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Study on Mining Technology of Shallow Hole + Deep Hole Shrinkage method with Ductule Grouting in Advance for Roof Reinforcement

Study on Mining Technology of Shallow Hole + Deep Hole Shrinkage method with Ductule Grouting in Advance for Roof Reinforcement

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Abstract. In this paper, the mining method of shallow-hole and deep-hole shrinkage with ductule grouting in advance for roof reinforcement is studied, which is used to recover steeply inclined thick orebody with broken roof. The main innovations in mining technology are as follows: the orebody is artificially divided into two layers, the upper layer is mined by shallow hole blasting, the lower layer is mined by parallel deep holes, both layers advance simultaneously from bottom to top, the mining face of the upper layer is 3.6 meters higher than that of the lower layer, thus forming an inverted step working face in which the mining operation and roof protection are carried out, using ductule to reinforce the broken roof by grouting in advance, and arranging bolts systematically to anchor the grouting reinforced layer into deep rock mass. By using the method of orthogonal test design and in-situ experiment, the reasonable construction parameters of ductule grouting are obtained, which have been applied to No. 7 stope of of stage 3840 in Qaza Lead-Zinc Mine in Yunnan Province and achieved good results. By monitoring the displacement of reinforced layer during and after mining, no obvious displacement and collapse are found, which proves the feasibility of this technology. The mining scheme has a good reference value for similar ore deposits with mining technical conditions.

1. Introduction

At present, non-coal underground mines mostly adopt caving method or filling method to recover steeply inclined thick orebody with broken roof. On the premise of taking measures to maintain the stability of roof, open stope mining method can be used to recover such orebody, for it is simple and economical, with high ore recovery and low dilution rate[1, 2].

The No. 3 orebody of Qaza Lead-Zinc Mine is a steeply inclined thick orebody with an inclination of 62-83 degrees and an average horizontal thickness of 12.65 meters. The orebody and surrounding rock are generally stable, and the sublevel stoping mining method has been used in the mine. But in stage 3840, an oblique fault intersects with the hanging wall, which resulting in a fracture zone with a length of about 1400 m and height of 18 m to 34 m in the roof. The roof often collapses in the mining process, and the ore grade of Pb+Zn is only 3.21%, which seriously affects the economic benefits of enterprises. Therefore , the method of "shallow hole + deep hole shrinkage with ductule grouting in advance " has been designed to solve this problem.

2. Mining method

2.1 Introduction to the Mining Method

The mining method is shown in Figure 1, where (a) the cross section in the middle of the stope and (b) the cross section of pillars between stopes.
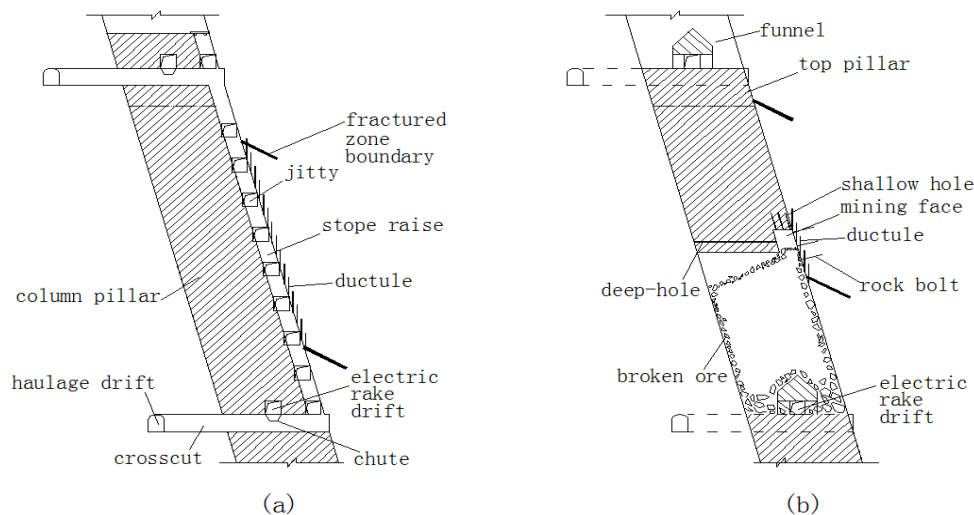

Figure 1. Cross section of stope.

