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INTENSIFICATION OF THE PROCESS OF BENEFICIATION OF MINERALS BY VIBRATION-IMPACT ACTIVATION

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ABSTRACT. Experimental data are presented on the use of vertical vibrating mills (MVV) for the preparation of mineral feedstock for the beneficiation process. The influence of optimising the vibro-impact loading process on the parameters of the mineral beneficiation process is examined. Galena (PbS) was selected as the experimental sample. This is because the surface energy characteristics of PbS depend on the number and nature of defects in the crystal lattice, which is a key feature of material activation in MVV. The relationship between the degree of liberation (f) and the technological parameters of grinding in the vibro-mill (vibro-impact loading dose (D) and collision velocity (V_{imp})) is demonstrated. It has been found that even a slight change in the interplanar distances between atoms in the crystal lattice and the presence of a small number of impurity atoms of a foreign element significantly affect the chemical and adsorption activity of this mineral particle with respect to the components of the pulp's liquid phase and, ultimately, — the flotation activity.

Key words: *mechanochemistry, beneficiation, vertical vibrating mill, galena*

Introduction and theoretical background

When preparing ores for beneficiation, various types of grinding and classification equipment are used to break up the mineral grains. Broadly speaking, ore comminution is divided into batch processes carried out in crushing machines and continuous processes in grinding units.

In practice, the disintegration of mineral raw materials is carried out in machines using various crushing principles, with ball mills being the most commonly used.

The technological and chemical stages involved in preparing ore for processing are determined by factors such as particle size and shape, the presence of impurities, and defects in the crystal structure.

Until recently, the selection of grinding equipment was carried out using F. Bond's method, which is based on determining the grinding of material using the work index W_i (Bond, 1961). The Bond grinding index is the specific energy (in kilowatt-hours per metric tonne) required to grind material from an infinite particle size down to a specific fraction, 80% of which passes through a 100-micron sieve (Stamboliadis et al., 2011). W_i is a mathematical indicator of the hardness and grindability of a specific ore or rock – equation (1):

$$W_i = 44{,}5/a^{0,23} q_n^{0,820}\, 10\left(1/\sqrt{P} - 1/\sqrt{F}\right), \qquad (1)$$

where a is the mesh size of the sieve used to separate the final product, in μm; q_n is the quantity of the newly formed product «-a» per drum revolution, in kilograms per revolution; F and P are the 80% particle sizes of the feedstock and the final product, respectively, in μm.

This index serves as an industry standard for calculating the energy required to grind materials in the mining and cement industries (Malyarov et al., 2008).

Its calculation is based on the «size-energy» relationship

$$e_{1,2} = C\left(\frac{1}{x_2^n} - \frac{1}{x_1^n}\right), \qquad (2)$$

where n = 0.5, $x_1 = \infty$ and $x_2 = 100$, C is a constant, $e_{1,2}$ is the specific energy required to grind a particulate material from initial size x_1 to final size x_2.

By definition

$$e_{\infty,100} = W_i \qquad (3)$$

and, as a result,

$$C = 10W_i. \qquad (4)$$

In theory, for any material ground in a mill, the value of W_i found should be constant regardless of the measured dimensions x_1 and x_2 used to calculate the constant C by measuring the energy $e_{1,2}$. In practice, this is not the case because $n \neq 0.5$, and so a number of correction factors have been proposed to overcome this inaccuracy arising from assuming n = 0.5. Stamboliadis et al. (2011) proposed a simple method for calculating n using conventional grinding methods. The same calculation can be used to determine the true value of W_i and to define the potential energy state of the material at any size.

It is well known that grinding mills are characterised by low energy efficiency. This is because the energy required to create new surfaces accounts for less than 1% of the electricity consumed by ball mills. The remaining 99% is presumably dissipated as noise, vibration and heat; it is also possible that energy is lost to the atmosphere or used to heat the pulp. Bouchard et al., (2017) present the results of a study aimed, on the one hand, at characterising energy outputs in industrial grinding plants and, on the other, at identifying the potential for energy recovery from grinding plants. Agnico-Eagle Goldex Division, the Canadian Malartic mine and the New Gold New Afton mine participated in the studies. The results showed that, on average, 79% of the supplied electrical energy is converted into heat absorbed by the pulp, 8% is lost through the drive system, and around 2% of the energy is transferred to the surrounding air. The analysis showed that 91% of the energy consumed is lost as heat. The authors suggested that energy could be recovered using appropriate technologies or integrated

energy management systems; however, this topic requires further study.

