INTRODUCTION

Longwall mechanized top coal caving mining (LMTCCM) is the most universal method for the mining of extra-thick coal seams in China. A gob-side entry (GSE) as walking, ventilation, and transportation passage is extremely important for the efficient production and safety of coal mines. The coal pillar width directly affects the stability of the GSE. The importance of the coal pillar width is mainly reflected in the following three aspects: (a) It influences the abutment stresses, surrounding rocks and occurrence environments of GSEs; (b) it affects the mining recovery rate of coal resources, and (c) it could isolate gangues, water, fires, and harmful gases in the gob. Thus, the coal pillar is an essential part for longwall mining system, and its reasonable widths are the primary factor to be analyzed before the GSE development.

A coal seam with a thickness greater than 8 m is called extra-thick coal seam in China. Extra-thick coal seams account for ~1/4 of China’s annual coal production capacity of 3.394 billion tons. In the past, studies of the coal pillar width mostly focused on thin (thickness ≤ 1.3 m), medium (1.3 m < thickness ≤ 3.5 m), and normal (3.5 m < thickness ≤ 8 m) coal seams. However, the huge mining space and high-intensity exploitation in LMTCCM in extra-thick coal seams lead to the mining pressure principles that are obviously different from coal seams with other thicknesses. Thus, the determination of a reasonable coal pillar width and the correlation with increasing coal seam thickness and mining height have become problems that must be solved.
A reasonable coal pillar width decreases the stress environment of the GSE and could effectively block gangue and toxic gases in the adjacent gob. At present, three common coal pillar widths are used in longwall mining: (a) wide-pillar; (b) narrow-pillar; (c) no-pillar (Figure 1). The width of the wide pillar is 20-50 m. It guarantees the stability of the GSE during the mining of thin, medium, and normal coal seams. However, the abutment stresses of extra-thick coal seams are large and their influence ranges are wide. Thus, the applicability of wide pillars in extra-thick coal seams must be further studied. The wide pillar inevitably wastes many precious coal resources. The waste in extra-thick coal seams is more than 2.3 times that in medium coal seams with the same pillar width. Without a pillar, the coal resources can be efficiently exploited, but the panels are affected by gangue, water, fires, and harmful gases in the gob. Therefore, artificial pillars must be constructed to isolate the gobs and support the GSE, resulting in a complex mining process and high cost. In view of the above-mentioned problems, the narrow-pillar has become a popular choice for the GSE in LMTCCM. Reasonable width designs of coal pillars must consider the specific mining and geological conditions of coal mines such as the properties of the coal and rock masses, environments surrounding the GSE, and in situ stresses. Thus, the correlation between the GSE stability and pillar width in the context of LMTCCM in extra-thick coal seams has been studied in this work to determine the reasonable pillar width to ensure the stability of the GSE and provide references for coal mines with similar mining and geological conditions.

Empirical, analytical, and field test methods are often used for the study and design of the coal pillar size. Carr et al comparatively tested pillars with different widths in three mines to determine a rational coal pillar width. Salomon presented an empirical design criterion and calculation for the pillar width to improve mineral excavations by pillar mining while assuming a roof control, which is widely used in analyses of the coal pillar strength and size. Mishra and Tang studied the stability of pillars using the displacement-discontinuity method and found that the pillar stresses increase and the safety factors of the pillars decrease during the caving process. However, when the caved zone was completely consolidated, both the stresses and safety factors did not change during the remaining extraction. Medhurst and Brown presented the relationship between the W-H ratio and coal strength and verified it using a uniaxial test. Yu et al analyzed the effects of geological factors, such as fissures, joints, and faults, on the stress and deformation of pillars using in situ measurements and pointed out that the deformation of the pillar increases and the load capacity decreases under weak geological ground conditions. Newman determined the pre- and postfailure moduli of the coal masses in pillars using field measurements and studied the relationship between the pillar stress and strain.

