



Risks of coal mining in fractured environments

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Abstract

The little understanding of mining-induced coal face behaviour under near-seam and thick strong strata embedded with pre-existing discrete fractures is addressed in this study. The deeper understanding of longwall stress distribution and resultant face behaviour (deformation, failure, spall, fall) are reliably achieved through panel-scale numerical simulations with realistic incorporation of discrete fractures in not only coal seam but also main roof strata. The study better characterises the shape of stress release zone, which should be “pear” rather than “elliptical” shape. The stress change in direction and magnitude is extensively demonstrated from coal seam to surrounding rocks. The front abutment stress is found up to 2.88 times the pre-mining stress and located 24 m ahead of coal face. The compressive stress is confirmed to re-develop in the reconsolidated rocks to its pre-mining value and can be greater in places. The study reveals that the coal face is more stable when the strength and elastic modulus of main roof rock increase. Similar face behaviour is manifested when the distance between coal seam and main roof increases. In contrast, the face is found to be less stable when the thickness and fracture density of main roof increase. Behind shield support, the top coal behaviour shows clear relationships with main roof rock strength, main roof strata thickness, and distance between main roof and seam floor. The study’s findings help engineers in better understanding longwall coal face behaviour in less favourable geo-mining conditions from which safe and environmentally sustainable coal mining can be better maintained.

Keywords Underground mining · Coal face behaviour · Near-seam strata · Thick strong strata · Discrete fractures · Roof strata characteristics

Introduction

Despite a pledge from 151 national governments to achieve net-zero emissions, global coal production reached a peak in 2023 and continues to increase until 2030 according to the governments’ plans and projections (SEI et al. 2023). This challenge forces coal operators to further improve their most productive and environmentally friendly mining methods such as conventional longwall and advanced longwall

with great cutting height and/or caving height. Since the coal reserves in favourable geological conditions for longwall mining become depleted, the mining in less favourable conditions such as under near-seam and thick strong strata, where coal reserves occupy a significant proportion worldwide but coal mine ground control is difficult, is of urgent need. Experiences from different coal industries show that under such geo-mining condition, critical coal face behaviours such as great stress concentration, spall, burst, collapse, and accompanying rock fall can occur, causing serious environmental impact and unsafety at work (Das et al. 2023; Gray and Gibbons 2020). For cases where thick strong strata are embedded with pre-existing discrete structures such as fault-associated fractures, the strata movement caused by coal mining significantly breaks these structures. This creates potential paths for methane gas emission or for water inrush flooding mining space, as occurred in, for example, Vietnam coal industry (Vang Danh Coal Company 2017). From a geotechnical perspective, an in-depth understanding of mining-induced coal face behaviour under near-seam and

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thick strong strata embedded with pre-existing discrete fractures is therefore important, providing coal operators with fundamental science to maintain sustainable coal extraction.

Mining-induced coal face behaviour under near-seam and thick strong strata has been well documented in the literature (Galvin 2016; Gray and Gibbons 2020; Zhu et al. 2024). The behaviour, however, was mostly caused by either massive or uniformly jointed overburden strata due to the strata's sedimentary nature. The face behaviour caused by the strata embedded with non-uniformly distributed structures such as fault-associated fractures has been poorly understood. Based on mining practice, a study commissioned by the Commonwealth of Australia (2014) stated that when fractures occur in relative strong and massive strata, their breakage releases considerable strain energy. The impact of breakage location on surface, rather than underground mining face, was recognised. From analysis of microseismicity data, Cao et al. (2020) confirmed the presence of pre-existing natural fractures in coal seam and surrounding rock mass, which can constitute discrete fracture networks. An associated model embedding discrete fractures in coal measure rocks was developed to analyse longwall mining-induced microseismicity, while coal face behaviour was not studied (Cao et al. 2018). The discrete characteristics of pre-existing fractures in coal seam were presented in Wang et al. (2017) and Le and Oh (2022). While the former focused on coal anisotropic behaviour and first roof weighting, the latter analysed coal face behaviour that is predominantly controlled by coal seam and immediate roof strata. Wang et al. (2020a) also analysed coal face behaviour in where the seam contains discrete fractures. The roof strata were, however, considered as continuum media without pre-existing fractures. In contrast, Arasteh et al. (2022) treated coal seam and roof strata as discontinuum media. However, because the synthetic rock mass method was used, realistic representation of pre-existing fractures was not performed in the study. Apart from coal mining, the discrete characteristics of fractures in sedimentary strata were shown in Wei et al. (2022). Although the research objective was not coal face, the study demonstrated the effect of randomly distributed discrete fractures on rock mass stability around an underground cavern. In general, all the above works revealed little understanding of coal face behaviour under near-seam strata embedded with discrete fractures. The reason for this is probably the insufficient consideration of strata pre-existing fractures, which can be further attributed to the great number and variation of the structures in practice and intrinsic limitation of each analysis tool.

This paper presents a detailed study of coal face behaviour in longwall mining under near-seam and thick strong strata embedded with pre-existing discrete fractures. The face (including its overlying top coal) behaviours from

a typical mining example and caused by variation of key main roof characteristics are investigated. Field measurement data of coal seam, roof strata and support behaviours are analysed that subsequently support a two-dimensional (2D) Distinct Element Method (DEM) simulation of longwall mining at panel scale for geomechanical analysis. The results from study can be used to improve panel design and face stability control towards environmentally sustainable coal mining in less favourable geo-mining conditions.

Vang Danh coal mine and field measurement

Vang Danh coal mine

Vang Danh coal mine is operated by Vang Danh Coal Joint Stock Company (a subsidiary of Vietnam National Coal and Mineral Industries Holding Corporation Limited—Vinacomin) and located at Quang Ninh coal field in the northeast of Vietnam (Fig. 1). The coal-bearing strata in the mine were formed in the Late Triassic, Norian to Rhaetian Age, Hon Gai Formation (T_3n-rhg) (Nguyen et al. 2022). According to Vinacomin Institute of Mining Science and Technology (2016), the mine contains two main fault systems that strike in meridian direction (F_{13} , F_{12} , F_{11} , F_{10} , F_8 , F_6 , F_5 , F_4 , F_3 , F_2 , F_1 , and F_0) and parallel direction (F_{40} , N_{20} , F_N , and F_M). A representative geological cross section along seam strike of the area being studied is presented in Fig. 2. The cross section shows that the area is limited by two main faults F_2 and F_3 . The stratigraphic sequence in some exploration drillholes is also incorporated in the figure.

Apart from Seam 8 whose roof strata are relatively easy to cave, Seam 7, Seam 6 and Seam 5 all have relatively strong and hard-to-cave main roof (Phung et al. 2022). At Seam 7 where longwall face instability and roof hanging have occurred, the seam has an average thickness of 9 m and dip angle of 10 degrees (Pham 2022; Vang Danh Coal Company 2017). The seam has an average overburden thickness of 371 m. The immediate roof of the seam is moderately bedded siltstone with an average thickness of 4.5 m. The immediate rock has an average Uniaxial Compressive Strength (UCS) of 53.42 MPa. The main roof is thickly bedded sandstone with an average thickness of 33 m and UCS of 104.46 MPa. The observation at exposed areas from previous open-pit mining and during roadway development confirms the presence of many panel-scale fractures whose strike direction is similar to that of Fault F_3 . This fault, as reported by Vinacomin Mining Geology Joint Stock Company (2016), plunges to the west (skewed to the north) with a dip angle of 60–70 degrees. An empirical assessment of Seam 7's panel stability indicated that the immediate roof was not sufficiently thick to fulfil the mined-out area if the



Fig. 1 Location of Vang Danh coal mine, Quang Ninh province, Vietnam (modified from Quang Ninh Portal [2025](#))

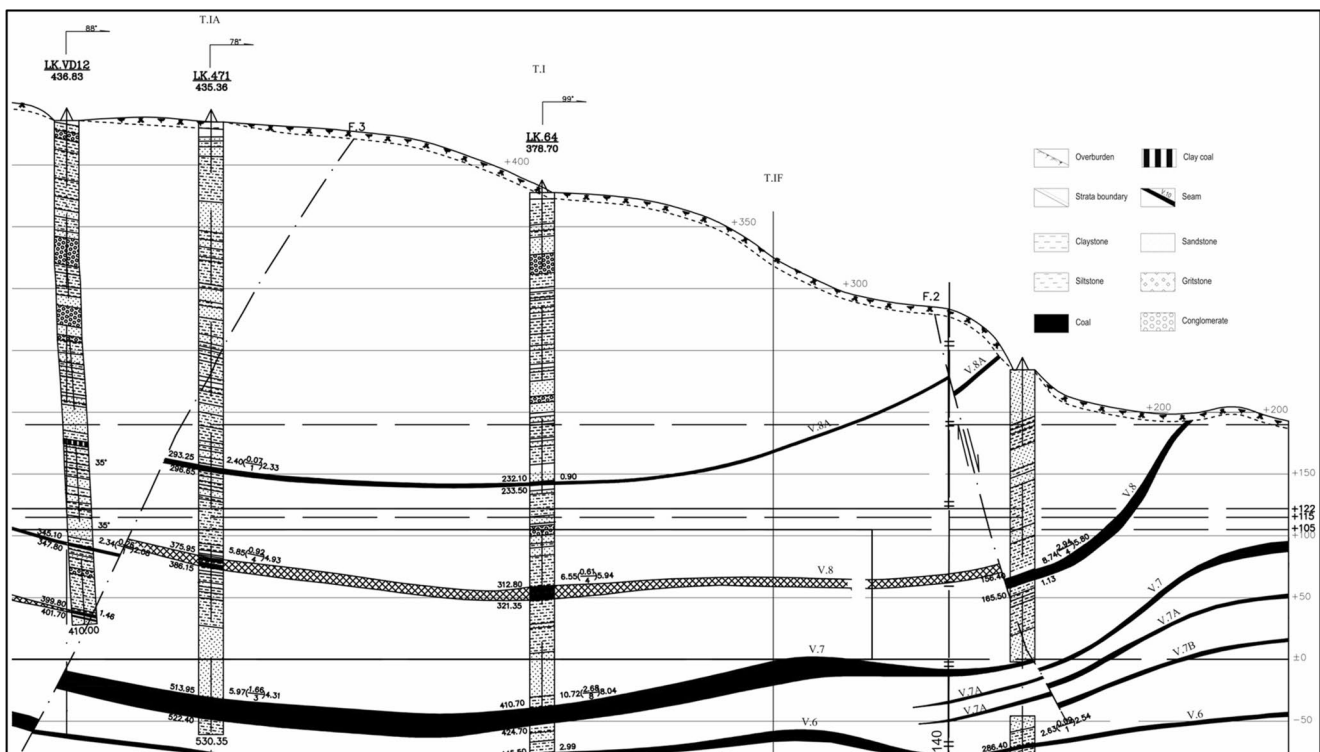


Fig. 2 Geological cross section along seam strike, Vang Danh coal mine (modified from Vinacomin Mining Geology Joint Stock Company 2016)

whole seam thickness was to be exploited (Pham 2022). The main roof would significantly impact the panel stability. The maximum cantilever beam length of immediate roof and main roof was calculated to be 0.7–3.2 and 43–75 m, respectively. The risk of roof weighting was identified.

Analysis of field measurement

A field measurement of shield support behaviour was done in the mechanised longwall panel I-7-5 at Seam 7. The panel used 4-leg shields with a maximum support capacity of 4400 kN (maximum leg pressure of 35 MPa) and a working height in the range of 1700–2800 mm. Because there was no real-time monitoring of shield behaviour due to lack of instrumentation, the shield leg pressure was manually measured by reading the two pressure gauges attached to front and rear legs. The piston displacement was manually measured at the same time. To avoid interruption of production and maintain a consistent time, the measurement was implemented at the beginning of each working shift before shearer operation. According to the panel design, there was three shifts per 24 h in which one shift performed one

mining cycle with a web cut of 0.6 m in bi-di mode. The cutting height (in bottom coal section) and caving height (in top coal section) were 2.7 and 6.3 m, respectively. The measurement period was from May 2023 (panel commencement) to September 2023, corresponding to 150 m of face retreat. The leg pressure and piston displacement of a shield operating at the mid-panel width are shown in Fig. 3.