To date, sufficient experience has been gained worldwide in the operation of various types of disintegration equipment in mineral processing, and this is taken into account in the design of crushing machinery.

One of the main indicators of grinding efficiency in ball mills is the specific throughput q_d (t/(m³·h)) for the newly formed nominal particle size class (– d):

$$q_d = (Q/V)(\beta_d - \alpha_d) = q_o \Delta\beta_d = q_o{}^{+}\Delta\beta, \qquad (5)$$

where q_d is the specific capacity of the mill or grinding plant for the newly formed design particle size class d, t/(m³·h); Q is the mill capacity based on the feed rate, t/h; V is the working volume of the mill or mills in the grinding plant, m³; q_0 – specific load based on the feed material (feed rate of fresh feed material per unit volume of the mill), t/(m³·h); $q_0{}^{+} = q_0 (1 – \alpha_d)$ – specific load for the coarse fraction (+d) in the feed material, t/(m³·h); $\Delta\beta = \beta_d – \alpha_d$ – increase in the content of the design particle size class, in units; β_d – relative proportion of the design particle size class in the ground products, in units; α_d – the same in the feed (Malyarov et al., 2012).

The q_d index presented above is a technical parameter that merely assesses the formation of the target particle size class and is not related to energy consumption. Furthermore, it does not take into account the formation of other, larger particle size classes that appear simultaneously with it. A very important characteristic of the grinding process is the grinding energy efficiency e_d for the newly formed particle size class (-d):

$$e_d = Q(\beta_d - \alpha_d)/N = q_d V/N = q_d/N_V, \qquad (6)$$

where e_d is the energy efficiency of grinding, tonnes/(kWh); N is the total power consumed by the mill's motor (kW), N_v is the specific power consumption of the mill motor, kW/m³.

A research article (Wentao et al., 2020) focuses on predicting particle size during grinding, which is an effective method for optimising the grinding process. Cassiterite polymetallic sulphide ore and lead-zinc ore were selected as the subjects of the study. The authors investigated the mechanism for predicting particle size based on the falling weight method and experimental data on batch grinding, taking into account the theory law of media motion in a ball mill and the population balance model. The results obtained show that the particle size distribution of the ground products at various energies and ore particle sizes was determined using the falling-weight method, whilst the grinding parameters ~~A~~ and ~~b~~ were calculated using a regression approximation method. A correlation was then established between the data from the falling-weight experiments. It was found that this correlation can also be applied to the correlation obtained for grinding products during media movement in a ball mill. This allows the tasks of determining the grinding function and the classification function to be solved effectively using MATLAB computing platform. The reliability of applying the particle size prediction model during grinding has been demonstrated. All the results obtained have been confirmed by experiments on periodic grinding.

In the paper of Kondratets et al. (2019), an equation was derived analytically that links the operational parameters of the ball mill and the material being ground to the parameters of the primary rod mill, which determine the energy efficiency of ore grinding.

A mathematical model has been analytically derived for energy-efficient ore grinding in a ball mill with three-phase movement of the grinding media, which remains invariant to changes in the length of the grinding media due to wear. The model enables the energy efficiency of grinding large ore chunks to be assessed on the basis of the volume of ground material obtained. The mathematical model includes such constant parameters as the cross-sectional areas of the rod-type primary transducers, the initial length of the rod-type primary transducers, the length of the base section where the strain gauges are located, the Young's modulus of the material of the primary transducers, as well as variable constants determined by the material being ground. Furthermore, a relationship for determining the length of the main rod-type primary transducer has been derived analytically, on the basis of which the height can be estimated.

