Oversize Yield in Underground Mine Development

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Abstract: Stoping, which is the initial link in the chain of the production processes of coal-face operation, significantly determines the performance of handling and mining-and-conveyor equipment, as well as losses and ore contamination. The size of the broken ore determines the cost of its mechanical crushing before lifting. Besides, the quality of breaking has a direct effect on the efficiency of ore enrichment. This influence manifests itself mainly in a significant increase in the cost of mechanical crushing and grinding of the ore, which are the most energy-consuming processes in ore dressing. The share of breaking in the direct financial costs of hard ore development, depending on the rock strength, reaches 20 – 35 %. Assuming that the main purpose of blasting operations in the mine is the destruction of the rock massif with creating the conditions for high-performance execution of the subsequent production processes of ore extraction, one should be careful about cost savings on breaking. The fact is that the proposed activities that reduce the cost of drilling and blasting may result in general deterioration of the quality of breaking. Therefore, the criterion of efficiency of any variant of blast hole drilling is to be an integrated indicator of the overall economic result of the entire production process chain, from coal-face operation to enrichment. Naturally, under certain conditions, when the effect of the results of stoping has local nature, the calculations can be made not for the entire production complex. This work analyzes the existing methods of calculating the parameters of blast hole drilling (BHD) in longhole stoping and considers the mining factors that affect the efficiency of stoping in similar geological conditions, namely, the oversize yield and the cost of coal-face operation. In practice, there are no methods for calculating the parameters of BHD that would be recognized as uniform and quite satisfying the needs of enterprises in all regions and types of mining. Depending on the local conditions, the home-developed methods proven by the practice are used, which are improved with experience. This work shows the experience in ore stoping at the Zapolyarny mine, and provides the results of industrial observations of the actual oversize yield.

Keywords: chamber, blast hole drilling, borehole, charge, explosion, oversize.

I. INTRODUCTION

Blast hole drilling in the underground conditions is characterized by a variety of geological conditions of occurrence and the used systems of development, a large number of production processes, a wide range of changes in the diameter of blasting boreholes and wells, and the use of various equipment and methods of drilling, the versatility of blasting methods, the need to support or collapse the overlaying rock, the use of methods and tools that reduce losses and ore contamination, etc.

The existing technology of blast hole drilling in coal-face operation does not always ensure the required quality of rock mass crushing, which results either in high oversize yield or in ore (rock) over-crushing. There may be several reasons for these phenomena: errors in calculating the parameters of the borehole location grid, incorrect choice of the type of explosive for the massif, high or low consumption of explosives, etc.

The destruction of a rock massif is the initial link in the chain of production processes, the efficiency of which largely determines the performance of handling and mining-and-conveyor equipment, and indirectly influences the losses and ore contamination. The efficiency of ore explosive breaking also directly affects the cost of mechanical crushing during enrichment.

While increasing the amount of blasting, the oversize yield often increases. It could also depend on the deterioration of the geological conditions with increasing the mining depth, incorrectly chosen parameters of blast hole drilling caused by the desire to reduce production costs, etc. This phenomenon increases the uncontrolled growth of additional support operations costs, which, in general, affects the economic efficiency of the enterprise.

It is therefore very relevant to solve the problem of predicting the granulometric composition of the yield of the rock mass at the stage of design, based on the known geological and mining data [1].

II. PROPOSED METHODOLOGY

A. Block Diagram

a.1 Influence of the borehole locations on the oversize yield

In boreholes, the ore is usually stoped in layers, the boreholes are arranged in rings, or parallel to each other in the layers, and to the surface of the stoped massif.

This surface may be adjacent with the free space that is sufficient for increasing the amount of blasted ore, or with the rock mass shattered into pieces (shrinkage stoping).

In practice, multirow (two – five rows) short-delay blasting of borehole sets is widely used. The ore is stoped in horizontal, vertical, or inclined layers.

In the layers of ore to be stoped boreholes may be drilled in any direction – horizontal, inclined, and vertical (ascending and descending). The mutual arrangement of the boreholes may be parallel, parallel-contiguous, or ring- or beam-shaped [2, 3].

a.2 The effect of the borehole diameter on the oversize yield

The correct choice of the borehole diameter and the parameters of the borehole grid arrangement determines the efficiency of stoping and all subsequent processes. The definition of rational borehole diameters and their arrangement grid should be closely related to the ore

fragmentation during stoping since the degree of fragmentation affects other processes of ore extraction.

