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INVESTIGATION OF MECHANOCHEMICAL LEACHING OF NON-FERROUS METALS

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Abstract: The research deals with metal extraction from off-grade ores and concentration tailings. There are provided results of simulating parameters of reagent leaching of metals in the disintegrator according to the metal recovery ratio. The research substantiates the method of waste-free processing of chemically recovered ores. Recovery of metals into solution is the same both under multiple leaching of tailings or ore in the disintegrator and agitation leaching of tailings or ore previously activated in the disintegrator with leaching solutions. The time of agitation leaching is more by two orders of magnitude than that of the disintegrator processing. Recovery of metals into solution is most affected by the content of sodium chloride in the solution. Then, in decreasing order, go the content of sulfuric acid in the solution, the disintegrator rotor rpm and L:S ratio.

Key words: Experiment, simulation, activation, concentration tailings, disintegrator, leaching

1. INTRODUCTION

Mineral mining for industrial branches is characterized by large-scale operations, application of powerful equipment, the extent of impacts on the environment and involvement of off-grade materials into production accompanied by increased wastes after processing (Liu et al., 2012; Sinclair and Thompson, 2015; Dmitrak and Kamnev, 2016). The threat of environment pollution can be decreased by non-waste utilization of processing tailings. Known concentration processes do not meet this requirement. They can be updated due to introduction of chemical and hydrometallurgical concentration processes (De Oliveira et al., 2014; Lyashenko et al., 2015; Golik et al., 2015c). In 1950s, a new technology called substance activation by high energy won the recognition. The energy acting with the velocity of 250 m/sec, the substance properties change and it is able to react with other substances (Khint, 1981).

Scientific papers dealing with geotechnological processes in concentration are topical. Yet, leaching by itself does not make extraction non-waste. Therefore, development of technologies combining chemical and mechanical processes is particularly promising (Golik et al., 2015b; Jordens et al., 2013; Gazaleyeva et al., 2017). The process of metal extraction from off-grade mineral materials can be facilitated by changing properties of substances in the disintegrator when metals are extracted, while crystals are destroyed under the action of reagents. The research aims to experimentally substantiate this concept (Morkun and Tron, 2014; Golik et al., 2015a; Golik et al., 2015e; Golik, 2013).

2. LITERATURE ANALYSIS AND PROBLEM STATEMENT

Methods of ore mineral concentration differ in the kinetics of recovery in apparatuses that realize physical principles of disintegration. The ore recovery degree during concentration in different apparatuses changes from 54% to 80%.

The study (Zhang et al., 2019) provides an approach to improve the recycling of zinc and iron and effectively reduce environmental risks from electric arc furnace dust. In this article, a hydrothermal reduction method was proposed to recover Zn selectively from electric arc furnace dust. The total Zn leaching efficiency was calculated to be 89.7%, producing Zn-impregnated leach solution and a leach residue rich in Fe. However, this process also has its shortcomings owning to larger liquid-solid ratio in the leaching process.

In the work by Xie et al. (2019), the feasible method for the extraction of lead from lead-containing zinc leaching residues was proposed. The effects of reaction time, reaction temperature, calcium chloride concentration, liquid-solid ratio, and pH were studied. It was shown that the calcium chloride concentration and the liquid-solid ratio have a significant influence on lead leaching rate and the optimized technological conditions were obtained.

The research by Wang et al. (2018) presented the impacts of mechanochemical activation on the physiochemical properties of lithium cobalt oxide powders of cathode materials from spent lithium-ion batteries, and analysed the relevant effects of these changes on the leaching efficiency of lithium and cobalt and the leaching kinetics of powders. The results revealed the superiority of mechanochemical activation in the following levels of changes in the powders. The physical properties included a decrease in the average particle size, an increase in the specific surface area, and the appearance of a mesoporous structure change.

In the research by Cetintas et al. (2018), the method for nickel recovery from lateritic ore was proposed. The method consisted of mechanochemical conversion in the presence of a reagent, ammonium carbonate or sodium hydroxide, then followed by an acid leaching. The study presents the recovery of nickel from lateritic ore at low temperature and lower acid consumption to overcome the disadvantages of traditional metallurgical processes such as

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high temperature, high acid concentration, high cost and material requirements. The optimization conditions for both mechanochemical conversion and atmospheric leaching processes were evaluated by response surface methodology to clearly demonstrate the advantages of the proposed method over conventional methods.

The research by Fan et al. (2018) gave the results of the selective extraction of Fe and Li from spent batteries via an environmentally friendly mechanochemical process with oxalic acid. With the use of a mechanochemical treatment and water leaching, the Li extraction efficiency can be improved to 99%. To investigate the reaction mechanism and determine the optimum reaction conditions, various parameters, including rotation speed, milling time, and ball-to-powder mass ratio were investigated.

In the work of Minagawa et al. (2018), the influence of planetary ball milling and vertical stirred ball milling on the leaching of a copper ore containing copper sulphate and covellite was investigated. A mixed experimental-simulation approach to correlate the kinetic parameters of leaching to the collision energy during grinding was used. The leaching of the ore occurred through three subsequent steps. It was confirmed that the rate constants for the leaching of covellite increased due to an occurred mechanochemical reaction.

In the research by Yang et al., 2017, mechanochemical activation was developed to selectively recycling Fe and Li from cathode scrap of spent batteries. By mechanochemical activation pre-treatment and the diluted leaching solution, the leaching efficiency of Fe and Li can be significantly improved, respectively. Through process and the mechanochemical activation mechanisms, the effects of various parameters during Fe and Li recovery were comprehensively investigated, including activation time, cathode powder to additive mass ratio, acid concentration, the liquid-to-solid ratio, and leaching time.

The work by Granata et al. (2018) addressed the mechanochemical activation of chalcopyrite by vertically stirred ball milling to enhance copper dissolution in leaching. The contribution of each activation mechanism to leaching enhancement was assessed by two-stage leaching. The increase of copper extraction in the first stage leaching confirmed the increase of soluble copper due to mechanochemical oxidation. Second stage leaching highlighted a larger dissolution of copper from chalcopyrite and enhanced kinetics upon activation.

The general drawback of the known technologies of processing tailings utilization is incomplete metal extraction resulting in accumulated secondary tailings in the vicinity of mining enterprises (Golik et al., 2015d; Ghorbani et al., 2016; Wang et al., 2015).

3. RESEARCH AIM AND TASKS

This research is aimed at investigating a mechanism and parameters of leaching off-grade metallic ores by experimental substantiation of metal recovery by leaching in the disintegrator and simulation of the process parameters providing the evidence of non-waste processing of chemically recovered ores.

