

Article

Field and Numerical Investigation of Water-Conducting Fractured Zone Development in a Deep Coal Seams with Variable Thickness at the 14,321 Working Face

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Abstract

This study characterizes the development of the water-conducting fractured zone at the 14321 working face of the Gubei Coal Mine, located in Fengtai County, Huainan City, Anhui Province, China, where segmentally differentiated mining was adopted in response to variations in coal seam thickness. The 4.2 m section was extracted by fully mechanized mining, whereas the 6.2 m section was extracted by fully mechanized top-coal caving with a cutting height of 4.7 m and a top-coal thickness of 1.5 m. Distributed optical fiber monitoring showed a four-stage strain evolution of steady response, slow increase, continuous increase, and abrupt change in the 4.2 m section. In the 6.2 m section, the strain response indicated progressive failure from the deeper part of the borehole toward the borehole opening. The measured caving zone heights were 13.4–13.6 m and 25.4 m, respectively. The DOFS-inferred upper boundaries of the potential water-conducting fractured zones were 41.4 m in the 4.2 m section and 58.0 m in the 6.2 m section. The 58.0 m result was further supported by surface-borehole drilling-fluid-loss observations, whereas the 41.4 m result was not independently verified by hydraulic testing. Numerical models constructed using the corresponding engineering conditions reproduced plastic zone heights of approximately 43 m and 66 m. These differences represent the combined influences of mining height, extraction configuration, mining intensity, and local geological variation. Because mining height and mining method changed simultaneously, their individual contributions cannot be separated in the present study. The results provide case-specific evidence for evaluating roof fracture development and water hazard prevention in deep coal seams with variable thickness.

Keywords: deep coal seams; water-conducting fractured zone; site-specific mining configurations; distributed fiber-optic monitoring

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1. Introduction

With the gradual depletion of shallow coal resources in China, coal mining continues to extend to greater depths. Deep coal seam mining is faced with an extremely complex geological engineering environment characterized by “high ground stress, high water pressure and strong mining” [1]. During the mining process of deep coal seams (such as coal

seam 1), the roof will undergo severe deformation, damage and collapse under the influence of mining, and develop upward to form a water-conducting fractured zone [2]. Accurately mastering the development height and temporal and spatial evolution law of water-conducting fractured zone is a prerequisite for mine water disaster prevention, water conservation mining and roof disaster prevention [3]. Recent studies have further revealed the development patterns of water-conducting fractured zones under complex aquifers and their direct control over water inrush mechanisms [4]. Especially in the engineering practice of near-horizontal coal seams, the fundamental purpose of implementing different mining technologies is not because of the change in coal seam thickness, but to effectively control the damage degree and fracture development range of overlying rock, so as to safely and scientifically improve the upper mining limit of near-horizontal coal seams [5].

Numerous studies have investigated mining-induced roof failure, the development of water-conducting fractures, and the associated monitoring and evaluation methods. In terms of the influence of different mining processes on roof rock damage and fracture development, a large number of studies show that under the same coal thickness, the height of water-conducting fractured zone caused by high-intensity one-time full-height mining such as fully mechanized top-coal caving is usually significantly higher than that caused by layered fully mechanized mining [6,7]. This difference is essentially due to the greater mining intensity and more concentrated stope space during fully mechanized top-coal caving mining, which intensifies the unloading and energy release of roof strata, thus driving the roof fracture to extend upward significantly. Therefore, under aquifer conditions, a fully mechanized top-coal caving face may have a greater water inrush risk than a conventional fully mechanized mining face, which requires special height prediction and prevention and control. In addition, the failure mode of overlying strata in fully mechanized top-coal caving face of large mining height is also different from that of conventional mining height. Similar simulation experiments found that the caving fractured zone of the overlying rock under fully mechanized top-coal caving mining was trapezoidal expansion, and the fracture distribution along the strike showed “m” shape characteristics [8,9], indicating that the roof fracture not only developed highly upward, but also expanded widely in front of and behind the work. According to the development law of water-conducting fractured zone, Liu et al. [10] deeply discussed the control effect of mining height on roof failure, and quantitatively determined the internal relationship between the maximum development height of water-conducting fractured zone and mining thickness. Recent large-scale data analysis and prediction models have also corroborated the decisive impact of mining height on the fractured zone in complex combined-strata structures [11]. Sun et al. [12] systematically revealed the evolution law of overlying rock fractures under repeated mining under the control of multiple key layers by means of theoretical analysis, similarity simulation, numerical simulation and field monitoring, and clarified the key control role of cumulative damage of roof and the superposition effect of mining-induced stress field on fracture morphology. Jiang et al. [13] combined theoretical analysis, numerical simulation, and field measurements to investigate the water-conducting fractured zone in a fully mechanized top-coal caving face, demonstrating that the coordinated movement of rock-layer groups and competent supporting strata controls upward fracture propagation. Their study focused on the final fractured zone height at a single working face, whereas the continuous strain evolution and differences between mining sections remain less well understood. In terms of the fracture evolution and numerical simulation analysis of overlying strata, the numerical calculation methods of continuous and discontinuous media have been widely used to reproduce the stress field evolution and fracture propagation process. Li et al. [14] used FLAC3D to conduct numerical simulation, combined with the independently developed roadway surrounding rock excavation unloading model test system and explored in depth the evolution law of surrounding rock deformation and failure characteristics under different unloading

rates. According to the mining conditions of near-horizontal coal seams, Liu et al. [15] constructed the numerical calculation model of the overlying strata of near-horizontal coal seams based on FLAC3D, systematically analyzed the stress distribution and failure characteristics of the overlying strata, and realized the qualitative and quantitative determination of the development height of the overlying strata caving zone and water-conducting fractured zone in the working face. Li et al. [16] used FLAC3D to establish a three-dimensional numerical model, systematically analyzed the influence of roadway surrounding rock stability under different in situ stress distribution forms, and defined the environmental coefficient of mining stress ETA. The greater the environmental coefficient of mining stress, the greater the surrounding rock stress environment. The range where the ETA coefficient is greater than 0.2 is called the damage risk area. In the field monitoring technology of roof deformation and fractured zone, distributed optical fiber sensing technology (DOFS) has become an important development direction of deep rock movement monitoring in recent years with the advantages of long distance, continuous distribution, high sensitivity and resistance to harsh environments. Liu et al. [17] used distributed optical fiber sensing to dynamically monitor the damage evolution of a weakly consolidated coal floor under dynamic pressure, confirming the effectiveness of DOFS in capturing mining-induced failure. Zhang et al. [18] applied the distributed optical fiber testing technology to the monitoring of surrounding rock morphology and structure, and successfully realized the dynamic perception of surrounding rock deformation distribution characteristics in the process of deep coal mining. Wang et al. [19] comprehensively applied Brillouin optical time domain reflectometer (BOTDR) and fiber Bragg grating (FBG) sensing technology to carry out real-time quantitative monitoring of roof movement. The research pointed out that the roof was mainly tensile failure during mining, and the strain accumulation was mainly concentrated in the rock interface and the interior of weak rock stratum. Liu et al. [20] further combined 3D digital image correlation (3D-DIC) with FBG to characterize the deformation and damage of deep rock specimens. Zhu et al. [21] studied the deformation and failure state of the roof above the goaf by using Brillouin optical time domain analysis (BOTDA) technology. The results showed that the strain zoning characteristics of optical fiber monitoring had a good corresponding relationship with the deformation zoning characteristics of the roof; Sun et al. [22] used Brillouin optical time domain analysis (BOTDA) to dynamically monitor the internal deformation of the model, and compared and analyzed the evolution characteristics of strain field and geoelectric field under mining conditions, so as to reveal the development mechanism of rock deformation and failure; Yuan et al. [23] analyzed the roof deformation characterization mode based on DOFS through simulation model test, discussed the influence of optical fiber embedding rules on model test in numerical simulation, and elaborated the characterization mechanism of the roof structure zoning process. The influence of optical fiber embedding on rock displacement is revealed in the model test; Sun et al. [24] used Brillouin optical time domain reflectometer (BOTDR) and borehole resistivity comprehensive monitoring to analyze the “optical electrical” evolution law of roof during mining. Recent field applications have demonstrated that DOFS can continuously capture mining-induced overburden deformation and assist in identifying the upper boundaries of the caved and fractured zones [25].

Although the distributed optical fiber sensing technology and numerical simulation have been applied in the field of rock movement, comparative field evidence from different sections of the same variable thickness working face under their actual site-specific mining configurations remains limited. Based on this, this paper comprehensively uses the field-distributed optical fiber sensing technology and numerical simulation and other means to carry out in-depth characterizations of the fractured-overburden responses in the two engineering sections under their actual mining configurations. By installing long-

distance monitoring boreholes in the roof of the working face and implanting sensing optical cables, combined with optical fiber demodulation and data analysis software system, the strain response characteristics of overlying rock under real mining conditions are obtained. This study aims to characterize and compare the roof-fracture responses observed in two sections of the same variable thickness working face under their actual mining configurations. Because mining height and mining method vary simultaneously between the two sections, the comparison is intended as an engineering case analysis rather than a controlled assessment of the independent effect of mining method.

2. Geological Setting and Mining Conditions

Gubei Coal Mine is located in Fengtai County, approximately 23 km northwest of central Huainan City, Anhui Province, China (Figure 1). The coal-bearing strata are mainly Permian in age, and the geological conditions are relatively complex. The 14321 working face is located northwest of the mine shaft station and extracts the No. 1 coal seam. The minable length of the working face is 1286 m, the face width is 197 m, and the average coal thickness is 5.4 m. The coal seam dip angle is 1–11° with an average of 5°. According to the Chinese classification of coal seam inclination, it is therefore a near-horizontal coal seam, with only local sections entering the gently dipping range. It belongs to the deep mining working face. The lithology of the roof and floor of the 14321 working face is relatively complex, and there is no pseudo roof development. The immediate roof is mainly mudstone with a thickness of about 0.8 m, which is prone to collapse. The main roof is sandstone with relatively hard lithology. The immediate floor is siltstone and the main floor is fine sandstone (Table 1). Affected by the geological sedimentary environment, the coal seam thickness of the 14321 working face has significant trend (or tendency) variation characteristics. The average thickness of the coal seam in the front section is about 4.2 m, and the thickness of the coal seam in the back section is about 6.2 m. In order to realize the “full extraction” and efficient recovery of coal resources, the working face adopts the segmented differentiated mining technology: for the 4.2 m thick area, the layered fully mechanized mining technology is adopted. Aiming at the 6.2 m thick area, the dynamic adjustment was made to the fully mechanized top-coal caving technology, and the continuous and safe mining under the condition of complex variable thickness coal seam was realized. In the 6.2 m thick section, the term “top coal” denotes the upper portion of the coal seam that is not extracted by the shearer but is instead caved and recovered through the rear coal-discharge opening of the support; it is therefore distinct from the immediate roof. The shearer cutting height is 4.7 m, and the top-coal thickness is 1.5 m, resulting in a total extraction thickness of 6.2 m. The corresponding cutting-to-caving height ratio is approximately 3.1:1.

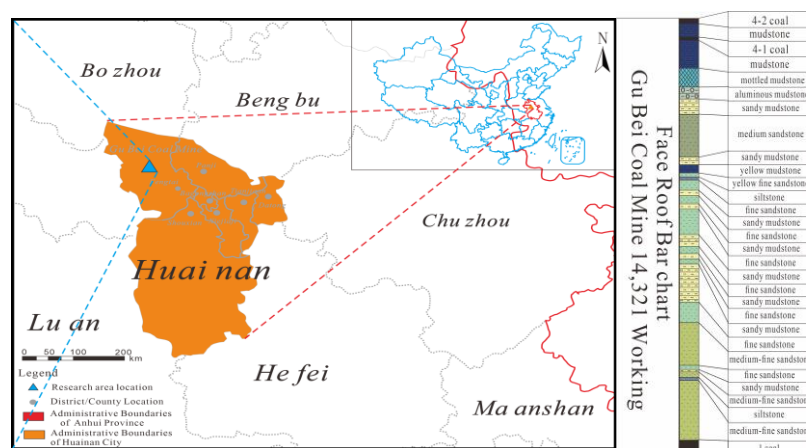


Figure 1. Location map of the study area and borehole column diagram.

Table 1. Lithological characteristics of the roof and floor strata.