Research (Ma et al. 2022 a), which focused on polymetallic complex ores from Tongken and Gaofen, revealed that the particle size of the ground products significantly influences the technical and economic performance of subsequent separation operations. Tests were conducted using the JK falling-weight method, periodic grinding, and a kinetic population balance model of grinding on the Simulink platform. The grinding characteristics of the two ore types and methods for predicting the particle size of the ground products were investigated. The results showed that the impact crushability of the ores from Tongken and Gaofen corresponds to 'medium' and 'soft' degrees, respectively. To improve the accuracy of the methods used in the study, the PSO–BP method was proposed, which offers high prediction accuracy and high model processing speed; however, it should be noted that the accuracy and speed of the iterative results depend on parameters such as the threshold value. The process of solving parametric problems for each forecasting method is based on various simplifications and assumptions. Therefore, for practical application, appropriate hypothetical theoretical models should be selected depending on the properties of the ore.

In another study of the same research group (Ma et al. 2022, b), again the Tongkeng and Gaofeng polymetallic complex ores were used as the subjects of investigation, and the effects of grinding patterns were examined using a single-factor experiment, whilst the grinding parameters were optimised using the response surface method for the two ore types. The results show that grinding time is a significant factor influencing particle size distribution. The optimal grinding concentration for Tongken and Gaofeng ores is 70% and 75%, respectively. The effect of the mill loading ratio on the Gaofeng ore is negligible. Rotational speed has a negligible effect on the technical efficiency of grinding. The regression model equations obtained using the response surface method are highly significant, and the relative prediction errors are within 1%, indicating the high reliability of the approximation equations. Optimal grinding conditions were determined for the ores: for Tongkeng ore, a grinding time of 5.4 minutes, a grinding concentration of 67% and a filling ratio of 35%; and for Gaofeng ore, a grinding time of 3.8 minutes, a grinding concentration of 73% and a filling ratio of 34%.

Fine grinding in a ball mill equipped with an agitator, which on the one hand improves ore liberation but on the other is associated with high energy consumption, is examined by Elbendari et al. (2025). According to the authors, energy consumption can be minimised by optimising operating conditions. This study investigated the influence of key

operating parameters, such as grinding time, agitator tip rotation speed, solid phase concentration and feed size, on grinding efficiency. Grinding efficiency was assessed based on specific energy consumption during the grinding of Egyptian copper ore in a mixer mill. Experimental data showed a linear correlation between the natural logarithm of the cumulative retained fraction and particle size (μm). As a result of the research, a model was proposed that accurately describes the particle size distribution (PSD) at various solid concentrations and grinding durations.

On the other hand, mechanochemical transformations occur during the grinding of materials. The dynamics of mechanochemical transformations are expressed by the relationship between the formation of a particular product and the excitation dose. Under real-world conditions, a dependence on the duration of mechanochemical treatment is most commonly observed. The activation dose (*D*) is determined by the formula (Butyagin et al., 1986):

$$D = N_v \cdot t \tag{7}$$

where *D* is the dose of vibro-impact excitation, N_v is the specific power consumed by the mill motor, kW/m³; t is the processing time, s.

The relationship between the vibro-impact activation of the material and the energy yield is determined using the formula:

$$G = W / N_v \tag{8}$$

where *G* is the energy output, mol/MJ; *W* is the specific reaction rate per unit mass of the material being ground, mol/(kg·s); N_v is the specific power density, representing the amount of mechanical power supplied per unit mass of the material being processed. In this system of units, N_v has the dimension MJ/(kg·s) (which is equivalent to MW/kg). If power is measured in the more familiar units of W/kg, a factor of 10^{-6} is usually introduced into the formula to convert to MJ.

It has been confirmed that mechanical activation of aluminium oxide – the grinding of material (Andrić et al., 1999) – alters the size and structure of its particles; this results in an increase in the system's potential energy, as well as in the material's chemical activity, due to an increase in surface area and the concentration of crystal lattice defects. Studies show that the properties of the processed materials change. Control of the mechanical activation process and the properties of activated materials is utilised in various technologies (sintering, chemical synthesis, hydrometallurgy, etc.).

