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Original paper

Stress environment and roof support needs for roadways ahead of the coal face: Case study from Ha Lam coal mine

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Abstract: Underground coal mining is characterized by confined working conditions and significant occupational safety hazards. Specifically, development headings driven along the coal seam ahead of the longwall face present a high risk of roof falls and challenging roof control conditions. **The aim** of this study is to analyse the stress distribution and the development of plastic failure zones in the surrounding rock mass, based on the mining conditions of Seam No. 10 at the Ha Lam coal mine. **The relevance** of the work is driven by the critical need to ensure the stability of development headings under the abutment pressure influence of the advancing longwall face, in order to enhance both safety and extraction efficiency. The main method employed involves numerical modelling of the rock mass stress-strain state, validated with in-situ monitoring data. **As a result**, it was established that the roadway ahead of the longwall face is consistently subjected to an asymmetric stress regime, with the horizontal-to-vertical stress ratio exceeding 2.5. The plastic zone in the surrounding rock develops asymmetrically between the two ribs of the roadway, leading to an unbalanced load. The magnitude of the stress and the extent of the plastic deformation zone around the ventilation roadway are significantly greater than those around the haulage roadway. Consequently, a supplemental support strategy is required to enhance roadway stability as it approaches the longwall face during mining operations. This paper provides a reliable foundation for the Ha Lam mine and other underground coal mines in Vietnam in developing plans to ensure roadway stability, improve production efficiency, and enhance occupational safety.

Keywords: coal face, roadway, numerical simulation, stress distribution, plastic deformation.

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Оригинальная статья

Напряженно-деформированное состояние и требования к креплению для выработок перед угольным забоем: пример угольной шахты «ХаЛам»

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Резюме: Подземная добыча угля связана с ограниченными условиями работы и значительными рисками безопасности труда. В частности, горные выработки, проводимые по угольному пласту впереди очистного забоя, представляют собой зону повышенного риска обрушения кровли и сложностей с управлением кровлей. **Целью** данного исследования является анализ распределения напряжений и развития зон пластической деформации во вмещающих породах на основе условий отработки пласта № 10 шахты «Ха Лам». **Актуальность работы** обусловлена необходимостью обеспечения устойчивости подготовительных выработок в зоне опережающего влияния очистных работ для повышения безопасности и эффективности добычи. В качестве основного метода использовано компьютерное моделирование напряженно-деформированного состояния породного массива, верифицированное данными натурных наблюдений. **В результате** установлено, что выработка, расположенная впереди очистного забоя, постоянно находится в условиях асимметричного напряженного состояния, при этом коэффициент соотношения горизонтального и вертикального напряжений достигает значений свыше 2,5. Зона пластических деформаций развивается асимметрично между двумя боками выработки, что приводит к неравномерной нагрузке. Величина напряжений и степень распространения зоны пластической деформации вокруг вентиляционной выработки значительно больше, чем у транспортной выработки. Следовательно, необходим план дополнительного крепления для повышения устойчивости выработки по мере ее приближения к очистному забою. Данная статья служит надежной основой для шахты «Ха Лам» и других подземных угольных шахт Вьетнама при разработке планов по обеспечению устойчивости горных выработок, повышению эффективности добычи и безопасности труда.

Ключевые слова: угольный забой, выработка, численное моделирование, распределение напряжений.

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Introduction

The longwall mining method is the primary approach adopted by underground coal mines [Babyr, 2024; Zuev et al., 2020; Szurgacz, 2024]. It is considered one of the safest, most productive, and efficient methods for coal extraction. In this approach, the stability of roadways along the seam plays a crucial role in production efficiency and occupational safety [Kuranov et al., 2020; Kongar-Syuryun et al., 2025], especially in the area ahead of the coal face and at the intersection between the coal face and the roadway along the seam [Esterhuizen et al., 2019; Zhu et al., 2022]. According to studies, this area is affected by the superimposition of static stress, lateral stress, and front support stress at the coal face. The roadway along the seam is susceptible to deformation due to the combined effects of support pressure exerted by the advancing coal face and the tendency of roof strata to shift into the excavated space. This results in a significant risk of collapse, as documented in references. Additionally, this area has a large space due to the intersection of the coal face and the roadway along the seam, serving as a location for multiple transport equipment installations, pedestrian access, and ventilation [Burtan et al., 2022; Kongar-Syuryun et al., 2025].

Therefore, a critical task for underground coal mines is maintaining the stability of the roadways to ensure safe and efficient production [Linh et al., 2021; He et al., 2023]. The stress distribution patterns, surrounding rock displacement, and deformation of the roadway at the coal face-roadway intersection and ahead of the coal face in Vietnam's underground coal mines have yet to be the subject of comprehensive study. Consequently, the measures for ensuring roadway stability have been based on production experience rather than scientific grounds [Kovalsky et al., 2025]. Currently, single hydraulic props are used as the reinforcement method in these areas at underground coal mines in Vietnam (see fig. 1). However, the issue of maintaining roadway dimensions for production and safety, while reducing repair costs, remains unresolved.



