle with classifiers. The scheme is used in factories with high productivity.

*Three-stage scheme*grinding is rarely used and only when the need for additional grinding after stage II.

Autogenous grinding schemes can be single-stage or two-stage. Single-stage stages are used to obtain relatively large products - 60-70% class -0.074*mm*, and often products typical for grinding in rod mills, i.e. -3 *mm*.

Control questions

1. What parameters determine the effective operation of mills?
2. What formula is used to determine the critical speed of the mill?
3. Depending on rotation speed and duty cycle
drum with grinding medium, what are the operating modes of the mill?
4. What is the cascade mode of mill rotation and how does it happen?
ore grinding?
5. What is the waterfall mode of mill rotation and how does it happen?
ore grinding
6. What is mixed mode mill rotation?
7. What formula is used to determine the optimal rotation speed?
mill drum?
8. What is the relationship between the sizes of the grinding
environment and ore size?
9. What volume of the mill should be filled with balls?
10. What factors determine the wear of balls and mill lining?
11. How does pulp density affect mill performance?
12. How is the workshop equipment started and stopped?
13. What types of grinding schemes are used in processing plants?
factories?
14. Define the concepts of “open grinding cycle” and
"closed grinding cycle"
15. Based on the number of stages, what types of grinding schemes are divided into?

Lecture No. 8 HYDRAULIC CLASSIFICATION

Purpose and principles of hydraulic classification. Theoretical foundations of hydraulic classification. Classifier. Hydrocyclone**.**

Basic terms: speed deposition, hydraulic classification, grain density, medium density, Rittinger formula, Stokes formula, Allen formula, classifier, moving medium, sand fraction, chamber discharge opening, chamber classifier, What are the main parts of the gravitational hydraulic classifier, upward flow, pulp, drain, spiral classifier , receiving pocket, hydrocyclone, drain pipe, sand nozzle, supply pipe.

8.1 Purpose and principles of hydraulic classification

*Hydraulic classification*called the process of separating a mixture small particles of different sizes, shapes and densities into separate classes according to the rate of sedimentation of particles in a water flow.

The purpose of hydraulic classification, like screening, is to obtain classes with a certain range of grain sizes. Hydraulic classification is fundamentally different from screening in that each class obtained by hydraulic classification simultaneously contains large grains of light minerals and small grains of heavy minerals that have the same falling speed in water.

The particle size of the material subjected to hydraulic classification does not exceed 13*mm*for coals and 3-4*mm*for ores

Hydraulic classification can be a preparatory or auxiliary operation, independent. As an independent operation, hydraulic classification is used for washing granular material from clay and silt after the disintegration of manganese, tungsten, rare metal and other ores and placers.

Hydraulic classification can be a preparatory operation if separate enrichment of each class is necessary (for example, by gravitational methods) or an auxiliary operation in ore grinding schemes to separate from the crushed product requiring additional grinding

8.2 Theoretical basis of hydraulic classification

The theoretical basis of hydraulic classification is the patterns of falling of mineral grains in water.

The speed at which particles fall in a medium depends on their size, shape, grain density and density of the medium. Larger particles with high density fall faster than smaller particles with low density. However, the falling speed of a large particle with a high density can decrease significantly if it has a flat shape, since in this case the resistance of the medium increases.

There are two main types of resistance of the medium: dynamic and viscous.

The speed of falling particles during hydraulic classification is influenced by both types of resistance, but the degree of their manifestation when falling different grains is not the same.

When large particles fall at high speed, the *dynamic* resistance, and when small particles fall - *viscous*.

Terminal speed of fall in water $_0$, *m/s*, grains larger than 1 *mm* can be determined using the Rittinger formula:

$$_0 = \sqrt{(\ - 1000)}, \tag{8.1}$$

Where *R* - numerical coefficient (for water *R*=0.16, for air *R*=4.6); *d* - diameter of the spherical grain, *m*; – grain density, kg/m^3.

For grains with a particle size of less than 0.1 *mm* the final speed of fall is determined by: Stokes formula:

$$_0 = \ ^2(\ - 1000), \tag{8.2}$$

Where *S* - numerical coefficient (for water *S*=545, for air *S*=30 278). To determine the final falling speed of grains of intermediate size (0.1-1 *mm*) Allen's formula is applicable:

$$_0 = \ \sqrt[3]{(\ - 1000)^2}, \tag{8.3}$$

Where *A* - numerical coefficient (for water *A*= 1.146, for air *A*= 40.6).

The final falling velocities of spherical grains in water, calculated using formulas (8.1) - (8.3), exceed the actual ones, since all mineral grains entering the hydraulic classification after grinding have a different shape - flat, angular, oblong, round, etc.

8.3 Classifiers

Classifier - an apparatus for separating the starting material into two and more class in size without the use of a screening surface. The separation process carried out is based on the difference in speed constrained fall of large and small particles suspended in a stationary or moving medium. In the classifier, the pulp is divided into two or more products (fractions) of different sizes. When divided into two products, the larger product is called the sand fraction, abbreviated as sand, and the smaller one is called drain. Division into three or more products (fractions) is carried out in multi-product classifiers.

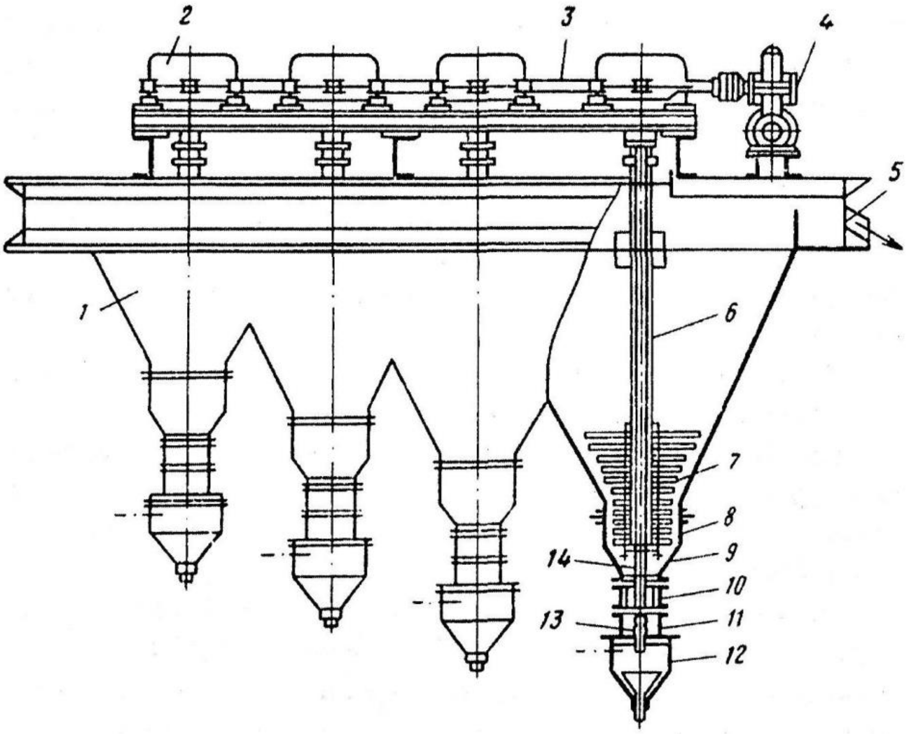
According to the operating principle, classifiers are divided into gravitational, in which separation occurs under the influence of gravitational forces (pyramidal, conical, elevator, scraper), and centrifugal, in which particle separation occurs using centrifugal forces (hydrocyclones, arc screens).

Depending on the design type of the classifier, the movement of the pulp (or added water) can be directed vertically, horizontally, along the axis of the classifier or along a spatial spiral

Gravity hydraulic classifier-classifier, in in which the source material is separated by size in a cylindrical, conical or pyramidal container by settling.

Hydraulic chamber classifiers are widely used for preparatory classification of material before gravity enrichment (for example, concentration on tables). The classifiers consist of four, six, or eight chambers, each of which maintains a different updraft velocity, allowing multiple equi-incidence classes to be obtained.

The hydraulic chamber classifier (Figure 8.1) consists of four chambers 1, increasing in size from the point of loading the source material to the drain threshold 5. The lower part of each chamber consists of a cylindrical zone 8, turning into a truncated cone 9, a classification pipe 10 and a water pipe 11, connected to the water supply network, and a receiver 12 for unloading settled material, which is discharged through an opening that is periodically opened using a ball valve 13.



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By changing the speed of the upward flow, the size of the material discharged from each chamber is regulated. Unloading of settled material from receiver 12 is carried out continuously.

The source pulp is fed into the narrow end of the classifier bath and forms a horizontal flow that merges through the threshold on the wide side of the bath. In the upper part of the bath, the material is classified according to size in a horizontal water flow, the speed of which decreases as the width of the bath increases from the loading end to the unloading end. In accordance with the decrease in the horizontal flow velocity, the size of the deposited grains decreases. The settling material enters pyramidal chambers, where it is continuously loosened by rotating mixers and exposed to rising water currents created by additionally supplied water. The washed sand fractions obtained in each compartment are uniformly unloaded in portions through conical nozzles, and fine grains go down the drain.

Four-section classifier height 2*m*and mirror area 3 m^2has a performance of up to25*t/h*.

Produced by industry hydraulic chamber classifiers of several standard sizes: KG-4, KG-6, KG-8. They have 4, 6, 8 cameras respectively. The width of the chambers (in plan) increases from the place where the material is loaded to the drain threshold.

The length of chamber classifiers, depending on the standard size, ranges from 3.7 to 7.4*m*, height - from 2.8 to 4.2*m*. Productivity on material with particle size less than 2*mm*ranges from 15 to 25*t/h*at water consumption 30-160*l/min*.

Advantages chamber hydraulic classifiers - automatic unloading of settled material using a mechanically rising rod with a ball valve, possibility of adjusting the classification.

Mechanical Spiral Classifier(drawing8.2.) consists of an inclined trough 1, in which one or two rotating shafts 2 with spirals 3 mounted on them are placed.

Separation is carried out in a horizontal flow into a large fraction - sands and a small fraction - drain.

The source material enters the lower part of the classifier through a receiving pocket in the side wall of the trough below the surface of the pulp contained in it. Large particles (sand) settle to the bottom of the trough and are moved by a rotating spiral to the upper part of the classifier to the discharge hole 4. Thin particles in the form of pulp overflow through the drain threshold 5.

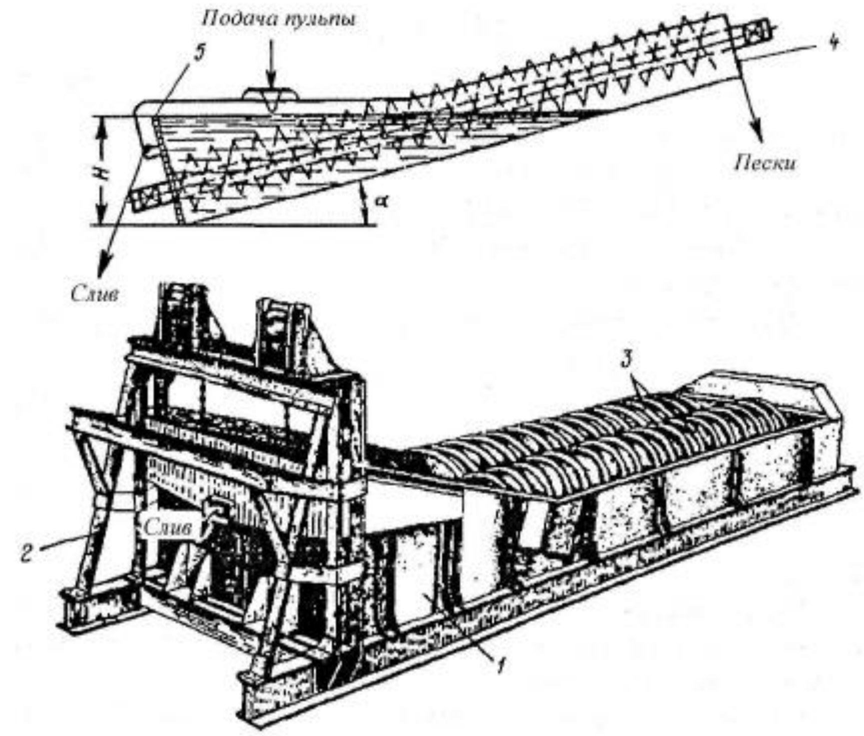

Figure 8.2 - Mechanical spiral classifier

Spiral classifiers are characterized by simplicity of design and maintenance, reliable operation, and high productivity.

The uniform and quiet rotation of the spirals provides good conditions for classifying the material and producing a clean discharge of high density.

*Performance*spiral classifiers is determined by two products: plum and sand. Draining capacity*t/day*, can be determined using the following empirical formulas:

for classifiers with a high threshold:

$$Q=m\ {}_{1}\ {}_{2}(94\ {}_{2}-16\ ) \qquad (8.4)$$

for classifiers with immersed spiral:

$$Q=m\ {}_{1}\ {}_{2}(75\ {}_{2}-10\ ) \qquad (8.5)$$

Where*m*- number of classifier spirals; $_{1}$- coefficient depending on drain size (for classifiers with a high threshold $_{1}$= 0.46+1.95, with immersed spiral $_{1}$=0.36+2.9); $_{2}$- coefficient depending on the density of the drain ($_{2}$=1.9+1);*D*- spiral diameter,*m*.

Sand productivity, t/day, is determined by the formula:

$$Q=135m\ {}_{2}\ {}_{3}n \qquad (8.6)$$

Where*P*- spiral rotation frequency,*min-1*.

Factories produce spiral classifiers with a spiral diameter from 0.3 to 3 *m* and trough length from 2.9 to 15.1 *m*.

8.4 Rules for the operation of mechanical classifiers

Starting the classifier is allowed only with the spiral raised. After starting the spiral, it is smoothly lowered to its normal working position.

It is allowed to supply power to the classifier trough with rotating spirals in the working position.

Putting the classifier into operation is not allowed:

- in the presence of strong vibration of the spiral shaft;
- when the drive mechanism is loosened;
- in case of partial absence of lining plates or their unreliable fastening;
- when the lifting mechanism shackle goes beyond the guide groove;
- without supplying water to the hydraulic seal of the lower support of the spiral.

Before stopping the classifier, it is necessary to stop the supply of ore to the mill with which the classifier is connected, completely exhaust the sand from its trough and raise the spiral to the non-working position.

In case of sudden stops of the classifier for unforeseen reasons, it is necessary to lift the spirals to the upper position in order to avoid their silting.

During the operation of the classifier, maintenance personnel are obliged to monitor the uniform supply of material to the classifier and prevent foreign objects from entering the trough

Hydrocyclones- devices for hydraulic classification finely ground materials in a centrifugal field created as a result of pulp rotation.

A hydrocyclone (Figure 8.3) is an apparatus consisting of 4 cylindrical and 5 conical parts. The internal surface of the device is protected from abrasion by abrasive particles by a lining made of stone (diabase) casting, polyurethane or rubber.

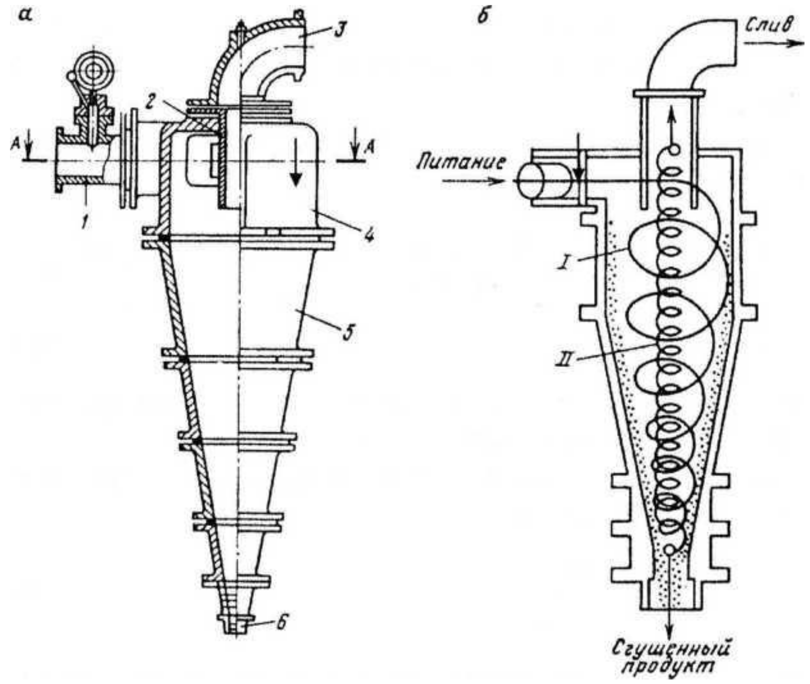

Figure 8.3 - Hydrocyclone (*A*) and the pattern of pulp movement in it*(b*)

The cylindrical part is closed on top by a cover 2, which has a central hole, to the flanges of which a drain pipe 3 is bolted. A sand conical nozzle 6 is also bolted from below to the conical part of the hydrocyclone. The cylindrical part of the hydrocyclone has a pipe 1, through which the source material is supplied under pressure. The pulp pressure at the inlet to the hydrocyclone is controlled by a pressure gauge. The initial pulp enters the hydrocyclone (Figure 8.3) under pressure through the supply pipe 1. Since the supply pipe is located tangentially to the cylindrical part 4 of the housing, the pulp receives rotational motion. The heaviest and largest particles, under the influence of centrifugal force, are thrown towards the walls of the apparatus and move downward in a downward spiral flow to the unloading nozzle 6 for sand. Small particles together with water form an internal flow, which rises upward and is carried out through the drain pipe 3.

The performance of hydrocyclones and the efficiency of material separation depend on many factors, the main ones of which are:*pulp pressure at the entrance to the hydrocyclone; internal dimensions of the sections of the supply pipe, drain and sand nozzles; diameter of the cylindrical part and cone angle of the hydrocyclone; density of the pulp fed into the hydrocyclone; characteristics of the separated material.*

As the diameter of the hydrocyclone increases, its volumetric productivity increases. However, it should be taken into account that the smaller the feed size, the smaller the diameter of the hydrocyclone should be.

As the size of the supply pipe increases, the volumetric productivity of the hydrocyclone increases proportionally. However, when

Increasing volumetric productivity, larger particles are carried away into the drain.

With an increase in the diameter of the outlet (sand) nozzle, the discharge output decreases, the volume of the condensed product increases, the solid content in it decreases, and the entrainment of fine particles through the lower nozzle increases.

The pressure at the inlet to the hydrocyclone affects the performance and quality of separation. To obtain thin drains, the pressure must be at least 150-200*kPa*.At pressure 30-50*kPa*the particle size in the plum increases significantly.

Hydrocyclones for classification and thickening are distinguished by a small cone angle of 10-20°, an elongated conical part and a shortened drain pipe.

The main technological parameters for classification and thickening: solid content in the diet - 10-20%; minimum separation size - 0.01*mm*; supply pressure - up to 0.4*MPa.*

The solid content of the thickened coarse-grained hydrocyclone product can be obtained in a wide range - from 300 to 900*g/l.*

Performance W,m^3/h,hydrocyclone on the original pulp determined by the formula:

$$W=30 \quad \sqrt{\quad} \quad \overline{\quad} \tag{8.7}$$

Whered_n- diameter of the supply pipe,*m*; - diameter of the drain nozzle, *m; N*- pulp pressure at the inlet,*Pa*.

Boundary size h,*m*, classifications in a hydrocyclone are determined by the formula:

$$h= \frac{0.005 \quad \text{With}\sqrt{\quad}}{P\sqrt[4]{\quad}\sqrt{\quad} - \Delta} \tag{8.8}$$

Where P- diameter of the lower (sand) nozzle,*m*;*D*- diameter hydrocyclone,*m*; =100/(*R*+1) - solid content in food %;*p=F:T* – liquid to solid ratio in nutrition*;* and Δ – density of particles and water, respectively, t/ 3.

In beneficiation practice, hydrocyclones are used to separate crushed materials into waste and sand, to desludge products, and to enrich certain types of ores.

To obtain a fine drain at relatively high For productivity, small-diameter hydrocyclone batteries are used. The batteries are either composed of individual hydrocyclones or have a special block-type design.

Control questions

1. What is hydraulic classification?
2. How is hydraulic classification fundamentally different from screening?
3. On what parameters does the rate of falling particles in water depend? environment?
4. Write Rittenger's formula and what does it determine?
5. What is the classifier intended for?
6. Based on the principle of operation, what types of classifiers are divided into?
7. What are the main parts of the gravitational hydraulic classifier?
8. What are the main parts of a spiral classifier?
9. Explain the working principle of the spiral classifier
10. How is the performance of a spiral coil determined? classifier?
11. What is a hydrocyclone?
12. What are the main parts of a hydrocyclone?
13. Explain the operating principle of a hydrocyclone
14. Hydrocyclone performance and separation efficiency What factors does the material depend on?

Section 3. Basic enrichment processes

Lecture No. 9 GRAVITATIONAL ENRICHMENT

General information about gravity enrichment. Jigging process. Jigging machines. Enrichment on concentration tables. Enrichment at the gateways. Enrichment on jet concentrators. Enrichment using screw separators.

*Key terms:*gravitational enrichment, gravity, force resistance, jigging, jigging machine, concentration table, sluice, chute, jet concentrator, screw separator, pneumatic enrichment, heavy medium, deck, segregation, jet chute, flush water.

9.1 General information about gravity enrichment

*Gravity enrichment methods*processes are called in which the separation of mineral particles differing in density, size or shape is due to differences in the nature and speed of their movement in the environment under the influence of gravity and resistance forces.

The following are used as the medium in which gravitational enrichment is carried out: water, heavy liquid or medium (for wet enrichment), air (for pneumatic enrichment).

All minerals can be divided into:

- heavy – density 4-8 to 19 t/m^3 (gold, cerussite $PbCO_3$, galena PbS, cassiterite SnO_2, wolframite $FeMnWO_4$);
- light – density < 2.7 t/m^3 (quartz, feldspar, coal, calcite);
- intermediate density – 2.7-4 t/m^3 (malachite, apatite, limonite). The

main gravitational processes occurring in the aquatic environment are called hydraulic classification - this is enrichment on jigging machines, concentration tables, sluices, chutes, jet concentrators, screw, cone and countercurrent separators. There is enrichment in the air - this is *pneumatic enrichment*, and also enrichment in heavy environments occurs.

9.2 Jigging process. Jigging machines

*Jigging*is the process of separating a mixture of mineral grains into density (difference in the rate of falling of mineral particles) in a water or air environment oscillating in the vertical direction.

Jigging can enrich minerals with a particle size from 50 to 0.25 *mm* for ores, and from 100 to 0.5 *mm*for coals.

The essence of the jigging process is to separate a mixture of mineral grains by density in an aqueous or air environment that oscillates (pulsates) relative to the mixture being separated in the vertical direction.

The initial product, which is a mixture of mineral grains of different densities (Figure 9.1), is fed to a sieve, through the holes of which an ascending and descending flow of water, variable in direction and speed, passes.

In the initial position at zero velocity of the upward flow, the grains of minerals are in a cohesive state. During the period of action of an upward flow moving at a speed *V*, greater than the speed of constrained fall of grains of a given density and size, the material is weighed and it is regrouped into density layers in accordance with the speeds of fall of various grains. During the period of downward flow, a similar process occurs, but the material sinks and becomes compacted. After a certain time, depending on the frequency and amplitude of oscillations of the water flow, the grains are completely separated into density layers; the densest are concentrated in the lower layer (on the sieve of the machine), and the lightest in the upper layer.

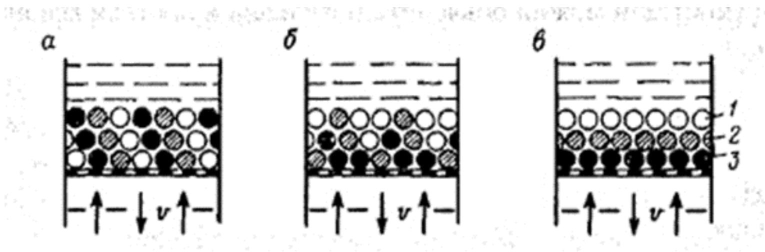

Figure 9.1 - Scheme of stratification of a mixture of mineral grains of different densities in a pulsating stream of water:

*a, b*And *V*- initial, intermediate, final state of the system; 1 - light grains; 2 and 3 - intermediate density and heavy

In real processes, the separation of particles during deposition occurs according to density and size. At the same time, the patterns of movement of mineral particles are very complex and do not have an unambiguous theoretical justification. Since the loosening of the layer during the jigging process is small, the separation of particles can occur both according to the patterns of separation in the suspended layer and according to the patterns of segregation.

A material of intermediate density is used as a bed (natural material - magnetite, hematite, feldspar, galena; artificial material - steel or lead shot, rubber balls, etc.).

The role of the bed is to selectively allow particles of heavy mineral to pass through and retain light grains.

Jig- machine for gravitational enrichment, in in which the source material is separated on a jigging sieve under the influence of vertical vibrations of liquid or air.

The jigging machine (Figure 9.2) consists of two interconnected compartments - the concentration compartment*I*and pulsation departments*II*. In the concentration department there is a sieve 1, on which the minerals are separated. In the pulsation department there is a device that imparts a reciprocating movement to the water with which the machine chamber is filled. The enriched mineral falls onto the sieve along with water, which transports it along the machine, distributing it in an even layer called a bed.

Bed 3 is the entire mass of material, joints and rock located on the sieve.

Through the holes in the sieve, the drive creates ascending and descending water flows that vary in speed and direction. As a result of repeated influences of ascending and descending flows, the bed is stratified: light minerals are carried by ascending flows to the upper layers, and heavy minerals, under the influence of gravity, overcoming the resistance of the environment, are concentrated in the lower layers of the bed. Due to the longitudinal flows of transport water, the bed moves along the machine to the unloading end of the sieve 2, where layer-by-layer unloading of enrichment products occurs.

Small heavy grains are unloaded into the machine chamber through a sieve, larger ones are moved along the sieve and unloaded through a slot at the end of the sieve, and light grains are removed along with the drain.

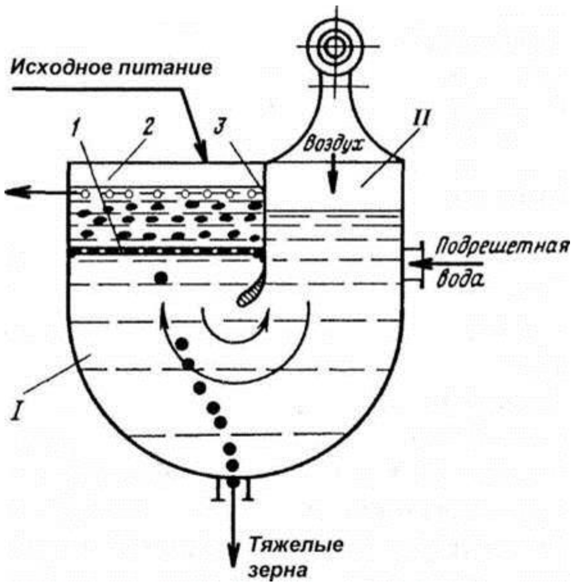

Figure 9.2 - Schematic diagram of a jigging machine

Jigging machines can be used to enrich ore with particle sizes from 0.25 to 50*mm*and coal size from 0.5 to 13*mm*. To increase the efficiency of beneficiation by jigging, the ore is screened into size classes and each class is enriched on a separate jigging machine.

According to the principle of operation of the drive, which ensures water pulsation in the concentration department, jigging machines are divided into piston, diaphragm, air-pulsation (pistonless) and with a moving sieve

9.3 Enrichment on concentration tables

Enrichment on concentration tables- gravitational process enrichment in a thin layer of water flowing along a slightly inclined flat deck, performing reciprocating movements in a horizontal plane perpendicular to the direction of water movement.

