NUMERICAL STUDY ON STABILITY OF RETAINED ROADWAY IN NON-PILLAR MINING AT NAM MAU COAL MINE

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ABSTRACT

By using numerical modelling method, the study aims to better understand the geomechanical behaviours of roof strata and stability of retained roadway in the non-pillar mining method at Nam Mau coal mine. The analysis of numerical model shows that when the face is far away from the retained roadway, the mining does not cause significant impact on roadway stability. When the face approaches the roadway, the vertical stress gradually increases around the roadway. The stress reaches a peak value when the face is from 10 to 20 m away and then rapidly decreases when the roadway complete falls within the stress relief zone of longwall. In combination with the insignificant concentration of horizontal stress, the surrounding stress condition induces the roadway to deform and fail when the face approaches roadway. The study finds that the roadway is significantly affected when the face is adjacent to it. The rock mass around roadway fails up to 25 m towards the unmined area. At the end of the upper panel mining, only immediate roof caves that form a caving line of less than 70 degrees to seam floor. The strong main roof does not cave but sag downwards, causing significant load on roadway. The results in this paper can serve as fundamental knowledge for efficient design of roof cutting techniques, contributing to reducing the cost of using non-pillar mining method into Vietnam coal industry.

Keywords: *gently inclined seam, non-pillar mining, retained roadway, numerical modelling, rock mass pressure*

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1. INTRODUCTION

The non-pillar mining technology was trialled in 2009 in China, based on the advanced understanding of roof strata behaviour and advances in face equipment [1]. The principle of the technology is the retention of the transportation roadway of an upper longwall panel for re-use as the ventilation roadway of the lower longwall panel [2]. The retention is performed by actively reducing the rock pressure on roadway and at the same time reinforcing the bearing capacity of roadway itself. The pressure reduction is implemented by cutting the console roof adjacent to roadway before longwall roof caving. The reinforcement is done by use of negative Poisson’s ratio (NPR) high pre-stressed constant-resistance large deformation bolt/anchor (CRLD). The layout of non-pillar mining is illustrated in Figure 1. Since the roof pressure is reduced while the bearing capacity is increased, the roadway remains stable and is ready for use for the lower longwall panel. The development cost for the lower longwall is accordingly reduced. The resources loss is also reduced that ultimately improves the production economic efficiency.

In Vietnam, most underground mines use the longwall mining method with coal pillars and therefore limited studies have focused on the reduction of resource loss within coal pillar. Amongst those studies, three main directions have been researched: (i) to drive a new ventilation roadway for lower longwall panel, (ii) to recover pillar at the same time with extraction of lower longwall panel, and (iii) to use artificial pillar (i.e., backfill, chemical). To present, only one national-level trial production project for use of non-pillar mining method in Vietnam was performed by Vietnam Mining Science and Technology Association [4]. The project results show that the non-pillar mining was successfully trialled at Longwall I-10-9, Khe Cham I, Ha Long coal mine. Compared to conventional method, the coal resource was extracted with an additional amount of 37 thousand tonnes. The coal loss ratio was reduced from 18.5% down to 5%. However, the trial project also reveals that the efficiency of the mining method is greatly dependent on the geological conditions of roof strata. The project reports that when the roof strata are competent, the drilling for roof cutting is extremely difficult, resulting in great cost for roof cutting.

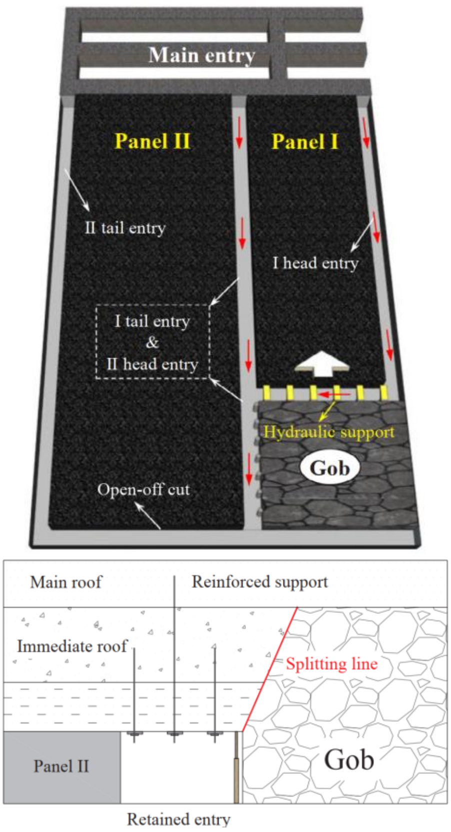

Figure 1. Layout of non-pillar mining technology [3]

