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A Study on the Characteristics of Complex Overhanging Roof Structures on the Go-Ahead Side in High-Intensity Mining and Roof Control: A Case Study of the 12-Shang-302 Working Face

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Title: A Study on the Characteristics of Complex Overhanging Roof Structures on the Go-Ahead Side in High-Intensity Mining and Roof Control: A Case Study of the 12-Shang-302 Working Face

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A Study on the Characteristics of Complex Overhanging Roof Structures on the Go-Ahead Side in High-Intensity Mining and Roof Control: A Case Study of the 12-Shang-302 Working Face

Abstract

In the Shendong Mining Area, under high-intensity mining conditions in shallow coal seams, the problem of intense mining pressure in open-face roadways caused by large-scale overhangs in thick, hard roof strata has become increasingly prominent. Taking the 12-Upper 302 working face at the Guoneng Shendong Burtay Coal Mine as the engineering context, this paper systematically investigates the structural characteristics of thick, hard roof overhangs on the open-face side and pressure relief control technologies by comprehensively applying on-site core sampling and testing, theoretical analysis of key strata, multi-point displacement monitoring, and roof pre-fracturing blasting tests. The results show that within a 40-meter range above Coal Seam 12 in the study area, there is one main critical layer and seven subcritical layers. Among these, the lower subcritical layer consists of coarse-grained sandstone with a thickness of 12.04 meters; it is located within a collapse zone with a height of approximately 16.8 meters. The overlying strata exhibit a “dual-zone” distribution comprising a collapse zone and a fracture zone, with the fracture zone reaching a height of approximately 54.6 meters. Multi-point displacement monitoring revealed that displacement changes were greatest at the lower measurement points and gradually decreased as the vertical height increased, indicating that the suspended roof structure is dominated by the critical lower layer. Fractures propagate in the direction of movement through a bottom-up layered transmission mechanism, while the direction of inclination forms a stepped lateral cantilever beam structure. The cantilevers exhibit periodic growth-fracture cycles of 8–12 m, and after fracturing, they transform into a masonry beam structure. Based on this masonry beam structure, this study elucidates the disaster-causing mechanism resulting from the coupling of static and dynamic loads in multi-key-layer composite suspended roofs. A long cantilever extending from a low-lying critical layer in a subsidence zone causes long-term concentration of high static loads on the coal pillar, while sequential fracturing of multiple subcritical layers in a fracture zone generates impact dynamic loads; the superposition of these two forces, transmitted through the narrow coal pillar, induces instability in the surrounding rock of the tunnel. Based on the characteristics of surrounding rock instability and failure, a zoned, graded, and targeted pre-fracturing stress relief plan is proposed. By employing a three-tiered drilling and fracturing method—low, medium, and high—the load is applied to the key controlling layer in the collapse zone and to sub-key layers at different levels within the fracture zone. This approach eliminates static loads at the roof and distributes dynamic loads, thereby achieving a layered, staged collapse of the roof. The study revealed the structural characteristics of the roof—characterized by “multiple key layers overlapping, low-level dominance, and a stepped cantilever”—in the goaf side under high-intensity mining, as well as the mechanisms underlying the resulting disasters. The proposed zoned and graded stress-relief plan can serve as a reference for the prevention and control of high mining pressure in goaf tunnels in mines with similar conditions.

Keywords: high intensity mining ; thick and hard roof ; suspended roof on the airside ; key stratum theory ; multi-base point monitoring ; pre-splitting blasting pressure relief

1 Introduction

The development of coal resources in western China is exemplified by the Shendong Mining Area, which is characterized by shallow burial depths, thin bedrock, and thick coal seams. Longwall mining faces generally employ full-length mining technology with high working heights, with

annual production per face reaching the 10-million-metric-ton level, thereby forming a “high-intensity mining” model [1]. However, under high-intensity mining conditions, the overlying rock exhibits extensive movement, rapid fracturing, and significant mine pressure. This is particularly true when the roof above a coal seam consists of multiple layers of thick, hard sandstone; in such cases, the hard roof cannot be caved as mining progresses, resulting in the formation of large-scale suspended roof structures on the sides of the goaf. Suspended roof strata transfer the overlying load through key strata to the coal pillars and the open-face roadways in the mining section, leading to frequent dynamic phenomena such as roof subsidence, floor heaving, and rib spalling, which pose a serious threat to safe production [2–5].

The key-layer theory proposed by Academician Qian Minggao provides a theoretical framework for understanding the movement patterns of overlying rock [6]. This theory posits that coal-bearing strata contain one or more relatively hard, thick rock layers (key layers), whose fracturing controls the movement and deformation of all or part of the overlying strata. Before fracturing, the key layers bear the overlying load as “plate” or “beam” structures; after fracturing, they form a masonry-beam structure, and their “S-R” stability conditions determine the intensity of mining pressure in the working face [7]. Under conditions involving multiple key strata, the failure sequence of each sub-key stratum and the load transfer paths directly affect the stress environment in the adjacent tunnel. Regarding the hazard posed by suspended roofs in hard rock, scholars both domestically and internationally have conducted extensive research [8–12]. In terms of the mechanical analysis of suspended roof structures, Geng Ming et al. [13] established a mechanical model for thick, hard lateral suspended roofs and determined that the optimal cutting position is 3.98 m into the inner coal pillar; Zhang Xin et al. [14] constructed a mechanical model for thick, hard roof strata in coal pillar-free working faces and derived expressions for the step distances of initial and periodic pressure waves. In terms of pressure relief control technology, roof pre-fracturing—which shortens the overhang length and reduces the applied pressure by creating predetermined weak planes—is the most widely used engineering method [15–21]; directional hydraulic fracturing technology has been successfully applied in mines such as the Burttai Coal Mine in the Shendong Mining Area, where it can limit the overhang area to within 30 m² and reduce mine pressure by approximately 17% [22,23]. In terms of monitoring technology, Zhu Weibing et al.[24] conducted a systematic review of the development of in-situ monitoring techniques for rock displacement within overlying strata affected by mining operations, pointing out that multidimensional real-time collaborative monitoring is an effective means of revealing the patterns of rock mass movement.

However, most of the aforementioned studies have focused on the overall movement characteristics of the overlying rock in the direction of advance, with insufficient attention paid to the layered movement patterns of the laterally overhanging roof structure at the edges of the goaf. Existing research has two main limitations. First, monitoring of overhanging roof structures is primarily based on strike-direction cross-sections; there is a lack of in-situ measurement data to support the vertical stratified displacement characteristics of the overlying rock along the lateral goaf, making it difficult to accurately characterize the spatial morphology and evolutionary process of overhanging roof structures. Second, most pressure relief control schemes are designed for single critical layers or uniformly layered roof structures, and there is a lack of systematic design criteria for layered pressure relief control under conditions involving multiple superimposed critical layers [25–29]. For complex scenarios involving multiple layers of thick, hard roof strata occurring in a

stacked configuration and developing in a stepped, laterally cantilevered manner under high-intensity mining conditions, there is an urgent need to establish a systematic research approach encompassing “detailed structural characterization—elucidation of failure mechanisms—layered, targeted stress relief.”

Based on the above analysis, this paper takes the 12-Shang-302 working face at the Bo'ertai Coal Mine as its engineering context and, through core drilling and mechanical testing, investigates the distribution of key strata in the overlying rock and their spatial coupling relationship with the “two zones.” Using multi-point displacement monitoring stations, we simultaneously recorded the layered displacements of the collapse zone and the fracture zone, revealing the stepped structural characteristics and evolutionary patterns of the laterally overhanging roof. Based on key-layer theory analysis, multi-point displacement monitoring, and engineering tests on roof pre-cracking blasting, this study explores the failure mechanisms caused by the coupling of static and dynamic loads in overlying roofs with multiple key layers. A zoned, graded, and targeted pre-cracking pressure-relief scheme is proposed, with the aim of providing theoretical and engineering references for the prevention and control of high mine pressure in open-face tunnels under similar conditions.