Figure 3 shows that within the first 150 m of face retreat, shield leg pressure at the mid-panel width fluctuated in the range of 16–25 MPa, while its piston displacement was in the range of 0.7–0.9 m. Within the first 65 m of retreat, shield leg pressure followed cycles of change (increase and decrease) in which the peak value gradually increased. As the face started retreating out of installation roadway, the first cycle of pressure change completed after 12 m of face retreat. The pressure increased and decreased in the range of 16–21 MPa. The piston displacement in this cycle decreased to a low value of 0.7 m (rear leg) before restored a setting value of 0.8–0.9 m. This pressure change cycle was due to the start and completion of initial (first) caving event of top coal reported by on-site engineer. The second change cycle completed after 25.8 m of face retreat. The pressure change was in the range of 18–24

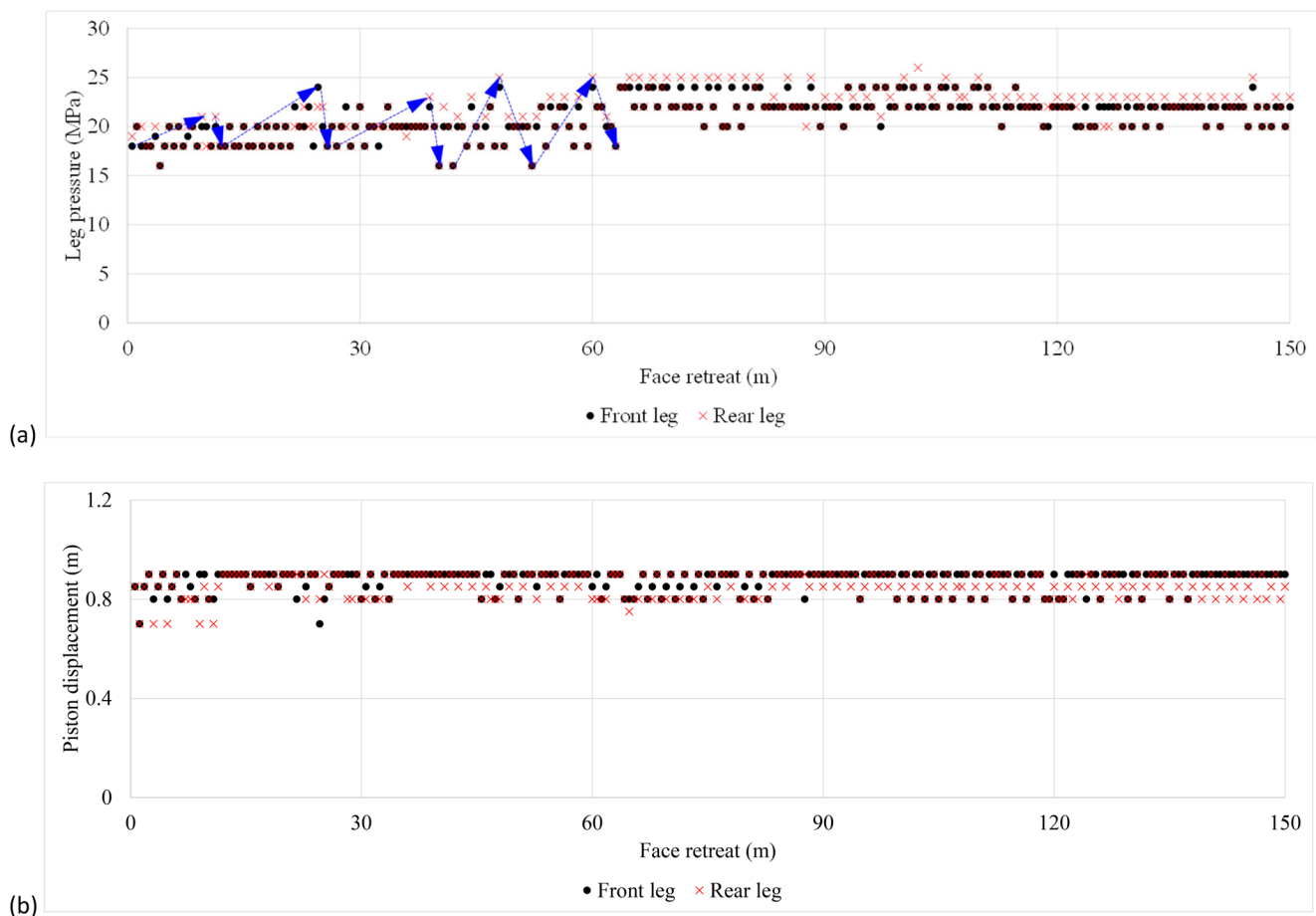


Fig. 3 (a) Leg pressure and (b) piston displacement of shield at middle width of Panel I-7-5

MPa, which was greater than the range in previous cycle. A low piston displacement value of 0.7 m was also measured in the cycle. This cycle was in accordance with the start and completion of initial caving event of immediate roof.

The next cycles were associated with the sagging and rupture of main roof layers. For example, after 39 m of face retreat, the shield pressure reached a peak of 23 MPa (rear leg) before dropping to 16 MPa in next two cuts. The pressure increased to next peaks of 25 MPa (rear leg) after 48 and 60 m, respectively, before dropping to a low value of 16–18 MPa. A decrease in leg pressure at 63 m of retreat and in piston displacement at 64.8 m of retreat could indicate the completion of initial rupture of all main roof layers.

For the last 85 m of retreat in the measurement period, the shield leg pressure followed a steady fluctuation that indicated steady-state movements of panel roof strata. The cycles of pressure change corresponded to an interval in the range of 16.2–18.6 m of face retreat. There was no noticeable decrease in piston displacement in this retreat period. The fluctuation of shield leg pressure in this study shows a similar trend to that in another panel at the same mine (Le et al. 2024), although the peak pressure was greater in the past study due to the monitoring location near gateroads.

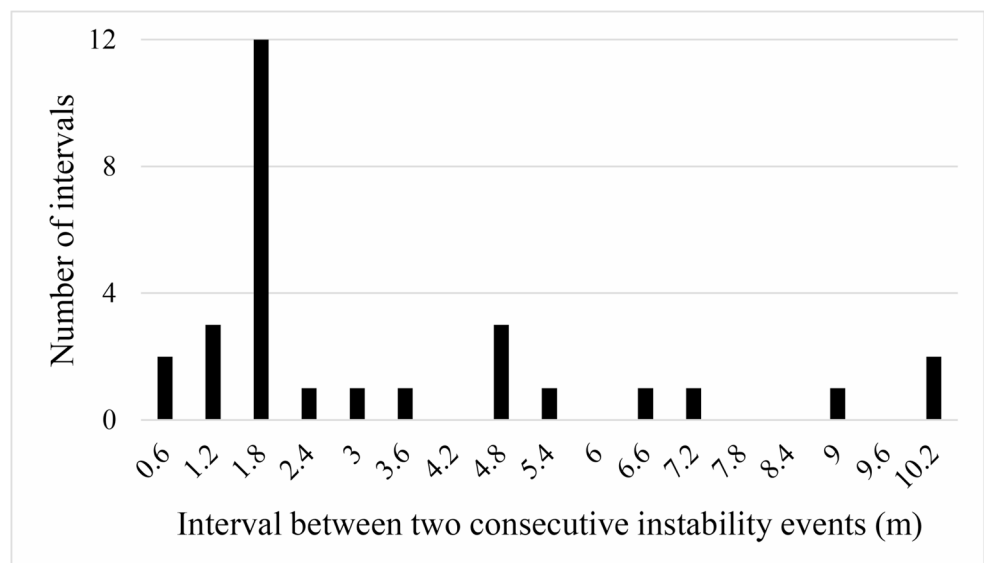
The face behaviour was additionally measured via the retreat interval between two consecutive face instability events (face spall and top coal fall above face) and the number of intervals, as shown in Fig. 4. Although the extent of each face instability was not measured due to safety rule, the monitoring shows that the face instability could occur after a minimum of one web cut (0.6 m, 2 events) to a maximum of 17 web cuts (10.2 m, 2 events). While the face instability commonly occurred after three web cuts (1.8 m, 12 events) due to cyclic caving of top coal and immediate roof, the instability also occurred in varied intervals of face retreat. These varied

intervals were likely associated with the hanging and rupture of main roof layers which contained discrete fractures.

Simulation of longwall under strata embedded with discrete fractures

The computer-based numerical simulation method is selected due to its capability of investigating a great number of varied geological structures within a limited time period. The 2D DEM-based code UDEC (Itasca Consulting Group 2019) is used to build a simulation model for studying longwall coal face and top coal behaviours. This is because the code is capable of simulating various rock behaviours (stress, deformation, failure, strength reduction, and detachment) from local scale (at face) to panel scale (roof strata). The Discrete Fracture Network (DFN) module built in UDEC is used as it can realistically simulate the pre-existing discrete fractures in rock strata. For determination and use of DFN parameters in this study, the flowchart for development of UDEC-DFN model in Le and Oh (2022) can be referred to. In the first step, field mapping and observation are performed at outcrop, roadway or coal face to collect data. In the second step, the data is analysed to determine fracture size, orientation, position and frequency/density characteristics. In the third step, a fracture template file including fracture size, orientation and position distributions is created. In the last step, different realisations of fractures are generated from the template depending on random seed. A realisation that results in initial caving event of top coal similar that in practice is finally incorporated in the UDEC model. Since the code is two-dimensional in nature, the coal face behaviour along the panel strike (whose dimension is much greater than that along panel width) is to be analysed.

Fig. 4 Number of and interval between face instability events



Model construction

The layout of Panel I-7-5 and location of cross section AA for longwall simulation is presented in Fig. 5. The cross section is located at the mid-panel width and along panel strike. The model construction started with the creation of model geometry. In vertical direction, only the rock strata that significantly affected coal face behaviour were explicitly simulated. The thickness of these strata, which can be roughly estimated by many empirical equations, were determined to be 120 m, including 9 m coal seam, 4.5 m immediate roof of siltstone, 33 m main roof of sandstone, and 73.5 m upper strata of siltstone. The rest of the overburden thickness, which was 260 m, was implicitly simulated by applying a vertical compressive stress of 6.81 MPa along the model top boundary. The

thickness of floor strata was simulated to be 45 m of siltstone, minimising the model bottom boundary's impact on the area of interest. The total model height was 165 m. In horizontal direction, the length of roof strata should be sufficient so that the main roof movement can fully impact coal face. From the analysis of strata movement in previous section, the retreat length was determined to be 150 m, including a 6 m width of installation roadway. The total model length was accordingly 600 m (five times the retreat length) to minimise the impact of artificial vertical boundaries on coal face behaviour. The model geometry is shown in Fig. 6.