Name of Top and Floor	Rock Name	Thickness (m)	Lithological Characteristics
Main roof	Fine sandstone	13–43.5 Average 24	Gray, fine-grained sandy structure, mainly composed of quartz and feldspar, with poor sorting and roundness
Immediate roof	Sandy mudstone	0–2.5 Average 1.2	Dark gray, mainly composed of clay minerals with argillaceous structure and sandy mixed structure
Immediate floor	Siltstone	2.1	Gray, with typical silty structure, with horizontal bedding or weak wavy bedding
Main floor	Sandy mudstone	13.8	Dark gray, mainly composed of clay minerals with argillaceous structure and sandy mixed structure

3. On-Site Optical Fiber Monitoring and Characteristic Analysis of the Water-Conducting Fractured Zone on the Roof

3.1. Principle of Fiber-Optic Sensing Monitoring and Construction of Monitoring System

To characterize mining-induced roof failure, field monitoring was conducted using distributed optical fiber sensing. Boreholes 1 and 2 were installed in the roof of the 4.2 m fully mechanized mining section, whereas Boreholes 3 and 4 were installed in the roof of the 6.2 m fully mechanized top-coal caving section. Distributed optical fiber sensing cables were installed in all four boreholes (Figure 2). During the whole field observation, the position of the orifice reference section remained unchanged. The intersection of the orifice and the roadway is selected as the origin of the coordinate system, the projection along the horizontal direction of the borehole is in the x-axis positive direction, and the vertical upward direction of the roof is in the y-axis positive direction. In this paper, the optical fiber strain data of Boreholes 1 and 2 in the thin coal area and Boreholes 3 and 4 in the thick coal area are selected for analysis. Borehole 1 was 80 m long and inclined upward at 45.6°. Its horizontal projection was oriented approximately 10° from the return airway toward the panel interior, providing vertical and horizontal control distances of 57.2 and 56.0 m, respectively. Borehole 2 was 55 m long and inclined upward at 28.5°, providing vertical and horizontal control distances of 26.2 and 48.3 m, respectively. Borehole 3 was 116 m long and inclined upward at 45.0°, with vertical and horizontal control distances of 82.0 m. Borehole 4 was 84 m long and inclined upward at 22.0°, providing vertical and horizontal control distances of 31.5 and 77.9 m, respectively. High density fixed-point distributed strain sensing optical cable is used to monitor roof damage. The distance between measuring points in Boreholes 1, 2, and 4 is 1 m, and the distance between measuring points in Borehole 3 is 2 m; the length of cable Borehole 1 is 100 m, the length of cable Borehole 2 is 70 m, the length of cable Borehole 3 is 170 m, and the length of cable Borehole 4 is 100 m. The spatial layout, borehole geometry, and fiber-optic sensing parameters of the four roof monitoring boreholes are summarized in Table 2. The optical cable is tied to a high-strength PVC pipe, and the front PVC pipe is processed into a shuttle shape, which is manually connected and pushed into the hole. The front end of the optical cable is fixed, and tape is wrapped every about 1 m along the pipe to prevent slipping. Three rounds of grouting are used for drilling, and after the final hole is drilled, the entire hole is grouted with a grouting pressure of 1.2 MPa. The exhaust pipe is required to for the grout to return to ensure the coupling between the sensor and the hole wall. The orifice is sealed with a locking disc and a dedicated gate valve. During the monitoring process, the mine optical fiber demodulator is used for daily monitoring to realize data acquisition, and the optical fiber demodulation and data analysis software system are used on the ground to process and analyze the field data.

Table 2. Spatial layout, borehole geometry, and fiber-optic sensing parameters of the roof monitoring boreholes.

Borehole No.	Monitoring Section And Face-Advance Chainage	Horizontal Orientation	Inclination (°)	Borehole Length (m)	Control Range, Vertical/Horizontal (m)	Fiber-Optic Sensing-Point Spacing (m)	Installed Fiber-Optic Cable Length (m)	Monitoring Function
1	Thin-seam mining section, 445.2 m	10° to the re-turn airway, toward the panel interior	45.6	80	57.2/56.0	1	100	Primary borehole for determining the upper boundaries of the caving zone and water-conducting fractured zone
2	Thin-seam mining section, 445.2 m	10° to the re-turn airway, toward the panel interior	28.5	55	26.2/48.3	1	70	Auxiliary borehole for locally cross-checking the caving zone and lower part of the fractured overburden
3	Thick-seam top-coal caving section, 1043.0 m	10° to the re-turn airway, toward the panel interior	45.0	116	82.0/82.0	2	170	Primary borehole for determining the upper boundaries of the caving zone and water-conducting fractured zone
4	Thick-seam top-coal caving section, 1043.0 m	10° to the re-turn airway, toward the panel interior	22.0	84	31.5/77.9	1	100	Auxiliary borehole for locally cross-checking the caving zone and lower part of the fractured overburden

Boreholes 1 and 2 were located in the 4.2 m fully mechanized mining section at a face-advance chainage of approximately 445.2 m, whereas Boreholes 3 and 4 were located in the 6.2 m top-coal caving section at a chainage of approximately 1043.0 m. Their horizontal projections were oriented at approximately 10° from the return airway toward the panel interior and were approximately aligned with the face-advance direction. The monitoring records therefore describe the temporal response of the roof as the working face approached and passed the corresponding borehole-controlled section. The boreholes in each section were designed as a deep–shallow pair rather than as four equivalent determinations of the fractured zone height. Borehole 1 was the primary borehole for determining the upper boundary in the thin-seam section, while Borehole 2 provided auxiliary verification of the caved zone and the lower fractured overburden. Borehole 3 was the primary borehole for determining the upper boundary in the thick-seam section, whereas Borehole 4, with a vertical control height of 31.5 m, was used only to verify the progressive failure process in the lower roof. Borehole 4 did not reach the upper boundary of the fractured zone and was therefore not used as an independent determination of the 58.0 m height. The monitored sections were selected outside the immediate influence ranges of the major mapped faults and

obvious tectonic disturbance zones identified from mine geological mapping and roadway exposure. No major mapped fault intersects the borehole-controlled sections. Nevertheless, the possible effects of small-scale joints and local variations in bedding properties cannot be completely excluded. The two borehole groups intersect the same regional roof-stratigraphic sequence. The immediate roof is predominantly weak sandy mudstone with an average thickness of approximately 1.2 m, whereas the main roof contains a relatively thick and competent fine-sandstone unit with an average thickness of approximately 24 m. The competent sandstone and the sandstone–mudstone interfaces constitute important mechanical controls on upward fracture propagation. The positions of persistent strain anomalies were therefore interpreted together with the local lithological interfaces. Because the thickness and elevation of these competent units vary along the panel, the resulting fractured zone heights may also exhibit spatial variation outside the monitored sections.

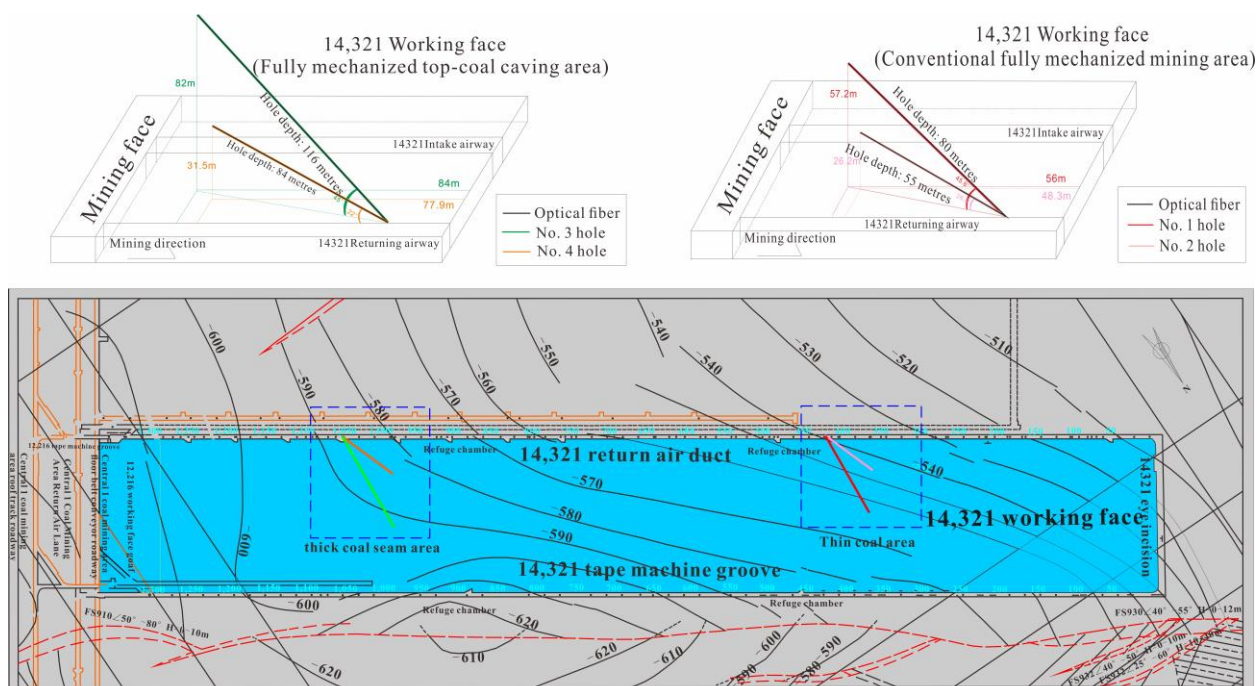


Figure 2. Geological profile and borehole location of working face in the study area.

Prior to mining, the overlying aquifers were dewatered by drainage measures; therefore, the working face was extracted under essentially drained conditions. Under these conditions, the measured heights represent the extent of the interconnected fracture network that would provide a potential water-conducting pathway if the strata were recharged, rather than a zone through which water was observed to flow during mining.

3.2. Fiber-Optic Strain Processing and Response Analysis in the Two Engineering Sections

3.2.1. Fiber-Optic Strain Responses in the 4.2 m Fully Mechanized Mining Section

Underground data collection was carried out according to the mining progress. The data acquisition is started when the mining face is about 120 m away from the observation hole, which is used as the background field for observation, and any data collected thereafter are compared with this. The optical fiber data shall be collected from the mining face about 100 m away from the final hole of the observation hole until the measuring line in the hole is completely destroyed or the collection is no longer allowed on site. When the working face is mined to a position 80–0 m from the final observation hole, the data shall be collected once a day until the observation line of the observation hole is completely destroyed or the collection is no longer allowed on site.

The strain response characteristics of Boreholes 1 and 2 are shown in Figures 3 and 4. The strain of the optical fiber in the fully mechanized mining area shows the evolution law of “steady slow increase continuous increase mutation” with the advance of the working face. The monitoring of Borehole 1 showed that before 18 November, the working face was 86.3 m away from the hole, the overall strain response of the optical fiber was small, and the overlying rock was not disturbed. From 20 November to 25 November, the working face was 77.9–58.9 m away from the hole opening, the pressure strain in the hole slowly increased, and the pressure strain at the hole length of 17.0 m increased to -458.6×10^{-6} , which was caused by the advance of the bearing pressure. From 26 November to 3 December, the working face was 55.6–24.8 m away from the orifice, and the compressive strain continued to increase. The stress concentration area was located at the hole length of 12.0–24.0 m and 51.0–80.0 m, and the compressive strain at 17.0 m was -3504×10^{-6} . On

December 4, the compressive strain in the 0–52.0 m section of the hole length rapidly changed to tensile strain, and the tensile strain at 15.0 m reached 2264×10^{-6} . On 5 December, the tensile strain at 15.0 m surged to 6981×10^{-6} , indicating that the roof collapsed. It increased to 9081×10^{-6} on December 6. The law of Hole 2 is consistent, and the tensile strain at 3.0 m of the hole increases from 97×10^{-6} to 7000×10^{-6} . Based on this, it can be judged that the hole length of the caving zone in the fully mechanized mining area is 19–28 m, and the vertical height is 13.4–13.6 m. The upper boundary of the persistent DOFS strain anomaly occurred at a borehole distance of approximately 58 m, corresponding to a vertical height of 41.4 m above the coal seam roof. This height is interpreted as the DOFS-inferred upper boundary of the potentially water-conducting fractured zone rather than as direct evidence of active hydraulic connectivity. In general, the strain in the fully mechanized mining area is led by the concentration of the advanced compressive stress, and marked by the sharp increase in the tensile strain before the caving. The paired boreholes show consistent strain-evolution characteristics in the overlapping lower-roof intervals, providing local cross-validation of the zonal interpretation. However, the upper boundaries were determined primarily from the deeper Boreholes 1 and 3.