The strike length of the stope is 50 m, in which the room length is 42 m, the pillar width between stopes is 8 m, and the stage pillars are 8 m high. The bottom structure is electric raking trench.

The ore body in the room is artificially divided into two layers along the thickness direction, and the working face is promoted from bottom to top at the same time. The upper layer is 2.5 m thick, stoped by shallow hole blasting. The lower layer is stoped by horizontal parallel deep hole blasting. The upper layer advances about 3.6 meters higher than the lower one, forming a stepped mining face. The working face is under the upper layer, 3 m high, in which roof protection, stoping operation are carried out. The mining operation cycle is shown in figure 2.

2.2 Ore recovery

2.2.1 Drilling. The shallow holes in the upper layer are drilled upwardly by YSP-45 driller, with the hole depth 2.0 m, hole diameter 38 mm, hole spacing 1.0 m, burden ranges from 0.8 m to 1.0 m. The V cuts are located at the midsection of the slice. The deep holes in the lower layer are drilled horizontally by YGZ-90 driller, with the hole diameter 90 mm, hole spacing 2.0 m, and burden 1.8 m. Each deep hole is parallel to each other.

2.2.2 Blasting. The detonating sequence of shallow holes is from the middle to both sides row by row, with the inter-row delay time 25 ms, and the V cuts holes are detonated firstly. Only one row of deep holes is detonated at a time, initiating simultaneous with the shallow holes but detonating 50 ms later than the shallow holes of final detonation. Of course, deep holes can be detonated one by one with a inter-hole delay time 25 ms in order to reduce blasting vibration.

2.2.3 Ventilation. JK55-2NO4.0 local ventilators are used to strengthen ventilation after blasting. After each blasting, the time of ventilation is not less than 30 minutes[3].

2.2.4 Ore drawing. After ventilation, about one third of the broken ore quantity of each blasting is released from the bottom of the stope. The amount of ore drawing per time depends on the expansion ratio of ore, the criterion of judgment is that keep the height of the working face 3.0 m. The remaining ore will be all released after completion of mining [4].

3. Ductule grouting in advance
In order to ensure the safety of working face and maintain the roof stability, ductule grouting technology is used to reinforce the hanging wall.

3.1 Layout of grouting holes
In the working face, the grouting holes are drilled by YT28 rock driller at an outward angle of 15 to 20 degrees along the ore-rock contact line of the upper wall.

The grouting holes patterns are staggered (Figure 3). The holes in the same row have the same depth and collar elevation, and each hole is parallel with others. The row plane is neat and parallel with adjacent rows, and the overlap length is not less than 1 meter along the length direction.
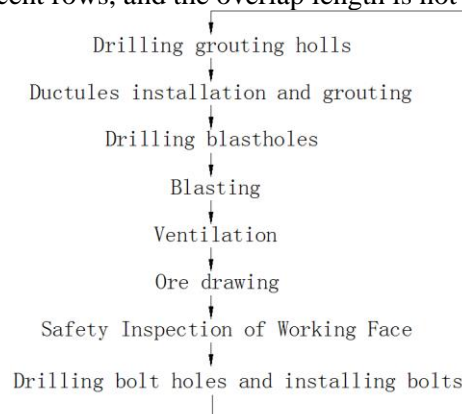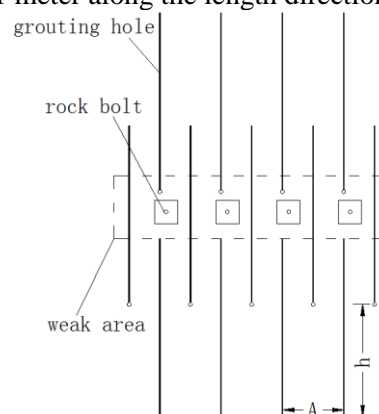

Figure 2. Mining operation cycle.

Figure 3. Layout of grouting holes.

3.2 Grouting experiment
In addition to the grouting diffusion radius, other grouting hole network parameters, such as hole spacing, row spacing and hole-collar spacing can be calculated by formulas. The grout diffusion radius is the most critical parameter, and there are many factors affecting it. This project uses the method of orthogonal experiment design and obtains the reasonable value of this factor through in-situ experiment. In the design of orthogonal experiment, some important factors such as water cement ratio, final grouting pressure and grouting aperture are considered. The design is shown in Table 1.