The geological structures in model included bedding planes, vertical joints, and discrete fractures, as shown in Fig. 6. Outside the area of interest, only bedding planes and vertical joints were assigned. While the bedding spacing

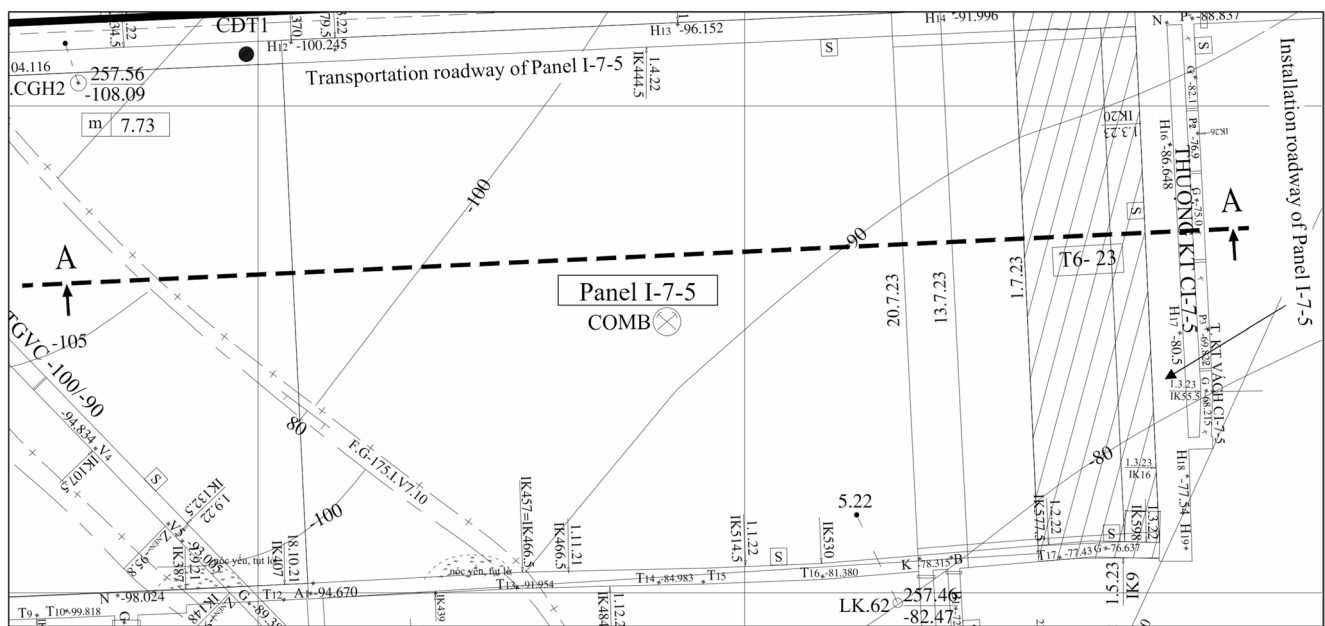


Fig. 5 Layout of Panel I-7-5 and location of cross section AA for simulation

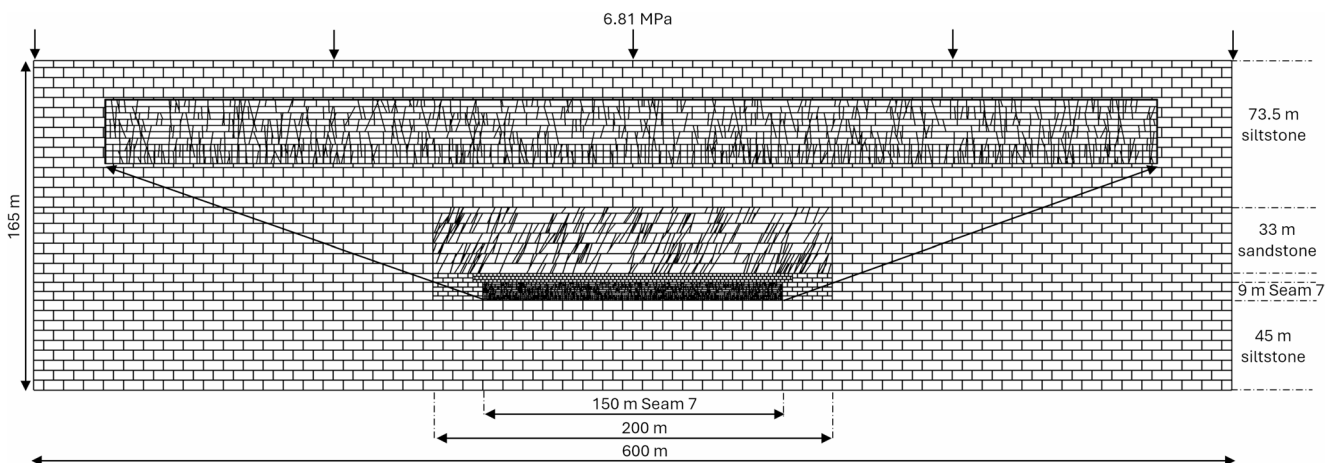


Fig. 6 Simulation of Panel I-7-5, Vang Danh coal mine

was not greater than 5 m, the vertical joint spacing was one to two times the bedding spacing in the same strata. Inside the area of interest, the bedding planes were assigned to coal seam, immediate roof and main roof with a spacing of 0.9, 1.5 and 5 m, respectively. In main roof strata, a network of pre-existing discrete fractures was generated using the DFN module in 200 m length at model centre. Based on the description of panel-scale fractures in previous section, the fracture orientation was observed to follow Gaussian distribution with a mean dip angle of 115 degrees (from the positive x-axis) and a standard deviation of 5. The fracture position and fracture size, due to limitation in observation area and lack of scaling exponent parameter, were assumed to follow uniform distribution. The fracture size limit was defined so that one fracture may cut one or several bedding planes. Since the mass density for stopping fracture generation was not able to measure in main roof, it was estimated to be one-fifth to one-third of the density measured in coal seam, based on field experience (Pham 2012). In coal seam, another fracture network was similarly generated. Based on the observation at in-seam roadway's sidewall and for model simplification, the fractures were assumed to be more or less vertical in 150 m length at model centre. The density of coal fractures was 0.7 m/m^2 , as roughly measured in the field. Note that in coal seam, fictitious vertical joints were assigned to the bottom coal section for web cutting.

The intact blocks in model were assigned with the strain-softening constitutive law. This law can simulate both block failure (via plasticity indicators) and post-failure strength reduction associated with longwall mining. The specific parameters required for the constitutive law were determined as follows. The block cohesion and friction were calculated from the UCS value of rock at panel scale using Mohr-Coulomb failure criterion. The panel-scale UCS value was derived from its laboratory-scale value using a reduction factor, accounting for the heterogeneity and presence of micro-structures within rock in the field. The laboratory UCS was tested by industry (Vinacomin Mining Geology Joint Stock Company 2016), while the reduction factor was adopted from empirical studies (Wilson 1980). The bulk and shear moduli were derived from rock elastic modulus at panel and a Poisson's ratio of 0.25. The panel-scale elastic modulus was derived from the laboratory-scale UCS using a modulus ratio (McNally 1996) and a reduction factor commonly used in past simulations (Mohammad et al. 1997). The tensile strength of rock was obtained in a similar manner. For softening behaviour, the critical strain and strength reduction rate were adopted from the literature (Le and Oh 2022; Zhou et al. 2023). The adoption was due to (i) the lack of test apparatus for post-peak strength parameters of rock mass and (ii) the similarity in settings of large-scale longwall mining simulation between the studies. Note that as the

softening behaviour is associated with mesh-dependence, the zone size in block should be kept consistently through simulations after model calibration. For the geological structures, the area-contact Coulomb slip constitutive law was assigned due to its simplicity. The stiffness of structures was derived from a trial-and-error process considering solution time and block interpenetration. Since tensile strength and cohesion of structures were almost zero in the field, their friction controlled the strength and were adopted from Pham (2012). The parameters of blocks and geological structures used for simulation are summarised in Table 1.

The pre-mining state of model was set by fixing the bottom and side boundaries and initialising the gravity-induced in-situ stresses. The vertical stress was equal to horizontal stress, as analysed from a nearby sedimentary basin in Vietnam (Vo and Chang 2020). The out-of-plane stress was additionally assigned to avoid material failure in this direction. It should be noted that the top boundary was assigned with compressive stress, as described earlier.

Model alternation and monitoring

One face cut or web cut (model alteration) was simulated by removing one meter depth of coal face in strike direction. After each cut, the shield support was installed near the new coal face, and the model was then cycled to an equilibrium state. During the cycling, any coal blocks spalled from face and fallen/caved from top coal section were recovered by deleting them from the model. The recovery areas were both ahead of shield and $3.0 \times 2.7 \text{ m}^2$ immediately behind the shield (Fig. 7). Before the next face cut, the support in previous cut was deleted and would be re-installed near new coal face. In this simulation, the shield support was represented by the built-in UDEC support members. The support had a supporting area of 4 m^2 , yield force of 2.94 MN, setting force of 2.35 MN, and stiffness of 130 MN/m.

Different behaviours of model were monitored. The magnitude of vertical stress along the mid-seam height was measured after each 20 m of face retreat. In addition, the magnitude of vertical stress at fixed locations (A1, A2, B1, B2), which were at 2.7 and 4.5 m above seam floor and 36 and 72 m away from installation roadway, were recorded as the face retreated (Fig. 7). The vertical and horizontal displacements of four block initially at the above locations were monitored. The number of coal blocks spalled from face and fallen from top coal section ahead of shield support (defined as face instability blocks) were counted in each face cut. Ahead shield support, the face instability rate was defined by dividing the number of face instability blocks by the number of pre-mining blocks within an average of 100 m coal seam. Behind shield support, the top coal recovery rate was defined by dividing the sum of top coal blocks

Table 1 Parameters used for simulation of panel I-7-5, Vang Danh coal mine

Coal measures	Block properties				Structure properties								
	Density (kg/m ³)	Uniaxial com- pressive strength (MPa)	Angle of internal fric- tion (degree)	Cohesion strength (MPa)	Tensile strength (MPa)	Critical strain (%)	Residual cohesion strength (MPa)	Residual tensile strength (MPa)	Elastic modulus (GPa)	Pois- son's ratio	Normal stiffness (GPa/m)	Shear stiffness (GPa/m)	Friction angle (degree)
Upper strata	2670	13.36	32	3.70	2.67	0.1	0.74	0.27	5.01	0.25	10	1	20
Main roof	2670	34.82	33	9.46	5.22	0.1	1.89	0.52	9.80	0.25	10	1	30
Immediate roof	2670	13.36	32	3.70	2.67	0.1	0.74	0.27	5.01	0.25	10	1	15
Coal seam	1650	4.50	22	1.52	1.13	0.5	0.30	0.11	1.58	0.25	10	1	15
Floor strata	2670	13.36	32	3.70	2.67	0.1	0.74	0.27	5.01	0.25	10	1	20

recovered during mining by the number of pre-mining top coal blocks. The top coal recovery rate was measured in the interval from 40 to 140 m away from installation roadway, minimising impact caused by panel ends. The UDEC built-in compiler FISH was used to write functions for the monitoring.

Model calibration and validation

Since several simulation inputs were derived from assumptions or previous simulation experience, the model was calibrated and validated to minimise the impact of the derivation on model outputs. The model was calibrated against the initial and cyclic caving events of coal seam in two steps. In the first step, the calibration was implemented by adjusting the random seed when generating DFN in coal seam. The random seed, as mentioned earlier, was adjusted to generate a DFN realisation that facilitated top coal caving. In the second step, the mechanical damping scheme in UDEC, which damps the motion for model equilibrium and thus affects coal face behaviour, was used. The damping coefficient was adjusted to match the seam caving against that in the field. The calibrated model induced the coal seam to start and complete its initial caving within the range of 6–16 m of face retreat. The simulated cyclic caving occurred readily after each face cut, similar to that observed in practice.

The calibrated model was run for further face retreats to validate against the caving and rupture of overburden strata. The simulation result shows that the first caving of immediate roof occurred after 26 m of face retreat, which was very close to that in the field (25.8 m). The rupture of the lowest main roof layers occurred after 39 m of face retreat, which was only 2 m less than that analysed in previous section. Another validation of the model was done regarding the force exerted by shield support. The simulated force mostly ranged from 1 to 3 MN, which covered the corresponding range of 1.34–2.09 MN at the panel. As analysed in the next section, the formation of stress release and concentration agreed with those in past studies. This also contributed to the model's validity. It should be emphasised that as the yield valve was not simulated, advanced simulation of support is recommended for future study for more realistic support behaviour.