Fig. 1. Current support conditions at the coal face-roadway intersection and ahead of the coal face in some underground coal mines in Vietnam

A number of studies have been conducted on a global scale with the objective of gaining insight into the deformation mechanisms of roadways and the technology employed for the control of surrounding rock. A mechanical model was developed to provide support for roadways ahead of the coal face [Esterhuizen et al., 2020]. This was achieved by analysing the distribution of support pressure in hard roof rock, with the proposed solution being the use of cable bolts in place of single hydraulic props. A computational model

was established to evaluate the support capability and potential for roof rock subsidence. A methodology for determining the parameters of the support system to secure the roadway ahead of the coal face was proposed [Tyulyaeva et al., 2024]. In their research works [Liu et al., 2022; Sasaoka et al., 2020; Fan et al., 2023], the authors introduced active support technology, utilising cable bolts throughout the entire lifespan of the roadway. This was achieved by establishing a prediction model for the deformation of surrounding rock under the impact of longwall mining. Furthermore, the stress and deformation of the surrounding rock at the coal face-roadway intersection and ahead of the coal face were investigated, with the suggestion that the support structure of the roadway should be determined based on the stress distribution and deformation characteristics of the surrounding rock [Qiang et al., 2023; Mao et al., 2020]. The mechanical properties of the surrounding rock were studied when the roadway was supported by hydraulic props combined with cable bolts, resulting in the proposal of a heterogeneous support strategy [Koteleva et al., 2021]. A corresponding support theory was proposed for the roadway position ahead of the coal face, along with a method for relocating hydraulic supports. This was done in order to address issues where the roof and roadway anchor system fail due to high support pressure and significant roof subsidence [Zhao et al., 2024, Kongar-Syuryun et al., 2025]. Their research indicated that enhancing the adaptability of support systems in roadways ahead of the coal face could improve stability. In addition, a movable support device for roadways ahead of the coal face was developed, along with advanced mechanized support principles [Guo et al., 2024; Kongar-Syuryun et al., 2025].

The preceding analysis demonstrates that the extant research agenda is oriented towards elucidating the relationship between the deformation of the surrounding rock and the support system. However, it should be noted that geological conditions vary significantly, which in turn gives rise to notable differences in the deformation patterns of the surrounding rock. The deformation of roadways in the coal face-roadway intersection area and ahead of the coal face, as well as the stress distribution of surrounding rock in Vietnam's underground coal mines, are subjects that have yet to be adequately addressed in the academic literature. It is therefore evident that further research is required in order to develop advanced, safe and efficient support technologies. The present study focuses on Seam 10 at the Ha Lam coal mine in Vietnam. In-depth analysis is conducted on the stress distribution and deformation of the roadway situated ahead of the coal face, as well as at the coal face-roadway intersection. The findings of this research will serve as a foundation for the development and implementation of mechanized support systems in this region.

Methods

Characteristics of the Study Area

The Ha Lam coal mine is located in Ha Long City, Quang Ninh Province, Vietnam. The coal seam in the area has a gentle dip. The roof and pillar rocks are primarily composed of siltstone and shale, while in many areas, the pillar and roof rocks are sandstone, which is of low strength and unstable. The geological drill column is presented (see fig. 2). The study area is the mechanized longwall face CGH 10.3 of Seam 10, with a thickness ranging from 2 to 4 meters. The coal seam thickness across the entire longwall face is relatively stable, with an average depth of approximately 300 m. The immediate roof of the seam is shale, with a thickness varying from 0 to 6 meters. The preparation of the mining block is conducted through two parallel roadways along the seam, one of which is designated to serve the adjacent mining block (see fig. 3). The coal pillar between the

two roadways has a width of 25 m.

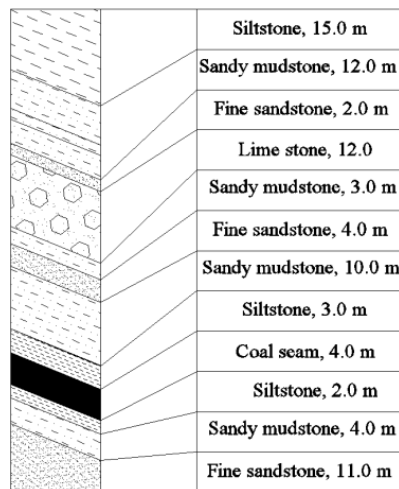


Fig. 2. Geological drill column in the study area

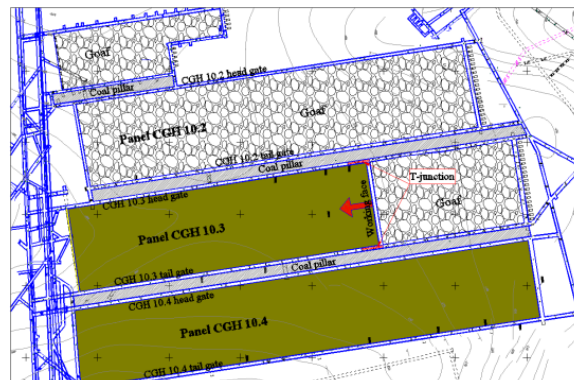


Fig. 3. Roadway layout in the study area

Occurrence Range of Hazardous Mining Stress Zones and Load Impact on Support Structures at the Intersection Area Ahead of the Coal Face

