Investigation of the possibility of obtaining concentrate production targets based on a mathematical model of an ferrum ore processing site

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This paper presents the methodological foundations for the compilation of a mathematical model of the circuits of the apparatus at the concentrating plant. With its help, it is possible to investigate the change in qualitative and quantitative indicators depending on the loading of the mill at the initial stage of grinding and the properties of the ore. The well-known approaches to the creation of enrichment site models based on the use of regression analysis, multilayer artificial and deep neural networks are considered. The works on the study of enrichment schemes using laboratory equipment are also presented, their disadvantages are shown. Lebedinsky mining and concentrating plant with its enrichment site No. 1 (one half-section) was selected as the object of modeling. An automatic control system (ACS) for loading a wet self-grinding mill (MWS) is considered, which includes a proportional integral (PI) regulator for the flow of ore fed from the conveyor. Mathematical models of aggregates of the enrichment site and their parameters for three grades of ores are presented. In the Matlab envferrumment, a study was conducted on its computer model, which consisted in sequentially changing the power assignment of the MWS to the PI controller, which corresponded to a load of 35 % to 50 %. An array of steady-state values of transients of the main indicators of the work of the enrichment site was obtained, on the basis of which static characteristics were constructed reflecting the relationship between filling, capacity and productivity for the initial ore, the finished class, between filling and the total ferrum content in the concentrate, energy intensity. Comparative results are given on the indicators obtained using the model of the enrichment site for various grades of ores while maintaining a constant loading of the MWS mill and choosing its optimal filling. It was shown that maximal productivity is achieved in the second case, for the initial ore (the finished class): the energy intensity is minimal, and the total ferrum content in the concentrate is within the permissible values.

Keywords: ore enrichment, mathematical model, Matlab, Excel, regression analysis, productivity, energy intensity, total ferrum, concentrate, PI-controller.
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Introduction

Active development and putting into practice digital technologies takes place in the mining and metallurgical industry during last years. High-quality preparation of raw materials for steel making plays especially important role in mining operations. Creation of computer models, which are able to realize forecast of qualitative and quantitative parameters of enrichment products, to assess possibility of concentrate production with preset content of a useful component at maximal volume of processing, to operate in the “adviser” mode with an operator technologist in the field of varying the load on the section (in order to provide planned parameters of the technological process), is rather actual.

The work [1] can be emphasized among recent investigations in the field of models development for concentrating production. It considers approach to creation of the model of enrichment section at Stoilensky mining and concentrating plant (Stoylensky GOK) on the base of use of regression analysis, multilayer artificial and deep neural networks. Ore and water consumption were used as initial entrance effects on grinding process, while capacity consuming by a drive engine of the mill is considered as the exit. Current of spirals and drain density are controlled in the classifier. Other aggregates, such as sump, hydrocyclone and deslimer, are presented by the models, where pulp density is both entrance and exit parameters. Pulp level is additionally controlled in a sump. The model of magnetic separation is presented by relationship between Fe content in concentrate and pulp density of the entrance flow. Absence of feedback in the chain “mill – classifier”, which provide return of coarse class back to additional comminution, is a deficiency of the developed model of enrichment section. The work also does not include clear demonstration of simulation results in the process of imitation of ore feed with different properties, and regarding use of automation systems.

The methods of physical simulation on the base of comminution, classification, separation using laboratorial equipment are also known [2, 3]. Smaller influence of random errors, possibility of varying the factors in more wide range, in comparison with industrial tests, are considered as their advantages [4]. But there are also disadvantages, such as large time of processing, expensive equipment, absence of several aggregates in the laboratorial enrichment scheme; they take place in real conditions. Mathematical simulation is characterized by low cost and quickness of obtaining the results.

Aim of research

The aims of this research are:

1) development of the mathematical model for ferrum ore enrichment section, which:

- differs from the a.m. analogue by more complete description of the components;
- allows to check possibility of achievement of preset quality of products for maximal section productivity by ore and finished class;
- provides possibility to conduct investigations for imitation of variability of raw matrial properties;

2) comparative analysis between parameters obtained with use of the suggested model and parameters obtained at the existing operating conditions of concentrating equipment.