A total of 72 test holes were drilled, i.e. three in-situ tests were conducted under each group of orthogonal test conditions. The grouting volume of each hole was recorded during grouting. The final grouting pressure reached the design pressure and then stopped. After 48 hours, the grouting diffusion radius was measured by RSM-YS5 ultrasonic detector. The results recorded in Table 1 are the average of three holes. According to the test results, the reasonable grouting parameters are: water cement ratio 1:1, grouting aperture 52mm, final grouting pressure 0.6Mpa.

3.3 Grouting construction
During construction, the hole space is 0.75 m, the collar space is 1.8 m, the depth of grouting holes is 3 m, the angle of insertion is 18 degrees, the water cement ratio was 1:1, the grouting hole diameter is 52 mm, and final grouting pressure 0.6Mpa. The ductule is made of hot-rolled seamless steel pipe with an outer diameter of 42mm and a thickness of 3.5mm. The front end of the ductule is tapered, and the end of the ductule is welded with a stiffening hoop of 6 mm. The little hole with diameter of 8 mm is drilled around the ductule. There are four holes in each row with a row spacing of 0.3m. There are no grouting holes in the tail 0.5m of the ductule. The length of a single tube is 3.0m.

The ductules are plunged into the holes manually, and the jump hole(interval) grouting is used when grouting. If the hanging rock is rather broken, or severely weathered, or the water inflow is large, the method of grouting while drilling can be used. The initial pressure of grouting is controlled at 0.3 MPa and the grouting speed is controlled within 30 L/min to stop when the design grouting quantity is reached. If the pressure rises that the collar pressure has reached 0.6 MPa, or the flow rate decreases

rapidly, grouting operation should be finished although the grouting volume does not reach the calculated value.

Table 1. Orthogonal experimental parameters and in-situ experimental results.

Orthogonal experimental parameters			In-situ experimental results	
water cement ratio	final grouting pressure (Mpa)	grouting aperture (mm)	grout diffusion radius (m)	Average grouting volume of single hole (m^3)
0.8	0.5	45	0.44	0.073
		52	0.49	0.091
	0.6	45	0.51	0.098
		52	0.56	0.118
	0.7	45	0.55	0.113
		52	0.59	0.131
	08.	45	0.57	0.122
		52	0.61	0.140
1.0	0.5	45	0.55	0.113
		52	0.59	0.131
	0.6	45	0.61	0.144
		52	0.65	0.156
	0.7	45	0.65	0.155
		52	0.68	0.166
	08.	45	0.67	0.167
		52	0.71	0.182
1.2	0.5	45	0.65	0.155
		52	0.70	0.181
	0.6	45	0.68	0.173
		52	0.72	0.190
	0.7	45	0.70	0.181
		52	0.74	0.196
	0.8	45	0.73	0.199
		52	0.77	0.201

3.4 Further reinforcement of bolts

In order to strengthen the connection between grouting body and deep rock mass and improve the overall stability of roof, full-length friction bolts are arranged systematically in the weak grouting area as is shown in figure 3. The bolt insertion direction is perpendicular to the ductule. The bolt is arranged in the weak area at the middle of the two grouting holes, the bolt hole space is 0.75m and the row space is 1.8m. The bolt length is calculated as three times the minimum thickness of grouting overlap, which is 2.4m.

4. Roof deformation monitoring

The deformation of the newly exposed roof (broken zone after grouting support) in the test stope was monitored in time, and the monitoring data were analyzed in order to grasp the reinforcement effect of grouting on broken roof.

4.1 Monitoring method

Commonly used monitoring methods of roof stability in goaf include manual inspection, borehole method, multi-point displacement meter, cross-section convergence measurement, leveling measurement, pressure gauge, acoustic emission, microseism and other monitoring and testing technologies[5,6]. This project adopts the conventional displacement monitoring method with the advantages of simple operation, intuitive data and low cost[7,8]. LBY roof separator is used for displacement monitoring.

Monitoring points are located in the stress concentration area of the stope roof, that is, the main stress area in the new balancing process[9]. The exposed area of roof fracture zone in test stope is 42 m in length and 3 m in height. There are nine displacement observation points in three groups. The first group is located at the left end of the exposed area, 6 m away from the left edge, the second group is located in the middle of the exposed area, the third group is located at the right end of the exposed area, 6 m away from the right edge, and the three observation points in each group are arranged vertically with a distance of 1.2 m. Observation begins after the observation points are set up. Observations are made regularly twice a day and the observation time points are fixed. Records are made and data are analyzed in time. Observations stop two days after the roof deformation tends to terminate.