The concentration table (Figure 9.3) is a flat trapezoidal surface - a deck with narrow grooves. The decks are made of wood or aluminum and covered with linoleum (polyurethane), etc. The deck is installed at an angle of 1 -10° in the transverse direction and, under the action of the drive, makes asymmetrical reciprocating movements in the horizontal plane.

A thin stream of water is supplied in the transverse direction over the entire area of the deck.

Food in the form of pulp is supplied to the upper corner of the table through the loading tray.

Mineral particles arriving on the table deck are exposed to two main forces: the force of the flushing flow of water directed across the deck, and the inertial force of particles that occurs during the reciprocating motion of the deck and is directed along the deck.

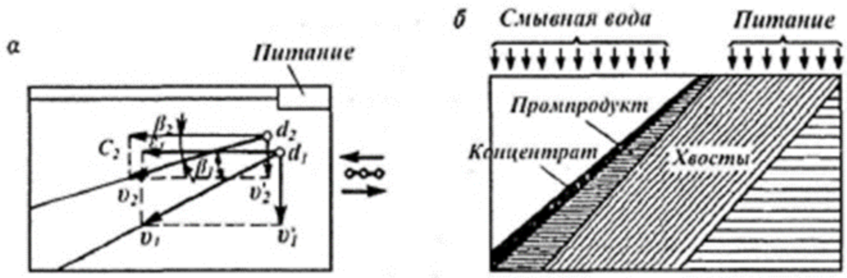

Figure 9.3 - Scheme of movement of grains of the same size, but of different densities*(A)*and distribution of enrichment products on the table deck*(b)*

The process of separating granular material on a concentration table is highly efficient, which depends on the density, size, shape of particles, hydrodynamics of water flow, parameters of the deck movement, segregation phenomena, etc.

Size range particles, effectively shared on concentration tables ranges from 0.04 to 3.0*mm*for ores and from 0.1 to 0 *mm* for coals.

Concentration table- separator for gravity enrichment, in which the separation of mineral particles is carried out in a liquid moving in a thin layer along a deck with grooves that perform directed vibrations.

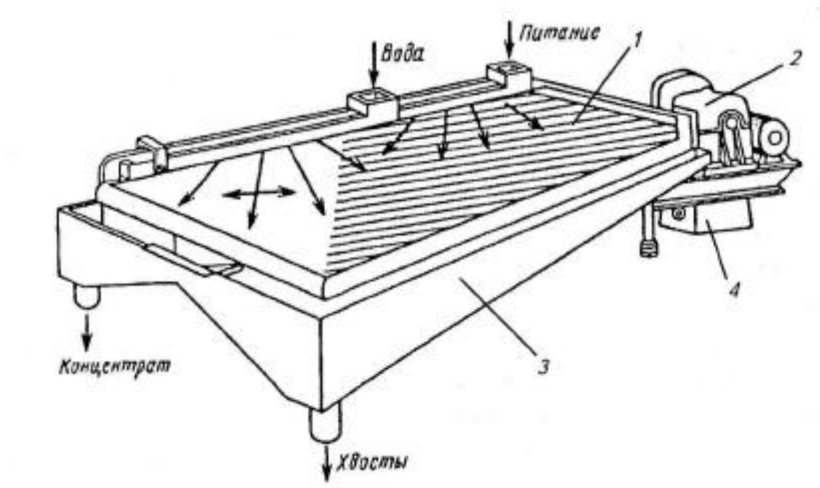

Figure 9.4 - Concentration table

The main components of a concentration table of any design include: deck 1, drive mechanism 2; support device 3 and deck tilt adjustment mechanism 4 (Figure 9.4).

Tables differ in the number and shape of decks, drive design, frequency and amplitude of deck vibrations, groove system and other features.

According to technological characteristics, tables are classified into *sandy* (for beneficiation of ore size -3+0.2*mm*) And*slurry*(for enrichment of product size -0.2 + 0.4*mm)*. Based on their design characteristics, they are distinguished: tables with a rectangular deck - support tables (SKO-1-7.5); tables with a diagonal deck - support (CKO-0.5; SKO-2; SKO-7.5; SKO-15; SKO-22; SKO-ZO; CKO-37; SKO-45). Sand and slurry tables differ in the design of the deck grooves.

Concentration tables are common in primary gravity enrichment schemes for tin, tungsten, and rare metal ores (for size class -3*mm*), as well as in schemes for finishing concentrates and middlings during the enrichment of gold ores, titanium-zirconium sands, etc.

9.4 Enrichment at locks

The gateway is the simplest device for the enrichment of ores with a low content of heavy minerals.*Enrichment at the gateways*- a process of gravitational enrichment in which the separation of particles is carried out in a layer of liquid moving along an inclined chute, the bottom of which has a rough or smooth coating.

A sluice is a separator for gravitational enrichment in which the separation of particles is carried out in a layer of liquid moving along an inclined chute with parallel sides, on the bottom of which stencils or coatings made of rough material (felt, coarse cloth, corrugated rubber, etc.) are laid (Figure 9.5).

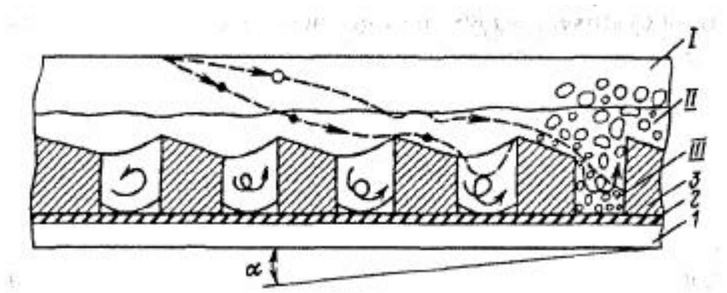

Figure 9.5 - Scheme of grain separation at the concentration gate:

1 - layer of suspended grains; II - layer of primary concentration; III - final concentration layer; 1 - bottom; 2 - checkmate; 3 – stencil

At the locks, placer ores of gold, tungsten, tin and rare metals are enriched.

The pulp, when liquefied L:T not less than (5-6):1, is fed into the head upper part of the sluice prepared for operation.

As the pulp moves through the sluice, the mineral mixture is separated according to density and grain size. First, grains of heavy minerals settle to the bottom of the sluice, which are concentrated between the stencils or are retained by the rough surface of the coating, while larger pebbles and boulders are carried away by the flow and roll along the bottom of the sluice. Along with them, grains of light minerals and sludge are carried away.

The operation of concentration sluices with a collecting coating is characterized by periodicity. As heavy minerals accumulate, the spaces between the stencils are completely filled with them and the power supply to the gateway is stopped. Then the sediment is removed by rinsing the airlock. Rinsing fleecy fabric is done by removing it and washing it in a special tank. The operation of removing sediment is very labor-intensive, and in newer locks the rinsing process is automated. After removing the concentrate, the gateway is again reinforced and put into operation.

The concentrate yield is tenths and hundredths of a percent. When applied to the beneficiation of poor alluvial materials, the sluice is characterized by a high degree of concentration.

*Main technological parameters of gateways*are: solid content in the pulp; flow depth; the angle of inclination of the gateway; type of bottom surface; gateway width. These parameters are selected depending from the properties of the enriched material and determine the ore enrichment indicators -*productivity, recovery and concentrate quality*.

Minimum ratio*F:T*for different slopes of the gateway is characterized by the following data:

Sluice slope	0.05	0.08	0.11	0.15
Minimum possible F:T	15.5	9.4	6.5	4.5

*Water consumption*during enrichment at the locks it fluctuates within a very wide range limits. When enriching fine material and a large slope of the sluice, the water consumption is 3-10m^3on1m^3ore, and during ore enrichment size 200-300 *mm*water consumption increases sharply - up to 100m^3water on1m^3 ore.

The flow depth is determined depending on the size of the enriched material.

*Picking frequency*concentrate ("bed replacement") depends on the content of the captured metal, on the volume of the bed between the stencils and for each specific case is established experimentally.

*Gateway performance*working on small material with a soft catching coating is 0.1-0.3$m^3/(m \cdot h)$,and without special coating - less than 0.1$m^3/(m \cdot h)$.Gateway performance, extracting lighter minerals (cassiterite, wolframite) is reduced by 2-3 times compared to those indicated.

Before enrichment, disintegrating devices and screens are installed at the locks. In some cases, disintegration and screening are combined in one apparatus - a dredge barrel, a scrubber drum.

9.5 Enrichment on jet concentrators

In jet devices, enrichment occurs in an inclined flow under the influence of gravity. Such devices include:

- jet chutes (tapered) and their various installations on a common frame (jet concentrators) (Figure 9.6,*A*);
- cone separators, which consist of one or more cones installed with the base up and representing, as it were, a single circular tapering trench (Figure 9.6,*b*).

The jet chute is short (up to 1000-1200*mm*) a wedge-shaped chute with a flat and smooth bottom. It is installed at an angle of 15-20° to the horizon, with the narrow part down.

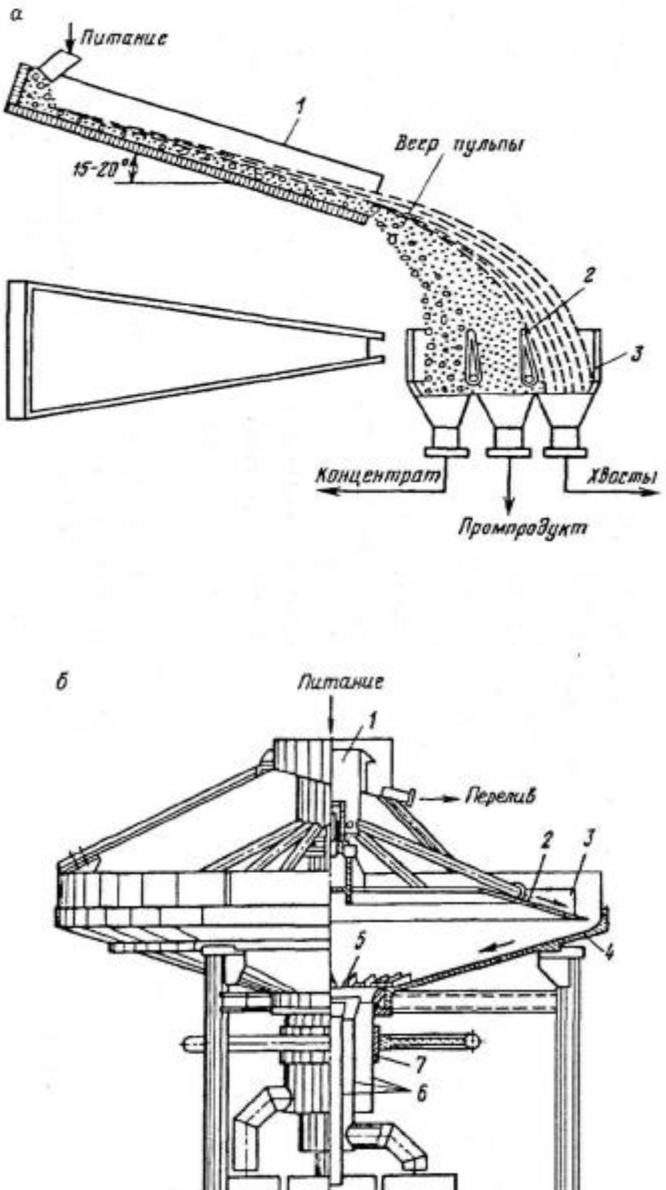

Figure 9.6 - Tapering gutter (*A*) and cone separator VDGMK (*b*):

1 - receiving tank: 2 - distribution cone ring; 3 - stabilizing partition; 4 - cone; 5 - tapering wedges; 6 - pipes for unloading products; 7 - steering wheel; 8 - receiving chutes; 9 – frame

The VDGMK cone separator (one-, two-, three-, five- or six-tier) has a working surface of an inverted cone with a base diameter of 2 or 3 *m* and an apex angle of 140-156°. The initial pulp is fed to the periphery of the cone, and the products are unloaded in the center using concentric pipe flow fan dividers. On the inside

On the surface of the cone, dividing wedges are located radially, forming tapering longitudinal grooves.

Fundamentally, the enrichment process on a jet apparatus comes down to the following:

• fine-grained and de-sludged material is fed to the top the wide part of the apparatus in the form of a dense pulp containing 50-60*%* solid;

• the movement of water along an inclined chute begins at a small speed (about 0.2-0.3*m/s*) and flow depth (1-2*mm*). As it moves down, its speed increases to 1-2*m/s*, and the flow depth is up to 7-12*mm*at the bottom edge of the gutter;

• flowing down an inclined chute, the granular mixture is stratified vertically so that small particles of heavy minerals end up in the bottom region, and larger grains of light minerals remain in the upper zones of the flow. The flow fan is cut at the bend after it leaves the chute using counter-cutting knives located in the receiver; in other cases, the fan is cut with vertical cutters. In jet devices, the concentrate is released continuously.

Currently, Reichert concentrators are produced, consisting of five and even seven cones, and several cones are made double, operating in parallel. The performance of such a concentrator is 80 *t/h*. Their height taking into account the distributors is 13-14*m*, and the mass is about 1.4*T.*

The tiered arrangement of concentration cones allows for a developed scheme of material enrichment (with cleaning of concentrators and control operations for tailings) with gravity transportation of all products.

Jet devices are most widely used in the practice of enriching sands from placers of marine origin. These sands are usually represented by a narrow range of particle sizes, with the heavy minerals in the free state generally being finer than the light minerals of the gangue - the heavy fraction has a particle size of -0.2 (0.4)
- 0.05, and light - after washing and screening -2 (3)*mm*.

When beneficiating ores from primary deposits, jet devices can be used in the primary, coarse beneficiation cycle in order to separate a certain part of the waste tailings at the beginning of the process.

As a result of the primary enrichment of rare metal sands, a collective concentrate is obtained with a yield of 9.0%, a content of total heavy fraction minerals of about 90%, a recovery of heavy fraction minerals of about 85%, and an enrichment efficiency above 80%.

9.6 Enrichment on screw separators

Screw separator- this is a separator for gravitational enrichment, in which the separation of particles is carried out in a liquid moving along a screw chute with a vertical axis (Figure 9.7). The pulp is fed into the upper part of the gutter and, under the influence of gravity, flows downwards in the form of a thin (6-15 *mm*) layer.

When driving along gutter screw separator particles distributed along the flow depth in accordance with their hydraulic size. At the same time, under the influence of circulating water flows, centrifugal and gravitational forces, particles move in the transverse direction: particles of smaller hydraulic size (mainly grains of light minerals) located in the upper layers belong to the outer side, and those located in the lower layers (particles of heavy minerals and large particles of light minerals)) - to the internal.

As a result of the transverse redistribution of grains, separate layers of flow are formed (concentrate, middlings, tailings), which acquire steady motion.

The main redistribution of grains ends after approximately two or three turns, after which the grains move at constant distances from the separator axis.

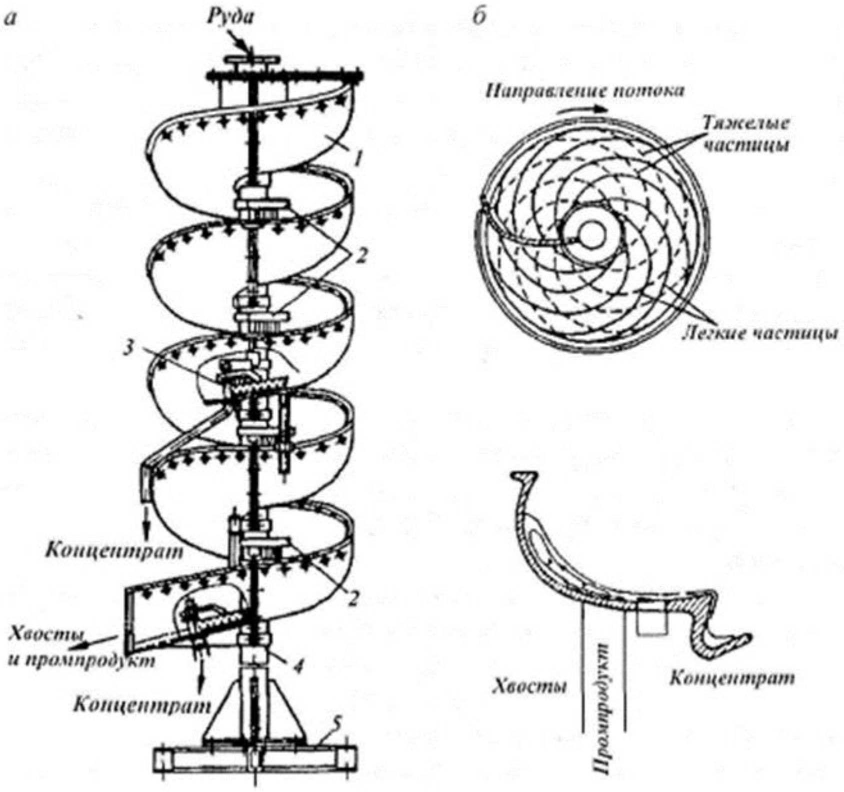

Figure 9.7 - Screw separator (*A*) and cross section of the gutter screw separator (*b*), showing the movement of particles:
1 - screw chute; 2 - water distribution boxes; 3 - cut-off; 4 - support pipe (column); 5 - support frame

Conclusion products enrichment carried out cut-offs, installed on separators of domestic design at the end of the last turn, and on foreign ones - on each turn of the gutter (up to three cut-offs on each turn)

*Technological parameters*operation of screw separators are: size; density and shape of mineral particles in nutrition; preparation of ore before beneficiation; solid content in diet; amount of flush water and productivity.

*Particle Shape*determined by the sphericity coefficient, with an increase in which the extraction of particles into the concentrate on a screw separator decreases. The most advantageous cases for beneficiation are those where waste rock particles have a high sphericity coefficient (rounded grains) and useful mineral particles have a low coefficient of sphericity (flat grains).

*Solid content in diet*when enriching placer sands usually maintained between 15-25% (by weight). When enriching primary ores, for example iron ores, it is higher and amounts to 30-10% solids in the main separation operation. When the pulp density exceeds the specified limits, its viscosity increases, as a result of which the release of heavy mineral particles slows down. The minimum acceptable solid content in food is 6-8%.

*Flush water consumption*significantly affects the degree of concentration material. Typically, flush water consumption for one gutter with a diameter of 600 *mm*amounts to0.3-0.6*l/s.*With excessive flow of wash water, particles of heavy minerals are carried into the main flow of the pulp; with low flow, particles of waste rock enter the concentrate.

*Performance Q*screw cages depends on the diameter spiral, number of gutters, material composition and size of the enriched raw material and is, for example, for a single-trough separator with a diameter of 600*mm*when enriching iron ores with a particle size of -0.3 (0.2)÷0*mm*about 1-1.4*t/h*.

Preparation of material before beneficiation consists of its classification (or screening) into size classes and desliming.

Screw separators are manufactured with an adjustable And unregulated pitch of turns.

The main design parameters affecting the operation of screw cages include:*diameter of the screw chute, its profile cross-section, number of turns, pitch of the screw chute, number of cut-offs and location of their installation.*

The simplicity of the device, the absence of a mechanical drive, high specific productivity, small area occupied by them and high reliability of operation have ensured that screw separators are widely used in the enrichment of titanium and titanium-zircon sands, gold placers, tin and tungsten ores.

Control questions

1. What is the gravity enrichment method?
2. What types of gravitational enrichment are there?
3. Which separation process is called jigging?
4. What is the essence of the jigging process?
5. What are the main parts of a jigging machine?
6. How does enrichment occur on the concentration table?
7. What are the main parts of a concentration table?
8. How are minerals separated in sluices?
9. What parameters does the gateway efficiency depend on?
10. How does enrichment occur in a jet concentrator?
11. What are the main parts of a screw separator?
12. How the enrichment products are removed from the screw separator?
13. What factors affect the operation of a screw separator?

Lecture No. 10 MAGNETIC ENRICHMENT

Are common intelligence Andclassification magnetic enrichment. Theoretical foundations of magnetic enrichment. Magnetic separators

*Key terms:*magnetic separation, magnetic field, magnetic product, non-magnetic fraction, permanent magnet, electromagnet, magnetic field strength, uniform magnetic field, inhomogeneous magnetic field, field gradient, magnetic susceptibility, magnetic induction, magnetic permeability, magnetic product.

10.1 General information and classification of magnetic enrichment

*Magnetic enrichment*is an enrichment in a magnetic field based on the difference in the magnetic properties of the separated components. The starting material for magnetic enrichment is a mechanical mixture of magnetic and non-magnetic bodies, which is separated into magnetic and non-magnetic products in an air or water environment.

Magnetic enrichment processes(magnetic separation) is processes of separation of mineral particles based on the difference in the magnetic properties of the separated components, carried out by changing the trajectory of the particles in a magnetic field.

Magnetic enrichment is carried out in magnetic separators, a characteristic feature of which is the presence of a magnetic field in their working area. When material moves through the working area of the separator under the influence of magnetic force of attraction $F_{magician}$minerals with different magnetic properties move along different trajectories, which allows magnetic minerals to be separated into a separate - magnetic - product, and non-magnetic - into a non-magnetic one (Figure 10.1).

In addition to the magnetic force, material particles moving through the working area of the separator are affected by a number of mechanical forcesF_{fur}- strength gravity and environmental resistance, molecular cohesion force, centrifugal force. The separation of materials by magnetic properties is carried out effectively in the case when the magnetic properties of minerals differ significantly from each other, and the magnetic forces acting on magnetic particles significantly prevail over the resistance forces.

The process of magnetic enrichment is characterized by the complexity of the phenomena of mass transfer, flocculation, adhesion, etc., which are caused by the participation in it of a large number of particles of the separated material, of different sizes with different physical properties.

A) *b)* *V)*

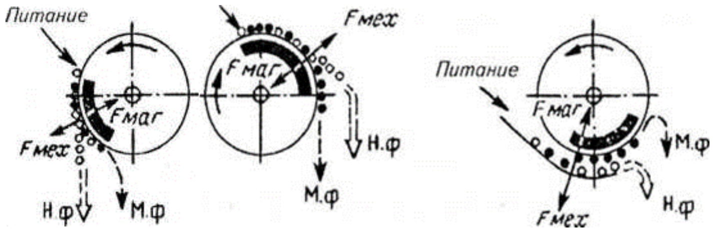

Figure 10.1 - Schemes for separating particles by magnetic properties:*a-c*-respectively deflection, retention and extraction of magnetic particles;
M.f. - magnetic fraction; N.f. - non-magnetic fraction

There are three known methods for separating particles based on magnetic properties:

- *deflection of magnetic particles*, at which the total flow of material, passing by the magnet is divided into two (Figure 10.1,*A).*The speed of particle movement, and therefore the productivity at a given magnetic force, can be quite high, but the separation efficiency is not high enough;

- *retention of more magnetic particles*in the direction of general flow perpendicular to the surface of the drum (Figure 10.1,*b*). The directions of particle gravity and magnetic force coincide, which ensures high recovery of magnetic particles;

- *extraction of more magnetic particles*from the stream as it passes under a magnet (Figure 10.1,*V*), while the magnetic fraction is of higher quality, but with a lower extraction of magnetic particles.

The deflection separation method is used for easily separated mixtures of granular materials with a large difference in magnetic properties. The angle between the directions of movement of the magnetic and non-magnetic fractions is called the fan opening angle. The results of mineral separation are greatly influenced by changes in the position of the divider relative to the fan.

To ensure stability of the separation process, it is necessary to increase the fan opening angle.

When particles are separated by the retention method, fan expansion ($\propto \geq 90°$) is achieved by changing the direction of the magnetic force and using the curvilinear movement of magnetic particles, and when separating by extraction, the countercurrent movement of the particles.

Magnetic enrichment processes are used in the processing of ferrous, rare and non-ferrous ores. Magnetic enrichment is most widely used for the enrichment of iron-containing magnetite ores.

10.2 Theoretical basis of magnetic enrichment

The separation of minerals by magnetic properties is carried out in magnetic fields created by permanent magnets or electromagnets.

The main force characteristic of a magnetic field is intensity. *Magnetic field strength H*is the force with which the field acts on a unit of positive magnetic mass placed at a given point in the field. The SI unit of magnetic field strength is ampere per meter (*Vehicle*).

According to the nature of the change in intensity, magnetic fields are divided into homogeneous and inhomogeneous. IN*uniform magnetic field* tension is the same in magnitude and direction, in*heterogeneous*- tension is not constant in magnitude and can change in direction (Figure 10.2). A uniform magnetic field arises between two unlike poles of a flat shape. Inhomogeneous magnetic fields are created between curved and angular poles. In a uniform field, a magnetic moment acts on a magnetic particle, under the influence of which the particle is oriented along the field lines. In a non-uniform field, in addition to the magnetic moment, a magnetic particle is acted upon by a magnetic force (attraction or repulsion) in the direction of increasing the magnetic field strength. The action of this force is responsible for the separation of magnetic and non-magnetic minerals.

*Inhomogeneous magnetic field*characterized by a field gradient (*grad N*), i.e. intensity of change in magnetic field strength:

$$gradH=—, \qquad 10.1)$$

Where*dx*- the distance at which the field strength changes by size*dH* .

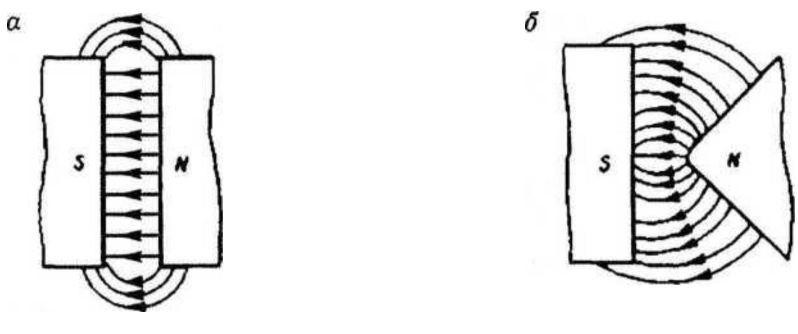

Figure 10.2 - Scheme of a homogeneous*(A*) and heterogeneous (*b*) magnetic fields

Magnetic properties minerals are characterized magnetic susceptibility and magnetic permeability.

Magnetic susceptibility- physical quantity characterizing the ability of a body to change the intensity of its own magnetization. There are volumetric and specific magnetic susceptibility. Volume magnetic susceptibility æ is equal to the ratio body magnetization *I* to magnetic field strength *N*, in which the body is located:

$$æ = ,–$$ (10.2)

Specific magnetic susceptibility - it's magnetic susceptibility per unit body mass:

$$=æ, \ –$$ (10.3)

Where - body density, *kg/m3*

Depending on the sign of magnetic susceptibility, all bodies are divided into diamagnetic and paramagnetic. Diamagnetic bodies, when introduced into a magnetic field under the influence of magnetic forces, are pushed into areas with lower field strength (æ > 0), paramagnetic bodies are drawn into areas with higher strength (æ < 0); in vacuum æ = 0.

Magnetic permeability - a value characterizing the ability substances change their magnetic induction *IN* under the influence of an external magnetic field:

$$= \ —$$ (10.4)

Where *IN* - magnetic induction, characterizing the magnetic field strength in a substance. Magnetic induction depends on the strength of the external magnetic field *N* and on the magnetization of the substance *I*:

$$B=H+4 \ ,$$ (10.5)

Magnetic permeability is related to magnetic susceptibility by the following relationship:

$$= 1 + 4 æ$$ (10.6)

For vacuum =1, for paramagnetic substances >1, for diamagnetic <1.

A mineral grain placed in a non-uniform magnetic field will be subject to a magnetic force $F_{magician}$, determined by the formula:

$$F_{magician}=m \ ,$$ (10.7)

Where - specific magnetic susceptibility of grain; *m* - grain weight; - magnetic field strength; - field gradient.

body magnetization*I*to magnetic field strength*N*, in which the body is located:

$$æ = ,– \qquad (10.2)$$

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$$F_{magician}=m \qquad , \qquad (10.7)$$

Where - specific magnetic susceptibility of grain;*m*- grain weight; - magnetic field strength; - field gradient.