Coal face behaviour under near-seam and thick strong fractured strata

It is widely known that an underground mining disturbs the initial equilibrium state of stress in rock strata, resulting in stress release and concentration zones whose dimensions increase to a certain level as the mining progresses. The

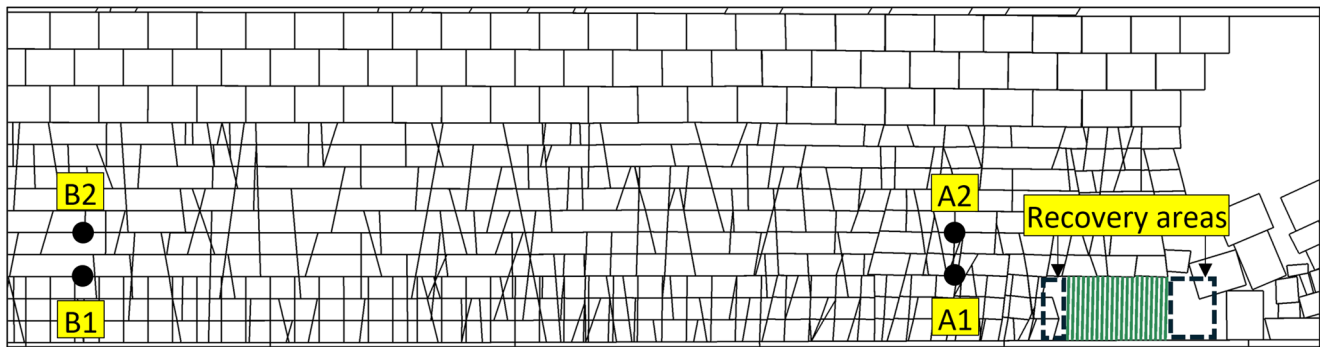


Fig. 7 Model support and monitoring

redistribution of stress is associated with the rock deformation, failure and displacement that would finally form a new equilibrium state of stress in underground. The change of stress and displacement occurs cyclically as the mining progresses, but the cycle characteristics may vary significantly depending on geological conditions (e.g., stress regime, rock mass properties) and technological design (e.g., panel geometry, equipment capacity, shearer and shield operation). An analysis of rock stress and displacement change under the condition of near-seam and thick strong strata embedded with pre-existing fractures is accordingly a prerequisite for better understanding of coal face behaviour.

Stress redistribution

An overview of stress redistribution caused by longwall mining can be seen through the redistribution of stress tensor in the Panel I-7-5 simulation (Fig. 8). Note that before mining, the major and minor principal stress tensor components were equal in magnitude, while their directions were mostly vertical and horizontal. Figure 8 clearly shows that once the coal was mined, the stress in the mined-out area and its close proximity was released and transferred to the surrounding area. For example, immediately above and below the mined-out area, the vertical component of stress tensor was significantly released, while the horizontal component could be increased and became the major principal stress (Fig. 8a–b). In contrast, right ahead of and behind the mined-out area, the vertical component was increased and became major principal stress, while the horizontal component was significantly released. The release of stress tensor in initial face retreats formed a stress release zone in “elliptical” shape. However, as the face retreated further, the release zone gradually turned to “pear” shape when the hanging roof caved or ruptured (Fig. 8c–f). The release of stress from caved/ruptured roof and the displacement of roof rocks (which was much greater than that of floor rocks) resulted in this pear shape. This simulation result better outlines the shape of stress release compared to that in previous

studies (Gao et al. 2014; Le and Oh 2022). The current simulation also shows that as the mining progressed, the stress release zone increased in its height and reached a maximum value after 80 m of face retreat (Fig. 8d). The height could not be greater (Fig. 8e–f) because the roof strata reached a steady-state movement. The current result is in line with that analysed from shield pressure in previous section. Since 80 m of retreat was significantly less than the total retreat (150 m), the model geometry was proven to be sufficient for studying coal face behaviour impacted by near-seam and thick strong fractured strata.

Inside the stress release zone, because more rock strata were broken and caved into the mined-out area, compressive stress started to re-develop in the rocks due to their own weight. After 80 m of face retreat, due to the bulking characteristics, the broken/caved rocks filled the mined-out area and connected the unbroken upper strata with floor strata. Under the movement of upper strata, the rocks were significantly re-consolidated, causing the compressive stress to greatly increase (Fig. 8d–f). Figure 8 shows that the maximum stress magnitude developed inside the stress release zone being 2, 4, 7.5, 7.5, 10 and 12 MPa after 20, 40, 60, 80, 100 and 120 m of face retreat, respectively. It should be noted that negative stress indicates compression due to UDEC convention. This result indicated that the rock stress was re-developed in the release zone and reached its pre-mining value (9.86 MPa at seam floor) after 100 m retreat. This means that a new adjacent working face can be safely operated when it is at least 100 m away from the current mined-out area. The result agrees well with the Vietnam mining practice in where the distance between two adjacent working faces is typically 100–400 m.

Outside and close to the stress release zone, the stress tensor changed in both direction and magnitude (Fig. 8). For the stress direction, the earlier analysis pointed out that above and below the release zone, the major principal stress was horizontal while the minor principal stress was vertical. In contrast, ahead of and behind the zone, the major principal stress was vertical, and the minor principal stress was

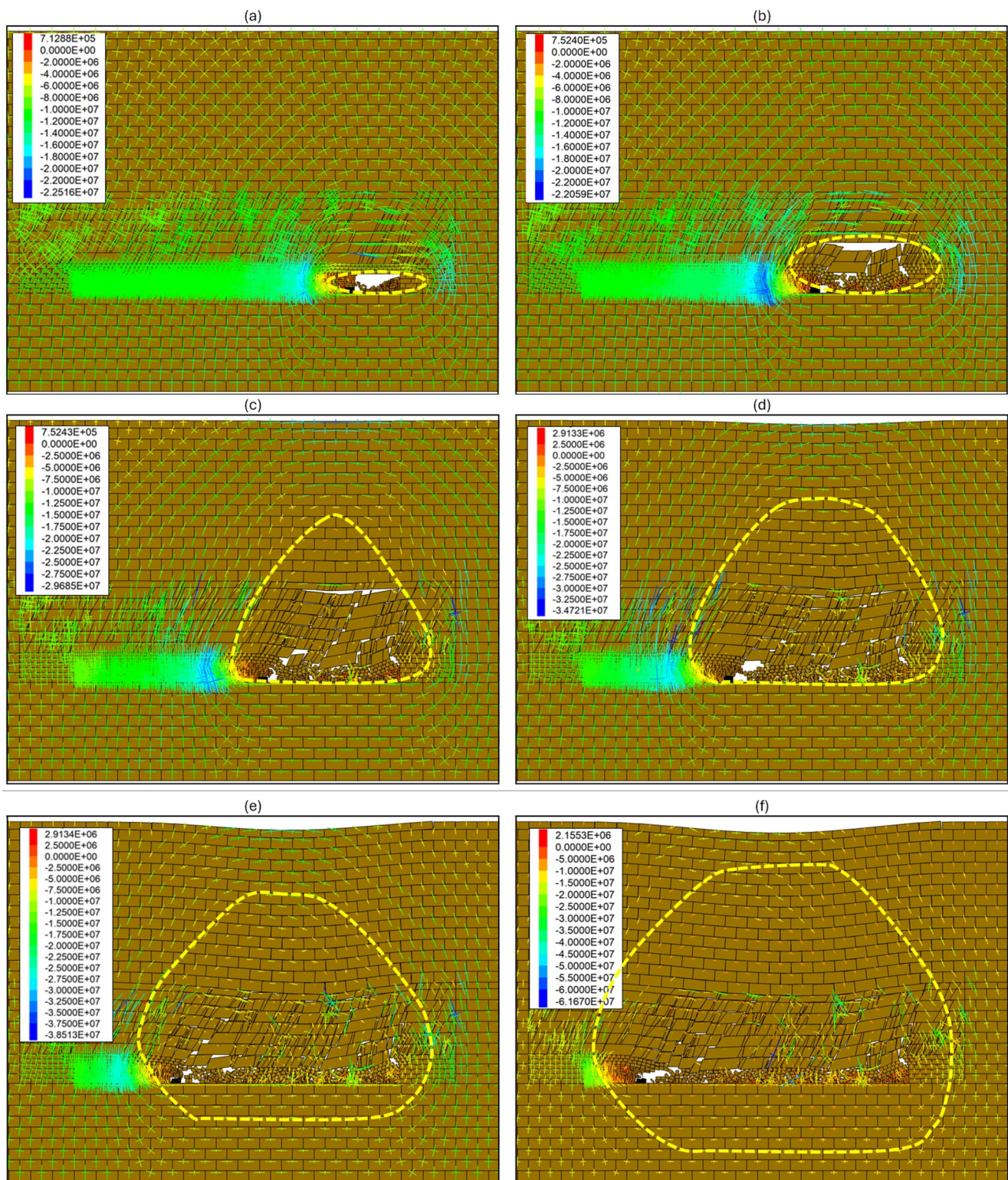


Fig. 8 Stress tensor at each 20 m face retreat

horizontal. In the transitional areas between the sides and top/bottom of release zone, the direction of major principal stress gradually rotated from vertical to incline and finally horizontal, or vice versa. Compared to past studies (Le et

al. 2018; Wang et al. 2020b), this study demonstrated the change in stress direction not only in coal seam but also in surrounding rocks. Figure 8 also shows that further away from the release zone, the stress tensor gradually recovered

its pre-mining direction. For the stress magnitude, the change was clearly seen above and at sides of the release zone. Above the release zone, initial mining formed a concentration of horizontal stress in immediate roof and main roof layers (Fig. 8a). As soon as these roofs caved/ruptured, the horizontal stress was released and transferred to upper rock layers for a new cycle of stress concentration (Fig. 8b). Similar to stress direction, the stress magnitude recovered its pre-mining value at further away from the release zone.

At sides of stress release zone, the change in vertical stress within coal seam was analysed because it directly impacted the mining. Figure 9 displays the vertical stress distributed at the mid-seam height and along panel strike. The figure shows that there was a concentration of vertical stress in unmined seam. The peak vertical stress was 19.5, 22.7, 23,

27.97, 28.42 and 25.51 MPa at 19, 20, 21, 24, 21 and 21 m ahead of coal face at each 20 m of face retreat, respectively. This means that the front abutment stress was up to 2.58–2.88 times the pre-mining stress and located within the range of 21–24 m ahead of coal face. When the face retreated more than 120 m, the peak vertical stress moved outside the area of interest where the zone size became inconsistent. Figure 9 also shows a stress concentration zone behind the mined-out area. Since there was no mining activity in this side, the peak vertical stress fluctuated in the range of 22–23 MPa and at 15–20 m behind the installation roadway. The stress concentration behind the mined-out area was less severe than that ahead of the area. The simulation results explain why at least 20 m long coal pillar is left at panel end for longwall mining in Quang Ninh coal field. Figure 9