To achieve this aim, indices of ore concentration technologies should be systematized, the theory and practice of recovery activation should be analysed and experimental leaching should be performed.

4. MATERIALS AND RESEARCH METHODS

To balance metal losses in ore concentration, magnetic, gravity and electrochemical separation methods are used, making it possible to obtain marketable products from concentration tailings.

The technology developed by the Institute of Ore Deposit Geology, Petrography, Mineralogy and Geochemistry of the Russian Academy of Sciences is intermediate between pyro- and hydrometallurgy as ore is broken down at 550°C at the normal air pressure with subsequent leaching. Specific low-temperature (100–550°C) interaction of processed ores and water chlorides is used. In this thermal interval under the normal air pressure, some minerals are destroyed and oxidized. The process is accompanied by transformation of metals into the forms dissolved in subsequent acid leaching.

The disintegrator consists of rotors and a basket (Fig. 1, 2).

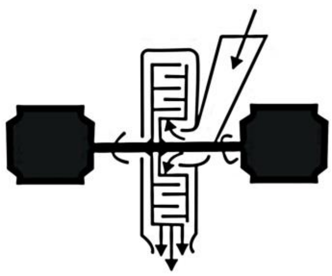

Fig. 1. The fundamental scheme of the disintegrator

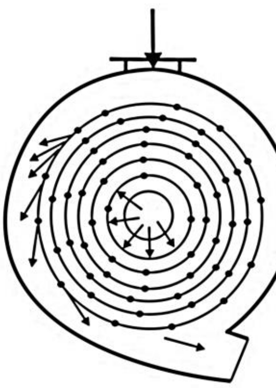

Fig. 2. Direction of forces in the working body of the disintegrator

The dashers are located between rotor disks so that each row of dashers of one rotor is located between two rows of dashers of the other. Materials are fed to the central part of the rotor and on the way to the periphery, they are beaten with the rotation velocity of over 500 rpm.

In mechanical ore processing, energy is accumulated and part of this energy is accumulated on the newly created surfaces and spent on chemical processes. In processing polycrystalline raw materials, they are destroyed on the boundary of phase separation as well. Processes of phase separation from ores in the disintegrator are significantly simplified and the end product yield increases. For the first time, the industrial disintegrator was used for 5 years at Shokpak deposit (Kazakhstan).

The industrial plant ДУ-65 was equipped with 200 and 250 kW engines, self-bushing rotors with a protective layer in the stowing complex net (Fig. 3).

The disintegrator produced up to 55% of the active class and

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combined with the vibration mill its yield increased up to 70%. The disintegrator was located in a three-level building with the foundation of 5×7 m. The disintegrator technology provided activity increase up to 40%. In disintegration products, the fraction of particles larger than 125–400 mkm is comparatively small and the fraction is less than 5 mkm. The values of specific slag surfaces of single and secondary grinding in the disintegrator differ by a factor of 1.4, while the content of fractions of less than 63 mm is much higher in case of secondary grinding.

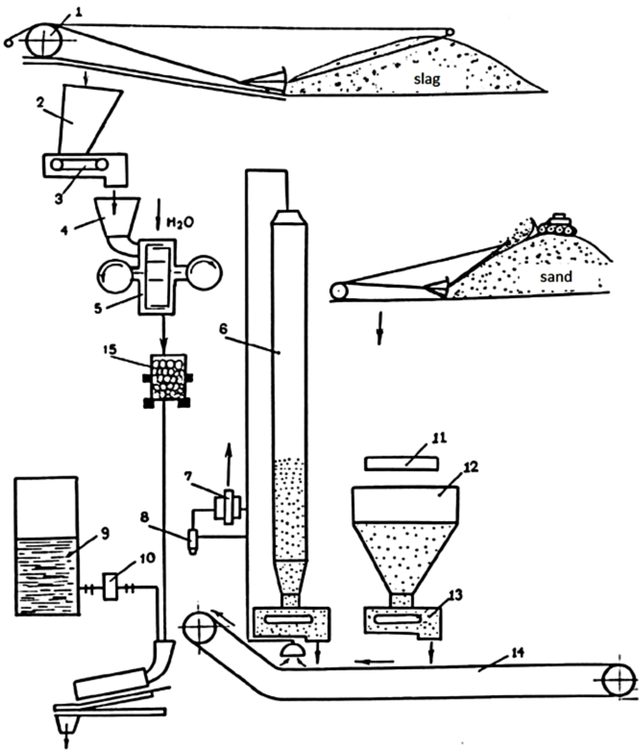

Fig. 3. The disintegrator in the net of the stowing complex: 1 – a scraper hoist; 2 – a storage bin; 3 – a metering unit; 4 – a feeder; 5 – a disintegrator; 6 – a silo tower; 7 – a fan; 8 – a water block; 10 – a pump; 11 – a screen; 12 – a sand bin; 13 – a metering unit; 14 – a conveyor; 15 – a vibration mill

Fig. 4. The assembled view of the disintegrator DES-11

It is explained by the fact that on the way from the basket centre to their periphery, ore particles are repeatedly accelerated and decelerated for milliseconds.

Metals were recovered experimentally in the disintegrator DES-11 (Fig. 4). The maximum capacity of the disintegrator in grinding materials of 2...3 g/sm^3, with the rotor rotation of 12,000 rpm is 1–10 kg/h, the maximum initial size of particles of the processes material is 2.5 mm, the maximum humidity of the ground material is 2%.

The research stages included: investigating concentration tailings; leaching tailings in percolators; leaching tailings in the disintegrator. 0.05 t of floatation tailings of Misur plant was processed. Tailings were screened (the cell size of 2.0 mm) and fed to the basket with the reagent solution. The efficiency of metal recovery was investigated through comparing leaching variants (***Błąd! Nie można odnaleźć źródła odwołania.***).