Work is called magnetic field strength F_P. The higher the specific magnetic susceptibility, the more forceful the magnetic field affects the mineral grain, all other things being equal. Mineral grains for which the magnetic force is greater than the sum of the opposing mechanical forces (gravity, inertia, centrifugal, environmental resistance, etc.) will be attracted to the poles of the magnetic system of the separator and extracted into the magnetic product. Mineral grains with low magnetic susceptibility practically do not change their magnetization, do not interact with an external magnetic field, and move in a magnetic field along a trajectory that depends only on mechanical forces. These mineral grains are released into a non-magnetic product.

Separation of mineral grains in the magnetic field of the separator is possible subject to the following inequality:

$$F_{magician} > \text{fur} \qquad (10.8)$$

Where fur- the resultant of all mechanical forces attributed to unit of grain mass and directed opposite to the action of specific magnetic forces.

Currently, magnetic enrichment in a constant inhomogeneous magnetic field, based on the difference in the magnetic susceptibility of the separated minerals, is most widely used.

In a non-uniform magnetic field, magnetic particles are attracted to the pole in the direction of convergence of the magnetic field lines, i.e. are drawn into areas with higher field strength, and particles of non-magnetic or diamagnetic minerals, on the contrary, will be pushed out under the influence of magnetic forces into areas with lower field strength. This ensures fairly effective separation of particles of magnetic and non-magnetic minerals in the working area of the separator.

In a uniform magnetic field, in which the intensity is the same in both magnitude and direction, the mineral particles will be subjected only to the influence of a torque that orients them parallel to the current lines of force. However, the particles will not move to the poles of the magnetic system. Therefore, only non-uniform magnetic fields are used in magnetic separators.

10.3 Magnetic separators

Magnetic (electromagnetic) separator- this is a separator for magnetic enrichment, in which the source material is divided into components according to the difference in magnetic susceptibility in a constant field magnets (or electromagnets).

Any magnetic separator consists of the following main structural units: magnetic system; a feeder for supplying ore to the working area of the separator; devices for transporting a magnetic product from the area of magnetic forces; drive and casing or bath. The design of individual components and the operating mode of various types of separators are characterized by great diversity.

The separation of minerals is carried out in the working area of magnetic separators. With the top feed, the source material goes directly to the working body - drum, roller, disk, etc., with the bottom feed - into the gap between it and the feed tray, the bottom of the bath or the pole piece. Magnetic particles, under the influence of a magnetic field, are attracted to the working element and are carried beyond the action of magnetic forces, where they are unloaded into receivers for the magnetic product. Non-magnetic particles slide under the influence of centrifugal forces and gravity along the surface of the working element, pole piece, tray or bath bottom and are discharged into non-magnetic product receptacles.

Depending on the design features of individual units, as well as technological and target characteristics, magnetic separators are usually classified into the following groups.

Depending on the purpose of the separator and the magnetic field strength, all magnetic separators are divided as follows:

- *separators with weak magnetic*field (magnetic strength fields from 70 to 120*kA/m*) - for isolating highly magnetic minerals from ores;
- *separators with strong magnetic field*(magnetic intensity fields from 800 to 1600*kA/m*) - for isolating weakly magnetic minerals from ores.

Depending on the nature of the environment in which the separation of minerals directly occurs, all magnetic separators are classified into dry ones for the enrichment of minerals in the air and wet ones for enrichment in the aquatic environment. For the enrichment of minerals with particle size from 3 to 50-100*mm*used dry, finer 3(6)*mm*- usually wet magnetic separation.

Depending on the direction of movement of products relative to each other, separators with direct-flow, countercurrent (C) and semi-countercurrent (SC) baths are distinguished.

According to the design of the main working body and the type of environment in which the separation occurs, separators are divided into: drum for wet separation (BM), drum for dry separation (DS), roller for wet separation (VM), roller for dry separation (DS) , disk for dry separation (DS).

10.3.1 Separators with weak magnetic field

*Wet magnetic enrichment.*For wet enrichment, the particle size material should not exceed 6 mm. Currently in wet practice

For magnetic enrichment of highly magnetic ores, mainly drum separators of the PBM type are used, which have a multi-pole system of permanent magnets (Figure 10.4).

The separator has a drum 1 with a six-pole magnetic system 2 made of permanent magnets (UNDK-24 alloy), a bath 4, a loading box 5, an overflow box for flush water 3. The outer surface of the drum is covered with rubber

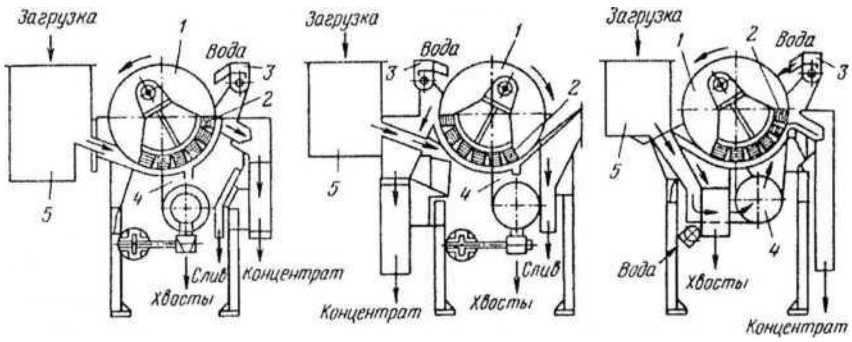

Figure - 10.4. Drum separator PBM-90/250 for wet enrichment ore: *1*- with direct-flow bath; *2*- with a counterflow bath; *3*- With semi-counterflow bath

The separator drive is mounted inside the drum, which makes it easier to replace the latter and increases its service life.

The separator PBM-90/250 is available in three versions: with direct-flow, counter-flow and semi-counter-flow baths.

The separator works as follows. The pulp is fed under a rotating drum and moves through the working area along a curved path. Magnetic minerals in the zone of action of the magnetic system are attracted to the drum and carried into the concentrate compartment of the bath. At the unloading point, the concentrate is washed off from the drum with water.

Non-magnetic minerals, having passed through the working area, are unloaded into the tail compartment of the bath. Products are removed from the separator through outlets with nozzles, the diameter of which is selected depending on the size of the feed and the productivity of the separator. The magnetic field strength on the surface of the drum of these separators is 90-100*kA/m*, at a distance of 50*mm*from the drum surface - 40-50 *kA/m*, the productivity of the separator depends on the type of bath, the properties of the raw materials and reaches 40-200*t/h*

*Dry magnetic enrichment.*For dry enrichment of highly magnetic materials ores up to 50 fines*mm*In order to separate dump tailings, one-, three- and four-drum separators with magnetic systems with permanent magnets (type PBS and PBSTs - with centrifugal unloading) and electromagnets (type EBS) powered by direct current are used.

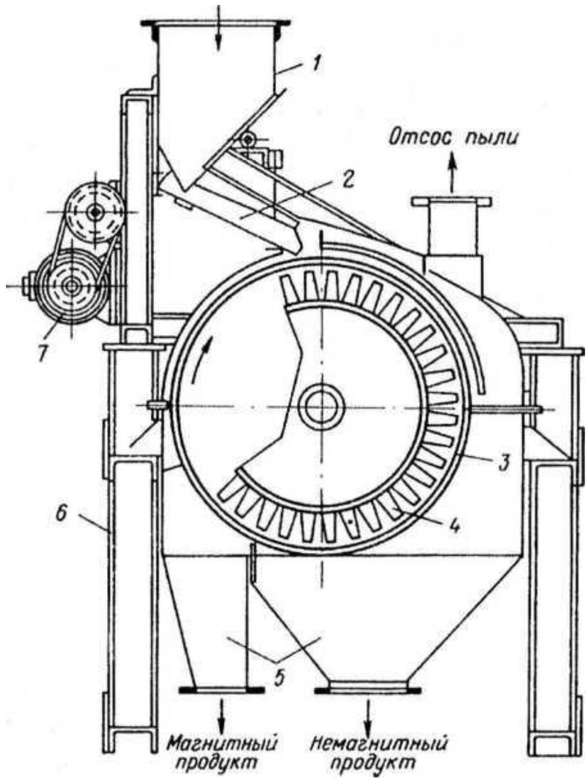

Figure 10.5 - Drum separator PBSTs-63/50 for dry enrichment ore

For dry separation small highly magnetic material separators of the PBSTs-63/50 (20SB-SE) type are used (Figure 10.5).

The drum shell 3 of the separator is made of non-magnetic stainless steel with a thickness of 1.2-2*mm,*The permanent magnets of the fixed magnetic system 4 are made of UNDK-24 alloy. The polarity of the poles alternates around the perimeter of the drum. Poles are set in 50 increments*mm*. The magnetic field strength at the surface of the drum is: against the middle of the poles - 115-125*kN/m*, against the gap between the poles - 84-92*kA/m*.

The separator works as follows. The original ore from hopper 1 is fed into the upper part of the drum using a vibrating tray 2 with a drive 7. The magnetic fraction is attracted to the surface of the drum and is unloaded into the hopper 5 for the magnetic product at the moment when the drum section leaves the zone of action of the magnetic system. The non-magnetic fraction is transported by the drum and unloaded into the non-magnetic product hopper. All separator components are mounted on frame 6.

10.3.2 Separators with strong magnetic fields

*Wet magnetic enrichment***.**Upper limit of ore size and material enriched by magnetic wet or dry method, 6*mm*. The separators use electromagnetic systems with a field strength of 40-144*kA/m*. This process is carried out mainly on roller separators of various designs operating in the extraction mode of magnetic minerals (bottom feed).

Figure 10.6 shows the basic design of a wet roll separator with two rolls operating in parallel, located at the same level on both sides of the magnetic system.

The two-roll electromagnetic separator 2EVM-30/100 (ERM-1) (Figure 10.6) consists of two rolls 4, four pole pieces 5, two cores with excitation windings 3, a loading device 1, right and left receiving baths 8 and 9.

The initial product from hopper 1 along tray 2 along with water is fed into the gap between roller 4 and pole piece 5 of the magnetic system. Grains of highly magnetic minerals, under the influence of magnetic forces, are attracted to the surface of the rotating rolls, and then washed off with water into the receiver 9 for the magnetic product. Non-magnetic grains under the influence of gravity through slot-like gaps in the pole pieces are unloaded into the receiver 8 for the non-magnetic product.

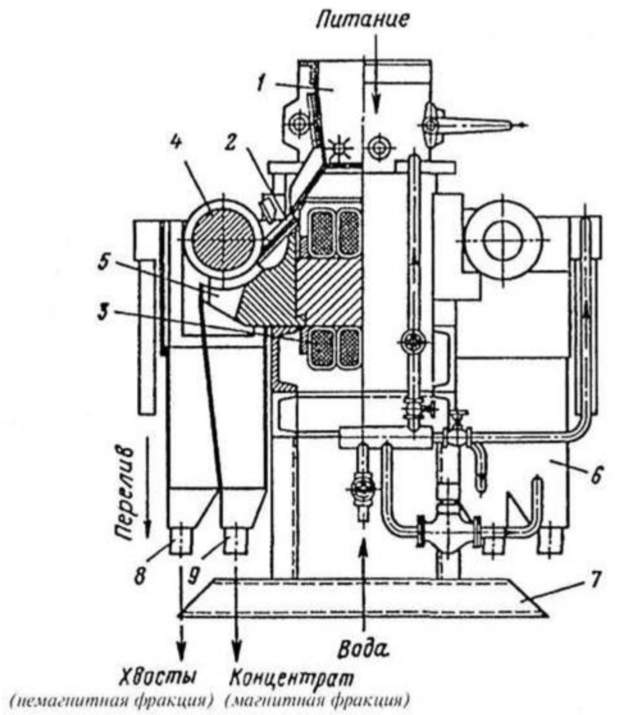

Figure 10.6 - Electromagnetic roll separator for weakly magnetic materials ore: 1-hopper for ore; 2-tray; 3-winding; electromagnet; 4-rolls; 5-pole pieces; 6-casing; 7-support frame; 8-receiver for non-magnetic product; 9-receiver for magnetic product.

*Dry magnetic enrichment.*For dry enrichment of rare metals and other weakly magnetic ores, separators of the following types are used: 2EVS-36/100, EVS-36/100, 2EDS-60/40. To extract ferrous impurities from glass, ceramic and abrasive raw materials, separators such as 6EVS-V-10/80, 2EVS-15/80, EVS-V-15/80 and some others are used.

10.3.3. High Gradient Separators

High-gradient separators differ from conventional drum magnetic separators by the presence in their working space of small carrier magnets (polygradient medium), in the gaps between which strong magnetic fields are induced. Balls of small diameters and drill shot (d= 6) are used as a polygradient medium÷8*mm*) and other small iron materials.

A feature of polygradient media is that, due to their small size, neighboring balls touch almost at a point. Therefore, even with a low field strength in the working space, magnetic saturation occurs at these points, and the adjacent areas are characterized by a very high volumetric gradient and, therefore, high field strength. In areas with high magnetic field strength, there is intense attraction and retention of finely divided weakly magnetic particles, while non-magnetic particles are filtered through the holes between the balls.

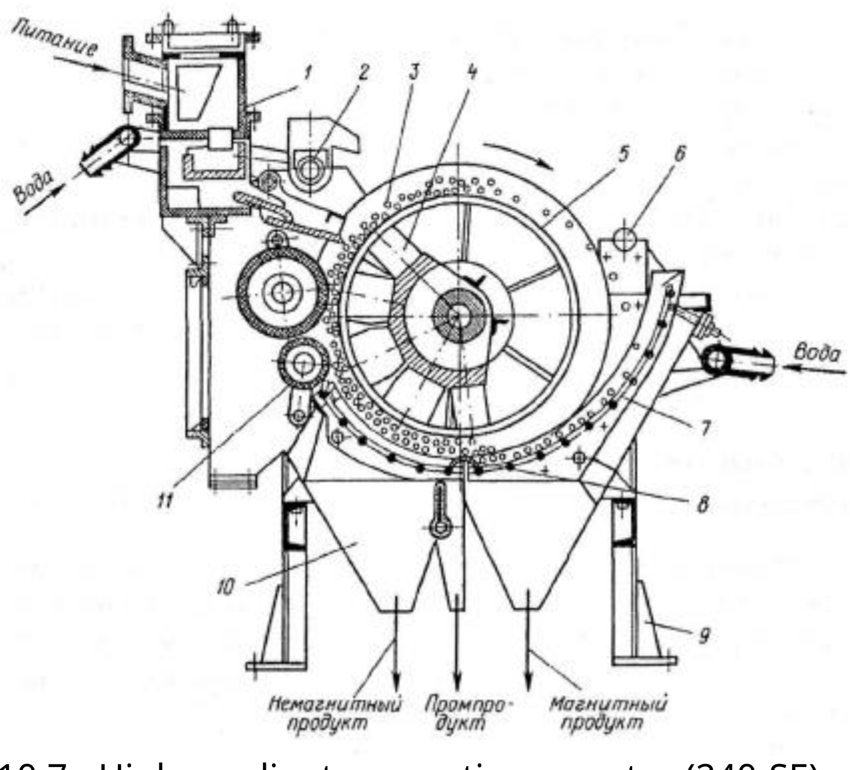

Figure 10.7 - High-gradient magnetic separator (240-SE):
1 - feeder; 2.6 - splashed; 3 - steel balls; 4 - magnetic system; 5 - drum; 7 - arc-shaped sieve; 8 - threshold; 9 - frame; 10 - bath; 11 - pressure rollers

The design and operating principle of high-gradient separators are presented in Figure 10.7. Inside the drum 5 of the separator, a five-pole magnetic system 4 is fixedly fixed, the working space of the separator is filled with steel balls 3. Between the bath 10 and the drum there is an arc-shaped sieve 7, which is covered by a threshold 8 in the middle part under the drum. The separator is equipped with a feeder 1, pressure rollers 11, splashers 2 and 6. All separator components are mounted on a frame 9.

The source material from the feeder in the form of pulp is fed to a layer of balls, which is held on the drum by the field of the magnetic system. Non-magnetic particles pass through the layer of balls and enter the tail section of the bath. Magnetic particles, held by the magnetic force in the channels between the balls, rise with them to the upper part of the drum, where the final washing of non-magnetic particles with water from sprayer 2 is carried out. The balls, together with magnetic particles, are transported by the drum and fall onto a sieve, where magnetic particles are washed with water from splashed 6.

Magnetic particles fall into the concentrate and partially into the industrial product compartments of the bath, and the balls, having passed through the threshold, again enter the zone of action of the magnetic field. Then the described cycle is repeated.

Main technological parameters, defining The effectiveness of the process of separating mineral complexes in magnetic separation is: the magnetic properties of minerals, density, size, shape of particles, design features of magnetic separators.

Control questions

1. What is the process of magnetic enrichment based on?
2. What is the essence of magnetic separation?
3. What forces act on particles of material moving through separator working area?
4. What methods of separating particles based on magnetic properties do you do you know?
5. For the enrichment of what types of ores is the process often used? magnetic separation?
6. What do you understand by magnetic field strength?
7. Tell us about homogeneous and inhomogeneous magnetic fields
8. Under what conditions does the separation of particles occur? electromagnetic separators?
9. What is a magnetic separator?
10. What are the main parts of a magnetic separator?
11. Tell us about the classification of magnetic separators
12. How does a wet enrichment magnetic separator work?
13. How does a high-gradient magnetic separator differ from other separators?

Lecture No. 11 ELECTRICAL ENRICHMENT

General information and classification of electrical enrichment processes. Electric separators. Technological parameters of electrical separation**.**

*Key terms:*electrical separation, electric field, electrical conductivity, dielectric constant, contact potential, particle charge, collecting electrode, receiver, drum, operating voltage, corona discharge, triboelectric separator, charger, electrification

11.1 General information and classification of electrical processes enrichment

Electrical enrichment(electric separation) based on differences in the electrical properties of the separated minerals and is carried out under the influence of an electric field.

Electrical enrichment processes are processes in which the separation of mineral particles that differ in electrical properties is due to the difference in the nature and trajectories of their movement in an electric field.

Differences in the electrical properties of minerals are manifested in their electrical conductivity, dielectric constant, contact potential, triboelectric, pyroelectric, piezoelectric effects and different abilities, under the influence of certain physical influences, to acquire electrical charges of different magnitudes or signs.

Charging of particles of the separated material (Figure 11.1) can be carried out by contact with a charged electrode, ionization in the electric field of a corona discharge, electrification by friction, changes in temperature, pressure and other methods. The choice of particle charging method ensures the greatest difference in the electrical properties of the main separated minerals and thereby the maximum efficiency of electrical separation.

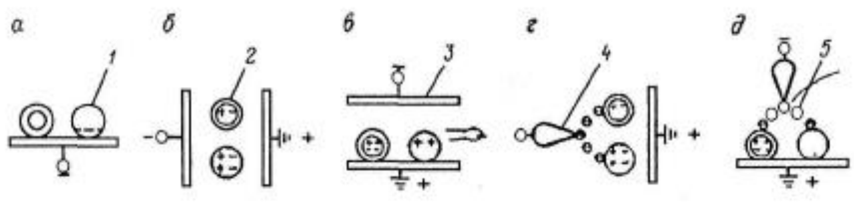

Figure - 11.1. The main methods of charging particles in electrical processes separation:

A- touch:*b*- induction;*V*- combined;*G*- gas ions; *d*- gas ions and discharge

Depending on what electrical properties are used as a separating feature, the following processes of electrical enrichment are distinguished: electrical separation, electrostatic, dielectric, triboelectric, triboadhesive, electrical classification by size and shape.

*Electrical separation*is the process of separating minerals particles, based on the difference in the magnitude of their electrical charges, by changing the trajectory of movement of these particles in an electrostatic field or the electric field of a corona discharge.

*Electrostatic separation*is the process of separating particles into electrical properties, depending on which the trajectory of these particles changes under the influence of an electrostatic field.

*Dielectric separation*is the process of separating minerals particles based on the difference in their dielectric constant.

*Triboelectric separation*is a process of mineral separation particles, based on the phenomenon of the triboelectric effect, which manifests itself during electrification by friction or contact.

*Triboadhesive separation*is a separation process based on differences in the forces of adhesion of particles to the electrode, in particular, on the difference in electrical components in adhesion.

Largest industrial application got processes, based on the difference in electrical conductivity and the ability of minerals to acquire different charges during contact electrification.

Based on electrical conductivity, minerals are divided into three groups:

• conductors with electrical conductivity 102-103*cm/m* (native metals, graphite, many sulfides, magnetite, hematite, rutile, etc.);

• semiconductors with electrical conductivity 10-108*cm/m* (bauxite, garnet, limonite, siderite, chromite, etc.);

• non-conductors (dielectrics) with electrical conductivity 10-8 *cm/m*(diamond, quartz, feldspar).

Minerals of each of these groups are characterized by a specific resistivity value. Conductors include minerals with a resistivity of less than 109*Ohm/m*, to non-conductors - more than 1012 *Ohm/m*.

In an electric field, conductors and non-conductors behave differently. When a conductor comes into contact with a charged body, due to good conductivity, the conductor acquires a charge of the same name and is repelled from the charged body, while in a dielectric there is only a displacement of charges and the orientation of electric dipoles in the direction of tension fields. As a result, conductors and dielectrics move along different trajectories and are separated into products with different contents of valuable components (minerals).

11.2 Electric separators

Electric separators are distinguished by the method of electrical separation, the characteristics of the electric field and the nature of the movement of material in the separation zone.

In accordance with the classification of electrical separation methods, electrical separators are divided into the following main groups: electrostatic; corona and corona-electrostatic; triboelectrostatic, pyroelectric and dielectric.

Based on the design of the main working body of the separation zone, separators are distinguished into drum, tray, chamber, blast, disk, and fluidized bed separators.

Electric separators have low energy consumption, do not use industrial water, do not pollute the air, and can be fully automated and controlled. They consist of three main parts:

• charger, or electrifier, in which they charge minerals;

- the actual separating part, in which the particles are
- separated; high voltage unit.

The charger and the separating part can be structurally combined or made separately.

Structurally, depending on the type of charged and deflecting electrodes, etc., electrostatic separators are divided into drum, chamber, cascade and plate (Figure 11.2).

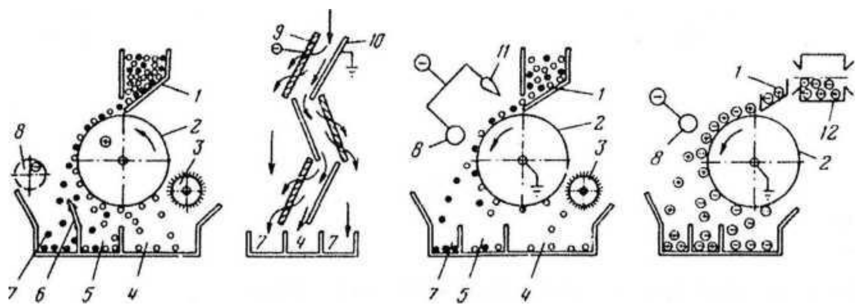

Figure 11.2 - Schemes of electrical separators:
A- drum;*b*- plate cascade;*V*- corona-electrostat- chesical;*G*– triboelectric

Electrostatic separator- this is an electric separator, in in which the source material is divided into components according to the difference in their electrical conductivity in an electrostatic field.

Electrostatic drum separator(drawing11.2,*A*) consists of a housing, a drum collecting electrode 2, a deflection electrode 8, a cleaning brush 3, and a hopper 7.

The source material from hopper 1 is fed in a thin layer onto a charged drum 2. Upon contact with the drum, electrically conductive particles are charged with the same charge and, pushing off from the drum, move along curvilinear trajectories into the electrically conductive fraction receiver 7. Non-conductive particles are charged more slowly and fall without deviating the trajectory into receiver 4 as a result of cleaning the surface of the drum with brush 3. A mixture of grains of different electrical conductivity enters receiver 5.

Electrostatic plate (cascade) separator(drawing 11.2,*b*) consists of parallel, inclined, zigzag plate electrodes. The lower plate electrodes 10 are smooth, the upper 9 are louvered. One row of plate electrodes is grounded (+), and the other is mounted on insulators and connected to a high voltage source (-). The operating voltage on the electrodes is about 70*kV*.

The source material sequentially passes through a cascade of plates, exposed to an electric field. Non-conducting minerals slide along the plates 10 and are discharged into the receiver 4, the conductive minerals come off the smooth electrode and, having passed the shutters of the opposite electrode 9, roll into the receiver 7.

The speed of movement of particles of the separated material along the plates is determined by the angle of inclination of the plates. The electric field strength and trajectories of separated particles in the interelectrode space of the separator are determined by the distance between the plate electrodes.

Corona electrostatic separator- it's electric a separator in which the source material is separated into components according to their electrical conductivity in a combined corona discharge field and electrostatic field.

Corona-electrostatic drum separator(drawing11.2,*V*) consists of a drum precipitation, corona and deflection electrodes. It simultaneously uses a corona discharge field and an electrostatic field.

The source material from feeder 1 is fed evenly across the entire width in a thin layer onto a rotating grounded drum-type precipitation electrode 2 and transported into the zone of action of the electric field of the corona discharge formed by the corona electrode 11. Here, each particle of the mixture acquires a charge, the sign of which corresponds to the sign of the corona. Since the particles are in contact with the surface of the grounded collecting electrode during charging, they are discharged simultaneously with the charging of the particles. Particles with high electrical conductivity (conductors), leaving the corona discharge zone, quickly give up their residual charge to the collecting electrode and centrifugal forces are thrown from the drum into the receiver for conductors 7. Particles with lower electrical conductivity give up their charge more slowly to the collecting electrode and come off from it later than the conductors. Non-conducting particles that did not have time to discharge during 3/4 of a revolution of the collecting electrode are cleaned from it with a brush and fall into receiver 4.

Thus, the particles, depending on the rate of transfer of their charge to the collecting electrode, determined by their electrical conductivity, have different coordinates of the points of separation from the surface of the drum. The formation of a fan of particles is facilitated by the electrostatic field generated by the deflecting electrode 8.

Triboelectric separator- this is an electrical separator, in in which the starting material is divided into components according to the difference in acquired triboelectric charges in an electrostatic field.

Triboelectric drum separator (Figure 11.2,*G)*has a charger 12 separated from the separating part. Charging of minerals is carried out in drum or other type devices by electrification by friction as a result of contact of minerals with each other. The electrifier is equipped with a heater for heating the material to 120-300*°C*, therefore, for minerals prone to pyroelectric electrification, the pyroelectric effect may be of auxiliary importance in creating charges. Separation occurs in an electrostatic non-uniform field of constant polarity with a strength of 2-4 *kV/cm,* created between a metal grounded drum 2 and a cylindrical deflecting electrode 8, to which a high voltage is applied (15-50*kV*). The sign of the voltage is selected taking into account the sign of the charge acquired by the minerals during electrification.

It also happens*pyroelectric and dielectric separators*. *Pyroelectric separators*- this is an electric separator, in in which the source material is divided into components according to their pyroelectric electrification in an electrostatic field.

*Dielectric separator*is an electric separator in which the source material is divided into components according to their dielectric constant in a dielectric liquid located in an electrostatic field.

11.3. Technological parameters of electrical separation

Main technological parameters, defining the effectiveness of the electrical separation process is: the difference in the electrical conductivity of the separated minerals; designs and operating principles of the separator; material and granulometric composition of mineral raw materials; technological mode of the process.

The greater the difference in values *electrical conductivity* separated minerals, the more significantly they differ in the rate of charging (on a charged electrode) and discharge (on a collecting electrode), the magnitude of residual charges and the trajectory of movement in the working space of the separator, the easier it is to separate them.

