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Substantiation of parameters of mine working drivage with blasting technique and cleaning charges in advance cutting holes

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Abstract: Drilling and blasting operations (D&B) are the most difficult and laborious operations in the process of mineral extraction, requiring permanent development and upgrading of methods and equipment. The aim of the study is to substantiate the parameters of drilling and blasting drivage of horizontal and inclined (up to 12o) mine workings on the basis of high-performance self-propelled equipment and new designs of box cuts with cleaning explosive charges (0.2 kg of 6ZhV ammonite), placed in advance holes of 65 to 105 mm in diameter. The paper presents the results of the analysis of practical experience and scientific achievements in the field of drilling-and-blasting rupture of solid media and continuum mechanics. The need for new designs of box cuts, reliability of which in the formation of high-quality (clean) cut cavity reaches 0.95-1.00, was substantiated. New design options of box cuts have been developed, the peculiarity of which consists in provision of sufficient compensation (peripheral) volume with the use of blast hole cut charges for blasting rupture of trapezoidal partitions, with the compensation volume factor of 2.50 to 1.34. Promising areas of research were shown using the example of complicated structure ore deposits and large faults in the Kirovograd ore district and crystalline rocks of the Ukrainian shield at the following mines of Ukraine: PJSC KZhRK, CJSC Sukhaya Balka (Kryvyi Rih), Vostok-Ruda LLC, SE VostGOK (Zheltye Vody), CJSC ZZhRK, etc. Promising areas of research were proposed using the example of complicated structure ore deposits typically formed at the junction of large faults; besides, the use of environmentally friendly emulsion explosives and self-propelled emulsion explosive chargers was recommended.

Keywords: ore mass, mine workings, drivage with blasting technique, box cut, cleaning charge, safety, performance

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Обоснование параметров буровзрывной проходки горизонтальных горных выработок с подчищающими зарядами взрывчатых веществ в опережающих скважинах вруба

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Аннотация: Наиболее сложными и трудоемкими в добыче полезного ископаемого, требующими постоянного изучения и совершенствования технологии и технических средств для ее осуществления являются буровзрывные работы (БВР). Целью исследования является обоснование параметров буровзрывной проходки горизонтальных и наклонных (до 12º) горных выработок на базе высокопроизводительного самоходного оборудования и новых конструкций призматических врубов с подчищающими зарядами взрывчатых веществ (ВВ) по 0,2 кг аммонита марки 6ЖВ, размещенных в опережающих скважинах диаметров от 65 до 105 мм. В работе представлены результаты анализа практического опыта и научных достижений в области буровзрывного разрушения твердых сред, механики сплошных сред. Обоснована потребность в новых конструкциях призматических врубов, надежность работы которых по образованию качественной (чистой) врубовой полости достигает 0,95–1,00. Рекомендованы варианты новых конструкций призматических врубов, особенность которых заключается в обеспечении каждой из них достаточным компенсационным объемом для работы шпуровых врубовых зарядов ВВ на разрушение трапециевидных перегородок с коэффициентом компенсационного объема от 2,50 до 1,34. Показаны перспективные направления исследований на примере рудных месторождений сложного строения и мощных тектонических разломов Кировоградского рудного района и кристаллических породах Украинского щита на шахтах Украины: ПАО «КЖРК», ЧАО «Сухая Балка» (г. Кривой Рог), ООО «Восток-Руда», ГП «ВостГОК» (г. Желтые Воды), ЧАО «ЗЖРК» и др. Предложены перспективные направления исследований на примере рудных месторождений сложного строения с типичными образованиями в узлах сочленения мощных тектонических разломов и применения экологически чистых эмульсионных ВВ, а также самоходных зарядчиков эмульсионных ВВ.

Ключевые слова: рудные массивы, горные выработки, буровзрывная проходка, призматические врубы, подчищающий заряд, безопасность, эффективность

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Introduction

Drilling and blasting operations (D&B) are the most difficult and laborious operations in the process of mineral extraction, requiring permanent development and upgrading of methods and equipment [1, 2]. Construction of mine workings requires implementation of a whole cycle of mining operations, among which drilling and blasting operations are primary [3, 4]. Taking into account the parameters and various purposes of the workings (transport, ventilation, etc.) is also important in the design and implementation of D&B [5– 7]. High quality of drivage is a key prerequisite of safe operation throughout a mine working life cycle [8, 9]. This work is a continuation of the authors' research, the main scientific and practical findings of which are most fully presented in [10–12].