Today, specialists have no consensus about the expediency of using small or large diameter boreholes [2 - 17], which makes the issue of studying this parameter very important.

a.3 The effect of the specific explosives consumption on the oversize yield

The quality of ore crushing varies with changing the borehole diameter and the specific explosives consumption for stoping. With the same specific explosives consumption, the oversize yield increases with increasing the borehole diameter.

The dependencies of oversize yield on the specific explosives consumption for stoping (with the borehole diameter between 44 and 180 mm) shown in Figure 1 have the shape of hyperbolic curves.
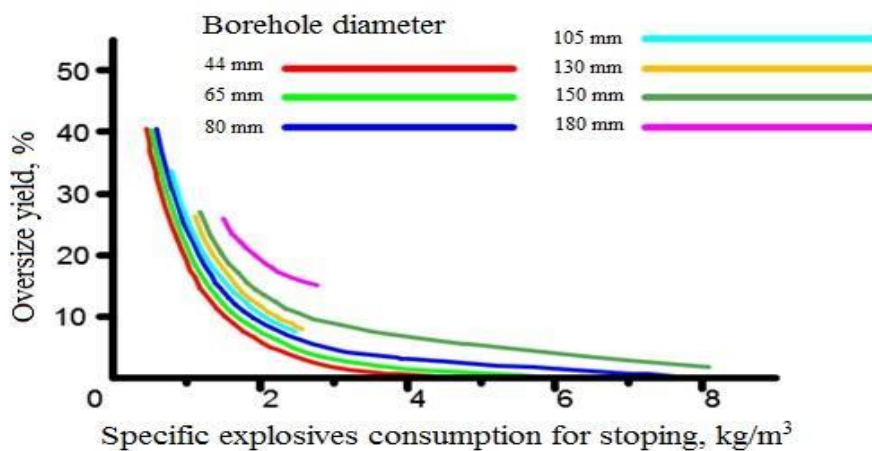
Fig. 1. The dependence of the oversize yield on the specific explosives consumption during ore stoping in blastholes

Increasing the specific explosives consumption, regardless of the borehole diameter, sharply reduces the oversize yield only up to a certain limit, after which the quality of crushing changes insignificantly. With increasing the borehole diameter, only increased specific explosives consumption for stoping is observed. With the same specific explosives consumption, the best fragmentation is achieved by longhole stoping with decreased borehole diameter. This

is ensured by a more uniform distribution of explosives in the destroyed massif [4].

a.4 Dependence of the yield on the size limit

The quality of ore crushing in underground development is usually assessed by the oversize yield. With changes in the size limit, the oversize yield changes (with the same crushing quality).
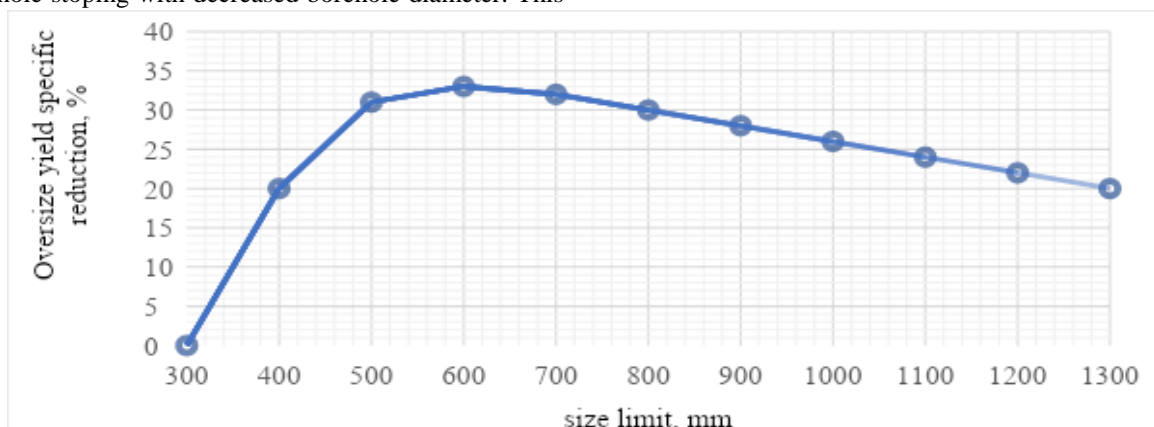
Fig. 2. The dependence of the specific reduction of oversize yield on the size limit

As can be seen in Figure 2, a particularly intensive decrease in the oversize yield occurs after increasing the size limit to 500 – 800 mm.