The quality of the products obtained during separation depends on *content separated minerals in the source material*. The less content it contains nonconductors, the higher the quality of the resulting fraction of conductive minerals. Fluctuations in the content of electrically conductive impurities in the separated minerals disrupt the stability of the separation process and lead to a decrease in the quality of the concentrates. The efficiency of the process and the quality of separation products deteriorate with an increase in the content of dust particles in the source material, therefore, before electrical separation, the material is usually subjected to thorough dust removal.

With increase *size* particles increases not only the amount of charge received in the field of a corona discharge or on a charged drum, but also the centrifugal force that separates them from the surface of the drum. This makes it difficult to clearly separate grains when separating a wide range of material sizes. In this case, a large non-conducting particle can come off the drum simultaneously with a smaller conductive particle and, conversely, very thin conductive particles will fall into the non-conducting fraction. Technological performance is significantly improved by electrical separation of narrowly classified material.

Structural characteristics separators provide significant impact on technological indicators. An increase in the rotation speed of the separator drum and an increase in centrifugal force contribute to the separation of conductors. However, its excessive increase can lead to the transition into the conductive fraction of those non-conducting particles that can no longer be held on the drum by the forces of electrical attraction. A decrease in the rotation speed of the drum leads to clogging of the conductive fraction with non-conductors, which manage to transfer their charge to the collecting electrode. Drum rotation speed with diameter 140-350 *mm* varies depending on the source material from 30 to 500 *rpm*.

With increase *electrode voltage* the difference in the charges of conductive and non-conducting particles increases and the results of their separation improve. The voltage at the corona electrode in modern separators is in the range of 35-50 *kV*, the maximum current in the interelectrode space is about 50 *mA*.

Decrease *distances between electrodes* increases the corona current and vice versa. The distance between the electrodes is set during the separation process and does not change during operation.

The productivity of each separator cell depends on the length of the drum, varying in different separators from 800 to 3000 *mm,* and the speed of its rotation. By increasing the rotation speed, you can increase separator performance, but the quality of the separation products may deteriorate.

Control questions

1. What is the electrical method of enriching minerals based on? fossils?
2. How can the particles of the separated material be charged? material?
3. Depending on the dividing criterion, what processes exist electrical enrichment?
4. Based on electrical conductivity, what groups are minerals divided into?
5. How conductor and non-conductor minerals behave in electric field?
6. What are the advantages of electric separators over other types? mineral beneficiation?
7. What are the main parts of electric separators?
8. Explain the working principle of electrostatic drum separator
9. Explain the operating principle of the corona electrostatic drum separator
10. On what factors does the electrical efficiency depend? separator?
11. How do dust particles affect the outcome of electrical separation?

Lecture No. 12 FLOTATION ENRICHMENT

General information and classification of the flotation process. Theoretical foundations of flotation. Purpose and classification of flotation reagents

*Key terms:*flotation, hydrophobic particles, aeration; foam, foaming agent, contact angle, floatability index, reagent, collecting reagents, activating reagents, hydrophobization, sulfidization, depressant, environment regulator, sludge, foaming reagents

12.1. General information and classification of the flotation process

Flotation enrichment (flotation)- this is a process of enrichment minerals, based on the selective adhesion of mineral particles to the interface between two phases; liquid - gas; liquid - liquid, etc. Depending on the phases involved in the process, flotation can be oil, foam, on a hydrophobic solid surface, on a fatty surface.

*Froth flotation*is a process by which hydrophobic particles stick to air or gas bubbles introduced into the pulp and rise with them to the top, forming foam, and hydrophilic particles remain suspended in the pulp.

*Film flotation*is a process in which hydrophobic particles, falling on the surface of a moving stream of water, they remain on it, forming a film, and hydrophilic particles sink.

*Oil flotation*is a process by which hydrophobic particles stick to the oil droplets in the pulp and float to the top, while the hydrophilic particles remain suspended in the pulp.

*Solid wall flotation*is a process of flotation of fine sludge (-10 *μm*) using a carrier - hydrophobic particles of flotation size, selectively interacting with the extracted sludge, while the resulting aggregates are subjected to conventional foam flotation.

*Ion flotation*is a process designed to extract from solutions of ions that form finely dispersed hydrophobic precipitates when interacting with collecting reagents.

*Vacuum flotation*is a type of flotation process with using gas bubbles released from solutions or suspensions in a vacuum.

*Electroflotation*is a process of flotation of mineral raw materials bubbles of oxygen and hydrogen formed during the electrolysis of water.

*Floccular flotation*is a flotation process characterized by extraction of particles in the form of flocs formed as a result of pre-treatment of particles with reagents.

*Foam separation*is a type of flotation process in which the initial heterogeneous mixture (suspension) is fed from above onto a previously prepared foam layer without destroying it.

Flotogravity- this is the process of enriching mineral raw materials, which consists of a combination of flotation and gravitational enrichment methods and in which a mixture of minerals treated with reagents is subjected to gravitational enrichment (on concentration tables, jigs, tapering chutes).

Currently, the most widespread process is *froth flotation*as the most universal enrichment method almost all types of minerals.

The flotation process is carried out in concentrators called flotation machines, the main structural elements of which are body 1, mixing and aeration device 2, foam 3 (Figure 12.1).

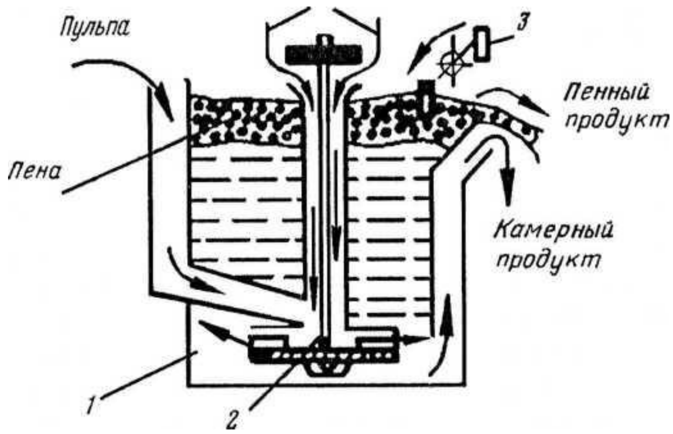

Figure 12.1 - Schematic diagram of the flotation process:
1 - body; 2 - mixing and aeration block; 3 – penogon

Flotation occurs in the following sequence of subprocesses:

• pulp, representing a suspension, is fed into the flotation chamber mineral particles in water;

• with the help of flotation reagents introduced into the pulp, conditions for the adhesion of particles of some minerals to air bubbles and, conversely, the adhesion of particles of other minerals to the bubbles is prevented;

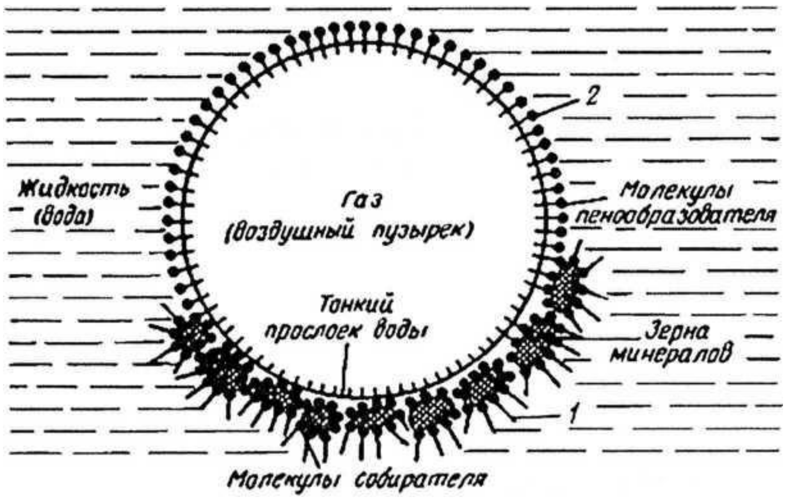

Figure 12.2 - Scheme of the formation of a mineralized bubble complex air - mineral particles in the pulp:
1, 2 - apolar and polar parts of the collector molecule, respectively foaming agent

• dispersed air is supplied to the pulp and it forms a large number of small bubbles, to stabilize which foaming reagents are supplied to the pulp;

• mineral particles collide with air bubbles and are fixed on them, forming a mineralized complex (Figure 12.2);

• mineralized bubbles float to the surface of the pulp, forming a foam layer, which is a moving medium consisting of gas bubbles tightly pressed against each other, with a mineral load;

• mineralized foam is removed from the pulp surface using special devices (foamers).

Typically, the beneficial minerals are transferred to the foam layer, while the gangue minerals remain in the pulp.

As a result, two products are obtained: foam and chamber.

The overall efficiency of flotation depends on the technological properties of the mineral raw materials and the efficiency of each of the constituent subprocesses.

12.2. Theoretical foundations of flotation

The basis of flotation is the formation of mineralized bubbles. The formation diagram of an aggregate consisting of one bubble and one mineral particle is shown in Figure 12.3.

In accordance with the second law of thermodynamics, the fixation of a particle on the interphase surface and flotation are possible if the free energy of the system after fixing the particle on the bubble E_2 there will be less free energy of the system before the particle is fixed E_1. In this case, the system from state 1 will spontaneously transition to state 2.

Free energy reserve of the system W_1 until the particle sticks to the bubble was:

$$W_1 = {}_{z-g}\ {}_{z-g} + {}_{t-w}\ {}_{t-w}, \qquad (12.1)$$

Where ${}_{z-g}$ and ${}_{t-w}$ - area of interface between liquid - gas and solid - liquid; ${}_{z-g}$ And ${}_{t-w}$ - surface energy on the same sections.

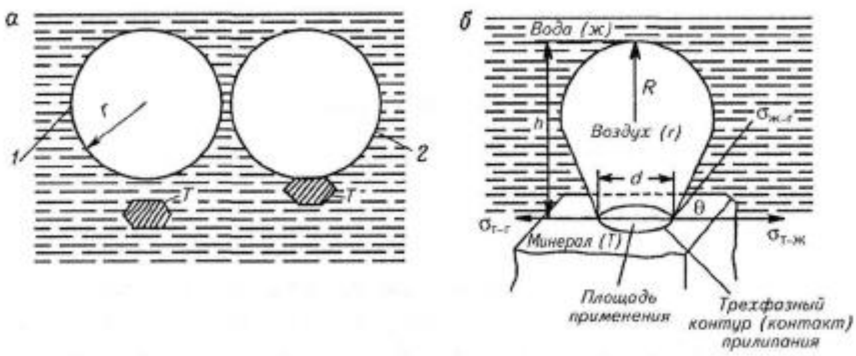

Figure 12.3 - Schemes for the formation of an air bubble-particle aggregate mineral (*A*) and acting forces (*b*):
1,2 – apolar and polar parts of the collector molecule, respectively and foam concentrate

Free energy reserve of the system W_2 after the particle adheres to bubble, related to the adhesion area of $1\ cm^2$, will be:

$$W_2 = ({}_{z-g} - 1) + {}_{t-g}1 + ({}_{t-w} - 1)\ {}_{t-w} \qquad (12.2)$$

Decrease in free energy of the system ΔW takes place provided:

$$\Delta W = W_1 - W_2 = {}_{z-g} + {}_{t-w} - {}_{t-g} > 0 \qquad (12.3)$$

Or

$$\Delta W = {}_{z-g} + {}_{t-w} = {}_{t-g} \qquad (12.4)$$

When fixing an air bubble on a solid surface from the conditions of equilibrium of a point on a three-phase solid-gas-liquid contact, the following relationship is valid (Figure 12.3);

$${}_{t-g} - {}_{t-w} = {}_{z-g}\cos \qquad (12.5)$$

Where - equilibrium *contact angle*, which is understood as the angle formed by the interface between two phases and the surface of the third phase and measured towards the liquid phase.

Substituting expression (12.5) into (12.4), we obtain an equation for determining the loss of free energy Δ *W* flotation system after fixation of a particle on a bubble under equilibrium conditions:

$$\Delta W(F) = z\text{–}g(1 - \cos\ ). \qquad (12.6)$$

The floatability indicator is the value *F*, characterizing the change in the surface energy of the system when a particle is fixed on the phase interface, related to a unit area of gas-solid contact. When >0, then Δ *F* > 0, i.e. particle adhesion to the bubble is possible if the contact angle is greater than zero.

From expression (12.6) it follows that the larger the contact angle, the more hydrophobic the surface of the mineral, the better its adhesion to the air bubble and the higher the floatability index.

The contact angle of wetting for various minerals can vary widely and is about 0° for natural hydrophilic quartz, 60-90° for coal, 75-85° for sulfides.

The flotation ability of minerals, i.e. the degree of wettability with water can be changed artificially by treating their surface with flotation reagents.

12.3. Purpose and classification of flotation reagents

Flotation reagents are called organic And inorganic substances that are introduced into the flotation process in order to regulate the floatability of minerals

Depending on their purpose, flotation reagents are classified into collectors, modifiers, and foaming agents.

Auxiliary reagents include regulators *pH* environment, foam modifiers, flocculants and dispersants.

Collection reagents - these are reagents, providing hydrophobization of the surface of mineral particles to be extracted into a foam product. A large number of organic compounds have been proposed as collectors

Collector reagents are divided into two large groups: ionic (dissociating into ions) and nonionic (not dissociating into ions)

Ionic collectors interact with minerals primarily on the basis of chemical adsorption, while nonionic collectors are attached to mineral particles by physical adsorption and adhesion.

Figure 12.4 shows the structure of the collector molecule, xanthate. The molecule consists of a polar group that does not interact with water, which chemically binds its molecule to the mineral and fixes it on the surface of the mineral, and the apolar group of the hydrocarbon radical *S4N9,*which imparts hydrophobic properties to the mineral, i.e. makes him difficult to wet with water.

As a result of hydrophobization, the three-phase wetting perimeter is consolidated, the rate of adhesion of air bubbles to the mineral is increased, and the adhesion strength is significantly increased.

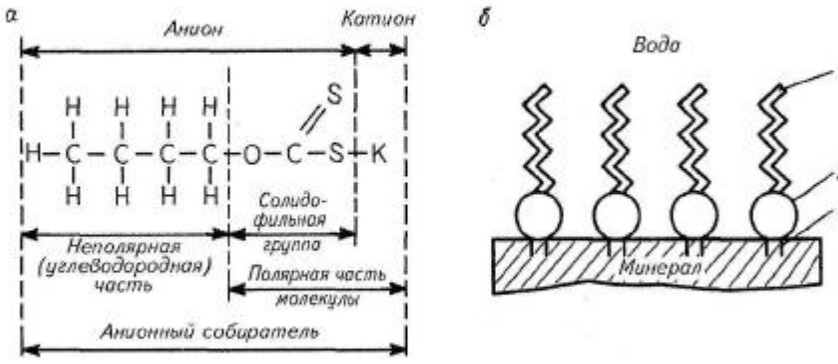

Figure 12.4 - Structure of the molecule of a heteropolar collector (xanthogen) (a) and its adsorption layer on the surface of the mineral (b):
I - hydrocarbon radical; 2 - solidophilic group; 3 - bonds of the solidophilic group with the mineral lattice

Activator reagents- these are reagents that create conditions, favoring the fixation of collectors on the surface of minerals.

There are three main mechanisms of action of activating reagents

Mechanism 1- formation of minerals on the surface is not a film interacting with the collector, on which the collector is actively fixed. For example, sulfidization of oxidized minerals using sodium sulfide (Figure 12.5, 1). Oxidized minerals themselves are unable to react with xanthate. However, when interacting with sodium sulfide, a film of metal sulfide is formed on their surface, on which the collector is fixed.

Mechanism 2- fixation of ions on the surface of minerals - activators, with which the collector then interacts.

For example, activation of quartz flotation by a collector with a carboxyl polar group by means of preliminary fixation of di- and trivalent metal ions on it (Sa^{2+}, Va^{2+}, $A1^{3+}$etc.) (picture12.5,*2a*) or activation of sphalerite flotation by copper ions (Figure 12.5,*2b)*.

Mechanism 3- dissolution and removal from the surface of secondary hydrophilic film followed by interaction of the collector with the freshly exposed surface.

For example, removing a film of iron hydroxides from the surface of pyrite using acid and then fixing xanthate on pyrite (Figure 12.5, 3).

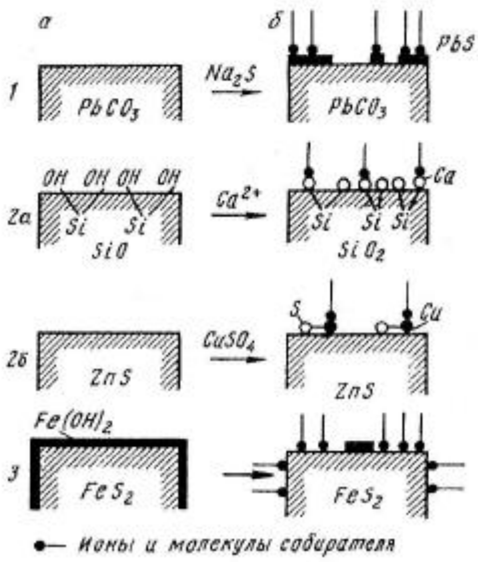

Figure 12.5 - Schemes of the mechanisms of action of reagents on the state
mineral surfaces:
a - before adding the activator; b - after adding the activator

Depressant reagents - these are reagents used for minerals by preventing hydrophobization collectors. They designed to increase the selectivity of flotation when separating minerals with similar flotation properties

There are 5 main mechanisms of action of depressant reagents.

Mechanism 1 - dissolution of a collector previously attached to the surface of the mineral. For example, suppression of flotation of copper sulfides (chalcopyrite, covellite, chalcocite, etc.) using cyanides as a result of the following process (Figure 12.6, 1).

Mechanism 2 - displacement of collector ions by depressor ions, which form a sparingly soluble hydrophilic compound with mineral ions. For example, suppression of galena flotation by sodium sulfide (Figure 12.6, 2). In this case, sulfur ions displace xanthate ions from the surface of the mineral.

Mechanism 3 - the formation of hydrophilic compounds by the depressant reagent on surface areas not occupied by collectors (without its displacement). As a result, the total hydration of the surface increases, and flotation worsens. For example, suppression by potassium dichromate (K_2Cr_2ABOUT$_7$) galena flotation (Figure 12.6, 3). In this case, the less hydrophilic areas of galena, not occupied by the collector, are covered with the resulting more hydrophilic lead sulfide chromates. The total hydration of the surface increases, and the floatability of galena decreases.

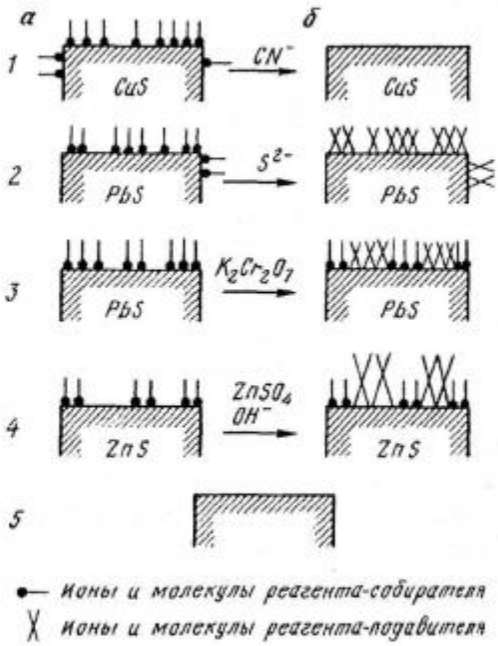

Figure 12.6 - Schemes of the mechanisms of action of depressant reagents on state of the mineral particle surface:
A- before adding depressant;*b*- after adding depressant

Mechanism 4- securing in areas free from the collector the surface of relatively large hydrophilic compounds formed by depressant reagents in solution and shielding the collector molecules, suppressing flotation. For example, suppression of sphalerite flotation by zinc sulfate in an alkaline environment created by soda (Figure 12.6,4). In this case, hydrophilic particles of basic zinc carbonate are formed in the solution, which adhere to the surface of sphalerite, reducing its floatability,

Mechanism 5- change in the properties of collecting reagents located in liquid phase of the pulp, leading to deterioration of their fixation on the surface of minerals. For example, binding of the collector into insoluble compounds, reducing the degree of dissociation of the collector with a decrease in the concentration of flotation-active anions of the collector, etc.

*Towards environmental regulators*regulators include*pH*environments and regulators ionic composition of the pulp.

Hydrogen index (*pH*) environment is the value of the logarithm of the concentration of hydrogen ions, taken with the opposite sign:

$$pH=-\log [H]^+ \qquad (12.7)$$

For acidic environments$pH < 7$, for neutral environment$pH = 7$, for alkaline environment$pH > 7$.

Many flotation reagents in aqueous solutions are highly hydrolyzed, and the ratio between the concentration of ions and molecules of flotation reagents largely depends on the alkalinity of the medium. For regulation*pH* media use acids, alkalis or their salts.

Regulators of the ionic composition of the pulp are used to remove ions from the pulp that interfere with the interaction of the collector with the surface of the floated minerals. For example, during flotation with sodium oleate, calcium and magnesium ions are harmful, since when they interact with sodium oleate they form sparingly soluble compounds that precipitate, as a result of which the concentration of collector ions in the pulp decreases and, as a result, flotation deteriorates.

The selectivity of the flotation process of lead minerals from lead-zinc ores is adversely affected by copper ions contained in the pulp and activating the flotation of sphalerite. As a result, sphalerite turns into foam along with galena.

Lime, soda and other regulators are used to bind unwanted ions. Regulating reagents are used in flotation and as sludge peptizers to reduce their negative impact on flotation results. Liquid glass, sodium hexametaphosphate, sodium tripolyphosphate, and some electrolytes are widely used as peptizing agents in flotation.

*Foaming agents*are reagents intended for increasing dispersion and stabilizing air bubbles in the pulp and increasing the stability of foam saturated with particles of the floated mineral. In addition, foaming agents slow down the rise of bubbles, and sometimes affect the collective action of reagents and the adhesion strength of particles to bubbles.

Foaming agents present yourself superficial active substances (surfactants) that can spontaneously adsorb on the water-air interface with a decrease in surface energy. Surfactant adsorption is oriented. In this case, the polar group is in water, and the hydrophobic radical is on its surface (Figure 12.7).

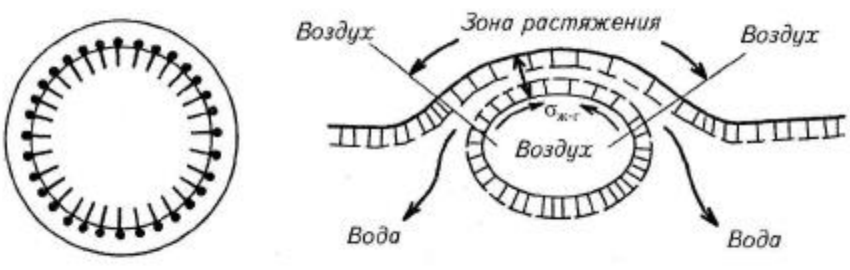

Figure 12.7 - Scheme of bubble stabilization by foaming agent molecules

Substances containing one of the following polar groups are used as foaming agents: hydroxyl (*HE*-),
carboxyl (CO_2– $_3$), carbonyl*(=C=O)*, amino group (*N.H.*$_2$),
sulfo group (OSO_2OHorSO_2OH).

The most widely used in processing plants are heteropolar compounds with a non-ionizing polar group:

pine oil, synthetic reagents OPSB, T-88, MIBK, cyclohexanol, etc.

Control questions

1. What is the process of flotation enrichment of minerals based on? fossils?
2. What types of flotation do you know?
3. What are the main parts of a flotation machine?
4. In what sequence does the flotation process occur?
5. What is contact angle?
6. For what purpose are reagents introduced into the flotation process?
7. What types of reagents are divided into?
8. What is the function of collection reagents?
9. What role do activating reagents play in the flotation process?
10. When is a medium considered alkaline?
11. What are foaming agents intended for?

Lecture No. 13 FLOTATION MACHINES. SCHEME FLOTATION

Flotation machines and their types. Flotation schemes.

*Key terms:*flotation machines, aeration, agitation, pneumomechanical flotation machines, pneumatic flotation machines, mechanical flotation machines, aerator, impeller, stator, suction chamber, direct-flow chamber, airlift flotation machine, flotation circuits, main flotation, collective flotation, cleaning, selective flotation

13.1 Flotation machines

*Flotation*called a flotation enrichment machine, in a chamber in which the starting material is separated in an aerated pulp into foam and chamber products according to the selective ability of minerals to adhere to floating air bubbles.

The classification of flotation machines into separate design types is based on the method of aeration and mixing of the pulp. Based on this criterion, all flotation machines are divided into three groups - mechanical (FM), pneumomechanical (FPM) and pneumatic (FP).

In the symbols of brands of flotation machines after the letters, the first number indicates the volume of the chamber in cubic meters.

Mechanical flotation machines are manufactured with chambers ranging in volume from 0.2 to 12.5 m^3, pneumomechanical - from 0.4 to 25 m^3, pneumatic - from 1 to 100 m^3.

The size of ore particles in the feed of mechanical flotation machines should not exceed 1 -1.2 *mm*, the largest particle size of minerals raised into foam is 0.2-0.3 *mm*.

The size of air bubbles varies widely and depends on the type of machine. The smallest bubbles (diameter 0.8-1 *mm*) are produced by mechanical type machines, and the largest (2.5-4 *mm*) - pneumomechanical machines.

IN*mechanical flotation machine*- aeration and mixing of the pulp is carried out by an aerator, sucking the air necessary for flotation directly from the atmosphere. Depending on the design of the aerator, machines are classified into impeller, ejector, and fluidized bed.

The most universal and widespread are mechanical flotation machines (type FMR - for ore, type FMU - for coal).

The Mekhanobr flotation machine (FMR) is used in many domestic ore processing plants. The FMR machine consists of sections including suction and direct-flow chambers (Figure 13.1).

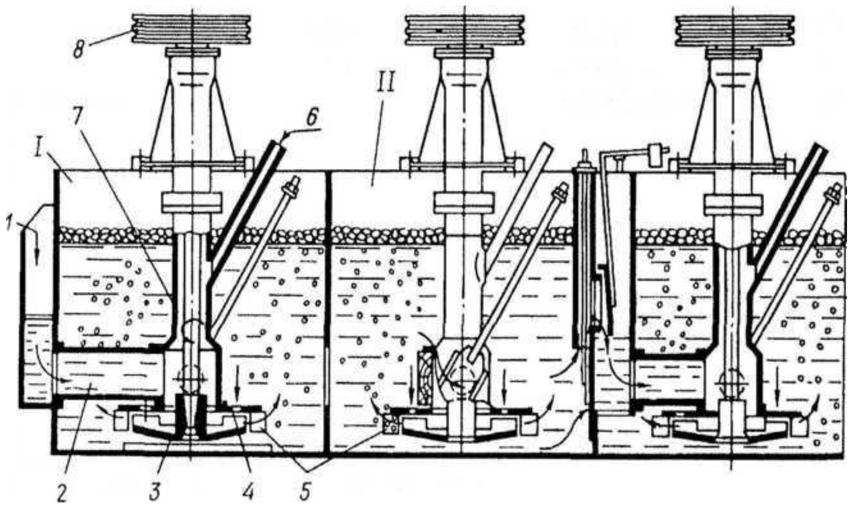

Figure 13.1 - Flotation machine “Mekhanobr” of mechanical type (lengthwise cut):

1 - suction chamber: II - direct-flow chamber; I - receiving pocket; 2 - pulp supply pipe; 3 - impeller; 4 - stator; 5 - guide vanes; 6 - pipe for air suction; 7 - aerator; 8 - impeller drive

The chambers communicate with each other through holes in the interchamber partitions. Each chamber has an aerator 7, consisting of a stator 4 and an impeller 3. The impeller consists of a vertical shaft, at the lower end of which an impeller disk with radial blades is attached. The impeller shaft is located in a central hollow tube and receives rotation through a V-belt transmission from an electric motor. At the bottom of the central pipe, a stationary over-impeller disk is attached, in the peripheral part of which there are guide vanes 5 located at an angle of 60° to the radius. The above-impeller disk with guide vanes is called a stator.