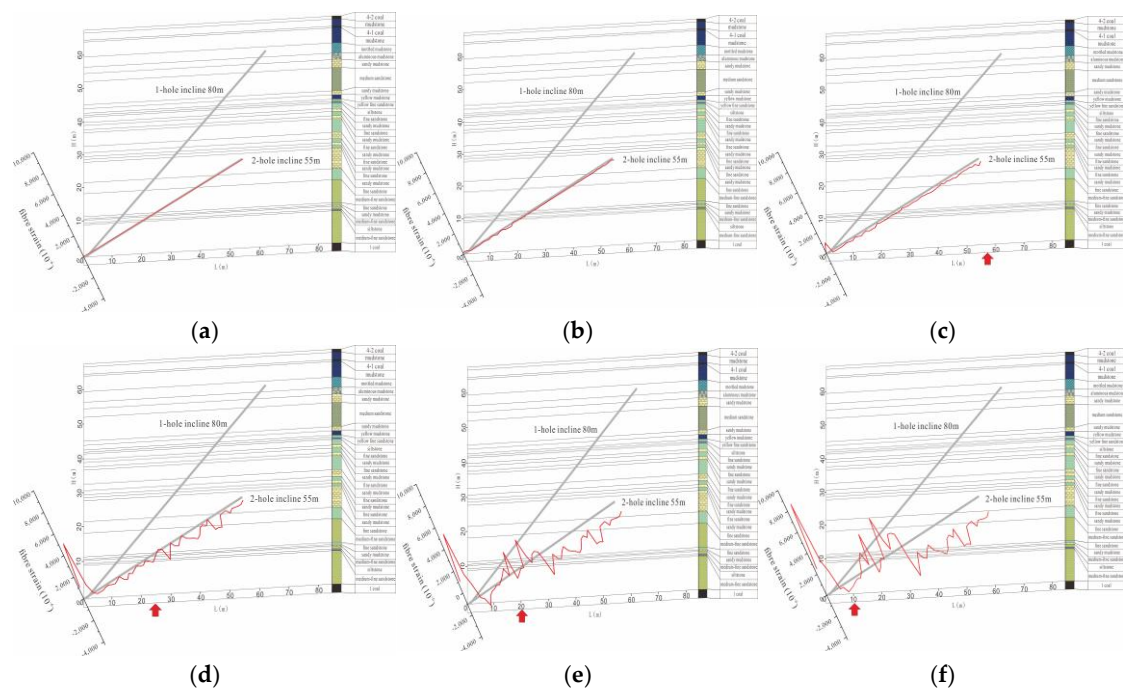
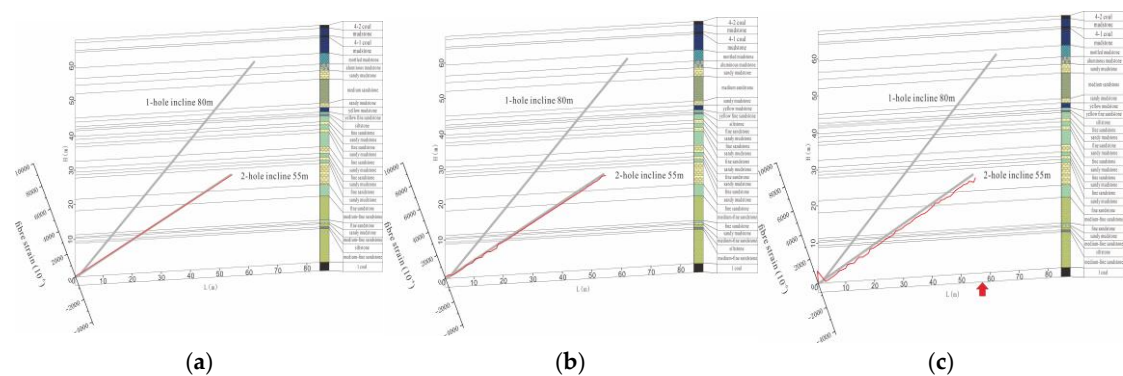


Figure 3. Optical fiber strain responses in Borehole 1 at face-to-borehole distances of (a) 169.0 m; (b) 58.9 m; (c) 38.7 m; (d) 24.8 m; (e) 21.0 m; and (f) 11.9 m.



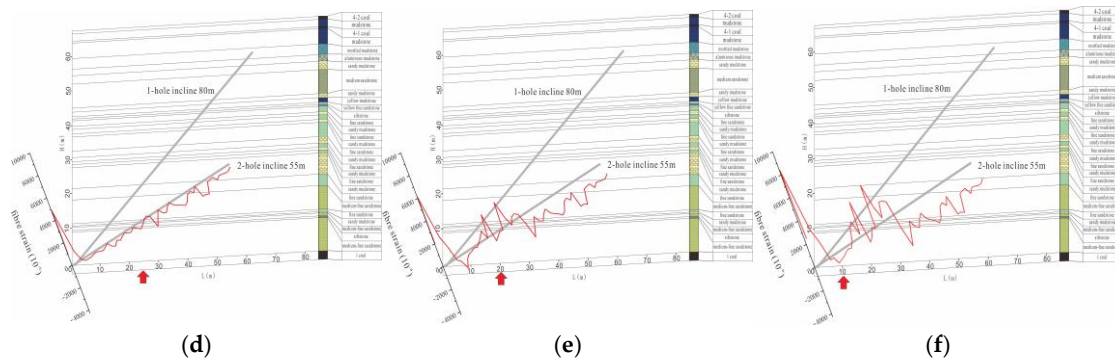


Figure 4. Optical fiber strain responses in Borehole 2 at face-to-borehole distances of (a) 165.1 m; (b) 103.0 m; (c) 88.1 m; (d) 24.8 m; (e) 21.0 m; and (f) 11.9 m.

3.2.2. Fiber-Optic Strain Responses in the 6.2 m Fully Mechanized Top-Coal Caving Section

Underground data collection was carried out according to the mining progress. The data acquisition is started when the mining face is about 180 m away from the observation hole, which is used as the background field for observation, and any data collected thereafter are compared with this. The optical fiber data is collected from the mining face about 180 m away from the end hole of the observation hole until the survey line in the hole is completely destroyed.

The strain response characteristics of Holes 3 and 4 are shown in Figures 5 and 6. The optical fiber strain in the fully mechanized top-coal caving area shows the evolution characteristics of large leading influence distance, obvious difference in layered sections and progressive collapse process. The monitoring survey of Hole 3 showed that on 22 April, the working face was 131.7 m from the borehole collar, and the strain response in the borehole was very small. From 23 April to 13 May, the working face is 128.7–72.4 m away from the hole opening, and the strain of optical fiber increases slowly. Along the hole length, it can be divided into three layers: the strain of 0–36 m increases slowly, the strain of 37–85 m increases rapidly, and the length of 86 m to the bottom of the hole is almost unchanged, reflecting that the influence of mining on rock mass in different layers is different. On 26 May, the working face was 60.3 m away from the hole opening, and the strain value at the hole length of 69 m increased sharply, resulting in a large tensile effect, which was analyzed to be caused by rock collapse. On 29 May, the working face was 55.9 m away from the orifice, and the strain at the hole length of 52 m increased again, indicating that the collapse gradually approached the orifice. On 6 June, the working face was 21.6 m away from the hole, and the tensile strain at the hole length of 29.0 m increased to 9576×10^{-6} . Based on this, it is judged that the hole length of the caving zone in the fully mechanized top-coal caving area is 36.0 m, and the vertical height is 25.4 m; the hole length of the water-conducting fractured zone is 82.0 m, and the vertical height is 58.0 m. Borehole 4 exhibited a strain-evolution pattern consistent with that observed in Borehole 3, and the strain is mainly the increase in tensile strain in the layer with the hole length of 30–68 m. Borehole 4 exhibited a strain-evolution pattern consistent with that observed in Borehole 3, with the principal tensile-strain anomalies occurring at borehole distances of 30–68 m. On 1 June, the tensile strain at a borehole distance of 7 m increased sharply to $1863 \mu\epsilon$, further confirming progressive failure in the lower roof. However, because Borehole 4 provided a vertical control height of only 31.5 m, it was used solely as an auxiliary borehole for examining the progressive failure of the caved zone and lower fractured overburden and was not used to determine the 58.0 m upper boundary. On the whole, the leading influence distance in the fully mechanized top-coal caving area is about 100 m,

which is significantly larger than that in the fully mechanized mining area. The strain increases in sections along the hole length, and the collapse progresses from the bottom of the hole to the orifice. The peak value of tensile strain is higher, and the height of the caving zone and water-conducting fractured zone are significantly higher than that in the fully mechanized mining area.

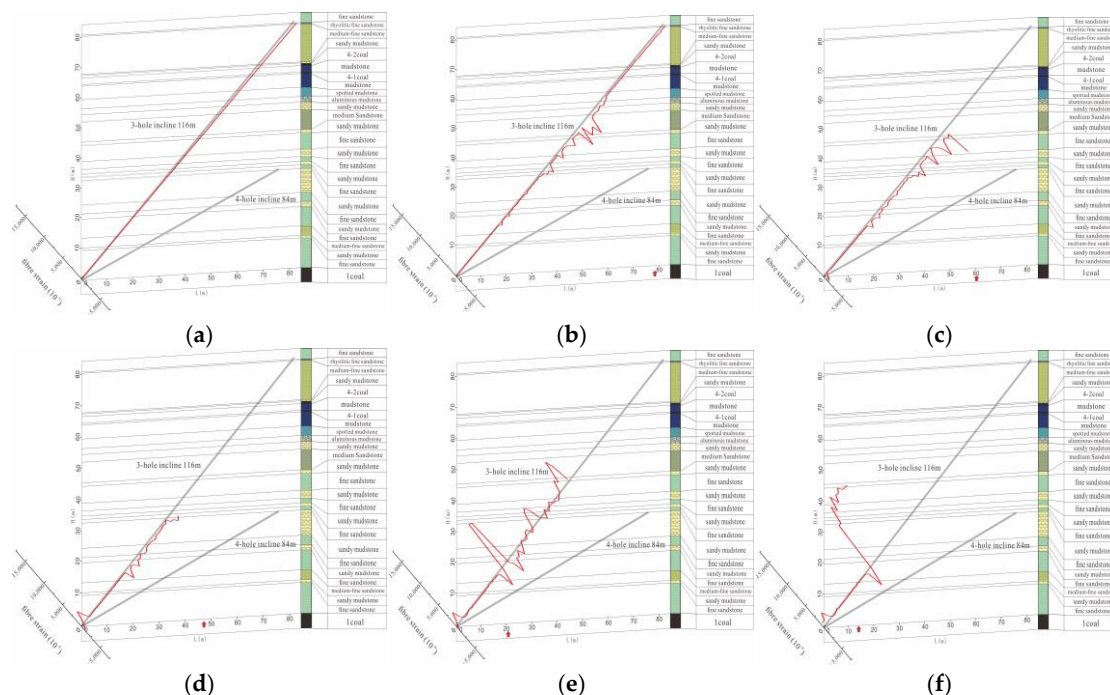


Figure 5. Optical fiber strain responses in Borehole 3 at face-to-borehole distances of (a) 169.0 m; (b) 77.8 m; (c) 60.3 m; (d) 47.6 m; (e) 21.6 m; and (f) 14.2 m.