Thus, a conclusion may be drawn that several key indicators significantly affect the oversize yield, by influencing which one can reduce the oversize yield to a satisfactory percentage [1 – 17].

The group of authors observed the actual oversize yield at the Zapolyarny mine of the Transpolar Branch of the PJSC MMC Norilsk Nickel, the results of which are shown below.

B.   Methods

b.1 General provisions about the location of the experimental work

The Zapolyarny mine of the transpolar branch of the PJSC MMC Norilsk Nickel is operating in the Northern and partly the Southern parts of the sulfide-nickel ores deposit Norilsk-1 belonging to the eponymous differentiated intrusion of gabbro-dolerites, more precisely, to its lower differentiates — axitic, picrite, and contact gabbro-dolerites. Currently, the main objects of development are disseminated ores in the intrusion. Other types of ores are mined along the way, as they are discovered, along with the disseminated ores.

The coal face is operated by the combined shrinkage-and-caving method with the two-stage and one-stage reserves stoping. The two-stage stoping consists of priority reserves stoping in the chambers followed by reserves stoping in the interchamber and panel pillars. The single-stage system consists of working out the reserves in a continuous front in bands with the width of 10 – 15 m without leaving pillars.

According to [18], stoping is performed at the coal face using rings of borehole charges. Stoping is performed to the open surface — the end stope. The end stope is located in the center or on side of the chamber. Its width should depend on the thickness of the stoped section, but should not be less than 3 m, and its length should be equal to the width of the chamber.

In some cases, the development of the end stope is made without making the cut-loose raise by the rings of descending and ascending boreholes. With the chamber height of 30 m and more, the cut-loose raise is made. Coning of the end stope with the use of a cut-loose raise is made with descending or ascending sets of boreholes.

Ore stoping in the chamber is made in vertical layers to the end stope by blasting explosive charges. Depending on the arrangement of the end stope (in the middle or at the end), the bilateral or unilateral procedure may be used for working the chamber. Ore stoping in the chamber is performed in sections (parts) one by one. After a sufficient compensatory space is created, the remaining part of the ore reserves in the chamber may be stoped in a single blast.

Boreholes arrangement in each ring depends on the borehole diameter, and the height and the width of the chamber. Regulations recommend assuming the following parameters of the boreholes [18]:

- with the diameter of 60 and 65 mm, the distance between the rows (the least resistance line) should be 1.1 to 1.5 m, and between the ends of the boreholes — 1.9 to 2.4 m; and

- with the diameter of 105 to 110 mm, the distance between the rows should be 1.8 to 2.4 m, and between the ends of the boreholes — 2.4 to 3.6 m.

The minimum length of the undercharge in the boreholes is 1.5 m.

The actual length of the undercharge in individual boreholes in a ring is not constant and amounts to about 30 % of the total borehole length.

b.2 Analysis and determination of the actual oversize yield

The methodology of the studies envisaged the measuring of the actual oversize yield after longhole stoping of the chambers at the Zapolyarny underground mine of the transpolar branch of the PJSC "MMC Norilsk Nickel, followed by recording the obtained information in a respective certificate.

The main provisions of the methodology of oversize fraction actual measurement are as follows:

1.   At least two chambers (the oversize yield of these chambers) are observed;

2.   Reducing the number of observations to the possible minimum, which would ensure obtaining the data with the accuracy suitable for further engineering calculations;

3.   Compliance with the design conditions for chamber stoping (borehole drilling grid; the number of boreholes in a ring; the depth and the angle of the boreholes; borehole diameter; explosives consumption) to obtain comparable data;

4.   Oversize pieces of rock from the monitored chamber are stored, if possible, in specially designated niches for measurement;

5.   The oversized pieces are measured in the presence of a representative of the organization where the authors work (FSAEI HE SFU) and the representative of the mine; the measurement is followed by photographing oversize pieces with a ruler with 10-centimeter increments;

6.   After the actual measurement of the oversize fraction, a consolidated certificate is made, which covers all main indicators of chamber stoping.

Below are the results of determining the actual oversize yield percentage at the Zapolyarny underground mine.

