# TABLE OF CONTENTS

Kondratets V.O., Serbul O.M.
INVARIANT SYSTEM OF AUTOMATIC STABILIZATION OF LIQUID FLOW RATES WITH PRESSURE VARIATION IN THE PIPELINE ............................................................... 3-19

Kondratets V.O., Matsui A.M.
MATHEMATIC MODELING OF CONCENTRATION OF COARSE ORE AS A MEASURE FOR POSSIBLE OVERLOAD OF A BALL MILL................................................................. 20-31

Morkun V.S., Tron V.V., Hryshchenko S.M.
METHOD AND DEVICE OF AUTOMATIC NON-DESTRUCTIVE CONTROL OF MAGNETIC IRON CONTENT IN SLURRY FLOW........................................................................... 32-39

Morkun V.S., Tron V.V., Ravinskaia V. O.
DISINTEGRATION OF ORE FLOCCULES BEFORE FLOATATION CONCENTRATION ON THE BASIS OF DYNAMIC EFFECTS OF CONTROLLED HIGH-ENERGY ULTRASOUND.......... 40-44

Morkun V.S., Tron V.V.
COORDINATED AUTOMATIZED CONTROL OF AN ORE-PROCESSING ENTERPRISE AS A TECHNICAL-ORGANIZATIONAL SYSTEM.......................................................... 45-49

Morkun V.S., Tron V.V., Paranyuk D.I.
ADAPTIVE CONTROL AND IDENTIFICATION OF DRILLING SYSTEM WITH CONSIDERING THE TYPE OF DRILLED ORE MATERIAL .......................................................................................... 50-54
INVARIANT SYSTEM OF AUTOMATIC STABILIZATION OF LIQUID FLOW RATES WITH PRESSURE VARIATION IN THE PIPELINE

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Abstract: The aim of the work is the development of an invariant system of automatic stabilization of liquid flow rates with a relative error not exceeding ± 1.0%, during fluctuations in pressure in the supply main with simultaneous reduction of its dimensions and material consumption.

The research methods used in the work are as follows: methods of the hydraulics theory; methods of the automatic control theory; methods of analysis; modelling methods; methods of the theory of errors, methods of probability theory and mathematical statistics.

Academic novelty lies in the fact that for the first time an invariant system of automatic stabilization of liquid flow compensates the negative influence of pressure fluctuations in the main line and, thus, provides high accuracy of functioning, is realized.

Practical significance of the performed studies is high, since the obtained results make it possible to stabilize the flow of water into the sand chute of a mechanical single-spiral classifier quite simply and with sufficient accuracy. It allows to stabilize the pulp discharge in a ball mill and significantly improve the efficiency of its operation.

The invariant system of automatic stabilization of the liquid flow rate contains a vertical uniflow hydraulic converter, an input pipe and a cylindrical float, which serves as a regulator. The system provides a water flow of 24.3 m3/h with a relative error of ± 0.85%.

Keywords: input pipe, hydraulic converter, cylindrical float, automatic regulator

Introduction. Ferrous metallurgy of Ukraine as a main raw material component consumes a concentrate of poor iron ores, requires its grinding and enrichment. Grinding of ore in concentrating mills is characterized by high energy intensity and operating costs. The results of further processing largely depend on this process, especially such as mill productivity, yield of the useful component, its content in the concentrate, and losses at the end. As a result of increased losses, domestic iron ore concentrates are slightly more expensive than foreign ones, which worsen its competitiveness in the world market. Comprehensive measures are taken to improve the situation, among which is very important the improvement of the automatic control of the grinding of the initial ore in the initial stages. Among them, it is important to automatically stabilize the pulp discharge at specified levels in ball mills with a circulating load. In these and other technological processes, it becomes necessary to automatically stabilize the flow of water with a sufficiently high accuracy. Since this task is not solved now, and as part of a common problem is one of the priority areas of science and technology in Ukraine and in the scientific subjects of higher educational institutions and, in particular, the Central Ukrainian National Technical University, the topic of the proposed publication is relevant.

It can be seen from scientific works [1, 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12] that scientists have long been engaged in the automation of ore grinding in concentrating mills, but the necessary efficiency has not yet been achieved. Therefore, now scientists are intensively continuing to carry out such studies. For example, only in one university of Ukraine scientists for the last period have published a number of important scientific works aimed at solving this industry problem [13, 14, 15, 16, 17, 18, 19, 20, 21, 22, 23, 24]. However, in recent years, no one has been able to solve the problem of stabilizing the flow of water into the process with the achievement of high accuracy, as required automatically ensure the ratio of ore/water in a ball mill [25]. It is established that it is rather difficult to measure the water flow into the sand chute of a mechanical single-spiral classifier due to its relatively small value and small diameter of the pipelines [26]. The relative error in stabilizing the flow of water into the sand chute should not exceed ± 1,0% [27]. Known devices of this type are not very diverse. The traditional approach to solving this problem cannot be realized for the following reasons: the complexity (the automatic control system (ACS) contains a water flow sensor, a regulating organ, an actuator valve, a setting device) the accuracy of stabilization and regulation (ACS of such purpose usually provides a control error ± (2,5 ... 3,0)% and more); the complexity of the technical...
implementation (usually to measure the flow it is used a diaphragm, which cannot be used on pipeline diameters of 50 mm or less). Promising for solving this problem can be vertical hydraulic converters in which water is supplied from the main pipeline and flows freely at the bottom. One of such known devices [28] contains a hydraulic converter in the form of a tank, an automatic water level controller that manages the difference in flow and outlet lines. The device with direct water level regulation by changing the flow rate in the line [29] is more perfect. The disadvantage of the considered devices is a relatively large value of the maintained water level and cross-section, the large metal capacity of the structure and the mass of accumulated water, the comparatively low accuracy of water level regulation. The most perfect is a device having a main line, a flow line, a hydraulic converter with the mouth, made in the form of a tower of comparatively low height and cross-section, and an automatic regulator of the level of indirect action [29]. However, its disadvantage is the relatively insignificant accuracy of stabilizing the flow rate of the liquid due to the decrease in the height of its column and the level deviation that is commensurate with it during the adjustment process due to sensor errors, transfer links and the automatic regulator under the influence of medium pressure fluctuations in the pipeline, and sufficient level control complexity and a certain dependence of flow rate on liquid temperature. Nobody has studied the possibility of increasing the accuracy of liquids dosing by such devices, reducing their size, material consumption.

The aim of this work is to develop an invariant system for automatic stabilization of the flow rate of liquid with a relative error not exceeding ±1.0%, with pressure fluctuation in the supply main while reducing its size and material intensity.

**Materials and Methods.** Like the known devices of this function, this system must have a hydraulic converter and an automatic regulator. Mechanically the hydraulic converter can be made of three connected cylindrical pipe sections of different diameters - the outlet tube, the channel and the storage tank. The scheme of the hydraulic converter is shown in Fig. 1. According to the scheme of the hydraulic transducer and sections 1-1 and 2-2, we write for the steady-state fluid motion the Bernoulli equation [30]

\[
 z_1 + \frac{P_1}{\gamma} + \frac{\alpha_1 v_1^2}{2g} = z_2 + \frac{P_2}{\gamma} + \frac{\alpha_2 v_2^2}{2g} + \Delta h_b , \tag{1}
\]

where \( z \) is the geometric height, that is, the distance from an arbitrary horizontal reference plane 0-0 to the point considered in the cross-section. The indices refer to the section numbers; \( \frac{P}{\gamma} \) is the piezometric height, corresponds to full or gage pressure; \( \frac{\alpha v^2}{2g} \) is the impact pressure; \( \Delta h_b \) is the loss of pressure to overcome the hydraulic resistances between the intersections; \( \gamma \) is the fluid density; \( g \) is the acceleration of gravity.

We rewrite equation (1) in the form of

\[
 \left( z_1 + \frac{P_1}{\gamma} \right) - \left( z_2 + \frac{P_2}{\gamma} \right) = \frac{\alpha_2 v_2^2}{2g} - \frac{\alpha_1 v_1^2}{2g} + \Delta h_b . \tag{2}
\]

From the scheme of the hydraulic converter it is seen that the left side of equation (2) is equal to \( H \). Taking into account, in addition, that \( v_1 = 0 \), equation (2) can be written in the form

\[
 H = \frac{\alpha_2 v_2^2}{2g} + \Delta h_b , \tag{3}
\]

where \( v_2 \) is the velocity of the liquid in the line (at the output of the hydraulic converter in section 2-2); \( \alpha_2 \) is the dimensionless coefficient, taking into account the non-uniformity of the velocity distribution.

Losses of the pressure are equal to the total losses of pressure along separate sections and depend on the local resistances of the converter, characterized by the loss coefficients. In the case of intake, the loss factor depends on the ratio of the areas of the pipelines and can be taken from the table [31]. For practical calculations, can be used the dependence [32]

\[
 \xi_{M2} = \frac{1}{2} \left( 1 - \frac{d_2^2}{d_1^2} \right) ; \tag{4}
\]

where \( n = \frac{S_2}{S_1} \) is the contraction ratio ( \( S_2 < S_1 \)).

For this case

\[
 \xi_{M1} = \frac{1}{2} \left( 1 - \frac{d_2^2}{d_1^2} \right) ; \tag{5}
\]

\[
 \xi_{M2} = \frac{1}{2} \left( 1 - \frac{d_2^2}{d_2^2} \right) , \tag{6}
\]

where \( d_1, d_2, d_3 \) are diameters of the hydraulic converter in the corresponding sections.
The loss of pressure on friction is determined by the Darcy coefficient $\lambda$. To determine the Darcy coefficient in the turbulent regime of fluid motion, it is first necessary to determine the resistance zone. It is most expedient to use a quadratic area of resistance.

Let us check the conditions for finding a quadratic resistance in the zone. This is done in several ways. One of them is the fulfillment of inequality [30]

$$\text{Red} \geq (\text{Red})_{\text{lim}}^{\ast},$$ (7)

where $\text{Red}$ is the Reynolds number; $(\text{Red})_{\text{lim}}^{\ast}$ is a limit value of the Reynolds number.

Taking into account the values of the parameters in expression (7), it can be represented in the form

$$\frac{v}{\nu} \geq \frac{500}{\Delta},$$ (8)

where $\nu$ is the kinematic viscosity coefficient of the fluid; $\Delta$ is the average height of the roughness protrusions on the pipe walls.

For technical pipes $\Delta$ is the average height of the roughness protrusions. Such an averaging geometric characteristic of $\Delta$ cannot be established by direct measurements of the roughness protrusions. Therefore find $\bar{\Delta}$, which is called the equivalent roughness. Its numerical value is given in the tables [30].

As the outlet of the hydraulic converter, taking into account the long working life, we take a steel, cut after many years of operation, a piece of pipe with a diameter of 50 mm. In it $\Delta=0.3$ mm [30], and $\bar{\Delta}=0.006$. Despite the fact that the obtained value of $\Delta$ is less than 0.007, to determine the Darcy coefficient, one can use the dependence [30]

$$\lambda_3 \approx 0.11\sqrt{\bar{\Delta}} = 0.0306.$$ (9)

Then $(\text{Red})^{\ast}_{\text{lim}} \approx 83000$. To satisfy condition (7), it is necessary to provide a certain water velocity at the assigned $d_3=50$ mm and the maximum value of the kinematic viscosity coefficient of the water $\nu=0.015188 \text{ m}^2/\text{s}$ [30]. Under these conditions, the velocity is $258 \text{ m}/\text{s}$. To ensure the conditions of non-exit from the quadratic zone of resistance, we take $v_3=2.8 \text{ m}/\text{s}$.

The condition (7) must also be satisfied in the channel of the hydraulic converter. Considering the long operating time of the device, we take as a channel a piece of a steel water pipe with a long operation time of 100 mm in diameter. For it we can take $\Delta=1.1$ mm [30], then $\bar{\Delta}=0.011$.

Given that the obtained value of $\bar{\Delta}$ is greater than 0.007, to determine the Darcy coefficient, we use the dependence [30]

$$\lambda = \frac{0.25}{\left(\frac{\Delta}{3.7}\right)^2} = 0.9009.$$ (10)

For the channel $\lambda_2 = 0.9009$, and the velocity is $v_2=0.7 \text{ m}/\text{s}$. The limit value of the Reynolds number $(\text{Red})^{\ast}_{\text{lim}} = 45454$. The real value of the Reynolds number while channel working is Red $=46083$, which confirms the fulfillment of the condition (7). The friction losses in the fluid reservoir can be neglected, since it is short and has a large diameter and a low velocity of liquid motion.
Taking into account the found coefficients and the reduced velocities of fluid flow into the converter channel, the loss of pressure will be equal to

\[ \Delta p = \lambda_2 \frac{l_2 \Delta v^3}{d_2^2} + \lambda_3 \frac{l_3 \Delta v^3}{d_3^2} \frac{2g}{d_2^2} + \frac{1}{2} \left( 1 - \frac{d_2^2}{d_1^2} \right)^2 \frac{\Delta v^5}{d_2^4} + \frac{1}{2} \left( 1 - \frac{d_3^2}{d_1^2} \right)^2 \frac{\Delta v^5}{d_3^4} \]  

(11)

Let us express the velocity in the branch pipe through the velocity in the channel and, taking into account the losses, we obtain a mathematical model describing the hydraulic converter in steady state of operation

\[ H = \frac{v_2^2}{2g} \left[ \alpha_3 \frac{d_3^2}{d_3^2} + \lambda_2 \frac{l_2}{d_2^2} + \lambda_3 \frac{l_3 \Delta v^3}{d_3^2} \right] \]  

(12)

This correspondence allows to find the maximum value of the height of the hydraulic converter for certain parameters. In turbulent flows with sufficient accuracy of the obtained results, we can assume that \( \alpha_3 = 1 \). Let us assign that \( l_1 = 0.1H; \) \( l_2 = 0.7H; \) \( l_3 = 0.2H \).

Substituting the received parameters and the found values of the quantities into equation (12), we obtain \( H = 0.875 \) m. This is the minimum value of the water level at which the hydraulic converter operates in the quadratic resistance zone. To guarantee the non-exit of the motion of the liquid from the quadratic resistance zone, we take \( H = 1 \) m.

### Table 1. Correspondence of the change in the water velocity in the channel of the hydraulic converter at different water levels

<table>
<thead>
<tr>
<th>( H, ) m</th>
<th>1.0</th>
<th>1.005</th>
<th>1.01</th>
<th>1.015</th>
<th>1.02</th>
<th>1.025</th>
<th>1.03</th>
<th>1.035</th>
</tr>
</thead>
<tbody>
<tr>
<td>( \Delta v_2 ), %</td>
<td>0</td>
<td>0.25</td>
<td>0.5</td>
<td>0.75</td>
<td>1</td>
<td>1.24</td>
<td>1.49</td>
<td>1.74</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>( H, ) m</th>
<th>1.04</th>
<th>1.045</th>
<th>1.05</th>
<th>1.06</th>
<th>1.07</th>
<th>1.08</th>
<th>1.09</th>
<th>1.1</th>
</tr>
</thead>
<tbody>
<tr>
<td>( \Delta v_2 ), %</td>
<td>1.98</td>
<td>2.22</td>
<td>2.47</td>
<td>2.96</td>
<td>3.44</td>
<td>3.92</td>
<td>4.40</td>
<td>4.88</td>
</tr>
</tbody>
</table>

From the data of Table 1 it can be seen that the dosing error when the liquid level deviates from the prescribed value increases to a lesser extent than the change, but it is significant. The error is permissible only when the liquid level is up to 20 mm. At large changes in the level, it is beyond the permissible value.

The considered correspondence is obtained at \( \alpha_3 = 1 \). In existing installations this coefficient can take values up to 1.13 [32]. For this case, the correspondence takes the form

\[ v_2 = \frac{2gH}{\alpha_3 \frac{d_3^2}{d_3^2} + \lambda_2 \frac{l_2}{d_2^2} + \lambda_3 \frac{l_3 \Delta v^3}{d_3^2} + \frac{1}{2} \left( 1 - \frac{d_2^2}{d_1^2} \right)^2 \frac{\Delta v^5}{d_2^4} + \frac{1}{2} \left( 1 - \frac{d_3^2}{d_1^2} \right)^2 \frac{\Delta v^5}{d_3^4}} \]  

(13)

This mathematical model of the hydraulic converter describes the water supply of the sand chute of a mechanical single-spiral classifier in the steady-state operating mode. The initial value of this process is the velocity of the fluid. It must be constant in time. The accuracy of the fluid supply is uniquely determined by the change in the velocity of the ambient in the channel of the hydraulic converter. In the correspondence (13), all the parameters are constructive constants, the liquid level \( H \) and \( \alpha_3 \) have another origin. They can change in the process of work, that are drilled factors that affect the accuracy of the costs of water sources.

We substitute the found and accepted values of the parameters of the hydraulic transducer in the correspondence (13) and obtain

\[ v_2 = 1,25339\sqrt{H}, \text{ m/s}. \]  

(14)

The value of the relative change in the fluid velocity in the channel of the hydraulic transducer with changes in the water level is given in Table 1. The base value is the velocity \( v_2 \) at \( H = 1 \) m.
diameter of the outlet branch pipe of 50 mm and an average water velocity at the outlet of 2,8 m/s. The water in the hydraulic converter can be fed with a pipe 50 mm in diameter and 2 m in length. The line will have two elbows with an angle of 90°, a crane and a control element. The control element is the outlet of the straight pipe section on the shield. As a shield we use a cylindrical float.

For this hydraulic water supply system, can be used equation [28]

$$P_2 = P_1 + \Delta P_\rho + \Delta P_{p.o.}, \quad (16)$$

where $P_2$ is the pressure of the ambient in the source; $P_1$ is the ambient pressure in the hydraulic environment; $\Delta P_\rho$ is a resistance of the liquid supply line; $\Delta P_{p.o.}$ is a resistance of the control element.