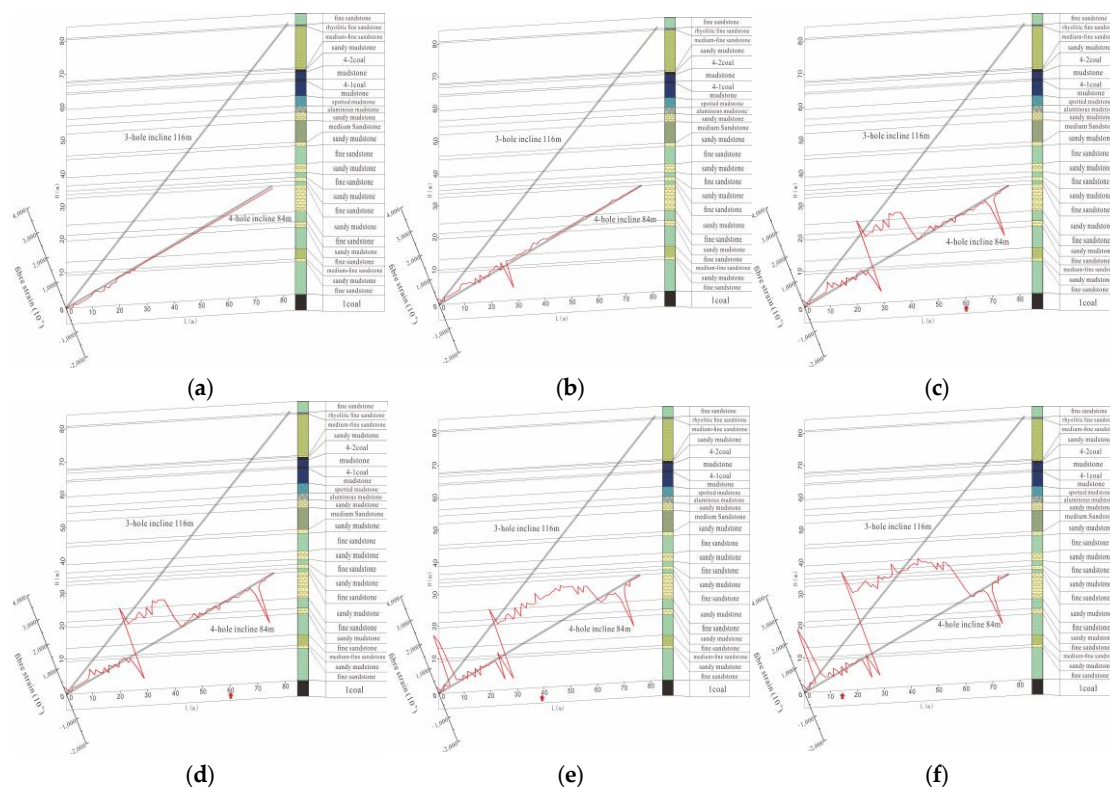
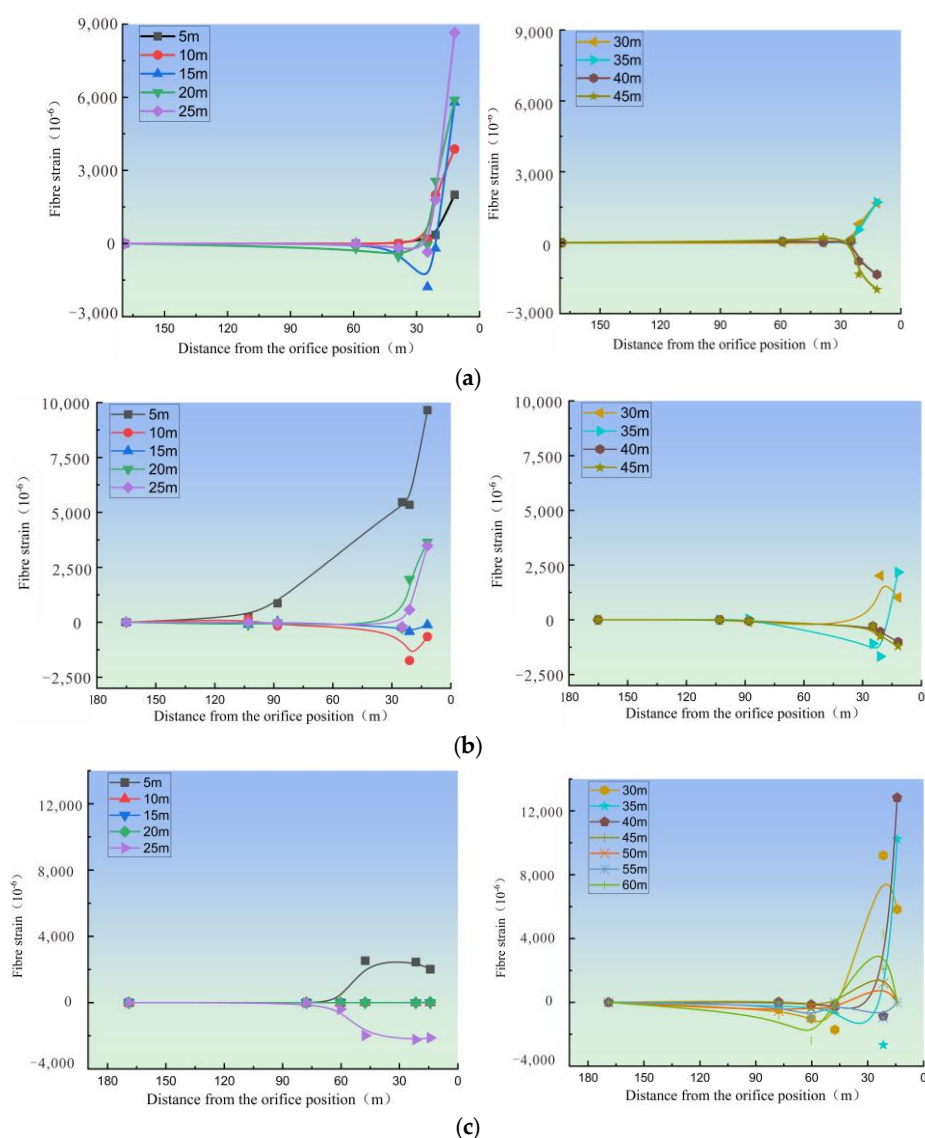


Figure 6. Optical fiber strain responses in Borehole 4 at face-to-borehole distances of (a) 169.0 m; (b) 108.9 m; (c) 77.8 m; (d) 60.3 m; (e) 39.2 m; and (f) 14.2 m.

The strain curve of optical fiber in fully mechanized mining area (Figure 7a,b) shows a four stage gradual evolution law of “steady slow increase continuous increase mutation”. The deformation is led by the concentration of advanced compressive stress and marked by the sharp increase in tensile strain before collapse. When the working face is far from the orifice (about 86.3 m), the roof is basically in the undisturbed original rock stress state, and the curve remains stable. As the working face approaches gradually (within 80 m from the hole), the roof is squeezed by the advance bearing pressure, and the compressive strain gradually increases in the hole. The compressive strain increases to -458.6×10^{-6} at the length of 17.0 m of borehole 1. When the working face is pushed to 55.6–24.8 m from the orifice, the compressive strain continues to increase, reaching -3504×10^{-6} at 17.0 m, and the stress concentration area is mainly distributed in the hole length of 12.0–24.0 m and 51.0–80.0 m. When the working face is pushed to the vicinity of the orifice, the roof strata lose support, begin to bend and sink and finally break, the optical fiber changes from compression to tension, and the tensile strain increases by thousands of microstrains. The tensile strain at 15.0 m of Borehole 1 suddenly increases from 2264×10^{-6} to 6981×10^{-6} and even 9081×10^{-6} ; at 3.0 m of Hole 2, it increased from 97×10^{-6} to 7000×10^{-6} indicating that the rock stratum has undergone substantial collapse and fracture. Based on this, it is determined that the hole length of the caving zone in the fully mechanized mining area is 19–28 m and the vertical height is 13.4–13.6 m, and the hole length of the water-conducting fractured zone is 58 m and the vertical height is 41.4 m.



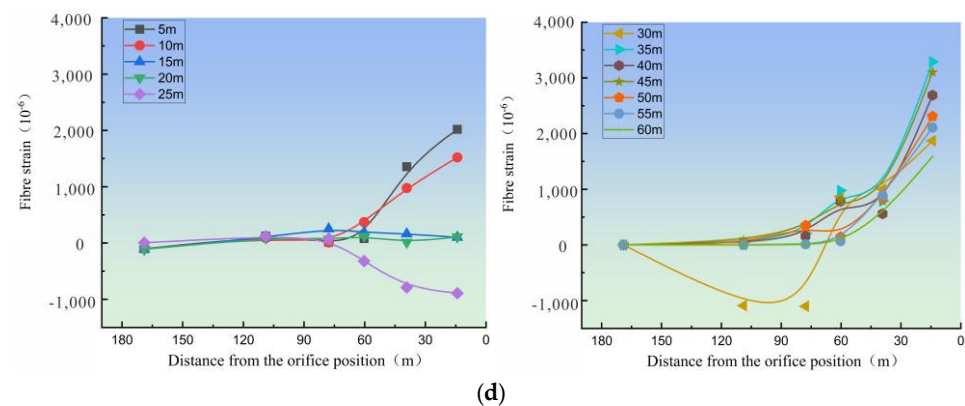


Figure 7. Variation curve of optical fiber strain in roof rock at different hole depths with advancing distance of working face during mining. (a) Borehole 1, (b) Borehole 2, (c) Borehole 3, (d) Borehole 4. Positive value represents tensile strain; negative value represents compressive strain.

Compared with the fully mechanized mining area, the strain curve of the optical fiber in the fully mechanized top-coal caving area (Figure 7c,d) shows the characteristics of “the leading influence distance is significantly increased, the difference between the layers along the hole length is obvious, and the peak value of the tensile strain is higher”, and the collapse process is obviously progressive. The stronger strain response observed in the 6.2 m top-coal caving section is mechanically consistent with its greater extraction thickness, larger mining void, and associated unloading intensity. However, because mining height and mining method varied simultaneously, this response cannot be attributed to the extraction method alone. A detectable strain response first appeared when the working face was about 130 m from the orifice, while the significant (effective) leading influence distance was about 100 m, which is obviously larger than that of the fully mechanized mining area. With the advance of the working face, the strain of Hole 3 increases in segments along the hole length: the hole length increases slowly from 0 to 36 m and rapidly from 37 to 85 m, and remains almost unchanged from 86 m to the bottom of the hole, reflecting that the rock mass in different layers is affected by mining differently. The collapse gradually progressed from the bottom of the deep hole to the shallow hole, and the tensile strain surged at the hole length of 69 m, 52 m and 29 m in turn. The tensile strain at the hole length of 29 m was as high as 9576×10^{-6} , and the tensile strain at the hole length of 7 m in Hole 4 also surged to 1863×10^{-6} . Based on this, it is determined that the hole length of caving zone is 36.0 m and the vertical height is 25.4 m, and the hole length of the water-conducting fractured zone is 82.0 m and the vertical height is 58.0 m. The locally inferred caved-zone and fractured zone heights were greater in the 6.2 m top-coal caving section than in the 4.2 m fully mechanized mining section, reflecting the combined influence of extraction thickness, mining configuration, roof lithology, and local mining-pressure conditions. The criteria used to distinguish the caving zone, water-conducting fractured zone, and bending subsidence zone are summarized in Table 3.

Table 3. Criteria for distinguishing the caving zone, water-conducting fractured zone and bending subsidence zone.

Zone	Fiber Strain Criterion	Drilling-Fluid Loss/Connectivity Criterion	Interpretation
Caving zone	Compressive strain reverses to tensile strain, with tensile strain increasing by several thousand $\mu\epsilon$ within 1–2 sampling intervals, or fiber breakage/strain step	Total drilling-fluid loss, consumption $> 9.6 \text{ m}^3/\text{h}$, borehole fluid level $> 200 \text{ m}$, drill dropping/air suction	Broken and collapsed rock, strongly connected

Water-conducting fractured zone	Sustained tensile/compressive strain anomaly below the caving threshold, extending along lithologic contacts	Markedly increased drilling-fluid loss but not total loss; loss-increase points coincide with lithologic interfaces	Interconnected fractures with potential water-conducting capacity
Bending subsidence zone	Strain close to the pre-mining background with no systematic trend	Drilling-fluid consumption close to background; no abnormal loss	Continuous bending, no significant connected fractures

Note: The hydraulic criterion was based on drilling-fluid loss during surface-borehole construction because the working face was mined after pre-mining dewatering and no natural water inflow was available.

3.3. Cross-Validation of Fiber Monitoring Results with Ground Boreholes

Surface borehole drilling fluid loss and water level observations were used to provide independent hydraulic evidence for the DOFS interpretation in the thick-seam section. Surface Borehole 2 indicated a fractured zone height of approximately 59.93 m, which is close to the 58.0 m DOFS-inferred boundary at Borehole 3 and therefore provides local cross-validation. Surface Borehole 1 yielded a greater height of 67.64 m, demonstrating spatial variability in fracture development within the thick-seam section. Consequently, the 58.0 m value should be regarded as the local DOFS-inferred height at the Borehole 3 monitoring section rather than the maximum height over the entire thick-seam area. The DOFS-derived and surface-borehole-derived heights of the caved zone and water-conducting fractured zone are compared in Table 4.