Research objectives and tasks

The aim of the study is to substantiate the parameters of drivage with blasting technique of horizontal and inclined (up to 12°) mine workings on the basis of high-performance self-propelled equipment and new designs of box cuts with cleaning explosive charges (0.2 kg of 6ZhV ammonite), placed into advance holes of 65 to 105 mm in diameter. This will ensure high-quality mine working drivage with the advance per round of at least 3.0–3.5 m, increasing the drivage rate from current 50–70 m/month up to 300 m/month per a tunneling system in the near future. To achieve this goal, the experimentally obtained data are compared with the calculated data. The research objective is development and performing of tests on new designs of box cuts with cleaning explosive charges (0.2 kg of 6ZhV ammonite), placed into advance holes for drivage with blasting technique of horizontal and inclined (up to 12°) mine workings in the conditions of a specific metal deposit [13].

Study of the rock mass characteristics at an ore deposits of complicated structure

The considered ore deposit of a complicated structure is a typical ore body at the junction of large faults in Kirovograd ore district in Ukraine. The combination of physico-mechanical properties of the rocks enclosing this deposit naturally determines geotechnical conditions under which development and face-entry drivages and stoping are to be safely performed. To provide the safety, assessing stress-strain state (SSS) of the rock mass (geotechnical monitoring) should be permanently performed by various methods and equipment, including field studies of the impact of man-made voids on the rock mass stability under the mine conditions [14]. When assessing the stability of mine workings, permanent assessing the rock mass weakening factor (the ratio of adhesion along the contacts of natural fractures to adhesion in intact solid rock) is required, this is decisive in determining the need and type of supporting these workings. Its value for the considered rock masses ranges 0.32 to 0.39 [15].

Deposits of this type demonstrate similarity in rock mass quality and, hence, similar practice of extraction and supporting are applicable to them. Mining at the deposit allow forecasting rock mass parameters at similar type deposits. The results of field measurements for the considered rock masses showed that the direction of the fractures, as a rule, coincides with the designed direction of the ore body extraction. In this case, fractures propagating at angle of more than 45° prevail [16].

The main processes occurring in rock mass after goaf formation (development workings and stopes), subject to monitoring [17]:

– formation of SSS of rock mass and its change over time;

– displacement (failure) of rocks, manifested in various forms;

– interaction of rocks and supports.

Rock pressure is produced by gravity – weight of vertical overlying rock column (up to the surface) and the shear stresses produced by tectonic movements. The forces of rock pressure objectively exist independently of the formation of man-made voids in a rock mass, but the processes of changing the integrity of surfaces are

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possible only after the void formation. Actual geomechanical rock mass force field at each point of the space coordinate grid is characterized by SSS values depending on the shape and parameters of mine workings. At permissible parameters, mine workings remain in stable safe state [18].

The manifestation of irreversible deformations changing the rock mass took place in the course of drivage of the exploration-and-production shaft in the form of fracturing and spalling of rock pieces from the wall at a depth of 870–880 m. This evidences potentiality of dangerous unloading of the rock mass (rock burst) after the formation of a goaf. Ore-enclosing rocks and ore bodies of such a deposit, composing the main mass, are very stable; exploration workings in them are capable to maintain their shape for a long time without noticeable signs of deformation, with the exception of appearing fractures and small spalls caused by changes in the natural stress field.

At the deposit, zones of rock weakness are collectors of static groundwater reserves and, at the same time, may cause cutter breaks (wedges) and inrushes in some tectonically disturbed areas in the places of clay gouge formation – in the discussed case, these are zones of Syenite and Sekushchy faults, whose thickness is 40-50 m. Hole drilling data show that zones of unstable rocks were found in all faults. In such fracturing zones, scabbing of large rock pieces at the contacts of the fault footwall and hanging wall. Therefore, in the course of drivage through these zones, rock scaling, and, in some cases, bolting are required [19].

There are no large gneiss masses at this deposit, but some gneiss is found within granites. Rockbursts, pressure bumps, and rock outbursts were not observed. In the course of mining, when approaching deep tectonic zones (faults), advance boreholes should be drilled to determine both watering and gas content. Control of gas content in the mine air should be systematically carried out by the ventilation and gas rescue services [20]. The main activities during mining should be aimed at reducing harmful aerosols content in the mine air to the safe level and minimizing their impact on operational personnel. 673 deep exploration boreholes have been drilled at the deposit, not taking into account underground drilling. All the boreholes are plotted to the abandonment map, but not all ones have been plugged [21].