Tab. 1. Results of the disintegrator testing

Indices	Testing stage			
	0	**1**	**2**	**3**
1. Tailings processed, t	0.04	0.09	0.03	0.04
2. Reduced mass of rotors due to wear, kg	7	0	0.1	0.5
3. Net metal consumption for wear, kg/t	0.01	0.01	0.01	0.01
4. Time for tailings processing, min	25	5	05	45
5. Plant capacity, t/h	0.01	0.04	0.03	0
6. No-load capacity of the external rotor, kW	0.4	0.4	0.4	0.4
7. No-load capacity of the internal rotor, kW	0.2	0.2	0.2	0.2
8. Processing capacity of the external rotor, kW	0.05	0.05	0.50	0.50
9. Processing capacity of the internal rotor, kW	0.7	0.7	0.9	0.8
10. Gross energy consumption of the external rotor, kW h/t	0.07	0.1	0.1	0.1
11. Gross energy consumption of the internal rotor, kW h/t	0.03	0.02	0.02	0.02
12. Gross energy consumption, kW h/t	0.5	0.3	0.2	0.3
13. Net energy consumption of the external rotor, kW h/t	0.20	0.16	0.16	0.20
14. Net energy consumption of the internal rotor, kW h/t	0.05	0.03	0.03	0.04
15.Net energy consumption, kW h/t	0.75	0.69	0.69	0.74

Ores of Sadon deposits were concentrated in heavy slurries. Tailings make 25–50% of the initial feed rate.

The ratio of components is following: coarse grained granites make 40%; porfirites make 30%, sandstones make 20%; vein materials make 8%; ore minerals make 2%. The content of base metals in tailings is following: pyrite – 1.4%; sphalerite – 0.6%; galena – 0.06%; chalcopyrite – 0.05%.

5. INVESTIGATION RESULTS

There were five stages of experiments: agitation leaching of tailings or ore in the percolator; agitation percolator leaching of tailings or ore previously activated in the disintegrator in the dry condition; leaching of tailings or ore by reagents in the disintegrator chamber; agitation leaching of tailings or ore previously activated in the disintegrator chamber with reagents during a single

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stage; leaching of tailings or ore previously activated in the disintegrator chamber with reagents during several stages.

The applied factors regulated during experiments include:

- the content of sulfuric acid and sodium chloride (changes depending on the level: X_1-1 - 2, 0 - 6 and 1–10 g/l, X_2 -1 - 20, 0–90 and 1–160 g/l);
- the ratio of L:S (liquid and solid phases) (changes according to the level: X_3 - 1- 4, 0 – 7 and 1–10),
- the time period of agitation leaching, X_4, (changes according to the level: X_4 -1 -0.25, 0–0.5, 1–1 hour, except for the experiments of the 3rd and 5th stages);
- the disintegrator rotors rpm, X_5, (changes according to the level: X_5 -1 -50, 0–125, 1–200 Hz, except for the experiments of the 1st stage),
- the number of leaching cycles, X_6, (changes according to the level: X_6 -1 -3, 0–5, 1–7) of tailings or ores in the disintegrator (only for the experiments of the 5th stage).

Leaching parameters include: the 0.08 active class yield, the chemical composition of the ores under study, the metal content in the solution, mg/l, metal extraction into the solution, %. The value of the indices in Sadon ores leaching made: the active class yield of 44.4%, the chemical composition, %: Al_2O_3 – 20.8; Si – 50. 29; S – 4.9; Cl – 0.06; Ag – 0.1; K_2O – 5,4; CaO – 2.5; TiO_2 – 0.79; Mn – 0.26; Fe – 7.5; Cu – 0.2; Zn – 3.9; Pb – 1.6.

To estimate the metal leaching technologies, the parameters of conventional (Table 2–4) and innovative leaching (Table 5–11), experimental data are determined and depicted graphically (Fig. 5–10).

Tab. 2. Results of agitation leaching of ore

#	Reagent content, g/l		L:S ratio	Time of leaching, h	Recovery into solution, ε, %	
	H_2SO_4	NaCl			Zn	Pb
1	2	20	4	0.25	26.63	0.25
2	10	20	4	0.25	38.25	0.50
3	2	160	4	0.25	12.50	8.50
4	10	160	4	0.25	18.25	11.75
5	2	20	10	0.25	28.75	2.50
6	10	20	10	0.25	57.81	0.50
7	2	160	10	0.25	22.19	13.13
8	10	160	10	0.25	25.63	15.00
9	2	20	4	1	23.75	0.25
10	10	20	4	1	38.00	0.50
11	2	160	4	1	13.38	7.50
12	10	160	4	1	21.75	14.75
13	2	20	10	1	56.56	1.88
14	10	20	10	1	57.50	1.25
15	2	160	10	1	23.75	14.38
16	10	160	10	1	20.31	11.88
17	2	90	7	0.625	28.00	12.25
18	10	90	7	0.625	57.19	14.88
19	6	20	7	0.625	60.59	0.88
20	6	160	7	0.625	30.41	25.00
21	6	90	4	0.625	29.25	8.50
22	6	90	10	0.625	30.00	17.50
23	6	90	7	0.25	29.31	13.56
24	6	90	7	1	29.97	14. 88

Tab. 3. Regression analysis of experimental data

Regression equation	Significance indices
$\varepsilon_{Zn}=39.02 +5.51X_1-11.09X_2 +5.6X_3+1.43X_4 +3.58X_1^2+6.48X_2^2-9.39X_3^2 -9.38X_4^2- 2.61X_1X_2 - 0.62X_1X_3 -1.86X_1X_4 -3.0X_2X_3-1.48X_2X_4 +1.41X_3X_4$	$R^2 = 0.8873$; $S_{ad} = 62.01$; F = 31.96
$\varepsilon_{Pb}=15.73 +0.58X_1+6.3X_2+1.42X_3 - 2.17X_1^2 - 2.79X_2^2-2.73X_3^2-1.51X_4^2 ++0.75X_1X_2 -0.89X_1X_3 +0.45X_2X_3-0.23X_3X_4$	$R^2 = 0.9023$; $S_{ad} = 9.37$; F = 25.89

Note: dimensionless variables are determined from the expressions:

$$X_1 = \frac{C_{H_2SO_4}-6}{4};\ X_2 = \frac{C_{NaCl}-90}{70};\quad X_3 = \frac{(L:S)-7}{3};\ X_4 = \frac{t-0,625}{0,375}; \tag{1}$$