Occurrence Range of Hazardous Mining Stress Zones:

In the longwall mining system, a hazardous stress zone (see fig. 4) always appears at the intersection between the coal face and the roadway along the seam and ahead of it. The cause is believed to be the advancement of the coal face and the collapse of the roof rock in the excavated space, which disrupts the structure of the roof rock mass and continuously creates new equilibrium stress states. At the front abutment stress zone ahead of the coal face, mining stress reaches a peak, generating a large load, and roof sagging occurs, directly impacting the support structures of the roadway along the seam. At the intersection between the coal face and the roadway along the seam, mining stress diminishes, resulting in roof rock collapse and a tendency for the roadway to experience severe compression and deformation [Phuc et al., 2022]. According to the collapse arch hypothesis, the occurrence range of the hazardous mining stress zone ahead of the coal face can be determined by formula (1):

$$L = 0,1 \sqrt{\frac{nmH(1+\cos\alpha)}{2f}} \quad (1)$$

In which: n – coefficient for calculating the influence of other longwall faces in the calculation area, determined by formula (2):

$$n = 1 + n_1 + \frac{m_1}{h_1} \quad (2)$$

where: coefficient accounting for the influence of adjacent longwall face (if influenced, $n_1 = 1$; if not influenced, $n_1 = 0$), $\frac{m_1}{h_1}$ – coefficient accounting for the influence of the adjacent upper coal seam; m_1 – thickness of the upper coal seam, in meters; h_1 – distance from the upper coal seam to the calculation area, in meters.

H – depth of the working area, in meters; α – inclination angle of the coal seam, in degrees; f – average firmness coefficient of the roof rock (according to the Protodiakonov scale).

Under the conditions of Seam 10 with a thickness of 4 m, located beneath the previously mined CGH 10.2 longwall face ($n_1 = 1$), with the upper Seam 11 having an average thickness $m_1 = 6.0$ m, an average distance between the two seams $h_1 = 15$ m, a working depth $H = 300$ m, a coal seam inclination angle $\alpha = 50$, and an average firmness coefficient of the roof rock $f = 4$, the hazardous stress zone range L is determined from formula (1) as approximately $L \approx 3.0$ m.

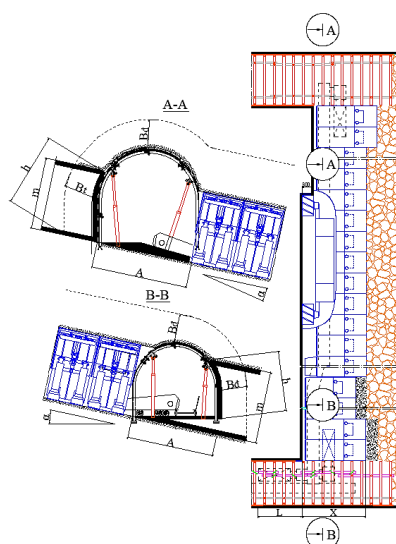


Fig. 4. Diagram for determining the range of the hazardous stress zone

Mining Pressure Acting on Support Structures in the Hazardous Stress Zone:

Mining pressure acts on the support structures of the roadway along the seam at the intersection of the coal face-roadway along the seam due to the load of both coal and roof rock in the collapse arch. The mining pressure on the support structures can be determined by formula (3):

$$q = \gamma_t B_t + \gamma_d B_d \left(1 + \frac{\gamma}{\gamma_d}\right) \quad (3)$$

where: γ_t – unit weight of coal, T/m^3 ; γ_d – average unit weight of roof rock, T/m^3 ; B_t – depth of the failure arch in the coal layer, which can be determined by formula (4):

$$B_t = \eta h \frac{\sqrt{2K_n \gamma_d H}}{1000 f_t k} \quad (4)$$

where: η – coefficient characterizing the inclination angle of the sliding prism; h – width of the exposed coal surface around the perimeter of the roadway, m; K_n – stress concentration coefficient based on the roadway shape, depending on the ratio of the roadway width in the inclined direction of the seam (A) to the width of the exposed coal surface (h); f_t – coal firmness coefficient; k – coefficient accounting for the reduction in coal firmness over time; B_d – height of the failure arch in the roof rock, determined by formula (5):

$$B_d = \frac{A \cos \alpha + 2B_t}{2kf_d} \quad (5)$$

where: A – roadway width in the dip direction of the coal seam, m; f_d – average firmness coefficient of roof rock in the collapse-prone area.