The resistance of the liquid supply line consists of the sum of the local resistances, which the crane maintains, the control element, the two elbows, and the resistance created by the friction of the liquid against the pipe wall.

We take a pipe with a diameter of 50 mm, which has been cleaned after many years of operation, for the execution of the water supply line, taking into account its long operation. For such a pipe, the equivalent roughness is $\Delta = 0.3 \text{ mm}$ [30]. Relative roughness is $\overline{\Delta} = \sqrt[3]{\Delta} = 0.006$ [30].

Taking into account that the obtained value of $\overline{\Delta}$ is less than 0.007, the correspondence [30] can be used to determine the Darcy coefficient

$$\lambda = 0.17 \overline{\Delta}^{0.75}, \quad (17)$$

according to which $\lambda = 0.0306$. The limit value of the Reynolds number is $(Re d)^{\infty} \approx 83000$. To fulfill the working condition of the water supply line in the quadratic resistance zone, it is necessary to provide a certain average velocity of the ambient, which for the maximum value of the kinematic coefficient of viscosity and $Re = 85000$ is $v = 2.8 \text{ m/s}$. The resistance of friction of the liquid against the wall of the pipe will be [32]

$$\Delta P_{rem} = \frac{\lambda}{d} \frac{v^2}{2g}, \quad (18)$$

where $l$ is the length of the pipeline; $g$ is the gravity acceleration.

Local resistance is determined by the formula [28]

$$\Delta P_{nl} = \xi_i \frac{v_i^2}{2g}, \quad (19)$$

where $\xi_i$ is the coefficient of local resistance of sections; $v_i$ is the velocity of the ambient in the sections.

Since the velocity in all local resistance is the same, their total value will be

$$\Delta P_{nl} = \frac{\xi}{2g} (\xi_k + 2 \xi_a), \quad (20)$$

where $\xi$ is the coefficients of local resistance of the crane and elbows, respectively.

Resulting resistance of the water supply line to the hydraulic converter is

$$\Delta P_n = \frac{\xi}{2g} \left( \xi_k + 2 \xi_a + \frac{\lambda}{d} l \right), \quad (21)$$

The coefficient of local resistance of an elbow bent at an angle of 90°, with a ratio of the pipe radius to the radius of curvature $0.3$ is equal to $\xi_b = 0.158$ [31]. The coefficient of local resistance of the crane is a variable constant, which can be selected. It has the meaning $-0.05; 0.29; 1.56; 5.47; 17.3; 32.6; 206$ respectively for the angles of rotation $5; 10; 20; 30; 40; 50; 60$ degree angles [31].

The resistance of the control element is defined similarly

$$\Delta P_{po} = \xi_{po} \frac{v_{po}^2}{2g}, \quad (22)$$

where $\xi_{po}$ is the coefficients of local resistance of the control element. It depends on the design features of the output from the pipe to the shield and the relative distance of the shield from the edge of the pipe. We take an output having a rounding pipe with a ratio of its radius to the line diameter of 0.5.

Remoteness $h$ of the shield from the edge of the pipe is difficult to affect the value of the coefficient of local resistance, but the initial zone of change $\xi_{po}$ is almost rectilinear. In such case $\xi_{po}$ takes on value $2.5; 1.3; 0.63$ with appropriate changes $\frac{h}{d} - 0.05; 0.07; 0.10$ [33]. For the assigned pipe diameter, the absolute distance will be $2.5 \text{ mm}; 3.5 \text{ mm}; 5.0 \text{ mm}$.

Taking into account the fact that the water outlet from the control element takes place in the atmosphere, the pressure difference $P_2 - P_1$ is supernormal (manometric) pressure $P_m$. This
makes it possible to represent equation (16) in the form
\[
P_M = \frac{\nu^2}{2g} \left( \frac{\xi_k}{\xi_k + 2\xi_3 + \frac{L}{d} + \xi_{po}} \right). \tag{23}
\]
From (23) we find the average velocity of water in the line
\[
\nu = \frac{2g \cdot P_M}{\sqrt{\frac{\xi_k}{\xi_k + 1.54 \xi_{po}}}}. \tag{24}
\]
The density of water varies within a narrow range, so it cannot significantly affect the average velocity of the ambient [30]. The main drilling is the change in water pressure in the pipeline.

Taking into account the values of the parameters, equation (24) can be represented in the form
\[
\nu = 14.007 \frac{P_M}{\sqrt{\xi_k + 1.54 \xi_{po}}}, \text{ m/s.} \tag{25}
\]
Equation (25) shows that for a certain value of \(\xi_k\) the change in pressure in the pipeline can be compensated to a certain extent by varying \(\xi_{po}\). For this we have to select the average value \(\xi_{po} = 1.3\) and at nominal pressure \(P_{MH}\) of the water in the pipeline find the necessary \(\xi_k\), which is ensured by a certain angle of the shut-off element of the crane. The condition for determining the coefficient of local resistance (position) of the crane will be equation
\[
\nu = 14.007 \frac{P_{MH}}{\sqrt{\xi_k + 284}}. \tag{26}
\]
The water level in the hydraulic transducers will be unchanged and will correspond to the set value if the average velocities in its branch pipe and the flow line are equal, taking into account the same diameters. This mode of liquid movement will be most appropriate, since there is no dosing error in it. Therefore, to find \(\xi_k\) it is necessary to take \(\nu = \nu_n = 2.8\) m/s, which is determined earlier. At the nominal water pressure in the enterprise main line \(P_{MH} = 4\) at the coefficient of local resistance determined by equation (26) is \(\xi_k = 97.26\), which is possible to ensure.

Taking into account the given \(\xi_k\) the equation (25) takes the form
\[
\nu = 14.007 \frac{P_M}{\sqrt{98.8 + \xi_{po}}}. \tag{27}
\]
It can be seen from equation (27) that the increase in liquid pressure in the enterprise network can be compensated by a corresponding increase at \(\xi_{po}\). The limits of possible compensation are determined using the equation, which follows from (27) where \(\nu = \nu_n = \text{const} = 2.8\) m/s
\[
P_M = 3.948 + 0.03996 \xi_{po}. \tag{28}
\]
Substituting in (28) the limiting values of \(\xi_{po}\) (0.63 and 2.5), we determine the range of variation of water pressure in the main line at which compensation occurs. It will be from 3.97 to 4.05 at. This is a very small range of pressure fluctuations. In actual practice of the enterprise, it is much wider.

Attention is drawn to the considerable amount of losses on the crane compared to other supports and the control element, which is caused by the large operating pressure of the enterprise pipeline. Considering this, we investigate the influence of the nominal pressure of the pipeline on the width of the range of pressure variation, at which compensation of its influence is possible. Performing similar calculations, we obtained the data that are listed in Table 2. In this case, the threshold pressure value was determined, at which the liquid supply in the adopted motion mode is impossible. Threshold pressure value \(P_{MH} = 2.84 \cdot 0.03996 = 0.11349\) at. Therefore, the minimum value of the nominal pressure in the pipeline in the calculations was taken somewhat larger - \(P_{MH} = 0.15\) at.

From the data of Table 2 it can be seen that the coefficient of local resistance of the crane decreases significantly when the nominal pressure in the enterprise pipeline decreases. At close to the limiting values of pressure, it becomes commensurable with the index of the control element. In the case of small \(P_{MH}\) the pressure variation range is compensated by greatly expanding, taking the best values from 17.83 % to 31.97 % referred to \(P_{MH\min}\). Thus, the proposed control element at low nominal pressures in the enterprise pipeline within the linearity of the static characteristic ensures compensation of parameter fluctuations arising in the production conditions.
To compensate the effect of pressure changes in the pipeline, an automatic regulator must be used. As a sensor element of the liquid pressure, it is possible to use a shield of the control element, made in the form of a circular plate of a sufficiently large diameter. The action force of the jet on the plane of the shield is [32]

\[ F = \gamma \frac{\pi \cdot d^2}{4} \cdot v^2 . \]  

(29)

In this correspondence, the average fluid velocity in the pipe is a function of the pressure in the pipeline. It is determined by the correspondence (24). Substituting (24) into (29), we have

\[ F = \frac{\pi \cdot d^2 \cdot g}{2} \cdot \frac{P_m}{\xi_k + 2\xi_3 + \frac{l}{d} + \xi_{po}} . \]  

(30)

It can be seen from expression (30) that the force of the jet on the shield depends linearly on the pressure in the pipeline. Depending on all parameters are constants except of \( \xi_{po} \), which can vary. As the distance to the shield increases, the coefficient of local resistance of the control element will decrease, and the force at the same pressure will increase. At the same time, the velocity of the liquid also increases.

When designing an automatic regulator, it is advisable not to convert the strength of the sensing element into another signal. Then it is convenient to carry out comparisons of the sensor signal directly on the shield of the control element, applying counter force. Such force can be a hydrostatic lifting force, taking into account that the vector of the pressure force of a heavy liquid on the surface of a body immersed in it is equal in magnitude to the weight of the fluid in the body’s volume and directed in the direction opposite to the weight force [34]. Such the body can be a cylindrical float located on the surface of a liquid which axis coincides with the axis of the hydraulic converter. The shield of the control element can be a circular plate or directly the top wall of the float of an invariant system of automatic stabilization of liquid flow (Fig. 2).

Force \( F \) (30) compensated by the force of the float, directed in the opposite direction. The effort of the float is

\[ F_n = \frac{\pi \cdot d_n^2}{4} \cdot \delta h_n \gamma - (m_n + m_e) g , \]  

(31)

where \( d_n \) – float diameter; \( \delta h_n \) – depth of float immersion in water; \( m_n \) – float mass; \( m_e \) – shield mass.

A certain immersion of the float into the water is provided by the condition \( F = F_n \), at which the specified level of liquid in the hydraulic transducers and the velocity of movement in the branch pipe (costs) are guaranteed.

According to the correspondence (30) using the data of Table 2 at a nominal water pressure of 0,15 at. there were forces acting on the float on the side of the flow entering from the pipe into the hydraulic converter, at various distances from the edge of the pipe to the top wall of the float within the operating range of its displacements. The results of the calculations are plotted in Figure 3, a. It can be seen from it that under conditions of constant water pressure in the pipeline, the force acting on the float in the different equilibrium position is the same and in this case is equal to 30,8 H. That is, the force does not depend on the distance \( h \) of the float from the edge of the pipe. This can be explained by the fact that the force \( F = \gamma \cdot Q \cdot v \) [32], where \( Q \) is the water flow rate, when the flow acts perpendicular to the surface of the

<table>
<thead>
<tr>
<th>( P_{MH} ), at</th>
<th>( \xi_k )</th>
<th>Equation</th>
<th>Pressure deviation, at</th>
<th>Relative pressure deviation, %</th>
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<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td>Lower limit</td>
<td>Upper limit</td>
</tr>
<tr>
<td>4,00</td>
<td>97,26</td>
<td>( P_m = 3,948 + 0,03996 \xi_{po} )</td>
<td>3,97</td>
<td>4,05</td>
</tr>
<tr>
<td>3,00</td>
<td>72,24</td>
<td>( P_m = 2,948 + 0,03996 \xi_{po} )</td>
<td>2,95</td>
<td>3,05</td>
</tr>
<tr>
<td>2,00</td>
<td>47,21</td>
<td>( P_m = 1,948 + 0,03996 \xi_{po} )</td>
<td>1,93</td>
<td>2,05</td>
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<td>1,00</td>
<td>22,18</td>
<td>( P_m = 0,948 + 0,03996 \xi_{po} )</td>
<td>0,97</td>
<td>1,05</td>
</tr>
<tr>
<td>0,5</td>
<td>9,67</td>
<td>( P_m = 0,448 + 0,03996 \xi_{po} )</td>
<td>0,47</td>
<td>0,55</td>
</tr>
<tr>
<td>0,3</td>
<td>4,67</td>
<td>( P_m = 0,248 + 0,03996 \xi_{po} )</td>
<td>0,27</td>
<td>0,35</td>
</tr>
<tr>
<td>0,15</td>
<td>0,91</td>
<td>( P_m = 0,098 + 0,03996 \xi_{po} )</td>
<td>0,12</td>
<td>0,20</td>
</tr>
</tbody>
</table>
float. The water flow rate \( Q \) in the system does not change. As the pressure decreases \( h \) increases, grows to the standard value of the velocity \( v \) and force \( F \) remains unchanged. If the pressure suddenly increases, then for a certain \( h \) the velocity and flow rate increase. The excess water flow into the hydraulic converter lifts the float to the new stable position with a smaller \( h \), but the force \( F \), applied to the float in the steady state does not change. Instantaneous changes in water pressure in the pipeline lead to an increase or decrease of the initial force. Changes in the initial value of the force acting on the float are shown in Fig. 3, b, if the pressure, for example, changes instantaneously with respect to the average value of 0,15 at. When the pressure decreases from 0,15 at to 0,12 at the initial force acting on the float also decreases from 30,8 H to 24,6 H. Instantaneous pressure increase in the pipeline from 0,15 at to 0,2 at increases the initial force to 41,1 H. Under any conditions, then, the force acting on the float comes to 30,8 H. The transition process is less prolonged with a small mass of the float. Massive floats are characterized by inertia and longer transition processes.

The system of automatic stabilization of water flow into the sand chute of the classifier has the following features. If the ratio of the loss factor \( \Sigma \Delta P \) in line 2 and the hydraulic transducers \( 10 \Sigma \Delta h \), is established by selecting the position of the crane shut-off device 1, which is determined by the relation

\[
\frac{\Sigma \Delta P}{\Sigma \Delta h} = \frac{S_T}{S_K} = \frac{P_M}{H \cdot \gamma},
\]

where \( S_T \), \( S_K \) is respectively, the cross-sectional area of the pipe and the channel of the hydraulic converter, the balance of water inflow to the hydraulic converter and its flow rate is ensured.

In this case, the specified value of the liquid level \( H \) in the hydraulic converter will be maintained. Judge from the theoretical substantiation of the system, this value of height should satisfy the condition

\[
H \geq \frac{25 \cdot 10^4 \cdot \nu^2 \cdot (4 \cdot \lambda - 1,62)}{25 \cdot 10^4 \cdot \nu^2 - 2 \cdot g \cdot A^2},
\]

where \( d_c \) is a channel diameter; \( \nu \) is the liquid kinematic coefficient of viscosity; \( \lambda \) is the Darcy coefficient.

In this automatic control system (Fig. 2) branch pipe 8, channel 7 and the accumulator 6 are made with a diameter ratio of 1:1,05-1:10:4,5-5,5. Under such conditions, we obtain the minimum dimensions of the object and the high accuracy of stabilizing the water flow. The adjustable object is static. The automatic controller realizes the principle of drilling control. As a drilling, fluctuations in the water pressure in the pipeline 3 serve. The edge of the 5 pipe and the upper surface of the cylindrical float 9 create a hydraulic resistance, the magnitude of which is determined by the distance from the edge of the pipe to the float. The nominal distance determines the magnitude of the vertical stroke of the float 9 and the resulting errors in the liquid level \( H \) in the hydraulic converter. Therefore, to ensure the
highest accuracy of stabilizing the water level, it is advisable to have as little value as possible of a given distance (float stroke). These requirements are met by a pipe-shield system with rounded edges and a ratio of 0.5 radius of rounding to the diameter of the pipe. For small deviations, the resistance coefficient of this system is linearly dependent on the removal of the surface of the float, which makes it possible to use it as the control element.

Increasing the accuracy of liquid flow stabilization is also facilitated by the correspondence between the diameters of the float, pipe, line and accumulator. If the diameter of the cylindrical float is 3...4 of the diameter of the feed line pipe and 0.71 of the diameter of the accumulator, the linearity is achieved between the value of the local resistance coefficient of the outlet of the straight pipe section of the liquid supply line from removal of the float and the sufficient cross section for passage of liquid into the channel of the hydraulic converter. Perform these functions the float can with a ratio of its height and parameters

\[ H_{fr} \geq \frac{4 \cdot M_{fl}}{\pi \cdot d_{fl}^2 \cdot \gamma} + \frac{5 \cdot d_f^2 \cdot P_{MH}}{d_{fl}^2 \cdot \gamma \cdot \sum \Delta P_B}, \]  

(34)

where \( M_{fr} \) is the float mass; \( P_{MH} \) is the lowest value of water pressure in the pipeline; \( \sum \Delta P_B \) is total losses when supplying water from the pipeline to the hydraulic converter; \( d_n \) is a float diameter; \( d_t \) is a pipe diameter.

The float of diameter \( d_n \) in the ambient of density \( \gamma \) depending on the depth of immersion, develops a buoyancy force \( F_B \). The dependence of the buoyancy force of the cylindrical float on the depth of immersion for various diameters is shown in Figure 4, from which it can be seen that floats of 100 and 150 mm in diameter cannot develop a buoyancy force that will overcome the force of the flow from the pipe. From the graph 1 in Figure 4 it follows that the cylindrical float with a diameter of 200 mm such a buoyancy force can develop and it is advisable to use it.