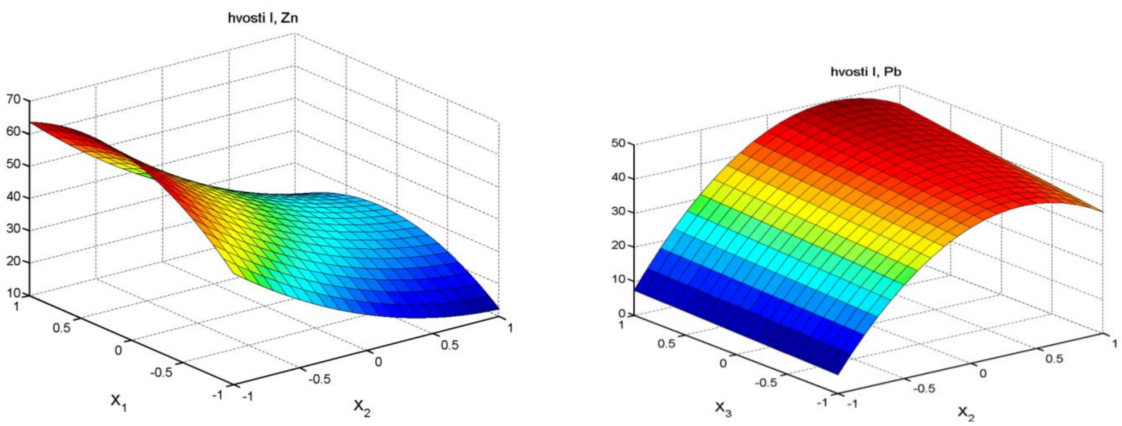

Fig. 5. Results of agitation leaching of ore: a) zinc; b) lead

Tab. 4. Results of agitation leaching of tailings

#	Reagent content, g/l		L:S ratio	Time of leach-ing, h	Recovery into solution, ε, %	
	H_2SO_4	NaCl			Zn	Pb
1	2	20	4	0.25	41.26	1.43
2	10	20	4	0.25	57.76	0.48
3	2	160	4	0.25	18.11	36.19
4	10	160	4	0.25	24.00	38.10
5	2	20	10	0.25	48.42	3.57
6	10	20	10	0.25	82.11	4.76
7	2	160	10	0.25	12.63	30.95
8	10	160	10	0.25	17.89	35.71
9	2	20	4	1	44.58	17.14
10	10	20	4	1	70.26	1.43
11	2	160	4	1	10.95	24.76
12	10	160	4	1	28.21	37.14
13	2	20	10	1	49.47	3.57
14	10	20	10	1	50.53	1.79
15	2	160	10	1	15.79	46.43
16	10	160	10	1	18.95	44.05
17	2	90	7	0.625	21.37	35.83
18	10	90	7	0.625	34.63	41.67
19	6	20	7	0.625	67.79	1.67
20	6	160	7	0.625	25.79	45.00
21	6	90	4	0.625	40.84	21.29
22	6	90	10	0.625	36.84	58.33
23	6	90	7	0.25	40.53	49.17
24	6	90	7	1	42.74	40.83

Tab. 5. Regression analysis of experimental data

Regression equation	Significance indices
ε_{Zn}=39.35 +6.76X_1-18.88X_2 -0.62X_4 – 11.6X_1^2+7.19X_2^2+2.03X_4^2 – -2.84X_1X_2 -1.39X_1X_3 -0.89X_1X_4 -2.04X_2X_3+1.00X_2X_4 -2.45X_3X_4	R^2 = 0.9393; S_{ad} = 46.93; F = 68.59
ε_{Pb}=42.43 +16.8X_2+2.68X_3 +0.93X_4 – 3.89X_1^2 - 19.31X_2^2+2.36X_4^2 + +2.12X_1X_2 -0.9X_1X_4 +1.73X_2X_3+1.04X_3X_4	R^2 = 0.8888; S_{ad} = 71.17; F = 30.19

Note: dimensionless variables are determined from the expressions:

$$X_1 = \frac{C_{H_2SO_4}-6}{4};\ X_2 = \frac{C_{NaCl}-90}{70};\ X_3 = \frac{(L:S)-7}{3};\ X_4 = \frac{t-0{,}625}{0{,}375}; \qquad (2)$$

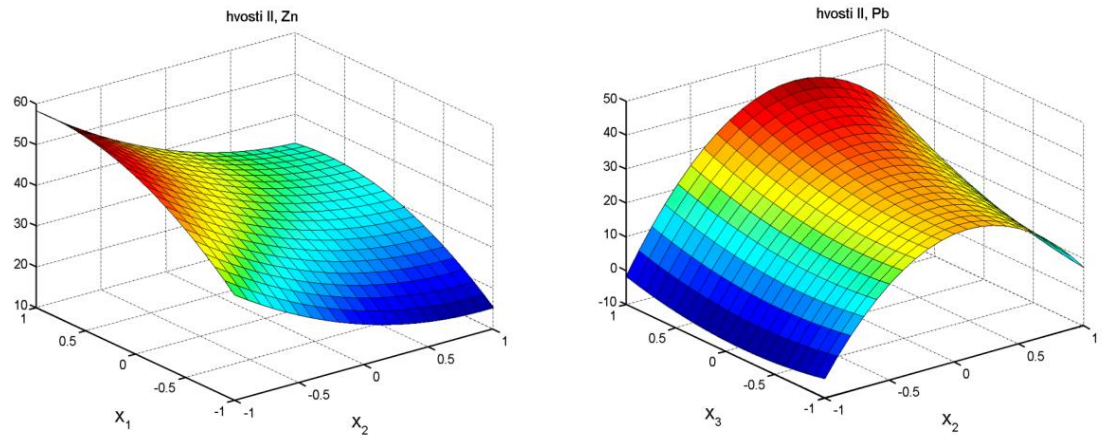

Fig. 6. Results of agitation leaching of ore: a) zinc; b) lead

Tab. 7. Results of agitation leaching of activated ore

#	Reagent content, g/l		Time of leach-ing, h	Rotor rpm, Hz	Recovery into solution, ε, %	
	H_2SO_4	NaCl			Zn	Pb
1	2	20	0.25	50	14.63	0.25
2	10	20	0.25	50	11.63	7.75
3	2	160	0.25	50	16.50	11.25
4	10	160	0.25	50	12.13	11.13
5	2	20	0.25	50	35.00	1.19
6	10	20	0.25	50	35.31	1.44
7	2	160	0.25	50	12.19	11.25
8	10	160	0.25	50	13.44	11.88
9	2	20	1	200	21.50	0.35
10	10	20	1	200	36.63	0.50
11	2	160	1	200	7.38	5.00
12	10	160	1	200	18.63	13.50
13	2	20	1	200	30.63	1.88
14	10	20	1	200	39.38	1.44
15	2	160	1	200	15.63	11.88
16	10	160	1	200	21.56	12.5
17	2	90	0.625	125	12.25	6.56
18	10	90	0.625	125	27.34	16.63
19	6	20	0.625	125	41.34	1.31
20	6	160	0.625	125	18.81	11.88
21	6	90	0.25	125	15.63	7.5