Lebedinsky mining and concentrating plant (Lebedinsky GOK) was chosen as the object for research. Let's consider enrichment section 1, one semi-section [5]. It includes the bin for ore delivery to a feeder, conveyor block for ore feeding to the wet autogenous grinding mill MMC-70x23, the mill trammel for extraction of ore pebble and the spiral classifier 1-KSN-24B which provides return of not completely ground material back to the mill. Drain from the classifier enters to the block of magnetic separators PBM-120/300 of the I stage of magnetic separation of ore material in magnetic fraction (commercial product) and non-magnetic fraction (tailings) by their magnetic susceptibility. The second grinding stage includes the ore-pebble mill MRG-40x75, its drain is also separated on the classifier of the same type. Classifier sands (ore scrap) are returned back to the MRG mill, while drain is forwarded to the II stage of wet magnetic separation (PBM-120/300); it extracts commercial product of the II stage (which unites in sump with commercial product of the I stage) and tailings. Pulp is pumped from sump in the battery of hydrocyclones GTs-500, their sands are returned back to the MRG mill for additional grinding, while drain enters the magnetic hydro-concentrator MGK. MGK drain is forwarded to the separators of the I stage and sands – to the deslimers MD-5A. MD sands enter in sump of the II stage and then they are pumped to the magnetic activator of suspension MAS; its drain is forwarded to sump of the I stage and sands – in magnetic separators PBM-120/300 of the III stage, where concentrate is obtained.

At present time, the grinding process is automatized rather well. Use of up-to-date control sensors of mill filling, which were developed by the institute "Soyuztsvetmetavtomatika" named after V. P. Topchaev [6], and algorithms of optimal control [7, 8] provide increase of productivity and lowering of energy consumption. Local control systems for a magnetic separator, e.g. with PI-controller [9] or with optimal regulator, which is set up via the method of dynamic programming [10], can be used for support of preset quality of concentrate. However, it can be applied only for stabilization of magnetite ferrum, but there are no flow analyzers for control of general ferrum. It can be used only by periodical taking the samples with consequent laboratorial analysis via well-known methods [11]. Thereby, at present time rather actual is the problem of creation of the systems, using relationship between variation of mill productivity and possibility of its prediction of productivity rise and ore load adjusting in such way that predicted value will correspond to the technical specification.

Methods and materials

Mathematical models of the aggregates at enrichment section, their parameters for three ore grades and designations of entrance and exit signals are presented in the **Table 1**. We shall call ore with majority of easily grinding fractions as grade No. 1, ore with majority of hardly grinding fractions as grade No. 3, and ore with approximately equal correlation of these fractions as grade No. 2. Parameters of differential equations, describing mill models, were revealed from the experimentally discovered curve of transition process $\varphi_{m1,2}(t)$. Relationships $I_{m1,2\_1}(\varphi_{m1,2})$, $I_{m1,2\_2}(\varphi_{m1,2})$, $P(\varphi_{m1})$, $g_{-0.071}(I_{m1,2\_1})$, $g_{-0.044m1,2}(I_{m1,2\_1})$ were obtained using regression analysis of the results of industrial experiment [12]. The models of feeder and conveyor were determined on the base of data from the work [13]. Parameters of the rest models were obtained analytically using reference information [14].

The model of enrichment section also includes automatic control system for MMC mill feed; this mill contains PI-controller with control law $\Delta U = k_p(P_p - P) + k_i\int_0^t (P_p - P)dt$ where ΔU – variation of control voltage to the feeder during the time t, B; k_p, k_i – coefficients of proportionality and integration of controller, V/Wt, V/Wt/h; P_p, P – preset and current values of capacity which corresponds to the definite level of mill loading, kWt. The coefficients k_p, k_i were determined using PID-tuner supplement, which is built in Matlab. Mill filling at the operating production site is usually supported on the preset level, which corresponds to the nominal operating conditions. It is not varied owing to the danger of overload or underload. The loading degree for the mill MMC-70x23 makes 43 % [13], though ores with different grinding ability require various and optimal filling.

At the same time, achievement of maximal productivity in initial ore, minimal energy consumption for restriction of general ferrum content in concentrate can be considered as a criterion of optimization. Let's determine optimal filling with use of the developed model of enrichment section via Matlab Simulink program. Experimental researches were conducted in the following way.