**Figure 3.** Dependence of the steady (a) and initial (b) force acting on the float on the flow side with constant values of pressure and its instantaneous changes in the pipeline

- Removing the top wall of the float from the edge of the pipe \( h \), mm
- Water pressure in pipeline \( P_{MH} \times 10^{-1} \), at
The work of the invariant system of automatic stabilization of the flow rate depends strongly on the mass of the float, which can be made from various materials. Metal cylindrical floats can be made with a wall thickness of 2 mm, and plastic with a wall thickness of 4 mm. The dependence of the mass of cylindrical floats made of various materials on their height is shown in Fig. 5. From Figure 5 it follows that the cylindrical float from steel is distinguished by a significant mass, which must be unjustifiably overcome during operation. Cylindrical floats of duralumin and polystyrene materials have a relatively small mass. The best is a cylindrical float made of polystyrene with a diameter of 200 mm and a height of 120 mm. At a height of 140 mm and a diameter of 200 mm, its mass is only 0.64 kg. From the graph (Fig. 4), it can be seen that the functions of the control element in the invariant system of automatic stabilization of the liquid flow, taking into account \( F = 30.8 \) H and mass \( M_n = 0.5 \) kg (graph 3 in Fig. 5), can perform the cylindrical float of polystyrene with a diameter of 200 mm and a height of 120 mm. The verification of this conclusion by formula (34) showed that the height of such a cylindrical float should be 118.6 mm. So, we take the cylindrical float made of polystyrene with a diameter of 200 mm and a height of 120 mm with a wall thickness of 4 mm. Since \( H_n = 120 \) mm, and \( l_1 \) in Figure 1 took up at the level of 100 mm, then it must be extended to 200 mm by reducing the length of the channel \( l_2 \).

The invariant system of stabilization of the liquid flow works as follows. In the steady-state operation mode, at a nominal water pressure in the pipeline, the set value of its flow is maintained. In this case, we will have an average value of the distance from the shield to the edge of the pipe, the average value of \( \xi po \), and the given level value in the hydraulic converter. This is typical in that the water velocities in the pipe and the branch pipe of the converter are the same and are approximately equal to 2.8 m/s.

If the pressure in the pipeline suddenly decreases, then the force \( F \), respectively (30) will be less, at the established force \( F_n \) will lead to the output of the float from the water, a decrease in the distance from the pipe to the shield and an increase of \( \xi po \). The reduced velocity of the liquid in the line will result in its short - delivery to the hydraulic converter, as a result of which the level will take a lower value, and then the float will drop down, providing \( \xi po \), at which the flow in the

**Figure 4.** Dependence of the buoyancy force of a cylindrical float on the depth of immersion, with diameters: 1 – \( d_P = 200 \) mm; 2 – \( d_P = 150 \) mm; 3 – \( d_P = 100 \) mm

**Figure 5.** Dependence of the mass of cylindrical floats on height, made of: 1 - steel; 2 - duralumin; 3 - polystyrene
water supply line at a lower pressure are restored to the nominal value due to an increase in the distance from the edge pipes to the shield. In the steady state, this distance will correspond to the largest displacement of the shield up to 5 mm, the deviation may be 1.5 mm. Within such limits, the liquid level \( H \) will also vary.

With increasing water pressure in the pipeline, a large flow force acts on the float and it immerses into the liquid to a certain depth before compensation by the opposing force. In this state, due to the higher velocity and the smaller hydraulic resistance, a greater amount of liquid flows during the unit of time, which fills the space between the float and the walls of the accumulator. The buoyancy force immediately increases, the float begins to move to the edge of the pipe. The equilibrium of forces is established at a slightly higher level of \( H \) and flows that are practically reduced to the previous value. The deviation of the level will be insignificant - up to 1.0 ... 1.5 mm.

An invariant system of automatic stabilization of water flow rate into the sand chute of the mechanical single-spiral classifier should provide the flow rate of 24.3 m\(^3\)/h. It has the parameters given in Table 3. The task of the last stage of the study is to determine the actual flow of water, its dependence on various factors, the error of dosing.

### Table 3. Parameters of the invariant system of automatic stabilization of the liquid flow rate of 24.3 m\(^3\)/h into the sand chute of the classifier

<table>
<thead>
<tr>
<th>Diameters, mm</th>
<th>Pipe ( d_I )</th>
<th>Float ( d_n )</th>
<th>Accumulator ( d_k )</th>
<th>Channel ( d_k )</th>
<th>Outlet pipe ( d_{\text{BR}} )</th>
<th>Liquid level ( H )</th>
<th>Float height ( H_h )</th>
</tr>
</thead>
<tbody>
<tr>
<td>mm</td>
<td>50</td>
<td>200</td>
<td>282</td>
<td>100</td>
<td>50</td>
<td>1027.6</td>
<td>120</td>
</tr>
</tbody>
</table>

Such a problem can be solved by using a measuring container and fixing the time of its filling. The justification of the method for determining the volume flow of water at the facility is as follows. The calculated water flow, provided by the invariant system of automatic parameter stabilization, can be determined in accordance with the correspondence

\[
Q = \frac{V_{MC}}{t_3},
\]

where \( V_{MC} \) is the measuring container volume; \( t_3 \) is the measuring container filling time.

The calculated water flow depends on the error of the invariant system of automatic stabilization and the error of the method itself, which is characterized by the accuracy of the determination of \( V_{MC} \), more precisely the volume \( V_{BO} \) of the input liquid, and \( t_3 \). Moreover, \( V_{MC} \equiv V_{BO} \). To assess the accuracy of the invariant system of automatic stabilization of water flow, it is necessary to minimize the error of the method, that is, to determine \( V_{BO} \) and \( t_3 \) as accurately as possible. Ways to improve the accuracy of the method is the implementation of measures to increase the accuracy of fixing the volume, increasing the absolute value of the volume, the accuracy of time fixing, increasing the time of filling the volume, automating the process of fixing the volume and time. These requirements are mostly met by a measuring container (Fig. 6), designed to test an invariant system of automatic stabilization of liquid flow. The tank 1 with a significant volume of 5 m\(^3\) with a lower cylindrical chamber 8 with a diameter of 320 mm is used as a measuring container. It has an upper neck 2 with a diameter of 455 mm, covered with a lid 3, in which an electrode 4 of the lower and 5 of the upper levels and a shield 7 are installed, which prevents accidental closing of the electrodes with water. Due to the narrow cylindrical chamber 8 and the shield 7, it is possible to accurately fix the beginning of the readout of the volume of the measuring container. The end of the readout of the volume of the measuring container is also provided precisely due to the relatively narrow neck 2 and shield 7.

The absolute error in determining the volume \( \Delta V \) is determined by the correspondence

\[
\Delta V = \Delta h_1 S_{UK} + \Delta h_2 S_r,
\]

where \( \Delta h_1, \Delta h_2 \) – respectively, the height of the wave in the cylindrical chamber and the neck when filling the measuring container; \( S_{UK}, S_r \) – respectively, the cross-sectional area of the cylindrical chamber and the neck.

Studies have shown that when the measuring container is filled with water from the invariant system of automatic flow stabilization,
The wave process is created. The average wave height is generally 10 mm and is practically the same in the cylindrical chamber and neck, that is \( \Delta h_1 = \Delta h_2 = \Delta h \). Taking this into account, the correspondence (36) takes the form
\[
\Delta V = \Delta h (S_{UK} + S_t).
\] (37)

The electrode-water-reservoir circuit closes randomly. That is, the electrode can connect with the top of the wave crest, with the hollow or at another point of the wave. The probability of closing the electrode with any point of the wave is the same. Therefore, the dispersion of the absolute deviation of the water level in the cylindrical chamber, which characterizes the error, will be 5 mm. The absolute error in the volume is 0.009438 m\(^3\). The relative error in determining the volume in a cylindrical chamber or neck will be 11.74\%, and the total error will be 0.189\%.

The filling time can be determined in accordance with the frequency of the alternate current in the electrical network of 50 Hz and is maintained quite accurately. The maximum error will be characterized by one unaccounted impulse, depending on the phase of the contact closure by water and the phase of the beginning of the voltage wave in the electrical network. This error will be equal to
\[
\Delta t_{g_1} = \frac{0.02 \cdot Q \Pi}{V_{ME}} \cdot 100\% = \frac{Q \Pi}{V_{ME}} \cdot 2\% \approx 0.02 \cdot \frac{Q \Pi}{V_{ME}}, \] (38)
where 0.02 is a duration of one oscillation period

\( V_{ME} \) — voltage wave in the electrical network, \( s; Q \Pi \) is the flow, which provides an invariant system of automatic stabilization, m\(^3\)/s.

Taking into account the time error that occurs in the cylindrical chamber and the neck, the relative error in determining the filling time of the measuring container can be represented in the form
\[
\Delta t_{g_3} = \frac{4 \cdot Q \Pi}{V_{ME}} \cdot \%, \] (39)

The deviation in time \( \Delta t \) is also a random variable and will be characterized by dispersion that determines the error in measuring the filling time of the measured container. For this case \( \Delta t \) is 0.02 s, and the maximum absolute error \( t_{\max} = 0.04 \) s. The dispersion will be equal to \( \sigma_t = 0.0133 \) s, and the absolute error of time determination with confidence probability at \( \pm 3\% \) is 0.324\%.

The error in determining the flow rate, which is provided by the invariant automatic stabilization system, can also be affected by its temperature. At the certain ambient temperature, measuring container made of steel will assume a certain volume. A specific volume at the same temperature will also have water with a volume \( V_{ME} \) at a basic temperature level of \( +20^\circ\)C. Comparison of these volumes at different temperatures shows that the difference is quite small. The discrepancy between the volume of water and the volume of the measuring container is only 0.05\%, which allows us to neglect the effect of this factor.

As can be seen, the errors in determining the volume of water input into the measuring container and the time of its filling are sufficiently small and can be neglected in the first approximation, determining the calculation water flow by the formula
\[
Q = \frac{V_{BO}}{t_3},
\]
with a sufficiently high approximation. More accurate determination of the water flow by an invariant automatic stabilization system can be done with the correspondence
\[
\sigma_\Pi = \sqrt{\sigma_\epsilon^2 - \sigma_\phi^2},
\] (40)
where \( \sigma_\Pi, \sigma_\epsilon, \sigma_\phi \) are respectively, the mean square deviations of the invariant system of automatic stabilization of flow, experiment and method.

The mean square deviation \( \sigma_\Pi \) characterizes directly the error of the invariant system of
automatic stabilization of water flow. The mean square deviation $\sigma_e$ determined from the experiment, where the calculated water flow is determined by the formula $Q = \frac{V_{BO}}{t_3}$, considering that $t_3$ and $V_{BO}$ is the filling time and the amount of input liquid are determined without error. The mean square deviation $\sigma_m$ characterizes the error in the determination of $V_{BO}$ and $t_3$.

The measuring capacity is the basis of the facility for the testing of the invariant system of automatic stabilization of water flow into the sand chute of the classifier. The scheme of the facility is shown in Figure 7.

![Figure 7](image1.png)

**Figure 7.** Scheme of the facility for testing the invariant system of automatic stabilization of water flow:
1 – measuring container; 2 – electrical insulating lid; 3 – electrodes; 4 – invariant system of automatic stabilization of water flow into the sand chute of a mechanical single-spiral classifier; 5 – main pipeline; 6 – liquid accumulator; 7 – manometer; 8 – pump; 9 – reservoir; 10 – drain pipe; 11 – drain valve; 12 – pipeline for regulating the liquid pressure; 13 – valve for liquid pressure control

It contains a number of mechanical assemblies and the invariant system of automatic stabilization of water flow 4, operates cyclically. The facility is ready for operation with the closed valve 11. When the pump 8 is switched on from the reservoir 9 via the main pipeline 5, the water is fed into the invariant automatic flow stabilization system, fills it and goes to the measuring container 1. During the initial time, the liquid flow mode is set and then the bottom electrode 3 closes. After filling the measuring container, the upper electrode closes and the pump 8 can be switched off. Battery 6 smoothes the ripple of the liquid. By opening the valve 13 it is possible to regulate the water pressure in the pipeline, as indicated by the pressure gauge 7 indication. After filling the measuring container with water, the valve 11 is opened and it flows into the reservoir 9. The facility after draining the liquid is ready for the next experiment.

According to the scheme (Fig. 7) the facility for carrying out experiments was realized (Fig. 8). It accommodates two pumping units of type K 50-32-125-C, which have a pump 1 and an electric motor 2. Each unit provides a capacity of 12.5 m$^3$/h and a water lift to a height of 20 m. Together, the two units provide a yield of 25 m$^3$/h, which is slightly higher than the water flow, provided by an invariant automatic stabilization system 4. Water by pumps 1 is fed via pipeline 3 to the invariant system of automatic flow stabilization, and then to the measuring container 5. After the experiment, the water is drained into the tank 6. The volume of the measuring container is 5 m$^3$, and the reservoir is 7 m$^3$.

![Figure 8](image2.png)

**Figure 8.** Facility for experiments:
1 – pump; 2 – electric motor; 3 – main pipeline; 4 – invariant system of automatic stabilization of water flow; 5 – measuring container; 6 – reservoir

The automation of the processes in the facility
Fig. 8) is provided by means of the scheme shown in Figure 9. Structurally, it is implemented in the form of a panel of automation (Fig. 10). As a pulse counter, the СБ-1М/100 device was used, the relay Р1 and P2 – МКУ-48, electromagnetic starters – ГМЛ-4 1000×48. In the stand there are units with three-phase electric motors of АИР 80В 2ХУ2 type running at 2850 rpm and their barrels are directly connected to the barrels of centrifugal pumps.

During the experiment, close the drain valve (Fig. 7), pre-setting the required water pressure in the pipeline. The automation panels include automatic circuit breakers that supply power to the transformer, the rectifier of the relay supply, the control circuit of the pump drives. This operation is carried out after the voltage is applied to the automation panel, as indicated by the illumination of the warning light. After that, press the “Start” button on the button station (Figure 10). Electromagnetic starters K1 and K2 (Fig. 9) operate and become self-locking, closing the contacts 1K1 and 1K2. In this mode, the pumps can run as long as desired. Emergency stop can be performed by pressing the “Stop” button. In normal operation of electric motors and pumps, water begins to fill the measuring container. When the DNR lower level electrode closes with the electrode, the relay P1 is triggered, switching the counter of the LI impulses by the contact 1P1, which closed (Fig. 9). The pulse counter determines the time of filling the measuring container with water. When the DVR upper level electrode closed, the relay P2 actuates, which opens the impulse counter with the contact 1P2 and stops the pumps with the contact 2P2. The number of impulses can be counted on the scoreboard according to the position of the arrows. The actuate time of the relays P1 and P2 is chosen to be the same, which ensures that no errors are introduced in determining the filling time of the measuring container. When the drain valve is opened, water flows into the tank, the upper level electrode is out of the liquid, the relay P2 is de-energized, the contacts 1P2 and 2P2 return to the closed state and the circuit returns to its original position. It is ready for a new experiment after the leakage of water and the closing of the drain valve.

In the experiments, the volume of the measuring container \( V_{МЄ} \) did not change and was 5 m\(^3\), the filling time was found from the number of \( n_i \) impulses, and the flow of the invariant automatic stabilization system was determined by the formula

\[
Q_\Pi = 18 \cdot 10^4 \frac{V_{МЄ}}{n_i}, \text{ m}^3/\text{h.} \tag{41}
\]

The foregoing confirms the possibility of implementing an invariant system of automatic stabilization of water flow into the sand chute of a mechanical single-spiral classifier.
mechanical single-spiral classifier and its verification for the accuracy of dosing.

Results. The mathematical model of the hydraulic transducer (13) is obtained thanks to the repeatedly proven hydraulics in practice. It determines the average speed of water flow from the converter, which must be unchanged for its parameters and physical constants. Variable in the model (13) can only be the water level in the hydraulic converter and the flow coefficient $\alpha_3$. The coefficient of flow $\alpha_3$ can not change, because the operating conditions of the hydraulic inverter are constant. Therefore, only change in the average velocity of water $v_z$ from the hydraulic converter will determine the error of its dosing. It is shown that the error is permissible only when the liquid level deviates from the selected value up to 20 mm. Provide an acceptable error of stabilizing the flow of water is possible by automatically maintaining its level in the hydraulic converter.

It is shown that the water level in the hydraulic transducer are unchanged and correspond to the set value when the average velocities in its branch pipe and the flow line are equal. The main drilling influence here is the change in water pressure in the pipeline. It is better to use the principle of drilling control. It is most expedient to build a hydraulic system, where a cylindrical float simultaneously performs the functions of a sensing element, a comparison element, a control element and an actuator. Such a system does not require an external power source. It works best at low pressures in the flow line. In the system, the pressure varies from 0,12 to 0,2 at. The obtained theoretical correspondences (30) and (31) of the acting forces make it possible to realize this automatic control system. It is established that in the steady state ($P_M = \text{const}$) the force acting on the float on the flow side of the pipe is also unchanged under sudden pressure changes and numerically equal to 30,8 H. With instant changes in pressure in the pipeline, the initial force acting on the float changes, which causes it to move. That is, the system carries out a transient process, where the vertical position of the float, the local resistance coefficient of the control element and the water level in the hydraulic converter changes. The minimum size of the system and its high accuracy are achieved when the branch pipe, channel and accumulator are made with a diameter ratio of 1:1,05 - 1,10: 4,5 - 5,5. Increase in accuracy is also facilitated by the relationship between the diameters of the float, pipe, flow line and accumulator. The functions listed in the system the float can perform under the conditions of realization of formula (34), obtained using the exact dependencies that constitute the proven foundations of the classical sciences. The created invariant system of automatic stabilization of the liquid flow corresponds to the condition (34). Mathematical modelling of the float obtained its best parameters, which correspond to the condition (34). It should have a diameter of 200 mm, a height of 120 mm, a wall thickness of 4 mm. It should be made of polystyrene.

An original facility for testing the invariant system of automatic stabilization of water flow into the sand chute of a mechanical single-spiral classifier and the method of its investigation developed. It is established that the error in $Q_n$ determination arises from the change in the dosing conditions by an invariant system of automatic stabilization of liquid flow and changes of the fixed water volumes $V_{BO}$ in separate experiments which do not exactly correspond to the volume of the measured container. Some influence is rendered also by inaccuracies of time fixing of filling of measured container.

In the course of the study, 120 experiments were carried out at different values of water pressure in the pipeline and its temperature. The ambient and liquid temperature varied from 5 °C to...
to 45 °C, and the liquid pressure varied from 0.34 to 0.45 MPa. The experimental data were recorded in a working journal.