2 Project Background

2.1 Overview of the Work Surface

The upper 302 working face of 12 is located in the third panel of Buertai Coal Mine. The inclined length of the working face is 310 m, the strike mining length is 4674.6 m, and the mining area is 1.4491 million m^2. The average mining height of the coal seam thickness is 4.9 m, the dip angle of the coal seam is 1-5 °, which is a near-horizontal coal seam, and the buried depth is between 380 m and 450 m. The working face is arranged along the strike, advancing along the tendency, and all the goafs are treated by the caving method. The east side of the working face is the goaf of the first mining face of 12 upper 301. The working face is left with a 20 m wide section coal pillar. The 12 upper 302 auxiliary transport crossheading is adjacent to the 12 upper 301 goaf, and the 12 upper 303 auxiliary transport crossheading is the next working face crossheading. The relative position of the working face and the layout of the roadway are shown in Figure 1.

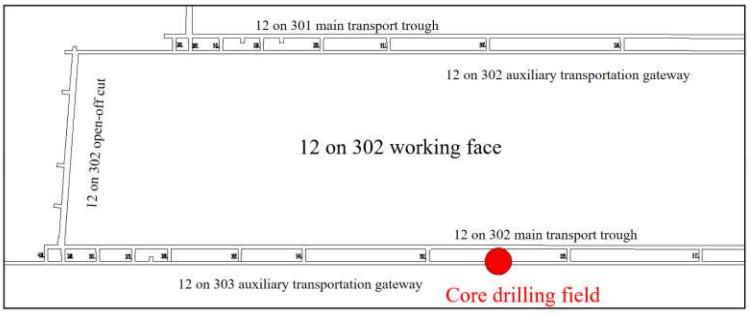


Figure 1 Relative position of working face and roadway layout diagram

2.2 Occurrence characteristics of rock strata

The immediate roof of the upper 302 working face of 12 is mainly composed of siltstone with a thickness of 3.27 m, a coal seam with a thickness of 0.52 m, and a coarse-grained sandstone with a thickness of 12.04 m. The coarse-grained sandstone of the basic roof is gray-white, argillaceous cementation, local calcareous cementation, thick bedded, mainly composed of quartz and feldspar, with high strength and good integrity. The direct bottom is 1.42 m siltstone. The characteristics of coal seam roof and floor are shown in Table 1.

Table 1 coal seam roof and floor characteristics

Top layer	Thickness/m	Rock name	Lithological characteristics
main roof	12.04	kern stone	Gray white, argillaceous cementation, local calcareous cementation, thick layer, mainly quartz and feldspar.
immediate roof	0.52	coal	Black, mainly dark coal, containing silk charcoal, semi-dark briquette.
immediate roof	3.27	siltstone	Light gray, argillaceous cementation, thin layer of fine sandstone, nearly horizontal bedding development.
direct bottom	1.42	siltstone	Light gray, argillaceous cementation, thin layer of fine sandstone

2.3 Characteristics of mine pressure appearance

In the mining process of 12 upper coal seam, the mine pressure is relatively strong. The 12 upper 303 auxiliary transportation roadway and the 12 upper 302 auxiliary transportation roadway both have dynamic phenomena such as roadway roof subsidence and floor heave. The analysis shows that the main controlling factors of mine pressure include thick roof, mining depth, ground stress direction and section coal pillar [30]. The buried depth of 12 upper 302 working face is more than 380 m in the range of about 1600 m from the open-off cut. When the working face is mined to the overlying area with thick and hard rock strata, the static load and dynamic load formed by the fracture of multi-thick roof are transmitted to the gob-side entry. When the total stress of surrounding rock exceeds the resistance and failure ability of coal-rock mass and supporting structure, the roadway has significant deformation and failure. The thick hard roof can maintain a large area of hanging without caving, the suspended area is large, and the advanced stress concentration degree of the working face is high ; when the suspended roof reaches the limit, it breaks in a large area, causing strong vibration of coal and rock mass, inducing floor heave and large-scale coal wall failure.

3 Discrimination of key strata and causes of suspended roof structure

3.1 Coring and rock mechanics test

The shunting chamber between 33 # joint roadway and 32 # joint roadway in 12 shang 303 auxiliary transport crossheading is selected as the sampling position. The borehole is designed with an inclination angle of 59 °, a length of 47 m, a diameter of 75 mm, and a 1 m coring drill pipe. The qualified cores (single section length $\geqslant$ 150 mm) are described, recorded and numbered, and transported to the ground for sealing and preservation. Some samples are shown in Figure 2. According to the lithology exposed by the borehole column, the borehole column diagram is drawn, as shown in Figure 3.



Figure.2 Core rock samples taken from shunting chamber between 33 # and 32 # roadways

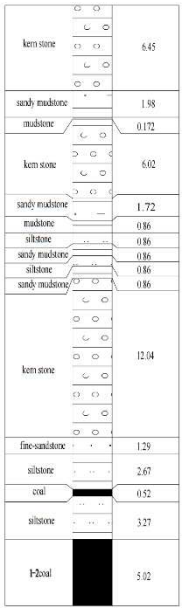


Figure.3 Columnar diagram of core rock sample in shunting chamber between 33 # roadway and 32 # roadway

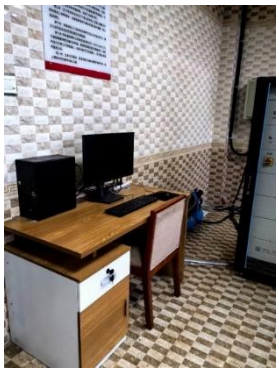
In the laboratory, the rock block is processed into a standard cylinder sample, and the uniaxial compressive test is carried out by using the multi-field coupling rock triaxial test system to determine the compressive strength and elastic modulus of the rock (as shown in Figure 4). The test results show that the compressive strength of coarse-grained sandstone is 18.2 MPa and the elastic modulus is 0.5441 GPa. The compressive strength of siltstone is 25.85 MPa and the elastic modulus is 1.5004 GPa. The compressive strength of sandy mudstone is 17.0 MPa and the elastic modulus is 0.4506 GPa. The siltstone has the highest strength and stiffness, which constitutes a relatively hard bearing layer. Although the strength of coarse-grained sandstone is slightly lower, its thickness is large (12.04 m) and its integrity is strong, which also has the characteristics of key strata. The sandy mudstone has the lowest strength and belongs to the weak rock stratum, which plays a role of loading and filling in the overburden movement.



(a) partial rock specimen



(b)Multi-field coupling rock triaxial test system



(c) Data analysis system

Figure. 4 Multi-field coupling rock triaxial test

3.2 Identification of key layers

According to the borehole histogram of 33 # joint roadway and 32 # joint roadway shunting chamber, combined with the correction results of columnar information within 40 m above the coal

seam of BK220 borehole, the key strata are identified based on the key strata theory. The identification of the key stratum is based on the deformation coordination condition : if the first layer (the lowest hard rock layer) deforms synchronously with the nth layer, the first layer is the key stratum, and the load of the overlying n − 1 layer is borne by the first layer [6]. The key stratum judgment diagram is shown in Figure.5.