Y_d – allowable roof sag without requiring additional support, $Y_d = 30\text{--}45\text{ mm}$; Y – roof sag due to the influence of the coal face, determined by the empirical formula (6):

$$Y = \frac{5nm(1+\cos\alpha)}{2\sqrt{\frac{nmH(1+\cos\alpha)}{2f}}(V+C)} \cdot \left(\sqrt{\frac{nmH(1+\cos\alpha)}{2f}} + X \right)^2 \cdot 10^{-3} \quad (6)$$

where X – coal face width, m; V – average daily coal face advance rate; C – width of a shear cut, m.

Under the conditions at coal face CGH 10.3, the calculated mining pressure acting on the support structures is $q = 22,3 \text{ T/m}^2$.

This calculation result is quite consistent with field measurements at the transport roadway. However, on the ventilation roadway side, there are significant deviations (due to the absence of parameters reflecting the influence of the coal pillar and the space mined out by the adjacent coal face). To clarify further, the method of numerical simulation analysis has been applied to this study.

Development of the Numerical Simulation Model

The Flac3D software was used to simulate the longwall mining process. Flac3D, developed by ITASCA, is designed for continuous modeling of geomechanical mining environments. Based on the geological conditions of Seam 10 at the Ha Lam coal mine, a typical geological model was established. The lithology and thickness of the strata were selected and the chosen mechanical properties of the surrounding rock (see table 1). The vertical and horizontal displacements at the base of the model were fixed at zero, while the horizontal displacements of the four sides were constrained. Based on the rock mass density of 0.025 MN/m^3 , a vertical static stress of 7.5 MPa was applied to the top boundary of the model to simulate the static load of the rock mass. Gravity was included in the model, and the Mohr-Coulomb failure criterion was used for the numerical simulation (see fig. 5).

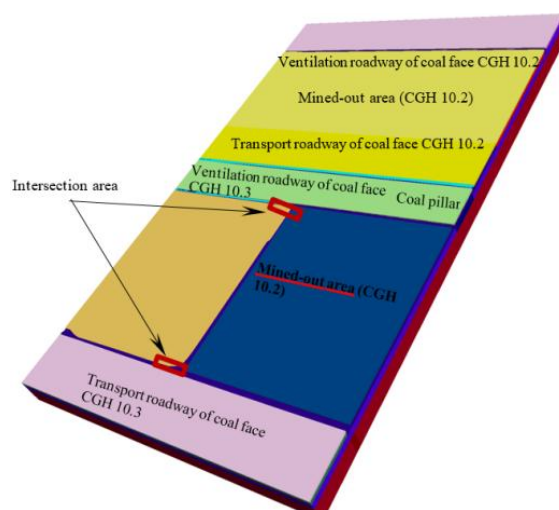


Fig. 5. Numerical simulation model

The simulation model was constructed based on a case study of Seam 10 at the

Ha Lam coal mine. The seam has an average thickness of 4.0 m, with an immediate roof and floor consisting of shale with a thickness of 3 m. The main roof is siltstone with a thickness of 10 m. The roadway is located 300 m below the surface, with a width of 4 m and a height of 3 m, and the coal pillar width is 25 m.

Table 1

Lithology and mechanical properties of rocks applied in the model
[Cai et al., 2024]

Parameter name	Sand-stone	Silt-stone	Shale	Coal
Tensile Strength (MPa)	1.6	0.9	1.2	0.4
Bulk Modulus (GPa)	7.456	2.333	1.822	0.748
Shear Modulus (GPa)	3.244	0.955	0.607	0.484
Poisson's Ratio, ν	0.31	0.32	0.35	0.26
Cohesion, c (MPa)	3.2	2.1	1.8	1.5
Friction Angle, ($^{\circ}$)	34	30	26	19
Unit Weight (kg/m^3)	2780	2550	2250	1450

The numerical simulation process was conducted in the following manner: (1) Setting up the static stress state and achieving the initial equilibrium state; (2) Excavating the transport and ventilation roadways along the seam at the coal face; (3) Mining the CGH 10.2 coal face; (4) Mining the CGH 10.3 coal face. During the simulation, the stress-strain state of the transport and ventilation roadways along the seam was extracted [Kongar-Syuryun et al., 2024; Kovalskiy et al., 2024; Tyulyaeva et al., 2024].