In the process of processing the experimental data by the methods of mathematical statistics, was obtained the regression equation

\[ Q_p = -2 \times 10^{-11} Q_\Pi + 24.3, \] (42)

where \( Q_p \) is the calculated (design) water flow by an invariant automatic stabilization system, \( m^3/h \); \( QP \) is the water flow, which took place in the experiment, \( m^3/h \).

The coefficient of correlation by expenditure was \( r_\theta = 1.82 \times 10^{-14} \). Its small value is due to the fact that measurements were made only at one water flow rate and did not cover a certain rather wide range of its variation. It is established that the invariant system of automatic stabilization of water flow is practically not affected by the temperature and pressure of the liquid in the range of their variation.

The method of dispersion analysis was used to evaluate the accuracy of the invariant automatic water stabilization system. The average quadratic deviation of water flow from the results of experiments is \( \sigma_Q = 0.0678557 m^3/h \). Let us estimate the error in determining the water flow rate \( \sigma Q = 0.0043984 m^3/h \). We establish the confidence interval of the error in determining the water flow rate with a probability of 0.995 (\( t = 2.85 \)). In this case we have \( \sigma_Q \pm t \frac{\sigma_Q}{\sqrt{T20}} \), let us take its right value + \( \sigma_Q = 0.069 m^3/h \). According to Gauss' law, the threefold value of the mean square error corresponds to a confidence probability of 0.9973. In this case \( 3 \sigma_Q = 0.207 m^3/h \), and the relative error of automatic stabilization of the water flow rate is \( \pm 0.85 \% \), which corresponds to the requirements for an invariant system for stabilizing water flow into the sand chute of a mechanical single-spiral classifier.

**Conclusions.** Thus, the invariant system of automatic stabilization of water flow into the sand chute of a mechanical single-spiral classifier contains a controlled object, a uniflow hydraulic converter and a float, automatic level control of the moving fluid in it. The cylindrical float is under the action of a jet of water flowing to the upper surface of the pipe, and the buoyancy force of the aquatic environment. It simultaneously implements the functions of the sensing element, the comparison element, the control element and the actuator, which do not require additional power supply. The developed theoretical justification makes it possible to implement such a control system that realizes the principle of drilling control and ensures high accuracy and the lowest metal intensity. By modelling the main part of the float system, its parameters are justified, which guarantee the operability and high accuracy of stabilizing the water flow. The float should be made of polystyrene with a diameter of 200 mm, a height of 120 mm and a wall thickness of 4 mm. The diameter of the pipe supplying water is 50 mm, the outlet pipe is made the same. The diameter of the channel is 100 mm, and the water level in the hydraulic converter is 1028 mm. The water flow is 24.3 \( m^3/h \), and the relative error of its automatic stabilization is \( \pm 0.85\% \).

The prospect of further research is the development of invariant systems of automatic stabilization of the flow of liquid into various technological processes and their application to the ore-concentration plant with various types of mechanical single-spiral classifiers.

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MATHEMATIC MODELING OF CONCENTRATION OF COARSE ORE AS A MEASURE FOR POSSIBLE OVERLOAD OF A BALL MILL

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Abstract: The purpose of the work is a mathematical modeling of the coarse ore concentration as a measure of the possible overload of a ball mill on the basis of the energy of effective destruction of the material. This work is based on the methods of the ball mill theory, idealization method, modeling method, analysis methods, methods of ore milling technology, methods of the theory of elastic material destruction.

The scientific novelty is represented by estimating the overload of a ball mill with coarse ore concentration in a pulp through energy-efficient destruction of solid matters in the shell of the process unit.

The practical significance of the obtained results is high, since they enable to provide a prospect to advance in the development of technical means for evaluating the overload of a ball mill.

It is shown that the trend of changing all the criteria for the concentration of coarse ore in the pulp is similar. The amount of coarse ore at certain circulating load increases linearly with the increase in the productivity of a ball mill. Similar patterns are observed in the mill shell. Concentration of coarse ore in a pulp is a measure of the overload of a ball mill with ore. The concentration of coarse fractions in the mill may be affected by a change in output power.

Keywords: ore milling, optimal circulating load, output, overload, evaluation.

Introduction. Considering the fact that the share of iron ore extraction in iron industry would not reduce, iron-ore concentrates extracted from low-metal content ores at ore-processing plants become more popular. Automation of production processes has significantly increased the efficiency of ore-processing plants. However, the cost price of iron-ore concentrates of domestic plants is higher than foreign analogues. This fact has negative effect on competitive ability of such product at the world market. One of the methods for reduction of cost price of domestic iron-ore concentrate is the automation of production process. In view of this fact, this article is topical since it is aimed at solution of one of the tasks of this problem. The critical nature of this topic is also confirmed by state documents focusing on energy and materials saving in ore mining and the inclusion of these tasks into science topics of Central Ukrainian National Technical University.

Means for automation of ore processing process have been developed for a long time [1, 2, 3, 4, 5, 6, 7, 8]. In later period, these studies are continued in works [9, 10, 11, 12, 13, 14, 15, 16, 17, 18]. Moreover, studies are continued and devoted to solution of separate critical issues. Work [19] focuses on the criticality of the first stages of milling cycle, i.e. magnetic separation while work [20] places emphasis on methods for automatic monitoring and control of iron mass fraction. Work [21] is devoted to creation of adaptive control of iron ore milling process under uncertain characteristics of the facility. Work [22] pays special attention to robust control of these processes. Work [23] is devoted to quality management system used in ore streams of iron-ore careers.

The stagnation in the development of ore-processing management systems is conditioned by the absence of primary means for process flow control [11], relatively low reliability of many information means and their relatively high value [12]. These means include devices for monitoring ball mill overloading with ore. Modern widely used sound ranging instruments have a lot of disadvantages [24]. The use of combined devices that include sound ranging instruments and vibration signals do not make difference.

They cannot determine the overloading of ball mills with ore precisely. Therefore, it is necessary to keep these mills underloaded. This event has negative effect on the performance.

According to the authors [11], it is necessary to use parameters, which can be monitored automatically and characterize the efficiency of material destruction process, in mill shell to find the way out. However, for this purpose, it is necessary to determine material destruction measure that characterizes overload. Nobody has performed studies for justification of this parameter.
The purpose of this work is mathematical modelling of the concentrations of coarse ore as a measure of ball mill overloading based on energy of efficient material destruction.

**Materials and Methods.** One of the critical indexes of ball mills that operate in the first mill stage is base ore performance that can be determined by the formula [25]

\[ Q = k D (2.5...2.6) L, \]

where \( k \) is a coefficient that depends on the nature of ore; \( D \) is mill diameter, m; \( L \) is mill length, m. Using the dependence (1), it is necessary to determine the performance of ball mill with \( D = 4 \) m, \( L = 5 \) m in different operation conditions specified in [25]. Let’s assume that the size of base ores is equal to 85% grain: - 25 mm, - 19 mm, - 12 mm and – 6 mm, and this ore is milled up to the size that is equal to 85% grain: - 0.208; - 0.107; - 0.074 mm. Let’s use \( k \) values specified in table 62 [25]. The dependencies of change of mill performance from the size of base ore at different value of milled materials are shown on figure 1 according to calculation data.

The size of base ore is accepted to be equal to the value so that average-weighted grade size is equal to average value of diameter \( d_c \) of respective grade with reduction of 15% excessive fractions in terms of size.

The performance of mill increases at certain size of milled material as you can see on figure 1 and decreases at reduction of base core size. The sensitivity to base ore size increases at the increase of the size of drilled material. Ore hardness also has significant effect on operation of ball mill. According to S.F. Shinkorenko [26] ores that refer to banded iron formation is divided into 2-5 groups from five hardness groups. It is possible to assume that banded iron formations refer to ores of average hardness and ores of excessive hardness. As is shown in work [11] on the example of rather thick ores against ores of medium hardness, relative performance of ball mills reduces and is equal to approximately 0.78. Considering this fact, figure 1 b shows the change in performance of the same mill upon processing of excessively thick ore. As you can see on figure 1 b, the performance of the mill in processing hard ores reduces with the increase of the feed size. However, the sensitiveness to the size is lower.

**Figure 1.** Mill performance dependency with \( D = 4 \) m, \( L = 5 \) m of the first milling stage from average-weighted size of ore at different size of milled ore (65% grain): 1 – 0.074 mm; 2 – 0.107 mm; 3 – 0.208 mm; a – medium hardness of ore; b – excessive hardness of ore

These dependencies allow selecting operation mode of ball mill. However, it is also necessary to determine certain circulating load since the increase of the latter results in the increase and performance of process unit [27]. The value of circulating load at set mill size depends on the ore crushing size and hardness. Circulating load increases with the increase of ore crushing size and hardness [28]. There are optimal values of performance and circulating loads that ensure maximum final product output [29].

Upon optimal performance of ball mill for certain materials, there is optimal value of circulating loads that is equal to 170-650%
Let’s model the process of change in concentration of coarse fractions in mill pulp at different operation modes.

Mathematical modelling is shown on the example of ball mill МШР 4.0 x 5.0. Let’s assume that the mill passes material that is mixed well within feed change range. Conditions of efficient mixing of materials in ball mill are described in work [33]. It is well known that granulometric composition of sands is not constant under different performances of ball mill. Sand size reduces with the increase of unit performance. Nonetheless, let’s assume that granulometric composition of sand product is unchanged, i.e. the size varies insignificantly and meets typical modes. Size yield of sand product is equal to: -0.056 mm – 2.4%; -0.074 mm – 14.1%; -3.0 mm – up to 100%. Weighted-average size of coarse sand fraction is equal to \( d_{sp} = 1.5 \) mm. Let’s take real product of ore processing plant with the following content as base ore: -0.056 mm (3.3%); +0.056 mm (0.4%); +0.07 mm (1.0%); +0.16 mm (1.4%); +0.3 mm (1.3%); +0.5 mm (1.4%); +0.8 mm (4.9%); +2 mm (2.2%); +3 mm (11.0%); +5 mm (26.0%); +10 mm (24.5%); +15 mm (11.6%); +20 mm (11.0%). Let’s use product of final size -0.074 mm (14.1%) and in pieces with average-weighted size of 1.5 mm (85.9%). Let’s consider mill feed in the form of three products: - 0.074 mm (3.7%); averaged-weighted size of fine grains \( d_f = 1.21 \) mm (23.2%) and averaged-weighted size of coarse grain \( d_c = 11.12 \) mm (73.1%). Let’s assume that coarse grain of feed is represented by spherical bodies 11.12 mm in diameter.

We will change the performance of a ball mill from 150 t/h to 180 t/h every 10 t/h and circulation load from 100% to 800% of the minimum ore flow every 100% in the course of mathematical modelling. Solid/liquid ratio will be maintained at the level of 80%. Let’s assume that ore density remains constant and equal to 3.3 g/cm³.

The purpose of experiments was to determine certain constant value of ore flow in mill at certain constant circulating load. In the course of the experiment, values that characterized process flow, i.e. density and content of solids in pulp around coarse ore, the concentration of coarse particles in one liter of mixture, share of layers of large spherical ore pieces in one liter of pulp at the area of 100 cm² were calculated by means of specially develop PC
software. Cube face size where each coarse ore pieces may be located at even distribution in material volume, mean distance between coarse pieces at even distribution and the distance from the edge of coarse ore piece to cube face are used as indexes of concentration of coarse particles.

Considering modelling results, it is necessary to point out the following. Density and the content of solids in pulp that surrounds coarse ore particles in mill drum (table 1) do not remain stable in different operation modes. The increase in circulating loads results in considerable increase of these indexes. Upon change in circulating load from 100% to 800%, the density of pulp that surrounds coarse solids increase from 2000 at base ore flow of 150 t/h to 2213 g/l. The content of solids increases from 71.7% to 78.6% at 80% for all material mass. The increase of ore mass flow in ball mill results in reduction of these indexes. At low circulating loads, the pulp that surrounds coarse pieces is relatively loose. When the ball applies pressure on material, fluid components easily passes through coarse particles and results in coarse fraction raise. High circulating loads cause the increase in density that results in fixing of coarse ore in fluid material mass.

Table 2 shows indexes of concentrations of coarse ore in pulp at charging area of ball mill at its different operation modes.

<table>
<thead>
<tr>
<th>Mass flow of base ore, t/h</th>
<th>Circulating load, percentage of initial ore consumption</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>100</td>
</tr>
<tr>
<td>150</td>
<td>2000</td>
</tr>
<tr>
<td>160</td>
<td>1989</td>
</tr>
<tr>
<td>170</td>
<td>1979</td>
</tr>
<tr>
<td>180</td>
<td>1970</td>
</tr>
</tbody>
</table>

Table 1. The density and content of solids in pulp that surrounds coarse ore in mill drum in different modes of its operation (density, g/l/solid content, %)

<table>
<thead>
<tr>
<th>Mass flow of base ore, t/h</th>
<th>Circulating load, percentage of initial ore consumption</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>100</td>
</tr>
<tr>
<td>150</td>
<td>15.32</td>
</tr>
<tr>
<td>160</td>
<td>15.18</td>
</tr>
<tr>
<td>170</td>
<td>15.03</td>
</tr>
<tr>
<td>180</td>
<td>14.89</td>
</tr>
</tbody>
</table>

Table 2. Values of concentration of coarse ore in pulp at charging area of ball mill at different operation modes

<table>
<thead>
<tr>
<th>Mass flow of base ore, t/h</th>
<th>Circulating load, percentage of initial ore consumption</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>100</td>
</tr>
<tr>
<td>150</td>
<td>4.20</td>
</tr>
<tr>
<td>160</td>
<td>4.06</td>
</tr>
<tr>
<td>170</td>
<td>3.91</td>
</tr>
<tr>
<td>180</td>
<td>3.77</td>
</tr>
</tbody>
</table>

Considering data of table 2, the trend of change in all criteria of concentration of coarse particles in pulp is similar. The increase of circulating load the values of these indexes increase. There indexes show decrease upon the increase of ore flow in mill. However, the sensitivity of criteria to the change in circulating load differs. The change of circulating load from 100 % to 800% results in the increase of cube face size by 1.63 times when the average distance...
between coarse pieces and the distance from the edge of coarse spherical ore to cube face increase by 3.3 times. Obtained data show that the increase in circulating load results in reduction of concentration of solids in pulp. The increase in feeding of base ore in mill results in increase of the concentration of coarse particles in pulp.

The dependency of weight, volume, the quantity of ore pieces in pulp volume unit at charging area of pulp mill from base flow is shown on figure 2. Fig. 2 shows that the nature of dependencies is similar and pulp condition may be determined based on any of them. Weight (figure 2, a), volume (figure 2, b) and quantity of ore pieces (figure 2, c) show similar behaviour only at change of the performance and circulating load of grinding cycle. For instance, the quantity of coarse ore at certain circulating load increases in linear proportion upon increase of the performance of ball mill. The sensitivity of dependency is not very high.

At low circulating loads, the quantity of coarse ore in certain limited pulp volume is high. It decreases quickly with the crease of circulating load.

For instance, upon the increase of circulating load from 100% to 200%, the quantity of coarse ore increases by 1.5 times. Upon further increase in circulating load, the process of change in quantity of coarse ores slows down. Obtained data show that it is possible to select necessary quantity of coarse pieces in full volume of pulp under any production conditions.

The pulp in ball mill can be also characterized by the quantity of layers consisting of coarse spherical particles. Let’s assume that spherical particles of similar size that is equal to 11.12 mm contact between each other at the points of protruding spheres that are located in line. Then 81 pieces will be placed within the area of 100x100 mm. Table 3 shows results of these pulp compositions that are peculiar to conditions of such studies. As you can see from table 3, the quantity of layers of coarse pieces decreases with the increase of circulating load. The increase in the performance results in the increase of the quantity of layers of coarse materials. At these conditions, the quantity of coarse material layers is equal to 3.43…3.74 at 100% circulating load and 0.76…0.90 at 800%. It is possible to set different values of this index by changing cycle operation mode.
Table 3. Share of layers of coarse spherical ore with averaged weighted diameter in one litre of pulp at different operation modes of ball mill and dense packing within the area of 100 cm$^2$

<table>
<thead>
<tr>
<th>Mass flow of base ore, t/h</th>
<th>Circulating load, percentage of initial ore consumption</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>100</td>
</tr>
<tr>
<td>150</td>
<td>3.43</td>
</tr>
<tr>
<td>160</td>
<td>3.54</td>
</tr>
<tr>
<td>170</td>
<td>3.65</td>
</tr>
<tr>
<td>180</td>
<td>3.74</td>
</tr>
</tbody>
</table>

Obtained dependencies of the change in quantity of coarse solids that may be located almost within one layer at the area of 100 cm$^2$ are shown on figure 3. The dependency 4 shows (fig. 3) that the quantity of particles increases linearly with the increase in base ore feed in ball mill but doesn’t reach complete filling of 100 cm$^2$ area within output change. Such mode corresponds to mill underload. The mode that corresponds to dependency 3 may be nominal mode. The dependency 2 may correspond to the mode that characterizes the limit of possible mill overload since the area contains the quantity of coarse particles that exceeds one layer. The mode for dependency 1 corresponds to the area of considerable overload. This is conditioned by the fact that effective ore grinding takes place at high speed of loading of particles that are destroyed [11]. It is impossible to reach high speed of loading of solid particles since they will be compacted and loaded gradually upon contact with the ball while taking basic energy of grinding body and reducing its speed. Upon creation of such structures of coarse ore, the particles will not be destructed completely and, as a result, coarse solids will accumulate with further mill overload. Particles of coarse material located in one layer will be loaded at high speeds. This will ensure their efficient destruction. Under such conditions, coarse fractions will not accumulate in the drum and, as a result, will not cause overload.