Mañosa J. et al. (2024) investigated the mechanical activation of kaolin as a supplementary cementitious material with the aim of optimising grinding parameters using the response surface methodology. The study employed the Ritveld method to assess the influence of rotation speed and grinding time on the amorphous phase content, the reduction of crystalline kaolinite, and the incorporation of impurities into the activated clay. The results showed that regulating the grinding parameters effectively enhances the activation of the clay, allowing its use in low-carbon cements.

An article (Vugrin et al., 2023) discusses the optimisation of mechanochemical reactions through the energy generated by inelastic collisions between balls in a mill; it was found that chemical transformations occur during the grinding of the reaction mixture. The motion of the grinding media leads to physical mixing of the reactants, resulting in a chemical reaction. It was found that the energy input increases with the mass of the ball, its average velocity and the frequency of impacts, but decreases with the hardness of the grinding media due to more elastic collisions.

A study investigated the synthesis and filling of crystallographic sites in $MgAl_2O_4$ - with and without mechanochemical activation (Corlett et al., 2022). Without mechanochemical activation, heating to 1200 °C and above leads to the formation of a single spinel phase, regardless of whether the powders were mechanically activated or not. Neutron diffraction analysis was used to determine the filling of cation sites and showed that mechanical activation leads to a lower degree of cation site inversion compared to non-activated materials, indicating that the powders were closer to thermodynamic equilibrium. This is the first study in which neutron diffraction has been used to characterise the effect of mechanical activation on the filling of crystallographic sites in magnesium-aluminium spinel.

A number of studies (Bakil et al., 2025, Loudari et al., 2025, Korkmaz and Kayıran, 2025) have focused on improving the efficiency of grinding processes whilst reducing energy consumption. Őze et al. (2023) investigated the influence of various grinding parameters (ball-to-powder ratio (BPR), material of grinding media and grinding chambers, rotation speed and mill type) on minimising energy consumption during the mechanochemical activation of kaolinite. X-ray diffraction, Fourier-transform infrared spectroscopy, inductively coupled plasma optical emission spectroscopy, scanning electron microscopy and specific surface area measurements were used to investigate the influence of grinding parameters. It was found that the addition of just 25% (by mass) of trass reduces the specific energy requirement for complete amorphisation of kaolinite by 56%. The use of steel grinding media (instead of ZrO_2 media) had a negligible effect on the kinetics of kaolinite amorphisation, but allowed 30% more samples to be mechanochemically activated at the same BPR ratio and specific energy requirement. The use of four-chamber grinding instead of single-chamber grinding significantly reduced the specific energy requirement for complete and incomplete (α = 0.9) amorphisation of kaolinite. Overall, using steel grinding media, a BPR ratio of 14:1, four-chamber grinding and incomplete amorphisation of kaolinite, it was possible to reduce the specific energy requirement by 94%.

Shan et al. (2022) used measurements of specific surface area (using the BET method), X-ray diffraction (XRD), scanning electron microscopy (SEM), energy-dispersive X-ray analysis (EDX) and laser particle size analysis to investigate the mechanism of mechanical activation of titanium-containing domain slag. The results showed that the specific surface area of the slag increases upon activation compared to the unactivated sample. A decrease in the intensity of the diffraction peaks corresponding to perovskite ($CaTiO_3$) is observed, which is mainly due to an increase in lattice distortion and a decrease in the unit cell volume. With an increase in the mass ratio of the beads to the mass of the material, the mechanical activation time and the rotation speed during mechanical activation, the activated titanium-containing blast furnace slag underwent intensive agglomeration and sintering. Based on the results obtained, the authors concluded that the optimal parameters for the mechanical activation process are a ball-to-material mass ratio of 20:1, an activation time of 170 min and a rotation speed during activation of 400 rpm, respectively.

It is well known that higher grinding speeds and greater powder fineness are achieved in mills with higher ball energy – namely, vibratory and planetary mills.

In global practice, the most widely used design is that of a vibrating mill with a horizontally positioned grinding chamber and an inertial vibrator, the housing of which performs oscillatory movements along a trajectory close to a circular one. In this case, the material is ground as a result of impact and abrasion, with the grinding media acting on the surface layers of the particles.