The observations were performed during the stoping of the ore massif from transportation and supply gates TDSH 41-7 and TDSH 42-7 (Fig. 3).
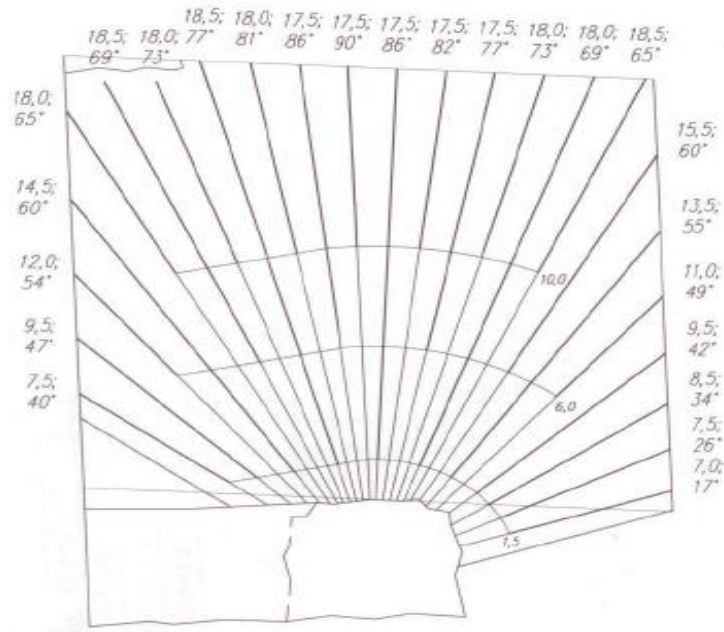
Fig. 3. A project of monitored ring No. 25 in ore massif TDSH 42-7

The stoped ore massif was represented by picrite, taxitic, and contact gabbro-dolerite. The rocks were large- and medium-block, strongly and moderately fractured. Rock strength f was 10 – 14. Rock fracturing was heavy.

The oversize pieces were stored in a designated area. The appearance of oversize pieces is shown in Figure 3.

a)

b)

c)

d)

e)   **f)**

a, b, c, d, e – the appearance of oversize pieces before secondary crushing; f – the appearance of oversize pieces after secondary crushing

Fig. 3. Appearance of oversize pieces after stoping in ring No. 25 TDSH 42-7

The main characteristics of the ore massif longhole stoping are shown in Table 1.

The actual oversize yield in ring No. 25 was 33.11 %, and in the entire developed ore massif — 30.7 % (Table 2).

Table 1. Technological characteristics of longhole stoping of the ore massif TDSH 41-7 and TDSH 42-7

Works location	The total number of boreholes	Borehole depth, m	Total boreholes length, m	Distance between boreholes in a row, m	Specific explosives consumption, kg/m³	Rock mass separation, m³	Total explosives, kg	Rock mass yield, m³/m
TDSH 41-7 (rings 1, 2, 3)	42	6.5 – 20.0	759.0	1.5	0.82	2,443.0	2021.0	3.22
TDSH 41-7 (rings 4, 5)	15	7.0 – 19.0	185.0	1.5	0.39	1,553.0	607.5	8.39
TDSH 41-7 (rings 6, 7)	16	6.5 – 19.5	275.0	1.5	0.38	1,600.0	608.0	5.82
TDSH 41-7 (rings 8, 9)	16	6.5 – 19.5	185.0	1.5	0.37	1,616.0	608.0	8.74
TDSH 41-7 (ring 10)	16	6.5 – 19.0	285.0	1.5	0.84	728.0	610.0	2.55
TDSH 41-7 (rings 11, 12)	48	6.5 – 19.0	702.5	1.5	0.40	1,543.0	624.0	2.20
TDSH 41-7 (rings 14, 15)	26	7.0 – 19.5	372.0	1.5	0.52	1,643.0	853.0	4.42
TDSH 42-7 (ring 16)	22	7.0 – 21.0	362.5	1.5	1.52	767.0	1,171.0	2.12
TDSH 42-7 (ring 17)	22	7.5 – 20.5	356.5	1.5	1.84	767.0	1,411.0	2.15
TDSH 42-7 (rings 21, 22, 23)	35	7.0 – 20.5	702.0	1.5	0.77	2,369.0	1,817.5	3.37
TDSH 42-7 (ring 24)	17	6.5 – 19.0	353.0	1.5	1.20	736.0	888.5	2.08
TDSH 42-7 (ring 25)	24	7.0 – 18.5	349.0	1.5	1.49	755.0	1,092.0	2.09