Upon falling of the ball, one, two, three, four etc. ore pieces may be located in destruction area in certain situations. The largest ore volume under the ball before destruction will correspond to closely packed parties of cubic shape. Figure 4 shows interaction of falling ball with coarse ore fractions.

Figure 3 The dependency of coarse solids $N$ that may be placed within the area of 100 cm$^2$ upon their extraction from the pulp during grinding process at different circulating loads and change in base ore feed :: $1 - Q_{uw} = 500\%$; $Q_{aux,min} = 600\%$; $3 - 700\%$; $4 - 800\%$

Figure 4. Position of falling ball before destruction (a) and after destruction (b) of ore: 1 - lining; 2 ore particles; 3 - ball

Under the highest concentration of coarse ore particles, solid block 2 (figure 4, a) with the height that meets the size of one cubic particle $d_p$ will be located in destruction area. Let’s assume that solid block has cylindrical shape of $d_p$ height...
and $r$ radius (figure 4, b). Its volume will be as follows

$$V_{PT} = \pi r^2 d_p.$$  \hspace{1cm} (2)

It is under the pressure of falling ball 3. Of right-angled triangle $A_1OB$

$$r^2 = d_p \left(2R_K - d_p\right).$$  \hspace{1cm} (3)

Considering (3), the volume of solids being under the pressure of falling ball will be as follows

$$V_{PT} = \pi d_p^2 \left(3R_K - d_p\right).$$  \hspace{1cm} (4)

Under the effect of falling ball, material begins destruction from central part. Further, its peripheral parts also start destruction while contributing to resistance. Since the energy of falling ball is excessive, the latter almost reach the lining (figure 4, b). In this case, solid volume appears to be destructed. This volume corresponds to the share of the ball submerged into solids. The volume of ore that is destructed under the effect of falling ball at the highest space filling at effective area of milling body will be as follows

$$V_{PP} = \frac{\pi d_p^2}{3} \left(3R_K - d_p\right).$$  \hspace{1cm} (5)

Fine ore in watery environment are pressed out under the ball to the peripheral area through openings between coarse classes. Under-milled solids formed due to partial destruction of coarse pieces at peripheral areas of the ball will remain in the volume of existed ore cylinder. The volume of remaining under-milled coarse solids in ball area can be calculated through the following dependency

$$V_{PP} = \frac{\pi d_p^2}{3} \left(3R_K - 2d_p\right).$$  \hspace{1cm} (6)

Solids volumes $V_{PT}$, $V_{PP}$, $V_{PP}$ allow approximate characterization of the process of interaction between solid and ball in the mill under maximum ore load.

To use these dependencies, it is necessary to justify the limitation, considering conditions of possible partial destruction of ore particles of maximum size at peripheral areas of the ball. The position of end points on the surface of the ball where coarse ore will be destructed partially may be justified by means of figure 5. It is necessary to create the force that would not push off ore fraction to the peripheral areas upon ball falling to ensure the destruction.

End point, at which above-mentioned events do not take place, will be B point on ball surface (fig. 5). This point relates to ball surface, is projected at straight angle to OB radius and should be placed under the angle of $45^\circ$ against AC straight line. Forces aimed at compression of ore fraction take place in the points located from the left of B and push out forces take place in the points located from the right of B. Therefore, B point is and end point where it is possible to determine limiting conditions for the size of ore pieces. Maximum size of ore piece will be equal to the length of section BE, i.e. $BE = d_p$.

$$d_{P\text{max}} = R_K - A_1O = R_K \left(1 - \frac{1}{\sqrt{2}}\right).$$  \hspace{1cm} (8)

Thus, it is necessary to use the limit (8) for each ball diameter upon the use of the dependences (4), (5) and (6). However, it should be specified. Ore of maximum size will be in the section under the effect of two sections $A_1B$, i.e. $2A_1B = \frac{2}{\sqrt{2}}R_K$. Solids will be effected by the part of sphere with $AA_1$ height. In section, it is limited by circle with $A_1B$ radius at maximum size of solids. The reduction in size of solids results in reduction of circle radius. This circle be located in parallel to the circle with $A_1B$ radius. Therefore,
maximum quantity of solid particles in destruction areas will be determined by the area of circles that are parallel to geometrical figure of the largest radius \( A_1B \). Circle radius and area are determined by the size of solid fractions.

The dependencies of ore volumes that are under the pressure falling ball, destruct and remain under-milled from the size of their fractions are shown in figure 6. Considering the dependencies (fig. 6), it is possible to assume that the volume of destructed material and under-milled ore fractions make the volume that is under the pressure of falling ball. Ratio between volumes of milled and undermilled material changes with change in the size of ore fraction. The share of undermilled material reduces with the increase of the size of ore fraction. This fact suggests the uncertainty of the dependency of the volume of destructed material and material that is under the pressure of falling ball, i.e. concentration of solids.

Measuring abilities of this approach can be characterized by ratio of these volumes based on dependencies (4) and (5)

\[
V_d = \frac{V_{pt}}{V_{pp}} = \frac{3(2R_K - d_P)}{(3R_K - d_P)}.
\]

The dependencies of change in volume ratios \( V_d \) from ball size and ore fractions are specified in table 4. Values at ore fraction size \( d_P = 4 \) mm were taken as basic values of volume ratio. Data of table 4 suggest the mismatch between volumes of destructed material and that, which under pressure of falling milling body, may reach considerable values at relatively small ball sizes and considerable ore size. This mismatch increases with the increase of solid size. If balls have considerable sizes, these indexes improve significantly for the same material. They are most perceptible when the size of ore fraction is equal to 0-12 mm. Thus, ore loading should be identified at certain distance from charging throat of the mill since the particle size reduces and ball diameter increases at certain distance from it.

**Table 4.** The relation between the volume of destructed material and solid volume that is under the pressure of falling ball at different sizes of the ball

<table>
<thead>
<tr>
<th>Parameters</th>
<th>Size of ore pieces, mm</th>
<th>4</th>
<th>8</th>
<th>12</th>
<th>16</th>
<th>20</th>
<th>24</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>( R_K = 30 ) mm</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>( V_d )</td>
<td></td>
<td>1.9535</td>
<td>1.9024</td>
<td>1.8462</td>
<td>1.7838</td>
<td>1.7193</td>
<td>1.6364</td>
</tr>
<tr>
<td>( \Delta V_d ), %</td>
<td></td>
<td>0</td>
<td>2.61</td>
<td>5.50</td>
<td>8.69</td>
<td>12.24</td>
<td>16.23</td>
</tr>
<tr>
<td></td>
<td>( R_K = 40 ) mm</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>( V_d )</td>
<td></td>
<td>1.9655</td>
<td>1.9285</td>
<td>1.8888</td>
<td>1.8462</td>
<td>1.8000</td>
<td>1.7500</td>
</tr>
<tr>
<td>( \Delta V_d ), %</td>
<td></td>
<td>0</td>
<td>2.85</td>
<td>5.5</td>
<td>8.69</td>
<td>12.24</td>
<td>16.23</td>
</tr>
<tr>
<td></td>
<td>( R_K = 50 ) mm</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>( V_d )</td>
<td></td>
<td>1.9726</td>
<td>1.9437</td>
<td>1.9130</td>
<td>1.8806</td>
<td>1.8462</td>
<td>1.8095</td>
</tr>
<tr>
<td>( \Delta V_d ), %</td>
<td></td>
<td>0</td>
<td>1.46</td>
<td>3.02</td>
<td>4.66</td>
<td>6.41</td>
<td>8.27</td>
</tr>
</tbody>
</table>

It is possible to identify the loading of mill with ore (determination of the coarse solid concentration) at certain sensitivity of the process. According to expression (5), first derivative \( \frac{dV_{pp}}{dd_P} \) characterizes the sensitivity of the process. It is equal to

\[
\frac{dV_{pp}}{dd_P} = \pi d_P (2R_K - d_P) \quad (10)
\]

This dependency suggests that the sensitivity of the process depends on ore size \( d_P \) and ball size \( R_K \). At mill input, the sensitiveness is ensured by maximum size of ore particles but it is
considerably reduced due to small sizes of balls. At the certain distance from charging throat, the size of ore particles reduces. This fact results in reduction of the sensitivity. However, it is recovered due to larger diameter of balls in these sections of the drum. It may be concluded that the sensitivity along the drum changes within relatively broad range and is sufficient almost in its any section.

It should be noted that it is possible to identify solid size using the dependency (5) upon setting ball mill to the mode that is close to maximum load. The dependency of the volume to be destructed from solid size at certain ball diameters is shown on figure 7. Figure 7 shows that it is possible to determine ore size in given drum section at certain dependency that is determined by ball size through known destructed solid volume \( V_{PP} \). The size of balls in certain drum section is known and may be remained constant.

These dependencies were obtained at the highest solid concentration in destruction area that actually meets to conditions that are close to overload. In real conditions, solid concentration will be smaller. However, considered dependencies describe this process since the reduction in concentration can be considered by calculating loosening coefficient. Therefore, obtained dependencies are valid and can be used in practice.

Upon movement of the material through mill drum, the size of solid particles reduces. Average weighted size of coarse solid can be determined at certain distance from charging throat of the mill according to milling kinetics. Average weighted diameter of coarse solids in certain point of the length of the mill can be calculated by the following formula [32]

\[
d_{ci} = d_{c aux} \cdot e^{-k[t \ln(t+1)]}^m,
\]

where \( d_{ci} \) is average weighted diameter of milled material; \( d_{c aux} \) is average weighted size of output material; \( e \) is a base of natural logarithms; \( k, m \) are coefficients associated with ore strength ; \( t \) is milling time.

The quantity of pieces with average weighted diameter is as follows [34]

\[
R = R_0 \cdot e^{-k\left(\frac{d_{c aux}}{d_{ci}}\right)}^m,
\]

where \( R_0 \) is a quantity of average weighted pieces of coarse material in final product.

By means of equation (11) and (12), it is possible to determine average weighted size of coarse pieces and their quantity. These data may be used for determination of the conformity between processes at the output of ball mill and at the certain distance from it. The concentration of coarse but smaller particles in this point will also determine the loading of ball mill with ore and its overload.

Thus, the concentration of coarse ores in pulp volume is a measure for loading and overloading of ball mill. It characterizes the process directly at the input. It is better to determine it at certain distance from loading throat while operating smaller but better homogenized solid particles. It is possible to affect the concentration of coarse pieces in pulp volume at certain concentration of ore through value of initial feed.

**Results.** It has been demonstrated that the performance of ball mills at certain size of milled material increases with the decrease of base core size and, vice versa, it reduces in case of use of stronger ores. The sensitivity to the size of base ore increases with the increase of the size of milled material and reduces when stronger ores are used. The output of process aggregate increase with the increase of circulating load. However, its value at set coarseness is determined by reduction ratio and ore strength. The increase in reduction ratio and ore strength results in the increase of circulating load. There are optimal values of performance and circulating load that ensure the highest output of finished product. In general, circulating load is equal to 100...200% of initial feed. The range of change in circulating load at certain content of prepared size in ball mill
Solid density and content in pulp that surrounds ore particles on mill drum change in different operation modes. These indexes increase considerably with the increase of circulating load. If circulating load is 100...200%, the density of pulp between coarse pieces of ore is 2000...2100 g/l and slightly reduces up to 1970...2077 g/l with the increase of mill output from 150 to 180 t/h. Such pulp density results in fixing of coarse ore particles in fluid material mass.

The process of interaction of solid and milling bodies in the mill under maximum ore loading may be characterized by solid volume that is under pressure of falling ball \( V_{Fr} \), ore volume destructed under the effect of ball and volume \( V_{P3} \) of residual undermilled coarse solids in effective area of the ball. Maximum size of destructed ore pieces is determined by the diameter of falling ball.

Upon change in the size of ore pieces, the ratio between volumes of milled and undermilled material remains unchanged. This suggests uncertainty of the dependency of the volume of destructed material and material that is under the pressure of falling ball. This index improves considerably and ensures sufficient precision of identification of concentration of coarse pieces that meets mill loading or overloading with material upon the increase of ball size and reduction in ore size. Considering the fact that the coarseness of ore pieces reduces and the ball size increases in a ball mill with the increase of the distance from discharging throat, the overload of process aggregate with ore should be 1/4 ...1/3 of drum length. This does not contradict to the sensitivity of detection of solid concentration since it changes within broad ranges and is sufficient almost in any section.

It has been disclosed that change of all criteria of concentration of coarse ore in pulp is similar. The increase in circulating load results in reduction of the concentration of coarse ore in pulp. The increase of base ore feed slightly increases ore concentration. For instance, the mean distance between coarse spherical pieces of ore, at their smooth distribution within the pulp, is equal to 4.2...6.78 mm at circulating load of 100...200%. This value reduces slightly within 3.77...5.75 mm with the increase of the output of ball mill from 150 to 180 t/h.

The nature of change of mass, volume and quantity of coarse ore in pulp volume unit is similar and any of these features may be used for determination of the material condition at the output of ball mill. For instance, the quantity of coarse pieces of ore shows linear increase at certain circulating load with the increase of the output of ball mill. At small circulating loads, the quantity of coarse ore in certain pulp volume is rather high. It drops significantly with the increase of circulating load. When circulating load increases from 100 to 200%, the quantity of coarse particles of ore in the same volume reduced by 1.5 times. At optimal circulating load, it is possible to adjust the quantity of coarse particles of ore in limited pulp volume by the change in output of ball mill.

The pulp in ball mill may be characterized by the quantity of ball, conventionally made of coarse spherical pieces that are located in certain volume of material. The quantity of coarse ore balls reduces with the increase of circulating load. The increase in output result in the increase in the quantity of balls of coarse material. It has been shown that this index may be used for obtaining underload and overload of ball mill and nominal load that ensures optimal output. It is possible to set different values of this index by changing the feed of base ore. Thus, the concentration of coarse particles of ore in the pulp is a measure of underload, rated operation and overload of ball mill.

This also relates to material milling process at certain distance along the drum of ball mill. The concentration of coarse but smaller ore in certain drum section will also determine mill load and overload. Here, a certain delay will take place but the quantity of coarse ore will be high and their smoothness will be better since ball mill is almost ideal mixer where the pulp mixes smoothly while reducing the content of coarse ore in fluid material even at starting points of movement.
Considering above-mentioned facts, it is possible to conclude that the concentration of coarse particles of ore in pulp volume is a measure of load and overload of ball mill. These indexes can be determined at certain distance from loading throat when smaller but better homogenized coarse material. The volume of initial feed of mill can affect the concentration of coarse pieces.

The validity of obtained results is confirmed by the fact that use process flow models are based on conditions and actual materials verified in practice. Assumptions made upon modelling do not exceed the limits applied in modelling of this processes and their results are confirmed experimentally.

**Conclusions.** Thus, mathematical modelling of the concentration of coarse ore as a measure of possible overload of ball mill suggest the following. There are optimal values of performance and circulating load that ensure the highest output of finished product for each process type of ore. In general, circulating load is equal to 100...200% of initial load and determined by the range of variation of strength and coarseness of base ore at constant coarseness of material at process unit discharging. A circulating load of 100...200%, the density of pulp is equal to 2000...2100 g/l and slightly reduces up to 1970...2077 g/l with the increase of mill output from 150 to 180 t/h. Such pulp density results in fixing of coarse ore particles in fluid material mass.

The process of interaction of solid and milling bodies in the mill may be characterized by solid volume that is under pressure of falling ball, ore volume to be destructed and volume of undermilled material. The diameter of falling ball determines maximum size of completely destructed body. Therefore, at certain size of milling body, the ratio between destructed and non-destructed volumes of ore changes upon variation of the size of coarse pieces milled material. This index levels the increase of ball sizes and reduction in ore size and makes the evaluation of concentration of coarse ore at certain distance from loading throat valid. The sensitiveness of measurement of load or overload of ball mill is sufficient in this case.

The nature of change of all criteria of concentration of coarse ore in pulp is similar. For instance, the quantity of coarse pieces of ore shows linear increase at certain circulating load with the increase of the output of ball mill. The same situation is observed in different section along mill drum. The concentration of coarse ores in pulp volume is a measure for load, rated load and overload of ball mill. It characterizes the process directly at the input. It is better to determine it at certain distance from loading throat while operating smaller but better averaged solid particles. It is possible to affect the concentration of coarse pieces of pulp volume at certain concentration of ore through value of initial feed. It is better to determine these indexes at certain distance from loading throat while operating smaller but better homogenized solid particles. It is possible to affect the concentration of coarse ore in pulp volume at certain concentration of ore through value of initial feed.

The subject of future studies is the modelling of coarse ore destruction in ball mill.

**References**

METHOD AND DEVICE OF AUTOMATIC NON-DESTRUCTIVE CONTROL OF MAGNETIC IRON CONTENT IN SLURRY FLOW

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Abstract The object is to investigate the automatic non-destructive control of iron ore slurry characteristics in ore concentration and to propose the method and the device of automatic non-destructive control of magnetic iron content in slurry flow.

The methods of systematization and analysis of specifics of ultrasonic waves propagation, mathematical and computer modelling, computer technologies and experimental investigations provide the results.

The scientific novelty consists in method of controlling the magnetic iron content in the slurry flow improving based on the Lamb waves propagation parameters assessment. The expression for determining the fraction of the ferromagnetic component in the slurry solid phase which contains the ratio of the measured Lamb waves amplitudes both in the magnetic field and without it and also contains the ratio of the Lamb waves amplitudes in the presence of the slurry and pure water in the measuring module (with no magnetic field was considered).

The practical significance of the studies is a flow chart of the device based on the Lamb waves propagation for controlling the magnetic iron content in the slurry proposing.