When the Mekhanobr machine operates and the impeller rotates, a vacuum is created in the space above the impeller under the stator disk, as a result of which pulp is sucked into the impeller through the hole in the central pipe and in the stator, and air from the atmosphere is sucked through pipe 6. The air, together with the pulp, is thrown into the chamber by the blades of the rotating impeller through the gaps between the stator blades, while it is crushed into tiny bubbles.

Hydrophobized mineral particles, colliding With air bubbles are fixed on their surface, air bubbles loaded with mineral particles rise to the upper zone of the chamber, where they form a layer of foam. The foam product is continuously removed by a rotating foam gun. The pulp enters the cavity of the impeller of the suction chamber from receiving pocket 1 or from an intermediate pocket. IN

In the direct-flow chamber, the pulp flows through a window in the interchamber partition. In a machine, suction and direct-flow chambers can alternate through one chamber, or several suction or several direct-flow chambers can be installed in series. The chamber product passes into the direct-flow chamber. A multi-chamber machine can be easily assembled from separate two-chamber sections, depending on the capacity and flotation scheme.

Pneumatic flotation machine- car, in the cell in which aeration and mixing of the pulp is carried out by an aerator fed by compressed air from an external source. Based on the type of aerator, machines are classified into impeller and vibration.

In pneumomechanical machines (Figure 13.2), compressed air is supplied to chamber 1 through the radial holes of the hollow shaft 3 into the sub-impeller zone of the finger aerator 2. Pneumomechanical blocks are not suitable for receiving circulating products.

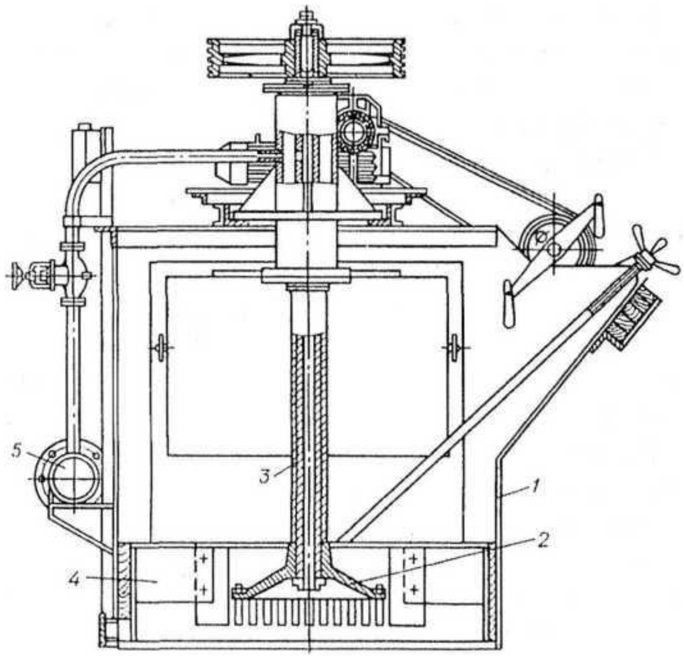

Figure 13.2 - Pneumo-mechanical flotation machine:
1 - camera; 2 - aerator; 3 - hollow shaft; 4 - stator; 5 - compressed air pipeline

The suction chambers of these machines are mechanical without additional compressed air supply. In the absence of mechanical chambers (cascade with machine sections arranged in steps along the length, direct-flow pneumomechanical), the pulp is fed into the flotation machine under hydrostatic pressure, and instead of unloading pockets, cascade transition boxes are installed, providing a back-up of the pulp at the entrance to the next chamber.

Machines with one-sided foam removal are manufactured in right and left versions. Right-handed machines (type FMR, FPR) have power supply comes from the right, if you look at the machine from the side where the foam product is removed; in left-handed machines (FPMGMO, FMR25S and FMR63S), on the contrary, power is supplied from the left, and foam is unloaded from the right along the flow of the main power. Converting a right machine to a left one and vice versa is carried out without additional parts.

Pneumatic flotation machine- flotation machine, in in which aeration and mixing of the pulp is carried out by compressed air supplied from an external source. Pneumatic flotation machines are mainly represented by airlift machines and foam separation machines.

Airlift flotation machine- pneumatic flotation a machine in which compressed air is supplied to a stationary airlift device.

A deep airlift machine (Figure 13.3) of the trough type has an aeration compartment 1 and flotation compartments 2 along its entire length, limited by partitions 5. From the longitudinal air manifold into the machine are vertical cabins 3, which have rubber tips 4 at the ends, which are simple valves that prevent pulp getting into the nozzles after stopping the supply of compressed air and clogging them.

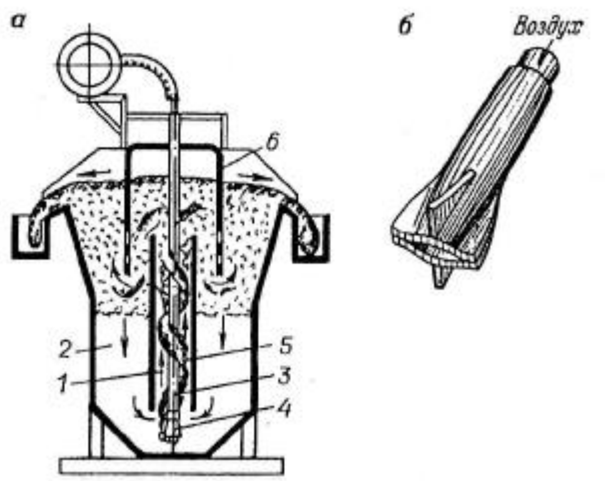

Figure 13.3 - Deep airlift machine: a - section; b - rubber tip (valve)

The compressed air coming out of the nozzles passes through more than half of the aeration compartment in a continuous stream or in the form of air plugs and only then breaks up into large bubbles. Small bubbles are also produced in vortex flows created by partitions 6.

Flotation foam separation machine- it's pneumatic flotation machine, into the chamber of which the original pulp is fed directly onto the foam layer. This is what makes it fundamentally different from other designs.

More hydrophobic particles are retained in the foam, while less hydrophobic particles, under the influence of gravity and flowing water, pass through the foam and fall out of it.

The design of the foam separation machine is shown in (Figure 13.4). It consists of a trough-type flotation chamber 7, along the middle of which there is a loading device 2, made in the form of a grooved divider, which evenly distributes power along the length of the machine. Below the surface of the pulp at a depth of 150-200*mm*There are 3 tubular aerators (a series of rubber tubes with tiny holes) into which compressed air is supplied. In the presence of a foaming agent, a layer of fairly stable foam is formed on the surface of the pulp. A small amount of water is applied to the initial section of the foam using spray 4, which enhances the precipitation of hydrophilic particles from the foam.

Foam 5 with particles retained in it is removed by gravity or with the help of paddles through the thresholds 6 into the gutters. In the case of flotation of water-soluble salts, the mother liquor is released from the foam through meshes 8. Tailings are removed through unloading device 7.

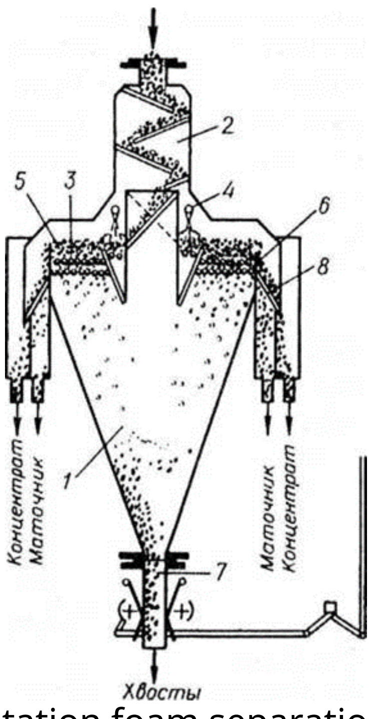

Figure 13.4 - Flotation foam separation machine

1- trough for concentrate; 2 - pacifier; 3 - camera; 4 - pulp distributor; 5 - ejector; 6 - air disperser; 7 - valve

*Column flotation machine*is round or square chamber about 1 wide *m*and about 7-9 in height*m*(Figure 13.5).

The power is supplied above the middle, but below the foam layer, which occupies about 1/3 of the column height, and moves from top to bottom. Aeration is carried out below using various aerators with small holes. The simplest option is rubber tubes with holes. IN

The column carries a countercurrent of falling particles and rising bubbles, and due to the long path traveled by the bubbles, they collide with particles much more often.

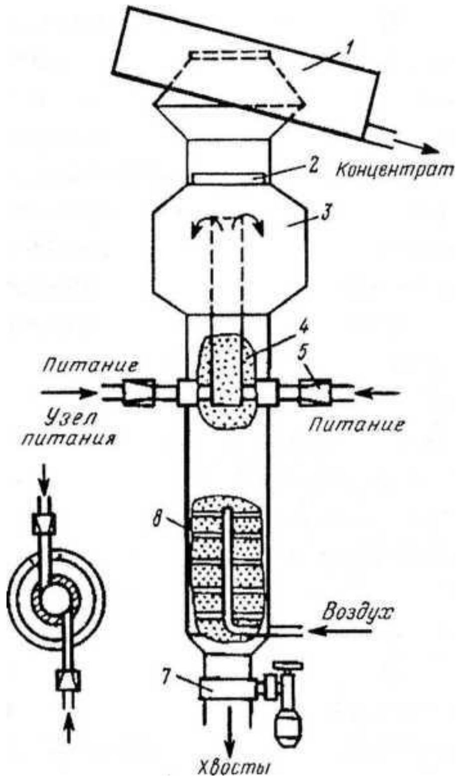

Figure 13.5 - Column flotation machine:
1 - concentrate trough; 2 - pacifier; 3 - camera; 4 - pulp distributor; 5 - ejector; 6 - air disperser; 7 – valve

The advantages of pneumatic type machines include: extreme simplicity of design; absence of rotating parts, wearing parts and assemblies; low metal consumption; ease of operation. Their disadvantages are: the need to use air injection units for air supply and pumps for pumping industrial products; relative unreliability of aerators, limited application (only for simple flotation enrichment schemes).

The group of flotation machines with variable pressure includes vacuum and compression flotation machines.

*In vacuum flotation machines*pulp, previously processed with reagents and saturated with air, it enters a vacuum chamber, in which (due to a decrease in pressure above the pulp) bubbles of dissolved air are released on the surface of hydrophobic particles, leading to their flotation. Hydrophilic particles settle to the bottom of the chamber and are discharged into a tailings collector. Vacuum flotation is mainly used to separate fine slurries.

In compression machines flotation is carried out as a result release of very small bubbles when pressure above the pulp decreases. Before being fed into the machine, the pulp is saturated with compressed air at excess pressure (3÷4) 10^5 *Pa*, and then fed into a machine bath, in which, when the pressure above the pulp drops to atmospheric pressure, dissolved gases are released in the form of microbubbles directly on the surface of the particles. Compression machines are widely used for flotation treatment of industrial wastewater.

In electroflotation machines pulp aeration is carried out by water electrolysis account. During electrolysis, very small bubbles are released (0.05-0.2 *mm*) hydrogen and oxygen, providing effective flotation of thin suspensions and flakes. The use of electroflotation machines is effective for ion flotation and wastewater treatment. Electricity costs are 0.3-0.4 $kW\ h/m^3$ liquid to be cleaned

13.2 Flotation schemes

Flotation scheme is called a combination of individual flotation operations. Existing flotation schemes are very diverse. They depend on the flotation properties of the enriched raw materials and the requirements for the quality of enrichment products.

The following names of individual flotation operations are currently accepted.

Main flotation is a flotation operation of mineral raw materials, in which produces the final concentrate or concentrate sent to cleaner flotation. Sometimes a flotation circuit may have several main flotations (for example, when enriching polymetallic ores, one circuit may have main lead and main zinc flotation, etc.).

Cleaner flotation is a flotation operation that receives main concentrate or product of a previous flotation operation to increase the concentration of the isolated mineral.

Control flotation is a flotation operation following main flotation for the purpose of additional extraction of valuable components and production of waste tailings.

Stage is a part of a circuit that includes one operation ore grinding and the following group of flotation operations. There are one-, two- and three-stage flotation schemes.

Cycle flotation schemes are called a group of flotation operations, in which produces one or more finished (not subject to further flotation) products. Each stage of the circuit can have several cycles.

*During flotation of polymetallic ores*with receiving several concentrates, depending on the sequence of separation of useful components, a distinction is made between collective flotation, sequential-selective and collective-selective.

*Collective flotation*is a method of extracting certain minerals with similar properties into a common collective concentrate. For example, during the flotation of gold ores, gold and sulfide minerals are transferred into the concentrate.

*Sequential selective flotation*is the process of separation one mineral from a group of minerals into a concentrate consisting primarily of one mineral.

*Collective selective flotation*is a method that provides obtaining a collective concentrate consisting of several types of minerals, followed by its division into monomineral concentrates. Most often, the separation of common (collective) concentrates is preceded by their regrinding.

According to the diagram in Figure 13. 6,*A*There is no additional grinding of products during the flotation process. The opening of all mineral grains occurs during the process of preliminary grinding of the ore, so the scheme is single-stage. In addition, all flotation operations are united by the commonality of the used flotation reagents and reagent modes and the commonality of the goal, which is to isolate the valuable component contained in the ore in the form of a final concentrate, therefore the scheme is also single-cycle. Schemes of this type are typical for the enrichment of monomineral ores with uniform dissemination of minerals.

According to the sequential-selective flotation scheme shown in Figure 13.6,*b*, individual minerals are sequentially separated from the ore, with easy-to-float minerals, such as galena, and then difficult-to-float minerals, such as sphalerite, to be isolated first.

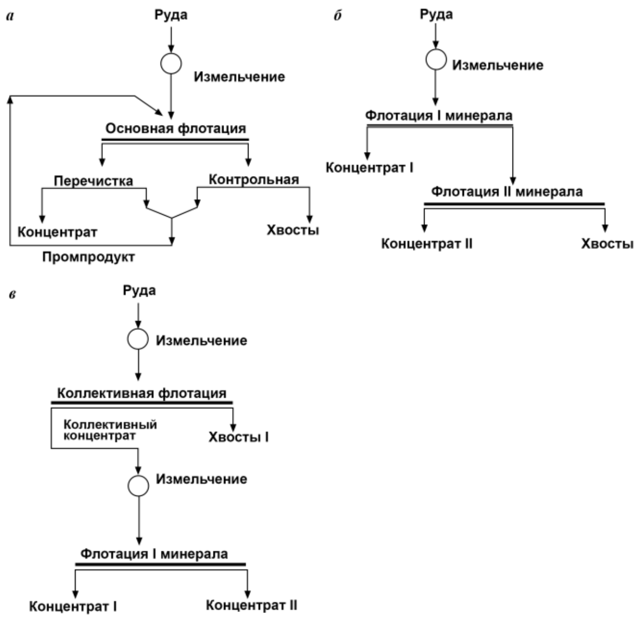

Figure 13.6 - Flotation schemes:
A– one-stage; *b*– selective flotation; *V*– collectively selective flotation

According to the collective-selective scheme (Figure 13.6, *V*) provides for additional grinding of the collective concentrate and its subsequent selective flotation. The scheme includes three independent cycles:

• collective flotation of two valuable components into a common concentrate;

• selective flotation of the first component;

• selective flotation of the second chain component. The rational distribution of flotation operations among individual chambers or groups of chambers of flotation machines is very important for organizing the process with the smallest number of chambers and transport units.

An example of a flotation scheme combined in one flotation machine, including main flotation, one concentrate cleaner flotation and one tailings control flotation, is shown in Figure 13.7

The main flotation is carried out in four chambers of the machine; the foam product (concentrate) enters the first cleaner flotation chamber (three chambers in total).

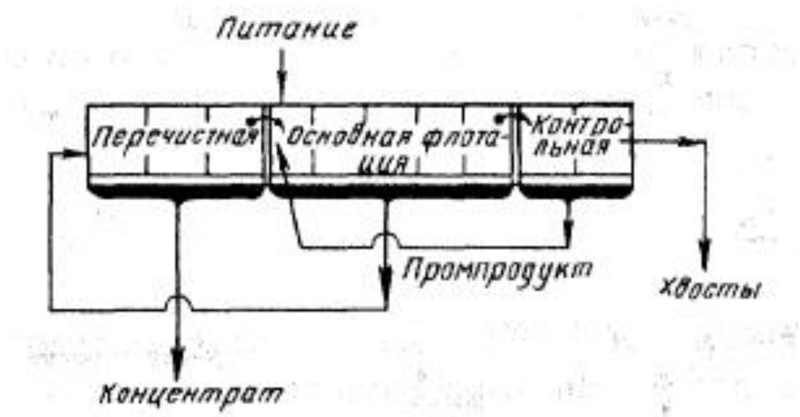

Figure 13.7 - Scheme of distribution of flotation operations in a flotation room car

Control questions

1. What is the function of a flotation machine?
2. What types of flotation machines do you know?
3. What are the main parts of a flotation machine?
4. In what order does the separation of useful minerals take place? flotation machines?
5. Explain the operating principle of a rotor-mechanical flotation machine
6. What are the main parts of an airlift flotation machine?
7. What are the main parts of a flotation column? car?

8. What is a flotation circuit?
9. What types of flotation operations are there?
10. Define the concepts of flotation stage and cycle.
11. What types of flotation schemes do you know?

Lecture No. 14 FACTORS AFFECTING THE PROCESS FLOTATION

Technological options flotation. Job flotation branches

*Key terms:*material composition, nature of impregnation, pulp density, pulp temperature, water composition, reagent mode, degree of aeration, bubble size, sludge, pulp density, floatability, reagent mode, bubble size, flotation speed, pulp level

14.1. Flotation technological parameters

Main technological parameters, defining the results of the flotation process are: material composition; the nature of inclusion and other properties of the mineral; granulometric characteristics of the solid phase; pulp density and temperature; water composition; reagent mode; degree of aeration and bubble size; flotation machine design, etc.

Material composition ores and coals determines their floatability.

Minerals are classified into several groups based on their floatability: *apolar metal minerals* (graphite mineral sulfur, etc.), *heavy metal sulfides* (minerals of copper, lead, zinc, molybdenum and etc.), *oxidized heavy metal minerals* (carbonates and sulfates of copper, lead, zinc, etc.), *oxides, silicates, aluminosilicates* and etc.

Flotation separation of minerals of various groups is carried out quite easily, often using one collector in the presence of a foaming agent.

For example, the separation of sulfide minerals from non-sulfide minerals (quartz, etc.) is usually a simple operation carried out by xanthate *pH 8.5*.

Flotation separation of minerals belonging to the same group is much more difficult to implement. In each specific case, the use of selectively acting reagents is required - depressors, activators, environmental regulators.

For example, in the selective flotation of copper-pyrite ores, the separation of sulfides can be achieved by adjusting *pH* environment in copper flotation with lime at a flow rate of up to 1.5-2 *kg/t* ore.

During the flotation of copper-lead-zinc ores, alkali cyanides (KCN, NaCN and Ca(CN) are used to suppress zinc (sphalerite)$_2$) at a flow rate of 200-300 *g/t*. Sphalerite depressors are also sulfite ($NaS0_3$), sodium thiosulfate ($Na_2S_20_3$) and sodium sulfide (Na_2S).

For the depression of secondary sulfides (bornite, covellite, etc.), a mixture of cyanide and zinc sulfate is used at a consumption of up to 5-6 *kg/t*.

When separating lead and copper, chromate salts, iron sulfate in combination with alkali metal sulfite, phosphates, etc. can be used to depress lead (galena).

An effective waste rock depressant is liquid glass at a flow rate of up to 100 *g/t* ore.

In reverse anion or cation flotation of iron ores, quartz and silicates are floated with a carboxyl collector (200-600 *g/t*) using lime as a waste rock activator (*pH* Wednesday until11), and the flotation of iron minerals is suppressed by starch (consumption up to 1 *kg/t*), metaphosphates, carboxymethylcellulose, etc. The consumption of depressants is 0.5-1 *kg/t*.

Source material size during flotation it should be like this, when in which the maximum proportion of useful minerals is freed from intergrowths with gangue minerals, and the size of floated particles must correspond to the lifting force of air bubbles.

Typically, flotation is carried out when the particle size of useful minerals is in the range of 0.02-0.6 *mm*. Ore particles ranging in size from 0.02 to 0.2 are most successfully separated by flotation *mm*. During flotation of coal sludge, graphite, potassium salts, particles with a particle size of up to 0.5-1.0 can be floated *mm*.

The maximum size of floated mineral particles depends on their hydrophobicity, density and shape. The source material should not contain either large grains, the flotation of which is impossible, or fine sludge (particle size less than 0.02*mm)*, the presence of which impairs the separation of minerals and increases the consumption of reagents.

The optimal grinding size is determined to a first approximation experimentally in laboratory conditions.

*Pulp Density*is one of the important factors influencing flotation, and represents the ratio of solid and liquid phases of the pulp.

Solid content in pulp*R, %,*determined by the formula

$$P=\frac{T}{T+F}100, \qquad (14.1)$$

Where *T*- mass of solid per unit volume of pulp; W is the mass of water in the same volume; T+W - mass per unit volume of pulp.

The solid content of the pulp in flotation practice ranges from 15 to 40%. In some flotation operations, preference is given to a more liquefied pulp; in others, on the contrary, the pulp is thickened.

With a high solid content in the pulp, its degree of saturation with air bubbles decreases, the flotability of large mineral particles worsens and the quality of the concentrate decreases as a result of more intense flotation of fine particles of waste rock (Figure 14.1). Flotation in liquefied pulp is carried out when it is necessary to obtain a high-quality concentrate.

Promotion*pulp temperature*in most cases it has a positive effect on flotation. At the same time, the solubility of a number of reagents (especially fatty acids and soaps) increases and their consumption decreases. At the same time, when using xanthates as collectors, such an effect is not observed and heating the pulp in this case is advisable only in winter.

During the flotation of sulfide minerals, oxidation processes and foaming can be controlled by changing the temperature of the pulp. Typically, cold water flotation requires more foaming agent.

*Water composition*significantly affects the flotation process, since it There may be various ions, dissolved gases, various colloidal and organic impurities that affect flotation and change the pH of the water.

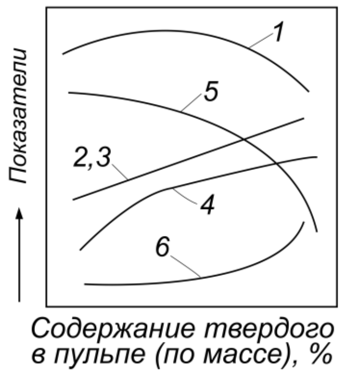

Figure 14.1 - Influence of pulp density on flotation:

1 - pulp aeration; 2 - volume concentration of reagents; 3 - residence time of the pulp in the machine; 4 - floatability of small grains; 5 - floatability of large grains; 6-grain abrasion

The main components found in natural waters are the following ions are: chloride*WITHI*-,sulfate*SO*2- 4,bicarbonate *NSO*3etc. The main dissolved gases are oxygen, carbon dioxide and hydrogen sulfide present in water in molecular form.

There are two main mechanisms for the influence of unavoidable ions on flotation:

• inevitable ions, fixing on mineral surfaces particles, change their flotation properties;

• unavoidable ions can react chemically with flotation reagents. For example, calcium ions form poorly soluble calcium soaps with fatty acids, and heavy metal ions with alkali metal xanthates form poorly soluble xanthates of the corresponding heavy metals.

Undesirable influence of unavoidable ions and gases on flotation is prevented and controlled by changing*pH*and application appropriate reagents. For example, soda and some other alkalis remove calcium and iron ions into insoluble compounds.

*Reagent mode during flotation*determined by assortment the reagents used, their consumption, the order of supply to the process and the duration of contact of the reagents with the pulp. The reagent regime is established experimentally based on the study of the flotation properties of the minerals of a given ore, the size of mineral particles, the composition of water and other factors.

The supply of reagents to the process occurs in the following sequence: first, regulator reagents are added to the pulp*pH*media, then reagents - activators and suppressor reagents, then collector and lastly foaming agent.

Pulp aeration- this is the process of saturation of flotation pulp air bubbles, carried out in flotation machines, air conditioners or other devices, by sucking it from the atmosphere or supplying it under pressure.

In industrial conditions, aeration is characterized by the air flow rate passing per unit time through a unit area of the horizontal section of the flotation chamber [$l/(m^2min)$] or through a unit volume of pulp [$l/(m^3min)$]. In different types of flotation machines it usually varies from 600 to 1300 $l/(m^2min)$.

Bubble size during flotation must satisfy the following conditions:

• lifting force of a mineralized bubble with a volume V_P must be more inertial forces F_i anti-rise;

• density of the mineralized vesicle P should be less pulp density and, because otherwise it will not float up;

• rate of rise of mineralized bubbles must be optimal and equal according to practice (5-15)10^{-2} m/s. At a lower speed, the bubbles will not have time to float to the surface of the pulp and most of them will go into the tails. For a high rate of rise of mineralized bubbles, large bubbles are required, which is associated with a decrease in the interphase surface and the probability of flotation;

• the ratio of bubble and particle sizes should ensure maximum probability W_c their collision and formation of a complex bubble - particle.

Thus, the minimum size of transport bubbles during flotation of grains of normal size must be at least 0.6 *mm*, and during flotation of grains of critical size at least 1-2 *mm*. It is desirable to have both very small bubbles, for example, bubbles released from the solution (for activation), and large “transporting” bubbles, at the same time in the pulp.

The kinetics of the flotation process is characterized by the dependence of the extraction of the flotating mineral 휀 from time to time *t*, i.e. 휀 $=f(t)$. Derivative d휀$/dt$ equal to the flotation speed at a given time and is determined tangent of the slope of the curve $\varepsilon=f(t)$.

Flotation speed determines the performance of flotation machines and allows you to analyze the influence of flotation conditions on the process.

14.3. Operation of flotation departments

Starting and stopping the flotation department. Before launch In the flotation department, the technologist checks the condition of the flotation machines, raises the thresholds, checks the plugs of the machines, the reliability of the transmission, the serviceability of the electric motors, and starts up the thickening and filtration equipment. Then he starts up the flotation machines, contact tanks and pumps, and only then starts the grinding department.

Depending on the complexity of the technological scheme in each individual case, the startup sequence of the section equipment is determined. Typically, cleaning flotation machines are started first, then middling, control and main flotation machines, starting with the last cells.

Once the pulp enters the contact tanks or directly into the flotation machines (where there are no tanks), it is necessary to feed the reagents into the tank or head chamber of the flotation machine. As the fresh pulp moves through the flotation circuit, other reagent feeders are turned on. When foam enters the chutes, the valves are opened, and water is supplied to the chutes in an amount that ensures normal pulp density in all operations of the flotation circuit.

Stopping the flotation department happens after stopping grinding departments. When the flow of pulp into the flotation stops, the reagent feeders are stopped in the reverse order of start-up. Water is stopped being supplied to the gutters as the foam stops flowing. As the end products stop coming out of the machines, subsequent devices are stopped, and the order is observed from the beginning to the end of the process. The pumps are stopped last. If the ore being processed produces dense sediments, then the flotation machines run idle for some time before stopping - they are exhausted. When the machines are fully used, it is necessary to maintain the required level of pulp in them for some time to ensure normal foam removal.

When stopping flotation machines, you must follow the reverse sequence: first you need to stop the main flotation machines, then the control, middling and cleaning flotation machines.