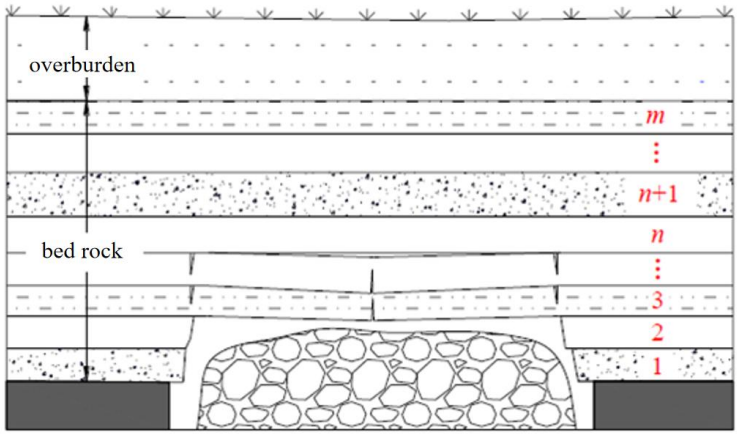


Figure.5 Key layer judgment diagram

The key layer identification adopts formula (1) and formula (2) :

$$(q_n)_{\max} < (q_{n+1})_{\max} \tag{1}$$

In which, the load of the nth layer of rock on the first layer of rock is calculated according to the formula (2).

$$(q_n)_1 = \frac{E_1 h_1^3 (\gamma_1 h_1 + \cdots + \gamma_i h_i + \cdots + \gamma_n h_n)}{E_1 h_1^3 + \cdots + E_i h_i + \cdots + E_n h_n^3} \tag{2}$$

In the formula : q_n is the load of the nth layer on the first layer ; E_i、h_i、and γ_i are the elastic modulus, thickness and bulk density of the i layer, respectively. When $(q_n)_{\max} < (q_{n+1})_{\max}$, the n+1 layer does not deform synchronously with the first layer, and the first layer is the key layer, which controls the n layers above it.

According to the mechanical experimental parameters (the formation bulk density γ is 26 kN / m^3), combined with the thickness of each rock layer in the histogram, the discriminant results are shown in Table 2 by Eq. (1) and Eq. (2). The discriminant results show that there are 1 main key stratum and 7 sub-key stratum in the overlying strata of 12 on 302 working face. The sub-key layer 1 nearest to the coal seam is 12.04 m thick coarse-grained sandstone with a breaking distance of 32.88 m. With the increase of vertical height, the fracture distance of each sub-key stratum increases gradually, and the fracture distance of the main key stratum (64.78 m thick coarse-grained sandstone) reaches 388.10 m. The superimposed occurrence of multi-layer key strata is the geological basis for the formation of complex suspended roof structure on the free side.

Table 2 The identification results of the key layer position of BK220 borehole

serial number	rock formation	thickness / m	vertical height from coal seam /m	breaking interval /m	key stratum position
5	kern stone	12.04	7.75	32.88	Sub-critical layer 1
16	siltstone	9.63	25.57	67.57	Sub-critical layer 2

17	kern stone	12.15	35.58	72.35	Sub-critical layer 3
19	kern stone	11.16	39.14	94.60	Sub-critical layer 4
21	sandy mudstone	23.98	54.16	115.07	Sub-critical layer 5
24	sandy mudstone	19.98	83.91	141.21	Sub-critical layer 6
28	kern stone	57.01	126.48	192.60	Sub-critical layer 7
41	kern stone	64.78	378.52	388.10	main key stratum

3.3 Formation mechanism of suspended roof structure under multi-layer key layer stacking

Under the condition of high strength mining of shallow coal seam in Shendong mining area, there are often incomplete ' three zones ' in overburden rock migration, that is, there is no bending subsidence zone, only caving zone and fracture zone. Based on the characteristics of mining height and rock fragmentation and expansion, the height of overburden failure zone is calculated, and the height of caving zone is calculated according to formula (3):

$$H_k = \frac{M}{K_p - 1} \tag{3}$$

In the formula : H_k is the height of the caving zone, m ; M is the mining height of coal seam, taking 5.02 m ; K_p is the bulking coefficient of the immediate roof strata, taking 1.3, and the theoretical height of the caving zone is calculated to be 16.8 m according to the bulking coefficient method. The fracture development of the 6 # measuring point (15.85 m) shows that the rock mass has been irregularly collapsed. Therefore, the actual top boundary of the caving zone is comprehensively determined to be about 16.8 m, which is consistent with the theoretical calculation.

According to the critical layer identification method, the 12.04 m thick coarse-grained sandstone is the sub-critical layer 1 (12.04 m thick coarse-grained sandstone), 7.75 m from the vertical height of the coal seam, and in the caving zone (16.8 m), indicating that the overlying rock of the 12 upper coal seam moves with the key layer.

The height of fracture zone is calculated according to empirical formula (4):

$$H = \frac{100M}{1.6M + 2.2} \pm 5.6 \tag{4}$$

The height of the fracture zone is calculated to be about 54.6 m, and there are multi-layer thick-layer coarse-grained sandstones in this area.

Combined with the above identification results of key strata, there is a significant hierarchical control coupling relationship between multi-layer key strata and the development of ' two zones ' :

(1) The main control mechanism of the caving zone : the sub-key stratum 1 is located in the caving zone as a whole, and its breaking and caving directly determines the development height and shape of the caving zone, which is the core main control layer of the near-field overburden movement and mine pressure appearance.

(2) The layered control mechanism of fracture zone : sub-key strata 2-7 are all present in the fracture zone. Each sub-key stratum acts as a rigid bearing layer to control the expansion range and development degree of its upper fractures, forming a layered failure structure of ' one key stratum and one fracture zone ', which is the root cause of the stepped development characteristics of the fracture zone.

(3) The overall boundary control mechanism : the main key stratum is located at the top of the fracture zone, and its integrity defines the upper limit of the development of the fracture zone, and controls the overall failure range of the overburden rock and the far-field stress transfer.

In summary, the spatial distribution of multi-layer key strata directly determines the layered

development form of ' two zones ', and also determines the structural characteristics of thick and hard roof overhanging on the free side of high-intensity mining.

4 Monitoring and evolution characteristics of hanging roof structure of thick and hard roof on free side

4.1 Multi-point displacement monitoring scheme

In order to master the lateral hanging roof structure characteristics in the mining process of the working face, a multi-point displacement monitoring station is arranged in the 12 upper 303 auxiliary transportation crossheading, which is located at 12 m in the direction of the 33 # joint lane to the main lane, about 592.5 m away from the cut. The monitoring system is composed of GUD5000 mine-used intrinsically safe displacement sensor and YHC4.5 mine-used intrinsically safe data acquisition instrument. It is fixed to the designated layer of the borehole through the anchor claw, and connected to the displacement sensor through the wire rope. When the displacement of the overlying strata changes, the resistance change of the potentiometer inside the sensor is converted into an electrical signal, which is transmitted wirelessly to the data acquisition substation, as shown in Figure 6.