Simulation and Analysis Results

Stress Distribution Patterns Ahead of the Coal Face and at the Coal Face-Roadway Intersection

In the area of the intersection and in advance of the working face, the stress field within the surrounding rock mass becomes disrupted [Han et al., 2023; Kazanin, 2023; Li et al., 2023]. The stress distribution patterns at the intersection and ahead of the working face (see fig. 6). It can be observed that the stress distribution is asymmetric due to the influence of front and lateral support pressures [Zeng et al., 2024; Linh et al., 2025; Linh et al., 2024]. The peak stress near the coal wall or the primary coal pillar is typically located at the intersection between the coal face and the roadway along the seam. Conversely, the peak stress near the coal wall of the CGH-10.3 coal face is distributed 15 m ahead of the face.

At the transport roadway of coal face CGH-10.3, on the left side, significant stress concentration occurs at the coal face-roadway intersection and extends over a 35-meter length of the roadway ahead of the coal face. The maximum vertical stress reaches 15.8 MPa at the intersection, then increases to 17.2 MPa and decreases to 15.7 MPa at distances of 9 m and 24 m ahead of the coal face, respectively. The maximum vertical stress concentration factor reaches $k=2.3$. Similarly, large horizontal stress concentration is also found at this location. The horizontal stress value reaches 7.89 MPa at the intersection, then increases to 8.6 MPa and decreases to 8.0 MPa at distances of 10 m and 25 m ahead of the coal face, respectively.

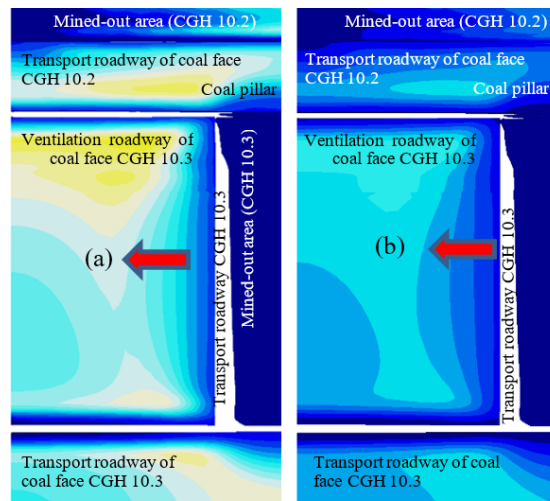


Fig. 6. Characteristics of stress distribution ahead of the coal face and at the coal face-roadway intersection: a – Vertical stress distribution; b – Horizontal stress distribution

In contrast to the left side, on the right side of the transport roadway, stress is significantly lower at the coal face-roadway intersection (vertical stress is 4.5 MPa, and horizontal stress is 2.3 MPa). The peak stress is found 25 m ahead of the coal face, with values of 16.4 MPa for vertical stress and 8.1 MPa for horizontal stress. Thus, it can be observed that due to the influence of support pressure, both vertical and horizontal stresses are distributed asymmetrically on the two sides of the roadway. This distribution leads to the formation of eccentric compressive forces acting on the supports, making them more prone to failure.

Stress concentration intensifies when there is a resonance between the front support pressure at the coal face and the support pressure formed by the previously mined-out space. Specifically, in this study, stress distribution around the ventilation roadway of coal face CGH-10.3 was examined. On the right side of the ventilation roadway (near the coal pillar), stress concentration was observed from the coal face-roadway intersection and developed over a 60-meter length of the roadway. At the intersection, the recorded vertical stress on the right side was 15.7 MPa (with a vertical stress concentration factor of $k=2.1$) and horizontal stress was 6.9 MPa. At a position 15 m ahead of the coal face, vertical stress increased to a peak of 18.3 MPa (with a stress concentration factor of $k=2.44$) and horizontal stress was 7.9 MPa. In contrast, on the left side of the ventilation roadway of coal face CGH-10.3, the vertical stress at the coal face-roadway intersection was only 2.2 MPa, and the horizontal stress was 1.05 MPa. This can be explained by the formation pattern of front support pressure at the coal face. The coal mass adjacent to the intersection lies within the failure zone due to the intense development of a fracture system, creating a stress relief zone. Subsequently, vertical stress peaks at 20.3 MPa (with a stress concentration factor of $k=2.7$) at a distance of 27 m, and then decreases to 19.3 MPa (with a stress concentration factor of $k=2.57$) at a distance of 50 m ahead of the coal face. Thus, vertical stress concentration on the left side is 1.1 times higher. Similarly, horizontal stress peaks at 10.1 MPa at a distance of 27 m ahead of the coal face. Horizontal stress remains high (9.9 MPa) even further from the coal face.

Thus, the asymmetric development of stress on both sides of the roadway was found in both the transport and ventilation roadways of coal face CGH-10.3. This formation mechanism causes uneven displacement of the rock mass, resulting in unbalanced forces

acting on the supports. Additionally, the sustained high horizontal stress is the primary cause of roadway compression ahead of the coal face.

When considering the stress environment in the roof rock above the roadway, the distribution between horizontal and vertical stresses in the rock mass (see fig. 7).