Justification of selection of equipment for mine working drivage

A tunnelling system (including self-propelled equipment) for drivage of horizontal and inclined workings includes [22]:

– jumbo drill rig;

– LHD.

Two options of tunnelling system with diesel engine undercarriage were considered:

1) AtlasCopco:

– Boomer 281 (282) drilling rig;

– ST 3.5 LHD;

2) Tamrock:

– drilling rig Minibur 1F;

– TORO 151 LHD.

Performance specifications of the considered systems for drivage of horizontal and inclined workings are presented in Table 1.

Productivity on loading and transportation of rock mass while driving mine workings and in working faces of Boomer 281 (282)/ST 3.5 system (Table 2) and Minibur 1F/TORO 151 system (Table 3).

Table 1

Performance specifications of the systems for drivage of horizontal and inclined workings

Indicator	Heading set of equipment (tunneling system) AtlasCopco		Heading set of equipment (tunneling system) Tamrock	
	Drilling rig	LHD	Drilling rig	LHD
Facility type	Boomer 281(282)	ST 3.5 (ST710)	Mini-drill 1F	TORO 151
Bucket capacity, m^3		3.6		1.75
Weight-carrying capacity, t, at γ = 1,66 t/m^3		5.2		2.5
Overall dimensions, mm:				
Length	11,620	8,824	8,500	6,970
Width	1,650	2,040	1,200	1,480
Height				
for transportation	2,100	2,104	1,850	1,235
at discharge		4,374*		
on duty		1,719		1,740
Engine:				
power, kW (hp)	42 (75)	149 (200)	30 (40)	53 (71)
air tyre	8,25R15	17,5×25	10×15	12×20
tank capacity, l	60	191	50	80
fuel consumption, kg/h	13.5	36	7.2	13
installed power, kW	63		55	
operating weight, t	9.3	18.2	7.0	8.7
rock-boring machine, type	COP 1432		1Gl300S	
water consumption, m^3/h	2.9		2.9	
noise level, dB (A)	<106		<106	
drilling-off area, m^2	10÷31		4÷18	
air consumption for face airing, m^3/s		17		6

Note. *From sill to bucket edge when unloading directly into transport vessels.*

Table 2

Boomer 281 (282)/ST 3.5 system performance

Indicator	Haulage distance									
	50	**100**	**150**	**200**	**250**	**300**	**350**	**400**	**450**	**500**
Hourly throughput, tph	107.2	89.8	77.6	67.8	60.8	55	49.8	45.8	42.2	39.4
Shift throughput, t/shift	536	448	388	338	304	272	248	228	210	196
Daily throughput, tpd	1,608	1,344	1,164	1,014	912	816	744	686	630	588
Yearly throughput, ktpa	386	322	254	244	218	196	178	164	152	142

Table 3

Minibur 1F/TORO 151 performance

Indicator	Haulage distance									
	50	**100**	**150**	**200**	**250**	**300**	**350**	**400**	**450**	**500**
Hourly throughput, tph	53.6	44.9	38.8	33.9	30.4	27.5	24.9	22.9	21.1	19.7
Shift throughput, t/shift	268	224	194	169	152	136	124	114	105	98
Daily throughput, tpd	804	672	582	507	456	408	372	343	315	294
Yearly throughput, ktpa	193	161	127	122	109	98	89	82	76	71

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Based on the analysis of advantages and disadvantages of the systems of the world's leading firms, the following conclusions can be drawn:

- high ergonomic characteristics of Atlas Copco LHDs and dump trucks are their advantage over Tamrock;

- productivity of Boomer 281 (282)/ST 3.5 of Atlas Copco in drivage is 2 times higher than that of Minibur 1F/TORO 151 system of Tamrock at the same distances of rock mass transportation;

- annual productivity of ST 3.5 LHD is 2 times higher than that of TORO 151;

- to mechanize blast-hole charging, it is advisable to use a self-propelled charging unit with diesel-engine drive.

Development of blasting pattern for mine working drivage

In practice, rational distances between the blast holes are calculated using the well-known formula of prof. V.N. Mosinets [13, 16]:

$$a = KW \text{, m,} \qquad (1)$$

where K is coefficient taking into account the blast hole designation, K = 1.0 - 1.3 for outer holes, K = 0.75 - for peripheral sill holes, K = 0.85 for peripheral roof holes, K = 0.95 for peripheral wall holes.