The size, shape, and strength of coal pillars; occurrence conditions; and in situ stress are not comprehensively considered in empirical and analytical methods that are used for the design of coal pillars. However, numerical simulations can meet the above-mentioned requirements. Esterhuizen et al developed a method of estimating the pillar strength and selecting a safety factor for design based on observations of stable and failed pillars by numerical models. Tewari et al studied the width and stability of coal pillars in the inclined coal seam of the Chasnalla Colliery of Indian Iron and Steel Company of the Steel Authority of India using numerical simulations. Jiang et al evaluated and modified the strength of the fractured rock mass and used the numerical software FLAC3D to study the deformation and failure characteristics of the surrounding rocks and the pillar sizes under the geological conditions of fractured rock masses. Bai et al simulated and investigated the damage characteristics of the GSE under hard roof conditions in the field and optimized the pillar width. Basarir et al predicted and analyzed the mining pressure load on the pillar and GSE using a three-dimensional

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**FIGURE 1** Three typical longwall mining schemes: (A) wide-pillar mining; (B) narrow-pillar mining; and (C) no-pillar mining
finite-difference technique. Kumar et al.\textsuperscript{28} studied the stability of coal pillars under extreme abutment stress in a 900 m deep mine using field tests and numerical simulations. Walton et al.\textsuperscript{29} used in situ back-analysis to determine numerical analysis parameters and substituted them into numerical simulations to study the coal pillar stability.

However, the above-mentioned studies on the width and stability of coal pillars mostly focused on medium and normal coal seams. There is a lack of research on the reasonable width of coal pillars to ensure the safety and stability of the GSE during LMTCCM in extra-thick coal seams. In view of the above-mentioned problems, a typical MDT mine in the context of the LMTCCM in 14-m extra-thick seam was chosen as the study background. The field investigations showed that the deformation and damage of the 5209 roadway with a 30-m wide pillar were serious during mining and a large amount of coal resources were wasted, indicating that the wide pillar is not suitable for LMTCCM in extra-thick coal seams (Section 2). Thus, the abutment stresses of coal seams with different thicknesses were compared and studied to determine the relationship between the width of the coal pillar and stress environment of the rock surrounding the GSE in Sections 3 and 4. Furthermore, the rational width of coal pillars for extra-thick coal seams was determined to guarantee the stability and safety of the GSE.

![Generalized stratigraphic column at the study site](image_url)

| Thickness (m) | Depth (m) | Lithology          |
|--------------|-----------|--------------------|
| 14.7         | 380.3     | Medium-fine sandstone |
| 7.3          | 395.0     | Coarse sandstone   |
| 1.1          | 402.3     | Mudstone           |
| 2.0          | 403.4     | Coal seam 2        |
| 3.3          | 405.4     | Mudstone           |
| 5.2          | 408.7     | Coal seam 3        |
| 1.9          | 413.9     | Mudstone           |
| 13.8         | 415.8     | Coal seam 5        |
| 5.4          | 429.6     | Mudstone           |
| 14.5         | 435.0     | Medium-fine sandstone |

![Layouts of the N5209, N5210, and N5211 panel](image_url)
CASE STUDY

2.1 Mining and geological conditions

The Madaotou (MDT) coal mine is in Datong City, Shanxi Province, China. The coal seam 5 is being mined at present. Its average thickness and buried depth are 13.8 and 415 m, respectively, and the dip angle is close to zero. The roof and floor of the coal seam 5 both comprise mudstones (Figure 2). The N5209 panel used in LMTCCCM was studied in this paper. The mechanical mining and top coal caving heights are 3.5 and 10.3 m, respectively. The longwall face and strike lengths of the N5209 panel are 150 and 1200 m, respectively. The N5210 panel is between the N5209 and N5211 panels, and the panel retreat has been finished. The coal pillar width between the N5209 and N5210 panels is 30 m, as shown in Figure 3.

The N5209 tailgate section is rectangular and has a width and height of 4 and 5 m, respectively. Bolts with 22 mm diameters were used for the roof and rib supports. The length of the bolts was 3100 mm in the roof and 2400 mm in the rib. The spacing of the bolts installed in the roof and coal ribs was 800 mm × 900 mm and 1000 mm × 900 mm, respectively. The bolts in the ribs were connected by steel meshes and ladder beams, and the bolts in the roof were connected by steel meshes and “W” belts. Cable supports with diameters of 21.8 mm and lengths of 8300 mm were used for the roof support; the spacing was 2100 mm × 900 mm. Figure 4 presents a cross section of the bolt and cable supports.