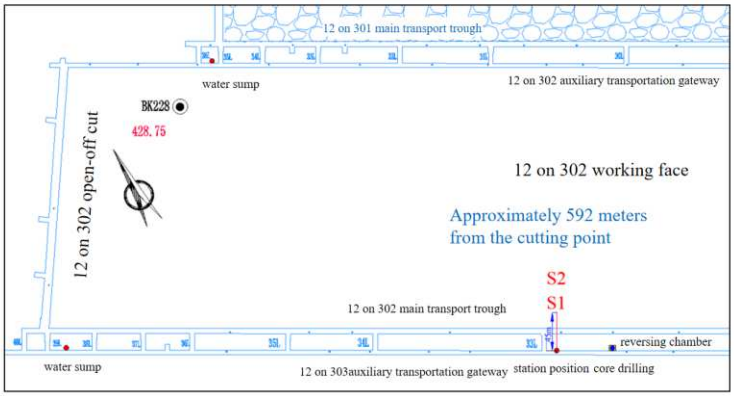


Figure.6 Monitoring station and coring drilling location map

The monitoring station set up 2 boreholes with a total of 8 measuring points. The dip angle of the high-level hole is 48 ° and the depth is 66.83 m. The dip angle of the low-level hole is 28 ° and the depth is 51.05 m. Covering the range of 7.8-46.5 m from the vertical height of the coal seam, across the three levels of the caving zone, the lower part of the fracture zone, and the middle and upper part of the fracture zone, corresponding to the upper and lower interfaces of each sub-key layer, to achieve simultaneous monitoring of zoning and stratification. Among them, the large-angle deep hole of the high-level hole covers the middle and high strata in the fracture zone, and the small-angle shallow hole of the low-level hole accurately monitors the main control key strata of the caving zone. The arrangement of each measuring point is shown in Table 3.

Table 3 Multi-base point displacement meter measuring point layer arrangement

Measuring point number	Depth of measuring point /m	Vertical height from coal seam /m	Corresponding to the main lithology	Corresponding to the key strata
1 # measuring point	66.83	46.50	siltstone	High rock strata in fracture zone
2 # measuring point	57.89	39.90	kern stone	Near sub-key stratum 2

3 # measuring point	45.87	31.02	kern stone	The middle rock stratum of fracture zone
4 # measuring point	39.41	26.23	sandy mudstone	The middle and lower rock strata of fracture zone
5 # measuring point	51.05	21.21	siltstone	Lower strata of fracture zone
6 # measuring point	39.72	15.85	sandy mudstone	Near the top of caving zone
7 # measuring point	32.28	12.36	kern stone	Upper part of sub-key stratum 1
8 # measuring point	22.67	7.80	kern stone	Lower part of sub-key stratum 1

The monitoring adopts the borehole axial displacement measurement method, and the subsidence characteristics are characterized by the vertical displacement of the rock layer converted by the borehole dip angle. The negative value of displacement represents the subsidence of rock strata in the direction of goaf, and the positive value represents the rebound in the direction of roadway. The mine intrinsically safe displacement sensor and data acquisition instrument are used to transmit data in real time, and the monitoring frequency is 2 h/time ; after the 2 # measuring point is calibrated by the same layer trend, the evolution law is consistent with the high-level measuring point, and the data is reliable. The layout of the monitoring layer is shown in Figure 7, and the installation is shown in Figure 8.

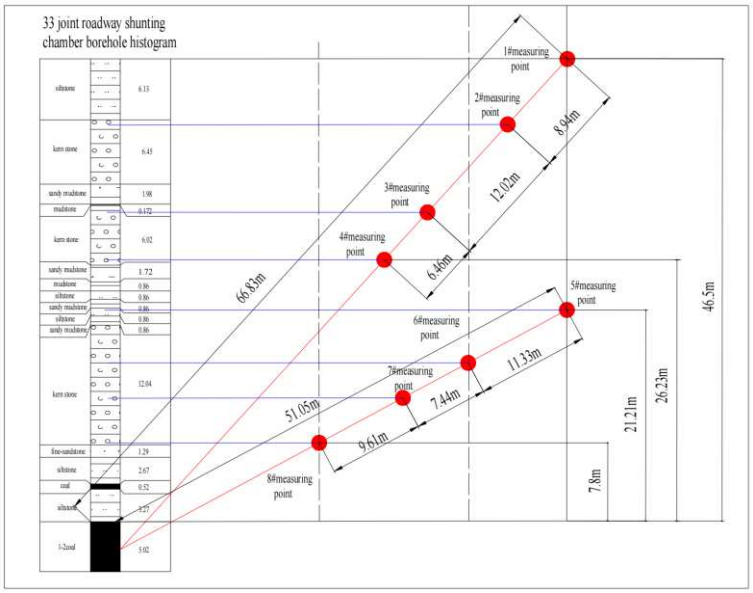


Figure.7 Multi-point displacement meter monitoring layer layout diagram



Figure.8 Installation diagram

The installation of the monitoring station was completed on January 7,2024. At this time, the working face was advanced by 513.7 m, and the station was 78.8 m ahead of the working face. As of January 27,2024, the cumulative advance of the working face was 672.5 m, and the station entered the goaf 80 m, with a cumulative advance of about 159.3 m.

The downhole data is wirelessly transmitted to the substation by the displacement sensor change acquisition instrument of each measuring point, and the substation displays the data of each measuring point in real time. The monitoring data of the substation on January 21,2024 is shown in Figure 9 (a total of 8 measuring points are installed corresponding to 01 ~ 08 transmission channels).



Figure.9 Real-time monitoring data of substation

4.2 Monitoring results and analysis

The continuous data of the built-in memory card in the substation are comprehensively analyzed. Combined with the cumulative advancement of the working face, the maximum displacement change value and fracture development of each measuring point are shown in Table 4.

Table 4 The maximum displacement change value and fracture development of each measuring point

Measuring point	Vertical height from coal seam /m	Maximum change value /mm	Fracture development
1#	46.50	−3	rupture
2#	39.90	−4	rupture
3#	31.02	−9	rupture
4#	26.23	−6	rupture

5#	21.21	−7	rupture
6#	15.85	+4	rupture
7#	12.36	+6	rupture
8#	7.80	−38	rupture

The monitoring data show the following rules :

(1) Low dominance : 8 # measuring point (located in the caving zone) has the largest displacement change, which is much larger than other measuring points, indicating that the low key stratum (sub-key stratum 1) is the main control layer of the suspended roof structure. The coarse-grained sandstone in this layer is 12.04 m thick and has strong integrity. A large area of cantilever beam structure is formed on the side of the goaf, and its breaking movement directly controls the mine pressure appearance of the goaf roadway.

(2) Displacement attenuation law : With the increase of the vertical height from the coal seam, the displacement change of each measuring point generally shows a decreasing trend, which is in line with the transfer characteristics of the overlying strata movement ' near field strong and far field weak ' [31]. The positive displacement (+4 mm,+6 mm) appears at the middle measuring point (6#, 7#). The positive displacement reflects the relative compression or separation closure rebound phenomenon caused by the rotary extrusion of the strata at the edge of the goaf. This phenomenon is particularly typical at the junction of soft rock and hard rock at the top of the caving zone.

(3) There are obvious gradients in the degree of fracture development of different zonal rock strata : the displacement of rock strata in the caving zone is large, the fracture is fully developed, and the integrity of rock mass is significantly damaged ; the displacement of rock strata in fracture zone decreases with the increase of strata, and the degree of fracture development gradually weakens, which reflects the hierarchical control effect of multi-layer key strata on fracture propagation.

4.3 Evolution process of strata movement

4.3.1 Direction : layered progressive fracture law

According to the monitoring data of multi-point displacement meter, combined with the advancing position of working face, the evolution process of rock strata along the strike of working face is drawn, as shown in Figure.10.From Figure.10, it can be seen that the fracture of overlying strata is characterized by progressive stratification from bottom to top, and the distance of fracture lagging working face increases gradually with the increase of strata. When the station lags behind the working face by 25.65 m, the immediate roof in the caving zone is first broken ; when the lag time is 31.2 m, the sub-critical layer 1 is broken, and the measured breaking length is basically consistent with the theoretical calculation value of 32.876 m, which verifies the accuracy of the key layer identification. When the lag time is 36.2 m, the separation layer appears in the high position layer ; when the lag time is 44.2 m, the rock breaking of each layer tends to be stable ; when the lag time is 68.98 m, the high rock strata in the fracture zone are finally broken.

