Research on the control technology of heavy layer hard roof blasting

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Abstract: In this paper, based on the typical geological features of heavy layer hard roof, a round a blasting weakening control of complete sets of technology, through the theoretical calculation and numerical simulation, analyses the rules of the thick layer of hard roof and the characteristic of mine pressure appear; Based on the theoretical analysis, the attenuation height of vertical blasting and the circular blasting distance are determined. By weakening roof blasting scheme design, pull cut blasting with two lane ahead of deep hole blasting weakening technology, this paper introduces hole arrangement, drilling tool, design parameters control and instrument. A complete set of technical measures to reduce the control of hard roof blasting in thick layer is formed, and the rationality of the technical measures is confirmed by field observation. The relevant research results have important practical significance and reference value to the hard roof management of similar geological conditions.

1. Introduction
Many researches have been carried out on the control of thick hard roof [1-5], especially on the management measures of blasting weakening roof, and many research results have been obtained [6-8]. High-yield and efficient work faces puts forward higher requirements. In this paper, combined with the typical geological characteristics of heavy hard roof, the characteristics of rock pressure and weighing step of heavy hard roof are analyzed. According to the engineering practice, based on the analysis of the weakening scheme of thick hard roof and the theoretical calculation of the control area, the weakening technology of cut-off slot blasting and two-lane advanced deep-hole blasting is organically combined. Emphasis is placed on the analysis of the technical parameters such as the height of roof treatment, the arrangement of boreholes, drilling design parameters, etc. The scientific rationality of the above technical measures is confirmed by field observation.

2. Analysis of Mine Pressure Behavior in Working Face

2.1 Engineering Survey and Rock Mechanics Parameters
The main coal of a mine is 29 coal, with an average thickness of 4.43m and partial parting 1 layer. Mechanical parameters of of coal seam roof and floor is listed in Table 1.

2.2 Numerical Simulation
According to theoretical calculation, the basic top breaking distance is 61.3m and the periodic breaking distance is 32.3m. The initial weighting equivalent Pe of basic roof is 1290 kPa. The weighting is very strong. The roof should be weakened in order to avoid the dynamic pressure.
Table 1. Mechanical parameters of each rock stratum of coal seam roof and floor

| Rock stratum       | h/m | σh/MPa | σv/MPa | E/GPa | μ   | σc/MPa | ϕ/° | ρ/kg/m³ |
|--------------------|-----|--------|--------|-------|-----|--------|-----|--------|
| Medium sand        | 8.1 | 87.7   | 8.80   | 25.12 | -   | -      | -   | 2.50   |
| Shale              | 2.32| 33.60  | 2.10   | 12.35 | -   | -      | -   | 2.20   |
| Sandy mudstone     | 3.75| 38.70  | 1.60   | 13.07 | -   | -      | -   | 2.20   |
| Mudstone           | 2.24| 25.70  | 2.00   | 11.14 | -   | -      | -   | 2.30   |
| Medium sandstone   | 10.31| 91.70 | 9.10   | 25.25 | 0.25| -      | -   | 2.50   |
| Siltstone          | 2.51| 49.60  | 4.88   | -     | 0.23| -      | -   |        |
| Shale              | 0.20| 32.80  | 2.05   | -     | -   | -      | -   |        |
| 2nd coal           | 4.43| 19.50  | 0.55   | 6.16  | 0.33| 2.12   | 33.10| 1.31   |
| Sandy mudstone     | 4.15| 30.56  | 5.00   | 27.27 | 0.27| 10.20  | 39.40| 2.71   |
| Argilaceous sandstone | 18.68 | 35.53 | 7.84   | 30.33 | 0.31| 18.40  | 34.20| 2.54   |

2.2.1 Model establishment and failure criteria

The thickness of coal seam is 4.43m, the basic top-middle sandstone is 10.31m and the direct top siltstone is 2.51m. The simulated mining depth is 230m, the thickness of roof is 90m, and the overlying strata are replaced by equivalent load. The width of the model is 240m, including 48m left on each side, the kerf width is 7.6m, and the simulated advancing distance is 136m. Mohr-Coulomb plastic constitutive model is used for the simulation.

2.2.2 Numerical simulation results analysis

Numerical simulation results is shown in Figure 1. The numerical simulation results show: when advancing for 16m, the peak value of advanced bearing pressure is about 6.72MPa, the stress concentration coefficient is about 1.40, and the direct roof caving for the first time; When it is estimated to be 28m, the peak value of the advance abutment pressure is about 7.43 MPa and the stress concentration factor is about 1.55. It is speculated that the free collapse of the basic roof may cause impact. When pushed to 38m, the basic roof can still remain intact. When pushed to 56m, the basic roof completely collapsed and the roof weighting was strong.

![Figure 1. Roof caving of working face with different advancing distance](image_url)
stress concentration factor is 1.47, and the suspension distance should not exceed 22m.

