

Mechanism and Prevention of Coal and Gas Outburst and Impact Ground Pressure Coupling Disasters in High Stress Extra - Thick Coal Seams: A Case Study

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Mechanism and Prevention of Coal and Gas Outburst and
Impact Ground Pressure Coupling Disasters in High Stress Extra
- Thick Coal Seams: A Case Study

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ABSTRACT

The interaction between coal and gas outburst and rock burst is a severe dynamic hazard in deep, high-stress mining of extra-thick coal seams. Using Weijiadi coal mine as a field case, we combine in-situ measurements, FLAC3D simulations and multi-source monitoring to examine how the coupled hazard develops and how it can be controlled. The measured stress field is dominated by horizontal stress; the maximum principal stress is 39.79 MPa. As the working face advances, stress in the coal pillar increases and reaches 59.67 MPa at the stop line, corresponding to a stress concentration factor of 4.19. Advance abutment pressure also increases. The transport-trough side, which is

adjacent to the goaf, develops the strongest concentration, with a peak value 3.88 times the original-rock vertical stress, a condition that favours gas enrichment. Fault activation further superposes mining stress: from the beginning to the end of extraction, the peak abutment pressure on the return-airway side and fault side increases by 24.5% and 20%, respectively. Mining stress varies positively with the drilling-cuttings desorption index K_1 and the maximum drilling-cuttings amount S_{max} , linking the stress and gas responses to the coupled hazard. A four-in-one prevention and control system, including fan-shaped-hole gas pre-drainage and static-pressure coal-seam water injection, reduces gas emission from the working face by 32% and keeps the upper-corner gas concentration below 1.0%. These observations provide an evidence base for assessing and mitigating coupled disaster risk in stress-concentrated zones of extra-thick coal seams.

Keywords: High stress extra - thick coal seams; coal and gas outburst; rock burst; coupling disasters; prevention measures

Highlights

- (1) Clarified the incubation system and stress field characteristics of coupled disasters.
- (2) Revealed the stress evolution law and the controlling effect of faults during the mining process.
- (3) Clarified the stress gas synergistic disaster mechanism of coupled disasters.
- (4) Verified the effectiveness of comprehensive prevention and control measures.

1 Introduction

As the cornerstone of China 's energy structure, safe and efficient mining of coal is crucial to national energy security and social stability ¹⁻³. However, the high ground stress, high gas pressure, complex geological structure and other extreme environments associated with deep mining make the dynamic disasters of coal mines show unprecedented complexity, coupling and destruction. The coupling disaster of coal and gas outburst and rock burst in high stress and extra-thick coal seam has become a major scientific and technological problem and safety bottleneck in the development of deep coal resources because of its suddenness, strong destructiveness and unpredictability. It poses a great threat to the safety of miners and causes serious economic losses and social impact ^{4,5}.

Coal and gas outburst is essentially the result of the combined action of ground stress, gas pressure and physical and mechanical properties of coal. It is a catastrophic dynamic phenomenon that high pressure gas breaks through the resistance of coal body in a short time, accompanied by a large number of coal and rock suddenly thrown into the mining space ⁶⁻⁸. Rock burst is mainly the sudden and violent release of elastic strain energy accumulated in coal and rock mass under high stress state, which leads to the instantaneous failure, ejection and strong vibration of coal and rock mass ⁹⁻¹¹. With the increase of mining depth, the engineering environment in the extra thick coal seam has undergone a qualitative change. High in-situ stress not only significantly improves the ability of coal and rock mass to store elastic strain energy as the dominant factor, but also provides a ' power source ' for the inoculation of rock burst. At the same time, it

greatly compresses the primary pores and fissures of coal body, reduces its permeability, makes gas migration difficult, and is easy to form abnormal high gas pressure area in local enrichment¹²⁻¹⁴. The application of graded optimized hydraulic fracturing technology in hard rock roof of goaf in extra thick coal seam, combined with microseismic monitoring and other technologies, effectively reduces the stress concentration.¹⁵ proposed the “key layer fracturing” technology to effectively solve the deformation problem of cross mining roadway by revealing the stress propagation mechanism of floor and the evolution law of surrounding rock plastic zone.¹⁶ studied the correlation mechanism of stress pore structure gas adsorption/desorption characteristics outburst risk for coal and gas outburst risk in roadway in non-uniform stress area.¹⁷ revealed the negative coupling law of gas emission and mining stress, and took the gas concentration ($\geq 0.05\%$) as the early warning index of rock burst, so as to realize "pressure detection by gas", and open up a new path for the prevention and control of dynamic disasters in deep high gas mines.¹⁸ clarified the synergetic disaster causing mechanism of static stress field and mining vibration load formed by roof clamping and rotation in steep and extra thick coal seam.¹⁹ proposed an intelligent classification and early warning system for rockburst based on meshing analysis of microseismic data and xgboost algorithm to achieve accurate positioning of roadway dangerous areas and quantification of risk levels. Scholars have carried out research on the disaster causing mechanism of coal and gas outburst and rock burst in ultra thick coal seams, as well as the stress gas coupling relationship, early warning indicators and prevention and control technology. However, the whole process of dynamic coupling

evolution of the two disasters under complex mining conditions and the multi field and multi-scale interaction mechanism are still not systematically revealed, and the real-time accurate early warning and dynamic prevention and control technology system for the coordinated occurrence of the two disasters still needs to be improved.

In recent years, scholars have also conducted in-depth research on the deep coupling effect of coal and gas outburst rockburst in the whole process from incubation, trigger to evolution, and its internal physical and mechanical mechanism, multi field spatio-temporal evolution and synergistic disaster causing mechanism²⁰⁻²³ revealed the stress gas two-way coupling catastrophe mechanism of deep high gassy coal seams. The high ground stress is not only the “ trigger source ” of rock burst, but also the “ invisible barrier ” of gas outburst.²⁴ established a three-dimensional energy analytical model of coal containing gas, and quantitatively revealed the energy control mechanism of gas pressure on coal destruction by introducing pore pressure to reconstruct the energy calculation system.²⁵ quantitatively established the mathematical model of “ dynamic damage → pore amplification → static strength attenuation ” for the first time through the true triaxial dynamic impact nuclear magnetic resonance quasi-static loading experimental chain, and solved the quantitative problem of damage evolution of coal under complex stress.²⁶ used simulation experiments to restore the real outburst process, revealed the dual dominant energy release mechanism of “ elastic energy+gas enthalpy ” triggered by in-situ stress, and established an energy driven outburst intensity prediction model.²⁷ revealed the three-stage development law of soft rock overburden fractures through multi-method fusion, established the quantitative relationship of

"mining stress dissipation energy fracture height", and cracked the disaster mechanism of overburden mining in medium thick coal seam. The above research mainly focuses on the coupling effect of stress, gas and energy in deep and specific conditions of coal seams, revealing the trigger conditions and evolution laws of complex dynamic disasters in deep and high gassy coal seams, but it still faces challenges such as the lack of universality of multi factor dynamic models, the lack of full-scale experimental verification, and the difficulty of engineering transformation of disaster criteria.

In view of this, this paper takes the coupling disaster of coal and gas outburst and rock burst in high stress and extra-thick coal seam as the research object, adopts the case study method, selects representative mine disaster cases, and focuses on the occurrence conditions, evolution law and coupling mechanism of coupling disaster through the combination of field investigation, theoretical analysis, numerical simulation and field monitoring. The research results can enrich the theoretical system of dynamic disasters in deep mines, provide scientific basis for accurate prediction and effective prevention and control of coupling disasters in high-stress and extra-thick coal seams, and have important theoretical value and engineering guiding significance for ensuring the safe and efficient mining of coal resources.

2 Overview of Study Area

2.1 Mine Overview

Weijiadi coal mine lies in eastern Pingchuan District, Baiyin City, China, approximately 7 km from the reference location. It forms part of the Weijiadi minefield in the eastern Baojishan mining area of the Jingyuan coalfield. The minefield extends

7.7 km along strike and 2.8 km down dip, covering 20.9941 km². The western boundary follows exploration line IX and the F₄₆ fault adjoining the Dashuitou and Baojishan minefields, whereas the eastern boundary terminates at exploration line XIX. To the south, the coal seam meets the F₁₋₂ overthrust fault; to the north, the boundary is defined by the intersection of the F₄₆ fault and the coal seam. Both folds and faults occur in the minefield. The principal faults trend NW-NWW, and the Baojishan syncline has a comparable axial direction, giving the area moderate structural complexity. The minefield structure is shown in Fig. 1a [27,28]. F₁₋₂, F₄₆ and F₃ faults impose compressional and torsional deformation on the northern and southern margins. Where structures approach roadways, roof falls may be accompanied by water and sediment inflow, complicating construction. The roof-fall material is loose, fractured breccia and is commonly associated with small faults. These fracture zones are generally about 0.8 m wide and wedge-shaped. The rose diagram in Fig. 1b shows that the small faults are mainly NE-trending and are mostly transtensional or transpressional in character; their angle to the F₁₋₂ faults is consistent with the combined action of NE-SW compression and NW tension.^{27,28}

The East 1100 working face is situated in the upper part of the East one mining area, within the influence zone of the F₁₋₂ overthrust fault. The seam has a monoclinic geometry and a complex structural setting. Its strike length is 1020 m and its dip width is 130 m, giving an area of 132340 m². Seam 1 is the principal minable seam. It is black, scaly and powdery, with well-developed endogenous fractures and an asphaltic lustre, and is classified as a coal and gas outburst seam. Seam thickness increases

southeastward along strike and decreases northeastward down dip. The total thickness ranges from 9.41 m to 24.84 m, with a mean of 18.38 m and a coal-content coefficient of 27.5%. The columnar section of Seam 1 is provided in Fig. 1c.

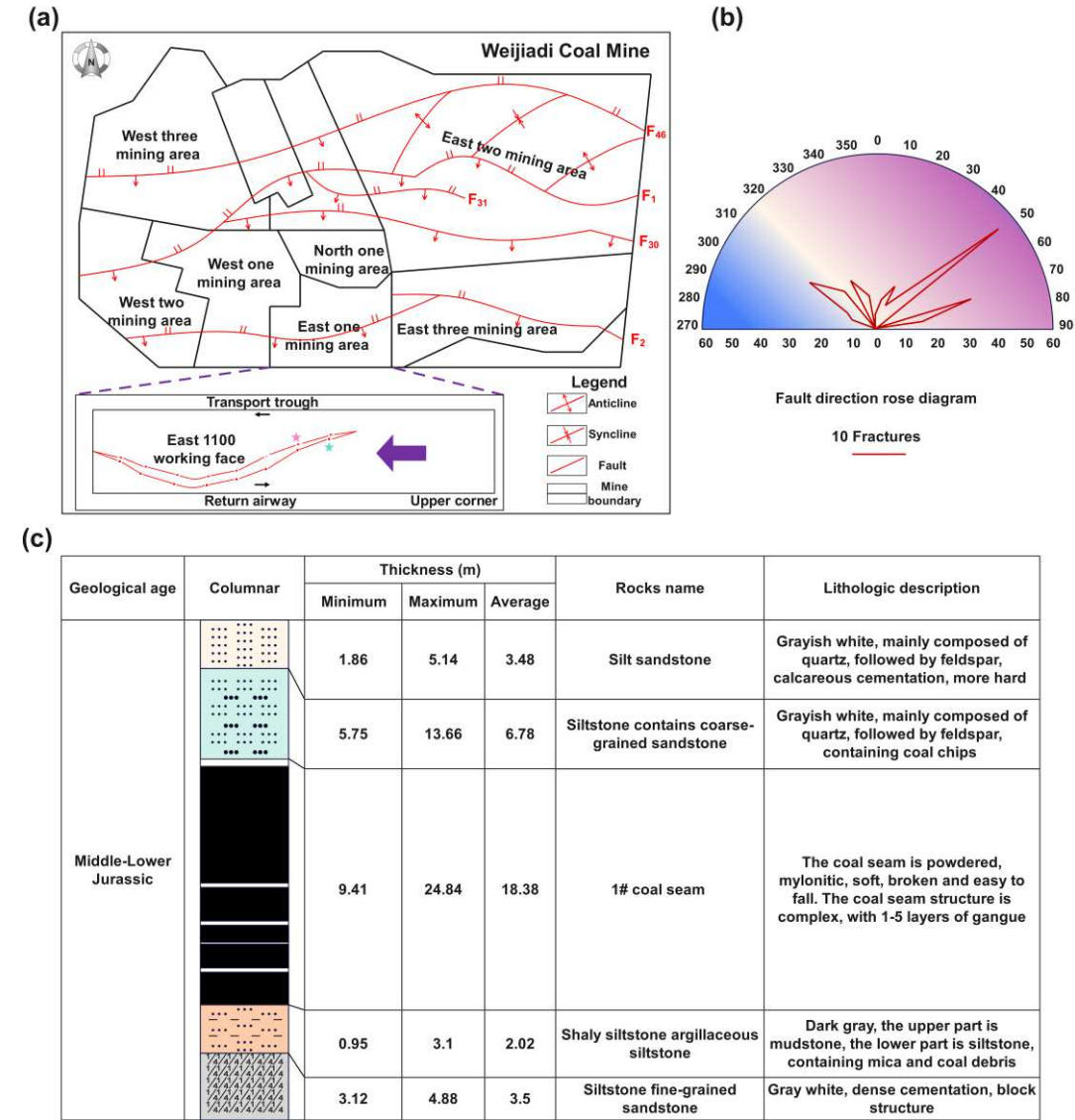


Fig. 1 (a) Minefield structure outline; (b) Fault direction rose diagram; (c) Coal seam histogram.