Table 2. The summary table on the results of accounting the actual oversize yield at the Zapolyarny mine of the Transpolar Branch of the PJSC MMC Norilsk Nickel

No.	The number of worked rings in chambers TDSH 41-7 and TDSH-42-7	The volume of the separated rock mass, m³	The volume of the rock mass loaded to the millhole, m³	The total oversize volume, m³	Explosives consumption for secondary crushing, kg	The average oversize yield, m³	The average oversize yield, %
1)	TDSH 41-7 (rings 1, 2, 3)	2,443.0	1,863.0	580.0	145.0	0.75	23.74
2)	TDSH 41-7 (rings 4, 5)	1,553.0	1,153.0	400.0	100.0	0.8	25.75
3)	TDSH 41-7 (rings 6, 7)	1,600.0	1,100.0	500.0	125.0	1.1	31.25
4)	TDSH 41-7 (rings 8, 9)	1,616.0	1,136.0	480.0	120.0	0.9	29.70
5)	TDSH 41-7 (ring 10)	728.0	508.0	220.0	55.0	0.85	30.22
6)	TDSH 41-7 (rings 11, 12)	1,543.0	1,023.0	520.0	130.0	1.2	33.70
7)	TDSH 41-7 (rings 14, 15)	1,643.0	1,123.0	520.0	130.0	1.0	31.64
8)	TDSH 42-7 (ring 16)	767.0	537.0	230.0	57.5	0.9	29.98
9)	TDSH 42-7 (ring 17)	767.0	467.0	300.0	75.0	1.3	39.11
10)	TDSH 42-7 (rings 21, 22, 23)	2,369.0	1,619.0	750.0	187.5	1.0	31.65
11)	TDSH 42-7 (ring 24)	736.0	526.0	210.0	52.5	0.95	28.53
12)	TDSH 42-7 (ring 25)	755.0	505.0	250.0	62.5	0.86	33.11
Totals for the developed workings TDSH 41-7 and TDSH 42-7						0.97	30.70

The results of the observations were used for statistical processing of the results, based on which the oversize yield could be predicted during subsequent working with the same technological parameters.

b.3 Statistical processing of the results obtained during monitoring

The results of measuring oversize pieces in chambers TDSH 41-7 and TDSH 42-7 were statistically processed by the maximum oversize dimensions and the volume of oversize pieces (including gross errors in the statistics of measurement). The initial data are shown in Tables 3 and 4.

Table 3. Characteristics of the ordered sample by the maximum oversize dimensions

No.	The number of blasted rings in boreholes	The number of oversize pieces corresponding to the largest dimension							The total number of oversize pieces
		70 cm	80 cm	90 cm	100 cm	110 cm	120 cm	130 cm and more	
1	3	15	16	12	10	14	12	25	104
2	2	16	12	12	8	13	12	13	86
3	2	10	10	20	9	17	12	7	85
4	2	9	11	13	14	12	16	10	85
5	1	4	16	8	10	10	10	13	71
6	2	15	13	10	7	8	11	15	79
7	2	15	11	12	11	15	15	5	84
8	1	10	7	8	5	3	6	4	43
9	1	10	9	5	6	10	5	1	46
10	3	15	13	16	10	13	1	10	78

11	1	11	12	11	14	0	7	8	63
12	1	13	10	16	9	1	3	6	58
Average		11.9	11.7	11.9	9.4	9.7	9.2	9.8	73.5
Variance		12.6	7	16.6	7.7	31.3	22.3	39.8	323.2
Average*		12	12	13.1	10.2	9.9	9.7	9.7	76.6
Variance*		15.3	3.5	13.4	5.9	35.1	26	12	106.8

The statistical analysis of the table is presented below. First, note that the number of oversize pieces in all tests, except for tests one, eight, and nine, differs from the average number by no more than 15.5 pieces, which is 21 %. Such tests (with a sharp deviation from the average) are called "outbreaks" and are usually not considered in the statistical analysis since this significantly increases the error in further calculations.

Note also that tests eight and nine with the least amount of oversize yield consisted of one explosion in one ring, while test one with the greatest oversize yield consisted of explosions in three rings, which was the maximum number of explosions in this series of tests. The series of tests consisting of two explosions provided results with the lowest deviations from the average.