In consideration of the (i) great coal reserves under strong roof in Quang Ninh coal mines [5], (ii) advantages of non-pillar mining method, and (iii) results from non-pillar mining trial project, this paper presents a study on the stability of retained roadway in non-pillar mining method under strong roof in Quang Ninh coal mines by using numerical modelling method and taking the geo-mining conditions from Nam Mau coal mine as a representative case. The study results should serve as fundamentals for design of roof cutting techniques, contributing to reducing the cost of using this advanced mining method in Vietnam coal industry.

2. MATERIALS AND RESEARCH METHOD

2.1 Nam Mau coal mine

Nam Mau coal mine is operated by Nam Mau Coal Company–a subsidiary of Vietnam National Coal and Mineral Industries Holding Corporation Limited (Vinacomin). The mine is located in Thuong Yen Cong ward (now Yen Tu ward), Quang Ninh province, which is within the geographical coordinates X = 38,500–41,000, Y = 367,700–371,300. The mine borders Bao Dai mountains to the north, Mieu Bong village to the south, Vang Danh coal mine to the east, and Yen Tu pagoda to the west (Figure. 2). The terrain of the area is mountainous with an average elevation of 450 m and gradually decreases from north to south.

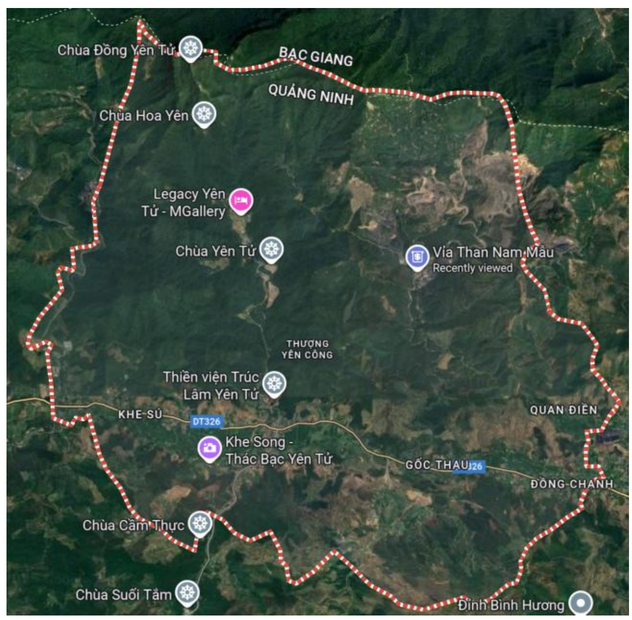

Figure 2. Location of Nam Mau coal mine within Thuong Yen Cong ward (now Yen Tu ward) [6]

The coal-bearing strata were formed in the Triassic–Jurassic Age, from Norian to Early Jurassic age (T_3n-J_1). The coal seams and rock strata plunge north with many minor folds changing seam dip angles (Figure 3). There are two typical faults, namely F.250 and F.400, which also change the dip angle of seams. At present, Nam Mau Coal Company extracts gently-to-steeply inclined coal seams (V9, V8, V7, V7T, V6a, V6, V5, V4) from level +125 down to level -50 by using longwall mining and caving method. Although most seam roofs have been assessed as easy to cave for ground control, practice shows that some panels have locally encountered hard-to-cave roof condition, as occurred at Seam V5. According to Pham [7], Seam 5 has an average thickness of 4,98 m. The siltstone immediate roof has a thickness ranging from 2 to 8 m and uniaxial compressive strength (UCS) of 32.4–74.5 MPa. The sandstone main roof has thickness of 5–35 m and UCS of 52.8–105.2 MPa. The dip angle of seam ranges from 10 to 80 degrees. A layout of longwall mining at Seam 5 in July 2025 is presented in Figure 4.