2.2 Observation of Surface Deformation in Mining Area

2.2.1 Observation Principle of Surface Deformation

Geological remote sensing applies modern remote-sensing techniques to geological mapping, resource exploration and analysis of geological processes. The

approach interprets the electromagnetic-radiation response of geological bodies at the regional scale together with complementary geological and remote-sensing information [29,30]. Coal extraction consolidates and compresses loose strata, producing a reduction in surface elevation. Conventional methods, including underground bedrock and layered marks, surface levelling and GPS, can provide high accuracy but depend on manually installed monitoring points. They are costly and difficult to deploy in complex or inaccessible terrain. Automated acquisition and processing are therefore increasingly important for tracking geological hazards and environmental deformation.^{29,30}

Remote sensing provides rapid, repeatable and spatially extensive observations of resource and environmental conditions. Improvements in data processing have made it possible to characterize the mining environment at successive stages and support its integrated management. Differential synthetic-aperture-radar interferometry (D-InSAR) is particularly suited to high-precision observation of slow surface deformation. It evaluates spatial coherence, unwraps the two-dimensional phase and converts radar-phase differences into surface-displacement information. Combining SAR, high-resolution satellite imagery and geological surveys allows regional monitoring to distinguish subsidence rates associated with mining disturbance and lithological variation, follow the temporal development of land subsidence and inform the adjustment of mining direction.

2.2.2 Observation Results

The vertical surface-deformation field of Weijiadi coal mine is mapped in Fig. 2.

Previously extracted East 1101, East 1102 and East 1103 working faces in the East No.1 mining area continue to subside, and their influence extends towards the unmined East 1100 and East 1104 working faces. East 1101 has a maximum coal thickness of 34.7 m and exhibits the greatest overburden failure and settlement. East 1100 lies next to the F_{1-2} reverse fault, where fault activation intensifies deformation. East 1104 is enclosed by faults and goafs and therefore shows an island effect. Strong overlying strata east of the transport crossheading limit settlement, but the associated mining disturbance may increase the likelihood of dynamic disasters.

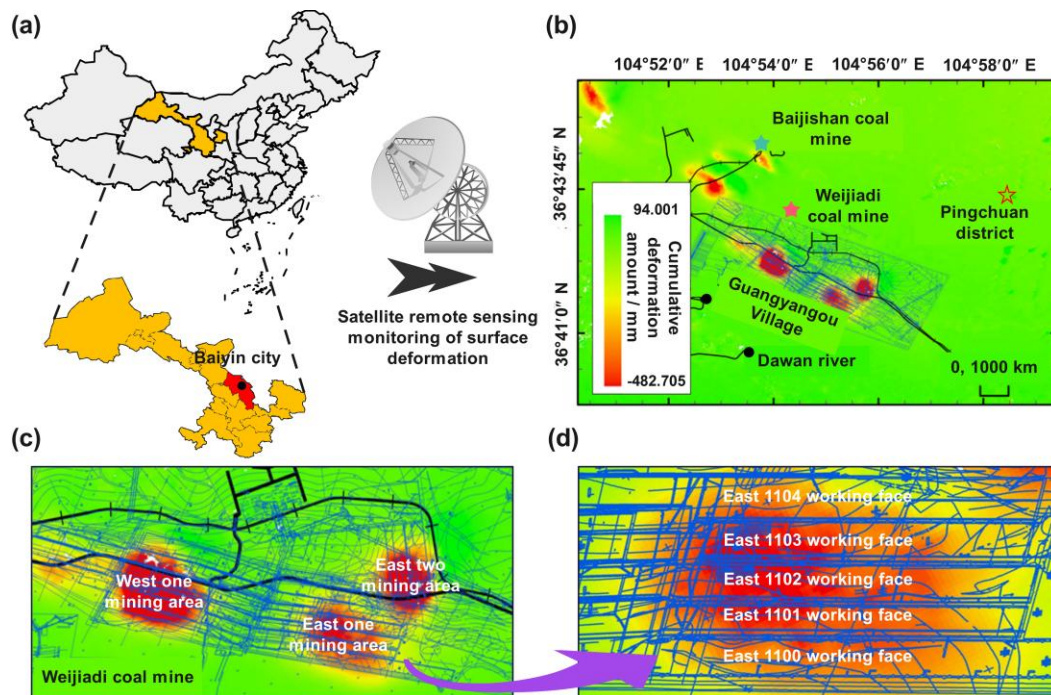


Fig. 2 Distribution of surface deformation areas in Weijiadi coal mine from the perspective of satellite remote sensing: (a) Geographical location of the study area; (b) Monitoring of cumulative surface deformation in Pingchuan mining area; (c) Monitoring of surface deformation in the main mining area of Weijiadi coal mine; (d) Surface deformation monitoring in east one mining area.

3 Analysis of In-Situ Rock Stress Test in East One Mining Area

3.1 Coal Rock Mechanics Parameter Test

The coal-rock mechanics programme measured uniaxial compressive strength,

elastic modulus and Poisson's ratio. Uniaxial compressive strength represents the ultimate load-bearing capacity under vertical uniaxial compression. Elastic modulus quantifies resistance to elastic deformation and is a principal mechanical-property parameter. Poisson's ratio is the lateral-to-axial deformation ratio during compression or tension and describes the lateral strain response of the material [31,32].^{31,32}

In-situ stress measurements were made at measuring points in drilling field 38 of the East 1100 return-air crossheading. A crawler-mounted, fully hydraulic tunnel drilling rig was used. The drilling tools included a $\Phi 127$ mm core bit, $\Phi 127$ mm core barrel, $\Phi 127$ mm conical reaming bit and $\Phi 36$ mm measuring-hole bit. The field procedure is shown in Fig. 3a. During drilling, cores were sealed in plastic packaging and transported for laboratory measurement of mechanical parameters (Fig. 3b).

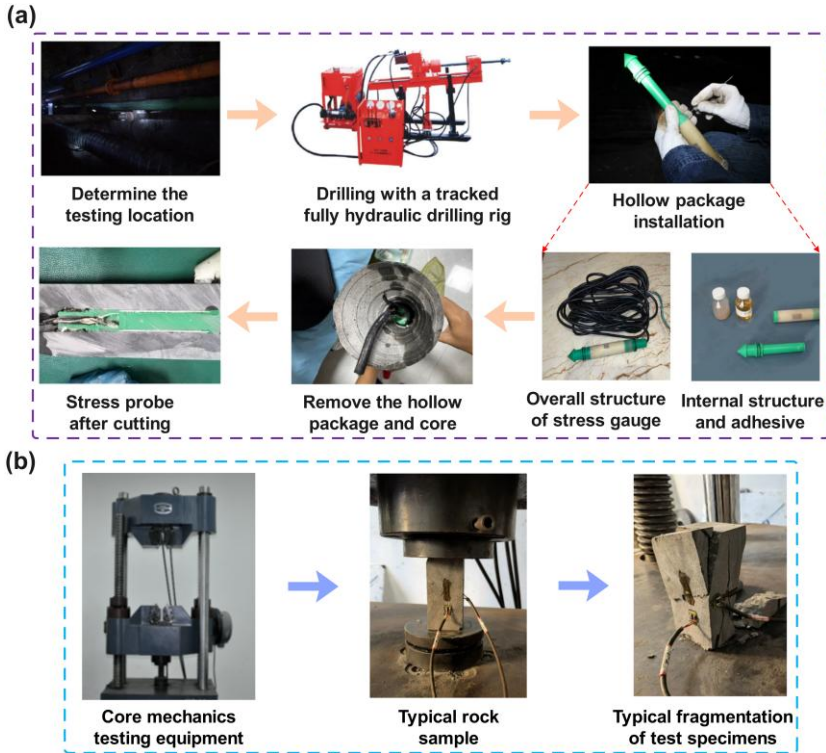


Fig. 3 (a) On site stress testing; (b) Coal rock mechanics index testing.

Figure 4 plots the measured stress against longitudinal and transverse strain. The

specimen elastic modulus and Poisson's ratio were obtained from these curves; the resulting values are listed in Table 1.

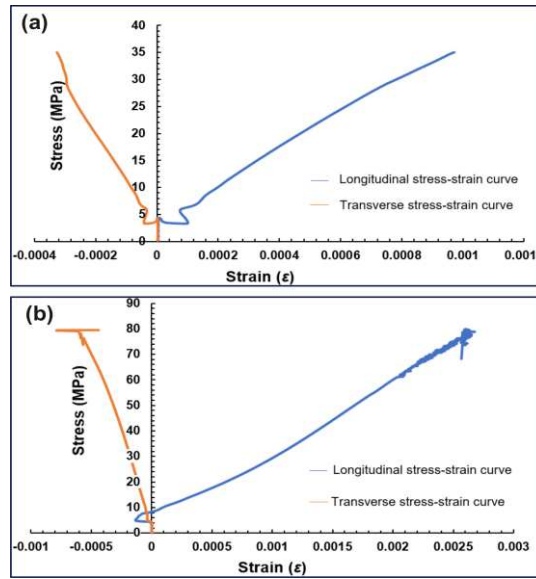


Fig. 4 Stress-strain test curve of specimen: (a) Specimen 1; (b) Specimen 2.

Table 1 Rock mechanics indicators of Weijiadi coal mine.

Testing location	Elastic modulus (MPa)	Poisson's ratio	Uniaxial compressive strength (MPa)
East 1100 return	32240	0.21	39.15
airway	26179	0.19	79.40

3.2 Calculation of In-Situ Stress Measurement Results

3.2.1 In-Situ Stress Distribution Law

The measured observations, borehole orientation, strain-gauge installation angle, elastic modulus, Poisson's ratio and related inputs were supplied to the hollow-inclusion strain-gauge stress-relief calculation programme. The resulting principal stresses and stress components for Weijiadi coal mine are reported in Tables 2 and 3.

Table 2 The measured in-situ stress in the east one mining area of Weijiadi coal mine.

Principal stress category	Principal stress value (MPa)	Azimuth angle (°)	Dip angle (°)
Maximum principal stress σ_1	39.79	221.98	7.76

Intermediate principal stress σ_2	24.78	-47.06	7.00
Minimum principal stress σ_3	13.69	264.55	-79.52

Table 3 The measured in-situ stress in the east one mining area of Weijiadi coal mine.

In-situ stress components σ_i , τ_{ij} (MPa)					
σ_x	σ_y	σ_z	τ_{xy}	τ_{yz}	τ_{zx}
31.01	32.92	14.33	7.29	-1.68	-3.32

Note : The in-situ stress component takes the geographic coordinate system, and its coordinate axis is that the x -axis points to the east, the y -axis points to the north, the z -axis is upward, and the compressive stress is positive.

The measuring point is at a vertical depth of 546 m, where the corresponding vertical stress is 13.81 MPa. The maximum principal stress is 39.79 MPa, above the typical values reported for other mining areas in China. The ratio of mean principal stress to vertical stress is 2.36, also exceeding the average level reported domestically and internationally. Because the maximum horizontal principal stress is greater than the vertical stress and the maximum principal stress is approximately 2.72 times the self-weight stress, the Weijiadi in-situ stress field is classified as horizontally dominated.

3.2.2 Analysis of In-Situ Stress Field Type

The relative ordering of the three principal stresses provides the basis for classifying a regional ground-stress field. The conventional categories are normal-fault, strike-slip-fault and reverse-fault stress types. A normal-fault stress regime corresponds to a geostatic or extensional field. Strike-slip and reverse-fault regimes correspond to geodynamic or compressional fields. The transition between normal-fault and strike-slip conditions is treated as a quasi-hydrostatic pressure field, representative of a relatively stable zone.