For many years, research has been conducted at the Dnipro University of Technology into the mechanochemical activation of solids under vibro-impact loading in a vertical vibratory mill (MVV). A distinctive feature of these mills compared to other types is that the grinding chambers perform oscillatory movements along a straight-line trajectory in a vertical plane. Such parameters ensure a predominantly vibro-impact effect on the material being ground, with the deformation zone penetrating the entire volume of the particle. Ball wear is minimal and occurs uniformly across the entire surface, whilst maintaining the spherical shape. This means that, in many cases, there is no need to use surfactants (Franchuk, 2010).

The process parameters for the dispersion of various materials were presented by Svetkina (2008). It has been experimentally demonstrated that the grinding process of ore and non-ore materials can be optimised using process parameters (the size of the process gap in the grinding chamber between the upper layer of grinding media and its cover (in static conditions), and the maximum collision velocity of the grinding media). It was found that grinding results in a directed change in physicochemical properties.

In view of the above, the use of a vibro-impact mill would be appropriate for flotation processes, in which the interaction of reagents with activated minerals is crucial.

Therefore, the aim of this study was to determine the relationship between the degree of liberation (*f*) and changes in the technological parameters of the vibro-impact mill.

Methods and materials

The experimental study was conducted on a laboratory prototype of MVV with a rigid eccentric drive, the schematic of which is shown in Figure 1.

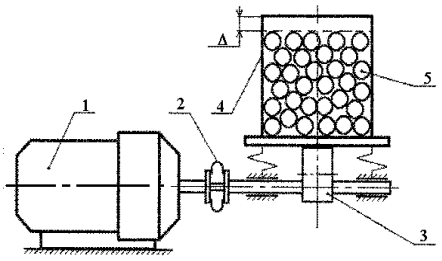


Fig. 1. Experimental setup diagram:
1 – electric motor; 2 – coupling; 3 – rigid eccentric vibration exciter; 4 – grinding chamber; 5 – grinding media

Electric motor 1 transmits rotation to vibration exciter 3 via petal clutch 2. This exciter imparts harmonic oscillations along the vertical axis to grinding chamber 4. The chamber oscillation amplitude (*a*) was 0.007 m, and the frequency (ω) was 16 Hz. The gap size Δ in the grinding chamber between the upper layer of grinding media 5 and its cover (in static conditions) was selected experimentally and was such that, at the specified oscillation parameters, it ensured, according to calculations (Anciferov et al., 2011) the maximum impact velocity and intensity of interaction between the grinding media and the grinding chamber. The computational model represents the process feed (grinding media), which is treated as a reduced unit mass interacting with the grinding chamber housing, without taking into account the impact time. The authors include dimensionless load coordinates in the mathematical models, as well as the clearance ratio between the grinding media and the chamber cover, and the maximum collision velocity (V_{imp}, m/s) and intensity of interaction between the grinding media and the grinding chamber. V_{imp} was calculated using the following formula:

$$V_{imp} = a \cdot \omega + \sqrt{2g\Delta}\,, \tag{9}$$

where *a* is the amplitude, m, ω – is the angular frequency, Hz, *g*=9.81 m/s^2, Δ is the gap in the grinding chamber between the upper layer of grinding media, mm.

Based on the experimental data, the starting point was chosen to be the technological gap Δ in the grinding chamber between the upper layer of grinding media and its cover (in static conditions) equal to $2.5 \cdot 10^{-2}$ m.

Therefore, experiments were conducted at various gaps (Δ). The process gap was varied by increasing or decreasing the volume of the grinding media. The dependences on changes in Δ are shown in Table 1.

Table 1 presents the process gap values established during vibro-impact activation of minerals and the corresponding impact velocities, calculated using formula (9).

Table 1. *Technological gap sizes and impact speeds*

Technological gap Δ, m	0.010	0.015	0.020	0.025	0.030
Collision velocity, V_{imp}, m/s	0.555	0.654	0.738	0.812	0.879

Further increasing the gap is undesirable, as it will disrupt the relationship between the grinding media and the mass of the material being ground, and could also result in the grinding media only impacting the bottom of the grinding chamber.