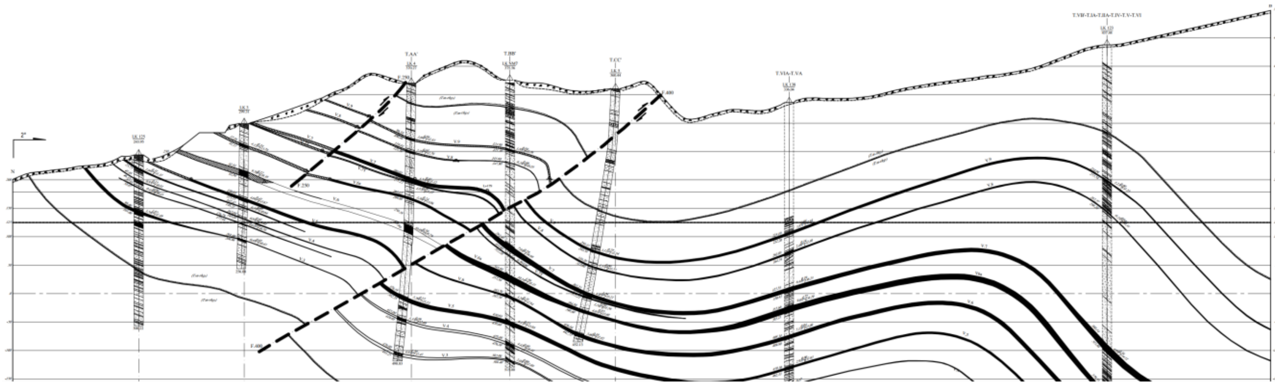

Figure 3. Cross-section along Line III, Nam Mau coal mine [8]

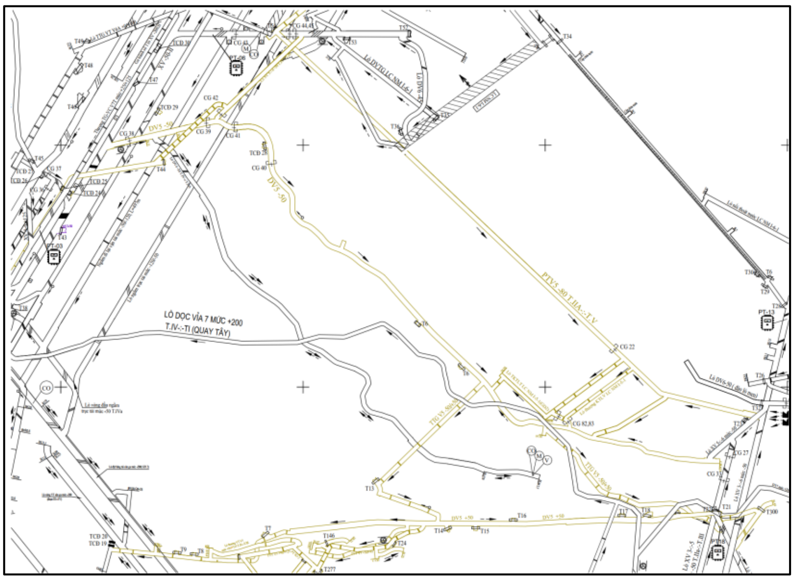

Figure 4. Layout of Panel I-5-1, Seam 5, Nam Mau coal mine [9]

2.2. Numerical modelling method

Numerical modelling is a research method particularly suitable for analysis of rock mass through the computation of mathematical equations and algorithms in computer. The use of numerical modelling in rock mechanics is important to understand the rock behaviours from micro scale to field scale within reasonable cost and time period. However, for proper use of the method, fundamental understanding of problem nature, simplification of geological characteristics, and judgement of users are needed.

For studying the rock mass surrounding the retained roadway in non-pillar mining method, a numerical method should be able to model the sedimentary strata at panel scale, the deformation and failure of retained roadway, and the caving or spalling of rock particles from host strata. For these reasons, the two-dimensional discontinuum-based code UDEC™ [10] is selected as the main analysis tool. UDEC™ models a problem as an assemblage of discrete blocks connected via contacts. The motion of blocks is governed by stress-strain law, while that of contacts is controlled by force-displacement relation. The calculation cycle in UDEC™ is illustrated in Figure 5. Detailed procedure for modelling a static problem is outlined in Figure 6.