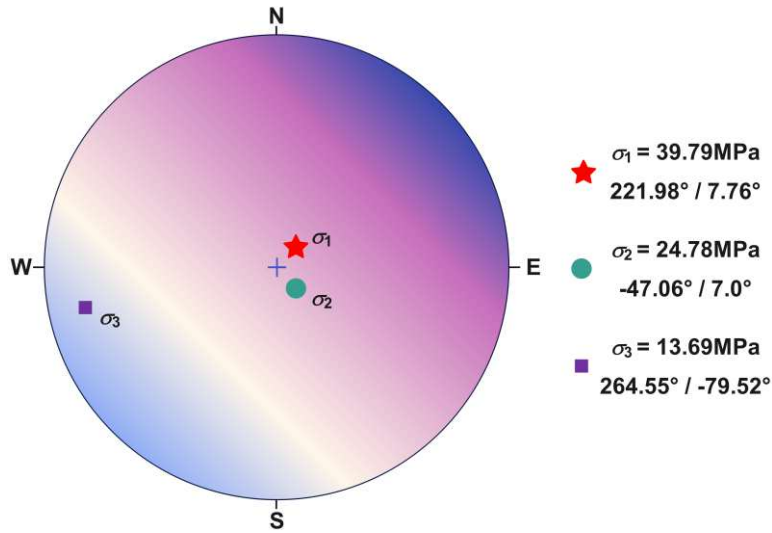


Fig. 5 In-situ stress value distribution of Weijiadi coal mine.

4 Evolution Law of Mining Stress and Coal Pillar Stress in East 1100

Working Face

4.1 Geometric Size of Working Face

The layout of the East 1100 working face and the sections along exploration lines 14, +14 and 15 are shown in Fig. 6.

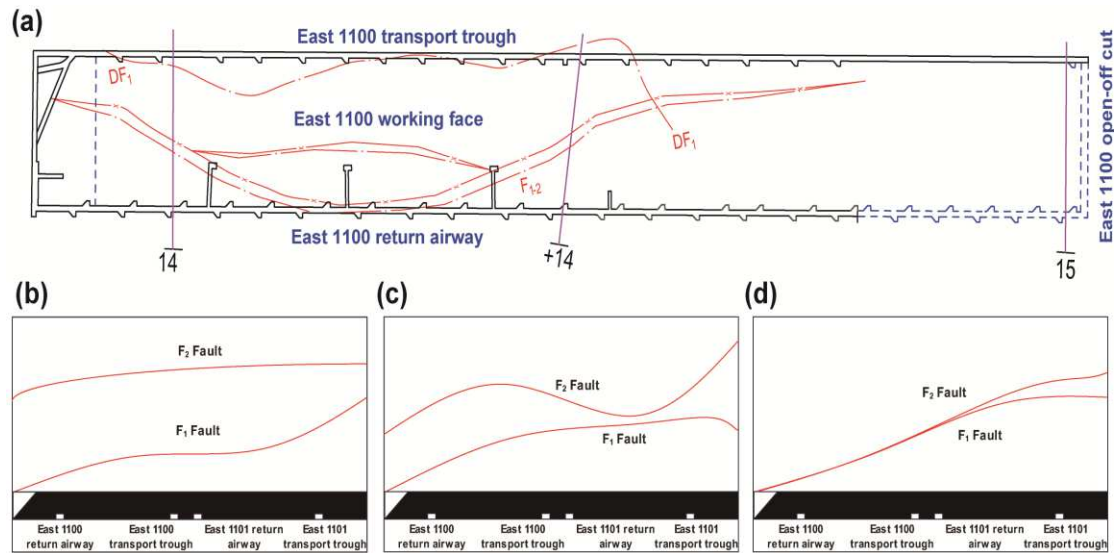


Fig. 6 (a) East 1100 working face layout; (b) 14 Exploration line profile; (c) +14 Exploration line profile; (d) 15 Exploration line profile.

4.2 Establishment of Working Face Mining Model

A finite-difference model was constructed in FLAC3D from the production and

geological conditions of the 1100 and 1101 working faces, with separate sections along exploration lines 14, +14 and 15 (Fig. 7). Each model is 378 m long, 300 m wide and 210 m high, and contains 792240, 644520 and 644520 zones, respectively. The simulated sequence follows field practice: the 1101 working face is extracted first, followed by the 1100 working face.

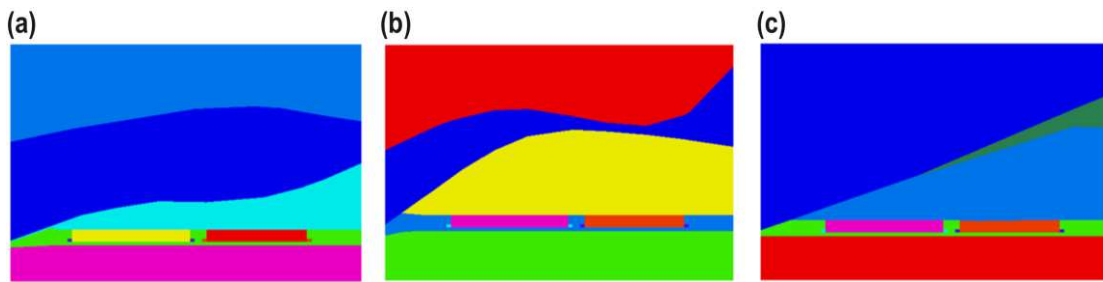


Fig. 7 Numerical model of working face: (a) 14 Exploration line; (b) +14 Exploration line; (c) 15 Exploration line.

The mechanical parameters of each coal stratum are shown in Table 4.

Table 4 The mechanical parameters of coal and rock are set in the model.

Rock position	Density (kg/m ³)	Bulk modulus (GPa)	Shear modulus (GPa)	Cohesion (MPa)	Internal friction angle (°)	Tensile strength (MPa)
Upper part of F ₂ fault	2387	7	13	4	36	5
Upper part of F ₁ fault	2387	7	13	4	36	5
Coal seam roof	1238	1.4	2.73	2.5	26	1.5
Coal seam	1238	1.4	1.55	1.6	28	1
Coal seam floor	1238	1.4	1.55	1.6	28	1

The model boundary constraints were specified as follows:

- (1) The two X-axis end boundaries were restrained in the X direction, so their X displacement was set to zero.

(2) The two Y-axis end boundaries were restrained in the Y direction, so their Y displacement was set to zero.

(3) The bottom boundary was fixed, with zero displacement in the X, Y and Z directions.

(4) The upper boundary was left free.

The applied boundary loads were defined as follows:

(1) A gradient stress ranging from 14.39 MPa to 18.91 MPa was applied along the model X axis.

(2) A gradient stress ranging from 7.79 MPa to 10.17 MPa was applied along the model Y axis.

(3) A gradient stress ranging from 12.45 MPa to 17.58 MPa was applied along the model Z axis.

4.3 Stress Distribution Pattern in the Region of Zone-Isolated Narrow Coal Pillars

The +14 exploration-line model for the East No. 1 mining area's 1100 working face was used to examine the small coal pillar during roadway development and subsequent face extraction. This analysis tracks pillar-stress evolution and evaluates the stability of the existing pillar. After the 1100 roadway was excavated, the vertical stress field shown in Fig. 8 developed a pillar peak of 25.23 MPa, equivalent to a stress concentration factor of 1.77.

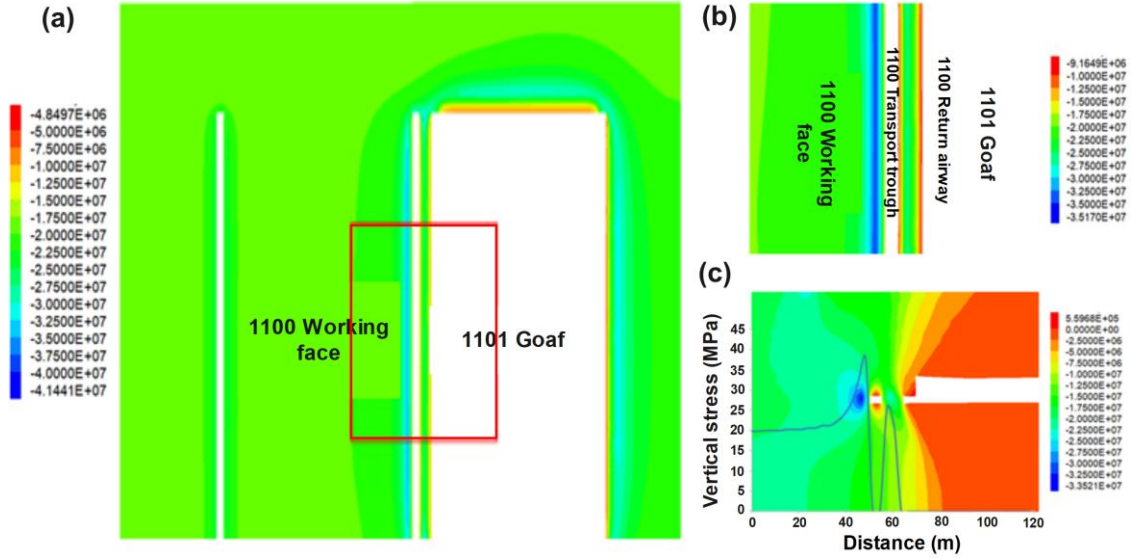


Fig. 8 (a) Vertical stress distribution diagram of coal pillar after excavation in the 1100 working face roadway; (b) Vertical stress plane diagram; (c) Vertical stress profile diagram.

The early mining stage was represented by the pillar-stress field when the 1100 working face was 50 m from the open-off cut (Fig. 9). At the working-face position, the pillar peak stress was 26.29 MPa and the stress concentration factor was 1.85. The maximum pillar stress, 27.04 MPa with a concentration factor of 1.89, occurred 5.88 m ahead of the face.

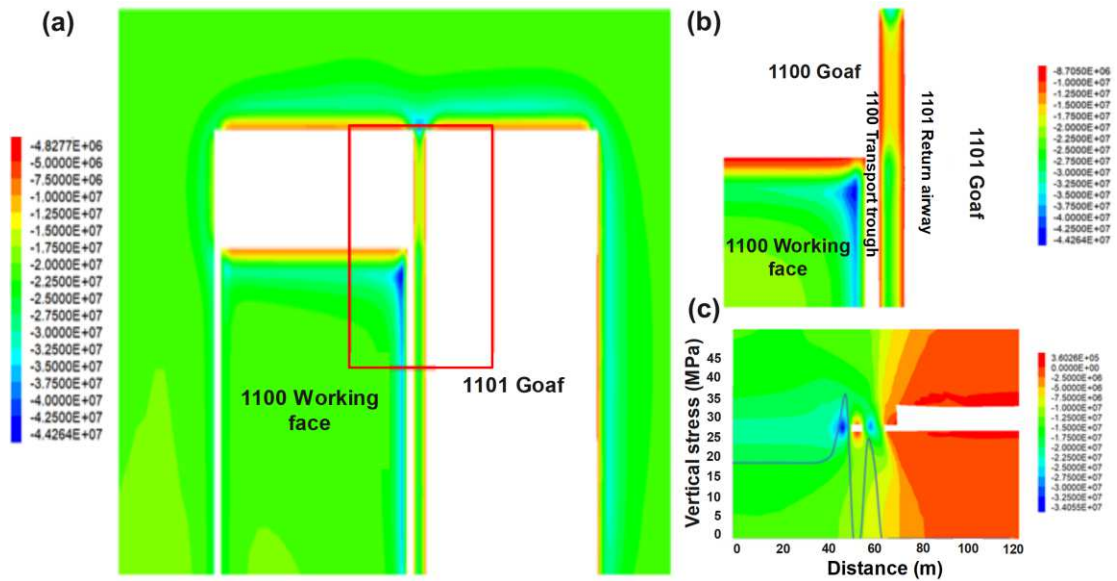


Fig. 9 (a) Vertical stress distribution diagram of coal pillar at a distance of 50m from the open-off cut on the 1100 working face; (b) Vertical stress plane diagram; (c) Vertical stress profile diagram.

The late mining stage was examined with the 1100 working face 50 m from the stop line (Fig. 10). The pillar stress at the face was 26.8 MPa, corresponding to a concentration factor of 1.88. At 6.7 m ahead of the face, the pillar reached its maximum of 27.89 MPa and a concentration factor of 1.96.