Regulation of the flotation department. When regulating During the operation of the flotation department, the technologist is guided by the regime map and technological instructions. At the same time, the technologist should take into account the following general provisions for regulating and conducting the process:

- the pulp must enter the flotation department evenly, in certain quantities; sharp fluctuations in the tonnage of processed ore and the volume of pulp cause overflows of pulp from flotation machines, lead to depletion of the concentrate and increase metal losses in the tailings;
- after starting the flotation machine, do not rush to raising the pulp level immediately after its arrival, since after some time the pulp begins to overflow into the gutters. If it is necessary to raise the levels in the chambers of the flotation machine, you must start from the last chamber, moving towards the head one. Lowering the level is done in reverse, starting from the first chamber and ending with the last;
- the highest level of pulp is usually maintained in machines main and control flotation to ensure higher recovery of useful materials in these operations. It is necessary to strive to ensure that the pulp level in these machines is such that

On the surface of the pulp, a layer of foam 40-75 mm high could form. The lower the height of the foam layer, the more particles of weakly floating materials and aggregates are entrained into the concentration trench, the higher the extraction of metals into the coarse concentrate.

Since a high level of pulp in main flotation machines is sometimes more difficult to maintain than the thickness of the foam layer, the thickness of the foam layer is regulated by changing the height of the drain thresholds for it. If the flotation machine has a device for regulating the air supply, then in the machines the main' For flotation, it is better to open the air channel completely. Removing foam by overflowing is harmful, since it depletes the foam products and increases the circulating load.

A lower level of pulp and a relatively high layer of foam is maintained in flotation cleaning machines, since high layers of foam contribute to better separation of mineral particles and produce higher quality concentrates. By regulating the air supply and the pulp level, it is possible to achieve a foam removal mode that will ensure the production of a concentrate of the required quality.

Very careful skimming should be done in the last chambers of each cycle of the main or control operation during selective flotation of polymetallic ores in order to avoid increased recovery of minerals subject to flotation in subsequent operations.

The flotation process in flotation machines is regulated in the following approximate sequence:

- during main flotation, first adjust the height of the drain thresholds for foam, then the air supply and pulp level in the machine or chamber and, finally, the supply of reagents (foaming agent);
- during cleaning flotation, the air supply is first regulated, and then the pulp level and then the drain thresholds for foam.

External signs of the normal course of the process are:

- in main flotation- in the first four to five chambers it is formed abundant, large bubbles (diameter of bubbles approximately 15-30*mm),*

- well-loaded foam with a metallic sheen. When draining from When put into a gutter, such foam breaks down well. Gradually, the size of the bubbles and their load decrease, and in the last chambers the foam becomes even less loaded;
- V*control flotation*- layer and load of foam in machines less. Loaded foam is present only at the beginning of the machine, then it becomes watery, translucent and at the end of the machine - transparent, below average coarseness;
- V*cleaner flotation*– in machines a well-loaded foam with bubbles of equal size.

You should not abuse the change in the supply of foam concentrate to regulate the foam output. This is only used in cases where regulating the operation of the flotation machine by other methods does not provide the required product yield.

If the flotation process is disrupted, the technologist must first find out the cause of the violation and then take measures to adjust the process.

Control questions

1. What parameters affect flotation efficiency?
2. How does the material composition of ore affect the outcome of flotation?
3. What are the requirements for ore size during flotation?
4. How does sludge affect the flotation process?
5. What do you mean by pulp density?
6. What formula is used to determine the solid content in the pulp?
7. How does pulp temperature affect flotation efficiency?
8. How does the composition of circulating water affect the outcome of flotation?
9. What is the essence of the reagent regime during flotation?
10. What is the procedure for supplying reagents to the flotation process?
11. What are the requirements for the size of pulp bubbles when flotation?
12. What is the procedure for starting up flotation equipment?
13. What is the procedure for stopping flotation equipment departments?
14.

Lecture No. 15 SPECIAL ENRICHMENT METHODS

Chemical enrichment. Radiometric enrichment. Enrichment by physical and chemical properties of minerals

*Key terms:*chemical enrichment, hydrochemical processes, chemical dissolution, leaching, pressure leaching, vat leaching, heap leaching, electrolysis, cementation, in-situ leaching, bacterial leaching, radiometric enrichment

15.1. Chemical enrichment

Chemical enrichment- area of technology for processing minerals fossils according to combined schemes, including chemical processes at the beginning, middle or end. The following processes are used: hydrochemical, thermochemical, pyrometallurgical, chloride and fluoride sublimation, sulfatizing, reducing, oxidative, segregation roasting, etc. Hydrochemical processes have received the greatest industrial use.

Hydrochemical processes- these are the processes of extracting valuable components or removing harmful impurities from ores and enrichment products by selectively dissolving them with aqueous solutions of chemical reagents. With the subsequent isolation of valuable components from the solution, high-quality products are obtained, often called chemical concentrates. In this case, physical and chemical dissolution are distinguished.

*Physical dissolution*is a process that occurs without changes in the composition of the soluble component or mineral.

*Chemical dissolution*is a process accompanied by change chemical composition of mineral components.

Physical dissolution is the basis for geotechnological methods for the extraction and processing of water-soluble salts: halite, sylvite, bischofite, etc. Chemical dissolution is used in the processes of extraction and processing of metals, their salts and oxides. The solvents used are oxygen (sulfuric, nitric, phosphoric, sulphurous) and oxygen-free (hydrochloric, hydrogen sulphide) acids, aqueous solutions of salts (soda, sodium sulfide, sulphate salts of alkali metals). The result of dissolution is the formation of a solution (in relation to the physical dissolution of salts - brine).

The processes of chemical dissolution are based on exchange reactions, redox reactions, complex formation, etc.

Physical and chemical dissolution is characterized by the presence of three stages of the process:

• supply of solvent to the surface of the mineral;
• interaction of solvent with mineral;
• removal of reaction products from the phase interface. The determining stage of dissolution kinetics can be any of the above.

Chemical enrichment includes the following processes: grinding of the source material; decomposition of ore minerals or their preliminary thermal treatment; mineral leaching; separating the solution from the solid and washing the precipitate; purification of the resulting solutions from impurities; separation and precipitation of extracted metals; drying and roasting of final products or remelting of electrolytically deposited cathode metal; regeneration of reagents and additional extraction of metals from waste solutions.

*Leaching*is the process of extracting one or more components from solid products (ores, concentrates, intermediate products, sometimes industrial waste) with an aqueous solution containing an alkali, acid or other reagent, as well as using certain types of bacteria.*Leaching is the main operation hydrochemical process.*

Leaching is used in the technology of extraction of non-ferrous (Al, Au, Cd, Cu, In, T1, etc.) and rare (Be, Li, Mo, Nb, Re, W, Ta, U, etc.) metals, with receiving B, Ge, Se, Te. Heap leaching is used to extract Ag, Au, Cu, U, underground - U, Ca, Ag, Co, Fe, Ni, etc., bacterial - Cu and U; the latter method is promising for processing silicate ores containing Al, As, Mn, Ni and other metals.

For leaching of ore minerals, solutions of sodium carbonate, ammonium carbonate, ammonia, potassium cyanide, caustic soda, sodium chloride, sulfuric, hydrochloric and nitric acids are used. Solvent concentrations range from 0.02% KCN for gold leaching to 30% HC1 for decomposition of scheelite concentrates or up to 94% $H_2SO_4$for decomposition of ilmenite, perovskite and niobium concentrates.

The extraction of metals into solution can reach 98-99% with insignificant dissolution (3-5%) of associated minerals. An increase in the leaching rate is achieved by increasing the temperature of the solution, using additives of oxidizing agents, reducing agents, bacterial cultures and their metabolic products, applying magnetic and electric fields, vibroacoustic and other influences.

Leaching is carried out *percolation, heapChanov, autoclave, and underground methods.*

*During vat (agitation) leaching*ground to50-90% class -0.074*mm*the ore passes through a series of series-connected vats, in which intensive mixing of the pulp is carried out by mechanical mixers, compressed air or a combined air-mechanical method.

A vat with a vertical mixing device (Figure 15.1) consists of a body 1 with an acid- and alkali-resistant lining 2 and a stirrer 3. The mixing device lifts the pulp in the middle part of the vat, while downward flows are observed along the periphery. When the pulp circulates, active contact of solid particles with fresh portions of the solution and intensive dissolution occurs.

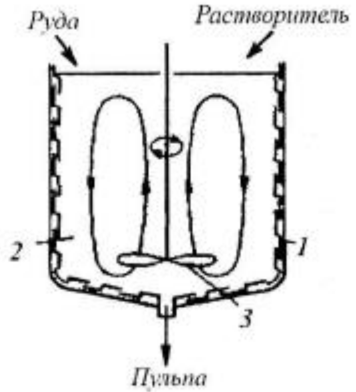

Figure 15.1 - Schemes of a vat with a mechanical mixing device type:
1 - body; 2 - lining; 3 - mixer (auger)

Autoclave leaching has found wide application in uranium, nickel, aluminum and tungsten industries. Autoclaves are metal vertical or horizontal vessels with a capacity of 5 to 130*m*, operating under pressure from 1 to 5*MPa* when heating the pulp through external heaters (steam jackets, electric heaters) or by blowing it in an autoclave with live steam, which also provides intensive mixing of the pulp. Increasing the temperature and pressure in the autoclave causes a significant increase in the leaching rate. To oxidize the leached components, for example, sulfides, oxygen, air, hydrogen peroxide, manganese salts, chlorine, etc. are supplied to the autoclave. Autoclaves are usually combined into batteries consisting of 5-10 devices, operating on the principle of continuous leaching of raw materials. The autoclave diagram is shown in Figure 15.2.

The high temperature and pressure in autoclaves increases the rate of chemical reactions, which makes it possible to carry out processes that under normal conditions proceed extremely slowly and incompletely. In addition, the tightness of the equipment reduces the loss of reagents and eliminates environmental pollution.

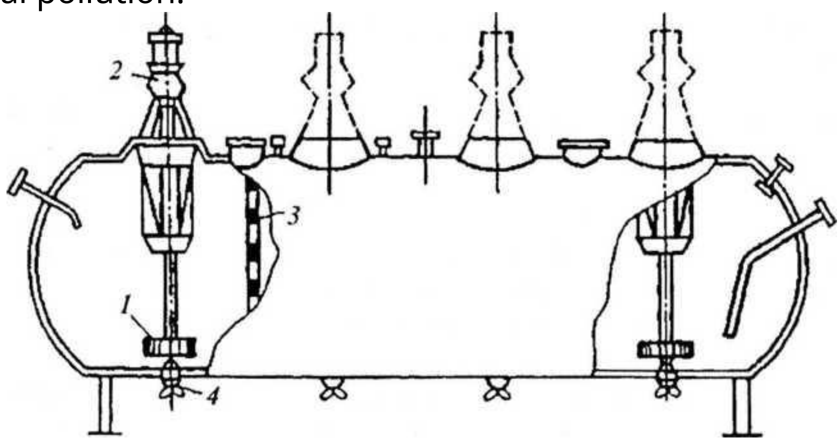

Figure 15.2 - Horizontal autoclave for ore leaching: 1 - impeller; 2 - impeller drive; 3 - partition; 4 - outlet valve

Percolation method leaching is carried out in vats without stirring. For leaching, vats with a capacity of 5-10 *thousand tons*. The length of the vats is up to 50*m*, width - up to 30-33*m* and depth - up to 5.5*m*.

The percolation method is used to process crushed, de-slimed ore with a particle size of -15÷1*mm*.

The percolation process of leaching in vats without stirring is used mainly for oxidized copper ores; it can also be used in combined technology for processing mixed (oxidized and sulfide ores).

The time for a full cycle of work on leaching copper ores is 8-13*days* (loading, leaching, washing, unloading). More than half time is allocated to the main operation - leaching. Copper recovery reaches 75-90% with sulfuric acid consumption in the range of 10-55*kg/t*.

Heap leaching(KV) recycle poor, off-balance sheet ores and overburden dumps with a low content of metals: copper (0.15-0.5%), gold (about 0.5*g/t*) or uranium (0.02-0.07% U_9O_8). In heap leaching, the ore is crushed to 120-400*mm*and is placed in the form of a stack (heap) up to 60 *m*, width up to 200*m*and up to 800 m long on a special waterproof platform (Figure 15.3).

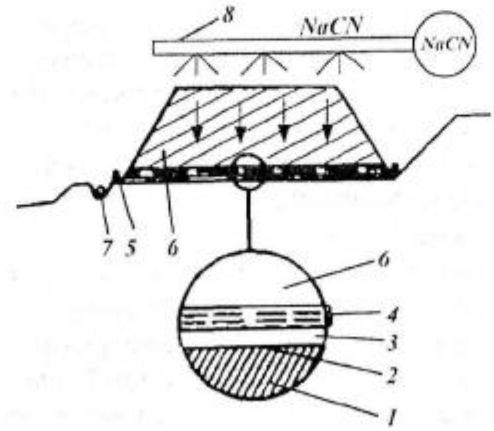

Figure 15.3 - Typical structure of a heap;

1 - foundation of the coating; 2 - polymer film; 3 - layer of sand (up to 50-100*cm*); 4 - protective coating (up to 50*cm*) with perforated pipes; 5 - berm (height up to 60*cm*); 6 - heap (stack); 7 - drainage trench with pipe; 8 - irrigation system

A layer of gravel, crushed ore, a mixture of clays and the sludge fraction of the mill tailings, bentonite clays and other materials are used as the base for the HF sites. The height of the base layer ranges from 100 to 600 *mm*. A waterproof coating (asphalt, plastic made from welded strips of acid-resistant polymer films) is laid on the base to prevent the loss of working and productive solutions and to ensure measures to protect the environment from pollution.

The surfaces of the stacks are irrigated with leaching solutions. Irrigation is carried out in various ways: by spraying. ponds, injection wells, irrigation ditches, or a combination of different methods. A solution of sulfuric acid is used as a solvent for leaching copper and uranium (*pH* 1.2-2.5), when leaching gold and silver - cyanide solution (*pH*10-10.5).

The leach solution percolates through the heap, becomes saturated with the recovered metal and is collected in a settling basin to settle clay and slurries before being sent to metal recovery.*Heap leaching*is a lengthy process and the resulting solutions characterized by a low metal content. Therefore, usually the entire cycle of saturation, collection and processing of the solution is repeated until the metal content in the solution will not reach the critical minimum value. For example, the metal concentration in uranium-containing solutions is usually 0.3-3.0*g/l*with pH 1.2-1.4. With a leaching duration of 30-80 days, the recovery of uranium and gold is 60-80%.

After heap leaching, the rocks are taken to dumps, and if the heaps are large, they are left in place and subjected to reclamation.

In some cases*heap leaching*gold and uranium ores are economically much more effective than other methods of extracting these metals, especially when developing deposits with insignificant ore reserves, as well as when processing old dumps or poor off-balance ores.

*Heap leaching*is the most suitable method for mining of small deposits of gold and silver due to low capital and operating costs, the possibility of effective environmental control and flexible management during operation. The ore should be accessible to cyanide solutions, contain little carbon that absorbs gold-cyanide complexes, and there should not be an excess of fine ore fractions that prevent the solutions from seeping through. Inexpensive waterproof substrates that can be quickly installed on site and an inexpensive process for extracting Au from activated carbon (resin) solutions are essential for the process to be successful.

Typically in the US, Au is leached from large heaps at a grade of 2.2 *g/t*Au. The ore is poured directly onto a waterproof film, then the top is sprinkled with a cyanide solution. Au extraction is carried out in absorption columns filled with charcoal, followed by gold desorption and coal regeneration. Gold is desorbed in hot alkaline solutions and recovered by electrodeposition. Leaching time is 60 days with 70% Au recovery.

The main factor determining the efficiency of the process is the permeability of the ore layer. One of the methods for organizing the permeability of the heap is to first fill out meter-high cones of ore. Subsequent layers of ore can be laid in the form of a heap with a height of more than 5 m. The base of the heap, according to practice, is more expedient to prepare from a disposable plastic waterproof film than on an asphalt base with a thickness of 200*mm*reusable.

*In-situ leaching*carried out at submission leach solution underground directly into the ore body or into a layer of specially prepared ore. The solution, which has seeped through the ore layer and is saturated with a valuable component, is pumped to the surface. There are two main options for underground leaching - borehole (shaftless) and mine. In the first case, a system of wells located in a certain way is used to supply the leaching solution and pump out the production solution, during the second - old or specially created mines, prepared underground chambers with collapsed ore, and for collecting the production solution - adits or drifts. When developing deposits of radioactive and non-ferrous metal ores, combined underground leaching systems are often used, including elements of borehole and mine systems.

The necessary conditions for the use of underground leaching are a sufficiently high permeability of the ore and the presence of an impermeable layer under the leached area, which ensures the collection of solutions sent for the extraction of metals. In other cases, special types of preparatory work are performed to create such conditions.

In-ground leaching is usually used when the depth of the ore body is no more than 600*m.*The use of the underground leaching method allows: to sharply reduce the volume of capital investments and the construction time of enterprises; increase labor productivity by 2-4 times; significantly reduce the harmful impact on nature (do not disturb the landscape, sharply reduce the amount of solid waste and harmful substances carried to the surface of the earth, it is relatively simple to restore waste areas).

In some countries, up to 10% U and up to 18% Cu are extracted using underground leaching.

*Leaching solutions are separated*from solid matter by condensation into ponds, vats and thickeners, filtration on vacuum filters or filter presses of various designs.

*Isolation of metals and their compounds from solution*carried out various methods. The most common of them are: electrolysis, cementation, production of insoluble compounds, hydrolysis, crystallization, sorption, extraction.

The choice of the most rational method must be decided in each individual case, taking into account a number of factors, of which the composition of the solution supplied for precipitation and the requirements for the purity of the final product are of paramount importance.

At*bacterial leaching*The ability of autotrophic bacteria (Thiobacillus ferrooxidans, Ferrobacillus thiooxidans, etc.) to absorb for their life activity the energy released during the oxidation of sulfides and thiosulfates of metals, sulfur, as well as during the transition of Fe^{2+}in Fe^{3+}. These bacteria contain substances that catalyze these reactions. As a result, H is formed$_2S0_4$or Fe salts^{3+}, which can be used as leaching reagents. The greatest activity of bacteria is observed at 30-35*°C.*

A prerequisite for bacterial leaching is normal pressure and temperature of the environment.

15.2. Radiometric enrichment

Radiometric enrichment- this is radiation enrichment, based on the separation of the source material as a result of measuring some type of radiation from the separated components. The starting material for radiometric enrichment is a mechanical mixture of solid particles that differ in the type and intensity of radiation.

The processes of radiometric enrichment (separation) include processes of separation of minerals based on differences in the intensity of emission, reflection or absorption of various types of nuclear physical radiation by mechanically changing the trajectories of the removal of particles from the material flow by special actuators.

For ores with natural radioactivity, differences in the radioactive properties of minerals are used; for radiometric enrichment of non-radioactive minerals, differences in the interaction of minerals with various radiations are used. In the latter case, radiation in a wide range of wavelengths is used as primary radiation. waves: -radiation ($<10^{-2}$*nm*); (-radiation ($=10^{-3}\div10^{-2}$ *nm*); neutron ($=10^{-2}\div 10^{-1}$*nm*) etc.

Any of these properties or any combination of them can be used when separating mineral particles.

In principle, radiometric separation can be represented as follows (Figure 15.4).

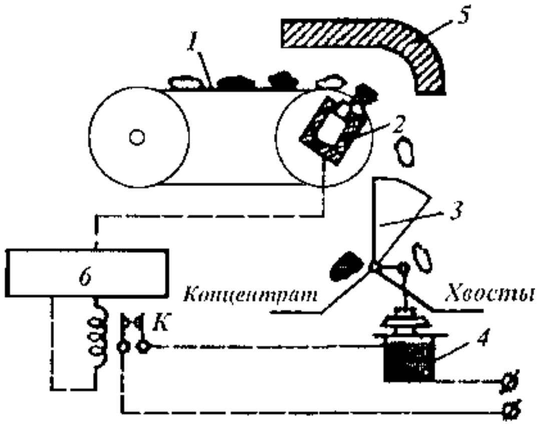

Figure 15.4 - Schematic diagram of radiometric enrichment:

1 - radiation source; 2 - object (portion, particle); 3 - radiation receiver; 4 - information processing unit; 5 - actuator for removing objects (portions, particles) from the flow; 6 - trajectory of movement of objects (portions, particles); 7 - trajectory of movement of the concentrate; 8 - trajectory of tails movement

Pieces of material 2 move in a monolayer along a belt or vibrating tray 6. Under the influence of external radiation from source 1, for example gamma radiation, pieces of material emit secondary radiation of varying intensities.

The intensity is measured using a special device 3, which transmits signals through an amplifier 4 to an actuator 5, which automatically separates pieces or fractions with increased or, conversely, decreased radiation intensity. As a result, the concentrate and tailings change their movement trajectories and are unloaded into the corresponding receivers.

Currently time known over twenty methods radiometric enrichment. About half of them are already used in industrial settings or are in preparation for implementation.

15.3. Enrichment by physical and chemical properties of minerals

*Enrichment in friction and shape*based on the use of differences in the speeds of movement of separated particles on a plane under the influence of various forces. The speed of movement of particles along an inclined plane (at a given angle of inclination) depends on the state of the surface of the particles themselves, their shape, size, humidity, density, properties of the surface on which they move, the nature of the movement (rolling or sliding), as well as the environment in which separation occurs. Particles can move under the influence of gravity (when moving along an inclined plane), centrifugal force (when moving along a horizontal plane of a rotating disk) and as a result of the combined action of gravity, centrifugal and friction forces (screw separators, see below). Effective separation by this method requires a narrow classification of the material by particle size.

Typically, friction and shape enrichment is used for fossil raw materials with a particle size of 10-100*mm*and is carried out in devices with a fixed (inclined planes, screw separators) and movable (drum, belt, disk, vibrating separators and screens) working surface. For example, planar separators are used for mica enrichment, screw separators are used for enrichment. mica, wolframite, cassiterite and magnetite, belt - for separating fine abrasive powders into fractions of different shapes and separating fine technical graphite from mica plates, vibration - for separating particles of powdered materials by size.

*Enrichment on fat*surfaces (fat process) based on selective fixation of certain minerals on a surface covered with a layer of fat. As pulp minerals flow through the fatty coating layer, hydrophobic particles adhere to it, and hydrophilic particles are removed by the flow of water into the tailings. This process is mainly used in finishing operations primary (rough) diamond concentrates released during the enrichment of diamond-containing raw materials. Fat coatings are mixtures that, depending on the properties of the ore and the temperature of the water, contain oil and machine oil in different proportions, sometimes Vaseline, paraffin, etc. The process is carried out on so-called fat tables. The continuously operating table is equipped with an endless rubber belt 1 m wide, stretched over two drums, which are mounted on a frame mounted on spring supports; the table can oscillate in the flow plane; the layer of fat with adhering particles is removed with a scraper under the unloading drum.

*Other methods.*For the enrichment of minerals they use Also *following methods*: photoneutron (beryllium ores), photometric (gold ores and non-metallic materials), photo- and x-ray luminescence (diamonds), gamma absorption (iron ores), neutron absorption (boron ores), etc. Differences in the physicochemical properties of the surfaces of the separated materials underlie flotation enrichment. A special type of physicochemical enrichment based on selective wetting of materials with mercury is amalgamation, used primarily. for the extraction of precious metals

All of the listed operations and methods of separation and concentration are used individually and in different combinations. Some general expenditure indicators (per 1 *T*ore to be processed): 2-4m^3water; quantity recycled water (in closed water supply systems) 90-95%; electricity costs 0.3-0.7*kWh*.

Control questions

1. What are hydrochemical processes based on?
2. What stages do hydrochemical processes consist of?
3. What processes does chemical enrichment include?
4. What is leaching?
5. How can the leaching rate be increased?
6. What leaching methods are there?
7. How is vat leaching carried out?
8. What is the advantage of autoclave leaching?
9. What is the essence of the percolation method? leaching?
10. How is heap leaching carried out?
11. What is the essence of in-situ leaching?
12. How are leaching solutions separated from solids? substances?
13. How metals are isolated and combined from solution?
14. For which ores is radiometric enrichment used?

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13. How metals are isolated and combined from solution?
14. For which ores is radiometric enrichment used?

15. What types of enrichment according to physical and chemical properties minerals do you know?

Section 4. Auxiliary Processes

Lecture No. 16 CONDENSATION

General information and classification of thickening processes. Drainage. Thickening.

Basic terms: thickening, dehydration, humidity, drainage, centrifugation, coagulation, flocculation, radial thickener, desliming, magnetic deslimers, plate thickener

16.1. General information and classification of thickening processes

Dehydration called the process of removing water (moisture) from products containing solid and liquid phases.

In the practice of enriching useful fossils greatest processes occurring in the aquatic environment are widespread. The final beneficiation products (concentrates and tailings) typically contain a lot of moisture.

Humidity W,%, determined by the ratio of the mass of water in the product to weight of wet starting product:

$$W=100(Q_1-Q_2)/Q_1 \quad (16.1)$$

Where Q_1- mass of wet product; Q_2- mass of dry product.

Depending on the moisture (water) content, products are divided into liquid (watered), wet, moist, air-dry, dry and calcined.

Liquid products (for example, pulp) are characterized by high liquefaction and fluidity. They contain at least 40% moisture.

Wet products contain less water (from15 to 40%) than liquid. Characterized by free flow of water.

Wet products are intermediate between wet and air-dry. Their moisture content ranges from 5 to 20%. They are not fluid.

*Air-dry*products retain hygroscopic moisture adsorption forces on the surface of particles in the form of a molecular film. The content of moisture retained by the material is determined by the physical and physicochemical properties of the substances (porosity, wettability, etc.).

*Dry*products contain only internal (constitutional) moisture).

The increased humidity of concentrates creates great difficulties during their transportation to metallurgical plants (especially in winter), increases transportation costs and increases the cost of their further metallurgical processing. The maximum moisture content of ore concentrates shipped by rail in winter is, %: for magnetite ore concentrates - 2-4; hematite and martite ores -3-5; brown iron ores -4-6; flotation concentrates of non-ferrous metal ores -5-12; For coal concentrates, the humidity is set to no more than 5% in winter and no more than 8-10% in summer.

In most cases, waste tailings from processing plants are also heavily watered, which increases the cost of their delivery to the tailings storage facility (in cases where there is no gravity transport) and requires the creation of large-capacity tailings storage facilities.

When factories organize full water circulation, water is separated from the solid phase and, if necessary, subjected to chemical or other treatment.

The following processes are used to dehydrate enrichment products:*drainage, thickening, filtration, centrifugation and thermal drying*.

An operation is sufficient to dewater coarse material. *drainage*, in which excess water is removed by gravity. Moisture is more difficult to remove from fine-grained material and several successive operations are required to dehydrate it. For dewatering such material, the following typical scheme is usually used:*condensation to humidity 30-50%; filtering to obtain a precipitate with a moisture content of 10-15% and drying.*Depending on the consumer's requirements for concentrates or technologies for further enrichment of industrial products at the factory, the moisture content of the dried material ranges from*0.5 to 5%.*

16.2 Drainage

*Drainage*called the process of removing gravitational moisture from lump and coarse-grained material by natural moisture seepage under the influence of gravity in the spaces between individual grains. Drainage is carried out in dewatering elevators, mechanical classifiers, screens, dewatering bins and drainage warehouses.

Dehydrating bucket elevators apply For transportation and dewatering of enrichment products obtained in jigging machines, and for dispensing products from settling tanks (for example, from bagger sumps). They are installed at an angle of 60-70° to the horizon and produce material with a moisture content of 25-30% and below. Used for dehydration of products with a particle size of at least 2*mm.*

Dehydration of products in mechanical classifiers occurs as they are transported along the bottom of the classifier. Depending on the size of the material and the operating mode of the classifier, it is possible to obtain products with a moisture content of 15-25%.

For the dewatering of products in a wide range of sizes - from lumps to sludge - the same types of screens are used as for screening. Slot-shaped sieves made of profiled brass or steel wire are most often used as the working surfaces of dewatering screens. Moving the material along the screen screen significantly intensifies drainage.

