Case study on the mining-induced stress evolution of an extra-thick coal seam under hard roof conditions

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Abstract
A strong mining disturbance may cause the superposition of local stresses and serious disasters such as crushed supports and severely destroyed roadways. To study the distribution law of mining-induced stress under hard roof conditions, an advance abutment pressure distribution model was developed. By monitoring the support pressure, a novel method of reckoning intensive factors was proposed, and then, the pressure distribution characteristics were obtained. An innovative in situ test of the mining-induced stress increment was performed. At the Tongxin coal mine, the field monitoring of no. 8309 working face revealed the following. (a) The peak position of the fully mechanized working face with a hard roof was 5.9 m away from the mining face in which the average intensive factor $K$ was 1.48. The pressure of the single props developed could be roughly divided into the fluctuation, slow increase, and stable stages. (b) A theoretical model of the stress evolution of the working face was proposed. Using an in situ uniaxial compression test, a law governing the variation of the advance abutment pressure was developed. The peak strength was reported to be 36 MPa, which is consistent with the advance prop pressure. (c) Mining-induced stress violently fluctuated during the process of top coal caving; the $K$ values in the supports in the middle of the working face and their rates of growth were apparently higher than those in the end supports. A Gaussian distribution of $K$ was established for this phenomenon. The results obtained provide first-hand data for safe and efficient mining under similar geological conditions.

KEYWORDS
intensive factor, longwall top coal caving, mining-induced stress evolution, support working resistance

1 | INTRODUCTION

The complexity of geological coal seam phenomena¹ such as extra-thick coal seams under hard roof condition leads to accidents such as support crushing,² high-degree weighting,³ rock burst,⁴,⁵ and coal and gas outbursts.⁶,⁷ Subsequently, these conditions impact fractures in top coal caving,⁸,¹⁰ which hinders the safe and efficient mining operations in longwall top coal caving (LTCC) regions.¹¹,¹² In many studies, the behaviors of hard roofs have been linked to their breakage; therefore, control methods have been investigated from the viewpoint of understanding the influence of roof...
presplitting, strata control, and evolution of mining stress. Li et al. reported that a hard roof was among the factors attributed for the failure of supports. Yasitli et al. performed a numerical simulation that demonstrates the substantial fracture of a 1.5-m-thick layer of coal just above the shield supports and the roughly broken top coal (3.5 m). Gao et al. reported that the abutment pressure, which can be attributed to mining operations, exhibited consistent trends for the entire time period. Based on in situ conditions, Alehossein et al. developed a yield and caveability criterion. Using field monitoring, Xie et al. reported that the total intensive factor of pressure comprises the intensive factor of pressure for both local and global mining mechanics and described that the former is generated from the coal caving process, and therefore, it cannot be neglected. For correcting the working resistance of hydraulic support, Guo et al. proposed a load estimation method, considering the suspension distances of key strata. Xie et al. examined the deformation and failure mechanism during the top coal caving process and discussed the changes in the advance abutment pressure. Tong et al. established the initial fracture mechanical model for a thick hard roof based on the long beam theory. He et al. noted that, while mining, a thick hard roof would result in a high stress concentration in the surrounding rocks and that its destruction would lead to a dynamic loading effect. Prusek et al. proposed that, for analyzing shield and roof strata interaction, a system that enables the characterization and mining conditions and is appropriate for shield capacity determination and selection should be developed. For relatively deep mining, Wagner reported that, for design and selection criterion, the dynamic loading situations should be considered and the ability of the support to absorb significant amount of energy should be included. Xue et al. proposed a statistical yield criterion to obtain the failure criterion for the discontinuous stress redistribution coupling. Wang et al. developed and meticulously validated a numerical model to investigate energy redistribution in coal seams and barrier pillars. Several studies have focused on assessing the impact of these dynamic phenomena on shield support loads, while certain studies suggested the development of new control methods for effectively reducing the abutment pressure control roof. However, as per the special geological conditions of a hard roof in extra-thick coal seams, stress distributions in LTCC and its support pressure require additional investigations. Datong mining area exhibits typical geological conditions: hard roof, hard coal, hard floor, large mining height (14.88 m), and high mining speed (5-8 m/day). A hard roof results in an intensive mining condition. Under the mining effects of a strong disturbance, the loading rate increases with rapid mining, thereby causing dynamic disasters. Severe strata behavior-related problems easily occur in the working face, causing theoretical and technical difficulties for conducting safe and efficient mining. Therefore, for deep mining, stress evolution analysis is crucial. To improve support plans around intersections, Sinha and Chugh studied the stress and strain distributions around an intersection. However, a study revealing the vertical stress distributions of large-space stopes under the condition of fully mechanized caving mining in extra-thick seams is important.