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#	Reagent content, g/l		Time of leaching, h	Rotor rpm, Hz	Recovery into solution, ε, %	
	H_2SO_4	NaCl			Zn	Pb
22	6	90	1	125	38.13	26.25
23	6	90	0.625	50	24.28	14.00
24	6	90	0.625	200	25.81	20.56

Tab. 6. Regression analysis of experimental data

Regression equation	Significance indices
$\varepsilon_{Zn}=26.82+2.8X_1-7.21X_2+4.81X_3+2.34X_4-6.99X_1^2+3.29X_2^2-1.74X_4^2-0.45X_1X_2+2.93X_1X_4-2.98X_2X_3-1.41X_2X_4-1.12X_3X_4$	$R^2=0.8582$; $S_{ad}=31.92$; F = 35.03
$\varepsilon_{Pb}=11.16+1.51X_1+4.68X_2+0.53X_3-0.81X_4+0.50X_1^2-4.50X_2^2-0.72X_3^2-0.94X_1X_3+0.60X_2X_3+0.24X_2X_4+0.81X_3X_4$	$R^2=0.8754$; $S_{ad}=6.87$; F = 23.92

Note: dimensionless variables are determined from the expressions:

$$X_1=\frac{C_{H_2SO_4}-6}{4};\ X_2=\frac{C_{NaCl}-90}{70};\ X_3=\frac{t-0{,}625}{0{,}375};\ X_4=\frac{f-125}{75}; \tag{3}$$

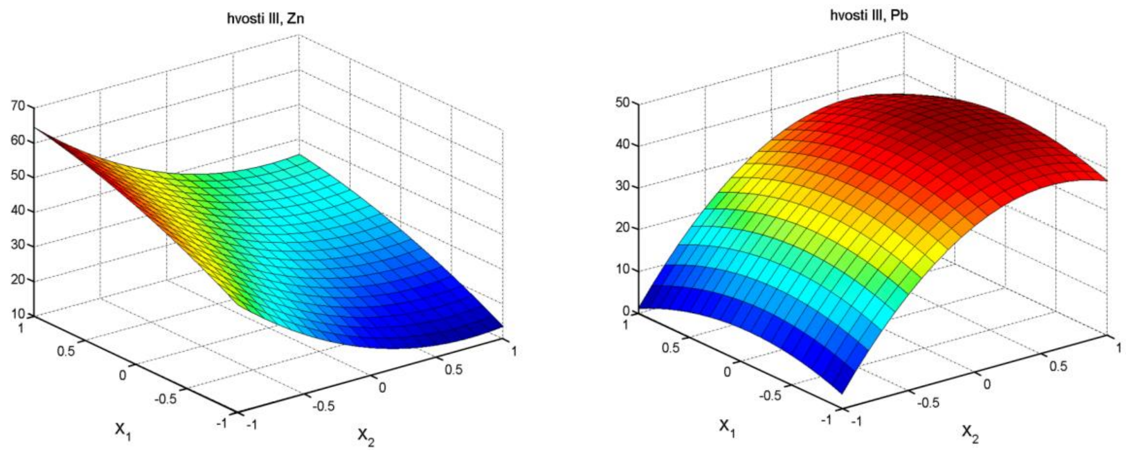

Fig. 7. Results of agitation leaching of ore: a) zinc; b) lead

Tab. 8. Results of agitation leaching of activated tailings

#	Reagent content, g/l		Time of leaching, h	Rotor rpm, Hz	Recovery into solution, ε, %	
	H_2SO_4	NaCl			Zn	Pb
1	2	20	0.25	50	19.37	0.81
2	10	20	0.25	50	56.00	0.95
3	2	160	0.25	50	6.74	17.62
4	10	160	0.25	50	28.21	2.38
5	2	20	0.25	50	50.53	3.21
6	10	20	0.25	50	61.05	2.62
7	2	160	0.25	50	4.63	35.71
8	10	160	0.25	50	13.68	38.10
9	2	20	1	200	28.63	1.43
10	10	20	1	200	66.11	0.86
11	2	160	1	200	1.01	3.33
12	10	160	1	200	27.79	37.14
13	2	20	1	200	50.53	2.98
14	10	20	1	200	51.58	6.67
15	2	160	1	200	6.74	36.90
16	10	160	1	200	15.79	38.10
17	2	90	0.625	125	16.21	30.00
18	10	90	0.625	125	43.47	25.00
19	6	20	0.625	125	60.42	2.83
20	6	160	0.625	125	23.58	4.42
21	6	90	0.25	125	36.11	28.33
22	6	90	1	125	34.63	39.17
23	6	90	0.625	50	36.84	33.33
24	6	90	0.625	200	28.00	45.00

Tab. 9. Results of agitation leaching of activated tailings

Regression equation	Significance indices
$\varepsilon_{Zn}=36.37+9.96X_1-11.56X_2+1.07X_3-6.53X_1^2+5.63X_2^2-1.00X_3^2-3.95X_4^2-1.21X_1X_2-5.79X_1X_3-4.16X_2X_3-0.74X_2X_4-1.15X_3X_4$	$R^2=0.9688$; $S_{ad}=24.88$; F = 102.17
$\varepsilon_{Pb}=29.91+1.1X_1+10.63X_2+6.15X_3+2.09X_4-2.41X_1^2-26.29X_2^2+3.84X_3^2+9.25X_4^2+1.21X_1X_2-0.72X_1X_3+3.21X_1X_4+4.81X_2X_3+1.08X_2X_4-1.00X_3X_4$	$R^2=0.8789$; $S_{ad}=86.00$; F = 10.52

Note: dimensionless variables are determined from the expressions:

$$X_1=\frac{C_{H_2SO_4}-6}{4};\ X_2=\frac{C_{NaCl}-90}{70};\ X_3=\frac{t-0{,}625}{0{,}375};\ X_4=\frac{f-125}{75}; \tag{4}$$