Table 4. Comparison and cross-validation of caved-zone and water-conducting fractured zone heights determined by DOFS and surface-borehole observations.

Mining Section	Monitoring or Verification Borehole(s)	Method	Caved-Zone Height (m)	Water-Conducting Fractured Zone Height (m)
4.2 m fully mechanized mining section	Boreholes 1 and 2	DOFS monitoring	13.6 m/13.4 m	41.4 m *
6.2 m fully mechanized top-coal caving section	Borehole 3	DOFS monitoring	25.4 m	58.0 m
6.2 m fully mechanized top-coal caving section	Borehole 4	DOFS monitoring	24.7	Not determined
6.2 m fully mechanized top-coal caving section	Surface fracture-monitoring Borehole 2	Drilling-fluid-loss and water-level monitoring	18.94 m	59.93 m
6.2 m fully mechanized top-coal caving section	Surface fracture-monitoring Borehole 1	Drilling-fluid-loss and water-level monitoring	25.58 m	67.64 m

* The value of 41.4 m represents the DOFS-inferred upper boundary of mechanically developed fractures with potential water-conducting capacity; it was not independently verified by hydraulic observations.

The boreholes were arranged along the advancing direction, so the strain records reflect the temporal evolution of roof failure as the face passed each borehole. The values of 41.4 m and 58.0 m should be interpreted as local DOFS-inferred upper boundaries of fractured overburden at the monitored sections rather than as spatially uniform or panel-wide

maximum values. The 58.0 m boundary was supported by surface-borehole hydraulic observations, whereas the 41.4 m boundary represents mechanically developed fractures with potential water-conducting capacity under recharged conditions.

4. Numerical Reproduction of Overburden Failure in the Two Engineering Sections

4.1. Construction and Parameters of the Site-Specific Numerical Models

In order to deeply analyze the stability of roof rock and the development characteristics of the water-conducting fractured zone of 14321 deep working face in Gubei Coal Mine under different mining methods, this study is carried out according to the technical idea of “field sampling physical and mechanical testing model construction and Parameter Inversion Numerical Simulation Analysis”. Combined with the field geological drilling data, the undisturbed core of coal seam 1 and its roof in the study area was obtained (Figure 8, left). Subsequently, the uniaxial compressive strength (UCS), elastic modulus and other rock mechanical properties of the above typical rock samples were tested in the laboratory (right of Figure 8), so as to accurately obtain the basic physical and mechanical indexes of the overlying strata and the lithologic units of the floor of the working face. To bridge the gap between laboratory core-scale properties and field-scale rock mass behavior, an empirical reduction procedure based on the Geological Strength Index (GSI) and Hoek–Brown upscaling theory were applied. The deep sedimentary strata in the study area exhibit moderate jointing and bedding development, corresponding to an average GSI of 65 ~ 75. Accordingly, the field deformation modulus E_{rm} and compressive strength σ_{cm} were downscaled to approximately 0.4 ~ 0.6 of their intact laboratory values (E_i and σ_{ci}). The equivalent elastic parameters (Bulk modulus K and Shear modulus G) were then converted via standard elasticity equations:

$$K = \frac{E_{rm}}{3(1 - 2\nu)}, \quad G = \frac{E_{rm}}{2(1 + \nu)} \quad (1)$$

The rock mass cohesion (c) and internal friction angle (φ) were estimated from the fitted Mohr–Coulomb parameters over the in situ stress range, while the field tensile strength (σ_t) was scaled down to 0.05 ~ 0.08 of the intact UCS. The final calibrated field-scale parameters are summarized in Table 5. These converted field-scale parameters not only quantitatively reveal the bearing and deformation characteristics of various rock masses under deep strong mining, but also provide reliable engineering boundary conditions and data support for the subsequent construction of FLAC3D numerical calculation model, the assignment of rock properties and the setting of boundary conditions, ensuring that the numerical simulation results can truly reflect the actual stress and failure evolution law of the 14,321 working face.



Figure 8. Rock sample coring and uniaxial compressive strength test.

Table 5. Mechanical parameters of coal and rock mass.

Serial Number	Lithology	Density (Kg/m ³)	Bulk Modulus (Pa)	Shear Modulus (Pa)	Cohesion (MPa)	Internal Friction Angle (°)	Tensile Strength MPa
1	Mudstone	2652	2.54×10^9	1.38×10^9	1.30	30	0.43
2	Carbonaceous shale	2572	4.10×10^9	2.58×10^9	1.87	32	1.1
3	Siltstone	2596	1.09×10^{10}	8.58×10^9	17.24	43	2.29
4	Sandy mudstone	2686	4.11×10^9	3.08×10^9	7.06	35	1.00
5	Coal	1400	1.90×10^9	9.30×10^8	0.84	27	0.28
6	Fine sandstone	2730	8.18×10^9	7.28×10^9	18.13	45	2.29
7	Medium sandstone	2607	7.34×10^9	6.62×10^9	13.02	42	3.68

The length (Y) × width (X) × height (Z) of the constructed model is 400 m × 400 m × 300 m, with a total of 435,200 units and 452,709 nodes. The calculation model is shown in Figure 9. In order to eliminate the influence of boundary effect in the process of numerical calculation, the distance between the working face and the model boundary is not less than 100 m, and the buried depth of the simulated coal seam is 600 m. Two site-specific numerical models were constructed to reproduce the corresponding engineering conditions of the 4.2 m fully mechanized mining section and the 6.2 m top-coal caving section. The models were used to examine whether the measured differences were mechanically consistent with the combined mining configurations. They were not designed as a controlled numerical experiment for separating the effects of mining height and mining method. The model height is 300 m and the simulated coal seam is buried at 600 m. Therefore, the top boundary of the model corresponds to a depth of 300 m below the surface. The 300 m of overburden above the model top was replaced by an equivalent vertical stress of 7.5 MPa, calculated as $300 \text{ m} \times 25 \text{ kN/m}^3$. Together with the self-weight of the 300 m model, the initial vertical stress at the coal seam level is approximately 15 MPa, consistent with a burial depth of 600 m. The mechanical parameters of coal and rock mass are shown in Table 5, and fix commands are applied around and at the bottom of the model to fix the boundary. The initial geostress field was calibrated based on in situ stress measurements in the mining district. At the current mining depth of 600 m, the vertical geostress is predominantly gravitational ($\sigma_v \approx 15.0 \text{ MPa}$), and regional tectonic stress is relatively relaxed. Given that the coal seam has an average dip angle of approximately 5° (near-horizontal seam) and lacks major fault structures, the horizontal in situ stresses were characterized primarily by Poisson lateral expansion ($\sigma_h \approx \frac{\nu}{1-\nu} \sigma_v$), yielding calibrated isotropic lateral pressure coefficients of $\lambda_x = \lambda_y = 0.5$ for the baseline model.

In deep mining engineering, horizontal stress anisotropy ($\lambda_x \neq \lambda_y$, or $\sigma_H > \sigma_v > \sigma_h$) can alter the spatial distribution of roof shearing and tensile cracking. When the maximum horizontal stress (σ_H) is oriented perpendicular to the face advance, horizontal confinement suppresses vertical tensile fracture propagation, potentially reducing the fractured zone height while expanding lateral bed-separation; conversely, when σ_H aligns parallel to face advance, stress concentration ahead of the face intensifies shear yielding. While the baseline isotropic setting captures the governing vertical unloading response, full-tensor tectonic anisotropy will be investigated in subsequent studies. The excavation step is 20 m, and excavation is carried out a total of 5 times. The FLAC3D model used in this study is a purely mechanical elasto-plastic model and does not include fluid-flow coupling or permeability evolution. The simulated plastic zone height is used only as a mechanical upper-bound indicator of mining-induced damage. Its proximity to the field-inferred fractured zone height indicates mechanical consistency, but does not demonstrate hydraulic connectivity.

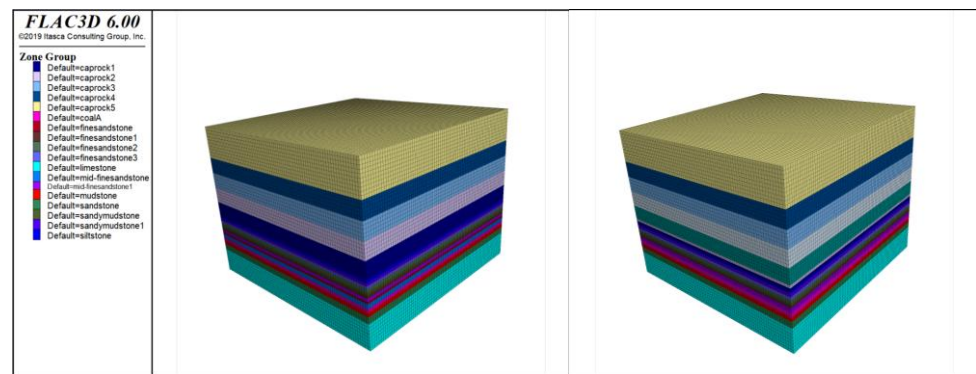


Figure 9. Three-dimensional site-specific numerical models: **(left)** the 4.2 m fully mechanized mining section; **(right)** the 6.2 m fully mechanized top-coal caving section.

4.2. Plastic Zone Development in the Two Site-Specific Numerical Models

The distribution of plastic zone during excavation is shown in Figures 10 and 11. Under the two mining methods of fully mechanized mining and fully mechanized top-coal caving, there are obvious differences in the development time sequence, spatial morphology and failure mechanism of the plastic zone of roof. Under the condition of fully mechanized mining, at the stage of 0–20 m excavation, the plastic zone of roof only develops locally along the strip near the cut hole. After 20 m excavation, the plastic zone extends upward and to both sides, forming a shallow arch failure zone with “low in the middle and high at the edge”, and the top rock mass appears as shear tensile composite failure. In the middle stage, the height of the plastic zone increased significantly and distributed in a “funnel shape”. In the later stage, it continues to transfer to the high layer, and the final development height is about 43 m, showing an “inverted funnel” structure. In the 6.2 m top-coal caving model, the combined effects of greater extraction thickness, a larger mining void, and the corresponding unloading intensity produced a more rapid upward expansion of the roof plastic zone: at the stage of 40 m excavation, the overlying rock appears as obvious plastic failure in a “basin” arch shape, and the failure is concentrated in the mudstone and sandy mudstone layer 15–20 m above the coal seam. When the excavation is 60 m, the height of the plastic zone is more than 50 m, showing a “funnel-shaped upward convergence” structure. When the excavation is 100 m, the plastic zone develops to 66 m above the roof, presenting a “bell-shaped” structure. The central rock mass is completely collapsed and broken, and the tensile failure proportion is significantly higher than that of fully mechanized mining. Overall, the 6.2 m top-coal caving model developed a higher and wider plastic zone than the 4.2 m fully mechanized mining model. This difference provides a possible mechanical explanation for the different damage extents observed between the two engineering sections. However, it represents the response of the combined site-specific configurations and should not be interpreted as the independent effect of mining method, which provides a possible mechanical explanation for the different damage extents observed between the two engineering sections. The plastic zone height was used only as a mechanical upper-bound indicator of the mining-induced fractured zone. To relate it to the water-conducting fractured zone, the simulated plastic zone height was calibrated against the field-measured and ground-borehole-confirmed fractured zone heights.

Not all plastic yielding contributes to hydraulic conductivity. In this study, the water-conducting zone was considered to correspond mainly to the region in which tensile plastic failure and interconnected shear failure developed, whereas isolated plastic zones without connection to the goaf were not interpreted as water-conducting. The simulated plastic zone height was therefore treated as an upper bound, and the connected water-conducting height was determined by the field and ground-borehole observations. In

terms of mining degree, while full subsidence of deep overburden typically requires extensive strike advance, the vertical propagation of the water-conducting fractured zone tends to reach mechanical saturation much earlier than full ground subsidence. As the face advanced from 60 m to 100 m (in five 20 m increments), the incremental vertical expansion of the plastic zone dropped substantially (less than 5% between 80 m and 100 m), demonstrating satisfactory asymptotic convergence for capturing the peak fractured zone height under the given panel boundary conditions.