### 2.3 Analysis of Strata Behavior Characteristics of Hard Roof
Due to the high strength and good integrity of the hard roof engineering mechanical properties, decided to face mine pressure behavior characteristics basically include: ①Thick and hard roof weighting has obvious time difference and step difference. ②The periodic breaking has large step distance and high dynamic load coefficient.

### 3. Blasting Weakening Scheme and Its Implementation Effect
Advanced deep-hole presplitting blasting in two lanes or special lanes (process lanes) can realize parallel operation of production and roof management, which is beneficial to high yield and high efficiency of the working face, and is simple and feasible in construction.

#### 3.1 Basic Top Weakening Control Area Analysis
According to the production needs of the working face, the basic top weakening treatment should meet two indicators: First, the basic top can be timely degraded and filled in the goaf, that is, it must maintain a certain weakening height in the vertical direction of the coal seam; The second is to prevent the occurrence of the phenomenon of working face pressure frame. In order to fully fill the mined-out area after the basic caving, the effective caving height $h$ value of blasting can be calculated according to formula (1):

$$H = M/(K_p - 1)$$  \hspace{1cm} (1)

Where: $H$ is the effective blasting height of the blasting, m; $M$ is the mining height, m; $K_p$ is the volumetric expansion coefficient after the rock is broken, 1.3.

According to the roof classification (industry standard of the Ministry of Coal Industry), there are:

$$L_p \leq \frac{P_z + 10.24N + 62.1 - 72.3h_m - 78.9B_c}{4.5}$$  \hspace{1cm} (2)

Where: $L_p$ is the cycle step, m; $P_z$ is the lower limit of the rated support resistance, the value is 810kN/m²; $h_m$ is the mining height of coal seam, with a value of 4.32m; $B_c$ is the top control width, with a value of 5.75m; $N$ is the direct top filling coefficient, $N=h/h_m=0.58$.

The calculation shows that the effective top-caving height of blasting is $H=14.4$m and the interval of periodic weighting step $L_p$ is 24.9m. The blasting weakening vertical height of the working face shall not be less than 14.4m, and the cyclic blasting step distance shall not be more than 25 m.

#### 3.2 Design of Roof Blasting Weakening at Cut
ZL-100 drill and 60mm drill are used to construct hole 1 with a hole depth of 16m, and bolt drill and 42mm drill are used to drill holes 2 and 3 with a hole depth of 9m and 5m respectively. Each group of holes is 3.0m apart, with two holes and one hole. The arrangement of the holes is shown in Figure 2.

![Figure 2. Schematic diagram of opening cut hole blasthole arrangement](image)
3.3 Weakening Design of Advanced Blasting in Two Roadways

1) Borehole layout and borehole design

The arrangement of blasting holes for deep-hole blasting in two roadways with hard roof is mainly divided into two parts: one is to cut off the connection between the upper part of roadway wall and the roof; the other is to cut off the connection between the roof and the roof. A group of blast holes are arranged at intervals of 20m in the transportation drift and the return air drift, each group is divided into upper and lower layers, each layer has 4 blast holes, the vertical distance between the upper blast hole and the coal seam is 15m, the vertical distance between the lower blast hole and the coal seam is 10m, and the diameter of the blast hole is 60mm. Advanced drilling is shown in Figure 3, and borehole parameters are listed in Table 2.

![Diagram](image_url)

Figure 3. Arrangement of advance boreholes in two roadways

Table 2. Blasthole parameters of roof caving with step spacing in two roadways

| Blasthole number | length/m | Elevation/° | Horizontal angle/° | Aperture/mm |
|------------------|----------|-------------|--------------------|-------------|
| A₁               | 30.2     | 38.9        | 22                 | 60          |
| A₂               | 30.2     | 38.9        | 22                 | 60          |
| B₁               | 32.3     | 36.6        | 34                 | 60          |
| B₂               | 30.3     | 28.3        | 34                 | 60          |
| C₁               | 49.5     | 30.2        | 71                 | 60          |
| C₂               | 48.2     | 24.7        | 71                 | 60          |
| D₁               | 49.5     | 31.1        | 78                 | 60          |
| D₂               | 48.2     | 25.5        | 78                 | 60          |

2) Advanced Deep Hole Blasting Position of Two Roadways

Pre-splitting time of the first group of deep-hole blasting: the optimal time should be in front of the cutting face. Pre-splitting time of deep-hole blasting in subsequent groups: The time or location of pre-splitting roof drilling blasting should be about 35m from the hole to the working face. When the hole bottom position is at least 10m away from the horizontal distance of the coal wall, start charging and detonate when it is 9m away from the coal wall.

3.4 Investigation of Implementation Effect

A CDW-60 bracket resistance recorder is installed on the front and rear pillars of 9 hydraulic supports in 9°, 18°, 27°, 36°, 45°, 54°, 63°, 72°, 81° face to monitor the roof weighting during the advancing process of the working face. The first borehole stress meter is arranged 45m away from the cutting hole, with an interval of 5m; The longest drilling depth is 12m and the shortest is 5m. Two roadway deformation observation points are respectively arranged at 55m and 80m ahead of the two roadways.