A laboratory (MVV) sample with a specific power of N_v = 394.7 kW/m^3 was used in the mechanical activation experiments.

Galenite (PbS) – a mineral prone to over-grinding – was selected as the subject of the study. The experiments involved the preliminary vibro-impact activation of the ore prior to the beneficiation process, comprising grinding in a ball mill, the determination of interplanar distances, and subsequent flotation.

For this purpose, an X-ray diffractometer (XRD) with a copper anode was used. The Bragg–Wolff formula was used to process the experimental data:

$$2d \sin\theta = n\lambda \tag{10}$$

where d is the interplanar spacing, in Å; θ is the angle of reflection, in degrees; λ is the wavelength of the emitter (1.5393 Å for copper); n is the order number of the line on the Debye diagram.

The angle of reflection was calculated using the following formula:

$$\theta = \frac{L}{4R} \quad (11)$$

where L is the distance between pairs of lines on the Debye diagram, m; R is the radius of the chambers ($5.7 \cdot 10^{-2}$ m). For the analysis, we used the concentrate from the first minute of the run and the galena flotation tailings with a particle size of −40 ± 15 μm.

To confirm the difference in the chemical activity of galena, we leached these products in 5% nitric acid and in a 5% solution of ammonium acetate. The galena sample weight for leaching was 0.1 g. The dissolution time of the mineral was 3 minutes.

For the flotation test, galena was used which had been activated by vibro-impact treatment (ground in a ball mill under the following operating conditions: Δ=25 mm and D=130.3) in a dry state and strictly classified within the following ranges: - 150 + 100 μm; - 100 + 74 μm; - 74 + 40 μm and - 40 + 15 μm. The latter fraction was prepared by separating the - 15 μm fraction through washing. A mixture containing 25% of each fraction was prepared for the experiments. The concentration of potassium butyl xanthate was 12 mg/L, 50 g/t of pine oil was added, and the pH was 8. The foam was skimmed off in fractions; after drying, each fraction was separated into the +74 and –74 μm classes using sieves. The –74 μm fraction was subjected to particle size analysis using a photometric method, which yields highly reproducible results. The +74 μm fraction was separated into narrower fractions using appropriate sieves.

In addition to factors determined by the conditions of interaction with reagents, the transition to foam is influenced by statistical factors that determine the conditions under which particles of different sizes attach to bubbles and partially detach from them as they fall back out of the foam layer; this is not the subject of this study.

Results and discussion

The strength of the chemisorption bond between the collector and the sulphide surface depends on the energy state of the cationic and anionic lattice sites, which determines the adsorption behaviour of the mineral surface. The energy characteristics of the sulphide surface depend on the number and nature of the disruptions to the strict periodicity of the crystal lattice.

As demonstrated in previous studies (Svietkina, et al., 2020), vibro-activation of minerals leads not only to surface ionisation but also to bulk ionisation. Consequently, the proportion of the covalent component of the bond between the metal and sulphur in galena may increase, which will weaken the interaction of water molecules with the mineral surface. This leads to the formation of surface areas with hydrophobic properties. From this perspective, it is of interest to perform X-ray structural analysis of the flotation products of the same mineral using the Debye–Scherrer method following activation in a high-pressure, high-temperature (HPHT) environment.

Table 2 presents experimental data on interplanar distances for the most intense lines in galena following its activation in MVV. The data presented in Table 2 show that the interplanar distances in galena following activation of the chamber product exceed those of the concentrate from the first minute of leaching. This indicates an increase in the ionic component of the chemical bond in the lattice of the activated mineral, as a result of which this galena particle will possess increased adsorption and chemical activity with respect to the ions and molecules of the liquid phase of the flotation pulp.

Table 2. *Values of interplanar distances for the most intense lines of the galena Debayagram following activation in the MVV*

№	Chamber product, d_1, Å	Concentrate, d_2, Å	$\Delta d=d_1-d_2$, Å
1	3.4964	3.4301	+0.0663
2	2.8861	2.8800	+0.0001
3	2.0531	2.0502	+0.0029
4	1.9047	1.8886	+0.0161
5	1.8223	1.8201	+0.0022
6	1.7159	1.7149	+0.0010
7	1.4259	1.4233	+0.0026
8	1.1103	1.1100	+0.0003

Leaching was carried out to confirm the changes in chemical activity. The leaching results are presented in Table 3.