2.2 Experimental rock mechanical properties

To determine the properties of the surrounding rock masses, rock samples were obtained by drilling into the coal seam 5 and roof strata of the N5209 tailgate and tested in the laboratory. The tests were conducted with a servo-controlled machine (TWA-2000). The tensile strength (σ$_{ti}$), compressive strength (σ$_{ci}$), cohesion (c$_i$), elastic modulus (E$_i$), friction angle (φ$_i$), and Poisson's ratio (ν$_i$) were obtained from uniaxial compression tests. The results are listed in Table 1.

2.3 Field monitoring and observation

2.3.1 The deformation and failure ranges of the surrounding rock

To evaluate the rationality of the coal pillar with 30 m, the deformation and failure ranges in the N5209 tailgate were determined.

Acoustic wave detection was used to measure the failure ranges based on the principle that the longitudinal acoustic wave propagates slowly in the fractured rock but quickly in the intact rock. The sampling frequency was 10 kHz, and the time precision was 0.1 μs. Three monitoring stations were installed in the N5209 tailgate during the initial stage of the N5209 panel retreat. The distances from the first, second, and third monitoring stations to the longwall face were 70, 300, and 600 m, respectively. The acoustic wave detection devices and borehole layouts are shown in Figure 5. The steps of acoustic wave detection can be described as follows: (a) The longitudinal acoustic wave propagation rate in the intact coal determined by laboratory tests is greater than 1.5 km/s; (b) three acoustic detection boreholes were drilled in the roof to a depth of 15 m, and 8 m deep boreholes were drilled in the yield and virgin coal ribs, respectively (Figure 5A); (c) the acoustic wave detection devices were placed in the boreholes, and acoustic waves were emitted and received (Figure 5B); and (d) the velocities of the longitudinal acoustic waves were recorded and studied. The borehole zone with a velocity below 1.5 km/s is damaged; the damage zone ranges are shown in Figure 9.

Forty monitoring stations were installed in the N5209 tailgate to monitor the deformation of the surrounding rocks during the GSE development and N5209 panel retreat. The monitoring steps are as follows: (a) Four pegs were permanently fixed in the roof, floor, and two ribs of the tailgate at every monitoring station; (b) the convergences of the roof, floor, and two ribs of the roadway were measured using tapes and rakes; and (c) a roof subsidence monitoring device was installed in the roof center at every monitoring station. Four monitoring points were installed in the roof. The spacing between adjacent monitoring points was 2 m (Figure 6). The subsidence of the roof was recorded every 2 days.

2.3.2 Results and analyses

Figure 7 shows the deformation and damage of the N5209 tailgate during the panel retreat. Roof subsidence, floor heave, coal pillar convergence, and support damage occurred due to severe mining pressures. The bolt-cable supports could
not guarantee the safety of the production in parts of the tailgate. Therefore, single-prop and wooden-stack auxiliary supports were used. However, single-prop sloping and steel-belt tearing occurred. The deformation and failure of the N5209 tailgate seriously affected the ventilation, transportation, and walking, hindered the efficient production of the panel, and caused potential safety hazards.

A typical roadway section was used to study the deformation and damage, because of the similar deformation during the N5209 tailgate development. As shown in Figure 8A, when the N5209 tailgate was developed, the maximum convergences of the roof and floor were 279 and 46 mm, respectively, which stabilized after 78 days. The maximum convergences of the coal pillar and virgin coal rib were 234 and 82 mm, respectively. Compared with the coal pillar, the deformation of the virgin coal rib is much smaller. As shown in Figure 8B, within 100 m of the longwall face, the deformation of the roadway decreases with increasing distance to the longwall face during the N5201 panel retreat. The closer the monitoring stations are to the longwall face, the greater is the deformation of the N5209 tailgate during the panel retreat. The maximum convergences of the roof, floor, coal pillar rib, and virgin coal rib were 562, 310, 490, and 268 mm, respectively. These data show that the deformation of the N5209 tailgate is large and the stability is poor during the N5209 panel retreat.