3.4.1 Initial Collapse of Direct Roof

According to the change time history of support resistance and field observation, the initial caving step distance of direct roof is 11.3m, the maximum average support resistance is 5372kN, and the average weighted resistance is 4128kN, so the weighting is not obvious. After the first collapse of the direct roof, the collapse body is broken with mining.
3.4.2 Basic Top Pressure Situation
When the working face is pushed to 34.79m, the basic top will be pressed for the first time, and the maximum average resistance will be 6344kN. The average step distance of the first cycle weighting is 23.59 m, the maximum average resistance is 6931 kN, the average step distance of the second cycle weighting is 18 m, the maximum average pressure is 6273 kN. The average step distance of the sixth periodic weighting is 23.44m, the maximum resistance is 7267kN.

Deep hole blasting effectively increases the fracture degree and fracture development degree of direct roof and basic roof, and alleviates the weighting strength. Blasting forces the direct roof to collapse with mining, which reduces the potential energy conversion during roof collapse and the dynamic load coefficient during weighting.

3.4.3 Pressure Distribution of Working Face
The average weighted resistance of frame 72 is 3258kN, the maximum average working resistance of frame 54 is 6085kN, and the maximum average resistance of frame 9 is 4052kN. The advance blasting of two roadways within 40m reduces the pressure step and weakens the pressure intensity. Within 50m from the middle of the working face, there is no blasting weakening, and the step distance and strength are larger than those at both ends, as shown in Figure 4.

The abscissa coordinate is different from the depth perpendicular to the coal wall, and the longitudinal coordinate is coal stress, as shown in Figure 5. The peak stress of return air roadway is at 10m, the pressure is 6.89MPa, the peak stress of transportation roadway is at 6m, the pressure is 10.6MPa.

3.4.4 Support and Convergence of Section in Two Roadways
The deformation of the two roadways is relatively small during mining, with the roof and floor moving closer to about 100mm and the two sides moving closer to about 110 mm. There is little difference between the two sides moving closer than the top and bottom plates. The main reason is that the mining depth is small, the stress of the original rock is small, the surrounding rock condition of the mining coal seam is good, the two sides of the roadway and the coal quality are hard, and the loosening circle of the two sides of the roadway is small. At the same time, due to the adoption of the above roof weakening measures, the mine pressure behavior strength of the working face and roadway has been obviously improved, as shown in Figure 6.
4. Conclusion
1) The simplified calculation results of beam type are as follows: The initial caving interval is about 61.3m, the periodic caving interval is about 32.3m, the initial weighting equivalent $P_e$ is 1290kPa, it is very strong.

2) The calculated vertical height of blasting weakening should not be less than 14.4m, and the step distance of cyclic blasting should not be more than 25m. This method has been applied to the regional planning of the weakening of thick and hard roof, and the effect is good.

3) The roof weighting is relatively mild in the blasting weakening coverage area, and the weighting is relatively strong in the middle area where the blasting weakening coverage is not available. The increase of cyclic blasting step has less influence on the cyclic weighting step, but has greater influence on the weighting performance of the working face.

References
[1] Msai, Tao Guangmei, Ma Jinqin. (2016) Application of directional hydraulic fracturing technology in hard roof of Jincheng mining area. Mining safety and environmental protection, (43) 2:84-86.

[2] Wang Tuo, Chang Jucai, Zhang Bing, Su Yafeng. (2017) Fracture characteristics and safety control of multi-layer hard roof in fully mechanized mining face. Journal of Underground Space and Engineering, (13) 1:339-343.

[3] Zhu Zhijie, Wang Hongkai, Zhang Hongwei, Lan Tianwei, Gaoming. (2017) Study on the law of rock pressure and control technology of multi-layer hard roof fully mechanized caving mining. Coal Science and Technology, (45) 7:1-6.

[4] Zhu Jiasheng, Zhang Zhaowei. (2018) Technical analysis of first caving in Bulianta Mine based on hydraulic fracturing method. Coal Mine Safety, (49) 2:77-80.

[5] Huang Bingxiang, Zhao Xinglong, Chen Shuliang, Liu Jiangwei. (2017) Hard roof hydraulic fracturing control theory and complete set technology. Journal of Rock Mechanics and Engineering, (36) 12: 2954-2970.

[6] Yang Juncai. (2017) High pressure water pre-cracking forced topping technology applied in Shandong mining area. Coal Mine Safety, (48) S1: 63-68.

[7] Wang Jinxin. (2018) Experimental study on deep hole pre-splitting blasting in the control of hard roof in fully mechanized caving face. Coal Mine Safety, (49) 1: 73-75.

[8] Jiao Zhenhua, Wang Hao, Lu Zhiguo, Zhang Bo. (2017) Deep hole pre-splitting blasting technology for thick hard limestone roof. Coal Science and Technology, (45) 2: 21-26.