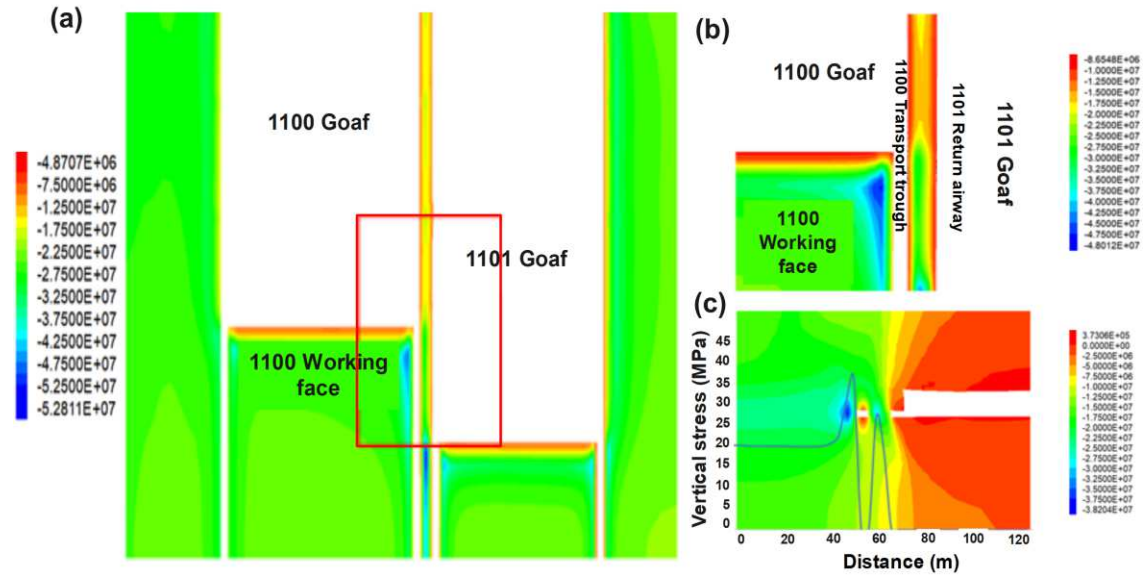


Fig. 10 (a) Vertical stress distribution diagram of coal pillar at a distance of 50m from the stop mining line on the 1100 working face; (b) Vertical stress plane diagram; (c) Vertical stress profile diagram.

At the stop line of the 1100 working face, the stress distribution shown in Fig. 11 gives a pillar peak of 34.43 MPa at the face, with a concentration factor of 2.42. The maximum value in the pillar is 59.67 MPa at a point 12.18 m ahead of the face, where the concentration factor reaches 4.19.

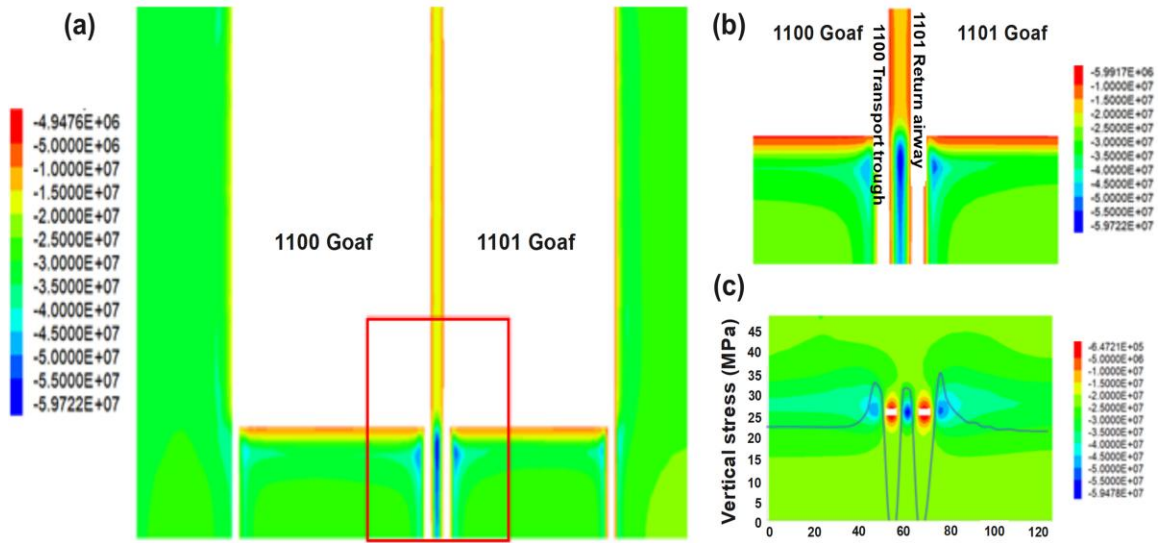


Fig. 11 (a) Vertical stress distribution diagram of coal pillar from the 1100 working face mining to the stopping line position; (b) Vertical stress plane diagram; (c) Vertical stress profile diagram.

Table 5 summarizes the pillar-stress evolution during extraction of the 1100 working face. Stress rises progressively during advance. Once the face reaches the stop line, the 1100 and 1101 stop lines coincide; the pillar changes from a condition with goaf on both sides to a condition without goaf, producing a sharp stress increase. The location of the maximum pillar-stress point also moves progressively farther ahead of the working face.

Table 5 Stress distribution at different positions of coal pillar in mining process.

1100 Working face mining position	Peak stress at the working face position (MPa)	Maximum stress peak of the whole coal pillar (MPa)	Distance between the maximum peak point of the coal pillar and the working face (m)
Unmined	-	25.23	-
50 m From the open- off cut	26.29	27.04	5.88

50 m Away from the			
stop line	26.80	27.89	6.7
Position of terminal			
mining line	34.43	59.67	12.18

320 **4.4 Stress Distribution Law of Top Coal Caving Face in Extra-Thick Coal Seam**

321 The East 1100 working face extends 1718 m along strike, and the overlying fault
322 geometry changes markedly along this length. Models at three exploration lines were
323 compared to determine how these structural differences affect mining-stress evolution.

324 **4.4.1 Mining stress at exploration line 14**

325 The plan-view mining-stress field for the 1100 working face along exploration line
326 14 is presented in Fig. 12. With the face 50 m from the open-off cut, the central advance
327 abutment pressure peaks at 29.85 MPa; the stress concentration factor is 2.24 and the
328 influence range is 33 m. On the transport-trough side, the peak abutment pressure is
329 42.72 MPa, or 3.20 times the original-rock vertical stress.

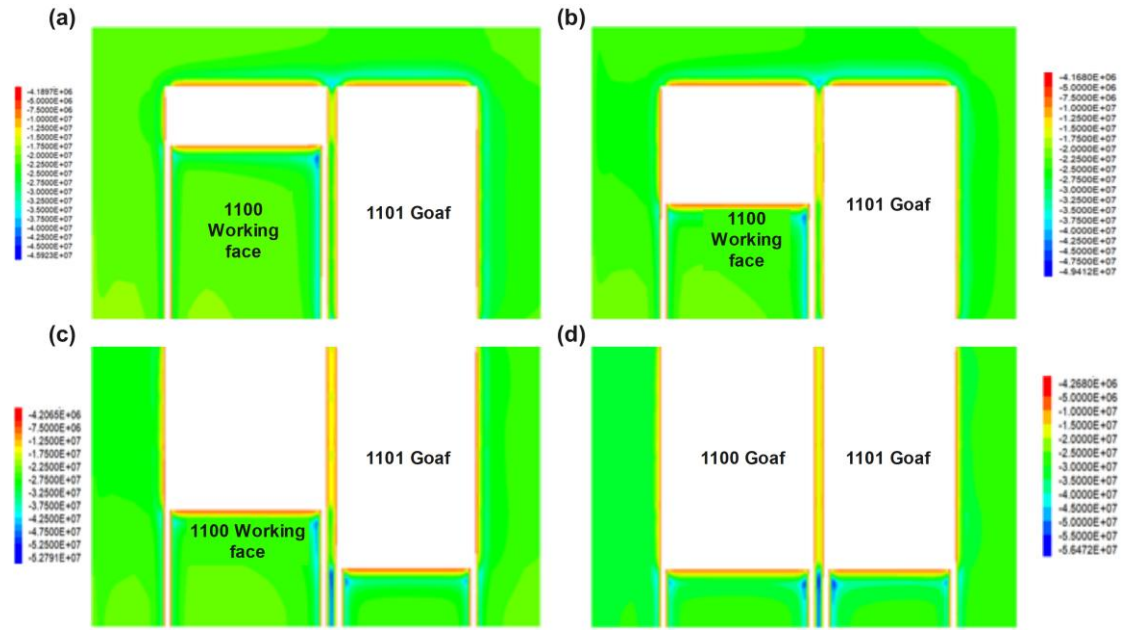


Fig. 12 (a) The mining stress distribution of the working face when the 1100 working face of the 14 exploration line is 50 m away from the open-off cut; (b) The mining stress distribution of the working face when it is 100 m away from the open-off cut; (c) The mining stress distribution of the working face when it is 50 m away from the stop line; (d) The mining stress distribution of the working face when it is far from the stop line.

At 100 m from the open-off cut, the central advance abutment pressure is 30.66 MPa, with a concentration factor of 2.30 and a 33 m influence range; the transport-channel-side peak is 43.97 MPa, or 3.30 times the original-rock vertical stress. At 50 m from the stop line, the corresponding central values are 33.26 MPa and 2.42, the influence range is 33 m, and the transport-trough-side peak is 47.86 MPa, or 3.59 times the original-rock vertical stress. At the stop line, the central peak is 36.03 MPa, the concentration factor is 2.70 and the influence range remains 33 m; the transport-channel-side peak is 51.74 MPa, or 3.88 times the original-rock vertical stress.

4.4.2 Mining stress at exploration line +14

Figure 13 shows the plan-view mining-stress field for the 1100 working face along exploration line +14. At 50 m from the open-off cut, the central advance abutment

pressure is 30.44 MPa, with a concentration factor of 2.21 and a 33 m influence range.

The transport-channel-side peak is 42.8 MPa, or 2.93 times the original-rock vertical stress. At 100 m from the open-off cut, the central peak is 31.44 MPa, the concentration factor is 2.28 and the influence range is 33 m; the transport-trough-side peak is 44.1 MPa, or 3.02 times the original-rock vertical stress. At 50 m from the stop line, the central peak is 32.58 MPa, the concentration factor is 2.36 and the influence range is 33 m; the transport-trough-side peak is 46.7 MPa, or 3.2 times the original-rock vertical stress. At the stop line, the central peak is 36.31 MPa, the concentration factor is 2.63 and the influence range is 33 m; the transport-trough-side peak is 48.3 MPa, or 3.31 times the original-rock vertical stress.

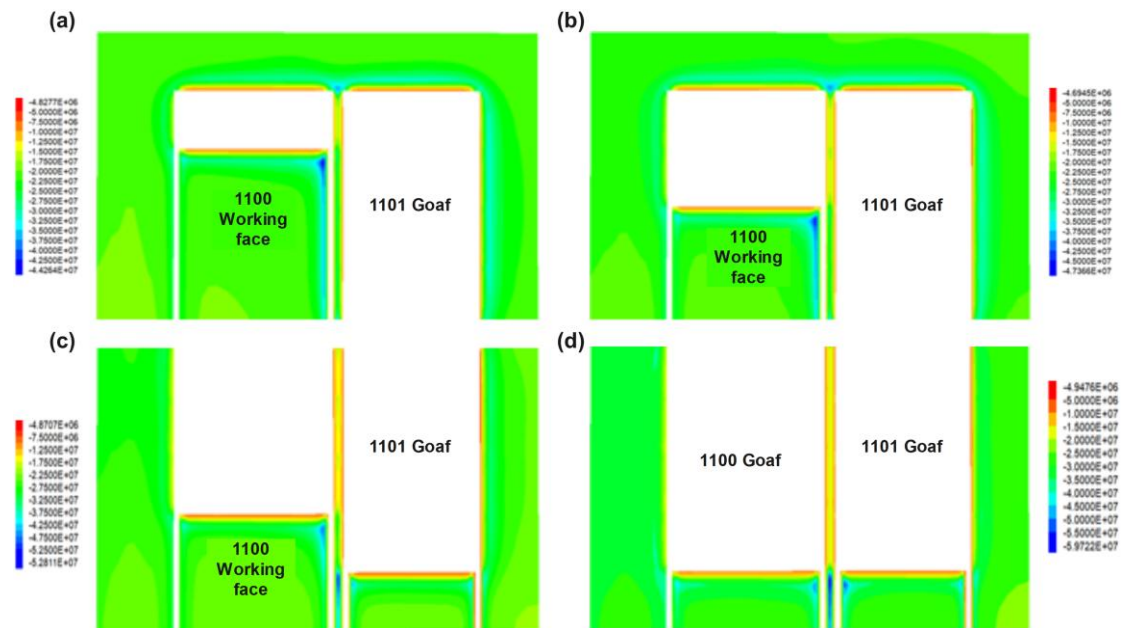


Fig. 13 (a) The mining stress distribution of the working face when the 1100 working face of the +14 exploration line is 50 m away from the open-off cut; (b) The mining stress distribution of the working face when it is 100 m away from the open-off cut; (c) The mining stress distribution of the working face when it is 50 m away from the stop line; (d) The mining stress distribution of the working face when it is far from the stop line.