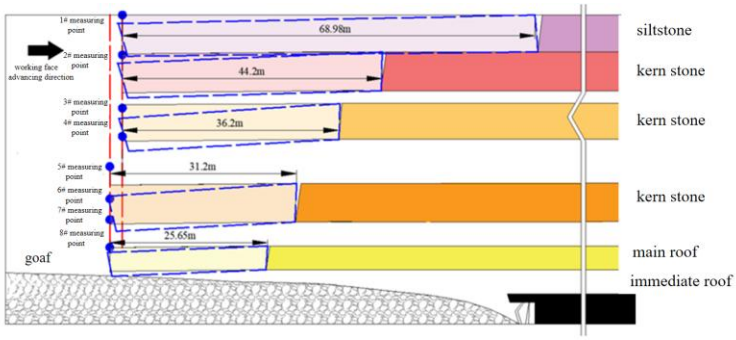


Figure.10 The evolution of rock strata recorded by each measuring point along the strike direction of the working face

4.3.2 Direction of inclination : stepped cantilever beam structure

According to the monitoring data of multi-point displacement meter, combined with the advancing position of the working face, the evolution process of rock strata along the working face is drawn, as shown in Figure 11. It can be seen from Figure.11 that the roof did not collapse immediately after breaking, forming a cantilever beam structure ; with the increase of the layer, the fracture line shifts to the side of the goaf, and the length of the cantilever gradually increases. The whole is in the form of a stepped lateral cantilever beam. The cantilever length of the caving zone is short and the fracture is early, and the initial cantilever length of the low immediate roof is only 25.65 m. The cantilever length of the fractured rock layer gradually increases with the increase of the layer, and the maximum lateral cantilever length of the high-level rock layer reaches 46.5 m, forming a large area of lateral hanging roof.

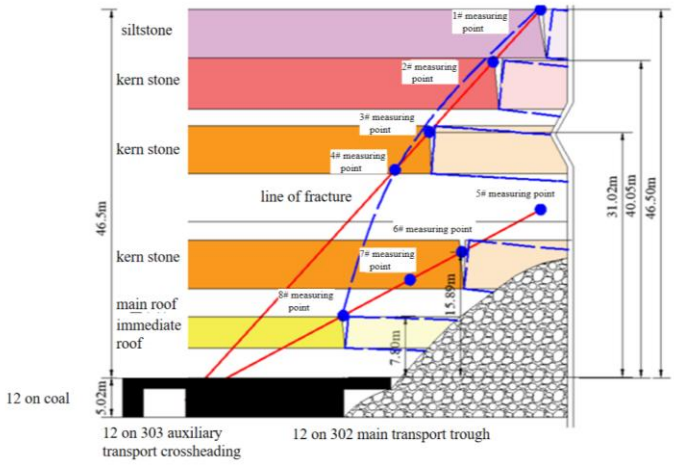


Figure.11 Evolution of rock strata along the dip direction of working face

4.4 Characteristics of suspended roof structure

From the above analysis, it can be concluded that the evolution of the lateral overhang reveals a complete ' cantilever growth → load transfer → fracture → masonry beam formation ' cycle. The strength of the coarse-grained sandstone of the main roof is large, the exposed state is prolonged, and the fracture is characterized by ' X ' type failure. The broken rock blocks squeeze each other to form a horizontal force, which generates friction between the rock blocks and forms a masonry beam structure. The upper and lower areas of the working face are damaged by the arc top, and the relationship between the masonry beam structure of the surrounding rock of the mining roadway is shown in Figure 12.

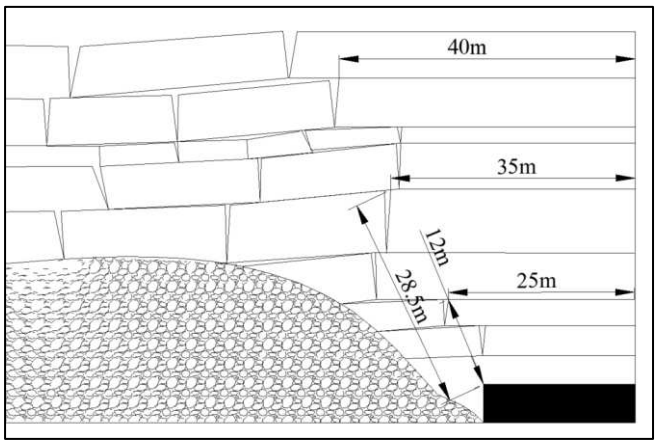


Figure. 12 Masonry beam structure relationship of surrounding rock in mining roadway

4.5 Strong mine pressure disaster-causing mechanism of suspended roof structure

Based on the identification and monitoring results of the key strata, the 12 upper 302 working face has the following characteristics of strong mine pressure disaster caused by the thick and hard roof hanging structure on the empty side :

(1) Multi-key strata superimposed overhang : There are 4 sub-key strata (sub-key strata 1 ~ 4) within 40 m above the coal seam, and the breaking distance of each key stratum increases from 32.88 m to 94.60 m, forming a sequence transfer pattern of ' low position first broken, middle position followed, high position then broken '. After the low key stratum is broken, the upper load is transferred to the middle key stratum, and the middle key stratum is cantilevered and broken again, which is transmitted upward in turn.

(2) Low-dominant lateral cantilever : Sub-key stratum 1 (12.04 m coarse-grained sandstone) is located in the caving zone and is the main control layer of lateral hanging roof. The length of the lateral cantilever increases and breaks periodically in the range of 8 ~ 12 m. The elastic energy released during the breaking is transmitted to the empty roadway through the coal pillar to form a dynamic load impact.

(3) Static load concentration effect : the lateral cantilever beam formed by the sub-key layer 1 in the caving zone continuously transmits the overburden load to the section coal pillar, forming a long-term high static load stress concentration environment ; the bearing capacity of 20 m narrow coal pillar is limited, and plastic failure is easy to occur under high static load, which leads to continuous deformation of roadway roof subsidence and floor heave.

(4) Static load-dynamic load superposition effect : The static load of the lateral cantilever (self-weight and overlying weak layer load) and the dynamic load of the mining (advance abutment pressure caused by the advance of the working face) are superimposed and transmitted to the floor through the limit equilibrium zone in the coal pillar. A large range of plastic deformation occurs in the floor, which is manifested as the floor heave of the roadway. The multi-layer sub-key strata in the fracture zone are broken in time series, and a large amount of elastic energy is released in each layer of key strata. When the fracture interval of the adjacent two key strata is small, the elastic energy release produces a superposition effect, forming an impact dynamic load, and superimposing the original high static load environment, it is easy to induce instantaneous strong mine pressure.

5 Roof control scheme

Aiming at the stepped lateral suspended roof structure formed by thick and hard multi-key strata and the strong mine pressure disaster-causing mechanism of ' static load concentration in

caving zone and crack driving load superposition ', this chapter puts forward the roof control idea of zonal fracturing, graded pressure relief and targeted disaster control. Directional pre-splitting blasting technology is used to target the key strata in different zones to realize the control of strong mine pressure in gob-side entry.