Thus, except for tests one, eight, and nine, we get the values shown in lines "average*" and "variance*", while the lines "average" and "variance" account for all 12 tests. Note that the value in the line "variance*" by the total number of oversize pieces shown in the last column is significantly lower than the corresponding value in the line "variance", which means smaller deviation values of the statistical range (the number of oversize pieces) from the average, and thus the accuracy of prediction for consequent tests.

In general, it may be argued with high probability that with further use of the current blast hole drilling parameters, most values (80 – 90 %) of the quantities of oversize pieces will vary between 12 ± 3 (i.e., 9 – 15) pieces for smaller values of the maximum size (70 – 90 cm), gradually decreasing to 10 ± 3 (i.e., 7 – 13) pieces for large values of the maximum size (100 cm and more) with the total number of 77 ± 15 (62 – 92) pieces.

Table 4. Characteristics of the interval series by the volumes of oversize pieces

No.	The number of blasted rings in boreholes	The number of oversize pieces corresponding to the largest volume							The total number of oversize pieces
		$0.16 – 0.35$ m³	$0.36 – 0.45$ m³	$0.46 – 0.55$ m³	$0.56 – 0.65$ m³	$0.66 – 0.75$ m³	$0.76 – 0.85$ m³	$0.86 – 0.95$ m³	
1	3	7	15	9	10	23	16	24	104
2	2	12	14	7	8	14	13	18	86
3	2	9	10	11	10	11	22	12	85
4	2	12	10	11	11	8	16	17	85
5	1	4	6	9	7	15	15	15	71
6	2	10	10	7	8	11	14	19	79
7	2	6	9	11	13	13	14	18	84
8	1	4	6	6	4	6	6	11	43
9	1	2	1	4	10	1	17	11	46
10	3	1	17	1	15	9	15	20	78
11	1	9	4	8	12	15	6	9	63
12	1	13	12	14	2	5	5	7	58
Average		7.4	9.5	8.2	9.2	10.9	13.3	15.1	73.5
Variance		16.4	21.9	12.3	13.4	33	26	25.9	323.2
Average*		8.4	10.2	8.8	9.6	11.2	13.3	15	76.6
Variance*		16.3	15.2	13.7	14.8	11.7	26.5	21.5	106.8

A similar pattern can be observed with the distribution of oversize pieces by their volumes, given that in this case, it is rather an inverse dependence of the average number of oversize pieces on their volume; with that, the dependence itself is less evident. By comparing, it may be concluded that pieces with a large maximum size do not always have the highest volume (i.e., have an elongated shape). The overall distribution pattern by the volumes of oversize pieces is similar to the previous one: with the currently used blast hole drilling parameters, the values of oversize pieces are expected to be 8 ± 3 to 15 ± 3, depending on their size.

C.   Algorithm

In the previous review of the existing methods for determining the fragmentation of stoped rock mass, it has been found that today there is no unified scientifically based policy for determining the parameters of blast hole drilling operations that would take into account the totality of factors that affect the results of stoping [1]. Typically, these methodologies do not take into account the mutual influence of the following factors: physical and mechanical properties of the massif (cuprous, disseminated, rich ore; backfill massif, upon opening the roof and/or sides of the working), the type of used explosives, charge diameter, charge structure, location of charge initiation, charge length, and the amount of undercharge, the length and the quality of tamping, interaction of simultaneously blasted charges. This explains the instability of blast hole drilling indicators, their low efficiency and,

consequently, increased oversize yield. This work discusses the effect of various geological factors on the oversize yield: borehole layouts, borehole diameter, specific consumption of explosives, etc.

Industrial observations revealed several features allowing to partially solve the problem of the high percentage of oversize yield.

Firstly, as can be seen from Table 1, the largest number of oversize pieces was formed after the first explosion, due to several reasons:

−   the explosion was made in closed space (i.e., the energy of the explosion could only work on one free surface); and

−   the greatest number of blast rings were involved in the explosion (three blast rings, with the total number of boreholes equal to 42), which determined the explosion of a large amount of explosives in the same closed space.

Secondly, it is clear that in the case of blasting only one ring, the number of formed oversize pieces was much less than in the case of blasting two or three rings. This is because microcracks formed during blasting of a large amount of explosives do not have the time to be created and accumulated in the destructed rock massif, whereby the rock holds tight above, and the risk of collapsing is minimal.

Thirdly, the possible reasons for the appearance of many oversize pieces after blasting may be the incorrectly chosen blast hole drilling parameters, namely, the distance between the blast rings.