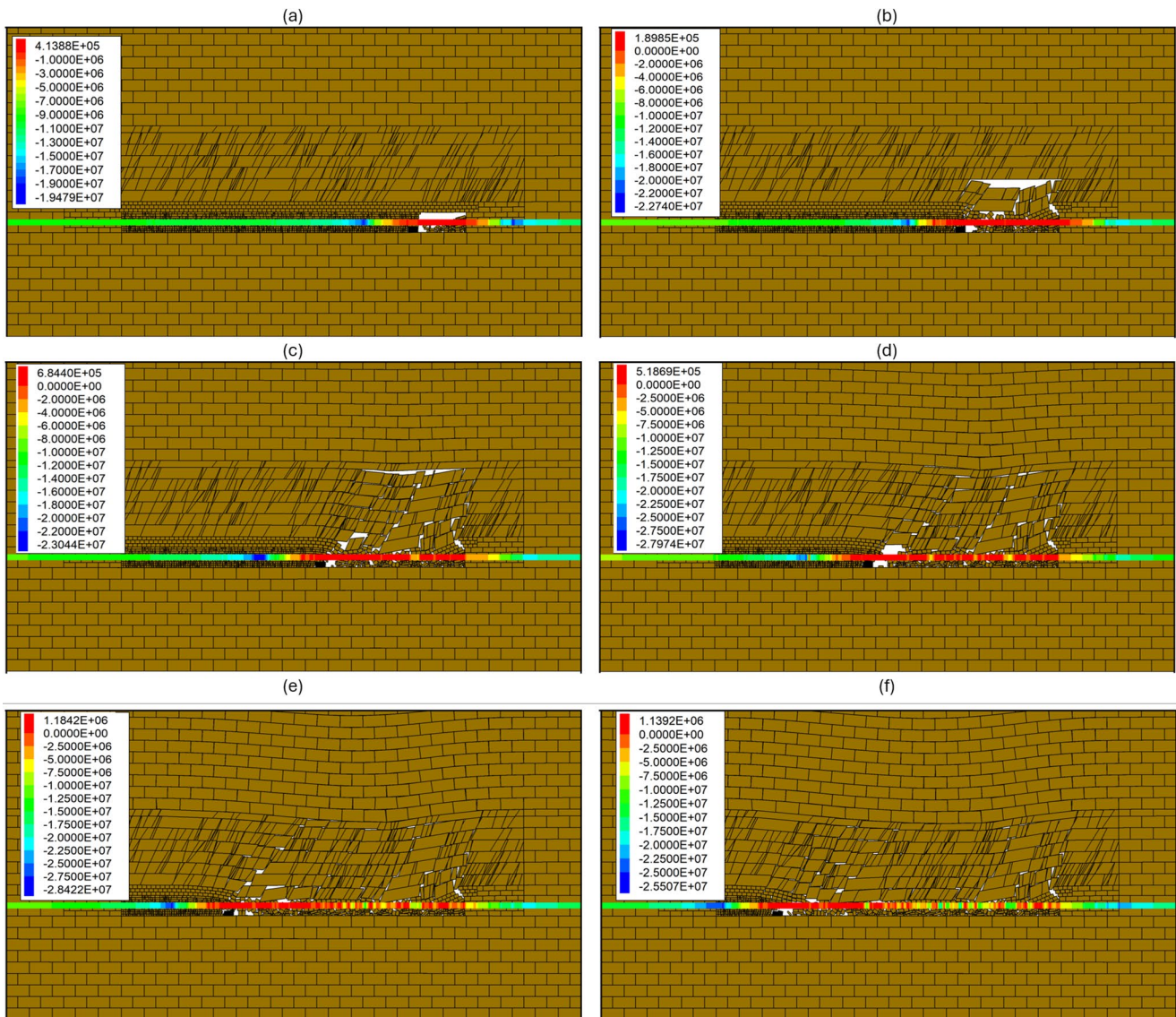


Fig. 9 Vertical stress at mid-seam height at each 20 m face retreat

additionally confirmed the re-development of compressive stress in mined-out area, which reached 12 MPa after 120 m of mining.

The change in vertical stress at fixed locations in coal seam is shown in Fig. 10. At locations 36 m away from installation roadway (A1 and A2), the change in vertical stress followed similar trends. When the face started retreating, the vertical stress at these locations increased and reached a peak value of 15–16 MPa after 14–16 m of retreat. This means that the peak stress was at 20–22 m ahead of coal face. When the face passed these locations, the stress rapidly decreased in next 10 m and fluctuated around zero in next 50 m of mining. This fluctuation was due to the fact that the monitoring locations were inside the caved but unconsolidated rocks. As the face retreated 85 m, the stress at Location A1 started to increase again. It recovered its pre-mining value after 110 m and reached a peak value (13 MPa) after 118 m of retreat. Similarly, at locations 72 m away from installation roadway (B1 and B2), the vertical

stress reached a peak value of 21.4 MPa at 23 m ahead of coal face. When the face passed these locations, the vertical stress also rapidly decreased and fluctuated around zero. The stress at Location B1 afterwards increased but less than 4 MPa. The results from Figs. 9 and 10 confirmed not only the stress concentration in unmined seam but also the stress re-development in mined-out area (Zhang et al. 2019). The stress in the mined-out area increased to its pre-mining value and was greater in places due to rock reconsolidation. The caved rock recovered its pre-mining stress after 100–110 m of face retreat, corresponding to 74 m distance between coal face and Location A1. This distance was about 0.2 times the overburden thickness, which was much smaller than the range of 0.3–0.4 times suggested by Whittaker (1974). The current study pointed out that for the mining under thick strong strata, the presence of pre-existing discrete fractures facilitates the caving of strata and reconsolidation of caved rocks. The time for commencing a new adjacent face can accordingly be shortened.

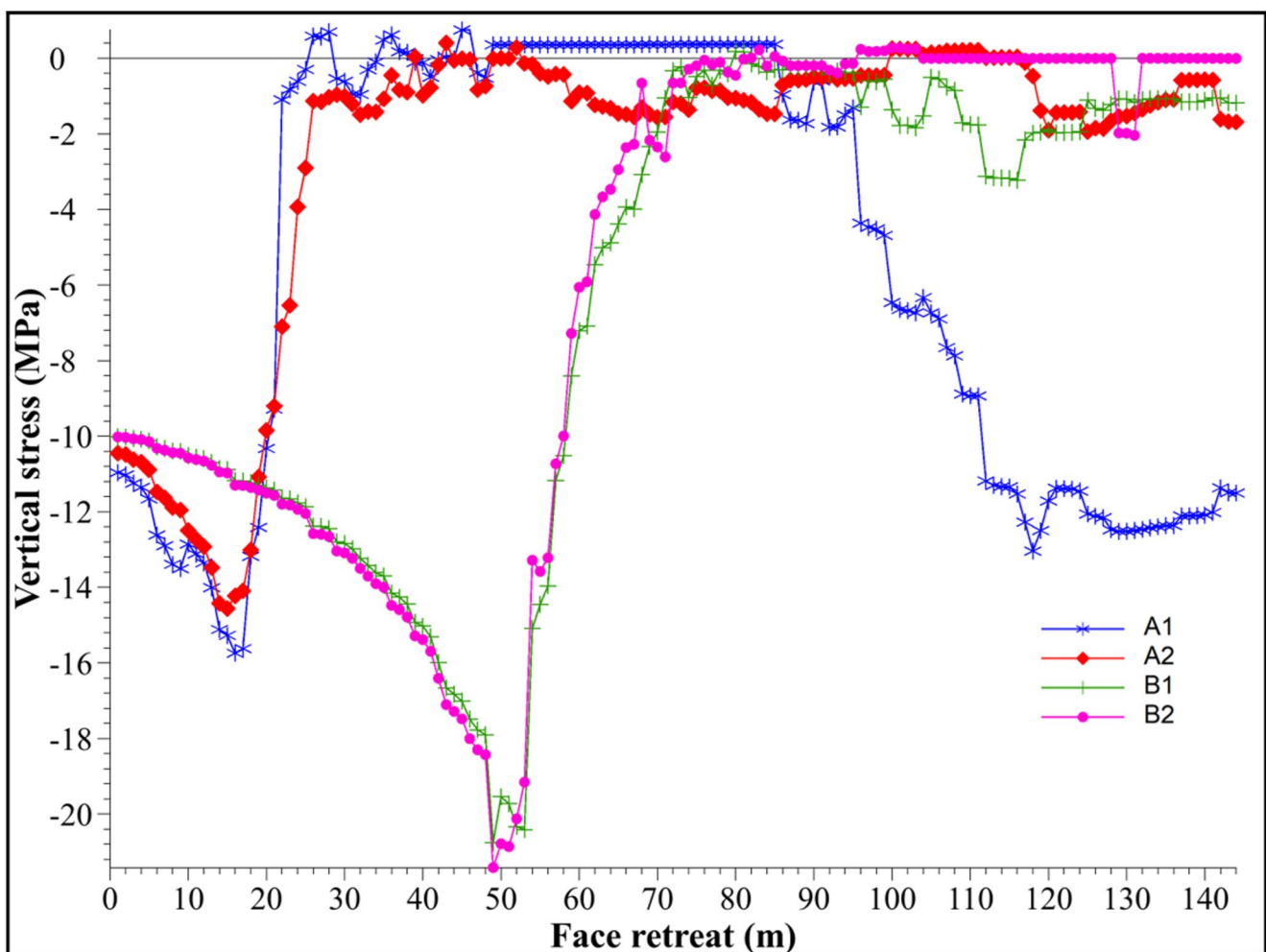


Fig. 10 Vertical stress at fixed locations during face retreat

Face deformation and failure

The face deformation was analysed through the displacement of four face blocks initially at the monitoring locations, as plotted in Fig. 11. In this figure the displacement was plotted versus numerical steps because it was recorded during cycling rather than after each face retreat. In terms of vertical displacement, Fig. 11a shows that the blocks initially at Locations A1 and A2 displaced downwards as the face retreated (negative value). Before the face passed these blocks, they displaced downwards a small value of less than 0.08 m. After the face completely passed the blocks (39 m of retreat), the blocks displaced downwards significantly. At the time of being mined (by web cut or recovery), the block initially at Location A1 (interface between bottom coal section and top coal section) displaced downwards 1 m, while the block at Location A2 (initially at mid-seam height) displaced downwards 4.5 m—being at seam floor. Similarly, before the face passed the blocks initially at Locations B1 and B2, these blocks displaced downwards less than 0.5 m. After the face completely passed the blocks (75 m of face retreat), they displaced downwards 0.9 and 3.8 m, respectively.

In terms of horizontal displacement, Fig. 11b shows that the blocks initially at Locations A1 and A2 displaced towards the mined-out area (positive value). Before the face passed these blocks, they reached a peak displacement of 0.45 and 0.6 m, respectively. After the face completely passed the blocks (39 m of retreat), their displacement remained unchanged because the blocks were cut or recovered. The horizontal displacement of the block at mid-seam height (A2) was greater than that at interface between bottom coal section and top coal section (A1). Similarly, the blocks at Locations B1 and B2 displaced towards the mined-out area, reaching a peak value of 1.1 and 1.4 m, respectively. After that, their displacement remained unchanged of approximately 1.1 m due to the coal cutting and recovery. The simulation results indicated that the longwall mining causes coal face to deform downwards seam floor and towards mined-out area. The deformation directions are consistent with the stress tensor directions in the simulation and agree with field observation.

The face failure was analysed through the failure of intact blocks and geological structures in unmined seam. The visual observation during simulation shows that in most face retreats, the failure extent of intact blocks was similar to that of geological structures. After each 20 m of retreat, the extent was 19, 20, 24, 24, 24 and 22 m from coal face. Except for the initial and final face retreats, the extent agrees well with the location of front abutment stress analysed earlier. In coal seam, most blocks and structures failed in shear, as explained in detail in Le and Oh (2022).