2 | MECHANICAL RATIONALE

On engineering scales, compared to laboratory tests, stress-strain measurements could be extremely and comparatively difficult, because directly obtaining the accurate advance abutment pressure is almost impossible. Nevertheless, certain geophysical methods are applicable for inverting the stress field. For example, a microseismic event can be considered; however, it only confirms the relative state of stress and its redistribution, thereby reflecting the considerable amount of work required for improving conventional methods. Herein, considering the actual situation of no. 8309 working face and drawing on Tan’s philosophy, a simplified model that combined laboratory and field tests for advance abutment pressure was established. We assume that in situ uniaxial specimen and the coal-rock mass synchronously deform. Typically, the stress in coal seams is expressed as follows:

$$\sigma_{h,CM} = \lambda \sigma_{v,CM},$$

where $\lambda$, $\sigma_{h,CM}$, and $\sigma_{v,CM}$ are the lateral pressure coefficient, horizontal stress, and vertical stress in the coal seam, respectively. And the increment of the horizontal stress, $\Delta \sigma_{h,CM}$, can be written as follows:

$$\Delta \sigma_{h,CM} = \lambda (\sigma_{v,CM} - \gamma H) = \lambda \Delta \sigma_{v,CM},$$

where $\gamma H$ is the self-weight stress of the overburden. Accordingly, assuming a plane strain problem along the mining

![FIGURE 2 Borehole of no. 8309 working face](image)
direction, the increment of the horizontal strain in coal seam, \( \Delta \varepsilon_{h-CM} \), can be written as follows:

\[
\Delta \varepsilon_{h-CM} = \frac{1 + \mu}{E} [(1 - \mu) \Delta \sigma_{h-CM} - \mu \Delta \sigma_{v-CM}],
\]

where \( \mu \) is Poisson's ratio. Considering that the maximum principal stress and the minimum principal stress at the test site are both horizontal, Poisson's ratio of coal seam is defined as follows:

\[
\frac{\mu}{E} = \frac{\Delta \varepsilon_{v-CM}}{\Delta \varepsilon_{h-CM}} = \frac{\Delta \sigma_{v-S}}{E \Delta \sigma_{h-CM}},
\]

where \( \Delta \varepsilon_{v-CM} \) is the increment of the vertical strain in the coal seam estimated by in situ uniaxial specimen. \( \Delta \sigma_{v-S} \) is the increment of the vertical stress of samples.

Solving Equations (1)–(4), \( \sigma_{v-CM} \) becomes:

\[
\sigma_{v-CM} = \gamma H + \frac{\Delta \sigma_{v-S}}{\mu(1+\mu)(1-\mu)-\mu} (\Delta \sigma_{v-S} \rightarrow t),
\]

which is a simplified distribution model of the advance abutment pressure in which \( \mu \) can be obtained using laboratory tests. Once the time-varying vertical stress increment \( \Delta \sigma_{v-S} \) is captured, the law of stress evolution can be determined. Generally, using conventional methods, the real-time monitoring of the increment of vertical stress \( \Delta \sigma_{v-S}(t) \) is difficult. For this purpose, the in situ loading test was designed. Because the increments of different initial stresses are different, the initial in situ stress should be applied in the field test. Note that this method has the advantage of repeatability.

For determining the mining-induced stress evolution in an extra-thick coal seam under hard roof conditions and for verifying the model's accuracy, subsequent experiments were conducted.