The authors note that additional study of these and related issues will greatly improve the quality of stoping at the enterprise.

During the process observations, the following conclusions have been made:

1)   the main factors that influence the oversize yield are the blast holes arrangement grid (the least resistance line and the distance between the ends of the blast holes along the contour of the chamber), as well as specific explosives consumption for stoping of 1 m^3 ore, depending on the diameter of blast holes;

2)   the presence of oversize pieces harms the efficiency of loading and transportation machines;

3)   the blast hole drilling parameters used at the Zapolyarny mine currently result in increased oversize yield, which increases the cost of ore development due to additional costs of secondary crushing; and

4)   it is necessary to comprehensively assess the oversize yield and to develop recommendations for minimizing it.

III.   RESULT ANALYSIS

One of the main indicators that determine the economic efficiency of the enterprise is the quality of rock crushing during blasting operations. Throughout the history of explosives, scientists sought to develop a model that would be able to predict and describe the size of rock pieces after blasting. Assessment of the efficiency of blast rock fragmentation is mainly determined by two parameters: the oversize yield and the degree of rock fragmentation.

Currently, there are many models [19 24] that provide an ability to predict both these parameters. However, as a rule, the proposed methods do not consider the mutual influence of several factors such as the physicomechanical properties of the massif, the type of the used explosive, charge diameter, charge structure, location of charge initiation, charge length, the amount of undercharge, the length and the quality of tamping, and the interaction of simultaneously blasted charges. This explains the instability of blast hole drilling indicators, their low efficiency and, consequently, increased oversize yield [1].

Rock stoping experience shows that even in progressive methods of BHD, it is not possible to eliminate oversize yield, which may reach 20 % or more. In this regard, there is an urgent task to eliminate these oversize pieces using the means and methods available at the enterprise. By the type of power supplied to the object of destruction, the following methods of crushing oversize pieces may be distinguished: explosive, mechanical, electrical, thermal, hydraulic, plasma, acoustic, chemical, and combined.

However, it is economically more feasible to prevent oversize yield in order not to waste time and money for removing them. However, the above models imply the maximum participation of various specialists in the calculation, which can affect the result. In recent years, due to the widespread use of personal computers, the demand for blasting calculation software has dramatically increased. As a result, there are quite many computer programs in the market today that are intended for solving various tasks related to blasting operations and production. Some of these programs are mentioned in [25, 26]. Most of these programs are currently used by various companies, but with the advent of more sophisticated and powerful personal computers, software technology is being improved; therefore, there is a need to constantly check for updated versions of the existing or new applications.

Currently, the dependence of the average piece of stoped rock mass on the use of blast hole drilling parameters and rock properties are calculated for each mining enterprise by the empirical principles based on the experience of similar enterprises, and its rational values at the operating enterprise — through a series of experimental and industrial explosions [1, 27]. The authors hope that with each passing year, more and more enterprises will adopt the use of specialized software products that will enhance safety and reduce the possibility of errors in determining blast hole drilling parameters as a result of human error.

IV.   CONCLUSION

Improvement of blast hole drilling is one of the ways of increasing the efficiency of the mine development. Depending on how well the blast hole drilling parameters have been calculated, the performance of the mining unit may be significantly improved.

Based on the foregoing, it can be concluded that the production volumes are steadily growing, along with the demand for creating more powerful explosives and for the development of new models (techniques) that facilitate prediction or determination of the granulometric composition of the stoped rock. During the implementation of the internal development strategy at enterprises, one should regularly perform analysis of technical and economic performance.

The analysis should contribute to changing the orientation of the economic policy of the company from the predominantly costly to the resource-saving and environmentally friendly one.

It is necessary to study many models of predicting the yield of granulometric composition of the stoped rock mass, which would allow obtaining a more detailed picture of the effect of the main factors on the results of blasting. The validity of this analysis depends not only on the perfection of methods but also on the efficiency of its implementation. At the present rate of the development of science and technology, production should be easily managed. Hence, the basic requirements for the analysis are its systematic nature, comprehensiveness, and promptness.

Thus, this paper presents the experience in stoping ore in a chamber development system at the Zapolyarny mine, from which a conclusion can be drawn that the blast hole drilling parameters used at the enterprise today do not fully ensure completeness and quality of chamber working. This fact has determined the need for further work on choosing appropriate blast hole diameter, drilling grids, and the explosive used.

V. ACKNOWLEDGMENT

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