4.4.3 Mining stress at exploration line 15

The plan-view mining-stress field for the 1100 working face along exploration line

15 is shown in Fig. 14. At 50 m from the open-off cut, the central advance abutment pressure is 30.14 MPa; the stress concentration factor is 2.18 and the influence range is 33 m. The transport-trough-side peak is 47.3 MPa, or 3.33 times the original-rock vertical stress.

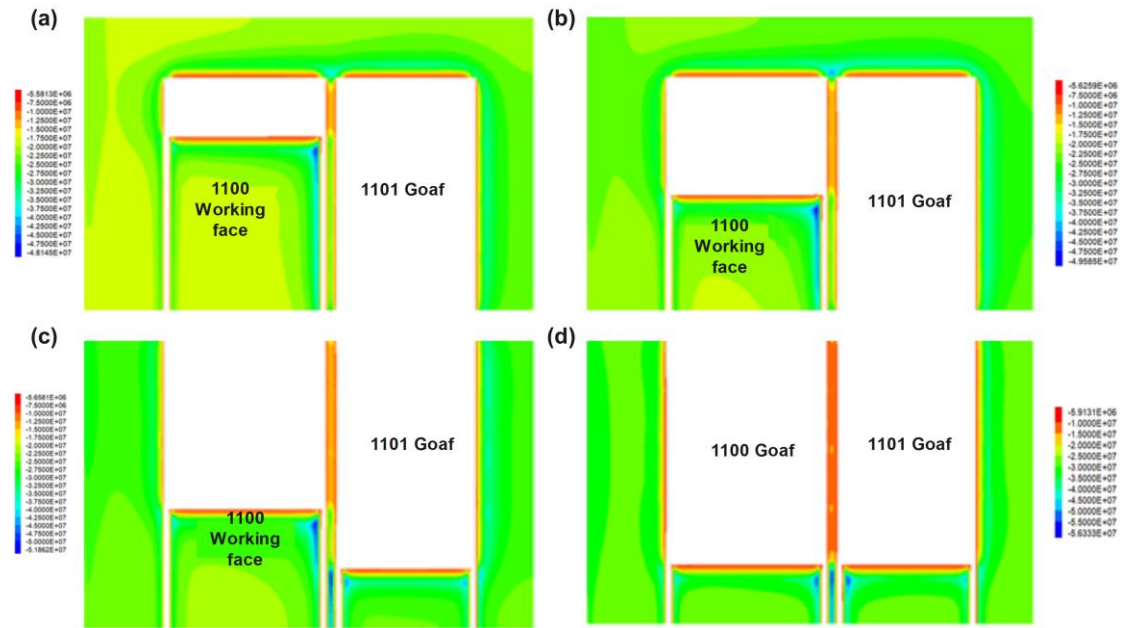


Fig. 14 (a) The mining stress distribution of the working face when the 1100 working face of the 15 exploration line is 50 m away from the open-off cut; (b) The mining stress distribution of the working face when it is 100 m away from the open-off cut; (c) The mining stress distribution of the working face when it is 50 m away from the stop line; (d) The mining stress distribution of the working face when it is far from the stop line.

At 100 m from the open-off cut, the central advance abutment pressure is 31.31 MPa, the concentration factor is 2.27 and the influence range is 33 m; the transport-trough-side peak is 48.3 MPa, or 3.4 times the original-rock vertical stress. At 50 m from the stop line, the central peak is 32.28 MPa, the concentration factor is 2.34 and the influence range is 33 m; the transport-trough-side peak is 50.5 MPa, or 3.56 times the original-rock vertical stress. At the stop line, the central peak is 35.40 MPa, the concentration factor is 2.56 and the influence range is 33 m; the transport-channel-side

peak is 53.4 MPa, or 3.76 times the original-rock vertical stress.

Across the three exploration-line simulations, the transport crossheading remains close to the 1101 goaf during extraction of the 1100 working face, and the advance zone near this crossheading forms the main stress-concentration area. The central advance-abutment-pressure peak increases as the face advances, whereas its influence range changes little.

4.5 Control Mechanism of Fault Geological Structure Zone on Dynamic Coupling Disaster

The influence of a reverse-fault zone on a coupled dynamic disaster can be considered through its geological structure, stress field and mining-induced interaction. A reverse fault forms under crustal compression, with the hanging wall moving upward relative to the footwall. Such faults are common in tectonically active settings and are closely associated with mine hazards. Their stress field is often heterogeneous, particularly near the fault zone: structural loading redistributes the original stress and can sharply raise local stress, creating conditions for dynamic disasters.

Figure 15a tracks mining stress at monitoring points east of the 1100 working face. As the face crossed the fault in January, stress rose from early January towards mid-month and reached its maximum around 16 January. It then declined overall from mid-January, falling rapidly between mid- and late January, followed by a smaller decrease from February to April before becoming stable.

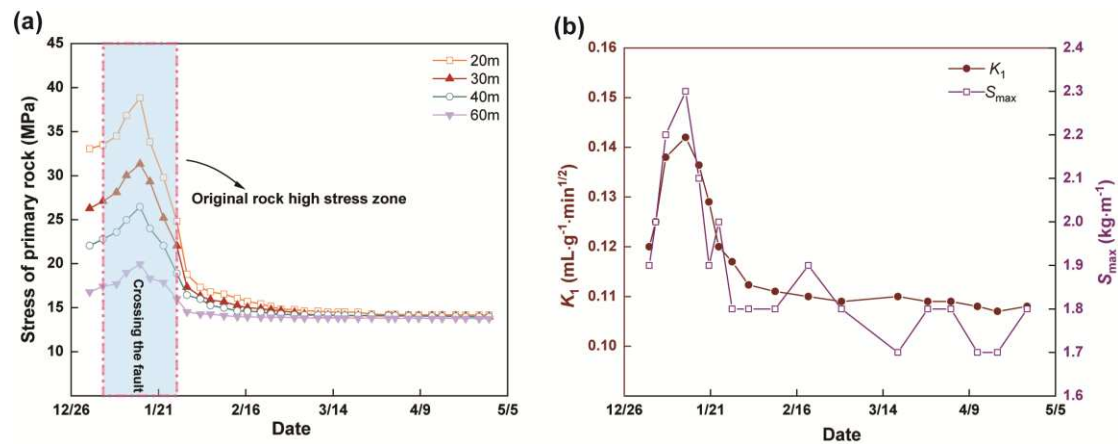


Fig. 15 (a) Stress change trend during mining period; (b) Variation trend of $S_{\max}$ and K_1 in a day during mining period.

Figure 15b presents daily $S_{\max}$ and K_1 values for the East 1100 working face from 1 January to 30 April. Both indicators increased initially, then declined and ultimately levelled off. Their rapid rise from early to mid-January was followed by a rapid decrease. The timing matches the stress response: passage through the high-stress original-rock zone produced a sharp stress increase and a corresponding change in the drilling-cuttings desorption index.

The relation between mining stress and K_1 or $S_{\max}$ at the point 20 m from the working face is fitted in Fig. 16. The outburst indicators respond to mining stress in a quadratic manner. As stress increases, K_1 and $S_{\max}$ rise, approach a relatively stable interval and then increase rapidly. The positive association suggests that mine pressure can be used to quantify the coal-gas response within a defined range, while gas-outburst indicators can in turn help identify the spatial distribution of mining stress.

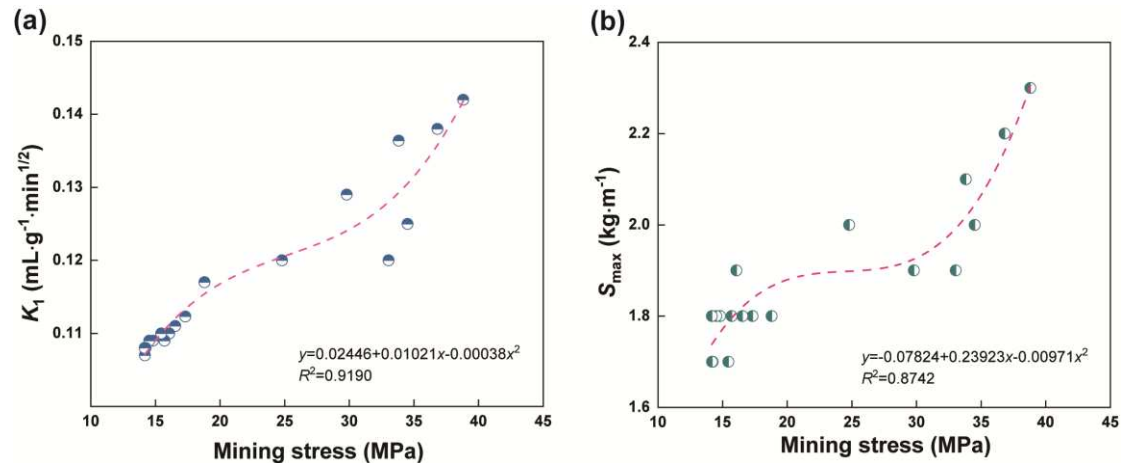


Fig. 16 (a) Fitting relationship between mining stress and K_1 ; (b) Fitting relationship between mining stress and S_{max} .

Taken together, the fault-structure zone produces a stress-rise region that intensifies concentration at the working face. Roof pressure increases by about 1 times, while maximum drilling-cuttings amount and K_1 also rise markedly. These concurrent responses indicate a substantially higher coupled dynamic disaster risk near the fault and support a controlling role for the fault structure.

4.6 Stress Superposition Effect of Mining in Fault Geological Structure Zone

4.6.1 Stress superposition in the return airway under the exploration-line-14 fault

Figure 17 shows the stress field in the 1100 return airway under the exploration-line-14 fault. The vertical-stress profiles identify the location of the working face's peak advance abutment pressure.

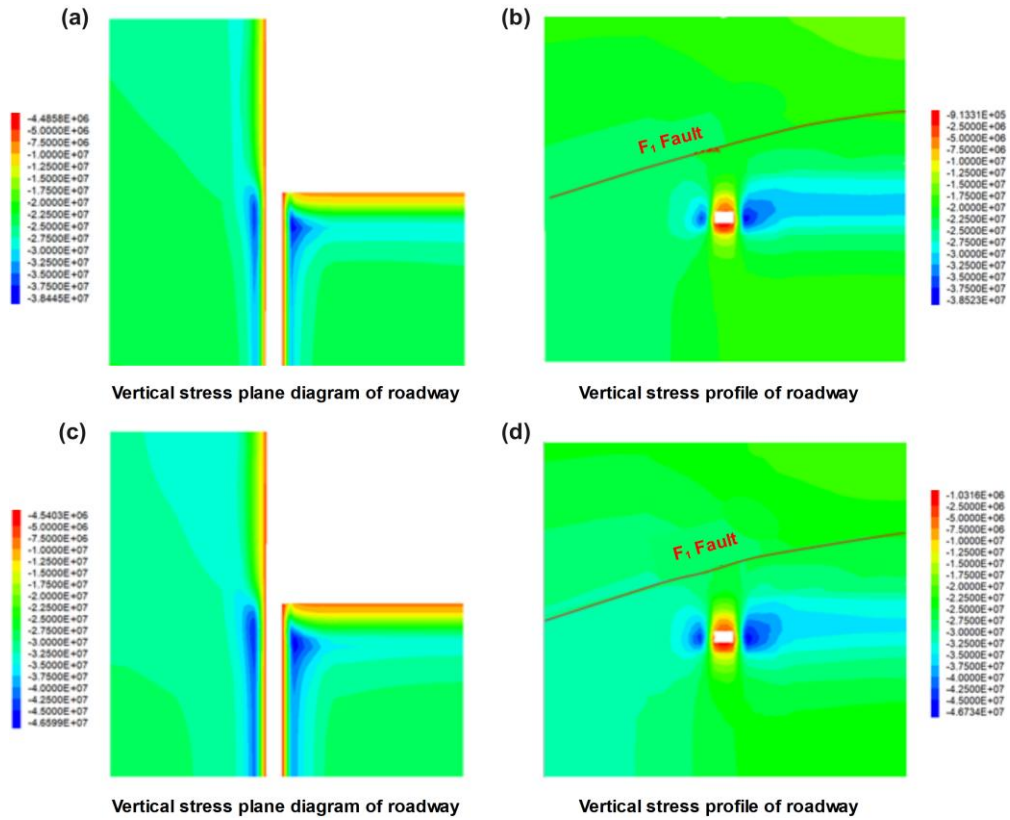


Fig. 17 (a), (b) Vertical stress in the roadway under the influence of faults during the initial mining period of exploration line 14; (c), (d) Vertical stress of roadway under the influence of faults in the late stage of mining on exploration line 14.