*Dewatering in silos*carried out mainly on coal preparation factories. These bunkers are rectangular (in plan) reinforced concrete cells with a pyramidal lower part and outlets equipped with special gates with perforated holes for water drainage.

*Drainage warehouses*are large buildings containers made of concrete and reinforced concrete, with a sloping bottom in which drainage ditches are laid.

They are used to remove moisture from the material after its preliminary thickening in settling tanks. Dehydration time - up to 24*h* and more. As a result of dehydration, humidity is reduced from 30 to 8.5%.

16.3 Condensation

By condensation called process dehydration watered fine-grained products by settling solid particles and releasing the liquid phase in the form of clarified overflow. The process is carried out in various types of settling tanks, thickening funnels and cylindrical thickeners.

When condensed into*settling tanks, conical and cylindrical thickeners* (drawing14.1) sedimentation occurs under the influence of gravity, while under steady-state conditions a clarification zone is formed in the upper layer of the thickener*A*, in the middle part there is a deposition zone*B*and below is the sediment compaction zone*G*. Sometimes an intermediate zone is distinguished between the last two zones*IN*. In the clarification zone, the movement of solid particles occurs in liquefied pulps according to the laws of free fall in an aqueous environment at a speed depending on the size and density of the particles. IN in the middle zone, particles accumulate, resulting in cramped falling conditions. In this case, small particles delay the deposition of larger ones, their falling velocities are equalized, and the particles in this zone are deposited in a cohesive mass. At the bottom there is a sediment compaction zone. In this zone, water is squeezed out of the sediment under the pressure of overlying particles and moves from bottom to top, the rate of sedimentation of particles practically becomes zero, the density of the sediment reaches a maximum and amounts to 43-44% by volume.

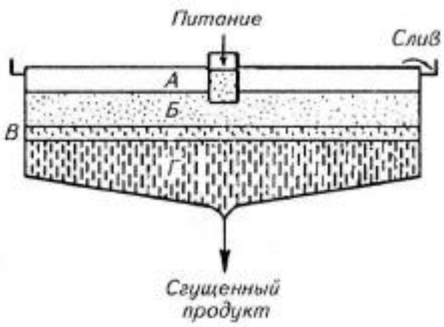

Figure 16.1 - Zones of pulp deposition in thickeners

Specific thickening area (in *m_2* on 1 *T* solid material per day) can be determined by the formula

$$s=(R-\ )/(\ ), \quad (16.1)$$

Where *R*-pulp liquefaction corresponding to the defining solid content; - liquefaction of the final (condensed) product; - rate of solid deposition during liquefaction *R*, *m/day*; - density of liquid pulp phases*, t/ m_3*; *R*, and - are determined experimentally.

Thickener depth determined by the formula:

$$H = h_1 + h_2 + h, \quad (16.2)$$

Where *h_1, h_2*, And *h*-height, respectively, of the zone of clarified liquid, zone pulp initial density (h_1= 0.3÷0.9 *m* and H_2=0.3÷0.6 *m)* and compaction zones.

The height of the compaction zone is determined by the formulas:

$$h=t(1/\ +R_{Wed}/\ )/(24s), \quad (16.3)$$

Where *t*- residence time of the solid phase in the compaction zone, *h*; -density solid, *t/m_3*; *R_{Wed}*-average liquefaction in the compaction zone (determined experimentally).

To accelerate thickening, special reagents are added to the pulp to cause coagulation or flocculation, i.e. sticking together of tiny mineral particles and the formation of relatively large, rapidly settling aggregates.

Most famous flocculant reagents are synthetic high-molecular substances: polyacrylamide, hypane, separan, aeroflocs, etc.

Cylindrical (radial) thickeners are most widely used for thickening various products at domestic processing plants. At iron ore factories processing magnetite ores, magnetic deslimers are used to desludge the pulp before magnetic separation.

Radial thickener (picture 16.2) represents cylindrical vat with mechanical unloading of condensed sludge. Depending on the design of the drive mechanism, thickeners are available with peripheral and central drive. Thickeners with a central drive can be single-, double- or multi-tiered, i.e. with one or more tanks installed one above the other, with a common central drive.

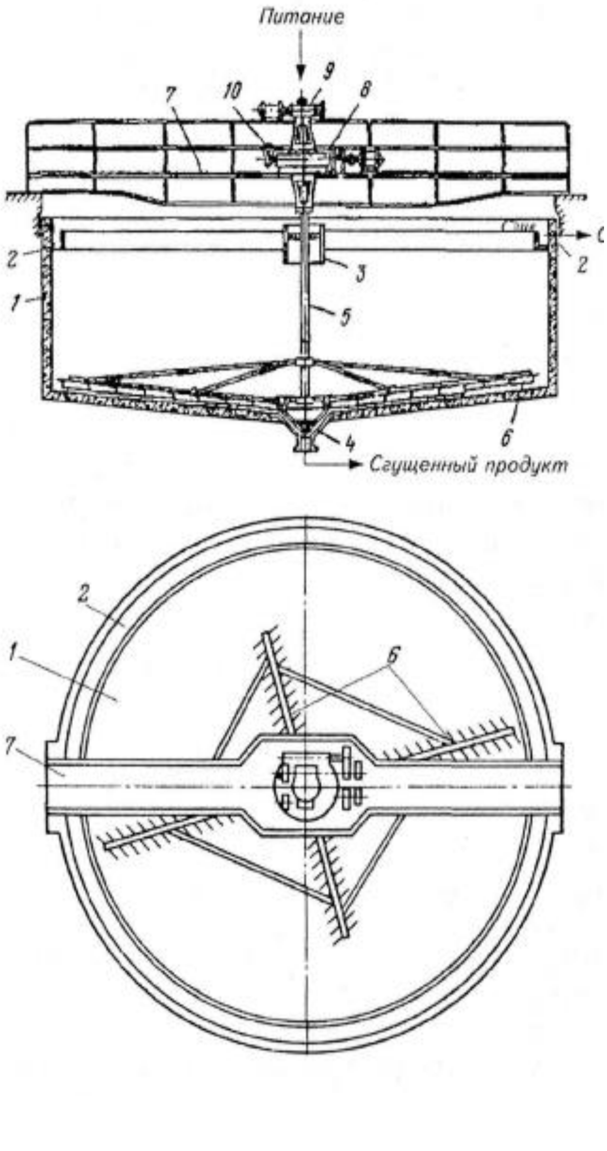

Figure 16.2. Single deck radial thickener with central drive:

1-chan; 2-drain gutter; 3-loading funnel; 4-discharge hole for condensed product; 5-central shaft; 6-rakes, 7-installation platform; 8-wheel drive; 9-lifting mechanism, 10-thickener overload indicator.

Radial thickeners with a central drive are available with a tank diameter from 2.5 to 18*m*; Thickeners with peripheral drive have a tank with a diameter of 18 to 100*m*and more.

The initial pulp is fed through a slurry line into the thickener through a loading funnel, the lower edge of which is below the pulp level in the vat. The solid particles contained in the pulp settle to the bottom of the vat, and the clarified water flows through the drain threshold into the annular peripheral chute, from where it enters the clarified water sump. The thickened sludge, using a rake frame, continuously moves along the bottom of the vat from the periphery to the center, where it is unloaded by a pump and then transported through slurry pipelines for subsequent processing in accordance with the technological process diagram.

The rotation of the rowing frame is carried out respectively from the central or peripheral drive.

*Magnetic dirt removers and hydroseparators*according to its design are in many ways similar to radial thickeners. They differ from them in that a magnetic system is built into the power box of these devices. Under the influence of the created magnetic field, magnetite particles contained in the incoming pulp are magnetized, enlarge, forming flocs, and quickly settle, and non-flocculated thin non-magnetic particles are removed by draining into a peripheral annular chute. Magnetic dirt removers are manufactured with a bowl with a diameter from 5 to 12.5*m.*

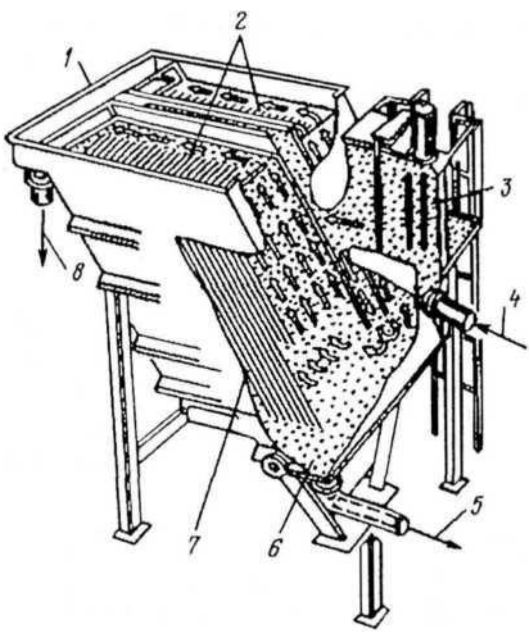

Figure 16.3. Plate thickener:

I - capacity; 2, 7 - plates; 3 - tank; 4 - sludge loading device; 6 - hopper: 8 - drain unloading

Plate thickener(drawing16.3) is a container 1 with numerous inclined plates 2, 7, which are located at some distance from each other and form a kind of channels.

The initial pulp 4 is fed into tank 3, where it is mixed with flocculants and moves from bottom to top between the plates, due to which the flow movement becomes laminar. Solid particles, under the influence of the resultant flow velocity and gravity, settle on the surface of the plates of each channel and slide down into the hopper 6, from where the condensed sludge 5 is discharged. The liquid (drain) 8 freed from solid particles moves upward and is removed through a collection chute. Thus, the deposited particles travel a small distance between adjacent plates.

The advantages of inclined plate thickeners include high productivity in a small footprint, no moving parts, low wear and tear and low operating costs. Such thickeners are manufactured with an effective deposition surface of 50; 100; 250; 1000m^2. The angle of inclination of the plates is 45-55°. For the manufacture of the device body and plates, ordinary or stainless steel and fiberglass are used. Thickener depth 2-3*m.* The distance between the plates is usually 50*mm.*

Control questions

1. What process is called dehydration?
2. Depending on the moisture (water) content, what types of products share?
3. Define the concepts liquid, wet, moist, air-dry and dry product
4. What processes are used to dehydrate foods? enrichment?
5. What process is called drainage?
6. Where is the drainage process carried out?
7. What process is called condensation?
8. How does the thickening process occur in thickeners?
9. What substances are added to the pulp to speed up the process thickening?
10. What are the main parts of a radial thickener?
11. Depending on the design of the drive mechanism, what types Do thickeners exist?
12. Explain the operating principle of the thickener
13. What is the difference between a thickener and a magnetic deslimer?

14. Explain the operating principle of a plate thickener

Lecture No. 17 FILTRATION. DRYING

Filtration. Centrifugation. Drying

Key terms: filtration, sediment, cake, filtrate, liquid phase, vacuum filter, filter cloth, filter press, drum vacuum filter, stirrer, hollow journal, disk filter, hollow shaft, sector, belt vacuum filter, filtration cycle, chamber filter press, suspension, centrifugation, drum dryer, drying.

17.1. Filtration

Filtering is the process of removing the liquid phase of pulp from using a porous partition under the influence of a pressure difference created by rarefaction of air or excess pressure. Solid particles retained by the filter surface are called *sediment* (kekom), and the water passing through the partition *filtrate*.

Special cotton, wool, nylon, nylon and other fabrics, as well as metal mesh with holes 0.15-0.25, are used as filter partitions *mm*.

The filtrate is released by creating a pressure difference on both sides of the filter surface (Figure 17.1).

Under the influence of pressure difference $\Delta P=P_1-R_2$ the liquid phase passes through the pores of the fabric, and the hard one is retained. Over time, the height of the suspension layer *N* will decrease, and the thickness of the sediment *h* increase. Thickness filter cloth h_0 is constant. Filtration will continue until until *H* will not be equal to zero. At this moment the height of the sediment layer *h* will be maximum. After this, a vacuum is maintained for some more time to dry the sediment, and then it is unloaded and the filtration cycle is repeated.

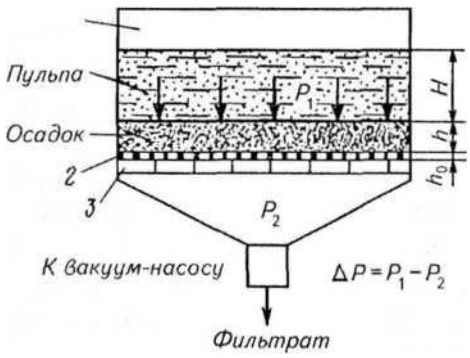

Figure 17.1 - Scheme of the filtration process:

1 - bath body: 2 - filter fabric; 3 - filter partition

The filtration rate of a liquid depends on the pressure difference, layer height, layer resistivity and liquid viscosity.

Depending on the method of creating a pressure difference, vacuum filters and filter presses are distinguished. Depending on the shape of the filtering surface, vacuum filters are divided into drum filters (with an internal and external filter surface), disk and belt. Vacuum filters are widely used for dehydration of enrichment products during the processing of various ore and non-metallic minerals.

*Drum vacuum filter*with external filter surface (Figure 17.2) consists of a drum 1, hollow axles 2, distribution heads 3, a bath 4, a stirrer 5 and a drive 6. A vertical partition 8 divides the drum into two sections isolated from each other. The outer surface of the drum is divided into shallow cells covered with perforated grids.

There are grooves between the gratings into which the filter fabric is sealed with rubber bands, so the cells are isolated from each other. The filter fabric is secured to the drum using soft steel wire, which is wrapped around the drum. The internal cavity of the drum is divided radially into sections, each of which is connected by pipes 7 to a trunnion. The drum is immersed in a bath equipped with a stirrer to prevent sedimentation of solid particles. Overflow pipes are provided in the side wall of the bath, with the help of which a constant level of pulp in the bath is ensured. Rotation from the electric motor is transmitted through a multi-stage gearbox to a drive gear mounted on the drum axle. The drum rotates on hollow cast iron axles in bearings mounted on the end walls of the bath.

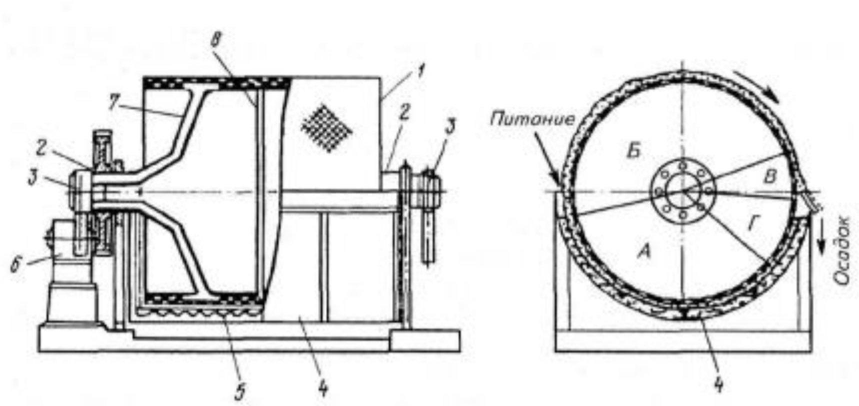

Figure 17.2 - Drum vacuum filter with external filter surface:

1 - drum; 2 - hollow pins; 3 - distribution heads; 4 - bath; 5 - mixers; 6 - drive

Distribution heads with replaceable washers are pressed to the ends of the hollow journals by springs, which serve to connect the internal sections of the filter to vacuum wires and pipes supplying compressed air and discharging filtrate.

The filtering process is carried out according to the following scheme. The pulp is fed into a filter bath, where solid particles are kept in suspension by swinging rakes. In the zone*A*The sectors of the drum are under vacuum, so a layer of sediment is deposited on the surface of the filter cloth. Water passes through the pores of the filter fabric and enters the inner cavity of the drum, from where it is discharged through the distribution head. Zone*B*-sediment drying zone. Under the action of a vacuum, air is sucked through the sediment, displacing the moisture contained in the pores. Zone*IN*-sediment removal zone.

The sectors located in this zone are connected to a compressed air line, which blows off sediment from the surface of the filter cloth. In the zone*G* the filter fabric is regenerated. The pores of the fabric are cleaned of solid particles using water or compressed air.

Drum filters with an external filtering surface, like disk filters, are manufactured in conventional (BOU, DU type) and acid-resistant (BOK, DK) versions for filtering fine-grained materials with an upper particle size limit of 65-70% class -0.074*mm*.

To filter larger materials, drum vacuum filters with an internal filter surface, linen fine vacuum filters and plan filters are used.

In drum vacuum filters with an internal filtering surface, filter sections with a total area of 10 to 40m^2 located on the inner surface of a solid drum with a diameter of 2.7 *m*and length from1.2 to 5.2*m*.

Disc vacuum filter(drawing17.3) consists of a hollow shaft 1 with disks 2 mounted on it, a distribution head 3, a device for removing sediment 5, a bath 6, a drive 7 and a stirrer 4. The disks are assembled from sectors that communicate through pipes 8 with the longitudinal internal channels of the shaft. The sides of the discs form a filter surface.

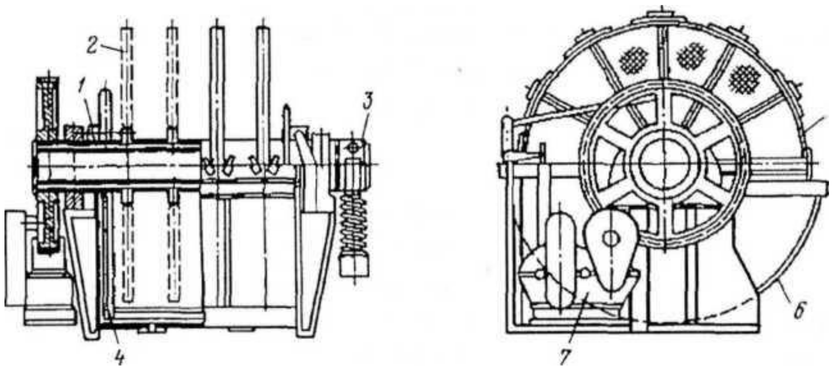

Figure 17.3 - Disc vacuum filter:

1 - hollow shaft; 2 - disks; 3 - distribution head; 4 - mixers; 5 - sediment removal device; 6 - bath; 7 – drive

The sectors are inserted into special sockets of the hollow shaft and secured to it using pins with linings. The distribution heads are pressed against the ends of the shaft, as with a drum vacuum filter, using springs. If the number of disks is less than six, the filter is equipped with one distribution head.

The shaft with disks is lowered into a bath of filtered suspension. During filter operation, the filter bath on the side where the sectors enter the pulp has pockets, on both sides of which there are knives reinforced with rubber or made from a conveyor belt. The knives clean the surface of the filter cloth from sediment that did not separate during blowing.

The operating principle of a disk vacuum filter is the same as that of a drum vacuum filter with an external filter surface. The pulp is fed into a bath equipped with an overflow pipe. When the disks rotate on the surface of the sectors immersed in the pulp, sediment is collected, then, as the sectors leave the suspension, the sediment is dried and removed from the surface of the filter fabric.

Belt vacuum filter(drawing17.4) is an endless rubber belt 5 with holes, covered with filter fabric and stretched onto drive 1 and tension 6 drums. The sides of the belt slide at a speed of 0.01-0.167*m/s*along two guide bars 3, and its middle part is adjacent to the grate above the vacuum chamber 2, connected by pipes to the filtrate collector.

The pulp comes from the feed tray 4, the resulting layer of cake is removed by a knife device 9 on the drive drum. The lower part of the belt, supported by rollers 7, can be washed by device 8 in order to regenerate the filter fabric.

Vacuum filters operate at a vacuum of 0.04-0.09*MPa*and compressed air pressure when blowing off the cake up to 0.05*MPa*. Their specific productivity increases, and cake moisture decreases with increasing vacuum and pulp temperature, material size and solid content in the filtered pulp, while reducing the content of sludge particles that clog the pores of the filter cloth.

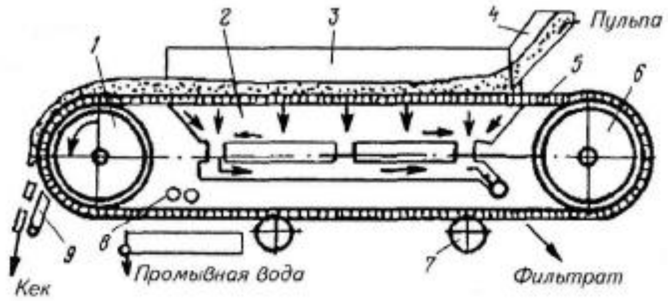

Figure 17.4 - Belt vacuum filter:

1.6 - drive and tension drums; 2 - vacuum chamber; 3 - guide strips; 4 - feeding tray; 5 - rubber tape; 7 - rollers; 8 - flushing device; 9 - knife device

The specific productivity of disk and drum filters with an external filter surface is 0.1-2 $t/(m^2 \cdot days)$ at cake humidity 8-25%, drum filters with internal filter surface - 0.6-115 $t/(m^2 \cdot days)$ at cake moisture10.5-14%, belt and plan filters - 0.3-10 $t/(m^2 \cdot days)$ at cake moisture9-20%.

Filter presses- periodic devices intended for filtering flotation waste and fine sludge in cases where other types of filtration equipment are ineffective.

There are two types of filter presses: chamber and frame. *Chamber filter press*(drawing17.5) consists of a package of compressed filter plates 1 with ribbed depressions on the surface.

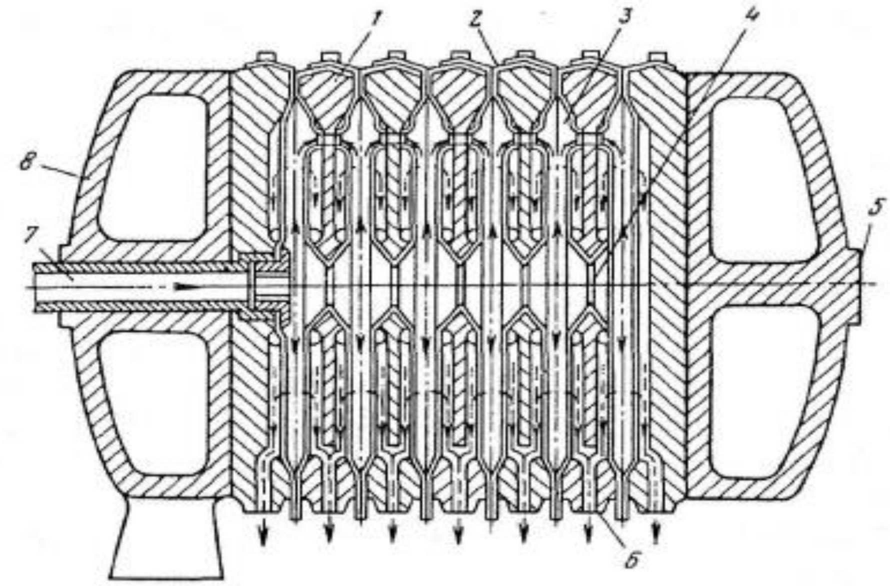

Figure 17.5 - Diagram of a chamber filter press:

1 - filter plates; 2 - filter fabric; 3 - cameras; 4 - central hole; 5 - movable plate; 6 - holes for filter release; 7 - initial supply pipe: 8 - fixed head plate

The plate has a central hole 4. Filter fabric 2 covers the plate on both sides and seals the plates together. Two adjacent fabric-covered plates form chambers 3, connected by central holes. The filter plates are supported by their side projections on the tie beams of the frame. The entire plate stack is compressed between two head base plates using a hydraulic or mechanical clamp.

The filtration cycle begins with closing the chambers. The filter plates, by means of a movable plate 5, pushed by a hydraulic or mechanical clamp, move towards the stationary head plate 8. The pressure in the clamping mechanism is higher than the operating pressure in the chambers, which ensures a reliable seal in the bag. The next phase is filling the filter with suspension. The initial suspension is pumped into the compressed package of plates through pipe 7 and enters the chambers through the central holes. The filtrate passes through the fabric (made of synthetic fibers), flows down the ribbed recesses and is discharged through the lower holes of the 6 plates to the side channel. Solid particles form sediment in the chambers.

The suspension is fed into the filter press using plunger, screw, membrane-piston or centrifugal pumps. During the initial period of filtration, when the chambers are filled with suspension, the pump operates with maximum flow at a relatively low pressure. After filling the chambers, the thickness of the sediment gradually increases, and the resistance also increases. During this period, the suspension is supplied at maximum pressure. After filling the chambers with sediment, the supply of suspension is stopped and compressed air is introduced through pipe 7, removing residual moisture from the pores. To unload the sludge, the filter plates are moved apart and the sludge from the open chambers is unloaded into the hopper.

In domestic coal preparation practice, chamber automatic filter presses from foreign companies with a filtration area of 570-600 have been used.m_2, providing productivity 7-9*t/h*and sediment humidity is 18-24%.

The advantages of filter presses are the ability to filter thin sludge, low sediment moisture, and almost pure filtrate. Flaws
- low productivity, design complexity.

17.2. Centrifugation

*Centrifugation*called the process of dehydration of small wet products and separation of the suspension into liquid and solid phases under the influence of centrifugal forces. The machines for carrying out these operations are called centrifuges, which, according to the principle of operation, are distinguished as filtering and sedimentation. The use of centrifugal forces during dehydration is caused by the need to accelerate processes that do not take place intensively or do not take place at all under the influence of gravity.

*Filter centrifuges with screw discharge of sediment*arranged as follows (Figure 17.6). The filter rotor 1 of the centrifuge rotates via a V-belt drive from an electric motor. A screw rotor 2 is placed in the internal cavity of the filter rotor, which rotates in the same direction, but with a lower frequency. Thanks to this, scrapers 3 mounted on the screw rotor in a spiral move the filtered material along the walls of the rotor 1 down to unloading.

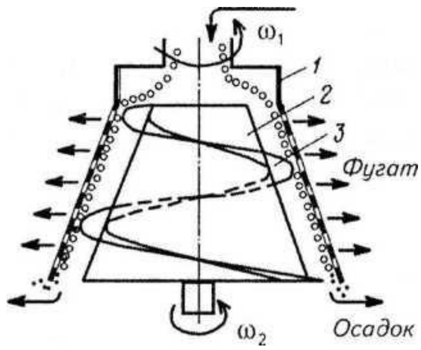

Figure 17.6 - Diagram of a filter screw centrifuge:
(1and 2- rotation speed of the filter and screw rotors, respectively)

The initial wet product enters the centrifuge onto the upper rotating part of the screw rotor, from where it is thrown onto the walls of the filter rotor by centrifugal force. From here, the auger scrapers gradually move the layer of material down towards the wide part of the rotor. As the solid phase moves along the inner surface of the filter rotor, it is dehydrated and water is separated by centrifugal force through the holes of the filter surface in the chamber of the centrifuge casing. The time it takes for the dehydrated product to move through the rotor of a screw filter centrifuge is determined by the performance of the screw (turn pitch and rotation speed) and is several seconds.

17.3. Drying

Drying- this is the process of dehydration of materials, including products enrichment based on the evaporation of moisture when heated.

Drying is the final stage of dehydration. It is used in cases where it is necessary to prevent the freezing of concentrates, to reduce the cost of their transportation over long distances, or when consumers of concentrates limit the moisture content to limits that cannot be achieved by thickening and filtering.

Various types of dryers are used for drying products: hearth ovens, tube dryers, shaft, electric, drum and drying ovens in a fluidized bed. At processing plants, drum dryers are most widely used.

Direct flow type drum dryer(drawing17.6) is a drum 5 with tires 4, with which it rests on support rollers 7. The drum is installed with an angle of 1-7° towards the unloading chamber 6.

The drum rotates from an electric motor through a gearbox 8, small 9 and ring gear 3.

The material is loaded through the loading pipe 2 from the side of the furnace 1. The inner surface of the drum is equipped with attachments 10, which loosen and lift the material to a certain height.