The formula is valid when the peripheral hole diameter ranges 50 to 200 mm. The disadvantages of peripheral blasting are as follows: the number of blast holes increases by 10–15%, and the charging labor costs increase. Based on the practice of peripheral blasting, the distance between the holes in the peripheral row should be adjusted for the depth of fracturing caused by blasting of the outer pre-peripheral hole charges of the latest row:

$$a_к \le (0{,}8-1{,}0)W_к \text{, m} \qquad (2)$$

where $W_к$ is the line of least resistance of the peripheraling charges, m, which on a field basis for hard rocks (Protodyakonov rock hardness index f = 16) is equal to 0.60-0.65 m. Thus, the distance between the peripheral blast holes is equal to $a_к \le (0{,}8-1{,}0)\cdot 0{,}6 \le 0{,}5-0{,}6$ m. It is taken at: a = 0.55 m on average.

The projected specific consumption of explosives is determined on the basis of calculation or experience in mine working drivage with certain type of explosive. The new explosive specific consumption for working drivage is adjusted according to the formula:

$$q_н = q \cdot K_{раб} \text{, kg/m}^3\text{,} \qquad (3)$$

where q is the used explosive specific consumption (ammonite No. 6ZhV), kg/m^3, $K_{раб}$ is the explosive performance factor, units;

$$K_{раб} = \frac{e_н}{e}, \qquad (4)$$

where $e_н$ is performance of the explosive used, cm^3. Performance of ammonite No. 6ZhV and grammonite 79/21 is equal to e = 360–370 cm^3 at charging density of ρ = 1.0–1.1 g/cm^3 (from 0.8 g/cm^3 for grammonite 79/21; granulite AS-4; AS-4V; AS-8; AS-8V up to 1.6 g/cm^3 for ammonite No. 1 for hard rock and other); when using other explosive type, its specific consumption is adjusted through the performance factor.

Two options of the equipment used for drilling operations are considered:

– AtlasCopco Simba M4C drilling rig;

– Tamrock SOLO 1L drilling rig (Table 4).

To effectively conduct blasting operations with ensuring minimizing seismic effect on the enclosing rock mass stability, the use of the following explosive types is recommended and regulated (Table 5).

Table 4

Technical characteristics of drilling rigs

Indicator	AtlasCopco	Tamrock
Facility type	Simba M4C	SOLO 1L
Rock-boring machine	COP 2550EX	TAMROCK 510LKh
Hole diameter	64–102	64–89
Optimum depth, m	51	30
Overall dimensions, mm		
Length	10,500	6,450
Width	2,350	1,670
Height		
For transport	2,875	2,150
In working position	2,965	2,750
Engine:		
power, kW (hp)	115 (156)	30 (42)
Installed power, kW	118	60

Table 5

Explosive seismic effect

Explosive type	Charge density g/cm^3	Explosion heat, kJ/kg	Velocity of detonation (km/s)	Seismic effect index
No. 6ZhV ammonite	1.0–1.2	4,305	3.6–4.8	1.0
Grammonite 79/21	0.80–0.85 (bulk density)	4,285	3.2–4.0	1.0
Granulite AS-4; AS-4V	0.8–0.9 (bulk density)	4,522	3.0–3.5	1.03
Granulite AS-8; AS-8V	0.8–0.95 (bulk density)	5,191	3.0–3.6	1.12
Ammonal A-200	0.95–1.1 (explosive cartridge)	4,932	4.2–4.6	1.07
Ammonal A-10	0.95–1.2 (explosive cartridge)	5,645	–	1.19
Hard rock ammonal No. 3	1.0–1.1 (explosive cartridge)	5,684	42–4.6	1.19
Hard rock ammonal No. 1	1.43–1.58	5,400	6.0–6.5	1.17

Development of box cuts with two holes having diameter of d = 65 mm

At peripheral blasting, the number of peripheral holes (along the cross-section contour) is determined on the basis of the previously justified parameters. The holes at the face include cut holes, the number of which depends on the face design. The greater the number of holes per a cut, the higher their performance. For finishing the cut cavity, auxiliary blast-hole charges are used, which determine the quality of rock mass breaking at the face and the length of advance per round. All blast holes outside the cut and the cross-sectional contour are evenly distributed in the face and called outer holes [23, 24].