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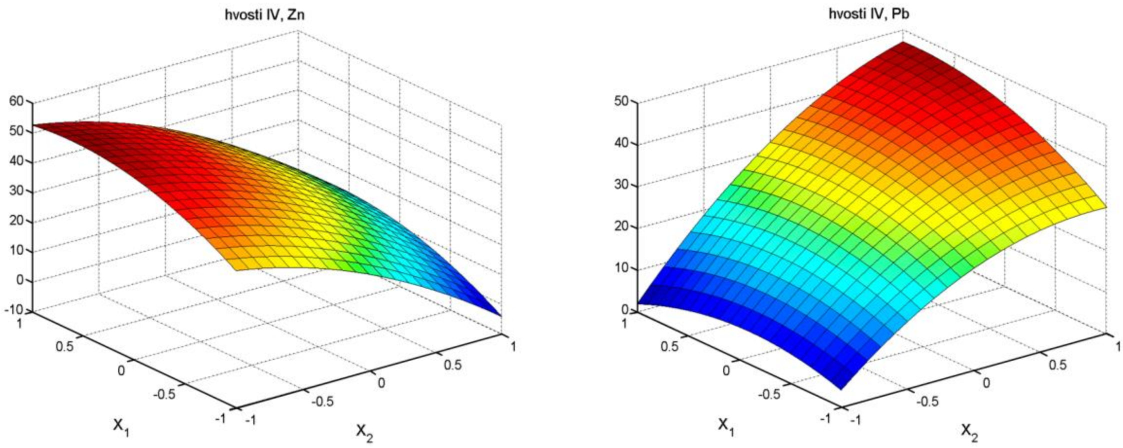

Fig. 8. Results of agitation leaching of activated tailings: a) zinc; b) lead

Tab. 10. Results of leaching ore in the disintegrator

#	Reagent content, g/l		L:S ratio	Rotor rpm, Hz	Recovery into solution, ε, %	
	H_2SO_4	NaCl			Zn	Pb
1	2	20	4	50	10.88	0.63
2	10	20	4	50	41.63	0.50
3	2	160	4	50	13.75	7.25
4	10	160	4	50	20.25	10.00
5	2	20	10	50	36.87	2.50
6	10	20	10	50	38.75	1.06
7	2	160	10	50	15.31	1.00
8	10	160	10	50	17.50	1.00
9	2	20	4	200	24.88	0.33
10	10	20	4	200	43.50	0.35
11	2	160	4	200	14.75	7.50
12	10	160	4	200	22.88	11.75
13	2	20	10	200	37.19	2.50
14	10	20	10	200	45.31	1.38
15	2	160	10	200	14.06	11.25
16	10	160	10	200	21.56	11.88
17	2	90	7	125	19.03	9.63
18	10	90	7	125	23.84	10.94
19	6	20	7	125	35.88	1.69
20	6	160	7	125	17.06	10.94
21	6	90	4	125	10.63	8.00
22	6	90	10	125	22.50	12.50
23	6	90	7	50	21.66	10.50
24	6	90	7	200	23.19	11.81

Tab. 11. Regression analysis of experimental data

Regression equation	Significance indices
$\varepsilon_{Zn}=20.40+4.92X_1-8.82X_2+2.55X_3+1.71X_4+1.03X_1^2+6.57X_2^2-3.84X_4^2+2.02X_2^4-2.19X_1X_2-2.77X_1X_3-2.53X_2X_3-1.02X_2X_4-0.61X_3X_4$	$R^2 = 0.9348$; $S_{ad} = 17.71$; $F = 69.49$
$\varepsilon_{Pb}=11.10+0.35X_1+3.46X_2+1.35X_4-0.79X_1^2-5.061X_2^2-0.82X_3^2+0.64X_1X_2-0.55X_1X_3+0.16X_1X_4-1.06X_2X_3+1.45X_2X_4+1.24X_3X_4$	$R^2 = 0{,}9254$; $S_{ad} = 3.69$; $F = 29.06$

Note: dimensionless variables are determined from the expressions:

$$X_1=\frac{C_{H_2SO_4}-6}{4};\ X_2=\frac{C_{NaCl}-90}{70};\ X_3=\frac{(L:S)-7}{3};\ X_4=\frac{f-125}{75}; \quad (5)$$

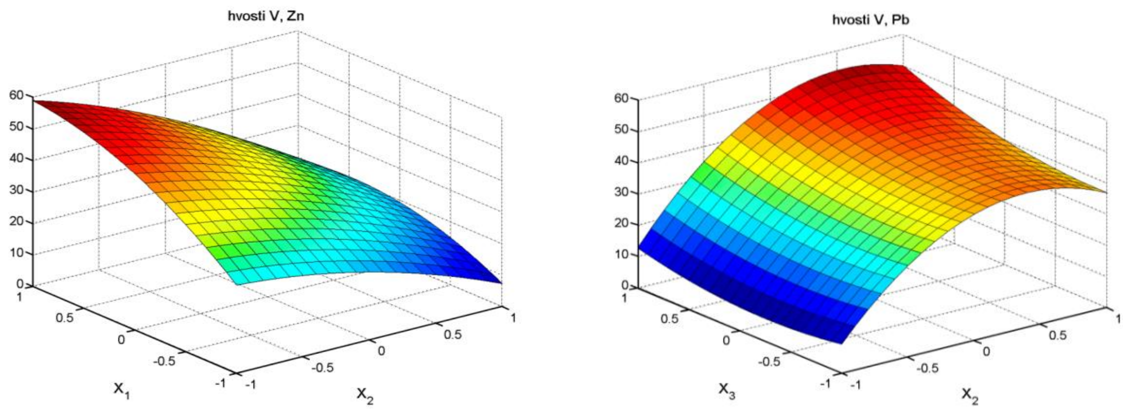

Fig. 9. Results of leaching ore in the disintegrator: a) zinc; b) lead

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On the next stage, parameters of reagent leaching of tailings in the disintegrator are investigated (Table 12 and 13, Fig. 10).

Tab. 12. Results of leaching tailings in the disintegrator

#	Reagent content, g/l		L:S ratio	Rotor rpm, Hz	Recovery into solution, ε, %	
	H_2SO_4	NaCl			Zn	Pb
1	2	20	4	50	26.95	0.33
2	10	20	4	50	78.74	0.95
3	2	160	4	50	10.95	27.14
4	10	160	4	50	27.37	40.50
5	2	20	10	50	47.37	4.76
6	10	20	10	50	54.74	1.79
7	2	160	10	50	6.32	40.48
8	10	160	10	50	15.79	35.71
9	2	20	4	200	32.42	0.71
10	10	20	4	200	61.47	1.43
11	2	160	4	200	13.47	27.14
12	10	160	4	200	27.37	40.00
13	2	20	10	200	42.11	5.95
14	10	20	10	200	52.63	1.55
15	2	160	10	200	12.63	44.05
16	10	160	10	200	65.26	18.38
17	2	90	7	125	22.11	39.17
18	10	90	7	125	36.11	28.33
19	6	20	7	125	58.95	1.67
20	6	160	7	125	23.58	47.50
21	6	90	4	125	35.37	34.29
22	6	90	10	125	29.47	42.86
23	6	90	7	50	32.42	38.33
24	6	90	7	200	26.53	40.83