Face spall and top coal fall

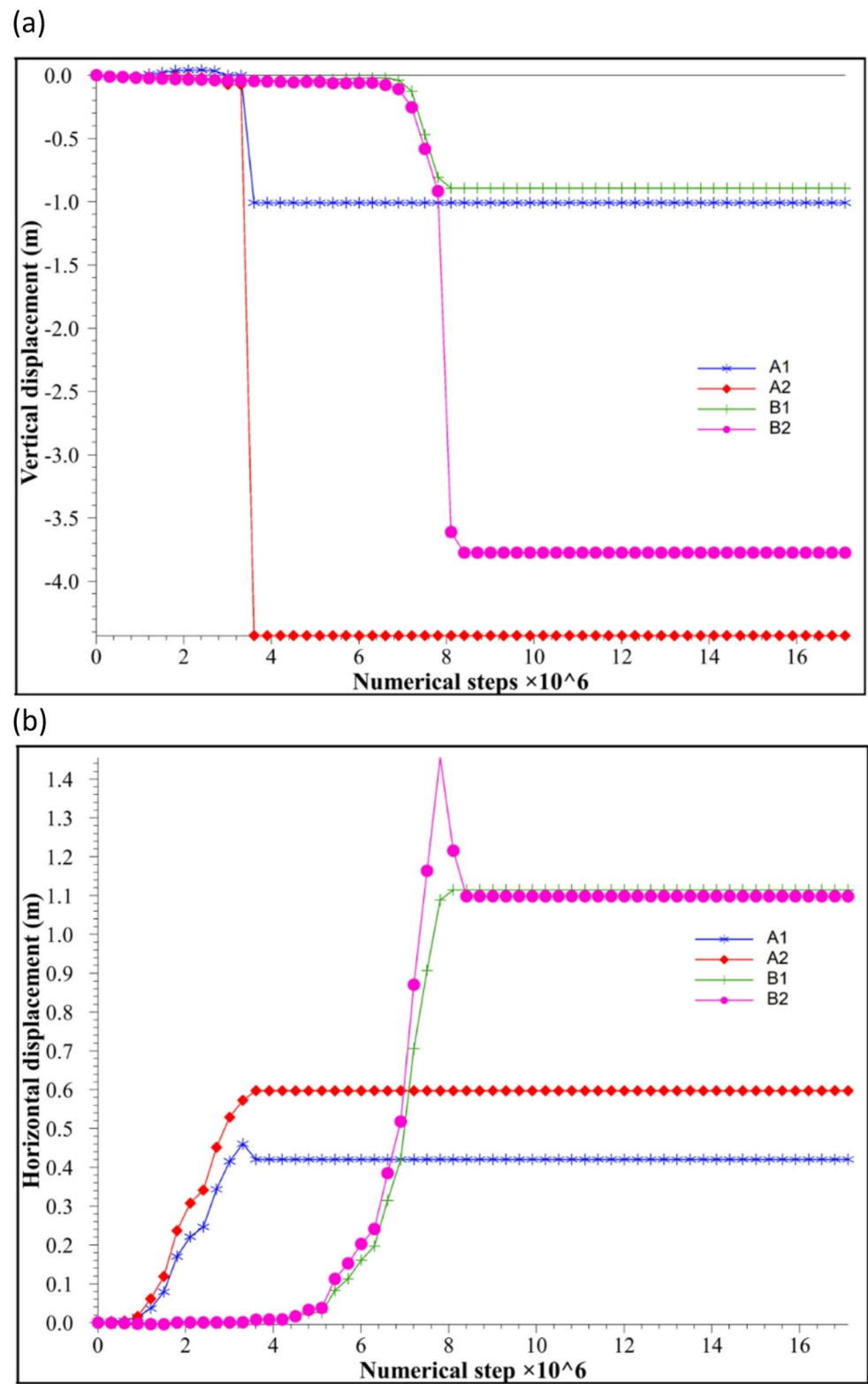
The number of coal blocks spalled from face and fallen from top coal ahead of shield support in each face retreat is displayed in Fig. 12. The figure shows that in a total of 144 face cuts, coal face remained stable in 64 cuts without any face spall and/or top coal fall, equivalent to 44% of the total cuts. Meanwhile, the face showed major instability in 33 cuts where both face spall and top coal fall occurred, equivalent to 23% of the total cuts. Although the total face instability events in the simulation were greater than that reported from practice (see previous section), the frequencies of stable and major unstable face events appeared to be reasonable according to on-site engineer's experience.

Amongst the major face instability events, severe events (defined as having greater than three spalled and fallen blocks) occurred in relation to roof strata stability. The first severe event occurred when the immediate roof was about to cave after 22 m of face retreat. The next severe events occurred when the main roof layers hung up and ruptured, which were after 49, 54, 57, 68, 69, 84, 86, 92, 94, 96, 100, 107, 109, 112, 120, 125, and 143 m of face retreat. The relationship between severe face instability and roof strata stability is explained as follows. Before caving or rupture, the roof layers hung up and behaved as cantilever beams. The beams caused increasing load on their fulcrum, which were at coal face and its proximity, and resulted in near-face coal mass failure. Once exposed, the failed coal mass could easily spall and fall along joints/fractures from the seam into mining voids under effect of gravity. Because the roof strata contained pre-existing discrete fractures, the hanging/rupture span of the roof layers was not consistent. The severity levels were accordingly different between the face instability events. Figure 12 shows that as the mining progressed further, the event became more severe because a greater number of roof layers hung up and ruptured. The figure also shows a greater number of top coal fall blocks compared to that of face spall blocks in each web cut. Since the geometry of geological fractures could affect this simulation result, a deeper interpretation for this comparison is recommended for future study.

Coal face behaviour impacted by main roof characteristics

Although the behaviours of coal face (including its overlying top coal) at Panel I-7-5 Vang Danh coal mine were analysed in detail in previous section, the behaviours may vary from place to place due to the inherent variability of their controlling geological parameters in practice. Previous publications mostly focused on longwall coal face behaviour in the

Fig. 11 (a) Vertical and (b) horizontal displacement of monitoring blocks



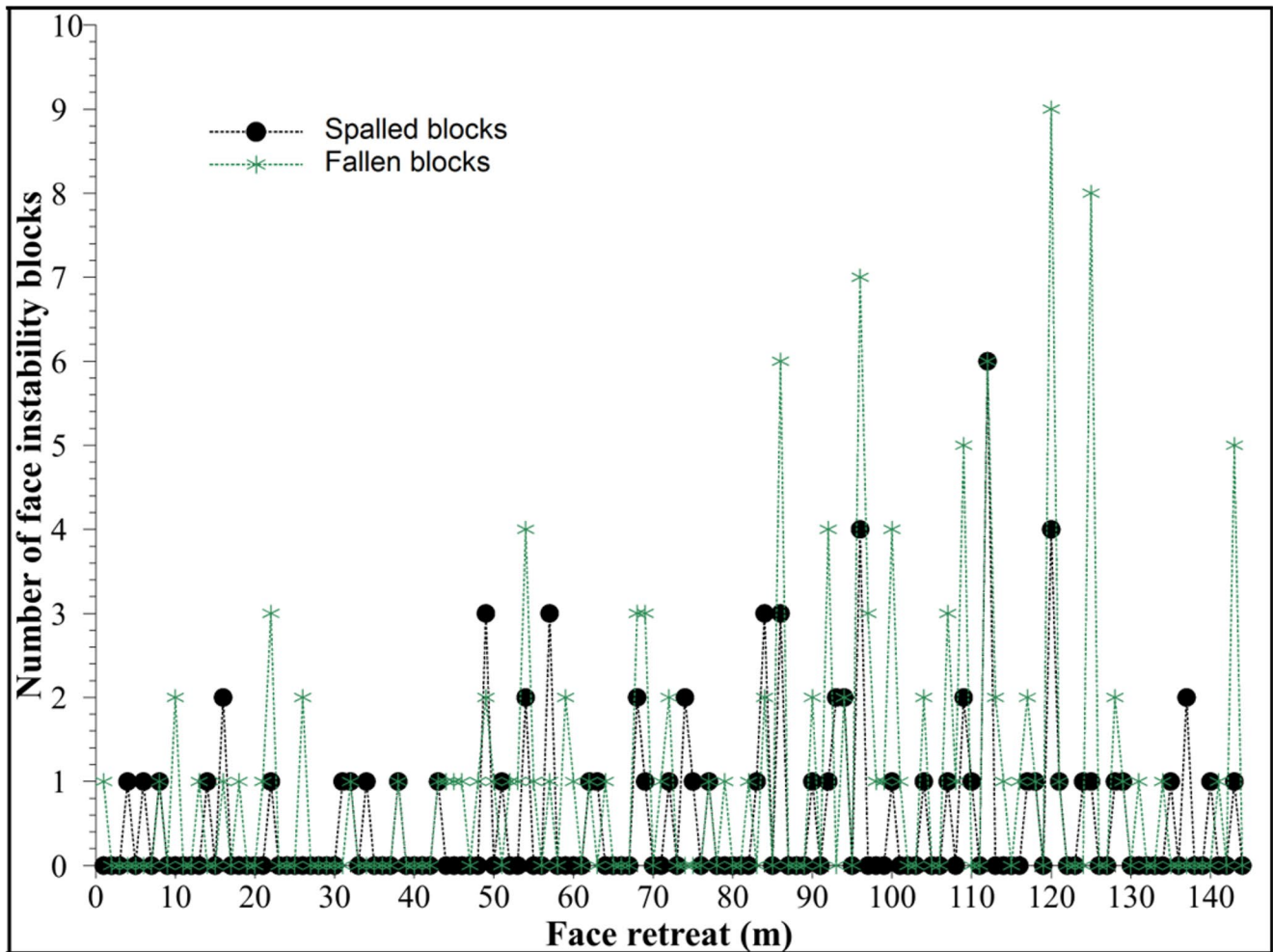


Fig. 12 Number of face instability blocks in each face retreat

variation of intact coal properties (Le et al. 2022) and in-seam structural characteristics (Le and Oh 2022). Furthermore, these studies were performed in favourable roof strata conditions and in limited extraction length. In such cases, coal face behaviour was predominantly controlled by the seam and immediate roof characteristics, rather than main roof characteristics. For the longwalls in where main roof strata are strong and near seam, sufficient movement of the strata should clearly control the face behaviour. In this paper, given a sufficient mining length to induce steady-state roof strata movement, the coal face behaviour under variation of main roof characteristics was analysed. Only the characteristics that directly impact coal face and can be collected in practice were selected for the analysis, as presented below.

Roof rock strength and elastic modulus

The coal face behaviour impacted by main roof rock strength was studied by running four additional simulations which had different main roof strengths while the

other rock material properties and simulation settings were the same with those in the Panel I-7-5 simulation. Note that this sensitivity analysis method was used for all the main roof characteristics in this study. The rock strength was from 17.41 to 52.23 MPa, which was derived from the representative laboratory-scale UCS range of rock at site. The coal face behaviour was monitored through the number of face instability blocks, face instability rate and top coal recovery rate (see previous section). The results in Fig. 13 (including those from the Panel I-7-5 simulation) show that when the rock strength increased, the face instability rate followed a sharp decreasing trend, from 15.06 to 10.35%. The associated number of face instability blocks significantly decreased from 214 to 147 blocks, equivalent to 31.3% reduction. Behind shield support, the top coal recovery rate also followed a sharp decreasing trend, from 87.56 to 71.23%. The simulation results indicated that as the main roof rock strength increased, the coal face and top coal became more stable. The better coal face and top coal stability can be explained as follows. As the rock strength