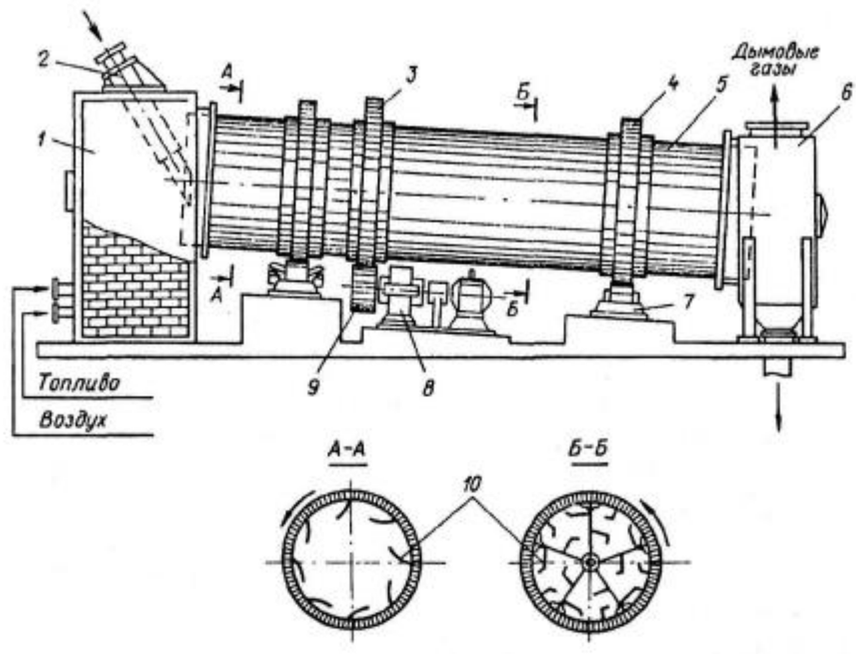

Figure 17.6 - Drum dryer:

1 - firebox; 2 - loading pipe; 3 - ring gear; 4 - bandage; 5 - drum; 6 -- unloading chamber; 7 - support rollers; 8 - gearbox; 9 – gear

The wet material is fed through the loading pipe into the drum, and coolant gas enters from the furnace. When hot gas comes into contact with the material, moisture evaporates, which, together with the gas, is removed by natural or forced (fan) draft.

As the drum rotates, the material gradually moves to the unloading chamber, from which it is unloaded with a moisture content of 1-5%.

Drying in tube dryers is mainly used in coal preparation plants. The drying installation consists of a firebox with a mixing chamber and a vertically installed firebox with a mixing chamber and a vertically installed pipe with a length of 14 to 35*m*and diameter 650-1200*mm*. Hot gases are sucked from the firebox through the lower end of the pipe by a fan-smoke exhauster and here the feeder is thrown into pipe source material. As the material moves up the pipe, it is dried and discharged into a cyclone. After cleaning in battery cyclones or a wet dust collector, gases are released into the atmosphere.

Drying in fluidized bed ovens involves the fluidization of bulk material under the influence of a slowly moving stream of hot gas, which transfers the material from a stationary state to a “boiling” state.

Fluidized bed furnace(drawing17.7) consists of two chambers: the lower - fuel-mixing and the upper - drying. The chambers are separated from each other by a metal or ceramic grille. The source material is fed by a paddle feeder 3 into the drying chamber 5. On the grate 2, under the influence of hot gas, a fluidized layer 30-45 cm high is formed. At the level of the fluidized bed, a pipe is installed through which dry material is removed from the dryer through the unloading device 7

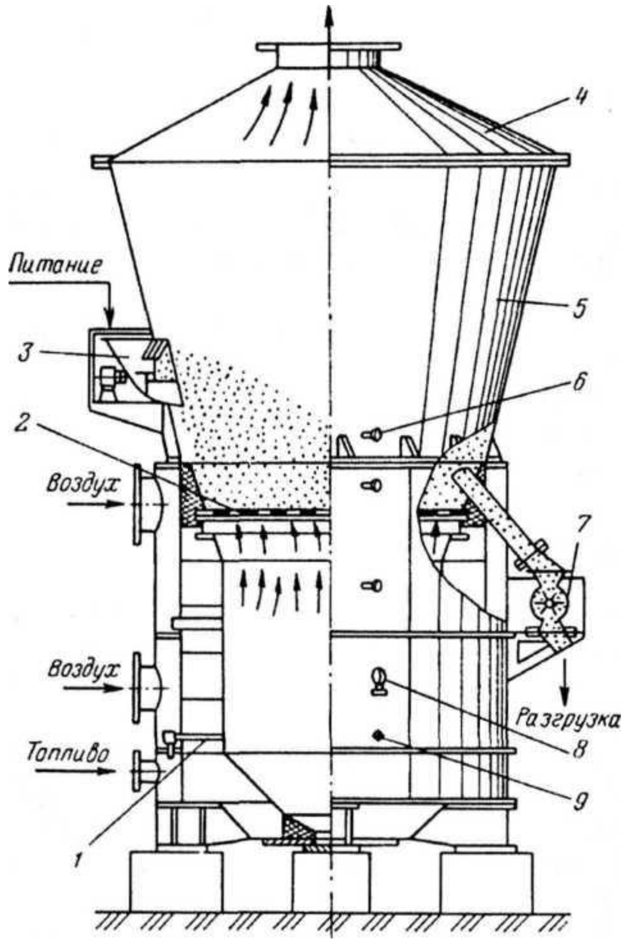

Figure 17.7 - Fluidized bed furnace:

1 - nozzle; 2 - grate; 3 - feeder; 4 - cover; 5 - drying chamber; 6 - thermocouples; 7 - unloading device; 8 - pressure gauge; 9 - ignition device

Control questions

1. What is the filtration process based on?

2. Define the concepts of sediment, filtrate
3. Explain the flow diagram of the filtration process
4. What types of filtration equipment do you know?
5. What are the main parts of a drum vacuum filter?
6. How does the filtration process work on drum filters?
7. What are the main parts of a disk vacuum filter?
8. Explain the operating principle of a vacuum belt filter
9. What are the purposes of filter presses?
10. Explain the operating principle of a chamber filter press
11. What is centrifugation?
12. What are the main parts of a drum dryer?

Lecture No. 18 PACKING OF MINERAL RESOURCES

Purpose and classification of agglomeration processes. Agglomeration. Pelleting. Briquetting

Key terms: agglomeration, agglomeration, pelletizing, briquetting, sintering, exhauster, thermal process, hygroscopic moisture, drum pelletizer, bowl pelletizer, mold, roller press

18.1. Purpose and classification of agglomeration processes

By agglomeration called processing processes of small classes minerals and concentrates into pieces (or granules, lumps) in order to prepare them for further more efficient use.

The agglomeration operation makes it possible to rationally use natural dust ores, concentrates, as well as some sludge waste from mining, processing and metallurgical industries. Pelleting processes include agglomeration, pelletizing and briquetting.

18.2. Agglomeration

Agglomeration is a process of thermochemical agglomeration of small ores, concentrates and flue dust by sintering them when heated.

The main area of application of agglomeration is the agglomeration of iron ore concentrates in order to obtain an agglomerate of a given chemical composition and metallurgical properties, such as: strength, porosity and reducibility, low content of harmful impurities, optimal basicity ($CaO:Si0_2$).

The production of agglomerate includes a large number of different operations, which can be reduced to the following basic diagram: preparation of components and charge for agglomeration - the agglomeration process itself - processing of the cake in order to obtain an agglomerate of specified properties.

Preparation of materials for agglomeration involves the calculation and preparation of a sintering charge (mixture) in order to obtain an agglomerate of a given chemical and mineral composition. Main component

The sintering charge is the ore part (concentrate) - 40-50%. Mandatory components of the charge: fuel (fine coke or anthracite) - 4-6% return (fine agglomerate) - 20-80%, moisture - 6-9%. To intensify the process and obtain an afluxed agglomerate, additional limestone, lime, dolomite, chalk and other additives can be added to the charge. All components of the charge must meet certain requirements for particle size distribution. The size of the ore part is no more than 8-6*mm*, fuel and limestone - no more than 8 mm. Before sintering, the charge is thoroughly mixed, moistened and pelletized in rotating drums.

The actual process of agglomeration of iron ores and concentrates is carried out, as a rule, in conveyor-type sintering machines, which are a conveyor consisting of individual trolleys moved along closed guides.

A diagram of the sintering process is shown in Figure 18.1. Bed 2 is loaded onto the grate of installation 1 (for example, onto the trolley of a sintering machine). After laying the bed, the mixture is loaded in a layer of 250-300*mm*.

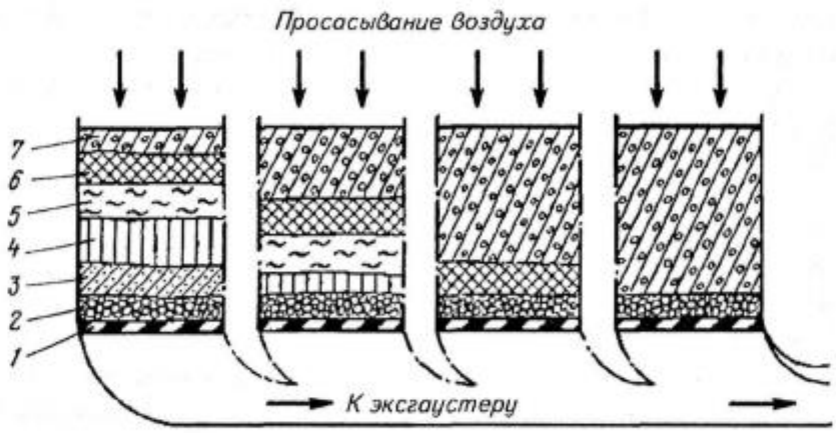

Figure 18.1 - Scheme of the agglomeration process:

1-grid: 2-bed; 3-zone of waterlogging; 4-drying zone; 5-heating zone of the charge; 6-combustion zone; 7-zone of finished sinter

Under the grate, the exhauster creates a vacuum of about 7-10*kPa*, as a result of which outside air is sucked from the surface into the layer. After turning on the exhauster, the top layer of the charge is ignited; This is done by sucking hot to 1473-1513*TO* combustion products formed during the combustion of a mixture of blast furnace and coke gases in a burner (or incendiary forge). The gas combustion products give off heat to the upper layer of the charge, removing moisture from it, and create conditions for the start of combustion of the charge fuel. Combustion is supported by air drawn from the atmosphere. Combustion zone 6 gradually moves from top to bottom (to the grate) at a speed of 10-40*mm/min*. When moving combustion zone until the bed, the sintering process ends. The process lasts 7-15*min*. A feature of the sintering process is that at any given moment combustion occurs only in a narrow layer of the charge (no more than 40 mm). Below the combustion zone there is a charge in which fuel particles cannot burn due to the low temperature, insufficient for ignition (less than 973 K), and the low oxygen content in the sucked gas.

In the fuel combustion zone (coke), mineral recrystallization processes occur. The physicochemical properties of the agglomerate are determined by the temperature in this zone.

Above the combustion zone there is a finished agglomerate 7, through which air is sucked. Cooling the agglomerate, the air heats up. The heat of the air is used in the underlying fuel combustion zone, where the temperature reaches 1673-1873*TO*. The finished sinter zone 7 is characterized by a completed thermal process. In this zone, partial oxidation of the iron ore materials of the finished sinter takes place with atmospheric oxygen - the transition of part of the magnetite to hematite.

The combustion zone is followed by the heating zone of the charge 5, in which the processes of decomposition of carbonates, hydrates, as well as reduction processes take place. Although in general the sintering process is carried out with excess air, microvolumes containing burning fuel particles are characterized by a reducing atmosphere.

In drying zone 4, the hygroscopic moisture of the charge evaporates. When drying, the lumps of the mixture, partially crumbling, compact it. For this reason, the drying zone represents the greatest resistance to the passage of gases through the charge. Waterlogging zone 3, located above the bed area, is characterized by high humidity. Overmoistening occurs due to condensation of water vapor and waste gases when they come into contact with the colder part of the charge.

The bed does not take direct part in the agglomeration process. It protects the grate from exposure to high temperatures, prevents the mixture from spilling through the gaps, and facilitates the removal of agglomerate from the grate after the end of the process.

The division of the sintering process into zones is conventional, since in reality it occurs continuously. The temperature of the gases sucked out by the exhauster for a long time is 323-333*TO*and only in the last minutes of the process it rises to 473-573*TO*. The sintering process is highly economical, since it ensures almost complete utilization of the heat of the exhaust gases and a significant part of the heat of the finished product.

As a result of sintering, sinter is obtained. The pre-cooled cake is crushed and sent to screening to separate the hot return (class 0-8*mm)*. Sinter with a particle size of more than 8 mm is sent for cooling and then for screening. After screening, sinter grades +50 and -50+13*mm* sent to the blast furnace shop, and class -13+8*mm*- for use as a bed on a sintering machine.

18.3 Pelleting

Pelleting - this is the process of agglomeration of moistened finely ground materials (60-80% -0.044*mm*), based on their ability, when rolled, to form spherical granules (pellets) without the use of direct pressure.

The principle of pelletizing finely ground ore concentrates is that when interacting with water, thin hydrophilic particles of the concentrate (−0.074*mm*) form separate aggregates - "germinal lumps", onto which, when the material moves in the rotating surface of the pelletizing apparatus, wet particles roll, forming spherical pellets. Under the influence of pressure arising at the point of contact of the pellets with the plane of rotation of the apparatus, they are compacted to form a homogeneous structure. The process of obtaining pellets from concentrates includes three main stages: *preparation of charge components for pelletizing; obtaining raw pellets; hardening firing*.

The initial components of pelletizing are concentrate, fine ore, limestone, binding additives, solid or gaseous fuel. Preliminary preparation for pelletizing consists of averaging and, if necessary, additional grinding of the concentrate, grinding fluxing and binding additives (bentonite, limestone, etc.). The use of strengthening (binder) additives in pelletizing practice is mandatory to increase the strength of raw pellets. Before pelletizing, the mixture is mixed in rotary or screw mixers.

Pelleting is carried out in drum pelletizers. or bowl

Drum pelletizer(drawing18.2) is a rotating drum 2, inside of which there is a scraper device 6 or a rotating cutter for cleaning the skull. The source material is fed into the drum by loading conveyor 1. The pellets from the drum are unloaded onto screen 3 to separate the fines. The fines are returned to the pelletizer by the conveyor system 4, and the finished commercial pellets are fed by conveyor 5 for firing in special furnaces.

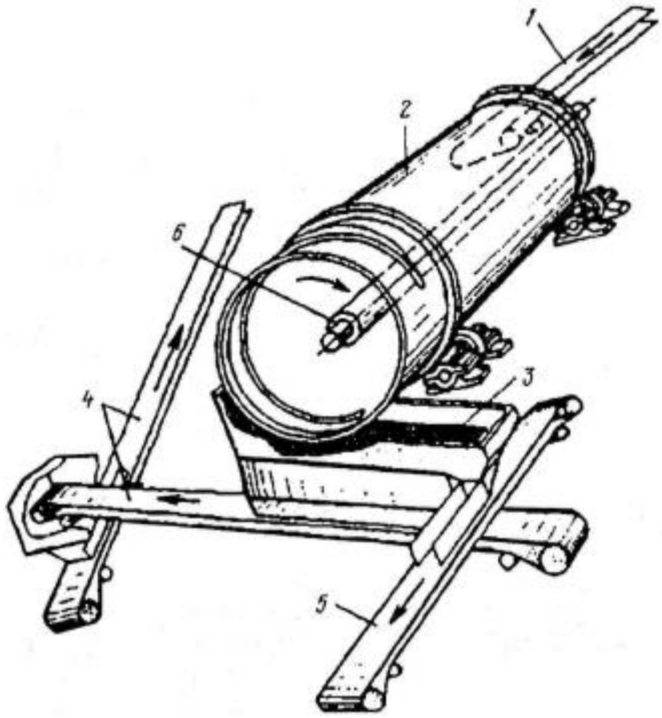

Figure 18.2 - Drum pelletizer:

1 - loading conveyor; 2 - pelletizer; 3 - screen for separating fines; 4 - conveyor system for returning fines to the pelletizer; 5 - conveyor for transporting pellets for firing; 6-scraper device or rotating cutter for cleaning the scull

Bowl (plate) pelletizer(drawing18.3) is a rotating bowl in the form of a disk with a side around the circumference, installed at an angle of 40-60% to the horizontal. The material is rolled at the bottom of the bowl. Under the influence of adhesion forces, the material clumps, the resulting pellets rise to a certain height and, rolling down, increase in size. The final pellet size is 10-16*mm*. Raw pellets must be strong enough to be transported from pelletizers to firing units. Pellets are considered suitable if they are not destroyed after 5-6 drops from a height of 300*mm*.

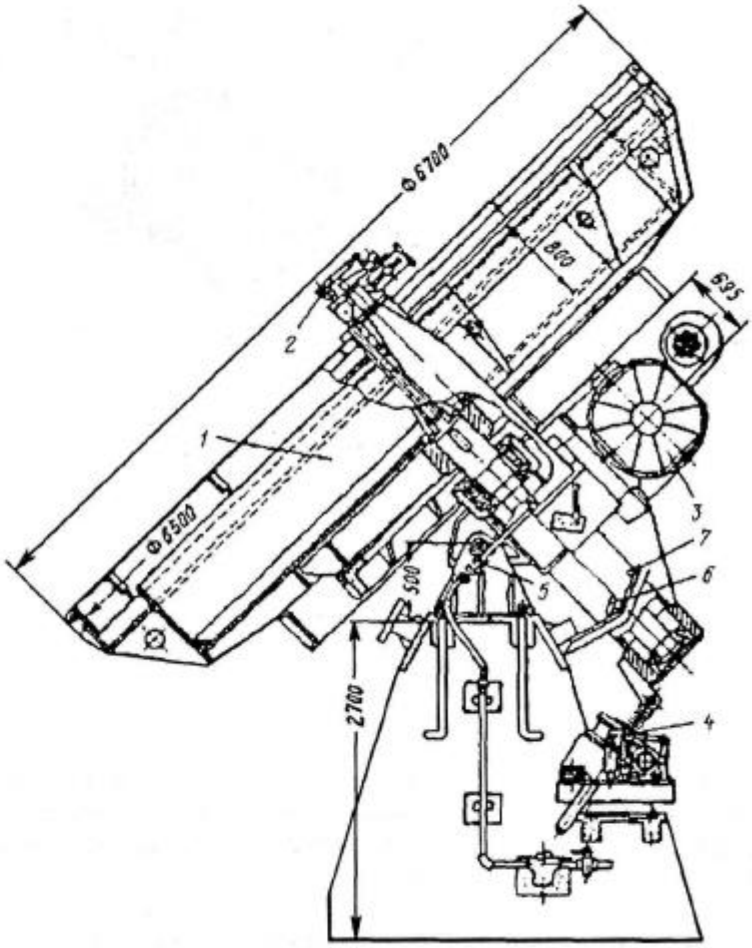

Figure 18.3 - Bowl pelletizer: 1-bowl; 2-installation of scrapers; 3-bowl drive; 4-mechanism tilt angle adjustment; 5-support; 6-copier; 7 way switch

The pellets should not crack during their hardening firing. To improve the properties of raw pellets, additives are added to the concentrate: bentonite, which increases resistance to crushing and cracking during rapid drying, limestone, which serves as a flux and increases the resistance of the pellets to impact during overload and crushing during drying. Bentonite is a highly dispersed clay capable of forming gels with a very developed surface. Strengthening firing of pellets is carried out after drying and consists of gradually heating them with furnace gases to a temperature of 1250-1300 *°C*.

Strengthening of pellets from magnetite concentrates occurs due to the oxidation of magnetite into hematite and subsequent recrystallization of hematite particles, solid-phase sintering of individual concentrate grains and partial formation of a slag binder during softening of the waste rock. Pellets from hematite concentrates are strengthened only thanks to the last three component processes not related to oxidation. The pellets are fired in various types of shaft furnaces, on conveyor furnaces machines, combined installations, including a moving grate and a drum rotary kiln. Cooling of pellets is carried out, as a rule, in the same units. The strength of fired pellets from concentrates of various types is in the range of 200-350*kg/pellet*.

Burnt pellets, received from finely ground concentrates, due to their high degree of oxidation and porosity, have good reducibility, high strength, uniformity in size and chemical composition, and have a high iron content. They can be stored, handled and transported without generating noticeable amounts of fines. To increase the technical and economic indicators of using pellets in blast furnace production, they are metallized in order to reduce iron oxides to metal at a temperature of 1493-1620*TO*. Methods of obtaining

metallized pellets differ in the composition of the charge, firing methods and equipment used.

The main objects of pelletization are finely ground iron ore concentrates. The size of concentrates obtained at a number of mining and processing enterprises processing low-grade iron ores varies from 95% class - 74*μm*up to 95% of the class - 44*μm*, which makes it difficult to agglomerate them by agglomeration and leads to large losses of metal during transportation and drying of the concentrate.

18.4. Briquetting

Briquetting- this is the process of agglomerating powdery, fine material in a confined space under the influence of mechanical forces (pressure).

The principle of briquetting small materials is that a briquette press compresses the source material in a mold, as a result of which small particles are combined into large briquette aggregates, the shape of which is determined by the configuration of the mold.

Depending on the method of binding particles into a briquette, a distinction is made between briquetting without binders and with the addition of binders (lime, clay, gypsum, coking coal, tar, etc.). In the first case, the particles come together under the action of molecular cohesive forces that arise during pressing. When briquetting with binders, the particles in the briquettes adhere to each other due to the adhesive ability of the additives.

The briquetting process includes preparatory operations, pressing and hardening of raw briquettes. Preparatory operations include crushing, grinding, screening, drying, roasting and mixing with a binder.

For briquetting itself, stamp, roller ring and revolving presses are used. Based on the specific pressing pressure, briquette presses are distinguished between low and medium (20-100*MPa*), high (100-150*MPa*) and ultra-high pressure (200-500*MPa*). The first group includes roller, table and rotary presses, the second - stamp presses, and the third - ring presses.

Roller presses(drawing18.4) are intended for briquetting ores and coals with binders and develop pressure up to 20-25*MPa*.

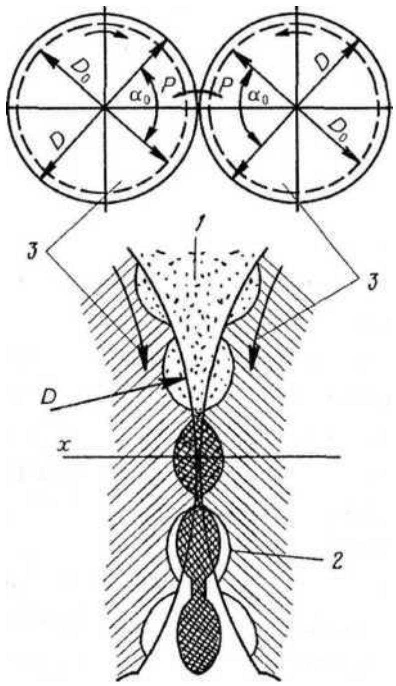

Figure 18.4 - Scheme of pressing a briquette charge in a roller press:

I - briquette charge; 2 - briquette; 3 - pressing rollers;*D*-diameter rollers;D_0- the nominal diameter of the circle on which they are located cells; *R*-pressure force of the rollers; 0- angle of direction of thrust forces between the rollers

Briquette charge arrives on pressing rollers from distribution bowl, in the bottom of which there are two rectangular holes above the rollers, where the charge is directed by a blade rotating in the bowl. In the chute through which the charge enters the rollers, a shield is installed that moves vertically to regulate the supply of the charge

The productivity, t/h, of a roller press is determined by the formula:

$$Q=0.006mBn, \qquad (18.1)$$

Where*m*- briquette mass,*G*;*IN*- number of cells on one bandage;*n*-frequency roller rotation,*With*$^{-1}$.

According to ore raw materials on roller press achieved productivity up to 100*t/h*.

The objects of briquetting are coals, fine ores and concentrates of non-ferrous and ferrous metals in cases where agglomeration and pelletization are unsuitable.

Briquetting of brown coals is carried out without binders. Coals are crushed to a particle size of 0-6*mm*, dried in steam tubular or gas pipe dryers to a moisture content of 15-20% and pressed under pressure 100-150*MPa*in stamp presses.

There are a number of requirements for lignite and coal briquettes. First of all, they must be moisture-resistant and heat-resistant, which characterizes their ability not to collapse before complete combustion. In appearance, briquettes must have a certain shape, weight, size and a smooth glossy surface, indicating the optimal moisture content of the dried product and optimal conditions for its pressing.

Brown coal briquettes for coking and semi-coking must have increased mechanical strength, heat and moisture resistance, and a compression resistance of at least 20*MPa,* moisture capacity (when immersed for 24*h*in distilled water) no more than 4% of the initial mass of the briquette.

Briquettes made with a binder should not stick together when stored for 3*h*in a thermostat at a temperature of 338 K under pressure corresponding to the pressure on their lower layers in railway cars. Commercial briquettes sent to the consumer should contain no more than 10% of fines measuring 0-25*mm*.

Technological scheme for producing lignite briquettes (Figure 18.5) is determined by the physical and mechanical properties of the processed coal, its particle size distribution and the requirements for the quality of briquettes.

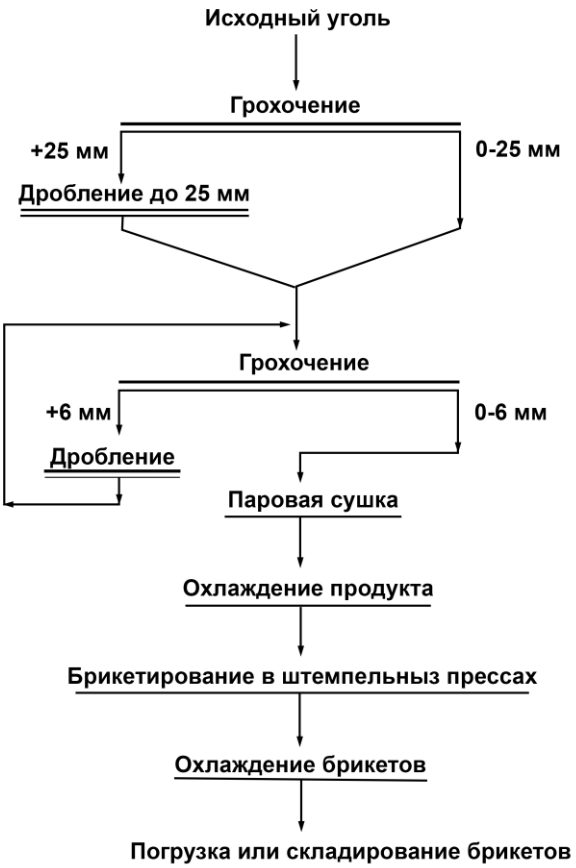

Figure 18.5 - Process flow diagram for producing lignite briquettes

The original coal with a moisture content of 55-56%, after screening and crushing to 6 mm, enters a drum steam dryer, where it is dried to a moisture content of 18-20%. To dry brown coal, drum steam and gas pipe dryers are used; In the former, superheated steam is used as a coolant, in the latter, flue gases are used.

Before pressing, fine coal is cooled to 313-323 K and briquetted in special presses under a pressure of 98-118 *MPa*.

For briquetting brown coals, stamp presses, which are periodic machines with an open matrix channel, are most widely used in industry.

Control questions

1. What is agglomeration?
2. What types of agglomeration process is divided into?
3. Define the concept of agglomeration process?
4. What operations does the production of sinter consist of?
5. Explain the scheme of the sintering process
6. What is pelletizing?
7. What are the main stages of the pelletizing process?
8. Explain the operating principle of the drum pelletizer
9. How does the pelletizing process take place in a bowl pelletizer?

10. What is the briquetting process?
11. How is the briquetting process carried out?
12. What are the requirements for briquettes?

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