Tab. 13. Regression analysis of experimental data

Regression equation	Significance indices
$\varepsilon_{Zn}=32.15+11.4X_1-14.04X_2+0.68X_3+1.85X_4-2.90X_1^2+9.25X_2^2-2.53X_4^2-0.39X_1X_2-1.95X_1X_3+1.32X_1X_4+1.47X_2X_3+4.84X_2X_4+3.61X_3X_4$	$R^2 = 0.8277$; $S_{ad} = 143.62$; $F = 18.06$
$\varepsilon_{Pb}=39.44-1.17X_1+16.76X_2+1.28X_3-0.55X_4-5.64X_1^2-14.81X_2^2-0.86X_3^2-4.09X_1X_3-1.42X_1X_4-0.42X_2X_3-1.00X_2X_4-0.82X_3X_4$	$R^2 = 0.9483$; $S_{ad} = 35.09$; $F = 44.58$

Note: dimensionless variables are determined from the expressions:

$$X_1=\frac{C_{H_2SO_4}-6}{4};\ X_2=\frac{C_{NaCl}-90}{70};\ X_3=\frac{(L:S)-7}{3};\ X_4=\frac{f-125}{75}; \quad (6)$$

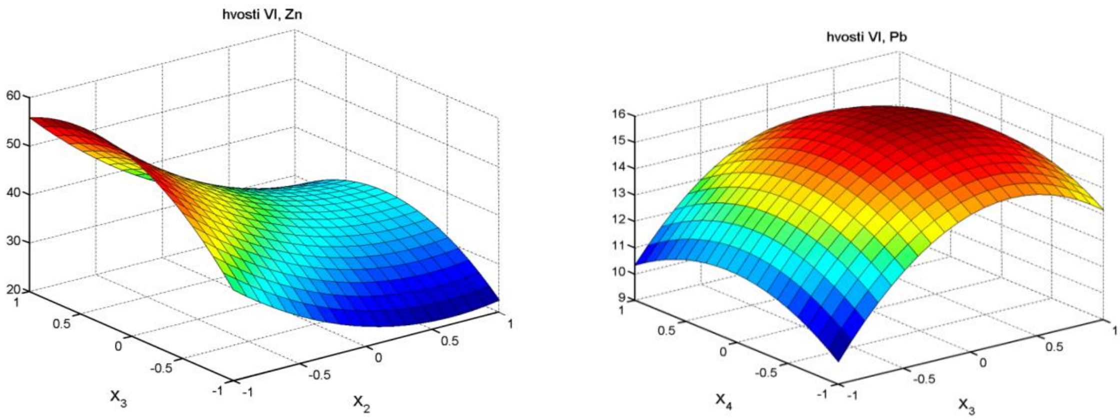

Fig. 10. Results of leaching tailings in the disintegrator: a) zinc; b) lead

Recovery of metals into solution in the disintegrator is equal during the time period by two orders of magnitude less than under agitation leaching.

6. DISCUSSION OF MATERIALS

To interpret the results of reaching, the method of linear multiple regression analysis is used. One parameter is variable, the rest are averaged in the intervals for Sadon ores: the content of H_2SO_4, g/dm^3, 6; the content of NaCl, g/ dm^3, 90; L:S ratio = 7; the leaching time T, h, 0.625; the rotor rpm, Hz = 125.

Recovery of metals into solution depending on the sulfuric acid concentration is interpreted by the graph in Fig. 11.

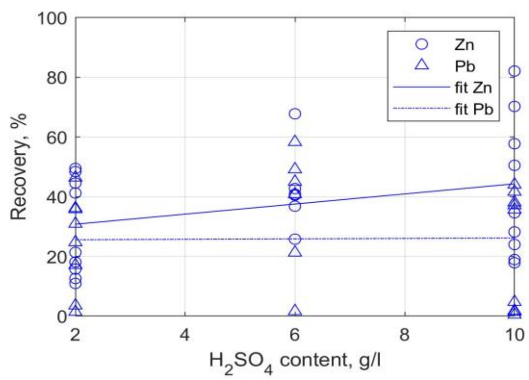

Fig. 11. Parameters of leaching metals by sulfuric acid

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You can see a sharp difference in leaching zinc (25–28%) and lead (25–80%) under equal concentration of the acid.

The parameters of recovery of metals by sodium chloride are interpreted by the graph in Fig. 12.

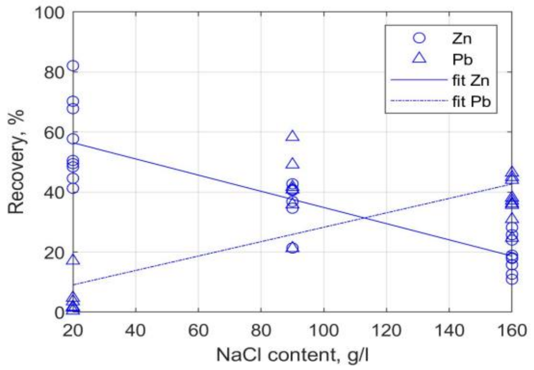

Fig. 12. Parameters of recovery of metals by sodium chloride

Recovery of lead and zinc into solution by sodium chloride is characterized by asymmetry of graphs with intersection in the range of values of 120 g/dm³ .

The parameters of recovery of metals depending on L:S ratio are interpreted by the graph in Fig. 13.

The L:S ratio influences metal recovery into solution in a dramatically different way. While this index remains almost the same for zinc, it increases greatly for lead.

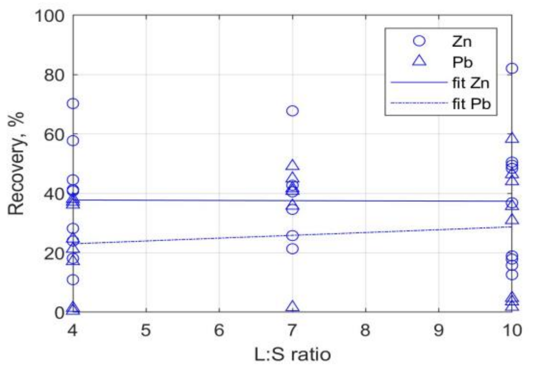

Fig. 13. Parameters of recovery of metals depending on the L:S ratio

The parameters of recovery of metals depending on time of leaching are interpreted by the graph in Fig. 14.