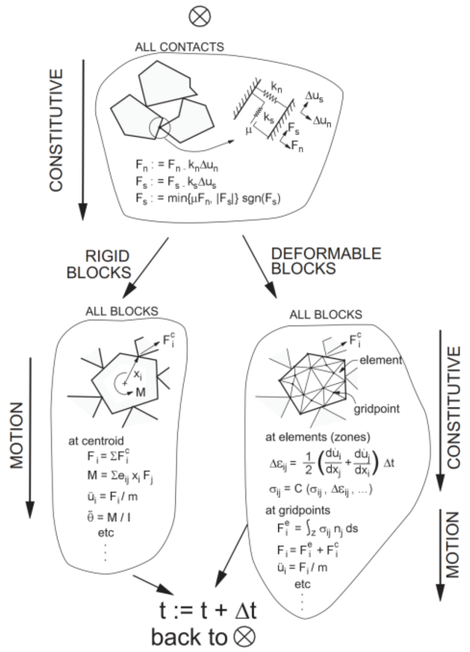

Figure 5. Calculation cycle in UDECTM [10]

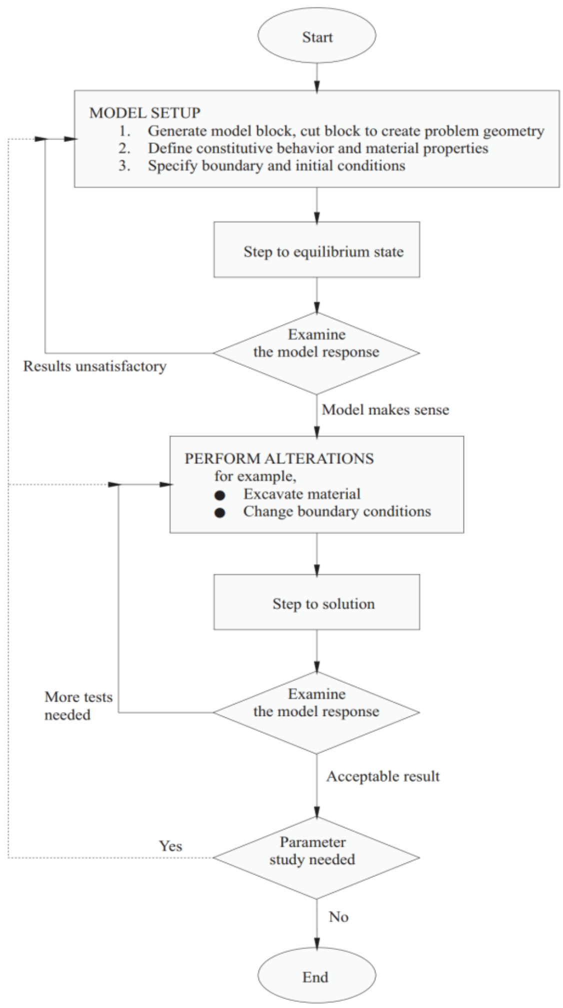

Figure 6. Procedure for modelling a static problem in UDECTM [10]

2.3. Development of non-pillar mining model

2.3.1. Model geometry and in-situ stress

The geometry of model is developed from the following considerations. Since UDEC^{TM} is two-dimensional as mentioned above, the cross section required for modelling is taken along the panel dip. Within the scope of Panel I-5-1, Seam 5 and rock strata have an average dip angle of 15 degrees. The panel width in dip direction is about 100 m. The retained roadway is modelled at location between the two panels, and it has a width of 3.7 m. The thickness of coal seam, immediate roof and main roof is 5, 3.4 and 21.2 m, respectively. Because the coal seam is inclined, the thickest overburden of 192 m is modelled explicitly. The remaining overburden of 238 m is modelled by a vertical stress applied on model top boundary (6,07 MPa). The in-situ stress at model bottom boundary ($y = 0$) is generated by the weight from 470 m strata thickness, with orthogonal principal stresses being equal. Due to sedimentary formation, all geological structures within panel are simplified to bedding planes and joints being perpendicular to the planes. In floor strata, the maximum bedding and joint spacing is 5 m and 10 m, respectively. In coal seam, the spacing is 1 m for both structures. In immediate roof, the spacing is 1.7 m for bedding planes and 3 m for joints. The spacing is 4 m for both structures in main roof. The explicit overburden is assigned with same structures as in the floor strata. The complete model geometry is presented in Figure 7.