4.2 Analysis of monitoring results

There are three observation points in each group. Each observation point records data twice a day. In order to avoid the influence of single data error, the average of 6 data per group per day is calculated. This observation lasts for 7 consecutive days. The results are shown in Table 2 and Figure 4. From Table 2, it can be seen that the deformation of the middle part of the roof is higher than that of the two ends, and the time lag for the stability of the displacement is about one day behind the end, but generally it tends to be stable on the 4th to 5th day after exposure (the 5th to 6th day after grouting).

Table 2. Roof deformation (mm).

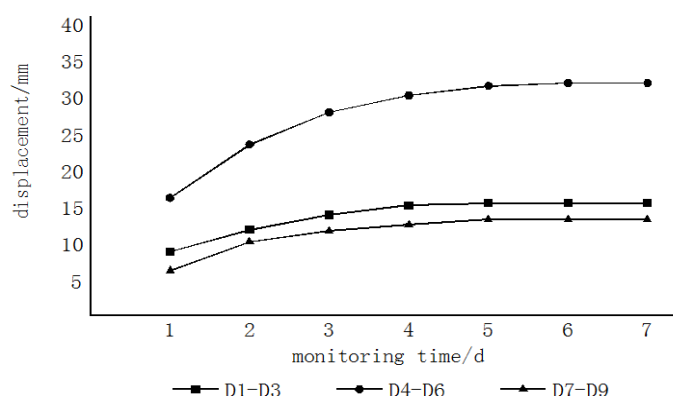| Observation point	Observation time (d)						
	1	2	3	4	5	6	7
D1-D3	8.7	11.6	13.5	14.8	14.9	15.0	15.0
D4-D6	16.0	23.2	27.7	29.8	30.9	30.1	30.1
D7-D9	6.2	9.9	11.4	12.5	13.1	13.2	13.2

Figure 4. Monitoring data of roof separation indicator.

From the analysis of Figure 4, it can be seen that the roof displacement has convergence, which proves that the ductule advanced grouting technology is effective for the reinforcement of broken roof. In addition, the test stope has been mined out for 41 days. After the stope is mined out for 7 days, no

obvious roof deformation and collapse have been found. The test stope has achieved ideal results, which further proves the feasibility of this technology.

5. Technology and Economic Assessment

The test stope roof breakage zone is 42m long, 28m high, cement consumption is 193.8t, ductules are 969, bolts are 896 sets, boreholes drilling are 5147m, and the total cost of roof maintenance is 1869,000 yuan. The average ore grade of Pb+Zn is 4.33%, which increases by 1.12%. It is estimated that the cost of beneficiation will be reduced by about 1.2296 million yuan.

The comprehensive technical and economic indicators of the test stope are shown in Table 3. Compared with the original mining method, the 1000-ton mining-cutting ratio is reduced by 2.22 m/kt, the comprehensive recovery rate is increased by 5.7%, the ore grade is increased by 1.12%, the mining efficiency is increased by 3.4 t/class, the production capacity is increased by 60 t/d, the block rate is decreased by 4.3%, and the direct mining cost is increased by 2.4 yuan/t.

Table 3. Comparison of comprehensive technological and economic indexes.

Mining method	1000-ton cutting ratio $(m\ kt^{-1})$	Comprehensive recovery rate (%)	Comprehensive ore grade (%)	Mining efficiency $(t\ class^{-1})$	Actual production capacity $(t\ d^{-1})$	Bulk percentage (%)	Mining direct cost $(yuan\ t^{-1})$
New method	3.12	88.3	4.33	24.8	580	2.1	54.5
Original method	5.24	82.6	3.21	21.4	520	6.4	52.1

6. Epilogue

This paper studies the mining technology of shallow hole + deep hole shrinkage method with ductule grouting the roof in advance to mining steeply inclined thick orebody with broken roof. The successful test has effectively solved the difficult problem of roof maintenance in the mining process of steeply inclined thick orebody with partial broken wall rock. Compared with the original mining scheme, it reduces the amount of mining and cutting work, and the rate of waste rock mixing, as well as improves the ore recovery rate and ore grade. Enterprises get better economic returns. The mining scheme has good application prospects and popularization value for mines with similar mining conditions.

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