As shown in Figure 9, the plastic failure depths of the roof, coal pillar rib, and virgin coal rib are 12.3, 7.8, and 6.1 m at the monitoring station A; 10.2, 6.2, and 4.8 m at the monitoring station B; and 10.8, 5.8, and 4.1 m at the monitoring station C, respectively.

The rocks surrounding station A are influenced by severe mining pressures such that the failure ranges are greater than those of stations B and C. Stations B and C are far away from the mining disturbance, and the failure ranges of station C are close to those of station B. The failure depth of the virgin coal rib is smaller than that of the coal pillar due to the great abutment stress on the 30-m coal pillar. Because the top coal in the roof is thick and weak, it can be easily separated from the overlying strata, and thus, the damage range of the roof is larger than that of the two ribs of the N5209 tailgate. The mining pressures cause the rock masses to separate and slip, and the fractures in the coal and rock masses are deep. Therefore, the failure ranges exceed the anchorage lengths of the bolts, which lead to the support failure of the bolts and cables.

In summary, the deformation and failure of the rock surrounding the N5209 tailgate with a yield coal pillar of 30 m

| Rock strata       | Lithology       | \(\sigma_{ti}\) (MPa) | \(\sigma_{ei}\) (MPa) | \(c_{i}\) (MPa) | \(E_{i}\) (GPa) | \(\varphi_{i}\) (deg.) | \(v_{i}\) |
|-------------------|-----------------|-----------------------|-----------------------|-----------------|-----------------|------------------------|---------|
| Roof              | Medium-fine sandstone | 2.93                | 83.4                  | 10.6            | 16.1            | 44                     | 0.23    |
| Coarse sandstone  |                 | 3.62                  | 113.4                 | 16.5            | 21.3            | 50                     | 0.21    |
| Mudstone          |                 | 2.14                  | 40.7                  | 6.4             | 5.4             | 30                     | 0.30    |
| Coal seam 2       |                 | 1.94                  | 38.1                  | 4.6             | 3.6             | 29                     | 0.30    |
| Mudstone          |                 | 2.52                  | 48.3                  | 6.9             | 6.9             | 32                     | 0.28    |
| Coal seam 3       |                 | 1.78                  | 37.5                  | 4.1             | 3.2             | 30                     | 0.31    |
| Mudstone          |                 | 2.47                  | 42.2                  | 6.4             | 4.9             | 31                     | 0.28    |
| Mining coal seam  | Coal seam 5     | 1.26                  | 33.8                  | 3.5             | 3.1             | 27                     | 0.30    |

**TABLE 1** Rock mechanical properties

**FIGURE 5** Failure range determination in the surrounding rock: (A) boreholes for acoustic wave detection; and (B) acoustic wave detection diagram
are severe during the N5209 panel treat. This seriously influences the ventilation and transportation, and there are risks of roof falling and rib spalling, which restrict the safe and efficient production. Thus, a 30-m coal pillar cannot guarantee the safe production of the N5209 panel in LMTCCM in extra-thick coal seams.

3 | SIMULATION MODEL

3.1 | Global model for the pillar width

Global models were built using the numerical simulation software FLAC3D. The principles of the plastic failure range and abutment stress were compared and studied to explore the relationship between the coal pillar width (8, 20, and 30 m) and the GSE stability. Finally, the rational width of the coal pillar for LMTCCM in extra-thick coal seams (14 m) was determined. The model size was 680 m × 300 m × 68 m. Considering the computational efficiency, the mesh between the N5209 and N5210 panels was refined (Figure 10). The vertical and horizontal displacements of the model bottom were fixed. The horizontal displacements of the four sides of the model were fixed. Based on the rock bulk density of 0.025 MN/m³, a vertical in situ stress of 10.25 MPa was applied to the top boundary of the model to simulate the overburden pressure. Gravity was applied in the model. The initial horizontal in situ stress was 1.2 times the vertical stress in the x- and y-axis directions.

A constitutive strain-softening model was used for coal seam 5. The Mohr-Coulomb and double-yield constitutive models were used for other strata and the gob gangue, respectively.