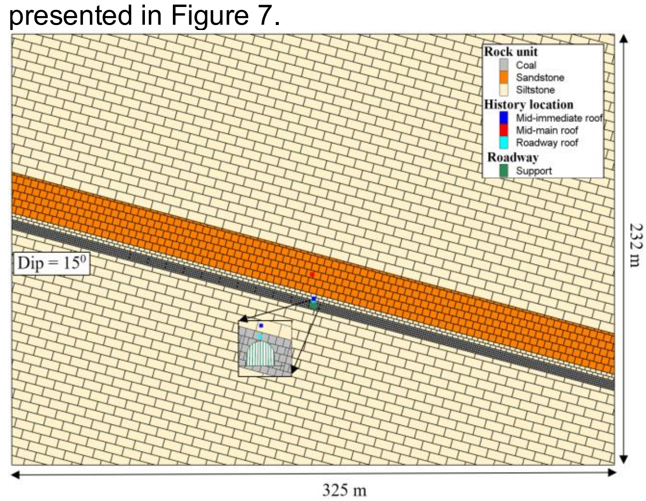

Figure 7. Model of non-pillar mining in panel dip direction

2.3.2. Material, roadway and extraction

Based on experience of modelling longwall, the blocks in the current model are assigned with strain-softening constitutive law. The contacts are assigned with Coulomb-slip law. The physical and mechanical properties of rock are originated from geological report and past studies [11, 12]. For the contacts, due to limitation in laboratory test, the normal and shear stiffnesses are obtained after a trial-and-error process. The cohesion and tensile strength are assumed to be zero as being very small in practice [13]. The properties are summarised in Table 1.

Table 1. Physical and mechanical properties of material

Material	Block						Contact		
	Density (kg/m^3)	Uniaxial Compressive Strength (MPa)	Young modulus (GPa)	Poisson ratio	Internal friction angle (degree)	Tensile strength (MPa)	Normal stiffness (GPa/m)	Shear stiffness (GPa/m)	Friction angle (degree)
Coal	1500	25	3.75	0.25	25	2.5	10	1	15
Siltstone	2600	45.6	9.12	0.25	32	4.6	10	1	20
Sandstone	2600	94.6	18.9	0.25	35	9.5	10	1	20

After setup, the model is run to reach an equilibrium. Prior to mining, the roadway is driven by removing rock particles within a horse-shoe shape. The roadway support is then installed by using support element structure available in UDEC^{TM}. The properties for modelling support behaviour are shown in Table 2. The model is run to reach the second equilibrium state after roadway excavation. For the extraction in dip direction, the longwall face is divided into 10 equal sections. The direction of mining is from high to low level.

Table 2. Displacement-force relation of roadway support

Value	Initial	Setting	Initial yield	Final yield	Collapse
Displacement (m)	0	0.005	0.01	0.2	0.8
Force (MN)	0	0.428	0.9	0.9	100

3. RESULTS AND DISCUSSION

3.1. Stress distribution around retained roadway

The stress distribution corresponding to extraction sections is analysed through the vertical stress (Figure 7) and horizontal stress (Figure 8). Figure 7(a) shows that prior to any underground excavation, the vertical stress is uniformly distributed through the area. The stress at model top and bottom boundaries are 6 and 12 MPa, as initially set up. After the excavation of roadway and prior to panel extraction, the stress inside roadway and vicinity is relieved, forming stress concentration zones at the two sides of roadway (Figure 7(b)). The maximum stress in the concentration zones reaches 1.4–1.5 MPa and at 5–6 m away from the roadway periphery. When the mining face is far away from roadway, the stress around roadway is not clearly affected. When the face moves closer to roadway, the stress concentration zone expands, approaches and affects the stress around roadway (Figure 7(c–e)). When the face is adjacent to roadway, the stress concentration zone, which was located at the left of roadway in preceding mining sections, moves to the right of the wall (Figure 7(f)). This means that there is a period that the retained roadway suffers great stress concentration, which will be analysed in detail later in this section.

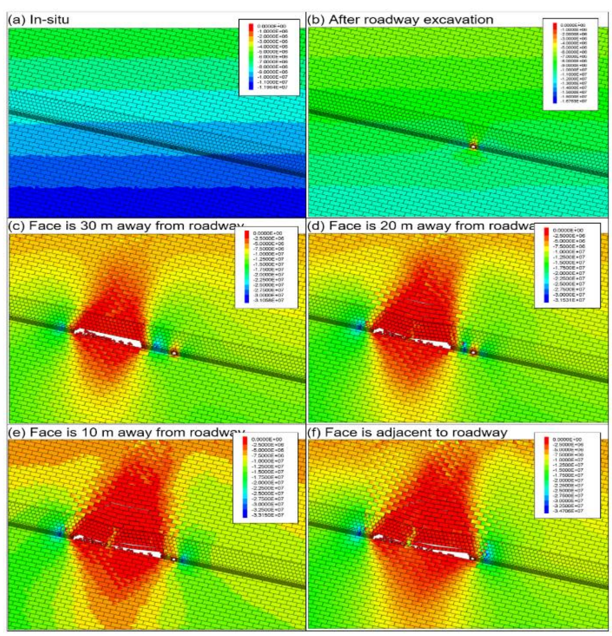

Figure 7. Vertical stress distribution according to progressive extraction (Unit: MPa)