5.1 The principle of zonal hierarchical control

According to the study of the hanging roof structure of the thick and hard roof on the air side in the previous chapter, the main factor affecting the mine pressure appearance such as the floor heave of the roadway is the overlying multi-thick roof. There are low, medium and high main control influence layers in the overlying multi-thick roof. Therefore, it is proposed to carry out pressure relief in advance for low, medium and high hard rock layers through roof pre-splitting blasting technology, so that the overlying thick roof can be broken in time during the mining process of 12 upper 302 working face, avoiding the formation of suspended roof and reducing the strength of mine pressure appearance. The core idea of roof cutting and pressure relief is to destroy the integrity of the hard roof by directional fracturing, make the roof of the goaf fall in layers and times, transfer the hanging roof load to the goaf, and reduce the stress concentration of the coal pillar and the empty roadway. The roof control scheme implements differentiated control for different motion characteristics and disaster effects of caving zone and fractured zone, as follows:

(1) Collapse zone top load reduction: the sub-key stratum 1 in the caving zone is the main control layer of near-field mine pressure, and the long-distance lateral cantilever formed by it is the core source of high static load of coal pillar. By breaking the roof of the layer, the lateral hanging roof structure is directly cut off, the roof is forced to break at a predetermined position, and the load is transferred to the interior of the goaf, and the concentrated static load of the section coal pillar is released from the root.

(2) Layered decoupling of fracture zone : dynamic load superposition caused by sequential fracture of multi-layer sub-key strata in fracture zone is the main cause of instantaneous strong mine pressure. Through the implementation of layered fracturing of sub-critical layers at different levels, the integrity of thick and hard rock layers is cut, the breaking distance of each layer is shortened, the breaking time sequence is changed, the energy superposition of multi-layer synchronous breaking is avoided, and the impact dynamic load strength is effectively weakened.

At the same time, the spatial position of the broken roof line is optimized and arranged in the plastic zone-fracture zone of the coal pillar to ensure that the key stratum of the caving zone can rotate gently after breaking, so as to avoid the impact dynamic load caused by sudden drop. The rock strata in the fracture zone are guided to collapse orderly from bottom to top, and finally a stable overburden bearing structure is formed, as shown in Figure.13.

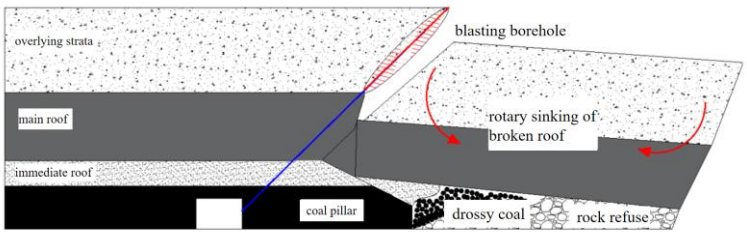


Figure. 13 Mechanical mechanism of targeted topping breaking

5.2 Partition control strategy and key parameter design

Combined with the mining state and the development stage of the suspended roof, the control

area is divided into the delayed pressure relief area and the advanced pre-control area, and the differential control is implemented ：

(1) Recovery lagging area : the lateral suspended roof structure has been completely formed, the deformation of the roadway continues to develop, and the control core is to quickly release the static load and inhibit the continuous deformation of the surrounding rock ;

(2) The advanced area without mining : the overburden structure has not been damaged by mining, the control core is to cut the rock structure in advance, limit the formation of long suspended roof from the source, and prevent and control the dynamic load impact disaster.

The core blasting parameters are designed based on the distribution of key layers and the height of the two zones, taking into account the safety specifications and pre-splitting effects. The key parameters are shown in table 5. All parameters are based on the core principle of ' accurately acting on the target key layer ' : the hole depth matches the vertical height distribution of the key layers in each zone, the sealing length meets the safety requirements of deep hole blasting and ensures that the fracturing section accurately corresponds to the target layer, and the hole spacing ensures the continuous penetration of the pre-splitting surface in the same layer.

Table 5 Blasting key parameter design table

key design	design value	design basis
blasthole diameter	94 mm	Matching on-site construction equipment and charging conditions
depth of blast hole	31-53 m	Corresponding to the vertical height of the key stratum from the caving zone to the upper part of the fracture zone
sealing length	13.5-27 m	Not less than 1 / 3 hole depth, meet the safety specification and precise explosion control.
charge linear density	3 kg/m^3	Adapt to the energy demand of thick and hard sandstone fracturing
spacing between blast holes	10 m	Ensure the continuous penetration of the pre-splitting surface of the key strata in the same layer

5.3 Drilling layout scheme targeting key strata

Based on the idea of zoning and grading control, the drilling arrangement adopts the design principle of ' layered targeting and deep and shallow matching '. According to the control target, it is divided into three types of drilling holes, which correspond to the key horizons in different zones [13,34] :

The low-position target hole : the hole depth is 31-35 m, and accurately acts on the sub-critical layer 1 in the caving zone. The core goal is to cut off the near-field main control hanging roof and release the high static load of the coal pillar ;

The median targeted hole : the hole depth is 40-45 m, acting on the sub-critical layer 2-3 in the lower part of the fracture zone, cutting the medium-thick hard rock layer, shortening the breaking distance, and weakening the elastic energy released by a single break ;

High-level targeted hole : the hole depth is 51-53 m, extending to the sub-key stratum on the upper part of the fracture zone, blocking the far-field stress transfer, and weakening the dynamic load superposition effect of multi-layer key stratum fracture. The typical section layout is shown in Figure 14.

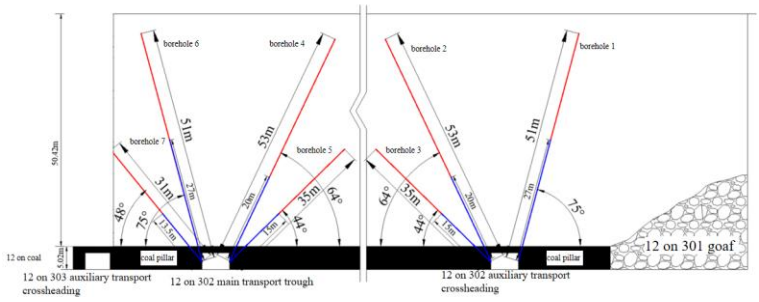


Figure. 14 Pre-splitting blasting drilling layout profile of roof

For different mining areas, the number of holes is adjusted on three types of basic boreholes :

The mining lag area : three boreholes are arranged in a single section, which correspond to the low, medium and high key strata respectively, and the layered pressure relief is mainly carried out for the formed lateral suspended roof.

Unmined advance area : 3-4 boreholes are arranged according to the adjacent goaf conditions. Under the condition of unilateral mining, three boreholes are set up to realize the advanced pre-control of the side roof of the working face. Under the condition of bilateral mining, a drilling hole above the coal pillar is added to realize the coordinated control of bilateral hanging roof.

5.4 Roof control effect is expected

Based on the above zoning and grading pre-splitting scheme, combined with the characteristics of overlying strata movement measured above, it is expected that the following control effects can be achieved :

(1) Shortening the length of the suspended roof : The pre-splitting weak surface breaks the hard roof at a predetermined position, and the lateral cantilever length is 85 m in the natural state, which significantly reduces the cantilever bending moment and load transfer coefficient [15].

(2) Layered and graded caving : through the low, medium and high three-layer fracturing, the multi-key layers are broken and collapsed in turn to avoid the dynamic load impact caused by large-scale synchronous breaking. Zhang Jian et al. [15] showed that pre-splitting blasting could shorten the first weighting step of the main roof by about 36 % and the periodic weighting step by about 30 %.

(3) Load transfer to goaf : after roof breaking, the load of hanging roof is gradually transferred to the caving gangue in goaf, and the lateral stress concentration factor of coal pillar is reduced. The field test of Geng Ming et al. [13] in Majiliang Coal Mine shows that the convergence of the two sides of the roadway is reduced by 39.6 % and the roof subsidence is reduced by 31.8 % after roof cutting.