A comparative evaluation indicates that the simulated plastic zone heights are slightly larger than the field DOFS measurements (43.0 m vs. 41.4 m under fully mechanized mining, a 3.86% discrepancy; 66.0 m vs. 58.0 m under top-coal caving, a 13.79% discrepancy). From a geomechanical perspective, this variation is attributed to two factors: (1) Physical definition distinction: FLAC3D plastic zones capture all continuum yield elements including micro-plastic dissipation, whereas DOFS identifies macro-scale through-going fractures and layer delamination ($\epsilon \geq 1500 \times 10^{-6}$). The uppermost boundary in FLAC3D naturally contains a non-interconnected micro-yield buffer zone. (2) Model assumptions: Under the intensive unloading of 6.2 m top-coal caving, the linear Mohr–Coulomb model without strain softening tends to produce diffuse boundary plasticity in high-level strata. In deep mining rock mechanics (600 m depth), a relative error below 15% falls well within the acceptable engineering tolerance, successfully validating the comparative overburden failure trends.

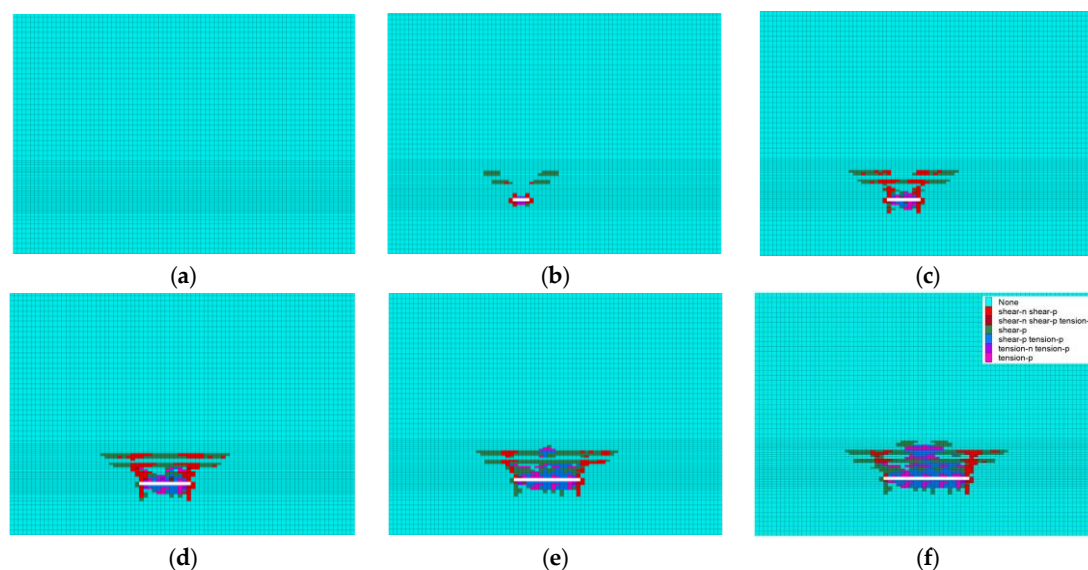
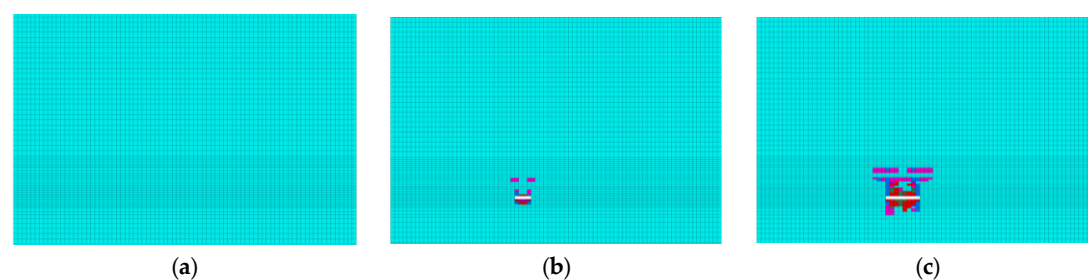


Figure 10. Plastic zone evolution in the 4.2 m fully mechanized mining model after excavation advances of (a) 0 m; (b) 20 m; (c) 40 m; (d) 60 m; (e) 80 m; and (f) 100 m.



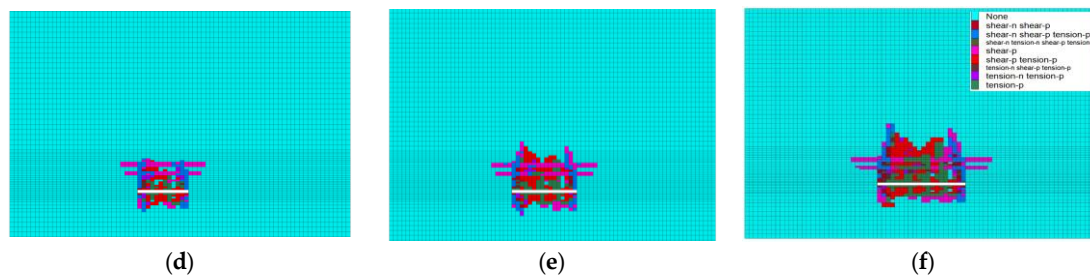
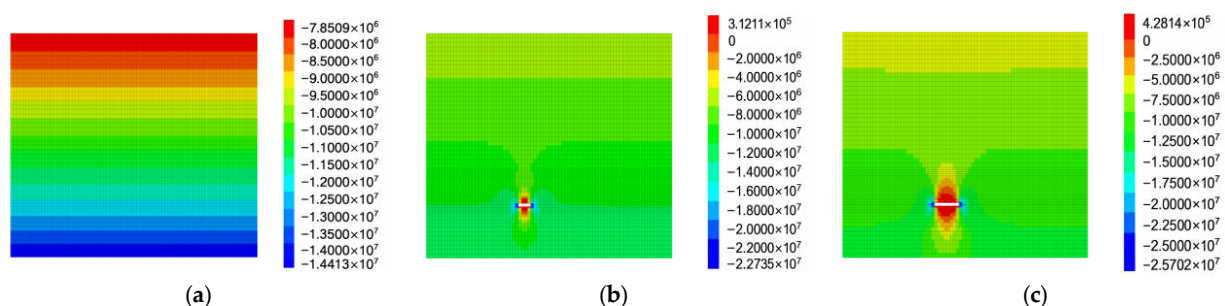


Figure 11. Plastic zone evolution in the 6.2 m fully mechanized top-coal caving model after excavation advances of (a) 0 m; (b) 20 m; (c) 40 m; (d) 60 m; (e) 80 m; and (f) 100 m.

4.3. Vertical Stress Distribution in the Two Site-Specific Numerical Models

The vertical stress distribution during excavation is shown in Figures 12 and 13. Under the condition of fully mechanized mining and fully mechanized top-coal caving, the vertical stress distribution of the roof is obviously different in the development process and influence range. In the initial state, the vertical stresses of the two models increase with depth in layers, and the isolines are horizontally distributed, without mining disturbance. The initial vertical stress level of the fully mechanized top-coal caving model is slightly higher, the maximum compressive stress is about 15.1 MPa, and the fully mechanized mining model is about 14.4 MPa. After 20 m excavation, a small range of tensile stress (unloading) concentration area appears directly above the goaf, and a high concentration of compressive stress appears at the edge of the cut and working face, and the mining stress begins to transfer to the roof. When the excavation is 40 m, the tensile stress zone expands obviously, and the compressive stress concentration zone at the edge increases synchronously. When the excavation reaches 60 m, the tensile stress zone in the upper part of the roof extends further upward, and the influence range of mining continues to expand. At the stage of 80–100 m, the tensile stress zone continues to extend and widen, showing an arched upward convergence shape. The compressive stress concentration zone migrates synchronously with the advance of the working face, and the vertical stress influence zone extends laterally as a whole. By comparison, it can be seen that the single unloading space of fully mechanized mining is small, the stress adjustment is relatively gentle, and the roof tensile stress zone is narrow and extended slowly. The fully mechanized top-coal caving has large mining space, intense unloading, stronger vertical stress redistribution, faster development, larger range and higher upward extension of the tensile stress zone. At 100 m, the area of the roof tensile stress zone is about 1.4 times that of the fully mechanized mining, and the compressive stress concentration is also higher. To sum up, the disturbance range of the vertical stress of the roof caused by fully mechanized top-coal caving is significantly larger than that of the layered fully mechanized mining, and the height and width of the tensile stress zone are larger, which is consistent with the law that the heights of the plastic zone and the water-conducting fractured zone are larger.



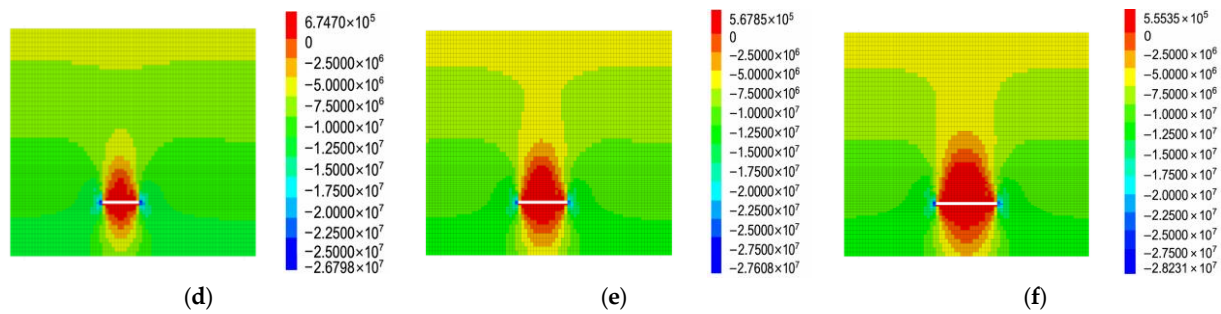


Figure 12. Vertical-stress evolution in the 4.2 m fully mechanized mining model after excavation advances of (a) 0 m; (b) 20 m; (c) 40 m; (d) 60 m; (e) 80 m; and (f) 100 m.

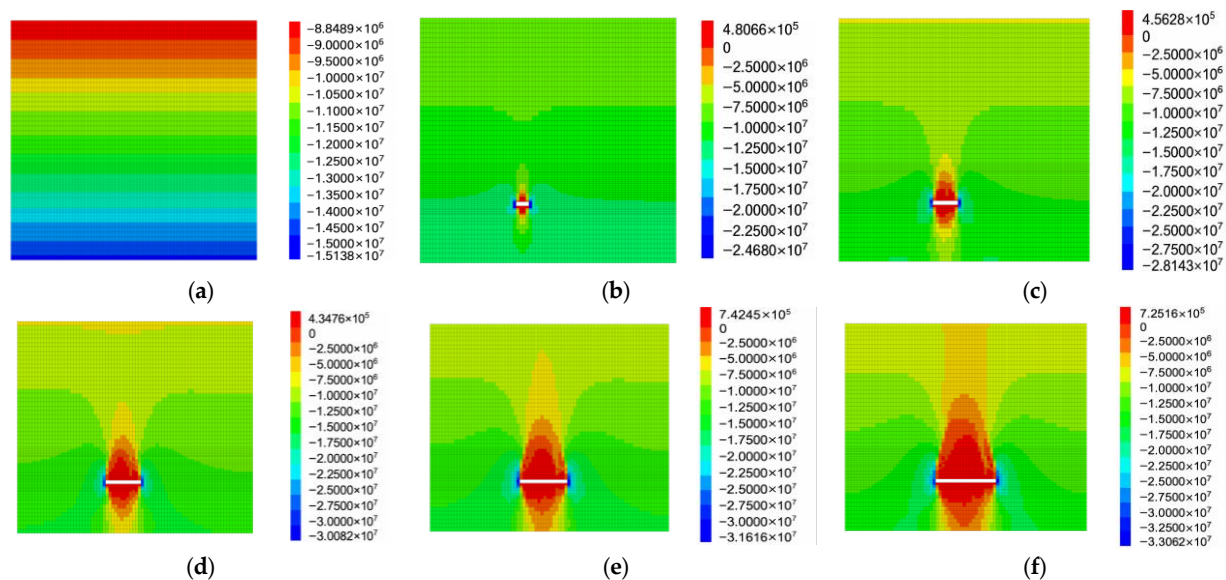


Figure 13. Vertical-stress evolution in the 6.2 m fully mechanized top-coal caving model after excavation advances of (a) 0 m; (b) 20 m; (c) 40 m; (d) 60 m; (e) 80 m; and (f) 100 m.