Similar to Figure 7, Figure 8 shows the horizontal stress distribution around the retained roadway. Figure 8(a) shows that prior to any underground excavation, the horizontal stress magnitude at roadway level is about 8 MPa, which is equal to the vertical stress magnitude. After the excavation of roadway, the horizontal stress within the roadway is relieved, but no stress concentration zone is formed at roadway sidewalls (Figure 8(b)). When the mining face is far away from roadway, the stress around it is not clearly affected. When the face moves closer to roadway, the effect becomes more apparent (Figure 8(c–f)). However, the horizontal stress concentration is clearly seen in the hanging roof strata, rather than in roadway roof. The high concentration of horizontal stress within longwall roof is a strong indicator of voussoir beam formation, which contributes to the delay of roof caving.

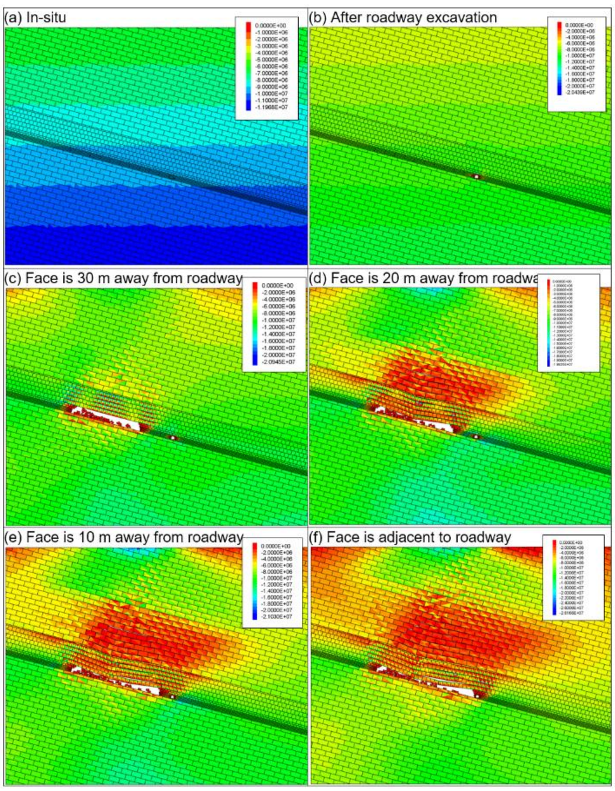

Figure 8. Horizontal stress distribution according to progressive extraction (Unit: MPa)

Detailed change of stress magnitude in roadway roof is monitored, as presented in Figure 9. For the vertical stress, Figure 9(a) clearly shows that as the mining face approaches the roadway (extraction span increases) the stress is increased in not only roadway roof but also immediate and main roofs. The vertical stress reaches maximum value in roadway roof and immediate roof when the face is 10 m away from roadway. It reaches maximum value in main roof when the face is 20 away. The maximum value in roadway roof, immediate roof, and main roof is 3.4 MPa, 6 MPa, and 12.2 MPa, respectively. After reaching the peak values, the stress at all monitoring points is rapidly decreased. This is because the points completely fall within the stress relief zone. For the horizontal stress, the stress change shows similar trend but less apparent compared to that of vertical stress (Figure 9(b)). The trend is clearly seen at the monitoring point in roadway roof, where the horizontal stress reaches its maximum value of 3.27 MPa when the face is 10 m away from roadway. In immediate and main roofs, the horizontal stress reaches its maximum value when the face is 20 away from roadway. The less apparent trend of horizontal stress change around roadway is because the roadway width is too narrow. The prominent concentration of horizontal stress is clearly seen in longwall roof where the panel width is much greater.