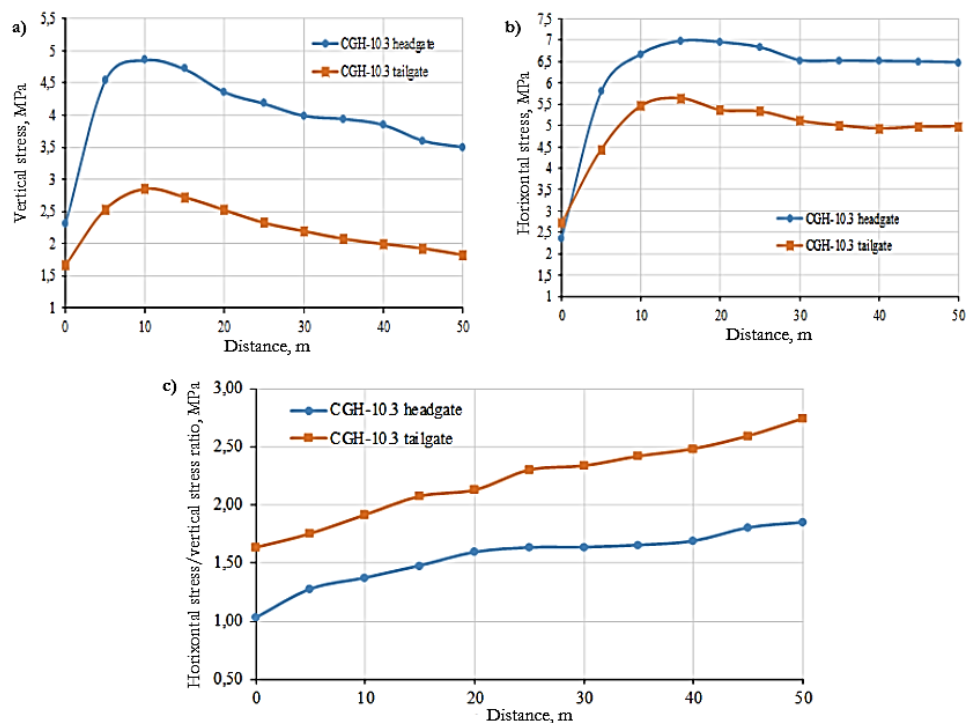


Fig. 7. Stress in the rock masses 3 meters above the roadway: a – Vertical stress distribution, MPa; b – Horizontal stress distribution, MPa; c – Horizontal/Vertical stress ratio, MPa

The reduced stress magnitude in the rock stratum located 3 m above the roadway is a consequence of its positioning within the stress relief region of the collapse arch structure. In the collapse arch, a system of fractures develops significantly in the roof rock, causing the rock layers to separate and crumple. Although the stress is not high, the entire rock mass within the collapse arch will form a direct load acting on the roadway's support system. The stress distribution still follows the pattern with peak stress located 10 m ahead of the coal face. Comparative analysis reveals elevated roof rock stress in the ventilation roadway relative to the transport roadway. This stress elevation is a consequence of the combined loading from dual support pressure zones in the ventilation roadway. The first component is the lateral support pressure from the adjacent mined-out area above, which acts on the coal pillar and ventilation roadway. This pressure formed during the coal extraction and roof collapse in the mining space of the adjacent CGH-10.2 coal face. As the CGH-10.3 coal face advances forward, the ventilation roadway falls within the second support pressure zone - the front support pressure of coal face CGH-10.3. The resonance of both support pressure zones expands the collapse arch above the roadway and increases the load from the roof rock acting on the support system. This explains why the support system of the ventilation roadway experiences more severe destruction and deformation compared to the transport roadway.

Horizontal stress is consistently greater than vertical stress in the zone ahead of the coal face (see fig. 7a and fig. 7b). As the distance from the coal face increases, the ratio of horizontal to vertical stress rises. At the intersection between the coal face and the

roadway along the seam, this ratio tends to approach the value of hydrostatic pressure ($k=1$). This may be because the roof rock mass has reached its critical displacement, causing a new equilibrium state to be achieved behind the intersection. With a horizontal-to-vertical stress ratio greater than 1, the rock mass tends to shift inward toward the roadway. This is the primary cause of roadway deformation and support system failure near the coal face.

Development Patterns of the Plastic Deformation Zone Around the Roadway

The distribution characteristics of the plastic zones in the surrounding rock at various positions of the ventilation and transport roadways of coal face CGH-10.3 (see fig. 8 and fig. 9). It can be observed that the shape and extent of the plastic zone in the surrounding rock of these two roadways are entirely different.