Experience shows that due to the impossibility of observing the calculated geometry of the cut due to violations occurring in the process of face drilling, the hole charges do not ensure cleaning of the cut cavity for the entire depth of the advance per round [25, 26]. As a result, the length of advance per round at low blast hole utilization rate. To improve the quality of the cut cavity cleaning, a cleaning charge is used, which is blasted last in the cut at a greater depth (300–400 mm) in peripheral hole. To clean the cut cavity, it is advisable to place the cleaning charge in a small-diameter peripheral hole (65, 74, 85 mm), if there is no confidence in effective cleaning the cut cavity without applying the cleaning charge.

The use of a cleaning charge in a cut increases the reliability of its cleaning and ensures high-quality cut formation up to 90–95%. To ensure horizontal and inclined (up to 12°) mine workings drivage advance up to 3.5 m and more, box cuts with two holes 65 mm in diameter (d) with bottomhole placed cleaning charges (0.2 kg of 6ZhV ammonite) were developed, one of which is shown in Fig. 1.

Study of perimeter (compensation) volumes for different cut options

Compensation (peripheral) volume factors for the different cut options during explosions for blasting the first charge of cut holes for peripheral hole: d = 105 mm, 2.5; d = 85 mm, 2.0; d = 74 mm, 2.14. At blasting of the fourth cut charge, the compensation (peripheral) volume factors for peripheral holes are as follows: d = 105 mm, 1.44; d = 85 mm, 1.36; d = 74 mm, 1.34.

It was found that action of the first charges in the cut bottom part occurs under conditions with excessive compensation volume (tight-face blasting), and the action of the last cut charges (the fourth in delay) occurs under weak compressing conditions. The main blast hole parameters are diameter (d_{bh}), length (l_{bh}) and depth (Fig. 2).

Correct drilling implementing the proposed cut designs with observing the above-described cut charge operating conditions ensures high-quality rock mass rupture with effective use of the entire face blast holes. The most effective cut options should be selected in the process of testing when implementing blasting patterns in the faces of horizontal and inclined (up to 12°) mine workings.

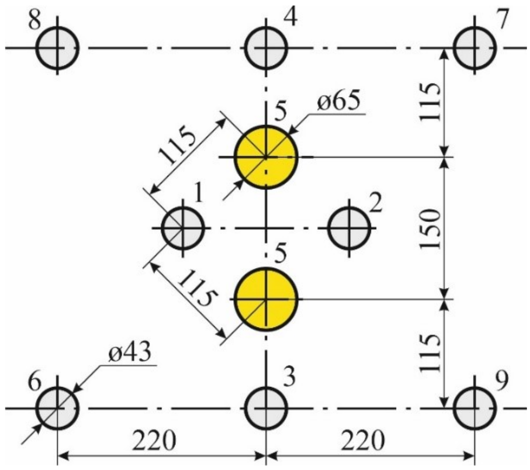

Fig. 1. Box cut with two blast holes with cleaning charges:
1, *2*, *3*, *4* – cut blast hole charges blasted in turn; 5 – cleaning charges (fifth delay); *6*, *7*, *8*, *9* – auxiliary blast hole charges

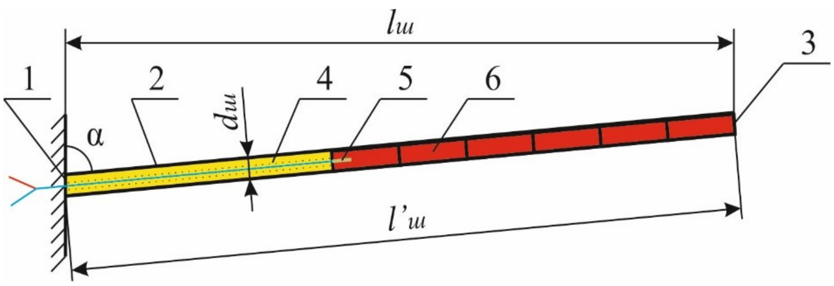

Fig. 2. Blast hole charge design:
1 – hole mouth; *2* – hole wall; *3* – hole bottom; *4* – stemming; *5* – electric detonator; *6* – explosive cartridge;
α - hole angle, degrees