The tasks for MMC mill capacity were varied consequently with 10 kWt step in the ranges 996.25 – 1146.25 kWt for the ore grade No. 1, 1056.25 – 1206.25 kWt for the ore grade No. 2, 1116.25 – 1266.25 kWt for the ore grade

Table 1. **Mathematical models of the aggregates of enrichment sections and their parameters for three ore grades**

Modeling aggregate	Mathematical description	Model parameters	Ore grade No. 1	Ore grade No. 2	Ore grade No. 3	Comments
Feeder	$Q_{in.c} = k_1 U,$	k_1, t/h/V	4.995			U – voltage on feeder engine brushes, V; $Q_{in.c}$ – ore consumption to conveyor, t/h
Conveyor	$Q_{in.m1} = Q_{in.c}\ (t - \tau),$	τ	0.0033			τ – conveyor retarding, h; $Q_{in.m1}$ – ore consumption in MMC mill, t/h
Mills MMC, MRG	$T_1 \dfrac{d\varphi_{m1,2}}{dt} + \varphi_{m1,2} = k_2 Q_{in.m1,2},$ $Q_{ex.m1,2} = \left(\dfrac{T_1}{k_2} - k_3\right)\dfrac{d\varphi_{m1,2}}{dt} + \dfrac{1}{k_2}\varphi_{m1,2}$ $I_{m1,2\_1} = A_1(\varphi_{m1,2} - B_1)^2 + C_1,$ $I_{m1,2\_1} = A_2\varphi_{m1,2} + B_2.$ $g_{-0.071} = D_1 I_{m1\_1} + E_1,$ $g_{-0.044m1,2} = D_2 I_{m1,2\_1} + E_2,$ $P = F\varphi_{m1} + G.$		MMC (MRG):			$\varphi_{m1,2}$ – mill filling degree, %; (1 – MMC, 2 – MRG); $Q_{in.m2}$ – pebble consumption in MGR, t/h; $Q_{ex.m1,2}$ – consumption at the mill exit, t/h; $I_{m1,2\_1}$ и $I_{m1,2\_2}$ – parameters of the first and second channels in control devices of active mills capacity , A; $g_{-0.071}$ – output of finished class -0.071 mm on MMC drain, %; $g_{-0.044m1,2}$ – output of finished class -0,044 mm on mills drain, %; P – average capacity, which is consumed by drive engine, kWt
		k_2, %/(t/h)	0.124 (0.43)	0.262 (3.37)	0.4 (6.31)	
		B_1, %	49.65 (43.92)	42.78 (37.77)	35.91 (31.62)	
		C_1, A	1.151 (3.28)	1.91 (4.12)	2.669 (4.96)	
		T_1, h	0.139 (0.383)			
		k_3, t/%	1.174 (2.04)			
		A_1, A/%²	-0.0055 (-0,028)			
		A_2, A/%	-0.0451 (-0.0053)			
		B_2, A	4.6746 (0.13)			
		D_1, A⁻¹	0.2656			
		E_1	0.2212			
		E_2	0.2603 (0.9016)	0.2603 (0.61)	0.2603 (0.3184)	
		G, kWt	646.25	706.25	766.25	
		D_2, A⁻¹	0.093 (0)			
		F, kWt/%	10			
Mill trommel	$Q_{dr.m1} = k_4 Q_{ex.m1}$ $Q_{in.m2} = Q_{ex.m1} - Q_{dr.m1}$	k_4	0.2085	0.135	0.43	$Q_{dr.m1}$ – consumption on MMC mill drain, t/h
Classifier of the I stage	$Q_{-0.071cl} = Q_{dr.m1} \cdot g_{-0.071}.$ $Q_s = Q_{dr.m1}\ (t - \tau) - Q_{-0.071cl}\ (t - \tau).$ $Q_{-0.044cl1} = Q_{dr.m1} \cdot g_{-0.044m1}.$		0.05			Q_s – consumption of sands, t/h; $Q_{-0,071cl}$ и $Q_{-0,044cl1}$ – MMC productivity for classes -0.071 mm and -0.044 mm, t/h; τ – retarding on the classifier, h
Magnetic separators of the I, II stages	$Q_{ex.s1} = k_5(Q_{-0.071cl} + Q_{dr.p}),$ $Q_{ex.s2} = k_5 Q_{-0.044cl2}$	k_5	0.55			$Q_{ex.s1,2}$ – pulp consumption at the exit of separators, t/h; $Q_{dr.p}$ – pulp consumption on MGK drain, t/h; $Q_{-0,044cl2}$ – MRG productivity for class -0.044 mm, t/h
Classifier of the II stage	$Q_{-0.044cl2} = Q_{ex.m2} \cdot g_{-0.044m2}.$ $Q_{o.s} = Q_{ex.m2}\ (t - \tau) - Q_{-0.044cl2}\ (t - \tau).$	D_2	0.0615	0.61	0.9016	$Q_{o.s.}$ – ore scrap consumption, t/h
		τ, h	0.05			
Sumps of the I, II stages	$T_2 \dfrac{dQ_{ex.s1}}{dt} + Q_{ex.s1} = k_6(Q_{ex.s1,2} + Q_{dr.p}),$ $T_2 \dfrac{dQ_{ex.s2}}{dt} + Q_{ex.s2} = k_6 Q_{s.d},$	T_2, h	0.021			$Q_{ex.s1,2}$ – pulp consumption at the exit of sumps, t/h; $Q_{s.d}$ – sand consumption of deslimer, t/h; $Q_{dr.p}$ – pulp consumption on MAC drain, t/h
		k_6	1			
Battery of hydrocyclones	$Q_{s.hc} = Q_{-0.071cl} - Q_{-0.044cl1},$ $Q_{dr.hc} = Q_{-0.044cl1} + Q_{-0.044cl2}$		-	-	-	$Q_{s.hc1,2}$ и $Q_{dr.hc1,2}$ – productivity for sands and drain of hydrocyclones, t/h
MGK	$Q_{s.g} = k_7 Q_{dr.hc},$	k_7	0.9			$Q_{s.g}$ – sands consumption on MGK, t/h
Magnetic deslimer	$T_3 \dfrac{dQ_{s.d}}{dt} + Q_{s.d} = k_8 Q_{s.g},$	T_3, h	0.38			-
		k_8	0.78			
MAC	$Q_{s.a} = k_9 Q_{ex.32},$	k_9	0.9			$Q_{s.a}$ – sands consumption MAC, t/h
Magnetic separators of the III stage	$\rho = aQ^2_{ex.a} + bQ_{ex.a} + c,$ $\beta = d\rho^2 + e\rho - f.$	a, h²/(t·m³)	-2.8·10⁻⁵	-0.0007	-0.0013	ρ – pulp density at the entrance of magnetic separator, t/m³; β – general ferrum content in the concentrate, %
		b, h/m³	0.00062	0.0374	-0.0449	
		c, t/m³	1.17	0.6717	0.794	
		d, %/(t/m³)²	-66.33	-204.08	-127.55	
		e, %/(t/m³)	137.7	427.55	260.97	
		f, %	-2.61	-156.07	-65.84	