Fig. 13 Coal face behaviour impacted by main roof rock strength

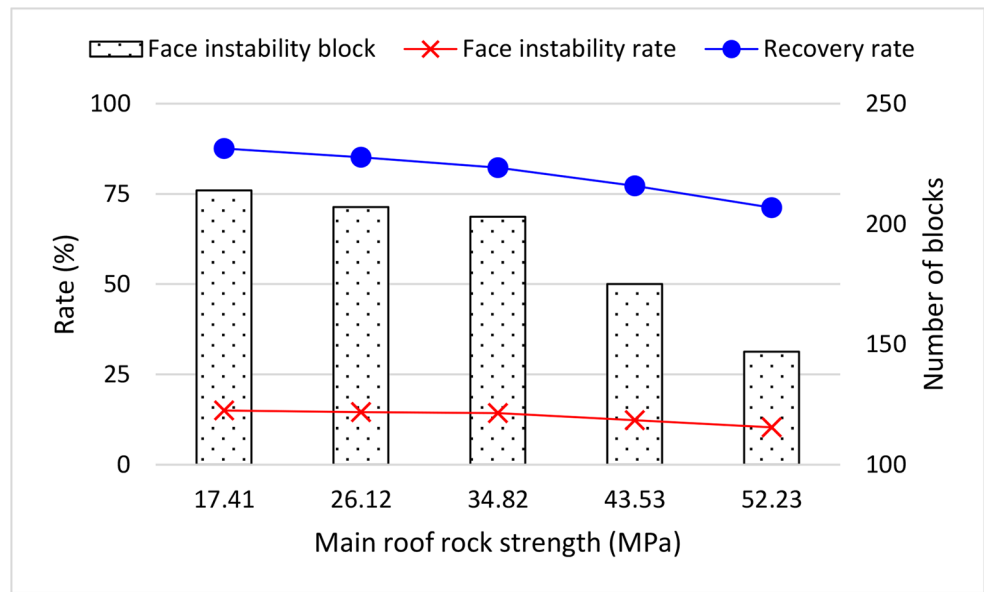
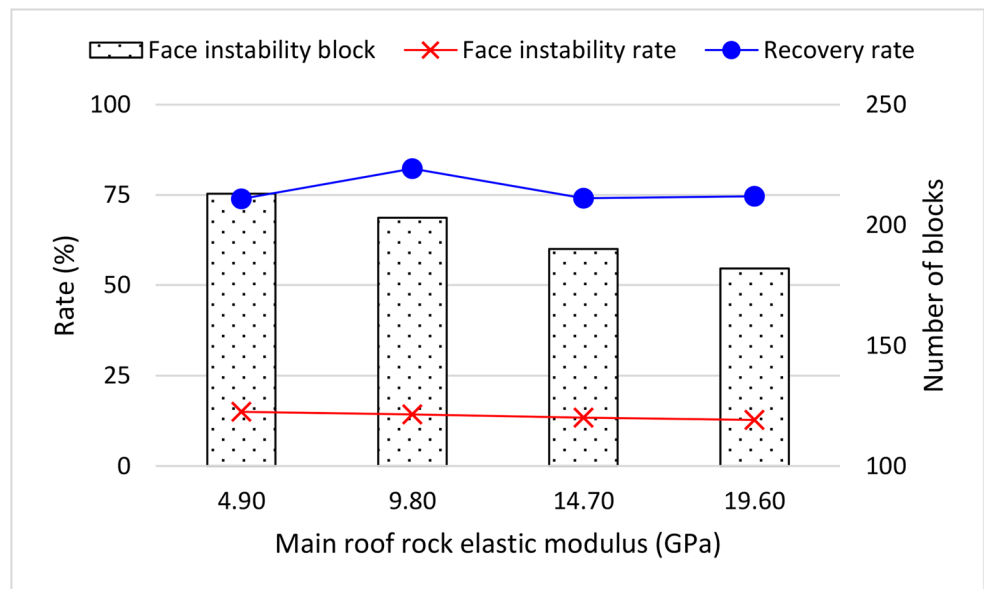


Fig. 14 Coal face behaviour impacted by main roof rock elastic modulus



increased, the overall strength of main roof strata increased, and they thus remained intact in longer interval before failed. While behaving as voussoir beams (before initial/main rupture) or cantilever beams (before each cyclic rupture), the longer intact strata transferred the load from roof rocks and their own weight to further ahead of coal face. The load on coal face and top coal was reduced, accordingly increasing the face and top coal stability. Note that in the case of strong massive strata, the rupture span of voussoir/cantilever beams can be significantly greater than that in the current study. The greater rupture span can better reduce the load on coal face at current retreats, but they exacerbate the stress concentration ahead of coal face and lead to worse face behaviour in latter retreats.

The coal face behaviour under different main roof rock elastic moduli was analysed by running three additional simulations. The rock elastic modulus was increased from 4.90 to 19.60 GPa, covering the representative range of the modulus in the field. The results in Fig. 14 show that when the rock elastic modulus increased, the face instability rate followed a slight decreasing trend, from 14.99 to 12.81%. The decreasing trend became more prominent for the number of face instability blocks, which decreased from 213 to 146 blocks and was equivalent to 31.5% reduction. Behind shield support, the top coal recovery rate did not show a clear change trend but fluctuated in the range of 73.87–82.29%. The better coal face stability can be attributed to the strata stiffness as follows. Under the same stress

regime and geometrical structure of main roof strata, as the elastic modulus of rocks increased, the overall stiffness of main roof strata also increased. The strata deformation and displacement accordingly decreased that better transferred the load from roof rocks and their weight to further away from coal face. Because the strata themselves were strong, the load reduction effect was maintained in a sufficient time period before strata failure and rupture, finally resulting in better face stability. It should be noted that as being restricted within elastic range, the load reduction effect caused by change in rock elastic modulus was less than that caused by change in rock strength. This was evidenced by the reduction rate of face instability blocks of 14.6% compared to that of 31.3% caused by rock strength. Behind shield support, the effect was slight that could not clearly impact top coal, causing a fluctuation in recovery rate in the figure.

Roof strata and bedding thicknesses

Two additional simulations with main roof strata thickness of 18 and 48 m were run to understand the resultant coal face behaviour. The lower limit (18 m) was considered to be sufficiently minimum that could still significantly impact coal seam behaviour (Trueman et al. 2010). Meanwhile, the upper limit (48 m) was the maximum sandstone thickness reported from the field (Vang Danh Coal Company 2017). The results in Fig. 15 show that when the main roof strata thickness increased, the face instability rate followed a slight increasing trend from 12.32 to 14.85%. The increasing trend was clearly seen for the number of face instability blocks, which increased from 175 to 211 blocks, equivalent to 20.5% increase. The relationship between the roof strata thickness and coal face behaviour is consistent with theoretical analysis and company report in Frith (2015). At the same time, the top coal recovery rate was significantly increased from 76.76 to 86.93%. The worse coal face and top coal stability was due to the fact that

as the main roof strata thickness increased, the load from the strata acting on coal seam subsequently increased. This caused the coal face and top coal to fail, spall and cave easier. It should be emphasised that the impact of main roof thickness on coal face behaviour is valid within the studied range. The thickness beyond which the impact of strata on coal face reaches limitation was not investigated in the current study.

The coal face behaviour impacted by different bedding thicknesses within main roof strata was investigated by running three additional simulations. The bedding thickness was in the range of 1–7 m, as commonly observed in Quang Ninh sedimentary rock strata (Pham 2012). The results in Fig. 16 show that when the bedding thickness increased, the face instability rate slightly increased from 11.33 to 13.73%. The face instability blocks increased from 161 to 195 blocks, equivalent to 21.1% increase. Behind shield support, the top coal recovery rate also increased from 73.12 to 83.17%. The worse face and top coal stability was mainly due to the limitation in generating discrete fractures in the simulation. In particular, while the total thickness of main roof strata was unchanged, thicker bedding thickness meant greater spacing between planes and accordingly smaller number of the planes. Because the fracture density was constant, the number and length of generated fractures was therefore increased. More fractures not only made main roof weaker (Le and Oh 2022) but also created more potential paths for roof caving or rupture. This reduced the span of roof caving/rupture that could not transfer the roof rock load far away from coal face. The thicker is the bedding thickness, the shorter is the roof caving/rupture span and the worse is the face and top coal stability. Note that the current result is only valid when the fracture density is constant, and the fractures intersect with other geological structures. The impact of bedding thickness on coal face behaviour should be further analysed in separate study.

Fracture density and location of main roof

Three additional simulations with the fracture density in main roof strata ranging from 0.1 to 0.4 m/m² were run to analyse the coal face behaviour. Along with the density of 0.2 m/m² used for the Panel I-7-5 simulation, the other density values were selected based on field experience (see previous section) and to avoid excessive simulation run time (caused by too many generated blocks). The different settings of geological fractures in main roof strata are illustrated in Fig. 17.

The simulation results presented in Fig. 18 show that when the fracture density in main roof strata increased, the face instability rate followed a sharp increasing trend, from

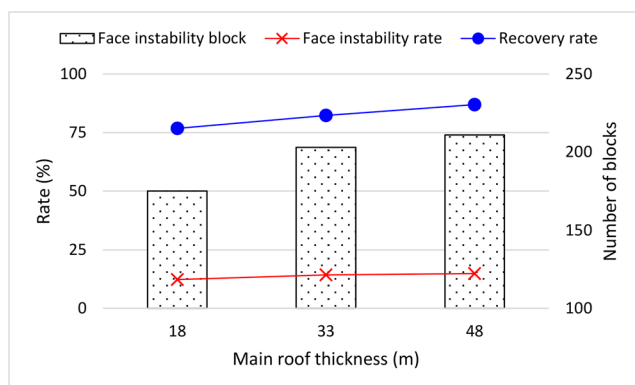
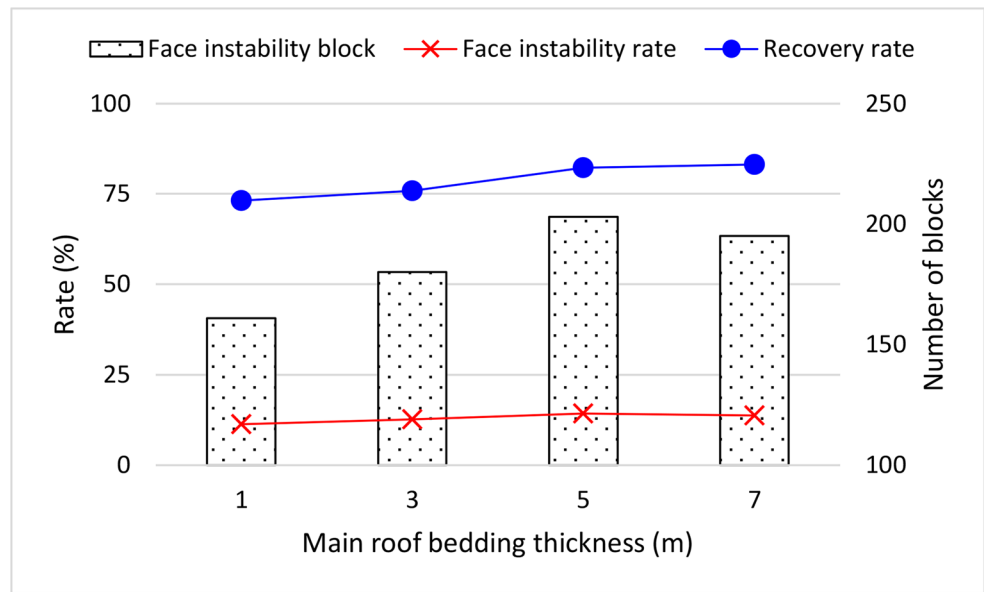


Fig. 15 Coal face behaviour impacted by main roof strata thickness

Fig. 16 Coal face behaviour impacted by main roof bedding thickness



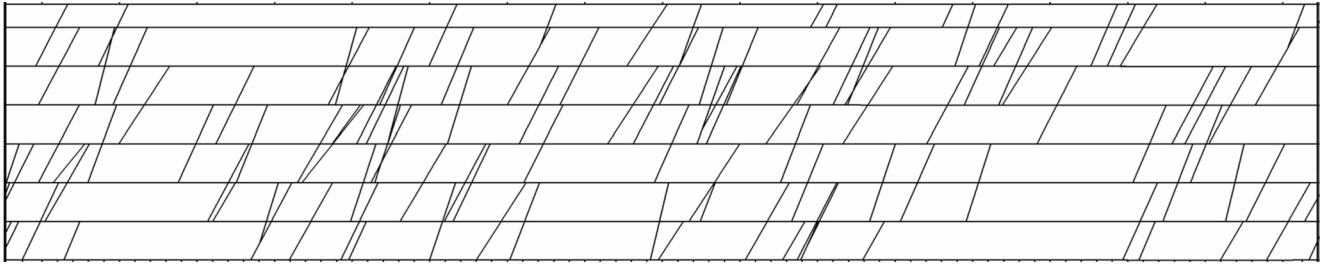
11.61 to 15.98%. The number of face instability blocks significantly increased from 165 to 227 blocks, equivalent to 37.6% increase. This worse coal face stability was associated with the greater number of fractures within roof strata. In particular, as the fracture density increased while the bedding spacing was constant, the number of fractures in main roof strata increased. As explained earlier, the shorter span of roof caving/rupture could not transfer the roof rock load far away from coal face, causing coal seam to spall and fall easier. The shorter roof caving/rupture span clearly increased the recovery rate behind shield support from 77.01 to 85.43% in the first three simulations before slightly decreased it to 82.91% in the last simulation.