At the beginning of mining, with the 1100 working face 100 m from the open-off cut, the peak abutment pressures on the inner working-face side and fault side are 38.52 MPa and 36.33 MPa, respectively; both peaks are 10 m ahead of the face. At the end of mining, when the 1100 working face reaches the stop line, the corresponding values are 46.73 MPa and 44.03 MPa, and the peak is 12 m ahead of the face.

4.6.2 Stress superposition in the return airway under the exploration-line +14 fault

Figure 18 presents the 1100 return-air chute stress field affected by the exploration-line +14 fault. The vertical-stress profiles mark the peak of the face's advance abutment pressure.

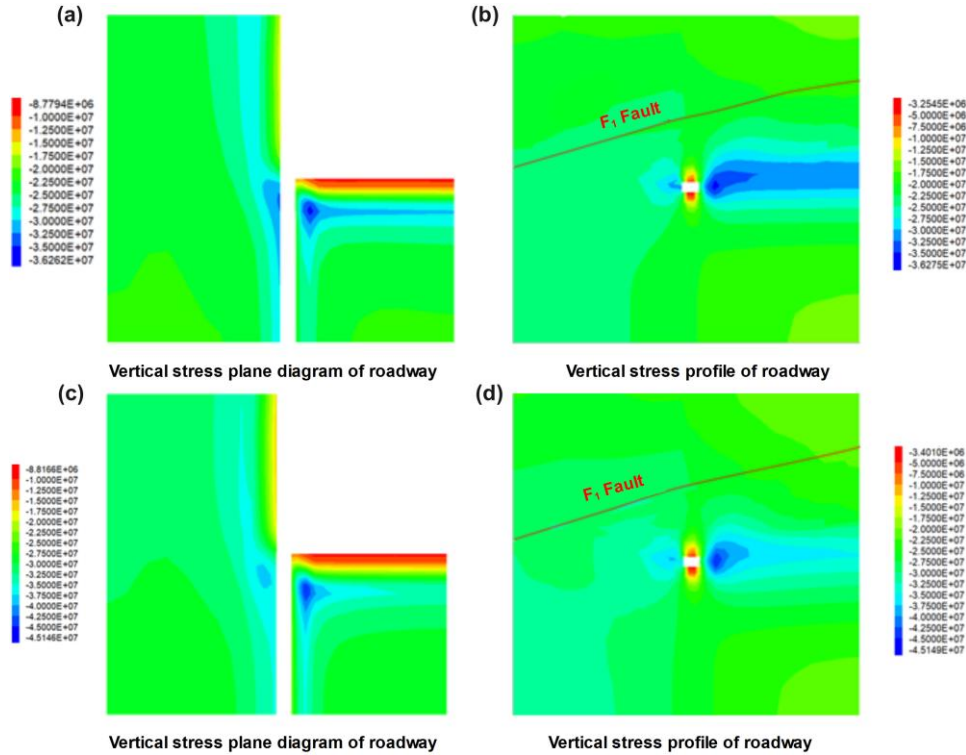


Fig. 18 (a), (b) Vertical stress in the roadway under the influence of faults during the initial mining period of exploration line +14; (c), (d) Vertical stress of roadway under the influence of faults in the late stage of mining on exploration line +14.

At the beginning of mining, with the 1100 working face 100 m from the open-off cut, the peak abutment pressures on the inner side and fault side are 36.27 MPa and 35.73 MPa, respectively, with the peak located 10 m ahead of the face. At the stop line of the 1100 working face, the two peaks are 45.15 MPa and 42.88 MPa, respectively, and the peak location is 12 m ahead of the face.

4.6.3 Stress superposition in the return airway under the exploration-line-15 fault

The vertical-stress distribution in the 1100 return airway under the exploration-line-15 faults is shown in Fig. 19; the profile indicates the position of the working face's maximum advance abutment pressure.

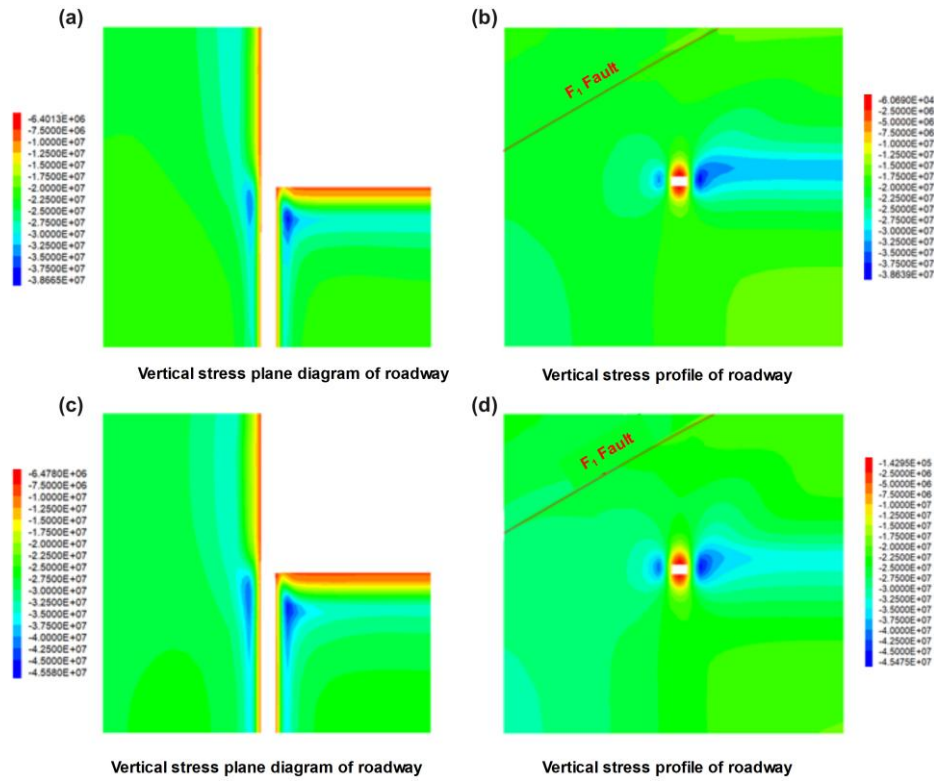


Fig. 19 (a), (b) Vertical stress in the roadway under the influence of faults during the initial mining period of exploration line 15; (c), (d) Vertical stress of roadway under the influence of faults in the late stage of mining on exploration line 15.

Early in mining, when the 1100 working face is 100 m from the open-off cut, the peak abutment pressures on the inner working-face side and fault side are 38.66 MPa and 33.27 MPa, respectively, and the peak is 10 m in front of the face. At the stop line of the 1100 working face, the corresponding peaks are 45.58 MPa and 41.58 MPa, with the peak located 12 m in front of the face.

The three exploration-line cases show a consistent pattern in the 1100 return airway. As the working face advances, peak abutment pressure in the 1100 return airway increases on both the inner side and the fault side, and the position of the peak moves progressively farther ahead of the face.

4.6.4 Fault slip during working-face mining

Mining of the 1100 working face induced movement along the fault. Figure 20

maps model vertical displacement at exploration lines 14, +14 and 15. The maximum fault slips are 0.46 m for the 1100 working face in the line-14 model, 0.52 m for the 1100 working face in the line-+14 model and 0.61 m for the 1100 working face in the line-15 model. The fault above the working face appears to limit roof movement during extraction: the closer the working-face roof is to the fault, the smaller the roof subsidence.

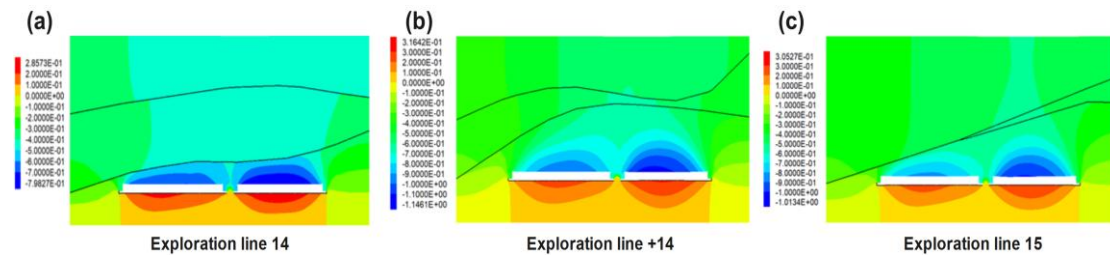


Fig. 20 Vertical displacement distribution diagram of the model.

5 Coupling dynamic disaster prevention measures

A comprehensive programme for gas and dynamic-disaster control should combine risk prediction, early warning, regional and local measures, and effectiveness testing. Following the four-in-one prevention and control system, the workflow used in this study is summarized in Fig. 21.

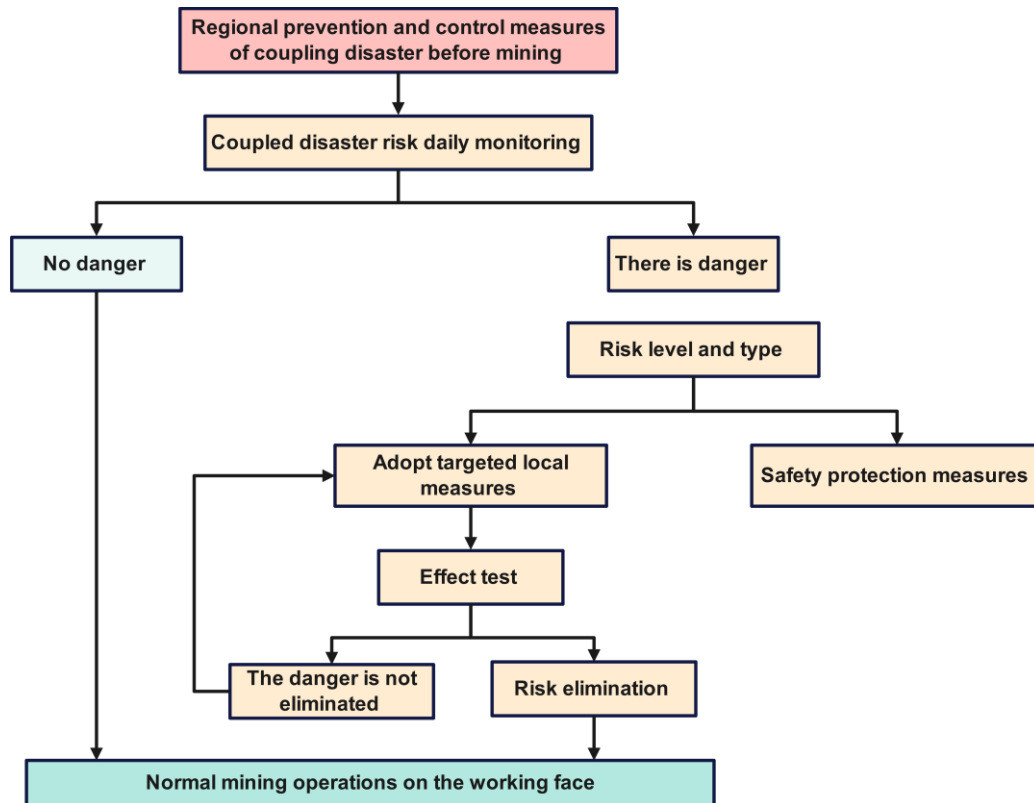


Fig. 21 Coupling disaster prevention technology flow chart.

5.1 Coupling Dynamic Disaster Risk Detection and Early Warning

The coupled-disaster initiation framework requires monitoring methods that are matched to the two energy sources and that resolve their distribution and migration in time. Because the sources differ in generation mechanism and spatial scale, no single method can provide complete information. The electromagnetic-radiation system surveys a zone extending 150 m around the mining face and is therefore suited to detailed monitoring of local critical areas, with warning value for rock burst and coal and gas outburst. Stress monitoring tracks the distribution and evolution of surrounding-rock stress in the mining space. Drilling-cuttings indicators reflect gas desorption and in-situ stress and can be used to assess coupled-disaster risk.