Table 3. *Results of experiments on the dissolution of flotation products from galena particles of various particle size classes (initial sample weight 0.1 g)*

Class, μm	Vibration impact dose D (at Δ=25 mm)	Impact velocity V_{imp}, m/s	Amount of dissolved substance, g	
			HNO_3	CH_3COONH_3
—40+15	0	0	0.0045	0.0022
	67.1	0.812	0.0108	0.0074
—74+40	0	0	0.0019	0.0015
	67.1	0.812	0.0089	0.0053
—100+74	0	0	0.0015	0.0009
	67.1	0.654	0.0047	0.0019
	67.1	0.738	0.0059	0.0023
	67.1	0.812	0.0066	0.0025
	130.3	0.812	0.0086	0.0027
	197.4	0.812	0.0092	0.0029
150+100	0	0	0.0009	0.0005
	67.1	0.812	0.0023	0.0016

Table 3 presents the results of our experiments, which show that the dissolution rate of activated galena is higher in all cases than that of non-activated galena. Consequently, the galena particles that will pass into the concentrate first are those with a higher proportion of covalent bonding, i.e. those activated by vibrational impact.

Since the flotation process involves the continuous extraction of the mineral into the froth product, it is to be expected that the mineral particles will be distributed in the froth over time according to their surface properties, and vibro-impact activation allows the adsorption properties of the minerals to be altered.

Figures 2 and 3 show the results of experiments to determine the flotation rate of galena in individual size classes during both individual and combined flotation, as well as the distribution of individual size classes in each fractionated

product. Analysis of these data shows that the –40+15 μm class has a higher flotation rate compared to the –150+100 μm class. The remaining classes have an intermediate flotation rate.

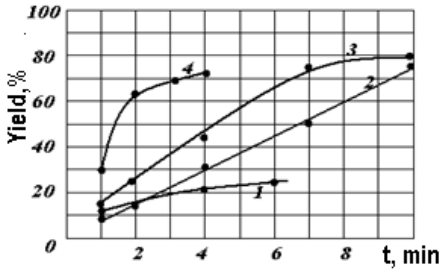


Fig. 2. Change in the flotation rate of galena of different sizes during separate flotation: 1 – -150…+100 μm; 2 – -100…+74 μm; 3 – -74…+40 μm; 4 – -40…+15 μm

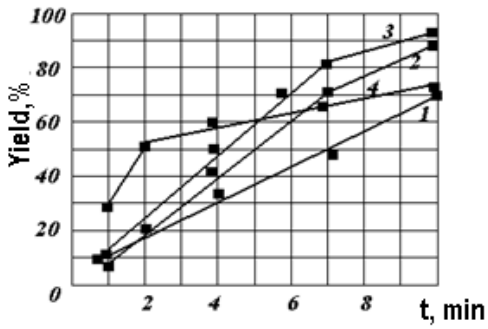


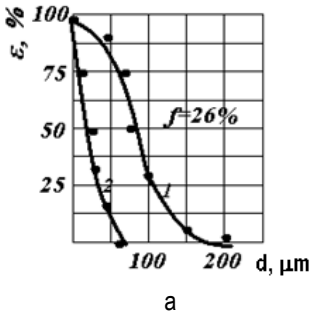
Fig. 3. Change in the flotation rate of galena of different sizes during combined flotation: 1 – -150+100 μm; 2 – -100+74 μm; 2 – -74+40 μm; 4 – -40+15 μm

The high yield of the –40 to +15 μm particle size class in the foam during the initial minutes indicates that this size class is more flotation-active. The increased flotability of the fine fraction is likely determined by two factors: an increase in the adsorption activity of particles as their size decreases, and the fact that larger particles are heavier and, all other things being equal, detach more easily from air bubbles. In addition, a certain quantity of the -40…+15 μm and -150…+100 μm classes remains in the tailings. The presence of the -40…+15 μm class in the flotation tailings can be explained by the distribution of particles according to their chemical bonds over the course of flotation. In this case, the activated galena in the tailings is characterised by a large interplanar distance and a higher value of the ionic component of the bond. The presence of galena particles of -150…+100 μm in the tailings is likely due to their high detachment force and insufficient density of the collector coating on the surface.