The results of the research investigate the regularities of the Lamb waves propagation in a metallic plate in contact with the iron ore slurry under various characteristics of the magnetic field was proposed. On the basis of the regularities of ultrasonic waves propagation in a metallic plate in the presence of the magnetic field, the authors analyze basic factors determining the magnetic susceptibility value, in particular, the size, form and initial magnetization of solid phase particles in the iron ore slurry. The method of controlling the magnetic iron content in the slurry flow is suggested based on the Lamb waves propagation parameters assessment.

Key words: automatic control, magnetic iron, ore slurry, ultrasound, magnetic field

Introduction. Currently, real time measurements of the useful component content in the slurry flow are essential for optimizing iron ore magnetic concentration. The methods of non-destructive on-line control based on ultrasonic, magnetic, microwave and radiometric means are considered the most promising. In particular, research work [1] considers application of ultrasonic volume waves to controlling milled material characteristics. It should also be noted that ultrasonic (elastic) waves have been widely used lately. The reason for this is their specific features like relatively large energy concentration in a wave because of its small localization layer and the possibility of receiving an ultrasonic signal from any point of the surface (including the curvilinear one) along which it propagates.

Materials and methods
1. Systematization and analysis of specifics of ultrasonic waves propagation
Let us consider basic types of ultrasonic surface waves and features of their propagation, primarily, with regard to the problems of controlling industrial slurry characteristics.

As is known, the equation of the isotropic homogenous elastic medium movement can be presented as follows [2-6]:

\[ \frac{\partial^2 \overrightarrow{U}}{\partial t^2} = \mu \nabla^2 \overrightarrow{U} + \rho (\lambda + \mu) \nabla \nabla \cdot \overrightarrow{U}, \]  

where \( \overrightarrow{U} \) is a displacement vector of the medium particles; \( \rho \) is the medium density; \( \lambda \) and \( \mu \) are elastic constants (the Lamé parameters) of the medium; \( \Delta \) is the Laplace operator.

The displacement vector is presented as:

\[ \overrightarrow{U} = \overrightarrow{U}_t + \overrightarrow{U}_r, \]  

where \( \overrightarrow{U}_t = \nabla \Phi \) ; \( \overrightarrow{U}_r = \text{rot} \overrightarrow{\Psi} \) and \( \Phi \) and \( \Psi \) are scalar and vector potentials. Thus, we obtain the equations describing longitudinal and transversal waves correspondingly:

\[ \rho \frac{\partial^2 \overrightarrow{U}_l}{\partial t^2} - (\lambda + 2\mu) \Delta \overrightarrow{U}_l = 0, \]  

\[ \rho \frac{\partial^2 \overrightarrow{U}_t}{\partial t^2} - \mu \Delta \overrightarrow{U}_t = 0. \]  

The ultrasonic Rayleigh waves are the most investigated and widely used among the known surface waves at present [4]. These waves propagate along the boundary of the solid space \( Z > 0 \) (Fig. 1, a).

For this case, the equations (3) and (4) can be transformed into a system of linear homogenous equations with relation to arbitrary constants \( A \) and \( B \).
The next type of ultrasonic waves is the waves inherently similar to the Rayleigh waves, yet, they are horizontally polarized. These are Love waves and they also satisfy the equation (1) being its linearly independent solution. It corresponds to the case when the wave vector is in the plane $XZ$ and the displacements are parallel to the axis $Y$, i.e. $U_y \neq 0$, $U_x = U_z = 0$ and $\partial / \partial y = 0$. Then, the equation (1) will look like

$$\rho \frac{\partial^2 \vec{U}}{\partial t^2} = \mu \Delta \vec{U},$$

where $\Delta \vec{U} = U_y \vec{v}_0$. The Love waves being the surface ones come into existence due to the addition of a solid layer to the semi-space and it becomes a load for the latter (Fig. 1, b). As weak heterogeneousness of the surface layer of the solid body often occurs in practice and can be easily created deliberately, the Love waves enjoy a wide practical application.

The third basic type of ultrasonic waves at the boundary of two semi-spaces is Stoneley waves [2]. Let the plane harmonic surface wave propagate in the direction of the positive axis $X$ along the plane boundary $Z = 0$ of two semi-spaces (1), (2) (Fig. 1, c). As before, we will suppose that a wave in each semi-space consists of a sum of longitudinal and transversal plane waves, each of them is a solution of the equations (3) and (4) with corresponding values $\rho, \omega, \mu$. The real root $K_o$ corresponds to the surface Stoneley wave and satisfies the condition

$$K_o > K_{t1}, K_{t2}.$$  

If one of the semi-spaces is a solid body and the other one is fluid, the equation describing the wave movement looks like

$$4k^2 q_{f1} = (k^2 + \omega^2)^2 = \frac{\rho_x}{\rho_1} q_{f1}^2,$$

where $\rho$ is the fluid density; $q_{f1}^2 = k_x^2 / k_x$. This equation differs from the Rayleigh equation (7) as it contains the right part that considers the influence of fluid on the solid semi-space. The thickness of layers of the Stoneley waves localization in the fluid makes

$$Z_o \gg \lambda_x,$$

in the solid body

$$Z_o \approx \lambda_x / 2\pi.$$  

We also refer the waves in planes including normal horizontally polarized waves (transversal normal waves) and normal vertically polarized waves or Lamb waves (Fig. 1, d) to surface ones. In
transversal normal waves, there is only one
transversal normal waves, there is only one
displacement component \( U_x \) parallel to the plane
surface and perpendicular to the direction of the
wave propagation, i.e. the deformation in the
transversal wave is a pure shear. The basic property
of transversal normal waves including the Lamb
ones is the fact that with the designed values \( \omega \)
and \( h \) only a certain number of waves can
propagate in the plate. The larger this ratio is
\[
2h/\lambda = \omega h/\pi c, \quad (13)
\]
the larger the number of waves is.

Specificity of the Lamb waves propagation
along the surface in contact with industrial slurries
is presented in detail in [5].

Besides the mentioned basic types of
surface ultrasonic waves, there are also several
varieties [4]: waves in a fluid layer on the solid
semi-space; waves in a solid layer on the solid
semi-space; quasi-volume waves in crystals;
Bleustein-Gulyaev waves (surface waves in crystals
with piezo-properties), etc. These varieties are
applied in the sphere of ultrasonic control over
characteristics of industrial slurries because of
their specific features.

Analyzing basic types of ultrasonic surface
waves we can draw the following conclusions. The
Rayleigh waves have the largest energy
concentration on the solid body surface, yet, the
characteristics of their propagation depend much
on the condition of the surface along which they
propagate. The Love waves are characterized by
their strong dependency on the heterogeneous
surface layer due to which they exist, this fact
making the measuring surfaces, along which they
propagate, vulnerable and, thus, "unstable". The
Stoneley waves (considered here) propagate in
both fluid and solid semi-spaces and their
component propagating in the fluid semi-space is
under the influence of the same perturbing factors
as the usual volume ultrasonic oscillations. For
example, one should expect a strong dependency
of their attenuation value on the content of
gaseous bubbles in industrial slurries.

Taking into account the fact that the walls
of technological tanks and trunks at concentration
plants are made of sheet metal, it is convenient to
apply the Lamb waves for ultrasonic control of the
media parameters in contact with them. These
waves are characterized by high energy
concentration and less subjected to perturbing
factors as the Rayleigh and Love waves are.

2. Basic regularities of ultrasonic surface
waves propagation in the metallic plate with the
magnetic field

Application of ultrasonic surface waves to
developing and realizing methods and technical
means of milled material characteristics control
allows considerable expansion of their potential
[7-9]. The basic expressions describing the Lamb
surfaced waves propagation in a metallic plate in
contact with the slurry flow are in [5]. From
research [6] it follows that the presence of the
magnetic filed causes additional rate attenuation
and dispersion of volume ultrasonic waves
propagating in the investigated medium.

Let us consider the influence of the
magnetic field on surface waves propagation. Let
this wave propagate in the solid perfectly
conducting semi-space with the constant
magnetic filed \( H_0 \). In this case, the elastic wave is
accompanied by variable electric and magnetic
fields and currents.

The elasticity equations and the systems
of Maxwell equations considering the elements
movement and the volumes of the conducting
semi-space in the magnetic field should be
satisfied [10]

\[
(\lambda + G)\nabla U + G\Delta U - \rho \frac{\partial^2 U}{\partial t^2} + \mu \nabla \times \left[ \nabla \times (U \times \mathbf{R}_o) \right] = 0, \quad (14)
\]

where \( \lambda, G \) are Lame parameters; \( \rho \) is the medium
density; \( \mu \) is the medium magnetic susceptibility;
\( U \) is the displacement vector in the wave. As in [5],
we present the displacement vector in the
following way

\[
\vec{U} = \nabla \phi + \nabla \psi, \quad (15)
\]

where \( \nabla \psi = 0 \). The scalar \( \phi \) and the vector \( \psi \)
potentials of the vector \( \vec{U} \) are connected with
the volume expansion

\[
\nabla U = \Delta \phi \quad (16)
\]

and revolution

\[
\nabla U = -\Delta \psi. \quad (17)
\]

The equation (14) solutions in the form
solutions in the form
corresponding to the plane harmonic waves
propagating in the direction of the axis \( x \) are the
expressions

\[
U_{x,y,z} = \frac{A, B, C}{k} e^{kz + i(kx - \omega t)}, \quad (18)
\]
where \( k \) is a wave number; \( \beta(k) \) is a function of \( k \); \( A, B, C \) are arbitrary constants. In case of the weak magnetic field the following solutions are valid \( [6] \)

\[
U_x = \frac{A}{k} e^{\beta(k)kz} e^{i(kx - \omega t)} \times e^{i(kx-\alpha t)} + \Delta \left[ \frac{\eta}{2} \right],
\]

\[
U_y = \frac{A(h_y)}{m} e^{\beta(k)kz} \times \left[ -e^{\beta(k)kz} + \frac{0.25\eta^2h_y \beta(k)kz}{(\beta(k)kz)^2 - \eta^2} \right] \times e^{i(kx-\alpha t)} + 0(h_y),
\]

\[
U_z = \frac{i\beta(k)}{k} \left[ -e^{\beta(k)kz} + \frac{2}{2 - \eta^2} e^{\beta(k)kz} \right] \times e^{i(kx-\alpha t)} + \Delta \left[ h_y \right]^2,
\]

where \( k_R \) is the wave number of the Rayleigh wave with \( H_0 = 0 \); \( \alpha \) is a correction to \( k_R \) because of the magnetic field; \( \eta = k_t/k_R \). For the metallic plate made of steel 12X18 H10T

\[
\alpha = \frac{2.09 - 0.3(h_y/h_x)h_y - 0.16h_y}{4 - 0.57h_x - h_y}. \tag{23}
\]

Thus, the one-component magnetic field changes the phase velocity of the wave and the two-component field changes the wave structure.

The magnetic field causes the third displacement \( U_y \) proportional to \( \sqrt{h_x - h_y} \) (Fig. 2, 3).

In strong magnetic fields, the displacements are determined by the following expressions \( [10] \)

\[
U_y = \frac{A}{k} e^{\beta(k)kz} e^{i(kx + \omega t) + \Delta(h_y^2)} + \frac{1}{2} \left[ 1 + \frac{h_y}{2(1 + h_x)(m - h_y)h_y} \right],
\]

\[
U_{x,t} = \Delta(h_x^2),
\]

\[
k = \frac{k_t}{1 + h_y^2} \left[ m - \left( \frac{3}{2} \right) \left( m + 1 \right) h_y - \frac{1}{2} \frac{h_x}{h_y} \right],
\]

\[
\beta = \frac{1}{2h_x} \left[ \left( 1 + \frac{1}{2(1 + h_y^2)} \right) \left( m - h_y^2 \right) \right].
\]

Expressions (24) and (25) indicate the presence of the surface wave connected with the magnetic field only. The possibility to use this wave depends on the magnetic field intensity. The wave only exists with those values \( h_x \) which satisfy the condition \( \beta > 0 \).

The above mentioned also indicates that in the two-component magnetic field (\( h_x \neq 0 \) and \( h_y \neq 0 \)) the volume (transversal) wave is transformed into the surface one.

---

**Figure 2.** Dependency of \( \alpha \) on \( h_x \) with various values of \( h_y \)

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35
3. Method of controlling magnetic iron content in the slurry flow based on measuring parameters of the Lamb waves propagation

The considered regularities of the Lamb waves propagation along the metallic plate in contact with the slurry flow in the presence of the magnetic field allow forming the method of controlling the magnetite-iron content in its solid phase.

In the presence of the magnetic field (the transversal one, $H_z \neq 0, H_x = 0, H_y = 0$) the Lamb wave will attenuate according to the law:

$$I_\nu = I_\nu(0) \exp \left\{ -k(1 - \alpha(H))l \right\} \times \exp \left\{ \frac{\left(1-W_{1}\right)p_{W_1}W_{\rho_{n_1}}}{\rho_{n_1}} C_1 l \right\}.$$  \hspace{1cm} (26)

Here, $I_\nu(0) = I_\nu(0) \exp \left\{ -k(1 - \alpha(H))l \right\}$ is the wave attenuation function in the absence of the contacting fluid medium (water, slurry, etc.), $\alpha(H)$ is the parameter considering the magnetic field impact (if $H = 0$, then $\alpha(0) = 0$). $\rho_{n_1}$ is the density of the plate material.

According to the above mentioned the parameter considering the magnetic field impact is determined by:

$$\alpha(H_2) = \mu_2 H_2 \text{Const},$$  \hspace{1cm} (27)

where $\mu_2$ and $H_2$ are magnetic permeability and intensity of the magnetic field in the plate along which the Lamb wave propagates. If the field is homogenous, the magnetic permeability of the controlled medium can be determined by the expression:

$$H_2 \approx \frac{(nJ_0)\mu_1}{\mu_2},$$  \hspace{1cm} (28)

where $H_2$ is magnetic permeability of the controlled medium, $n$ is the number of turns in a coil, $J_0$ is the current in a coil. It is presumed that $(l_1 >> l_2)$ and $(\mu_1 < \mu_2)$. Substituting (28) in (27), we obtain

$$\alpha(H_2) = \mu_1(nJ_0)\text{Const}.$$  \hspace{1cm} (29)

Then, the method of measuring implies the measurements of the wave amplitude in the presence of the magnetic field ($I_\nu(H)$) and without it ($I_\nu(0)$) and the value is calculated

$$\ln \frac{I_\nu(H)}{I_\nu(0)} = \alpha(H)kl = \mu_1[kl(nJ_0)\text{Const}],$$  \hspace{1cm} (30)

whence it follows that

$$\mu_1 = \frac{1}{kl(nJ_0)\text{Const}}\ln \left( \frac{I_\nu(H)}{I_\nu(0)} \right) =$$

$$= C_3 \ln \left( \frac{I_\nu(H)}{I_\nu(0)} \right)$$

where we find from the expression:

$$C_3 = \frac{1}{kl(nJ_0)\text{Const}}.$$  \hspace{1cm} (31)

According to the method of controlling the concentration of the slurry solid phase on the basis of changed parameters of the Lamb waves propagating along the metallic plate in contact with the slurry and the dependencies of iron ore slurry magnetization in the magnetic field mentioned above we can present the following expression for determining the fraction of the ferromagnetic component in the slurry solid phase.
\[ \eta = \left( \frac{\rho_{\text{ma}} - \rho_{\text{wo}}}{\rho_{\text{wo}}} \right) C = A \frac{(\mu_2 - 1)H}{\ln\left( \frac{I_{\text{v}}}{I_{\text{v}}} \right)} . \]  

(32)

Substituting (31) into (32) we acquire

\[ \eta = A \frac{(\mu_2 - 1)H}{\ln\left( \frac{I_{\text{v}}}{I_{\text{v}}} \right)} = (A n J_c) \frac{(C_s \ln\left( \frac{l_{\text{v}}(H)}{l_{\text{v}}(0)} \right) - 1)}{\ln\left( \frac{l_{\text{v}}}{l_{\text{v}}} \right)} = (33) \]

The expression (33) numerator contains the ratio of the measured Lamb waves amplitudes both in the magnetic field and without it and the denominator contains the ratio of the Lamb waves amplitudes in the presence of the slurry and pure water in the measuring module (with no magnetic field).

The conducted research indicates that the value calculated according to (33) does not depend on the solid phase concentration and at the same time it explicitly determines the magnetic iron content in the controlled volume of the slurry.

4. Device for controlling the magnetic iron content in the slurry flow

The device for controlling the magnetic iron content in the slurry flow (Fig. 4) consists of a multivibrator 1, starting univibrators 2, 16, 18, 36, generators 3, 17, 19, 37, forming prisms 5, 7, 23, 24, radiating transducers 4, 25, receiving transducers 8, 26, amplifiers 9, 27, temporary selection blocks 11, 29, forming univibrators 12, 28, pulse transducers 13, 30, analog-digital transducers 14, 31, a microcomputer 15, a data panel 42, coils 20, 38, a magnetic circuit 21, a measuring vessel 6, 22, an accumulating vessel 32, pipelines 33, 34, 35, valves 40, 41.

The device controlling the magnetic iron content in the slurry flow functions as follows. The iron ore slurry from the accumulating vessel 32 (the technological tank) along the pipelines 33, 34 enters the measuring vessels 6, 22. In the measuring vessel 22, it is affected by the magnetic field generated in the coil 22 and the magnetic circuit 21 by means of the generator 17 and the starting univibrator 16.

In the walls of the measuring vessels 6, 22 the Lamb waves are formed by means of the starting univibrators 2, 16, generators 3, 17, piezoelectric transducers 4, 25 and forming prisms 5, 23. Passing the fixed distance along the measuring vessels walls, the Lamb waves are transformed into electric signals by forming prisms 7, 24 and piezoelectric transducers 8, 26 and are amplified by amplifiers 9, 27. In the temporary selection blocks 11, 29, an information component of the received signals is distinguished by means of univibrators 12, 28 and it is transformed into a signal of direct current of the same amplitude by transducers 13, 30. By means of analog-digital transducers 14, 31 these signals are input into the microcomputer 15. The microcomputer 15 starts the system by means of multivibrator 1 and the data is output by the D-A converter 42.