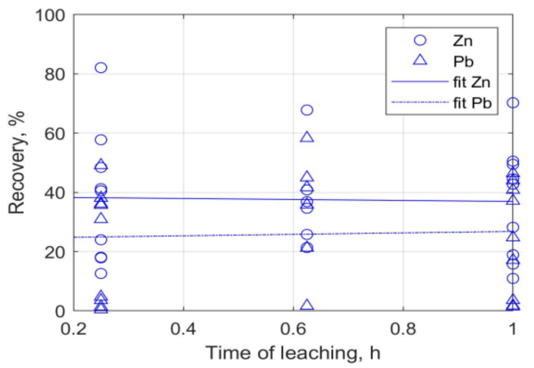

Fig. 14. Parameters of recovery of metals depending on time

While in the case of lead recovery, intensity increases with time, it reduces significantly for zinc.

The parameters of recovery of metals by sulfuric acid in the disintegrator are interpreted by the graph in Fig. 15.

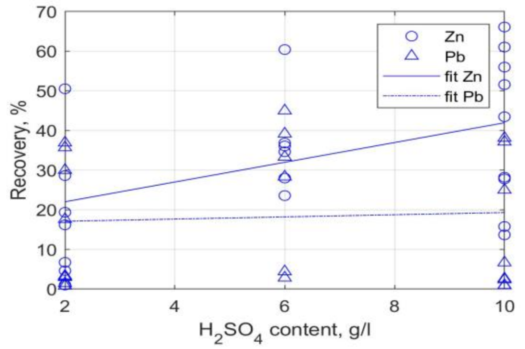

Fig. 15. Parameters of recovery of metals by sulfuric acid in the disintegrator

Orientation graphs of recovery coincide if the activity of zinc is much greater. The parameters of recovery into solution by sodium chloride in the disintegrator are interpreted by the graph in Fig. 16. Recovery of lead and zinc into solution by sodium chloride is characterized by asymmetry of graphs with intersection in the range of values of 140 g/dm³.

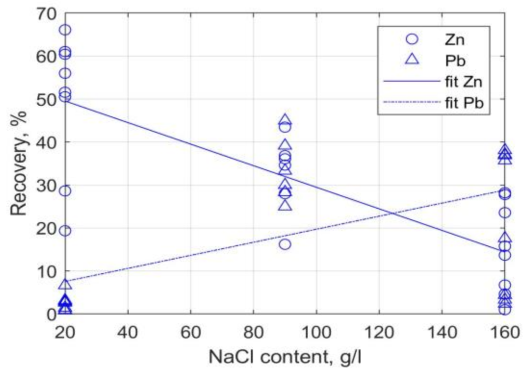

Fig. 16. Parameters of recovery of metals by sodium chloride in the disintegrator

The parameters of recovery of metals depending on the L:S ratio are interpreted by the graph in Fig. 17.

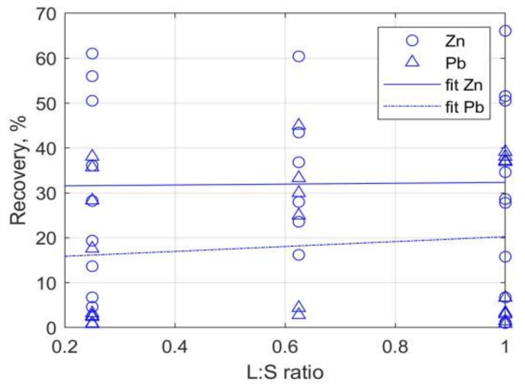

Fig. 17. Parameters of recovery of metals depending on the L:S ratio

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Orientation of the parameters of recovery of metals depending on the L:S ratio is different with lead being more active (Fig. 18).

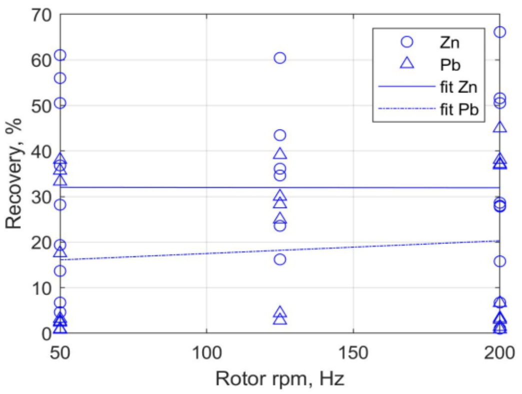

Fig. 18. Parameters of recovery of metals depending on the rotation velocity

The parameters of recovery of metals depending on the rotation are of the same orientation, while activity increases with the same pace.

Activation of materials in the disintegrator with subsequent leaching outside it increases metal recovery in comparison with conventional leaching:

- zinc leaching from concentration tailings by the factor of 1.36 and lead leaching by the factor of 1.13;
- zinc leaching from off-balanced ore by the factor of 1.63, and zinc leaching by the factor of 2.09.

The time period of leaching with simultaneous disintegration makes the first seconds. At the same time, the duration of agitation leaching changes from 15 to 60 minutes, that is, by two orders of magnitude higher.

The obtained results correspond to papers on mining modernization as for the environment and resource saving issues (Jarvie-Eggart, 2015; Sekisov et al., 2016; Komashchenko et al., 2016; Morozov and Yakovlev, 2016).

7. CONCLUSIONS

Recovery of metals into solution is close to the maximum and coincides:

- under agitation leaching of tailings and ore (Series 1);
- under agitation leaching of tailings or ore previously activated in the disintegrator with the leaching solution (Series 4).

Under agitation leaching of tailings or ore previously activated in the dry condition (Series 2) or under leaching of tailings or ore previously passed through the disintegrator once (Series 3), much fewer metals recover into solution.

Recovery of metals into solution is the same both under multiple leaching of tailings or ore in the disintegrator and agitation leaching of tailings or ore previously activated in the disintegrator with leaching solutions. The time of agitation leaching is more by two orders of magnitude than that of the disintegrator processing.

Recovery of metals into solution is most affected by the content of sodium chloride in the solution. Then, in decreasing order go the content of sulfuric acid in the solution, the disintegrator rotor rpm and L:S ratio. Zinc is easier to leach than lead.

The obtained regression equations adequately describe the experimental data with the significance level of 0.05 as the calculating values of the Fischer coefficient exceed their table values for each regression equation.

The content of lead and zinc in the solution almost coincides: under agitation leaching of tailings and ore (Series 1); under agitation leaching of tailings or ore previously activated in the disintegrator with solutions (Series 4). Under agitation leaching of tailings or ore previously activated in the dry condition (Series 2) or under leaching of tailings or ore in the disintegrator (Series 3), recovery of metals into solution is much less than in Series 5 and that confirms the expedience of changing the disintegrator design in order to increase the time period of activation in the disintegrator.

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