3 | FIELD TESTS

3.1 | Geological conditions

Figure 1 shows the field test scheme designed for no. 8309 working face, which has a capacity of 5 million tons. The cover depth was 476.3–601.0 m, and the average coal seam thickness was 14.88 m. The coal cutter and caving had mining thicknesses of 3.9 and 10.98 m, respectively; moreover, the layout ratio was 1:2.81. The coal seam had a relatively stable thickness in the working face, and its dip angle ranged between 0° and 3° or average value of 1.5°. The borehole histogram is shown in Figure 2.
3.2 Test scheme

The test was divided into three parts, that is, hydraulic support pressure, single prop pressure, and in situ loading test. The working resistances of the supports adopted a KJ216 monitoring system made by Uroica Mining Safety Engineering, Inc., China. The system transmitted the pressure values to the computer system, thereby realizing continuous online monitoring. The test schemes were as follows:

1. **Hydraulic support pressure in the working face.** A total of 118 supports were placed on the working face; as shown in Figures 3 and 4A, twelve pressure indicators were installed at support locations #8, #18, #28, #38, #48, #58, #68, #78, #88, #98, #108, and #112. As the mining face advance, pressure information was recorded using the KJ216 monitoring system.

2. **Single prop pressure at the tail entry.** DWX45-150/110 single hydraulic props were adopted for this test. Figures 3 and 4B show the designed 50-m advance support with a spacing interval of 1.5 m, and the monitoring data were obtained from June 1 to July 4, 2018.

3. **In situ loading test.** First, coal samples were selected from no. 8309 working face and transformed into standard specimens. Next, a rectangular cut (200 mm deep, 300 mm wide, and 400 mm high) was excavated at a distance of 100 m from the mining face and 1.5 m from the floor height. Figures 3, 4C, D show the site and scene images. Note that the rigid base, flat jack, GPD450M stress sensor, and specimen and loading indenter were sequentially installed from bottom to top, followed by the application of certain initial stress. Finally, as the mining face advanced, the stress changes were recorded using stress sensors.

4 PRESSURE EVOLUTION LAW OF THE WORKING FACE

4.1 Single prop pressure

In the mining process, the concept of the stratum pressure provides the mining mechanics conditions of coal and rock masses. Note that stratum pressure plays an important role in selecting the support of the working face to prevent roof caving. Because of the influence of the field-working conditions, measurements of 0 MPa often appear during the data acquisition process. Furthermore, because the initial supporting force of the individual props was > 10 MPa, the threshold of data filtering was set to 10 MPa. Figure 5 shows the denoized results of the individual prop pressure. As the mining face moved forward, the abutment pressure showed an upward trend, which could be roughly divided into three stages: fluctuation, slow increase, and stable stages. Within a range of 40-50 m from the mining face, where the coal was not affected by mining, the change in abutment pressure was insignificant, which induced a steady trend. Within a 5.9-40 m range, coal encountered mining influences and the pressure slowly increased. Finally, in a range of <5.9 m, both breakage and fragmentation occurred in certain areas with the abutment pressure exceeding the coal’s ultimate strength. Moreover, the bearing capacity of coal was affected
and the pressure of a single prop considerably varied. At this point, the maximum support pressure could be claimed to reach 36.6 MPa and the average $K$ was 1.48, which is at a high level. This indicates that intense mining condition led to the apparent breaking of the mining face along with a timely completion of the support of the surrounding rock roadway.

Figure 6 shows the pressure case for #1 support. Note that an advance of the mining face was accompanied by each station moving once in a certain support cycle. During the whole monitoring period, #1 prop moved three times into location marks M1, M2, and M3; however, the primary feature was the considerable drop in pressure. Then, with the mining face moving forward, the prop pressure exponentially fluctuated until reaching peak positions P1, P2, and P3 with corresponding values of 28.4, 32.1, and 32.9 MPa, respectively, for the three load cycles. During the monitoring period of 21.6-266.3 h, the prop experienced a violent fluctuation process, which reflected the adjustment process of the roof.

4.2 | Hydraulic support pressure

Longwall top coal caving realizes the simultaneous extraction of both coal and top coal. In a system comprising supports and roofs, the supports extend to the control roof through the top coal; the pressure of the roof is then simultaneously transferred to the support through the top coal. For safe and productive mining, effective working resistances and appropriate roof control technologies are two critical factors. Therefore, monitoring and regulating the hydraulic support pressure are important.