4.4. Vertical Displacement Distribution in the Two Site-Specific Numerical Models

The vertical displacement distribution is shown in Figures 14 and 15. Under the conditions of fully mechanized mining and fully mechanized top-coal caving, the vertical displacement of the roof is significantly different in the development process, spatial form and quantity. In the initial state, the displacement of the rock mass of the two models is zero as a whole, which is in a stable equilibrium state. Under the condition of fully mechanized mining, at the stage of 20 m excavation, the rock mass above the working face began to display the displacement dominated by subsidence, which was a semi oval distribution with the working face as the center, and the roof strata bent and subsided. At the stage of 40 m excavation, the displacement area expands upward, and there is obvious interlayer separation, which is funnel-shaped and extends laterally. At 60 m stage, the influence height of displacement increases significantly, and the displacement of settlement center increases significantly. At the stage of 80–100 m, the influence range of displacement is further expanded, and finally the caving zone, fractured zone and bending subsidence zone are formed. The roof is in the shape of subsidence funnel, and the maximum settlement of the upper boundary of the model is about 0.28 m. Under the condition of fully mechanized top-coal caving, at the stage of 20 m excavation, the subsidence is concentrated in the mudstone and sandy mudstone layer 10–15 m above the goaf, which is an oval depression, and the displacement is less than 0.1 m; at the stage of 40 m, the oval subsidence core area is formed, which is an arched subsidence profile; in the 60 m stage, the subsidence range extends to the height of 35–50 m, showing an inverted funnel shape. The displacement of the settlement

core reached 0.2–0.3 m at the stage of 80 m, and the influence height extended to more than 60 m. A typical “bell-shaped” settlement zone is formed at the stage of 100 m, with a maximum settlement of about 0.30 m. Compared with the 4.2 m model, the 6.2 m top-coal caving model exhibited more rapid displacement development, greater settlement, and a larger vertical influence range. The settlement pattern evolved from an elliptical depression to an arched and subsequently bell-shaped profile.

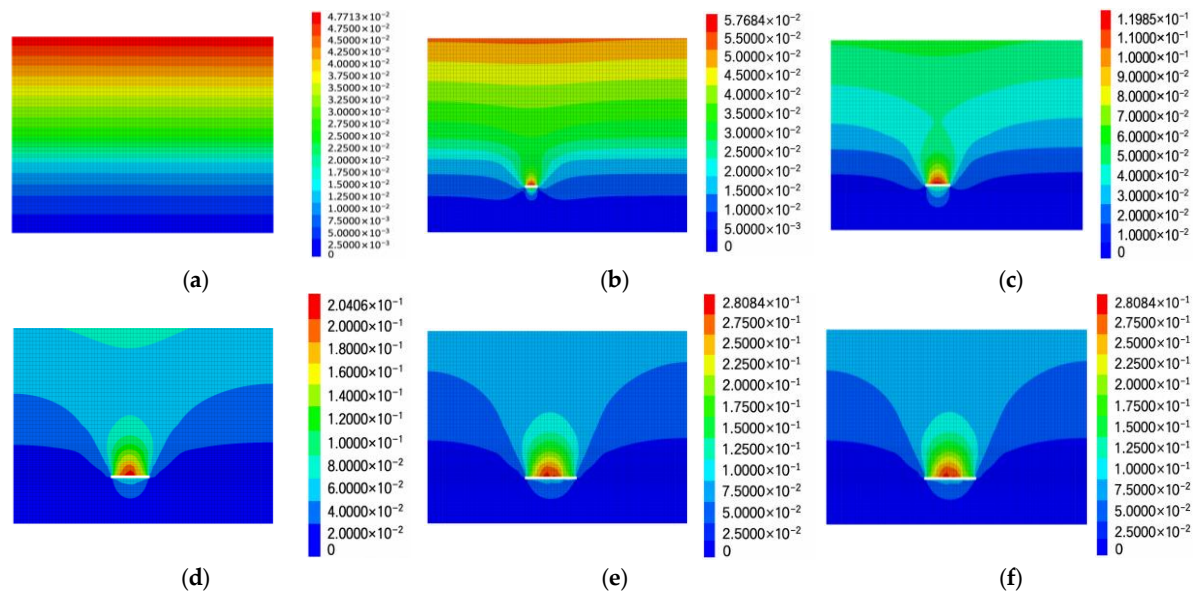


Figure 14. Vertical-displacement evolution in the 4.2 m fully mechanized mining model after excavation advances of (a) 0 m; (b) 20 m; (c) 40 m; (d) 60 m; (e) 80 m; and (f) 100 m.

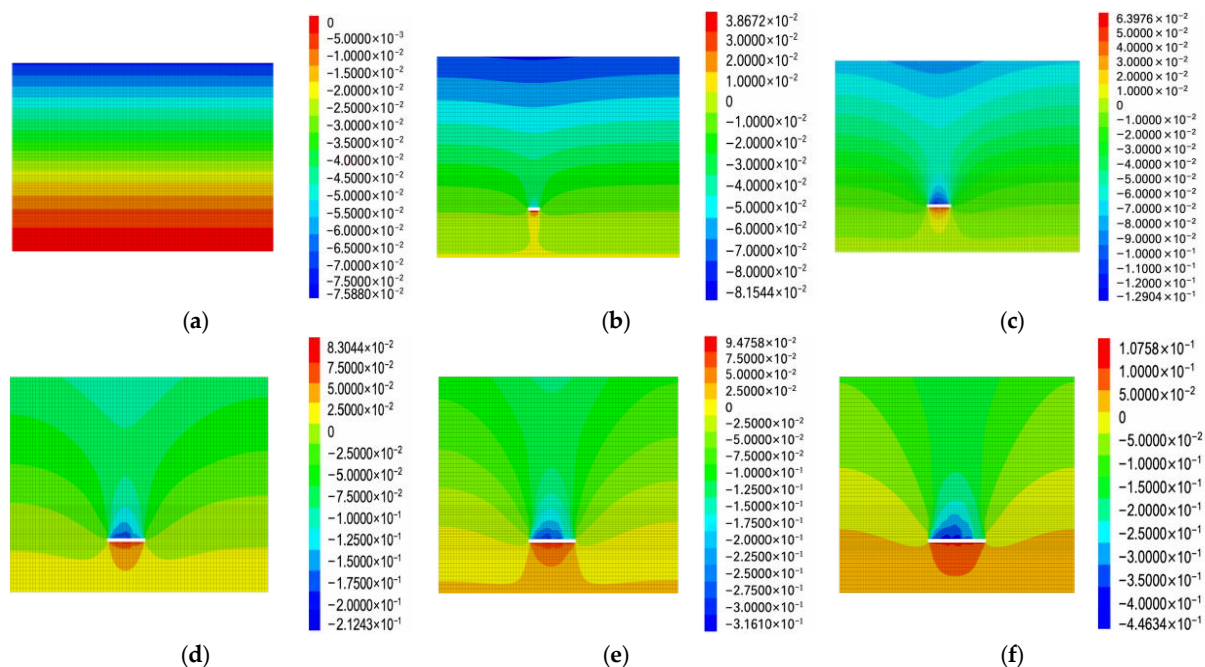


Figure 15. Vertical-displacement evolution in the 6.2 m fully mechanized top-coal caving model after excavation advances of (a) 0 m; (b) 20 m; (c) 40 m; (d) 60 m; (e) 80 m; and (f) 100 m.

4.5. Monitoring-Point Displacement Responses in the Two Site-Specific Numerical Models

At the same time, 18 monitoring points are set on the two models, including 9 monitoring points on the roof of fully mechanized mining face and 9 monitoring points on the

fully mechanized top-coal caving face, which are respectively located on the section of $x = 200$ m. The distribution of monitoring points is shown in Figure 16.

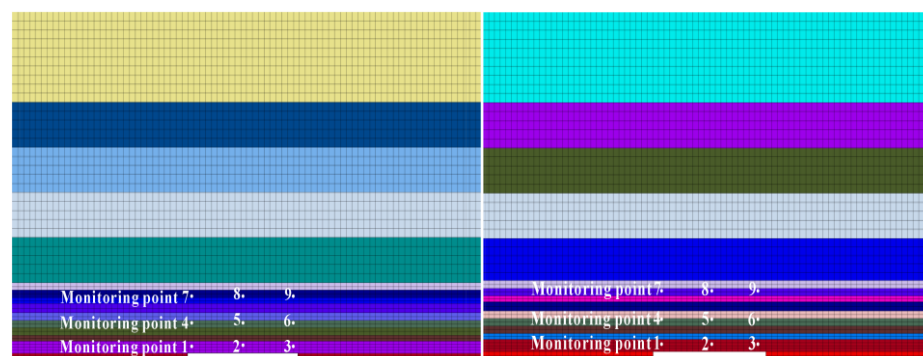


Figure 16. Monitoring point setting. (Left) Fully mechanized mining; (right) fully mechanized top-coal caving.

In order to further quantitatively compare the displacement responses of different failure zones of overlying rock under the two mining methods, Figures 17 and 18 show the evolution curves of displacement of monitoring points in fully mechanized mining and fully mechanized top-coal caving areas with excavation steps. Nine monitoring points are respectively arranged on the $x = 200$ m section of the two models, which are located at three elevations, 20 m, 40 m and 60 m away from the roof of the coal seam. Each elevation takes three positions at the center and both sides of the goaf along the width direction of the stope. In the fully mechanized mining area, the 20 m and 40 m elevations are located in the proper fractured zone, i.e., above the caving zone and below the upper boundary of the water-conducting fractured zone; in the fully mechanized top-coal caving area, the 20 m elevation is still within the caving zone, while the 40 m elevation lies in the fractured zone proper; the 60 m elevation lies in the bending subsidence zone in both areas. The monitoring results show that with the increase in excavation step, the continuous downward settlement occurs at all monitoring points, and the displacement response shows the characteristics of spatio-temporal differentiation of “large at the bottom and small at the top, large at the center and small at both sides”. The settlement above the center of the goaf at the same elevation is the largest and decreases to both sides. At different elevations, with the increase in the distance from the coal seam roof, the displacement value gradually decreases, and the response gradually lags behind, reflecting the attenuation and transmission of mining influence from the caving zone through the water-conducting fractured zone to the bending subsidence zone layer by layer.

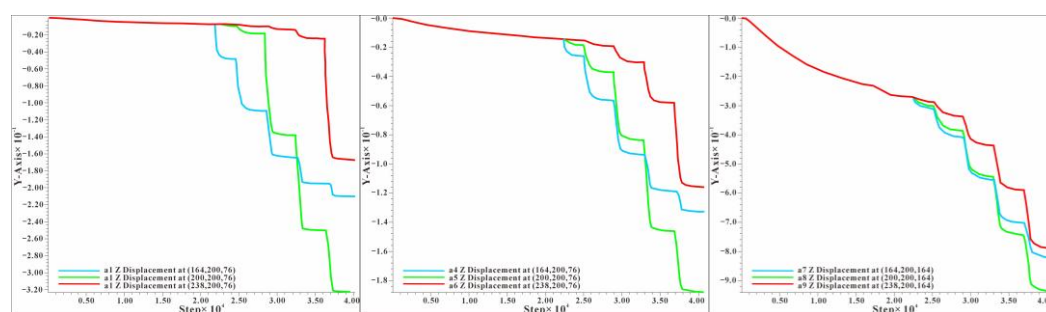


Figure 17. Displacement of monitoring points 1–9 in fully mechanized mining area.

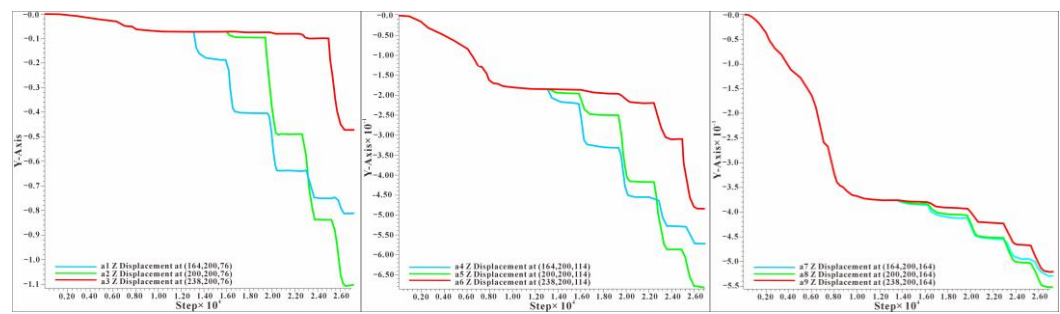


Figure 18. Displacement of monitoring points 1–9 in fully mechanized top-coal caving area.