No. 3. It corresponds to variation of charge within the range 35 – 50 %. Choice of the range is stipulated by the fact that PI-collector operates with linearized characteristic $P(\varphi)$ near the point $\varphi_o = 45$ %. When $\varphi > 50$ and $\varphi < 35$, large error is observed, it is equal to the module of differences of the values P, which were obtained from non-linear and linearized models; it exceeds the measurement error P.

At the same time, in each case the regulator processed control effect for the feeder, thus varying productivity by initial ore. Fluctuations of productivity values for the finished class (equal to -0.071 mm) on the drain of the classifier of the I stage, as well as of content of general ferrum in concentrate were observed. To provide data analysis and building of static characteristics, it is necessary to wait for finishing of transition processes, i.e. for the moment when they entered into so-called "narrow tube", which is determined by precision of calculation and rounding of Matlab program results. In such way the tasks for loading and capacity were fixed, as well as established values of productivity by initial ore, for the finished class -0.071 mm and values of general ferrum content in concentrate.

Obtained results and their analysis

The point diagrams reflecting relationships between $Q_{in.m1}(\varphi_{m1})$ (**Fig. 1**), $Q_{in.m1}(P)$ (**Fig. 2**), $Q_{-0.071cl}(\varphi_{m1})$ (**Fig. 3**), $Q_{-0.071cl}(P)$ (**Fig. 4**), $\beta(\varphi_{m1})$ (**Fig. 5**) were built using MS Excel program. Energy intensity of grinding process was calculated for each point as the relation $Э = P/Q_{in.m1}$, and the diagram $Э(\varphi_{m1})$ (**Fig. 6**) was built. Non-linear features of the $Q_{in.m1}(\varphi_{m1})$ relationship is stipulated by the fact, that $Q_{in.m1}$ rises with increase of capacity task value of a regulator up to the point of optimal loading (extremal value). Consequent increase of the task value leads to inefficient grinding and excessive consumption of sands. In this connection, regulator decreases $Q_{in.m1}$.

Based on building the trend lines, we can get analytical expressions describing above-mentioned relationships for different ore grades. It helped to calculate MMC mill loading, which is characterized by maximal productivity by initial ore and finished class during work with PI collector, as well by content of general ferrum in concentrate. Parameters during MMC loading, which is equal to 43 %, are calculated as well.