Table 6

Procedure for blast hole charges blasting at a face

Blast hole designation	Numbers		Blast hole delay degree
		delays	
Cut	1 2, 3 4, 5	0 1 2	0 0.025 0.050
Auxiliary	6–9 10–11 12–15 16, 17	3 4 5 6	0.075 0.100 0.150 0.250
Outer	18–27 28–41	7 8	0.500 0.750
Peripheral holes: wall and roof; sill (snake holes); sill (corner)	 42–61 62–68 69–70	 9 10 11	 1.000 1.500 2.000

Table 7

Procedure for blast hole charges blasting at a face

Indicator	**Value**
Rock	Albitite
Protodyakonov rock hardness index, *f*	16
Fracturing	Medium
Cross-section area, m^2	13.8
Cut type	Cylinder cut
Number of blast holes, total, and their length, mm	70
perimeter holes;	2, 3800 mm long
cut;	3, 3500 mm long
auxiliary;	12, 3500 mm long
outer;	24, 3500 mm long
peripheral;	29, 3500 mm long
Perimeter hole diameter, mm	65
Blast hole diameter, mm	40
Explosive	No. 6ZhV ammonite
Blast hole charge, kg	190.4
Cleaning charge weight in perimeter holes, kg	0.6
Total explosive specific consumption per blast	191.0
Total rock blasted, m3	45.88
Blast hole utilisation factor	0.95
Explosive specific consumption, kg/m^3	4.16
Blasting method	Electric with inverse initiation
Non electric delay detonators	Primadet type

Thus, the main initial data for designing safe mine working drivage, based on which blasting pattern is developed, are as follows [27]:

– geotechnical conditions, including Protodyakonov rock hardness index, *f*, rock mass blocky structure (type of fracturing), direction of bedding, etc.);

– mine working drivage rate, determining the advance per round length (up to 3.5 m);

– type of cut depending on the advance per round length; all the developed cuts are used for the advance per round length within the range of 2.5–3.5 m and more; the cut performance with the advance per round length more than 3.5 m shall be assessed after a pilot test.

The order of face hole blasting at blasting drivage of a horizontal mine working with cross-section of 4300 × 3600 mm is presented in Table 6, and the blasting conditions and parameters are given in Table 7.

Promising areas of research

Promising research area is application of emulsion explosives and equipment sets for mechanized drilling and charging of blast holes and boreholes in the course of mine working drivage. To date, the volume and scale of introducing of environmentally friendly Ukrainit-PP-2 emulsion explosive has increased significantly, with expanding geography and scope of its application at mines of Ukraine, including: PJSC "KZhRK", PrJSC "Sukhaya Balka" (Kryvyi Rih), LLC "Vostok-Ruda" (the city of Zheltye Vody), PrJSC "ZZhRK". Besides, the first experimental charges of a blast-hole ring (at the hole diameter of 89-105 mm and length of up to 30 m) were tested, and work is underway for improving the process of preparing the explosive components.

With the support of the Gosgorpromnadzor of Ukraine and the Kryvyi Rih Mining and Technical Inspectorate, commercial tests of Ukrainit-PP-2 emulsion explosive will continue at the mines of PJSC KZhRK and SE VostGOK. Some other companies of developed mining countries [10–12, 28] are also interested in testing on application of emulsion explosives.

Conclusions

1. It was shown that blast hole charge diameter is an important parameter when driving mine workings, since it determines the explosive content in the hole, the detonation velocity and distance of its transmission, the hole drilling rate and the labor intensity of drilling operations, the quality of contouring the mine working design cross-section, and economic indicators. It is unreasonable to apply blast hole diameter of more than 40–42 mm in drifting faces. Decreasing the blast hole diameter to 36 mm increases performance of mine working drivage with blasting technique.

2. It was determined that to provide an advance per round of at least 3.3-3.5 m, in addition to high-productive self-propelled equipment, new designs of box cuts are required, reliability of which in the formation of high-quality (clean) cut cavity reaches 0.95-1.00.

3. New designs of box cuts have been developed, the peculiarity of which consists in provision of sufficient compensation (peripheral) volume (advance holes with diameters from 65 to 105 mm, charged with cleaning charges of 0.2 kg of ammonite No. 6ZhV) for rupture of trapezoidal partitions with the compensation volume factor of 2.50 to 1.34.

4. The procedure for the development of blasting pattern is recommended, which ensures high-quality mine working drivage with the advance per round of at least 3.0–3.5 m, and increasing the drivage rate from current 50 – 70m/month up to 300 m/month per a tunneling system in the near future.

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