The coal face behaviour was finally analysed in the variation of main roof location. The distance between main roof and coal seam was increased from 13.5 to 43.5 m, falling within the range limit (maximum 70 m) causing significant impact on coal seam (Trueman et al. 2010). The results in Fig. 19 show that when the main roof was further above coal seam, the face instability rate decreased from 14.29 to 11.19%. The number of face instability blocks decreased from 203 to 159 blocks, equivalent to 21.7% reduction. At the same time, the top coal recovery rate significantly decreased from 82.29 to 63.69%. This relationship is consistent with practice reported in past studies (Frith 2015; Galvin 2016). It can be explained that the greater was the distance between main roof and coal seam, the thicker was the immediate roof. After caving, the thicker immediate roof better filled the mined-out area that reduced the impact from main roof strata movement on coal seam. The coal face and top coal therefore became more stable.

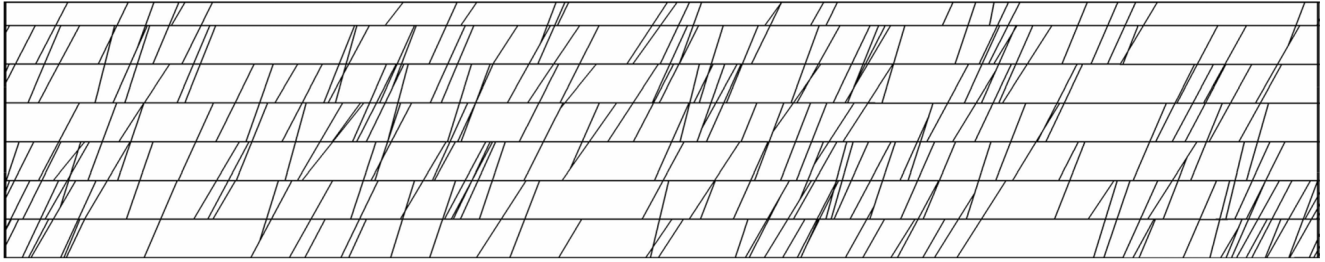
Discussion and conclusions

This paper presents a detailed study of longwall coal face behaviour under near-seam and thick strong strata embedded with pre-existing discrete fractures. Although the presence of discrete fractures in coal seam and roof strata is not uncommon in mining practices worldwide (Cao et al. 2020; Commonwealth of Australia 2014; Vang Danh Coal Company 2017; Wang et al. 2017; Wei et al. 2022), the coal face behaviour in such a condition remains little understood. The reason for this, as discussed earlier, could be attributed to the number and variation of the fractures in practice. Compared to previous studies, the current study realistically incorporates the pre-existing discrete fractures in not only coal seam but also main roof strata within a sufficient mining length (face retreat length). First, the presence of discrete fractures in coal seam enables coal blocks to completely spall from face, fall from top coal above face, and cave from top coal behind shield support. The complete detachment of coal blocks at face was not presented in past DEM longwall simulations due to the use of conventional joint set generator (Le et al. 2018; Shabanimashcool et al. 2014). Second, the presence of discrete fractures in main roof strata within a sufficient mining length leads to the cyclic steady-state movement of the main roof. The cyclic movement of roof strata results in “pear” rather than “elliptical” shape of stress release zone as stated in previous studies (Gao et al. 2014; Le and Oh 2022). The impact from roof strata movement on coal face is now fully observed compared to that in similar studies (Le and Oh 2022; Yao et al. 2017). The face behaviour in response to variation of key roof characteristics (e.g.,

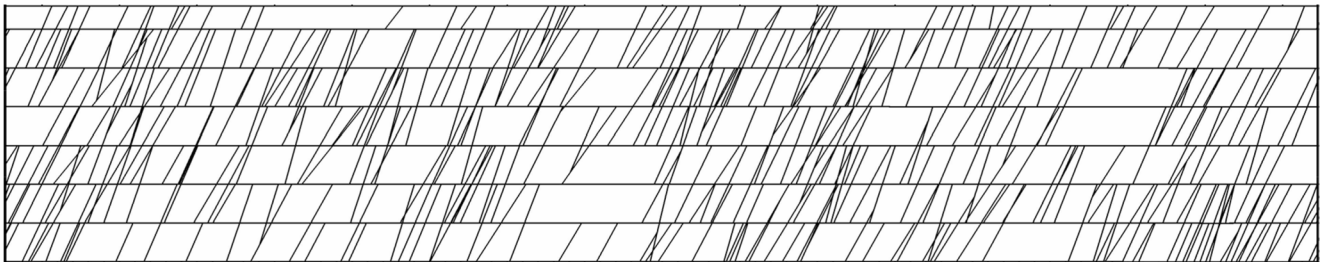
(a)



(b)



(c)



(d)

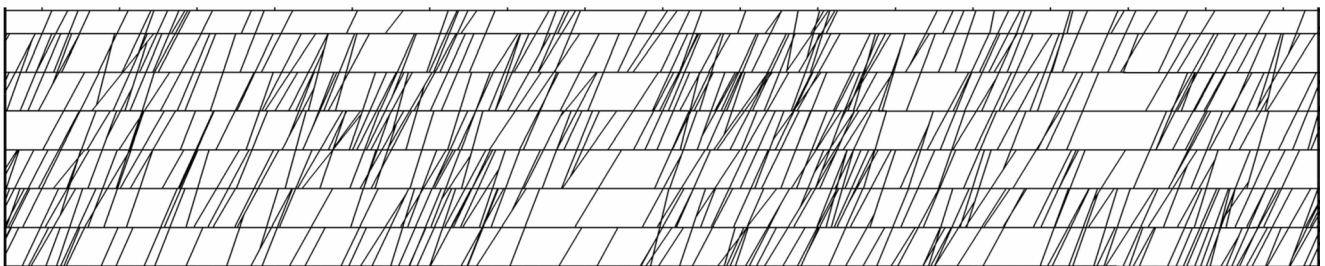


Fig. 17 Fracture density in main roof strata: (a) 0.1, (b) 0.2, (c) 0.3, and (d) 0.4 m/m²

fracture density), is studied for the first time in longwall stability research. The above advantages together make the coal face behaviour to be more reliably and sufficiently analysed in the current study. It should be highlighted that due to the limitation in observation area and lack of test apparatus, several assumptions were required for simulation inputs. The impact of the assumptions on the study's findings is minimised by strictly following simulation procedure and

reliably validating the simulation against on-site coal seam, rock strata, and shield support behaviours. More field stress data (e.g., real-time shield force and borehole stress) are required for future research for more rigorous validation of longwall simulation.

This paper presents a deeper understanding of mining-induced coal face behaviour under near-seam and thick strong strata embedded with pre-existing discrete fractures.

Fig. 18 Coal face behaviour impacted by main roof fracture density

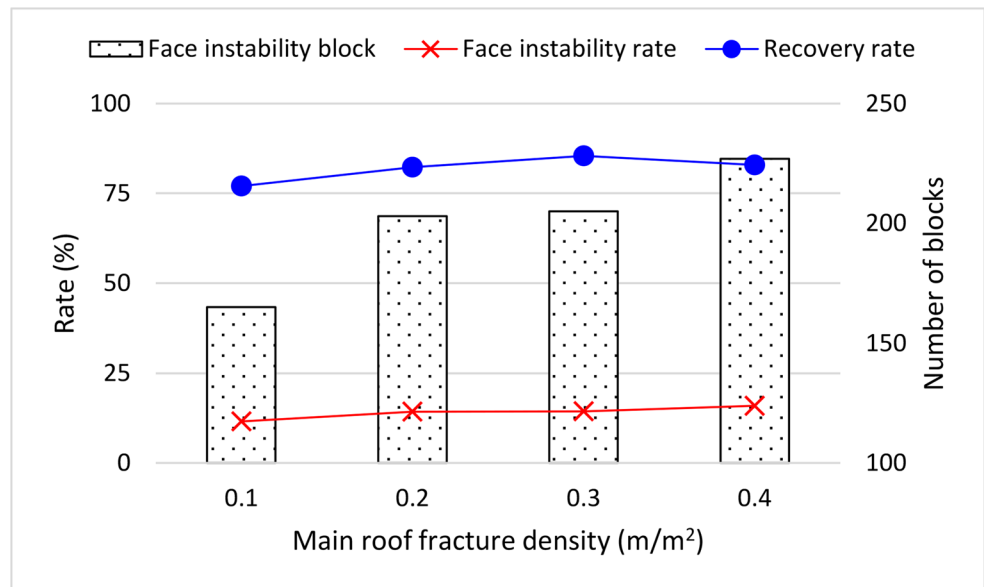
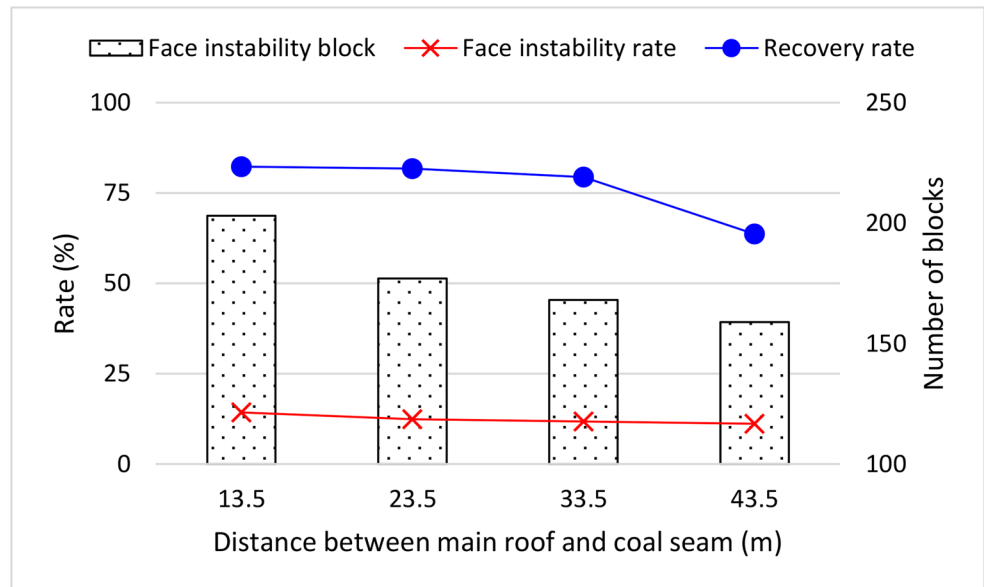


Fig. 19 Coal face behaviour impacted by distance between main roof and coal seam



The study first confirms the formation of stress release and stress concentration zones induced by longwall mining. The study then better characterises the shape of stress release zone, which should be “pear” rather than “elliptical” shape. Outside the stress release zone, the stress change in direction and magnitude is extensively demonstrated from coal seam to surrounding rocks. The front abutment stress is found up to 2.88 times the pre-mining stress and located up to 24 m ahead of coal face. Inside the stress release zone, the study confirms that the steady-state movement of roof strata causes the compressive stress to re-develop in the reconsolidated rocks in mined-out area. This stress can significantly increase to its pre-mining value and be greater in places. The coal face behaviour is demonstrated to closely relate

to roof strata stability. When the main roof characteristics vary, the study finds that the coal face is more stable when the strength and elastic modulus of main roof rock increase. Similar face behaviour is manifested when the distance between coal seam and main roof increases. In contrast, the face is found to be less stable when the thickness and fracture density of main roof increase. In addition to coal face, the top coal behind shield support shows clear relationships with main roof rock strength, main roof strata thickness, and distance between coal seam and main roof. The study’s findings help engineers in better understanding longwall coal face behaviour in less favourable geo-mining conditions from which safe and environmentally sustainable coal mining can be better maintained.

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Data availability Some or all data, models, or code that support the findings of this study are available from the corresponding author upon reasonable request.

Declarations

Competing interests The authors declare no competing interests.

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