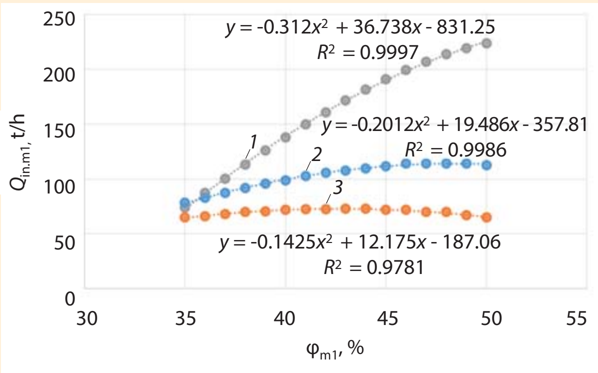

Fig. 1. Relationship between productivity by initial ore $Q_{in.m1}$ MMC and loading φ_{m1}

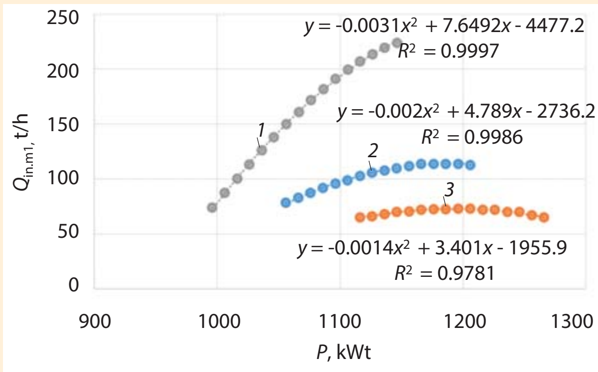

Fig. 2. Relationship between productivity by initial ore $Q_{in.m1}$ MMC and its average capacity P

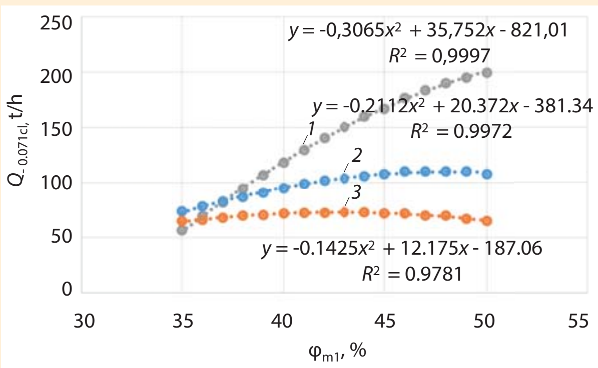

Fig. 3. Relationship between productivity by finished class $Q_{-0,071cl}$ MMC and loading φ_{m1}

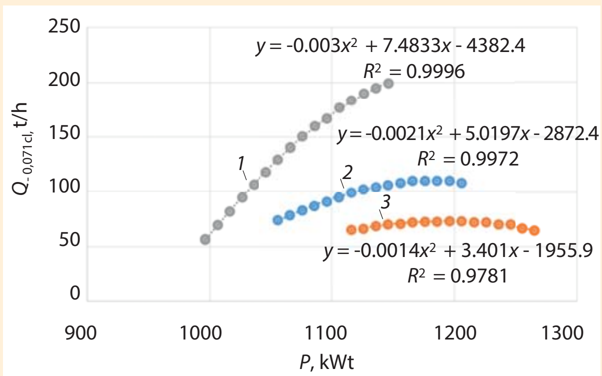

Fig. 4. Relationship between productivity by finished class $Q_{-0,071cl}$ MMC and its average capacity P

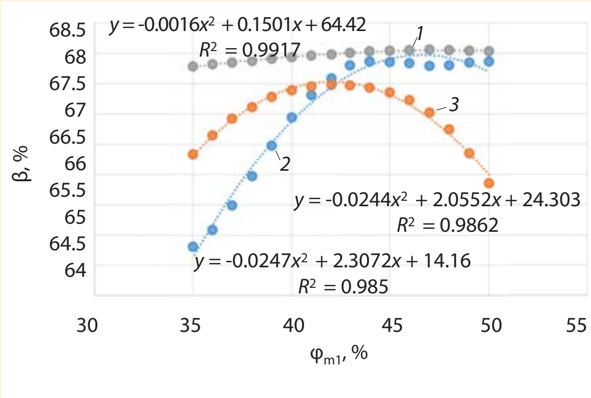

Fig. 5. Relationship between content of general ferrum in concentrate β and MMC loading φ_{m1}