Comparing the two mining methods, the displacement difference is the most significant at 20 m elevation. The monitoring points at 20 m in the fully mechanized mining area are located in the water-conducting fractured zone, and the rock mass is dominated by fracture development and bending separation, with relatively limited displacement. The elevation of the fully mechanized top-coal caving area is still in the caving zone, the rock mass has collapsed and broken, and the displacement is significantly greater, which directly reflects the damage difference in the fully mechanized top-coal caving zone is significantly greater than that of the layered fully mechanized mining. The 40 m elevation is located in the water-conducting fractured zone, and the displacement of the fully mechanized top-coal caving area is still greater than that of the fully mechanized mining, indicating that the fractured zone is more fully developed and the influence range is larger. The 60 m elevation all entered the bending subsidence zone, and the displacement value and difference decreased and tended to be consistent, indicating that the mining disturbance has gradually attenuated, and the influence of the two mining methods in the shallow roof converged.

Accordingly, the vertical gradient of displacement at monitoring points quantitatively depicts the zonal characteristics of roof failure from the perspective of deformation field: the displacement value from the caving zone to the water-conducting fractured zone and the bending subsidence zone decreases step by step, and the attenuation starting point of the fully mechanized top-coal caving area is higher and the attenuation amplitude is larger, which is confirmed by the difference between the plastic zone (about 66 m for fully mechanized top-coal caving and about 43 m for fully mechanized mining) and the height of the water-conducting fractured zone (58.0 m for fully mechanized top-coal caving and 41.4 m for fully mechanized mining). This shows that the vertical displacement gradient can be used as an auxiliary criterion to divide the caving zone, water-conducting fractured zone and bending subsidence zone, and provides a quantitative basis for optimizing the mining technology of deep near-horizontal and thickened coal seam, controlling the development height of the water-conducting fractured zone and preventing roof water damage.

5. Discussion

Based on the field DOFS measurements and site-specific numerical simulations, this study compares the roof damage responses observed in two sections of the same variable thickness working face under their actual engineering configurations. The observed differences between the two sections should be interpreted as the combined response to mining height, extraction configuration, mining intensity, and local geological variability. The mining height increased from 4.2 m in the fully mechanized mining section to 6.2 m in the top-coal caving section, representing an increase of approximately 47.6%. Meanwhile, the fractured zone height increased from 41.4 to 58.0 m, corresponding to an increase of approximately 40.1%. The resulting fracture-to-mining ratios were approximately 9.86 and

9.35, respectively. The similarity of these normalized ratios indicates that the greater absolute fractured zone height in the top-coal caving section is largely associated with the increase in mining height.

In contrast, the caving-to-mining ratio increased from approximately 3.2 in the 4.2 m section to approximately 4.1 in the 6.2 m top-coal caving section. The top-coal caving section also exhibited a greater leading influence distance, higher peak tensile strain, progressive failure along the borehole, and a broader bell-shaped plastic zone. These differences suggest that the larger extraction space and stronger unloading associated with the top-coal caving configuration may have intensified near-field roof collapse and modified the failure morphology. However, because mining height and mining method changed simultaneously, their individual contributions cannot be quantitatively separated. The present results therefore represent a comparison of two actual engineering configurations rather than a controlled assessment of the independent effect of mining method.

From the perspective of the origin of mechanical mechanisms, the difference in roof damage induced by fully mechanized mining and fully mechanized top-coal caving mainly stems from the spatial effects of the mining site and the different rates of stress unloading. Under the conditions of fully mechanized mining, the single unloading space is relatively limited, and the roof rock layer undergoes progressive bending, subsidence, and fracture under the action of advanced support pressure. This progressive failure mode is consistent with the latest mechanical evolution research, which indicates that the overlying strata above the goaf mainly exhibit bending stretch failure, forming distinct suspended rock layers and fractured zones [26]. This gentle stress adjustment results in a narrower tensile stress zone above the goaf and a slower upward extension. The overlying rock failure occurs in a layer-by-layer transmission mode, and the plastic zone ultimately presents a shallow arch shaped expansion characteristic of an inverted funnel. In comparison, the 6.2 m top-coal caving configuration produced a larger extraction void and a stronger unloading response in the numerical model. The numerical results show that the 6.2 m top-coal caving configuration was associated with a wider tensile stress zone and stronger compressive stress concentration near the working-face edge than the 4.2 m configuration. These differences are mechanically consistent with the combined effects of greater mining height, extraction configuration, and unloading intensity, but they cannot be attributed to the mining method alone.

From the perspective of composite roof lithology and key strata theory, the vertical propagation path of mining-induced fractures is fundamentally governed by the structural alternation of competent and weak rock units (Table 1). The immediate roof is composed of weak sandy mudstone (average 1.2 m), while the main roof comprises a thick, competent fine sandstone layer (average 24 m) with high stiffness and tensile strength (2.29 MPa). Under fully mechanized mining (4.2 m), the modest caving space induces coordinated, gentle bending of the thick sandstone key stratum; the weak bedding interfaces between mudstone and sandstone absorb tensile strains primarily through horizontal bed separation (delamination), effectively inhibiting vertical crack penetration and arresting the fractured zone at 41.4 m. In the 6.2 m top-coal caving section, the combined influence of greater extraction thickness, the larger mining void, the caving configuration, and local roof lithology is consistent with stronger pressure relief beneath the main sandstone unit. This combined response may have promoted more extensive shear–tensile damage and upward fracture propagation into the overlying weak mudstone units, resulting in the locally inferred height of 58.0 m.

The measured data of on-site distributed fiber-optic sensing technology and the FLAC3D numerical simulation results show broadly consistent trends, thereby supporting the proposed mechanical interpretation. In actual testing, the strain of optical fibers in

the fully mechanized mining area evolved through four stages—stable response, slow increase, continuous increase, and abrupt change—and the tensile strain surged to 6981×10^{-6} – 9081×10^{-6} before collapse. The strain response in the fully mechanized top-coal caving area exhibits characteristics of large leading influence distance and progressive collapse along the length of the hole, with more severe tensile failure and a peak tensile strain of 9576×10^{-6} . The significant difference in monitoring data is perfectly matched with the simulated plastic zone height and settlement zone displacement field. This conclusion is also consistent with the existing research on the violent movement of overlying strata in deep and thick coal seam mining, confirming that high-intensity unloading is the core cause of accelerating the expansion of fractures to high-level rock layers. The spatial representativeness of the borehole observations should be considered when interpreting the results. The boreholes were approximately aligned with the direction of face advance, allowing the temporal development of roof deformation to be captured as the face passed the monitored locations. However, each borehole pair samples only one local vertical section of the working face and does not resolve the complete three-dimensional variation in fracture development along the strike and dip directions.

Accordingly, the values of 41.4 m and 58.0 m represent local DOFS-inferred upper boundaries of fractured overburden at the monitored thin- and thick-seam sections, respectively. The 41.4 m value indicates a potentially water-conducting fractured zone without independent hydraulic verification, whereas the 58.0 m value is supported by surface-borehole drilling-fluid-loss observations. The paired boreholes provide local cross-checking of the strain evolution and zonal interpretation, but they cannot eliminate uncertainty caused by variations in roof lithology, key-stratum thickness and elevation, small-scale geological structures, or local mining-pressure conditions. This limitation is particularly relevant to the top-coal caving section, where strong pressure relief and progressive collapse may produce greater spatial variability. Additional boreholes distributed along the strike and dip directions, or complementary microseismic and three-dimensional geophysical monitoring, would be required to quantify panel-scale heterogeneity.

Based on the actual geological conditions of the 14,321 working face in Gubei Coal Mine, the case-specific findings provide a practical reference for fractured zone assessment and water hazard prevention in deep, near-horizontal, variable thickness coal seams. In the engineering practice of near-horizontal and variable thickness coal seams, the implementation of segmented differentiated fully mechanized mining and fully mechanized top-coal caving dual mining technology is not simply limited by the natural changes in coal seam thickness, but is aimed at scientifically regulating the degree of roof damage and fracture development range, thereby safely and effectively improving the upper limit of mining. Specifically, the working face adopts strict control of mining geometric parameters for different areas: in the thin coal area with a coal seam thickness of 4.2 m, fully mechanized mining technology is used; in the thick coal area where the coal seam thickens to 6.2 m, it is dynamically adjusted to fully mechanized top-coal caving.

Through this refined design of cutting height and coal discharge ratio, the working face effectively limits the spatial scale and roof disturbance intensity of a single mining operation when using the fully mechanized top-coal caving process. The field measurement finally shows that under the control of the above precise parameters, the vertical height of the water-conducting fractured zone in the thin coal area has been successfully suppressed at 41.4 m, and the vertical height of the water-conducting fractured zone in the thick coal area has also been stabilized at 58.0 m. This engineering case provides field evidence that in areas with high water inrush prevention requirements, such as facing complex roof conditions or approaching aquifers, by optimizing the mining process and setting mining parameters in a refined way, the roof damage caused by strong disturbance

in large space can be effectively weakened, while achieving efficient recovery of coal resources, the safety of improving the upper limit of mining can be effectively guaranteed, and scientific water conservation mining can be realized.

It is acknowledged that advancing the face to 200–300 m under varying geological parameters would provide deeper insights into supercritical mining states. However, this engineering study focused on validating the immediate monitoring array; the 100 m simulation accurately reproduces the critical period of intense overburden rupture and stabilizes the primary plastic zone boundary. Comprehensive parametric sensitivity tests and extended-distance modeling will be addressed in future work.

6. Conclusions

This study combined field DOFS monitoring and site-specific FLAC3D simulations to characterize roof failure in the 4.2 m fully mechanized mining section and the 6.2 m top-coal caving section of the 14321 working face. The principal conclusions are as follows:

(1) The DOFS measurements captured distinct temporal and spatial roof responses in the two monitored sections. In the 4.2 m section, the strain evolved through four stages: stable response, slow increase, continuous increase, and abrupt change. A compression-to-tension reversal was followed by tensile-strain peaks of 6981–9081 $\mu\epsilon$, indicating substantial roof collapse. In the 6.2 m section, the strain response showed a greater leading influence distance and progressive failure from the deeper part of the borehole toward the borehole opening, with a maximum tensile strain of 9576 $\mu\epsilon$. The corresponding caved-zone heights were 13.4–13.6 m and 25.4 m, respectively.

(2) Based on the site-specific DOFS criteria, the upper boundary of the potential water-conducting fractured zone was inferred to be 41.4 m in the 4.2 m section. In the 6.2 m section, a DOFS-inferred height of 58.0 m was obtained and was independently supported by a surface-borehole drilling-fluid-loss height of approximately 59.93 m. Because the overlying aquifers were dewatered before mining, these values represent the vertical extent of mechanically developed fractures with potential water-conducting capacity under recharged conditions rather than directly measured permeability or active water flow. The 41.4 m boundary is based primarily on the spatially continuous and persistent DOFS response and lacks independent hydraulic verification.

(3) The site-specific numerical models reproduced the main mechanical characteristics of overburden failure. The continuous plastic-damage envelopes reached approximately 43 m and 66 m in the fully mechanized mining and top-coal caving models, respectively. The former developed as a relatively shallow inverted-funnel-shaped zone, whereas the latter formed a broader bell-shaped zone with more extensive tensile and shear yielding. However, the FLAC3D models did not include fluid-flow coupling or permeability evolution. Therefore, the simulated plastic zones represent mechanical upper bounds and should not be interpreted as directly calculated water-conducting pathways.

(4) The observed differences reflect the combined effects of mining height, extraction configuration, mining intensity, roof lithology, and local geological variation. The mining height increased by approximately 47.6%, from 4.2 to 6.2 m, while the fractured zone height increased by approximately 40.1%, from 41.4 to 58.0 m. The similar fracture-to-mining ratios of 9.86 and 9.35 indicate that the greater absolute fractured zone height was largely associated with the increased mining height. In contrast, the increase in the caving-to-mining ratio from approximately 3.2 to 4.1 suggests that the top-coal caving configuration may have intensified near-field collapse. Because mining height and mining method varied simultaneously, their independent effects cannot be quantitatively separated.

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