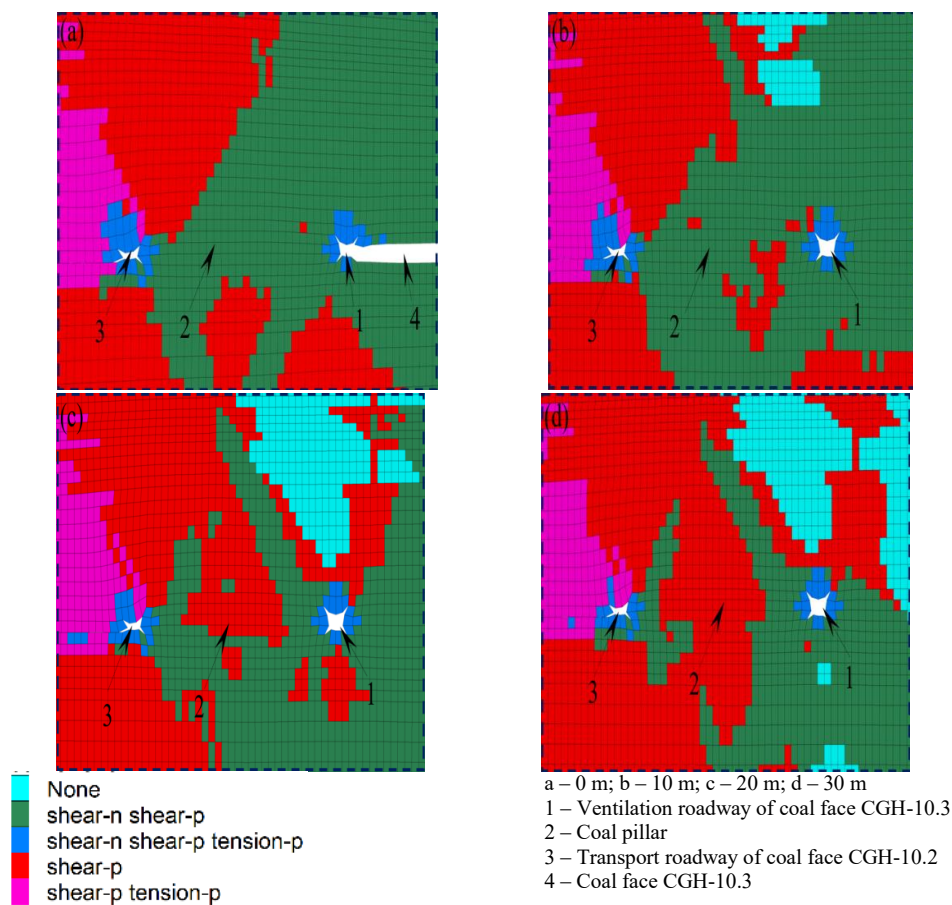


Fig. 8. Distribution of the plastic zone in the rock surrounding the ventilation roadway at different positions ahead of the working face

The plastic deformation zone in the rock mass within the ventilation roadway of coal face CGH-10.3 has reached a point where it covers the entire roadway area and coal pillar. The roadway is subjected to lateral mining stress originating from the previous coal face and front support pressure exerted by the current coal face. The extent of the plastic zone in the surrounding rock is markedly increased in the vicinity of the coal face. At the intersection, the plastic zone has developed extensively throughout the rock mass and coal pillar, resulting in significant deformation of the roadway. A minor elastic zone

has manifested in the roof rock above the coal face at a distance of approximately 10 m. Nevertheless, within a 15-metre radius surrounding the roadway, the plastic deformation zone, characterized by the formation of fractures, can still be observed. At a distance of 20 m ahead of the coal face, the elastic zone expands and persists in the roof rock, situated at a height of 3 m above the roadway. The plastic deformation zone in the rock mass continues to expand on both sides and beneath the roadway. At a distance of 30 m from the coal face, an elastic zone is visible on the right side and beneath the roadway. However, in general, the extent of the plastic zone in the vicinity of the coal pillar (on the left side) is greater than that on the coal wall side (on the right side), with both zones distributed asymmetrically. The extent of the plastic deformation zone is observed to span the entire coal pillar, extending for a distance of 25 m. The mining of the previous coal face has been identified as a factor that has disturbed the stress field in the surrounding rock mass. The ratio of horizontal to vertical stress is estimated to be approximately 1.5, and the direction of the principal stress is also observed to be skewed. As a result, the plastic zone in the surrounding rock exhibits an asymmetric distribution.