Preliminary de-magnetization of the iron ore slurry is performed by the impulse magnetic field of decreasing amplitude formed by the univibrator 36, the generator 37 and the coil 38. The valves 40, 41 are applied for controlling slurry and water flows to ensure the system operation and calibration.
Results. The ultrasonic control system was tested as part of the automated system of controlling iron ore concentration [11-17]. The calibration (preliminary adjustment) of the ultrasonic control system of the magnetic iron content in the slurry flow was performed as follows. In the absence of the controlled medium in the measuring vessel, the amplitude of the Lamb wave with the magnetic field \( I_{\nu}(H) \) and without it \( I_{\nu}(0) \) is determined and the value is calculated

\[
\ln \frac{I_{\nu}(H)}{I_{\nu}(0)} = \mu_1 [k l(n_0)\text{const}] = \frac{\mu_1}{c_3}.
\]  

(34)

The magnetic permeability for air is almost 1 and, then, the expression (34) will look as

\[
\ln \frac{I_{\nu}(H)}{I_{\nu}(0)} = \frac{1}{c_3},
\]  

whence it follows that
The conducted testing indicates that the given device ensures stable measurements of the solid and magnetic iron content in the ore slurry. The measuring error of the solid percentage in the slurry makes ± 0.25 % abs.; The measuring error of the magnetic iron percentage in the slurry solid phase makes ± 0.25 % abs.

**Conclusion.** As in processing lines of concentration plants, the walls of technological tanks and trunks are usually made of sheet metal, it is convenient to apply the Lamb waves to implementing ultrasonic control of the parameters of the media in contact with them. These waves are characterized by high energy concentration and are less subjected to perturbing factors than the Rayleigh and Love waves.

The parameters of the Lamb waves propagation in the metallic plate in contact with the iron ore slurry depend on the characteristics of the applied magnetic field and the magnetic susceptibility of the slurry. To measure the magnetic iron content in the slurry flow, one should measure the attenuation values of the Lamb waves propagating along the metallic plate in contact with the controlled medium in the presence of the magnetic field and without it.

We suggest the system of ultrasonic control of the magnetic iron content enabling us to maintain the optimal useful component content in the concentrated ore with changing qualities of the initial ore and the technological equipment conditions.

**References**

9. Morkun, V.S. Method of controlling parameters of slurry solid phase and device for realizing it. A.c. 1780412. MKI G 01 № 29/02. № 4012450/33; 23.01.86; DSP.
The research work focuses on the mathematical description of cavitation processes in the slurry, including cavitation ones. The research works of some investigations [1], it is revealed that in the process of ore crushing, hematite, martite, iron hydroxide and magnetite particles are fixed on the surface of quartz grains. As a result of some investigations [1], it is revealed that in the process of ore crushing, hematite, martite, iron hydroxide and magnetite particles are fixed on the surface of quartz grains. The stability of the mineral fixation depends on the efforts making them combine and mineral types. The fixed mineral particles are not removed fully from the quartz surface either by intensive radiation or by mechanical rubbing. The iron oxide particles can be removed from the surface of the quartz grains only when processing the slurry by ultrasound.

Materials and methods

1. Research and publication analysis

Under the action of ultrasound in the fluid medium, some physical, chemical and physical-chemical processes occur. They include cavitation, radioactive pressure and acoustic flows [2]. Cavitation is caused by the fact that all fluids are too sensitive to tensile forces. Under the influence of powerful ultrasonic oscillations in the fluid, the areas of contraction and vacuum appear. In the oscillation phase causing vacuum, a great number of gaps in the form of cavitation bubbles appear in the fluid, which close abruptly in the next phase of contraction.

The research work [3] indicates that cavitation processes intensify the cleaning of the ore particle surface from films and stuck pollutants. Under the action of powerful ultrasound in the fluid medium, there appear acoustic flows that intensively stir the slurry, reduce the thickness of the laminar layer at the boundary with the solid body and facilitate the removal of diffusive restrictions [4]. This process plays an important role in ultrasonic cleaning of the ore particle surface from mineral pollutants.

In mineral floatation in water previously processed by ultrasound the extraction of minerals into the froth product increases as...
compared to the floatation without water processing [5]. The research results of the ultrasound influence on minerals of titanium-zirconium deposits are presented in the research work [6]. It is revealed that the processing during 1-3 minutes with the oscillation frequency of 20 kHz and the intensity of 3.8 W/cm² radically activates mineral floatation.

To explain the reasons for the kinetics facilitation we suggest three hypotheses:

1) ultrasound makes the selective removal of particles possible [7];
2) ultrasound cavitation can modify the mineral particle surface by means of creating micro- and nano-bubbles and enhance the fixation of bubbles-particles [8];
3) ultrasound can influence on the probability of particles collision.

Thus, the analysis of the scientific sources reveals the efficiency of ultrasound application for intensifying the processes and solving certain technological problems in useful mineral concentration. The range of ultrasound application is great and its potential is not realized. The prospect of applying acoustic methods provides the opportunity of the efficient concentration upgrading.

2. Research results

The proportion between the mixture density and the volume fraction of the air \( \alpha \), according to the cavitation model considered in [9, 10] is described by the formula:

\[
\frac{\partial}{\partial t}(\rho) = (\rho_1 - \rho_{\text{vap}}) \frac{\partial}{\partial t}(\alpha),
\]

where \( \rho \) is the mixture density, \( \rho_1 \) is the fluid density, \( \rho_{\text{vap}} \) is the air density, \( \alpha \) is the volume fraction of the air. The volume fraction of the air \( \alpha \) is determined as follows:

\[
\alpha = f \frac{\rho}{\rho_{\text{vap}}}. \tag{2}
\]

The mass transfer rates are defined by such equations, if \( \rho \leq \rho_{\text{vap}} \) we have [9, 10]:

\[
R_e = C_e \frac{v_{ch}}{\sigma} \rho_1 \rho_{\text{vap}} \sqrt{\frac{2(\rho_{\text{vap}} - \rho)}{3(\rho_1)\alpha}} f, \tag{3}
\]

otherwise:

\[
R_c = C_c \frac{v_{ch}}{\sigma} \rho_1 \rho_{\text{vap}} \sqrt{\frac{2(\rho - \rho_{\text{vap}})}{3(\rho_1)}} f, \tag{4}
\]

where \( v_{ch} \) is the characteristic frequency, which is the approximation to the local turbulent geometry, \( C_e \) and \( C_c \) are empiric constants.

The general rate of the interphase mass transfer per volume unit considering the bubble density when applying the assumption about the equal bubble size is determined on the basis of the cavitation the Zwart-Gerber-Belamri model [11]:

\[
R = \frac{3\alpha \rho_{\text{vap}}}{R_b} \sqrt{\frac{2(\rho_{\text{vap}} - \rho)}{3 \rho_1}},
\]

where \( R_b \) is the bubble radius; \( \rho_{\text{vap}} \) is the air pressure. In the equation (5) the mass transfer rate of a unit volume is connected with the air phase density \( \rho_{\text{vap}} \) only. This equation is subject to the bubbles growth. To apply it to the process of bubble popping we use this generalized expression [9]:

\[
R_e = F \frac{3\alpha \rho_{\text{vap}}}{R_b} \sqrt{\frac{2(\rho_{\text{vap}} - \rho)}{3 \rho_1}} \text{sign}(\rho_{\text{vap}} - \rho), \tag{6}
\]

where \( F \) is the empiric calibrated coefficient. The rate of one bubble mass change is obtained as follows

\[
\frac{dm_b}{dt} = 4\pi R_b^2 \rho_{\text{vap}} \sqrt{\frac{2(\rho_{\text{vap}} - \rho)}{3 \rho_1}}.
\]

For \( n_b \) of the bubbles per volume unit, the volume fraction of the air is determined by the formula:

\[
\alpha = V_b n_b = \frac{4}{3} \pi R_b^3 n_b. \tag{8}
\]

With the growing volume fraction of the air, the density of the area where the bubbles appear increases. To simulate this process Zwart-Gerber-Belamri [11] suggested changing \( \alpha \) to \( \alpha_{\text{nuc}}(1-\alpha) \) in (6). As a result, we obtain the following form of the cavitation model for \( \rho \leq \rho_{\text{vap}} \):

\[
R_e = F_{\text{vap}} \frac{3\alpha_{\text{nuc}}(1-\alpha) \rho_{\text{vap}}}{R_b} \sqrt{\frac{2(\rho_{\text{vap}} - \rho)}{3 \rho_1}}, \tag{9}
\]

otherwise:

\[
R_c = F_{\text{cond}} \frac{3\alpha \rho_{\text{vap}}}{R_b} \sqrt{\frac{2(\rho - \rho_{\text{vap}})}{3 \rho_1}} f, \tag{10}
\]

We can achieve the stable cavitation mode as shown in [12] if the acoustic frequency is small (<1 MHz) and the pressure amplitude is much smaller than the air static pressure (101 kPa). Under these conditions, a bubble is in a steady cavitation state and oscillates near its initial radius on a periodic basis. To describe this process, we suggest using the empiric equation based on the Keller-Herring model [9, 13]:

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\[ R_0 \equiv \frac{3}{2} \left( M' u \right) \frac{u_m}{f_0^2}, \]  

(11)

where \( R_0 \) is the bubble radius, micron; \( f_0^2 \) is the acoustic frequency. It should be noted that under higher pressure the reaction of bubbles depends on the amplitude of the acoustic field pressure to a greater degree and, thus, the equation (11) is not possible under these conditions of “inertial cavitation”.

In the work by K. J. Carvell [13], the author obtains the expressions allowing us to calculate the initial bubble radius for the maximum expansion depending on the acoustic frequency and pressure amplitude:

\[ R_{optimal} = \frac{1}{\sqrt{0.0327F^2 + 0.0679F + 16.5P^2}}, \]  

(12)

where \( P \) is the pressure amplitude for the acoustic sinusoidal wave, MPa, \( f \) is the frequency, MHz; \( R_{optimal} \) is the initial optimal bubble radius, micron. For instance, if \( f = 1 \) MHz, \( P = 1 \) MPa, the optimal bubble radius is 0.2454 micron.

Having transformed the dependency (12), we obtain the quadratic equation which contains the function of the optimal frequency for a certain bubble size:

\[ 0.0327F^2 + 0.0679F + 16.5P^2 \left( \frac{1}{R_{optimal}} \right) = 0, \]  

(13)

in which \( F(R) \) with the known pressure \( P \) can be defined from the expression:

\[ F(R) = \frac{0.0679 \pm \sqrt{0.0679^2 - 4 \cdot 0.0327 \cdot (16.5P^2 \cdot \frac{1}{R})}}{2 \cdot 0.0327}. \]  

(14)

Thus, the cavitation mode with the known gas bubble size \( R_{optimal} \) is achieved under the influence of the high-energy ultrasound with the frequency \( F(R_{optimal}) \) on the slurry. Let us indicate the function of bubble size distribution by \( f(R) \). In this case, the value \( f(R) \, dR \) determines the fraction of the bubbles the size of which is from \( R \) to \( R + dR \).

Table 1 shows the values of the function \( f(R) \) used in the calculation. The function \( f(R) \) graph is shown in Fig.1 [14].

Table 1. Values of gas bubble distribution function

<table>
<thead>
<tr>
<th>Index</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>( R_{1m} )</td>
<td>0.0003</td>
</tr>
<tr>
<td>( R_{2m} )</td>
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</tr>
<tr>
<td>( R_{1m} )</td>
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<tr>
<td>( R_{2m} )</td>
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<tr>
<td>( R_{1m} )</td>
<td>0.005</td>
</tr>
<tr>
<td>( R_{2m} )</td>
<td>0.010</td>
</tr>
</tbody>
</table>

To simulate the acoustic signal propagation in the heterogeneous medium in the conditions of the changeable sound speed and the changeable density, the method k-space is used [15]. For the two-dimensional medium without losses, the equation looks like [15, 16]

\[ \frac{1}{\rho(r)c(r)} \frac{\partial^2 p(r,t)}{\partial t^2} = \nabla \cdot u(r,t) \]  

(15)

where \( u \) is the vector of the speed oscillation of the acoustic fraction with the components \( u_x \) and \( u_y, p \) are fluctuations of the acoustic pressure, \( \rho(r) \) is the medium density, \( c(r) \) is the sound speed in the medium; \( r \) is the vector of the coordinates \( x, y \). The wave equation of the second order looks as follows [15, 16]

\[ \nabla \cdot \left( \frac{1}{\rho(r)c(r)^2} \nabla p(r,t) \right) = \frac{1}{\rho(r)c(r)^2} \frac{\partial^2 p(r,t)}{\partial t^2} = 0. \]  

(16)

The result of the simulation of cavitation processes under the ultrasound impact (the impulse amplitude is 0.45 MPa, the impulse length is 2.75 of the cycle, the frequency is 6.25 MHz) is shown in Fig. 2. The result of the simulation of cavitation processes under the ultrasound impact (the impulse amplitude is 0.25 MPa, the impulse length is 5.5 of the cycle, the frequency is 5 MHz) is shown in Fig. 3.
Conclusion

The analysis of the major directions and approaches to intensifying ore concentration has revealed that application of high-energy ultrasound of certain intensity for preliminary iron ore slurry processing enables increase of the useful component output in the floatation product by disintegrating ore floccules.

The conducted investigation of the regularities of cavitation development allows forming the method of determining the optimal frequency of high-energy ultrasound to maintain cavitation processes in the slurry to disintegrate floccules and, thus, increase the efficiency of the floatation ore concentration.

References


COORDINATED AUTOMATIZED CONTROL OF AN ORE-PROCESSING ENTERPRISE AS A TECHNICAL-ORGANIZATIONAL SYSTEM

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Abstract: The primary objective of the paper is to develop a method of automatized control of ore-preparation and ore-dressing technological processes at the ore-dressing and processing enterprise, whose structural units have their own preferences in relation to technological and organizational processes. The methods for achieving this objective are as follows: analysis of domestic and foreign experience, systematization of available approaches and methods, methods of numerical simulation for synthesis and analysis of mathematical model, methods of mathematical statistics and probability theory for processing the results of experiments, methods of analytical design and computer simulation in the synthesis and analysis of control system, methods of system analysis in the development of control algorithms. The scientific novelty is in the method of coordinated automatized control of the ore-processing enterprise as a technical-organizational system consisting of agent and agent-controlled technological processes, both of which have fixed operation technology. Rational behavior of technical-organizational system participant under consideration is to minimize their own costs taking into account all available information about technological and organizational processes. Stressing the practical significance of the study, it was thus suggested to improve efficiency of the enterprise as a whole, as well as its certain structural elements including a control center, a mine, a crushing plant, an ore dressing plant. Results. The research sets out the results of the investigated approach to ore-dressing process control at the ore-dressing enterprise as a technical-organizational system in market economy terms with a view of activity and own preferences of structural elements of the ore-dressing enterprise. The automatized control process of technical-organizational system “control center – ore-dressing plant” was considered and general optimization model with restrictions was proposed. Simulation analysis makes it possible to conclude that the proposed approach of organizational and technical production management in the mining and processing operations allows to improve the control quality. Key words: coordinated control, automation, ore-dressing, technical-organizational system.

Introduction. One of the crucial aspects of effective automatized control of large industrial enterprises, for instance, an ore-dressing and processing enterprise is to provide optimal performance both for the enterprise as a whole and its certain structural elements such as a control center, a mine, a crushing plant, a processing plant. A large industrial enterprise should be viewed as a technical-organizational system in order to address the issue more effectively. Ore-processing technological processes and organizational interaction of individual structural units are used as a basis in designing an automatized control of such systems.

Materials and methods
1. Analysis of problem solving state
Methods of automatized control of ore-preparation and ore-dressing technological processes at ore-dressing and processing enterprise, based on classical approaches and intelligent control theory, are developed in [1-7]. Analysis of these works suggests that for improving automatized control efficiency of ore-preparation and ore-dressing technological complex it is necessary to take into account such factors as technological types ratio of processed ore and behavior of enterprise organizational processes – activity of its units. Control mechanisms of active system with a focus on organizational component of production processes subject to human factor are considered in papers [8-10]. In general, task of active system automatized control is described as follows.

In model constructing, control process is considered from the standpoint of control center [9]. Consequently, it is necessary to describe control center preferences and consider its decision making model under interaction with specified units (agents).

Agent and agent-controlled technological process have fixed operation technology. Dependence \( w(\cdot) \) of activity result on action and conditions is considered to be known to all participants of technical-organizational system (TOS) and cannot be changed [9]. Changing technology \( w(\cdot) \) of agent-controlled technological process as a passive object can be solved by methods of classical control theory. Agent preferences on set of possible performance are specified by its utility function \( v(\cdot) \), and activity result \( z \in A_0 \) depends on action \( y \in A \) and conditions
\( \theta \in \Theta \) in a certain way: \( z = w(y, \theta) \). Control law \( W(\cdot) \) is determined by the function \( w(\cdot) \) reflecting a structure of the passive controlled object and information \( I \), which has agent at time of decision-making.

Conditions set \( \Theta \) is known to all participants TOS and fixed. Decision-making model of control center is generally similar to above mentioned decision-making model of agent and is described by tuple, wherein subscript «\( \Theta \)» denotes a variable selected center \( \Psi_0 = \{ U_0, U_1, U_i, A_0, \Theta, w(\cdot), v_0(\cdot), I_0 \} \) \([9-10]\). Under uniformity of decision making models described in multi-level hierarchical systems control center can be regarded as a subject. It is controlled by a higher-ranking center and agent as a center, which controls lower-ranking agent. Control center actions are: \( U_0 \in U_0, u_0 \in U_0, u_1 \in U_1, u=(u_0, u_1, u_i) \in U_0 \times U_1 \times U_i \) is a control vector. Indices correspond to the control subjects: index «A» refers to institutional control, «V» - motivational, «I» - informational.