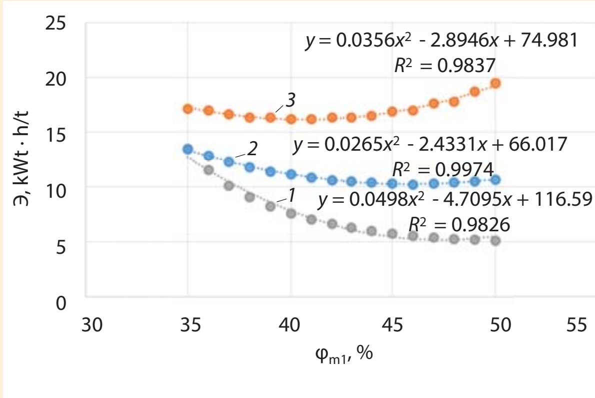

Fig. 6. Relationship between power intensity of MMC grinding Э and loading φ_{m1}

Table 2. **Parameters obtained using the model of enrichment section with support of permanent and optimal MMC loading for different ore grades**

Ore grade	MMC productivity by initial ore, $Q_{in1.m1}$, t/h	MMC productivity by finished class -0.071 mm, $Q_{-0,071cl}$, t/h	Loading φ_{m1}, %	Average capacity P, kWt	General ferrum content in concentrate β, %	Energy intensity Э, kWt·h/t
Supporting permanent MMC loading – 43 %						
No. 1	171.58	150	43	1076.25	68.01	6.1
No. 2	108.17	104.11	43	1136.25	67.64	10.47
No. 3	72.97	72.97	43	1196.25	67.51	16.36
Supporting optimal MMC loading						
No. 1	225.63	200.8	50	1146.25	68.05	5.53
No. 2	114.11	109.87	48.43	1190.61	67.89	10.43
No. 3	72.98	72.98	42.71	1193.4	67.52	16.33
Variation of parameter						
No. 1	54.05	50.8	7	70	0.04	-0.57
No. 2	5.94	5.76	5.43	54.36	0.25	-0.04
No. 3	0.01	0.01	-0.29	-2.85	0.01	-0.03

The obtained regression models have high determination coefficients (> 0.97). Dispersion of points relating to the trend lines is stipulated by rounding of the results, precision of calculation in Excel and Matlab, and not precise measurement of the established value for transition process. Comparative results for the parameters, which were obtained using the model of enrichment section, in the conditions of supporting permanent and optimal MMC loading for different ore grades, are presented in the **Table 2**. It includes established values afyer finishing of transition processes. "Minus" sign means decrease of the parameter in comparison with the operating mode, when MMC filling is supported only at the level 43 %.

According to the technical specifications, general ferrum content in concentrate at Lebedinskiy GOK should be 68.07 %, but at least not less than 67.5 % [15]. Thereby, according to the obtained data, this requirement can be realized for ores of all three grades, with supporting loading 43 % and optimal filling. However, in the last case, productivity by initial ore and by finished class for ore grades No. 1 and No. 2 is higher. Energy intensity is lower in comparison with the conditions, when permanent MMC loading is supported at the level 43 %; it is also possible to increase loading and capacity task. As for ore grade No. 3, the above-mentioned parameters are varying within their measurement error [13], that's why the existing procedure is optimal for this grade.

Conclusions

The following conclusions can be done as a result of conducted investigations. The developed model has complete mathematical description of aggregates. Its use allows to obtain static characteristics for evaluation of possibility of MMC optimal filling choice. This optimal choice provides maximal productivity for initial ore, for finished class of coarseness -0.071 mm, minimal energy intensity of grinding process and correspondence of general ferrum content in concentrate to technical specification. Parameters of ores with different correlation of easily and hard-grinding components can be obtained via use of PI-collector in the MMC automatic control system with possibility to vary mill capacity task. Permanent support of optimal filling will facilitate annual growth of productivity by ore, lowering of energy intensity and concentrate quality fluctuations.

Prospectively development of the system with the following properties is considered: it allows to predict general ferrum content in concentrate at grinding stage using variation of productivity by initial ore in MMC, not waiting for transition of processing material to the finishing concentration stage. For this purpose, it is necessary to develop an algorithm for identification of the static characteristic, which reflects correlation between MMC productivity and loading, as well as between MMC loading and average capacity. Based on this system, we can timely calculate variation of MMC productivity and loading and, respectively, tasks for collector average capacity, which is required for obtaining of aimed production parameters.

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