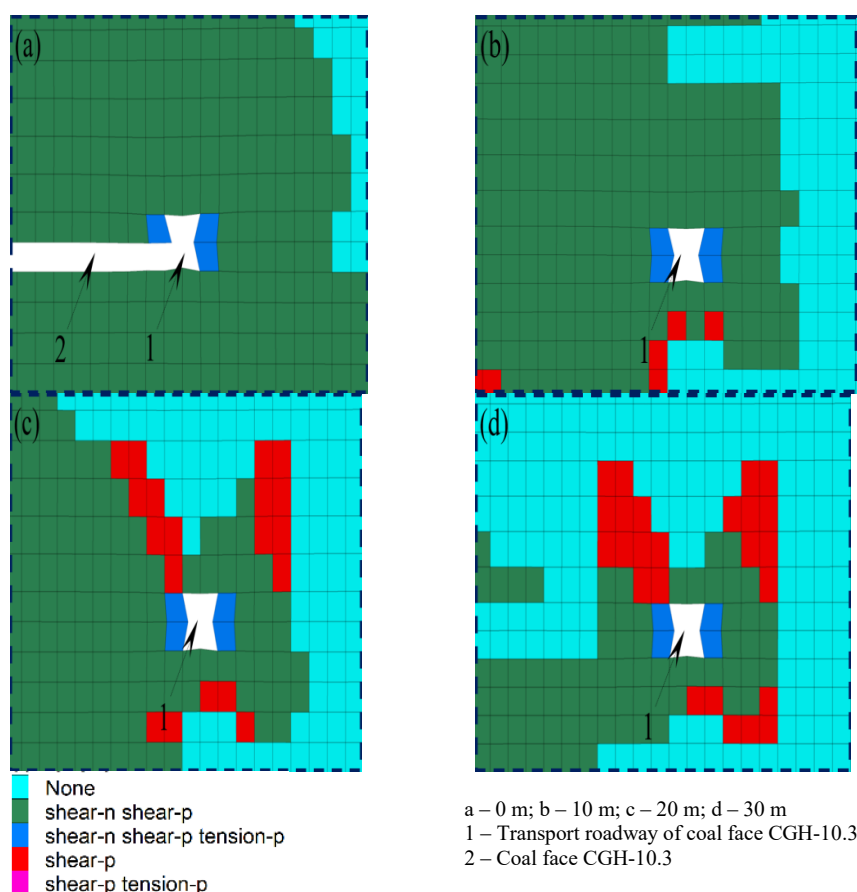


Fig. 9. Distribution of the plastic zone in the rock surrounding the transport roadway at different positions ahead of the working face

The transport roadway on both sides of coal face CGH-10.3 is comprised of solid coal, exhibiting no discernible impact from the mining activities associated with coal face CGH-10.3. The distribution of the plastic zone in the surrounding rock remains asymmetric in proximity to the coal face, extending approximately 10 m beyond the active working area. The plastic zone in the roof rock is primarily developed on the left side in the vicinity of the coal wall, which is in close proximity to coal face CGH-10.3. As the distance from the

working face increases, the asymmetric distribution of the plastic zone in the surrounding rock transitions towards a more symmetric distribution. The distribution of the roof, floor, and plastic zone on both sides of the roadway is symmetrical.

Therefore, in closer proximity to the coal face, the surrounding rock mass of the roadway is situated within a non-uniform stress field (horizontal/vertical stress ratio $\neq 1$), wherein the plastic zone in the surrounding rock transitions from symmetric to asymmetric. The horizontal-to-vertical stress ratio can exceed 2.5, with the plastic zone developing throughout the surrounding rock mass over a range of approximately 15 m. The zone of plastic failure primarily develops on the coal pillar side (in the ventilation roadway) and on the coal wall side of coal face CGH-10.3 (in the transport roadway). The zone of plastic deformation exhibits considerable growth within a distance of 30 m in advance of the coal face. Approaching the coal face results in an increase in the extent of the plastic deformation zone. This expansion increases the susceptibility to rock bursts, bilateral roadway compression, and support system deterioration. In light of these findings, it is possible to propose the implementation of enhanced support systems and the selection of appropriate supports in order to guarantee the safety of the roadway situated in advance of the working face.

Conclusion

The proposed technical solution serves as a reliable reference for Ha Lam coal mine and other underground coal mines in Vietnam in developing plans to ensure roadway stability, improve production efficiency, and enhance occupational safety. This solution allows for the delineation of the following conclusions:

- Due to the influence of front and lateral support pressures, the coal face-roadway intersection and the roadway ahead of the coal face are often in an asymmetric stress field. The ratio of horizontal to vertical stress in the roof rock of the roadway can reach up to 2.5, and the maximum vertical stress can be 2.57 times greater than hydrostatic stress. This leads to asymmetric damage in the roof rock of the roadway.
- The development range of the plastic deformation zone can extend over 15 m in the surrounding rock of the roadway and up to 30 m ahead of the working face. Particularly at the coal face-roadway intersection, the surrounding rock mass is almost entirely disturbed, with the collapse arch of the roof rock expanding and increasing the load on the roadway support system. Consequently, the risk of rock bursts and support failure increases on the roadway ahead of the coal face.
- Based on the case study of Seam 10 at the Ha Lam mine, Vietnam, within 30 m ahead of the coal face, the roadway must be supplemented with high-load bearing supports to ensure the stability of the roof rock in the plastic deformation zone. Specifically, suitable supports must be selected for the coal face-roadway intersection, where the surrounding rock mass of the roadway is completely disturbed.

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