(4) Improve the filling degree of goaf : The layered collapsed rock blocks fill the goaf in an orderly manner, providing support for the upper strata and shortening the load transfer path [34].

6 Discussions

6.1 Global comparison of failure characteristics of overlying strata and control effect of multi-layer interbedded structure

In this paper, the measured height of the caving zone is 16.8 m (caving ratio is 3.43), and the height of the fractured zone is 54.6 m (fracture ratio is 11.14). By comparing the results with the in-situ measured data under the same mining height conditions in the typical thick and hard roof mining area in the world, it can be seen that the collapse mining ratio of Daliuta mine (4.5 m mining height) in Shendong mining area is 3.38, and the crack mining ratio is 11.51 [31]. The collapse mining ratio

of Bulianta mine (5.0 m mining height) is 3.52, and the crack mining ratio is 11.26 [1]. The collapse mining ratio of Jinjitan mine (4.8 m mining height) in Yushen mining area is 3.52, and the crack mining ratio is 11.48 [3]. In the Sydney Basin, Australia (mining height is about 5.0 m), the ratio of fracture to mining is between 10.8 and 12.2 [32] ; the crack-mining ratio of similar thick and hard roof working face in Pocahontas mining area of the United States is also concentrated in the range of 11.0-11.8 [33]. The above cross-regional data show a highly consistent narrow convergence characteristic. When the mining height is 5 m and the main control roof is thick and hard sandstone, the caving-mining ratio is stable at 3.3-3.6, and the fracture-mining ratio is stable at 10.8-12.2. This macroscopic commonness shows that the main controlling factors of overburden failure height under the condition of thick and hard sandstone roof are mining height and rock mass fragmentation and expansion characteristics, and its brittle fracture mechanical behavior has the universality beyond the geological differences of specific mining areas.

Although there is a unified statistical law on a global scale, the differences in specific geological structures will still cause systematic deviations within the unified framework. In this paper, the crack mining ratio (11.14) is slightly lower than that of pure thick sandstone roof working face such as Bulianta Mine (11.26). The core control factor of this deviation is not the difference of geographical location, but the difference of lithologic combination. In this study, 1 main key layer and 7 sub-key layers are developed in the overburden within 40 m, and coarse-grained sandstone and sandy mudstone are frequently interbedded. The uniaxial compressive strength of sandy mudstone is only 17.0 MPa and the elastic modulus is 0.45 GPa, which is more than 3 times that of coarse-grained sandstone. This kind of soft and hard interbedded structure forms a multi-level ' energy dissipation interface ' during the longitudinal expansion of the fracture. When the tensile fracture encounters a layer of weak interlayer, its expansion driving force is partially absorbed and transformed into interlayer shear and separation deformation, thus blocking the fracture to a higher level. The study of the Pocahontas mining area also confirms that the multi-layer interbedded structure will significantly inhibit the final development height of vertical cracks and shorten the lateral cantilever length by about 15 % -20 % [33].This mechanism is consistent with the observation in this paper. In addition, the difference of rock strength is also involved in the regulation : the uniaxial compressive strength of coarse-grained sandstone in this test is 18.2 MPa, which is lower than the strength of coarse sandstone in some working faces of Yushen mining area (22 ~ 25 MPa). The relatively low strength of rock stratum makes the fracture more likely to occur, the fracture distance is smaller, and the development height of fracture zone is also slightly lower, which is consistent with the difference of measured results.

It is worth emphasizing that the above international comparative data are almost all based on the measured results of the overlying strata movement in the strike direction, that is, the breaking and caving law of the overlying strata above the stope during the advancing process of the conventional working face. In addition to providing vertical two-band data, this paper also uses 8-point multi-basepoint displacement monitoring stations to realize in-situ layered monitoring of lateral gob-side cantilevers in the tendency direction. The measured data show that the lateral cantilever length increases step by step with the increase of horizon. The initial cantilever length of the low immediate roof in the caving zone is 25.65 m, while the maximum lateral cantilever length of the high strata in the fracture zone is 46.5 m. This stepped structure intuitively verifies the hierarchical control effect of multi-layer key strata on lateral overburden : the low sub-key strata have short fracture distance, early fracture and small cantilever length ; the high sub-key stratum

has strong integrity, large breaking distance, and the cantilever length increases step by step. The discovery is less in the existing lateral gob-side overburden measurement research at home and abroad-most of the on-site monitoring schemes are only for the overlying strata or surface subsidence above the stope, and there is no direct observation on the layered motion characteristics of the suspended roof structure above the side of the coal pillar at the edge of the goaf. The measured data of the stepped lateral cantilever evolution process and the maximum cantilever length revealed in this paper make up for the deficiency of the lateral in-situ layered monitoring in the international shallow coal seam gob-side entry movement database.

In summary, the measured caving mining ratio and fracture mining ratio in this paper are in a reasonable range of data of similar mining areas in the world, which verifies the cross-regional consistency of the development law of ' two zones ' of thick and hard sandstone roof. The slight decrease of the crack-production ratio caused by the multi-layer soft-hard interbedded structure and the hierarchical evolution characteristics of the lateral stepped cantilever reflect the systematic modulation of the special geological structure on the unified law in the study area. More importantly, the in-situ layered monitoring data of the lateral cantilever provided in this paper supplements the in-situ evidence in this scenario. In the follow-up work, the author will introduce the lateral horizon correction coefficient into the existing formula for calculating the breaking distance of the key strata. The value range is 0.45 ~ 0.65, and its value increases with the increase of the horizon, so as to more accurately describe the hierarchical growth law of the lateral cantilever along the goaf, and further verify the hierarchical control effect of the multi-layer key strata on the lateral overburden.

6.2 Disaster-causing mechanism of multi-key strata superimposed suspended roof

Different from the condition of a single key stratum, the overlying multi-layer thick and hard rock layers on the 12 upper 302 working face are superimposed, and the breaking distance of each sub-key stratum increases from 32.88 m to 94.60 m, forming a differential breaking pattern. When the low key stratum is broken, the upper load is instantaneously transferred to the middle key stratum, resulting in a sudden increase in the cantilever load of the middle key stratum, which may induce the ' chain breaking ' effect. In the study of gob-side entry in extra-thick coal seam, Yan et al. [34] found that the superposition of ' vertical O-X ' fracture of near-field key strata and ' horizontal O-X ' fracture of far-field main key strata is the core cause of advanced deformation in the range of 200 m. In this paper, the large displacement change of 8 # measuring point (7.8 m layer) − 38 mm in the monitoring data is the direct characterization of the cantilever fracture of the low key stratum.

6.3 Limitations of key strata theory in the identification of high main key strata

The key stratum theory is the core theoretical basis of this study. The theoretical breaking distance of the mid-low sub-key stratum in this study is more than 96 % consistent with the measured value, which fully verifies the reliability of the theory in the identification of the mid-low strata. However, for the high-level main key strata with large thickness and shallow buried depth, the traditional discriminant method still has some limitations :

(1) The applicable boundary limit of the mechanical model : The traditional calculation of the breaking distance of the key stratum is based on the fixed beam theory and the implicit plane section assumption, which is suitable for thin rock strata with a thickness-to-span ratio of less than 1 / 5. In this paper, the thickness of the main key stratum is 64.78 m, the theoretical breaking distance is 388.1 m, the thickness-span ratio is close to the critical value of the thick plate, and the calculation error of the beam model will increase. At the same time, the heterogeneity and primary fracture distribution inside the thick rock layer will also lead to the deviation between the actual fracture

morphology and the ideal beam model.