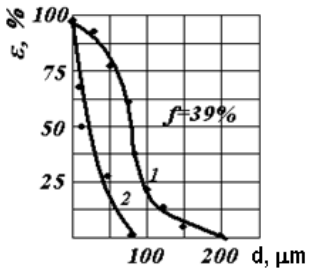
We determined the quantity of the mineral in the form of free particles, its total quantity and the degree of fragmentation using a graphical method, by plotting the size distribution curves for the mineral as a whole and for its free grains.

Figure 4 shows the size distribution curves for galena (curve 1) and its free grains (curve 2) before (a) and after vibro-impact loading (b). The experimental data shown in Fig. 4a and 4b indicate that vibration activation increases the degree of liberation for galena by 13%.

The grinding process in the vibrating impact mill depends on the process parameters under which the vibro-impact activation is carried out; consequently, the mineral liberation factor will also depend on these parameters.



a



b

Fig. 4. Characteristics of the size of galena (curve 1) and its free grains (curve 2) before (a) and after vibro-impact loading (b)

Figures 5 and 6 show the relationships between the degree of liberation (*f*) and the impact loading dose (*D*) and the collision velocity (V_{imp}).

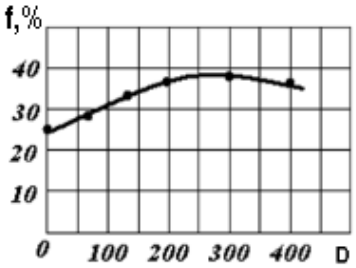


Fig. 5. Relationship between the degree of liberation (*f*) and the dose of vibro-impact loading, D, J/kg

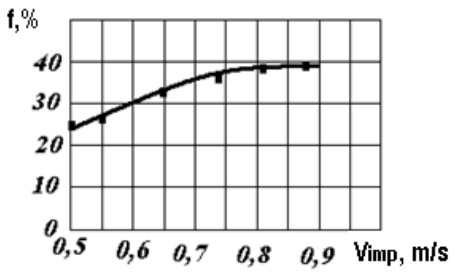


Fig. 6. Relationship between the degree of liberation (*f*) and the collision velocity V_{imp}, m/s

The relationships presented show that, during vibro-impact activation, there is a steady-state region in which the energy yield remains constant. The inhibition observed at high doses may be attributed to two factors: on the one hand, impurity

atoms within the mineral may cause the covalent component of the galena bond to become unstable; its proportion will vary in one direction or another, thereby causing varying degrees of interaction with the flotation reagents among galena particles of the same size; and, on the other hand, the predominance of ionic bonding makes the particles more chemically active towards the components of the pulp's liquid phase, which hinders their release into the froth. Nor can one rule out differences in the chemical activity of galena particles of varying sizes during the flotation process, which affects the rate of release into the froth, thereby determining the distribution of particles according to their surface properties across the flotation front. In addition, a study of the distribution of mineral particles along the flotation front (Nkadimeng et al., 2023) identifies additional technological factors regulating flotation by altering the feed particle size, time of conditioning with the collector and flotation water quality.

Conclusions

1. During vibro-impact treatment, a slight change is observed in the interplanar distances between atoms in the galena crystal lattice. These changes affect the chemical and adsorption activity of this mineral with respect to the components of the pulp's liquid phase, i.e. its flotation activity. This leads to an increase in the flotation rate.

2. It has been found that grinding the mineral in the MVV increases the degree of liberation *f* (for galena) from 26% to 39%.

3. Dependencies of the degree of liberation (*f*) on the technological parameters of the MVV have been obtained: on the vibrating impact loading dose (*D*) and on the collision velocity (V_{imp}), in which there is a steady-state region at constant energy output. As the vibrating load is increased, a decrease in the value of *f* is observed, which is attributed to changes in particle size and the presence of impurities in the mineral.

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