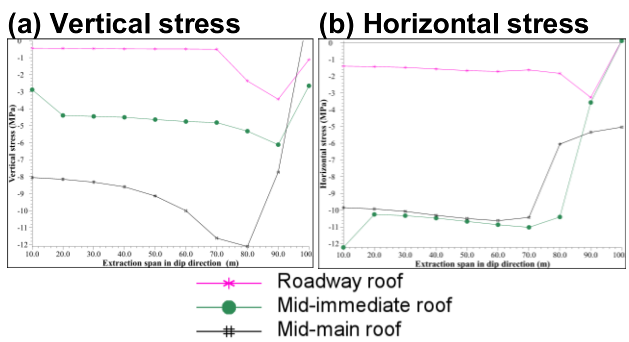

Figure 9. Stress change in roadway roof according to progressive extraction

3.2. Roadway deformation, failure and caving

The deformation of roadway is analysed through the displacement of monitoring points. In vertical direction, Figure 10(a) shows that when the mining face approaches roadway, the roadway roof increasingly displaces downwards. The maximum displacement is 0.23 m in roadway roof and 0.38 m in main roof when the face is adjacent to roadway. The vertical displacement is lower near roadway due to the supporting effect from roadway support. In horizontal direction, Figure 10(b) indicates that when the face is 30 m and further away from roadway, the roof displaces insignificantly. However, when the face moves closer to roadway, the roof significantly displaces towards the mined-out area. The maximum displacement is 0.08 m in roadway roof and 0.38 m in main roof when the face is adjacent to roadway. The horizontal displacement is greater near roadway because there is void space in mined-out area for the displacement. The deformation laws in the numerical model agree well with those reported from practice.

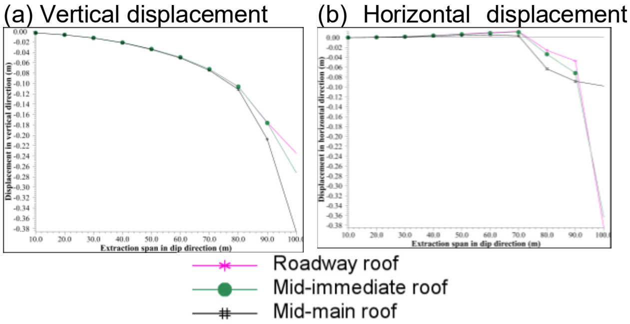

Figure 10. Displacement in roadway roof according to progressive extraction

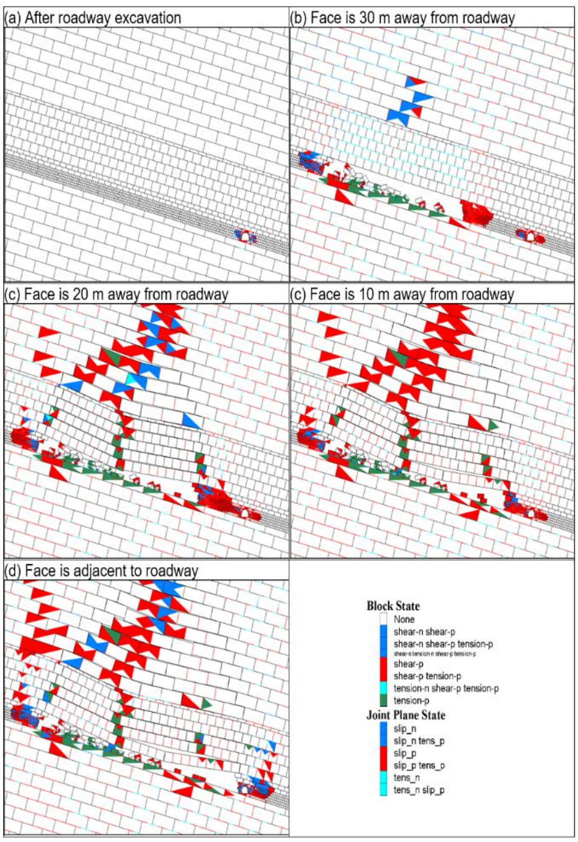

Figure 11. Deformation and failure of rock mass around roadway according to progressive extraction

The failure and caving of rock mass around the retained roadway are illustrated in Figure 11. The figure shows that after roadway excavation, the rock mass at sidewalls and roof fails (Figure 11(a)). The failure extent is 6 m away from roadway periphery, with the predominant failure mode being shear. Being consistent with the stress change, the roadway failure is not clearly affected when the face is far away from roadway. For the geo-mining condition being studied, the roadway failure becomes more apparent when the face is 30 m away from roadway (Figure 11(b)). When the face is 20 m away, the failure extends from the face to the roadway. When the face is adjacent to roadway, the rock mass around roadway fails towards the unmined seam, up to 25 m away from the sidewall. At this final stage, only immediate roof caves and forms a caving line of less than 70 degrees to seam floor. The strong main roof does not cave but sag downwards, causing significant load to deform and fail roadway as analysed above.