(2) The coordination hypothesis of load transfer is idealized : the key stratum theory assumes that the key stratum and its overlying strata are completely coordinated, and the key stratum bears all the overlying loads. However, there are multi-layer weak mudstone interlayers under the high main key stratum, and the difference of lithologic modulus is more than 3 times. The interlayer separation is easy to occur during the deformation process, which leads to the direct transmission of some loads to the lower strata through shear action. The actual bearing load of the main key stratum is less than the theoretical calculation value, and its breaking time will lag accordingly.

(3) The shallow buried boundary effect is not fully included : the main key stratum in the study area is only covered with thin loose layer, and the constraint effect of the surface free surface on the rock stratum is weak, and the main key stratum is easy to directly cause the surface step subsidence after breaking. The traditional identification of key strata is not included in the boundary conditions of the overlying free surface, which can not reflect the special law of ' breaking-surface linkage ' of shallow buried and thick main key strata.

The criterion of lateral stress scene is missing : the existing key stratum discrimination is based on the fixed force state at both ends of the strike direction, while the rock stratum along the empty side is the force state of the cantilever beam, and its fracture criterion and load transfer mechanism are different from the strike direction. The direct application of the strike discrimination results will overestimate the ultimate cantilever length of the lateral strata. The maximum cantilever length measured in this paper is 46.5 m, which is less than the theoretical fracture distance of the same layer, which also confirms this point.

In general, the critical layer theory has high discrimination accuracy for the middle and low sub-critical layers, and model correction and parameter adjustment are required for the high-thick main key layer and the lateral cantilever scene.

6.4 Applicable relationship between lateral cantilever beam structure and ' masonry beam ' theory

The theory of ' masonry beam ' proposed by Academician Qian Minggao systematically clarifies the mechanical mechanism of the hinged equilibrium structure formed by the horizontal extrusion pressure of the rock block after the basic roof of the longwall working face is broken, which is the core theory to explain the periodic pressure of the working face. The stepped lateral cantilever beam structure proposed in this paper focuses on the inclined overburden rock on the side of the gob-side entry. The two have an inherent relationship in the theoretical system, and there is a clear applicable boundary.

(1) Continuity of spatial evolution : The two are the manifestations of overburden fracture process in different regions. During the advancing process of the working face, the rock strata in the middle of the goaf are fully broken and the rotary hinge is formed to form a stable masonry beam structure. In the tendency direction of the edge of the goaf, the rock stratum is not completely collapsed by the support of the coal pillar, forming a dynamic lateral cantilever beam structure. As the working face advances, the lateral cantilever beam breaks after reaching the limit span, and the broken rock blocks rotate to the interior of the goaf, and finally integrate into the masonry beam structure system in the deep part of the goaf.

(2) The difference of mechanical mechanism : the masonry beam structure relies on the horizontal extrusion force between the rock blocks to form a hinged balance, and the load is mainly transmitted to the front coal wall and the rear gangue through the hinged point, and the stress

distribution is relatively dispersed ; the lateral cantilever beam structure relies on the support of the coal pillar to form a cantilever force. The load is concentrated on the coal pillar area at the root of the cantilever, and the stress concentration coefficient is significantly higher. This is also the fundamental reason why the mine pressure behavior of the gob-side entry is much stronger than that of the coal wall of the working face.

(3) The complementarity of engineering application : the masonry beam theory mainly guides the roof management problems such as the selection of working face support and the prediction of periodic pressure ; the lateral cantilever beam structure is aimed at the mine pressure control of the gob-side roadway, which provides a theoretical basis for the design of lateral roof pre-splitting and coal pillar width. For the narrow coal pillar gob-side entry, the traditional masonry beam theory is difficult to explain the lateral source of its strong mine pressure, and the lateral cantilever beam structure can effectively make up for this theoretical gap.

On the whole, the lateral cantilever beam structure is a supplement to the key layer-masonry beam theory system in the gob-side scene, which enriches the spatial dimension of the overburden structure theory.

6.5 Limitations and prospects

This study has the following limitations : (1) Only one station is arranged for multi-base point monitoring. The monitoring data reflect the movement characteristics of overburden rock at a single position, and there may be differences in different positions due to changes in rock thickness and buried depth. (2) Analysis of pressure relief effectBased on engineering experience analogy and theoretical inference, there is a lack of quantitative verification of comparative monitoring data before and after blasting. In the future research, monitoring stations should be added at different positions along the strike of the working face to carry out quantitative evaluation of mine pressure before and after blasting.

7 Conclusion

(1) There are 1 main key strata and 7 sub-key strata in the 40 m range of overlying strata in the upper 302 working face of Buertai Coal Mine. The low sub-key strata (12.04 m thick coarse-grained sandstone) are located in the caving zone (the height of the caving zone is about 16.8 m). The overlying strata are distributed in the ' two zones ' of the caving zone and the fracture zone, and the height of the fracture zone is about 54.6 m. The superimposed occurrence of multiple key strata constitutes the geological basis of the complex hanging roof structure on the free side, and the horizon target of pressure relief control is defined.

(2) Multi-point displacement monitoring reveals that the displacement of the low measuring point (7.8 m horizon) changes the most (− 38 mm), and the displacement gradually decreases with the increase of the vertical height, indicating that the suspended structure is dominated by the low key layer. The strike direction breaks the bottom-up layered conduction, and the tendency direction forms a stepped lateral cantilever beam structure, with a maximum lateral cantilever length of 46.5 m. The length of the lateral cantilever is a periodic growth-breaking cycle in the range of 8 ~ 12 m, and the masonry beam structure is formed after breaking. The ' S-R ' stability condition determines the strength of the mine pressure behavior of the roadway.

(3) The disaster-causing mechanism of strong mine pressure in gob-side entry under the coupling effect of multi-layer key strata and ' two zones ' is clarified. The stepped lateral suspended roof structure is the core control structure of strong mine pressure, and the disaster-causing effect is reflected in the superposition of static load and dynamic load. The long cantilever structure of the

low key stratum in the caving zone causes the long-term high static load concentration of the coal pillar, which is the fundamental source of the continuous floor heave deformation of the roadway. The elastic energy of sequential fracture of multi-layer sub-key strata in fracture zone is superimposed to form impact dynamic load, which is the direct trigger factor of instantaneous strong mine pressure. The two are coupled with the bearing conditions of narrow coal pillar, and finally induce the instability of surrounding rock.

(4) A zoning and grading targeted roof control technology suitable for multi-layer key stratum structure is proposed. Based on the control principle of ' breaking roof to eliminate static load and breaking dynamic load ', the low, medium and high layered targeted drilling design is adopted to act on the main control key strata of the caving zone and the sub key strata of different levels of the fracture zone respectively. The research results can provide reference for mines with similar conditions.

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Author Contributions

Conceptualization, Z.L; methodology, Z.L.; software, Z.L; validation, C.Y.D and S.J.; formal analysis, S.J.; investigation, C.Y.D; resources, S.J.; data curation, Z.L.; writing—original draft preparation, Z.L.; writing—review and editing, C.Y.D.; visualization, S.J.; supervision, S.J.; project administration, C.Y.D.; funding acquisition, C.Y. D.All authors have read and agreed to the published version of the manuscript.

Data Availability Statement

The data from the Buertai coal mine that support the findings of this study are available from Shenhua Energy Co., Ltd. Shendong Coal Branch, but restrictions apply to the availability of these data, which were used under licence for the current study and so are not publicly available. The data are, however, available upon request and with the permission of Shenhua Energy Co., Ltd. Shendong Coal Branch.

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Competing interests

The author(s) declare no competing interests.