Accordingly, the monitoring of the support pressure can effectively show the spatial distribution and evolution law of the stress field under intensive mining conditions. Figure 7 shows the pressures at the upper, middle, and lower hydraulic supports selected for 48h monitoring. During the coal caving process, the working resistance severely fluctuated. Most fluctuations occurred within 20-40 MPa and reached a maximum of 51.2 MPa. In fact, the working resistance of the hydraulic support was greater than that of the props, reflecting the strong fluctuations in the mining stress during caving and presents discernible periodic phenomena. Furthermore, during the monitoring period, the pressure rapidly dropped by 8-14 cycles, indicating that the mining face conducted 8-14 working cycles. In particular, the pressure increase rate of the middle part was generally higher than that of the end part. The pressure intermittently fluctuated, which could be attributed to two reasons: (a) the supports may not have been fully fit with respect to the top coal and resulted to fluctuations in the support pressure and (b) the caving process would cause local disturbances to the top coal, easily causing intermittent fluctuation characteristics.

4.3 | Preliminary study on the evolution of real mining-induced stress

Figure 8 shows the in situ uniaxial compression test, which is among the experimental methods used for obtaining the real evolutionary law of the mining stress. During the early stage, a relatively slight change or almost no fluctuation in the axial stress was observed on the coal sample. Subsequently, the coal sample underwent a weak disturbance stage in which the working face was far from the experimental position and mining stress fluctuated within a certain range. During the final stage of the experiment, the axial pressure sharply increased sharply. Figure 9 shows the results of the laboratory test in which brittleness was characteristically emphasized. The loading in the room was continuous, whereas that in the
field strongly fluctuated. As a result, the failure modes may be different. Figure 10 shows the failure mode of the uniaxial loading test. In terms of the failure mode, the coal sample F1 demonstrated an X-shaped shear failure characteristic. Both F2 and L6 are compression-shear failures. It is apparent that under the influence of mining, the axial loading continuously increases until the maximum shear stress upon the failure surface exceeds the shear stress intensity, at which point the coal sample experiences shear failure along this surface. L2 shows the characteristics of tensile failure. The Poisson effect of the rock material is responsible for the transverse tensile stress in the coal sample. Overall, the indoor specimen is relatively complete after failure. Because of the fluctuation in the mining stress, the damage to the field samples is greater, and the cracks are more numerous.

As per the geological conditions and laboratory test results, $\gamma H = 14\text{MPa}$, $\lambda = 0.7$, and $\mu = 0.226$. Along with Equation (5), the advance abutment pressure, as shown in Figure 11, can be obtained.

Figure 11 shows that the peak pressure at 36 MPa is almost the same as that of the advance single props, which indicates the accuracy of the model for evaluating the peak of the abutment pressure.

**Figure 7** Pressures of the supports at different positions. A, #8 (12 m to the head entry). B, #58 (100 m to the head entry). C, #108 (near the tail entry)

**Figure 8** In situ loading test

**5 | REGULARITY OF THE HYDRAULIC SUPPORT PRESSURE AT DIFFERENT POSITIONS**

A normal cycling operation corresponds to a load cycle of a support. Note that the hydraulic support pressure was crucial; therefore, Trueman et al.\textsuperscript{46} developed a software to accurately
delineate key features such as the set pressure, the intensive factor $K$, and the rate of loading $K/\Delta t$. Figure 12 shows the $K$ values and its growth rate of the support within 48 h at different locations where the distribution of $K$ was apparently concentrated at both ends. The $K$ growth rates of the middle supports were higher than those of the end supports. This demonstrates that a higher loading rate was experienced by the central supports and led to the probability of a dynamic catastrophe and a considerable increase in the extent of damage. During the monitoring period, the sound of coal burst, accompanied by vibration and falling debris, was occasionally heard.