5.1.1 Electromagnetic Radiation System

Measuring points were installed along both sides of the solid coal from 5 m in front of the face to a distance of 100 m, at 10 m intervals, giving 20 points in total. As the face advanced, additional points were installed. Measurements were taken every 2 days, or daily where a fault, fold or other geological structure lay ahead or where pressure increased (Fig. 22). For the East 1100 working face, the initial electromagnetic-radiation threshold was selected from local monitoring experience. After two months of observations, 1.5-1.8 times the maximum monthly mean was adopted as the provisional threshold and was updated as conditions changed.

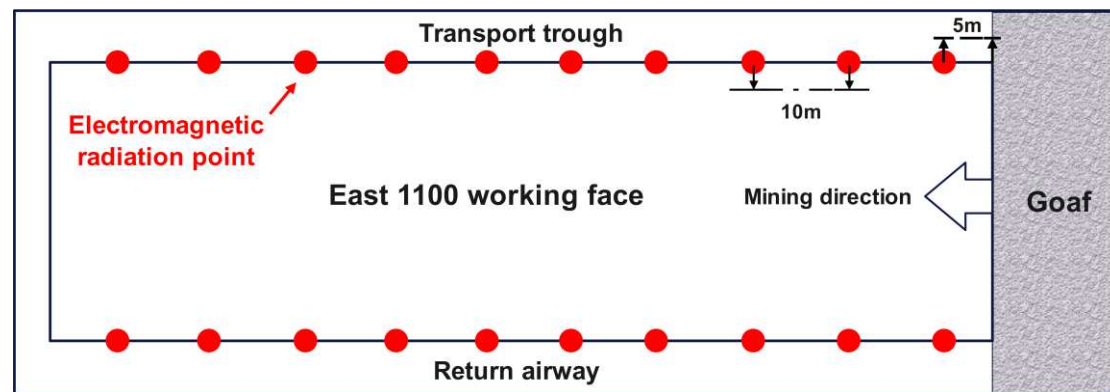


Fig. 22 Layout diagram of electromagnetic radiation measuring points in mining face.

5.1.2 On-Line Stress Monitoring System

The stress-monitoring system was installed to track impact hazard during advance. 13 stress meters were placed within 300 m ahead of the two working-face ends, with a group spacing of 30 m. Each meter was connected to a two-channel wireless data-acquisition unit through pressure-conversion components and a pressure transmitter. 13 such units were arranged in the two monitoring directions. A repeater was placed beside the outermost unit at Liangshun. Dedicated cable connected the repeater to the monitoring substation and the surface monitoring host in the East one mining-area

substation, allowing the substation data to be uploaded in real time. The system design is shown in Fig. 23.

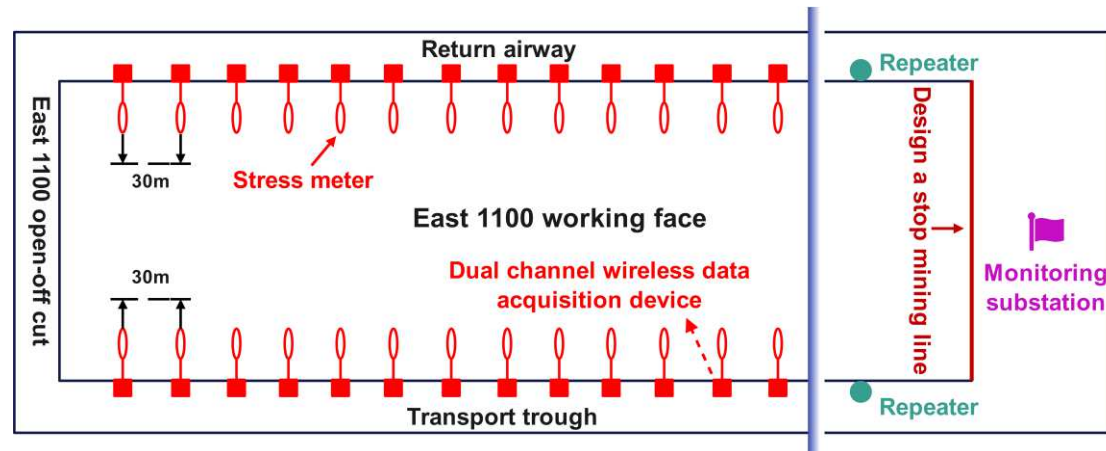


Fig. 23 On-line stress monitoring system.

5.1.3 Monitoring by Drilling Cuttings

At a heading face, one test borehole was arranged on each side for every 20 m of advance. Each borehole had a diameter of $\Phi 42$ mm and a depth of 12 m, was normal to the coal wall and followed the coal-seam dip; the opening height was selected according to field conditions. Three additional $\Phi 42$ mm boreholes, each 8 ~ 10 m deep, were drilled in the soft layer ahead of the coal-roadway heading face. One was placed at the face centre and parallel to the heading direction. The other two followed the inclined roof on either side, with final hole points 2 ~ 4 m outside the roadway contour. The arrangement is shown in Fig. 24a.

For roadways behind the heading face or during extraction, one test borehole was installed every 10 m, and the test interval extended more than 20 m beyond both sides of the hazardous zone. The borehole diameter was $\Phi 42$ mm, the depth 12 m and the orientation normal to the coal wall, with inclination following the coal seam; the opening height was selected from field conditions (Fig. 24b). During drilling, cuttings

were weighed for every 1 m of hole and the drilling-cuttings desorption index was measured every 2 m. Coupled-disaster risk at the heading face was judged from the per-metre maximum drilling-cuttings amount S_{max} and desorption index K_1 . When $S_{max} < 4.5 \text{ kg/m}$ and $K_1 < 0.35 \text{ mL}/(\text{g} \cdot \text{min}^{1/2})$, no coupled-disaster risk was assigned. When $S_{max} \geq 4.5 \text{ kg/m}$ or $K_1 \geq 0.35 \text{ mL}/(\text{g} \cdot \text{min}^{1/2})$, the face was classified as at risk.

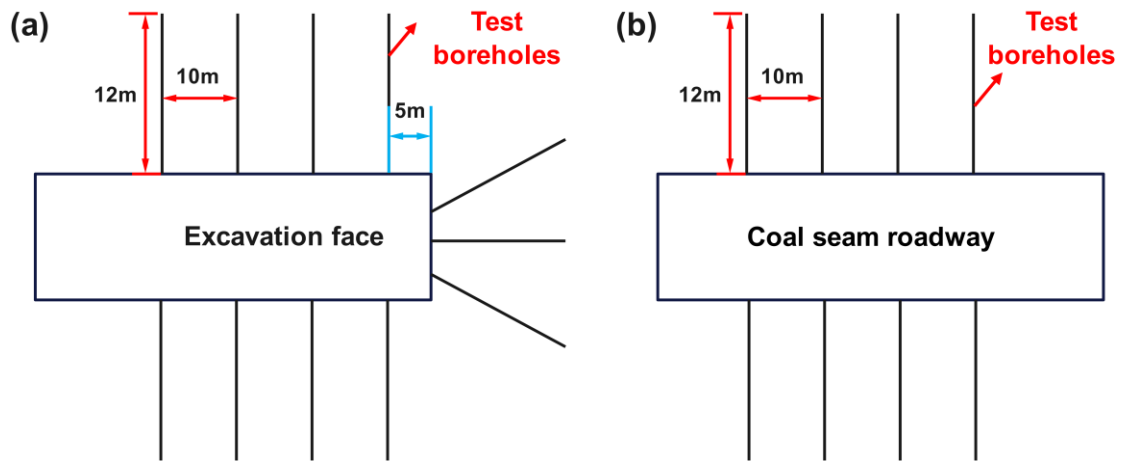


Fig. 24 (a) Determination of drilling layout by drilling cuttings method in heading face; (b) Checking borehole layout in dangerous area of roadway behind tunneling or during mining.

5.1.4 Displacement Surveying of Wall Rock

Deformation of the transport and return airways was observed during extraction of the East 1100 working face, with particular attention to roof separation at depth. Figure 25a records deformation along the East 1100 return airway. At 140 m, the roadway is 2.8 m wide and 2.58 m high and deformation is pronounced. Between 140m and 360m, both ribs and the roof of the East 1100 return-air crossheading show severe deformation; beyond 360 m, deformation is limited. The surrounding rock consists mainly of coal with a small amount of rock (Fig. 25b). Field observations indicate that the East 1100 return airway received 30 m of advance support, supplemented by individual supports and lace beams in the severely deformed zone (Fig. 25c). Gas at the

working-face upper corner was also monitored. It increased slightly during coal cutting but remained close to 0.2% at other times.

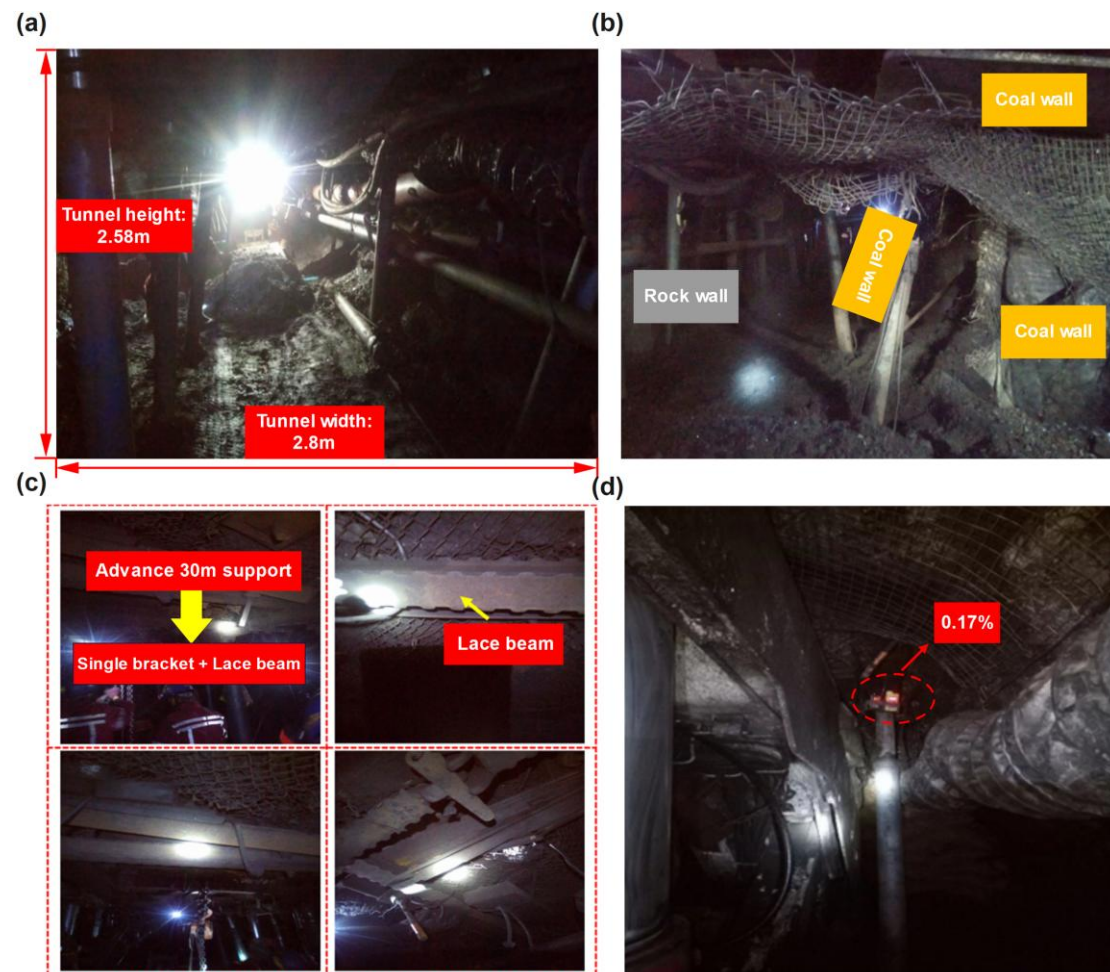


Fig. 25 (a) Deformation of the roadway at 140m in the return airway of the east 1100 working face; (b) Distribution of surrounding rock in tunnels; (c) Support method; (d) Gas concentration in the upper corner.

5.2 Prevention and Control Measures

5.2.1 Gas Pre-Drainage Measures in Fault Geological Structure Zone

In the East 1100 section, coal-seam gas pre-drainage ahead of connection of the two crossheadings was carried out from the transport trough. The East 1100 advance return-airway section extended at least 200 m, and at least three drilling fields within this section completed fan-shaped bedding drilling and pre-drainage of gas ahead of the

return-air crossheading (Fig. 26). Residual gas content or residual gas pressure was first calculated from gas extraction and emission in the test unit. After the required target was reached, the residual gas content or pressure was measured directly in the field and used to assess pre-drainage effectiveness. An area with residual gas pressure below 0.74 MPa or residual gas content below 8m³/t (6m³/t in the tectonic zone) was considered effective; otherwise, it was classified as ineffective.