In most models of technical-organizational system it is considered that the only role of control center is to implement the control, i.e. it does not have its own result of the activity. Thus, the result of center is generally considered as a result of agent activities \([8-10]\), which are controlled using modern approaches \([11-15]\). Since center preferences \( v_0(\cdot) \) are defined, including the set of possible outcomes \( A_0 \) agent activity, which are depended on agent actions and conditions, so control center function is to stimulate agent to choose certain actions. In case of uncertainty in the system, it is assumed that unit, using particular rule, eliminates uncertainty and makes decision in full awareness.

Thus, objective of this work is to develop a method of automatized control of ore-preparation and ore-dressing technological processes at the ore-dressing and processing enterprise, whose structural units have their own preferences in relation to technological and organizational processes.

2. Method of coordinated automatized control of a mining and-processing enterprise

Mineral processing of iron ore in Ukraine includes raw ore mining, crushing, milling and magnetic dressing as shown in Fig. 1. All these processes are realized at the mining and processing enterprises (MPE), including a mine, a crushing plant (CP) and an ore-dressing plant (ODP).

The task of the control center is the collection and analysis of mine, CP, ODP production indices and development a coordinated optimal plan of mining and processing of ore types. The most important unit of mining and processing enterprises is an ore-dressing plant, products of which determine efficiency of the entire plant.

Consider the automatized control process of technical-organizational system "control center – ore-dressing plant". In general, agent objective function is described as the difference between amount of stimulation proceeds and production charges \( f(y) = \sigma(y) - c(y) \), and center - the difference between sale proceeds and stimulating charges \( \Phi(y) = H(y) - \sigma(y) \) \([9]\). All components of the objective functions should be expressed to single monetary value.

It should be noted, that most of mining and processing enterprises process several technological ore types \([3]\). Quantity and quality of concentrate from various ore types differ with their characteristics: \( \rho_j \) - iron content in the j-th ore type; \( \eta_j \) - extraction of iron from the j-th ore type; \( \sigma_j \) - extraction of concentrate from the j-th ore type; \( w_j \) - specific energy cost per ton of processed j-th ore type \((j=1,\ldots,N_0)\), \( N_0 \) - the number of ore processed types.

Action of ore-dressing plant as an agent is performance of control center task for production of given concentrate volume \( y \). To fulfill the task an ore-dressing plant must process \( \overline{m} \) tone of \( N_0 \) ore types, which depends on the task of control center and the technological characteristics of ore types

\[
m_j = \frac{y_j}{\sigma_j}, \quad j = 1, N_0, \quad (1)
\]

where \( y_j \) is the mass of concentrate, produced by processing of the j-th ore type; \( \sigma_j \) is the concentrate extraction from the j-th ore type; \( N_0 \) is the number of ore types.

Rational behavior of considered technical-organizational system participant is to minimize their own costs in terms of all available information about technological and organizational processes. According to the characteristics of each ore type objective function of agent – ore-dressing plant is as follows

\[
c(y) = \sum_{j=1}^{N_0} \left[ \frac{y_j}{\sigma_j} w_j C_e \right] \rightarrow \min, \quad j = 1, N_0, \quad (2)
\]
where \( w_j \) is the specific energy consumption for processing one ton of \( j \)-th ore type; \( C_e \) is the price per 1 kW•h of electricity.

Objective function of control center is the sale proceeds of concentrate

\[
H(y) = C_e \sum_{j=1}^{N_r} y_j \rightarrow \max,
\]

where \( C_e \) is the contract price per ton of finished ore (concentrate).

Furthermore, the production plan should comply with the contractual obligations of the ore processing enterprise to produce and supply a certain mass of concentrate given quality:

\[
Y^{(l)} \leq y \leq Y^{(h)}, \quad \alpha^{(l)} \leq \alpha(y) \leq \alpha^{(h)}, \quad i = 1, N_i,
\]

where \( Y^{(l)} \), \( Y^{(h)} \) is respectively the minimum and maximum allowable value of concentrate volume; \( \alpha^{(l)} \), \( \alpha^{(h)} \) is respectively the minimum and maximum allowable value of iron content in the concentrate.

The iron content in the concentrate produced from several types of ore is described by the dependence

\[
\alpha(m) = \frac{\sum_{j=1}^{N_r} m_j \rho_j \eta_j}{\sum_{j=1}^{N_r} m_j \sigma_j} = \frac{\sum_{j=1}^{N_r} y_j \rho_j \eta_j}{\sum_{j=1}^{N_r} y_j}, \quad j = 1, N_r,
\]

where \( \rho_j \) – iron content in the \( j \)-th ore type; \( \eta_j \) – extraction of iron from the \( j \)-th ore type.

It should be noted, that the accessible volume of each type for each production line of ore-dressing plant for the \( t \) period is limited:

\[
0 < m_j(t) \leq m_j^{(h)}(t), \quad j = 1, N_r,
\]

where \( m_j^{(h)} \) is the mass of \( j \)-th ore type reserve.

Thus, the control center should select a system of stimulation \( u(y) \) to encourage an ore-dressing plant to fulfill given production plan \( y \).

Knowing that the ODP chooses the action, which maximizes its objective function, control center has to find such a stimulation system, that will maximize its own objective function. Meanwhile, it is considered that hypothesis of benevolence is satisfied. According to the mentioned hypothesis, ODP chooses one of several production plans, which give a global maximum for its objective function being the most favorable for the control center [8-9]. From the control center standpoint stimulation may not exceed the income received by it from ODP activity. From the ODP standpoint stimulation proceeds \( u(y) \) should not be less than production charges \( c(y) \) and reserved utility, which can be obtained by choosing a zero ore producing.

The Pareto optimal plan of control center is appropriate to use the recovery of expenses principle for an ore-dressing plant operation [9]. According to the principle, inducing the ODP for choosing action is enough to compensate it for production costs. Besides compensation costs, the center also pays a motivating premium \( \delta \geq 0 \). Therefore, in order for an ore-dressing plant to produce a required concentrate volume, the simulation from the center for the choice of this action should be at least

\[
u(y) = c(y) + \delta = \sum_{j=1}^{N_r} \frac{y_j}{\sigma_j} w_j \cdot C_e + \delta \quad (7)
\]

Since the stimulation proceeds is equal to ODP production costs, the optimal production plan \( y' \) is the plan, that gives maximum difference
between income of the control center and costs of ODP [9]. Consequently, the optimal realizable plan can be found by solving the following optimization problem

\[ y^* = \arg \max_c \{ H(y) \} \]

It should be noted, that the operational planning of mining and ore processing at enterprises is carried out periodically with several intervals. Thus, ODP objective function takes the form

\[ c(y, t) = \max_{t=1-T} \sum_{j=1}^{N_j} \frac{y_j(t)}{\sigma_j} w_j C_e \rightarrow \min, \quad j = 1, N_r \]

where \( T \) is the number of intervals in the planning period. The objective function of control center takes the form

\[ H(y, t) = \min_{t=1-T} \sum_{j=1}^{N_j} y_j(t) \rightarrow \max. \]

General optimization model in consideration of (8) takes the form

\[ \tilde{y}^* = \arg \max \left\{ \min_{t=1-T} \left\{ \sum_{j=1}^{N_j} y_j(t) \right\} \right\} \]

restrictions

\[ y^{(l)}(t) \leq y(t) \leq y^{(h)}, \quad t = 1, T \]

\[ a^{(l)} \leq a(y, t) \leq a^{(h)}, \quad i = 1, N, t = 1, T \]

\[ 0 < \sum_{t=1}^{T} m_j(t) \leq m_j^{(h)}, \quad j = 1, N_r. \]

Optimization problem (11) is solved using the Sequential Quadratic Programming (SQP) method.

**Table 1.** Characteristics of ore types

<table>
<thead>
<tr>
<th>Characteristic</th>
<th>Types</th>
<th></th>
<th></th>
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</tr>
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<tr>
<td>( \rho ), %</td>
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<td>II</td>
<td>III</td>
<td>IV</td>
</tr>
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<td>29,1</td>
<td>28,5</td>
<td>30,4</td>
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<tr>
<td>( \eta ), %</td>
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<td>78,1</td>
<td>75,4</td>
<td>75,1</td>
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<tr>
<td>( \sigma ), %</td>
<td>34,9</td>
<td>35,1</td>
<td>36,2</td>
<td>38,5</td>
</tr>
</tbody>
</table>

**Results.** A mining and processing enterprise requires in the plan period \( t=1, T \) days, making ore dressing of four ore types with the iron rate in concentrate 63,2-64,5%. Every day’s concentrate volume is required to be in the range of 1300-1600 tons. Testing the coordinated control method of ore mining and processing enterprise technological and organizational processes for the seven-day period (number of intervals \( T=7 \)) was specified. The cost for 1 kW•h of the electric power is 0,9733 UAH. The cost for 1 ton of a concentrate is 821,49 UAH. Characteristics of ore types are shown in Tab. 1.

The change in the main indices of the ore-dressing process is shown in Fig. 2-4. The function of the enterprise’s income (Fig. 2) is within the limits of 1-1,2 million UAH. The iron content in the concentrate (Fig. 3) throughout planning is within the set limits of 63,2-64,5%. Thus, the necessary output of a concentrate (Fig. 4) is provided.

**Figure 2.** Income of the enterprise

**Figure 3.** Iron content in concentrate

**Figure 4.** Amount of produced concentrate

Thus, the offered approach of organizational and technical production control at the mining and processing enterprise allows to improve the control quality.

**Conclusion.** The approach under consideration provides new prospects for increasing quality of automatized control of mining and processing enterprise as a technical-organizational system in market economy terms.
taking into account own preferences of each structural unit.

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ADAPTIVE CONTROL AND IDENTIFICATION OF DRILLING SYSTEM WITH CONSIDERING THE TYPE OF DRILLED ORE MATERIAL

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Abstract: The object is to investigate methods for forming a model for the system of adaptive control over drilling with the control object identifier.

The following methods have been used in the course of the research: analysis of domestic and foreign experience, systematization of available approaches and methods, methods of numerical simulation for synthesis and analysis of mathematical model, methods of mathematical statistics and probability theory for processing the results of experiments, methods of analytical design and computer simulation in the synthesis and analysis of control system, methods of system analysis in the development of control algorithms.

The scientific novelty consists in determining optimal parameters – number of membership functions, membership function type, inputs number - for neuro-fuzzy subsystem of drilling control system with considering the type of drilled ore material.

The practical significance of the studies is a structure of adaptive control and identification of drilling system with considering the type of drilled ore material including model of geological structure.

The results of the research investigate a strategy of the two-level adaptive control of borehole drilling under rapidly changing conditions which implies simultaneous drilling investigation and control. The subsystem of prediction is implemented on the basis of an adaptive neuro-fuzzy system. The applied neuro-fuzzy system realizes the Sugeno fuzzy inference in the form of a five-layer neural network of signal feedforward, the first layer of which contains the terms of input variables (the current signal value and its delayed values). It should be noted that the membership function type did not influence much on the prediction result. While processing and analyzing the current information about the latest characteristics of drilling and while forming the adaptive control it is reasonable to apply neuro-fuzzy structures with Gaussian membership functions.

Key words: drilling automation, neuro-fuzzy model, adaptive control.

Introduction. The creation of control for an object with uncertain parameters is an important problem of the automatic control theory. Non-stationary and uncertain parameters of control objects cause the necessity to create regulators with adaptable parameters ensuring the unchanged accuracy and quality of a system. The creation of the adaptive system with an identifier is aimed at forming a model-identifier of a control object on the basis of fuzzy and incomplete information [1].

Materials and methods
1. Analysis of research and publications

The quality of automatic control over technological processes on various stages of iron ore mining and processing can be improved by using the latest information about the technological process while controlling it [2-6, 12-15]. In this case, the information on the technological process development can be obtained both by its direct measurement and by using a mathematical model [2].

As drilling characteristics are random and non-stationary it is reasonable to apply the methods of adaptive control with an identifier of an object model while synthesizing this process control. The research is aimed at investigating the methods of forming a model for the system of adaptive control over drilling with the control object identifier [7]. In general, when forming the adaptive control of drilling rocks one should consider the fact that the control object is under the influence of the following input impacts: driving $X'(t)$, controlling $U(t)$ and disturbing $Z(t)$. The object's behaviour characterized by the output variables $Y(t)$ depends on a set of unknown parameters $\xi$ with the given set of admissible values $\Xi$ among which one should distinguish physical and mechanical characteristics of rock types. In this case, it is necessary to form the control that would ensure the given indices of drilling quality under all admissible values of the unknown parameters $\xi$.

The research is aimed at investigating the methods of forming a model for the system of adaptive control over drilling with the control object identifier.

2. Research results

Under rapidly changing conditions of borehole drilling one should use a strategy of the two-level adaptive control which implies simultaneous drilling investigation and control [7, 8].
When applied to drilling prospecting boreholes containing several rock types, one should include an extra block of the model formation into the control system structure (Fig. 1).

Figure 1. Adaptive system of drilling control

The information on the technological process development can be obtained not only by its direct measurement but also on the basis of interpretation of indirect factors [9-10]. In the course of the research, the following parameters are under control: drilling speed, rotation speed and torque. Besides the mentioned parameters, the research work [10] investigates the possibility of using the axis load while identifying the rock geological structure in drilling.

The predicting subsystem is realized on the basis of ANFIS – the adaptive neuro-fuzzy inference system [11]. The applied ANFIS realizes the Sugeno fuzzy inference in the form of a five-layer neural network of signal feedforward the first layer of which contains the terms of input variables (the current signal value and its delayed values). While forming the model the initial data selection is divided into two parts: training and checking.

The result of the adjustment of the membership functions in case of two or three terms of input variables is shown in Fig. 2.

The results of assessing the influence of the membership function number on the efficiency indices of the identification are presented in Table 1. The best results (the shortest time and the smallest standard error of prediction) are obtained in case of two membership functions with RMSE=0.0215 and the execution time of 5.2188 sec.

Figure 2. Membership functions of input variable terms

The result of investigating the influence of the membership function type on the identification efficiency indices is shown in Table 2.

The best results (tab. 2) are obtained when trapezoid membership functions (the minimum learning time) and the Gaussian membership functions (the minimum error) are used. However, it should be noted that the membership function type does not have any significant influence on
the prediction result. Later, the Gaussian membership function ensuring the minimum learning time is applied.

Table 1. Influence of the membership function number on identification efficiency indices

<table>
<thead>
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<th>Membership function number</th>
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Table 2. Influence of membership function type on identification efficiency indices

<table>
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<th>Convention</th>
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<th>Gbellmf</th>
<th>Psigmf</th>
<th>Trimf</th>
</tr>
</thead>
<tbody>
<tr>
<td>Learning time, sec</td>
<td>5.1125</td>
<td>5.2969</td>
<td>5.2344</td>
<td>5.1094</td>
</tr>
<tr>
<td>RMSE</td>
<td>0.0204</td>
<td>0.0215</td>
<td>0.0206</td>
<td>0.0214</td>
</tr>
</tbody>
</table>

The investigation of the influence of the deferred input number on the identification efficiency indices reveals (tab. 3) that the best results are achieved with three or four deferred inputs.

Table 3. Influence of deferred input number on identification efficiency indices

<table>
<thead>
<tr>
<th>Input number</th>
<th>2</th>
<th>3</th>
<th>4</th>
<th>5</th>
<th>6</th>
</tr>
</thead>
<tbody>
<tr>
<td>Learning time, sec</td>
<td>2.5469</td>
<td>3.0938</td>
<td>6.5313</td>
<td>24.1563</td>
<td>169.9531</td>
</tr>
<tr>
<td>RMSE</td>
<td>0.0554</td>
<td>0.0209</td>
<td>0.0214</td>
<td>0.0324</td>
<td>0.0326</td>
</tr>
</tbody>
</table>

The joint graph of the prediction initial data and results as well as the prediction error is shown in Fig. 5.

The fragment of the joint graph of the prediction results and the check data as well as the prediction error is shown in Fig. 6.

The dependency of the standard learning error on the number of the epochs for the training and checking selections are presented in Fig. 4-6.

Training of ANFIS was carried out by the method of back error propagation with the error level of 0 and the cycles number of 25. The volumes of statistical sampling and the
parameters of neuro-structures were identified on the basis of recommendations [7-8]. The N-number of elements of the statistical sampling, which required for training was determined by the next ratio
\[
\frac{2n(n + m)}{(n + 2\varepsilon_0)} \leq \Phi \leq \frac{10n(n + m)}{(n + 10\varepsilon_0)},
\]
where \(n\) – is the number of input signals; \(m\) – is the number of outputs; \(\varepsilon_0\) – is a relative error of neural network model.

The sample representativeness verification was carried out using the dependence proposed in [7], of the maximum error \(\hat{\varepsilon}_w\) on the volume of statistical sampling
\[
\hat{\varepsilon} = \frac{\Phi((p + 1)/2)}{2\sqrt{N}},
\]
where \(P\) – is the reliability level; \(\Phi()\) – is the Laplace function; \(N\) – is the elements number of statistical sampling. With a value of reliability level of \(P=0.9\) (90%) the appropriate level of significance is \((1-P)=0.1\).

Thus, the prediction error of the neuro-fuzzy model with four deferred inputs and two terms of input variables and ten training epochs is within 5-7%.

**Conclusion.** While processing and analyzing the current information about the latest drilling characteristics and forming the adaptive control it is reasonable to apply neuro-fuzzy structures with two Gaussian functions of term membership for each variable and 3-4 deferred inputs.

**References**