Generally, an increase in the mining speed induced the dynamic impact tendency of a catastrophe. Therefore, a reasonable mining speed can alleviate the degree of stratum behavior; moreover, as $K$ shows the dynamic fluctuation intensity of the roof, obtaining its accurate value $K$ is crucial. Moreover, for each load cycle, the pressure curve indicates a quite different interval time. This study considered such factors for a proposed method. Assuming that the average $K$ is constant, the formula for interval time can be defined as follows:

$$\frac{K}{\Delta t} \cdot \Delta t = \text{constant},$$  \hspace{1cm} (6)

where $K$ is the intensive factor and $\Delta t$ is the interval time. If the relation between $\Delta t$ and $K/\Delta t$ is drawn and fitted with a power
Accordingly, the average value of $K$ in Figure 13 was 1.15, which is slightly lesser than the conventionally calculated value of 1.17. The power exponent in the equation could show fluctuation in $K$. In particular, the average fluctuation range in the working face was 0.12. Figure 14 shows the $K$ factors at different positions of the working face with respect to the proposed method.

In the middle mining face, the $K$ value was significantly higher than that in the end supports, which is probably attributed to the support of the solid coal at both ends. Normally, support selection is focused on the support capacities of the end supports and the smaller capacities of the middle support. Nevertheless, the middle support may be faced with a higher loading rate. The Gaussian function fitting (Figure 15) formula of the $K$ value can be expressed as follows:

$$y = 1.1533x^{-0.988},$$

(7)

$$K(x) = 0.3425 \exp \left(-\frac{(x-85.64)^2}{13.86}\right) + 1.112 \exp \left(-\frac{(x-89.08)^2}{457}\right),$$

(8)

Because the middle support may face a significant dynamic load, their strengths and impact resistances should be considered during design and upon actual selection. If such strong supports experience difficulty in meeting the demands under thick coal and hard roof conditions, alternative measures, such as forced caving and water injection softening, may be considered.
Figures 16 and 17 show the nephogram of the support pressure and its frequency distribution in which the end supports were shown to be invalid for a long-term period because of the corresponding lower roof activities, whereas the middle supports were subjected to a higher pressure. In particular, pressure values were concentrated at 0 and 25 MPa. Figure 18 shows the short-term case in which three low-pressure zones were observed, namely, (a) and (c) show the low pressures of the end supports and b) shows the malfunctioning of the #68 hydraulic support. Figure 19 shows the support experiencing leakage; therefore, it could not function as an effective support (or a decreasing type), as described by Peng et al.47 and Prusek et al.

During LTCC, the pressure changes were different from those of traditional methods. Furthermore, a significant pressure fluctuation was observed in the rear leg in LTCC. Figure 20 shows a typical coal caving process occurs for #58. The entire process can be divided into three stages, namely, initial, intermediate, and final. The initial stage was described by the primary feature of gradually increasing pressure at the support. Such pressure tended to stabilize as the shearer gradually moved away under a growth rate that gradually slowed down. Next, in the process of coal caving, the pressure considerably fluctuated, which marked the intermediate caving stage. In the final stage, the shearer continued to approach after completing a footage cycle, and the pressure sharply increased until the peak was attained.

6 | CONCLUSIONS

Herein, the field test was aimed at investigating the regularity of mining vertical pressure, and the results are as follows:

1. An advancement in the mining face was accompanied by apparent increase in the single prop pressure, which can be roughly divided into three stages, namely, fluctuation, slow increase, and stable stages. Within 40-50 m from the mining face, the pressure showed a steady trend; within 5.9-40 m, it slowly increased; and finally, in the
range of <5.9 m, the coal entered the limit equilibrium area.

2. The working resistances of the hydraulic supports demonstrated periodic fluctuation characteristics, and each cycle presented an overall upward trend, which can be roughly divided into three stages. Accordingly, during the coal caving process, the working resistance of the hydraulic support was evidently higher than that of the single props, which reflects the severe fluctuations in mining stress.

3. A model of the stress evolution of the working face was established. The distribution of the advance abutment pressure was obtained as per in situ loading testing, which reflected the upward fluctuations of the pressure that ultimately reached the peak strength of 36 MPa at 5 m, which is consistent with the trend of the single prop pressure.

4. For the accurate calculation of the intensive factor $K$, a novel method was proposed. Using this method, the $K$ value and its growth rates of the hydraulic supports in the middle of the working face were obviously higher than those at the end supports.

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CONFLICT OF INTEREST
None.

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