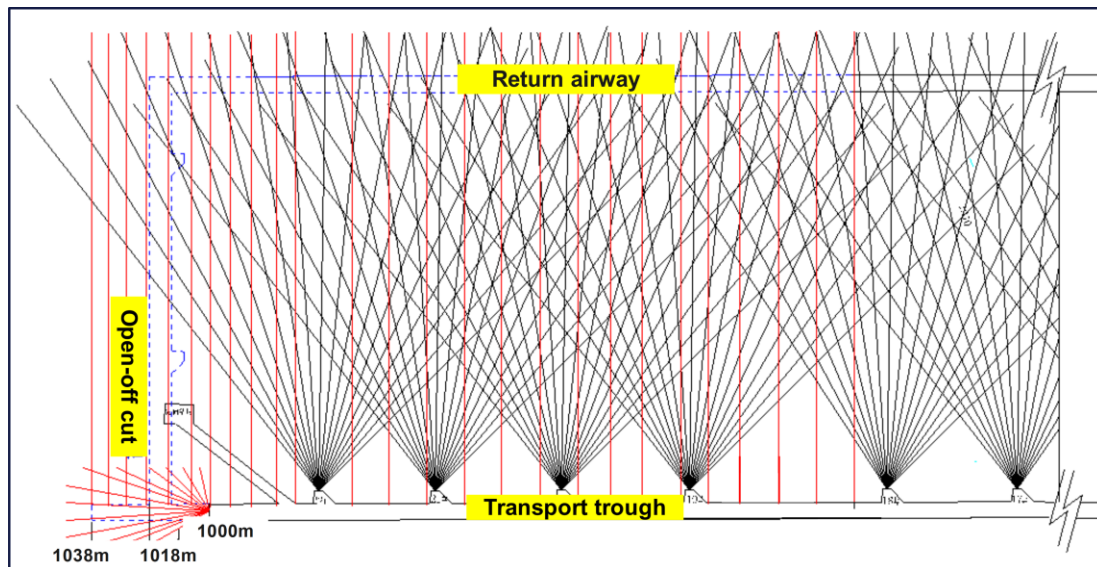
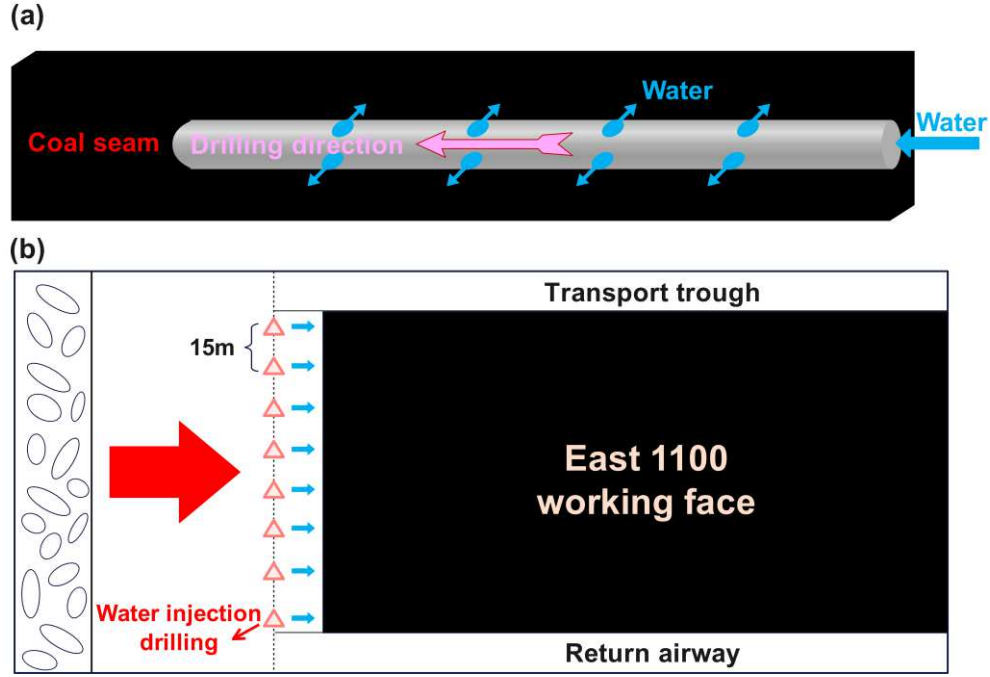


Fig. 26 Fan shaped hole pre extraction working face area gas.

5.2.2 Coal Seam Water Injection

Coal-seam water injection can use long, short or deep boreholes, with either static-pressure or dynamic-pressure water supply. A schematic of the injection model is shown in Fig. 27a. The selected method for the East 1100 working face was determined from seam permeability, the interaction between mining and injection, roadway layout, coal burial depth, the extent of roof-rock softening by water, field conditions and trial results. Static-pressure coal-seam water injection was therefore adopted.



ig. 27 (a) Schematic diagram of coal seam water injection model; (b) Static pressure water injection system for working face.

The injection parameters were selected to ensure complete and uniform wetting within the effective infiltration range of each borehole. Equation (1) gives the single-hole water-injection volume for the coal seam.

$$Q = \varphi X_1 X \rho \delta \quad (1)$$

where Q is the single-hole coal-seam water-injection volume (m^3); φ is the coal wetting coefficient ahead of the borehole, taken as 1.2; X_1 is the borehole spacing (m); X is the borehole length (m); ρ is coal density (t/m^3); and δ is the water injected per tonne of coal, $0.03 \text{ m}^3/\text{t}$.

The calculated single-hole injection volume for the working face is 6.9 m^3 .

Equation (2) defines the required coal-seam water-injection time.

$$t = \frac{Q}{q} \quad (2)$$

where t is the coal-seam water-injection time (h) and q is the injection flow rate, $4 \text{ m}^3/\text{h}$.

The calculated injection time is 1.7 hours.

The coal firmness coefficient at the East 1100 working face ranges from 0.15 to 0.34, indicating soft coal. Equation (3) was used to calculate the distance that water injection should extend ahead of the working face.

$$L = \frac{tv}{24} + L_t \quad (3)$$

where L is the distance of water injection ahead of the working face (m); v is the daily advance of the working face (m); and L_t is the distance between the borehole and the working face when injection is stopped, taken as 10-20 m.

The calculated value is $L = 15.07\text{m}$; the design value was rounded to 16m.

On this basis, the East 1100 working-face static-pressure injection system used 8 water-injection boreholes, spaced 15 m apart, with a single-hole depth of 10 m (Fig. 27b).

5.3 Effect Test

Changes in the East 1100 working-face outburst indicators provide an additional measure of the water-injection effect on gas control. Figure 28a shows the temporal variation in the daily maximum drilling-cuttings amount S_{max} and K_1 during a mining interval. Both indicators remained above their critical values during this interval, indicating that the coal-seam water-injection measures influenced the working-face outburst parameters.

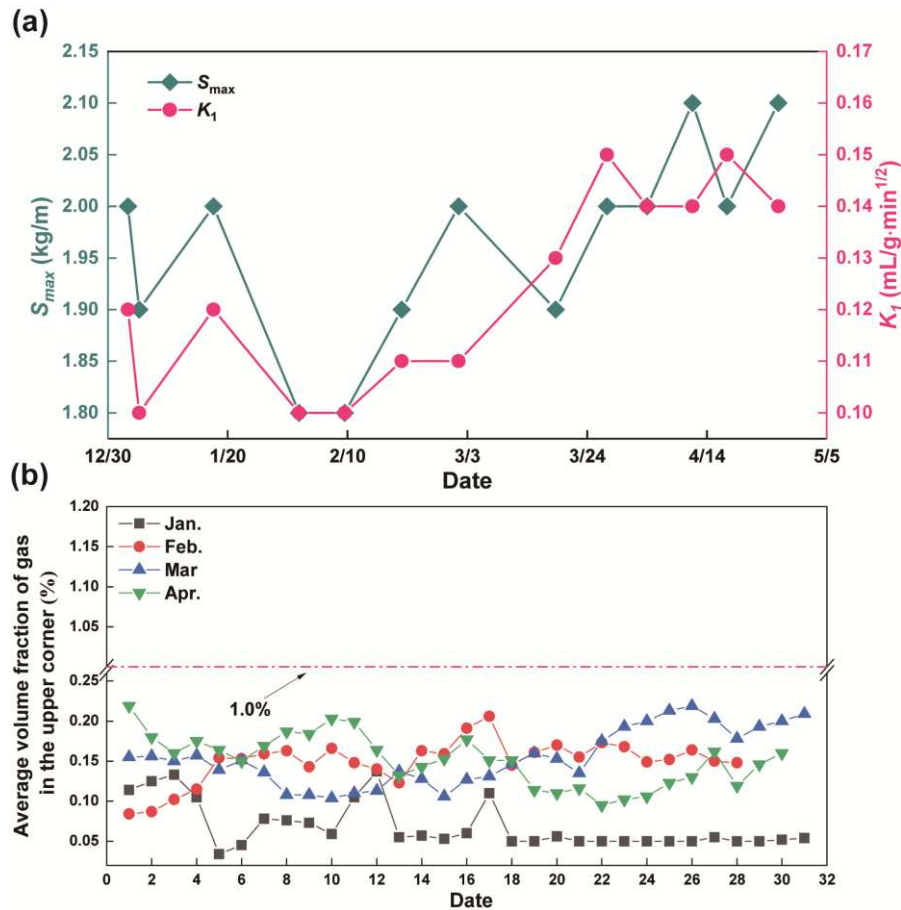


Fig. 28 (a) Variation trend of the maximum daily drilling cuttings S_{max} and K_1 values during the mining period of the east 1100 working face; (b) Change trend of average gas concentration in upper corner of working face.

The mean upper-corner gas concentration of the East 1100 working face through May and across the mining period is shown in Fig. 28b. From January to April, the concentration remained below 1.0% and did not reach the over-limit value, indicating controlled gas conditions during extraction. Water injection was one component of the broader control programme. As the East 1100 face advanced into deeper coal, upper-corner gas concentration increased gradually; the April mean was 0.082% higher than the January mean. Thus, static-pressure coal-seam water injection should be considered an important part of gas control.

Comparison of gas emission before and after implementation of the measures

shows a 32% reduction after treatment. The upper-corner gas concentration remained within the limit, and S_{max} and K_1 stayed at normal levels, supporting the effectiveness of the integrated control programme.

6 Conclusions

1. In a high-stress, extra-thick coal seam, high ground stress supplies the energy source for rock burst, while pore compression favours gas accumulation. Together these processes form a potentially unstable high-stress-high-gas-thick-coal system. At the East 1100 working face of Weijiadi coal mine, horizontal stress dominates; the maximum principal stress is 39.79 MPa and the field is a geodynamic (compressional) type, providing the stress basis for the coupled hazard.

2. Mining produces marked stress concentration in the narrow coal pillar. At the stop line, pillar stress reaches 59.67 MPa, the stress concentration factor is 4.19 and the peak lies 12.18 m from the working face. Stress continues to increase during advance, and the concentration factor near the fault reaches 3.88, increasing dynamic-disaster risk.

3. Reverse-fault activation superposes mining stress and raises the peak mining stress by nearly 1 times. K_1 and S_{max} also increase substantially in the fault zone and show a positive quadratic relation with mining stress. This correspondence identifies the fault as an important control on hazard triggering, while the field data support an association rather than a standalone causal proof.

4. Under the four-in-one prevention and control system, fan-shaped-hole gas pre-drainage, static-pressure coal-seam water injection and stress-electromagnetic-

radiation monitoring reduced working-face gas emission by 32%. Upper-corner gas concentration stayed below 1.0%, and Smax and K1 were below their critical values, indicating effective suppression of coupled-disaster risk during the monitored mining period.

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AUTHOR DECLARATIONS

Conflict of Interest

The authors declare no competing interests.

Author Contributions

Xiang Fu: Investigation (equal); Methodology (equal); Project administration (equal); Resources (equal); Supervision (equal); Visualization (equal); Funding acquisition (equal). Rongkun Liu: Conceptualization (equal); Data curation (equal); Formal analysis (equal); Software (equal); Writing-original draft (equal); Writing-review & editing (equal). Xuan Liu: Data curation (equal); Writing-review & editing (equal). Yujie Zhang: Validation (equal). Dawei Song: Methodology (equal). Yifan Wang: Supervision (equal).

667 **DATA AVAILABILITY**

668 The data supporting the findings of this study are available from the corresponding
669 author upon reasonable request.

670 **Funding Declaration`**

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