4. CONCLUSION

For better design and use of non-pillar mining technology into Vietnam coal industry, this paper presents a numerical study on the geomechanical behviours of roof strata and stability of retained roadway. Taking the geo-mining conditions from Nam Mau coal mine in Quang Ninh coal field as a case study, the study successfully develops a two-dimensional model of non-pillar mining method in seam dip direction. The analysis of model shows that when the face is far away from the retained roadway, the mining does not cause significant impact on roadway stability. When the face approaches the retained roadway, the vertical stress gradually increases around the roadway. The stress reaches a peak value when the face is 10–20 m away and then rapidly decreases when the roadway complete falls within the stress relief zone. In combination with the insignificant concentration of horizontal stress, the surrounding stress condition induces the roadway to deform and fail when the face approaches roadway. The study finds that the roadway is significantly affected when the face is adjacent to it. The rock mass around roadway fails up to 25 m away from roadway periphery. At the end of upper panel mining, only immediate roof caves that form a caving line of less than 70 degrees to seam floor. The strong main roof does not cave but sag downwards, causing significant load on roadway. It should be noted that the geomechanical behaviours in this study are limited to seam dip direction only where their impact on the next (lower) panel mining can be critical. Future studies may use a three-dimensional code for a more comprehensive analysis of behaviours. The results from study can serve as fundamental knowledge for design of roof cutting techniques, contributing to reducing the cost of using non-pillar mining method into Vietnam coal industry ❐

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AUTHOR CONTRIBUTIONS

Nguyen Trung Pham – Conceptualization and Review; Minh Khanh Pham – Data collection; Duong Quang Luu – Data analysis, Dung Tien Le – Modelling and Writing.

NGHIÊN CỨU MÔ PHỎNG SỐ SỰ ỔN ĐỊNH ĐƯỜNG LÒ DỌC VỈA TRONG KHAI THÁC KHÔNG TRỤ BẢO VỆ Ở MỎ THAN NAM MẪU

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TÓM TẮT

Sử dụng phương pháp mô phỏng số, nghiên cứu trong bài báo được thực hiện nhằm hiểu rõ hơn các biểu hiện địa cơ học của đá vách và sự ổn định đường lò chuẩn bị được giữ lại trong công nghệ khai thác không trụ bảo vệ, lấy điển hình ở mỏ than Nam Mẫu. Kết quả phân tích mô hình số cho thấy khi gương khai thác ở xa đường lò chuẩn bị, hoạt động khai thác không gây ảnh hưởng rõ rệt tới sự ổn định đường lò giữ lại. Khi gương khai thác tiến gần lại đường lò, ứng suất thẳng đứng quanh đường lò dần tăng lên. Ứng suất này đạt giá trị lớn nhất khi gương cách đường lò từ 10 tới 20 m và sau đó giảm mạnh khi đường lò lọt hoàn toàn vào vùng giải phóng ứng suất của lò chợ. Kết hợp với việc ứng suất ngang không tập trung rõ rệt, chế độ ứng suất xung quanh đường lò khiến đường lò bị biến dạng và phá hủy khi gương khai thác tiến lại gần. Nghiên cứu phát hiện thấy đường lò bị ảnh hưởng mạnh nhất là khi gương tiến sát đường lò. Khối đá xung quanh đường lò bị phá hủy tới phạm vi 25 m tính từ biên lò hướng về vùng chưa khai thác. Khi kết thúc khai thác lò chợ mức trên, chỉ có vách trực tiếp sập đổ tạo một góc phá hỏa dưới 70 độ so với trụ vỉa. Vách cơ bản bền vững không phá hỏa mà tách lớp hạ từ từ, gây ra tải trọng lớn lên đường lò. Các kết quả trong bài báo cung cấp những hiểu biết căn bản giúp thiết kế hiệu quả các kĩ thuật cắt vách, góp phần giảm chi phí áp dụng công nghệ khai thác không trụ bảo vệ vào công nghiệp khai thác than Việt Nam.

Từ khóa: *vỉa dốc thoải, khai thác không trụ bảo vệ, đường lò giữ lại, mô phỏng số, áp lực khối đá.*

@ Hội Khoa học và Công nghệ Mỏ Việt Nam