The excavation sequences of the roadway and panel are as follows: N5210 panel retreat → N5211 headgate development → N5211 longwall face excavation → N5211 tailgate development → N5211 panel retreat. The steps of the numerical simulation are as follows (Figure 11): (a) Simulation models are built for different coal seam thicknesses (5, 10, and 14 m) to compare and study the relationship between the abutment stress and coal seam thickness; (b) the grids generated in the models. Boundary conditions, constitutive relations, model materials, and initial stresses are applied to the models; (c) the models are run until the equilibrium is reached and the rationalities of the calculated results are analyzed. If they are reasonable, the simulation continues with the next step; otherwise, the model is modified and recalculated; (d) the N5210 panel is retreated and the overburden strata above the N5210 panel gob are calculated until they stabilize; (e) the tailgate, headgate, and longwall face of the
N5209 panel are developed; (f) coal pillars with widths of 8, 20, and 30 m are obtained in coal seams with different thicknesses during the N5209 tailgate development; and (g) the N5209 panel is retreated, and the deformation and plastic failure ranges of the surrounding rock are obtained for different pillar widths and coal seam thicknesses.

The width and height of the N5209 tailgate are 5 and 4 m, respectively. Considering the influences of the bolt-cable supports on the surrounding rock, the supports were realized using bolt-cable structure elements in FLAC3D. The bolt-cable arrangements are shown in Figure 4, and the parameters are listed in Table 2.

**FIGURE 8** Deformation of the rock surrounding the N5209 tailgate: (A) deformation during the development of the N5209 tailgate; and (B) deformation during the N5209 panel retreat

**FIGURE 9** Plastic failure ranges of the surrounding rock: (A) monitoring station A; (B) monitoring station B; and (C) monitoring station C

**FIGURE 10** Simulation model
Two lines monitoring the abutment stress were arranged in the middle of the coal pillar and virgin coal rib to determine the abutment stresses. A line monitoring the subsid-
ence was installed in the roof. Six monitoring points were installed on the surfaces of the roof and floor, respectively, to measure the deformation of the rocks surrounding the N5209 tailgate. Five monitoring points were installed on the surfaces of the virgin coal rib and coal pillar, respectively, as shown in Figure 12.

3.2 | Rock mass parameters

There are lots of fractures and joints in the in situ rock masses, but not in the intact rock samples.31 Medhurst and Brown20 pointed out that the fractures and joints could affect the strength of the rock mass, and the mechanical properties of intact rock samples obtained from laboratory tests are much larger than those of the in situ rock masses. Parameters obtained for the intact rock by laboratory tests cannot directly be used in the numerical simulation.32 Therefore, rock mass strength analysis software (RocLab 10.0) based on the Hoek-Brown failure criterion was applied to determine the parameters of the rock mass:

$$\sigma_1 = \sigma_3 + \sigma_{ci} \left( m_b \frac{\sigma_3}{\sigma_{ci}} + s \right)^a$$  \hspace{1cm} (1)

where $\sigma_{ci}$ is the uniaxial compressive strength of the intact rock; $\sigma_1$ and $\sigma_3$ are the maximum principal stress and minimum principal stress, respectively; and $m_b$, $s$, and $a$ are rock mass constants, which can be calculated as follows:

$$m_b = m_{ci} \exp \left( \frac{GSI - 100}{28 - 14D} \right)$$  \hspace{1cm} (2)

$$s = \exp \left( \frac{GSI - 100}{9 - 3D} \right)$$  \hspace{1cm} (3)

$$a = 0.5 + \frac{1}{6} \left( e^{-\frac{GSI}{15}} - e^{-\frac{20}{3}} \right)$$  \hspace{1cm} (4)

where $m_{ci}$ is the constant of the intact rock, $D$ is the disturbance factor, and GSI is the constant evaluating the fractured rock mass.

The uniaxial strength of the rock mass can be obtained by substituting $\sigma_1 = \sigma_3 = \sigma$ into Equation (1).

| Name   | $E_g$ (GPa) | $C_g$ (N/m) | $K_g$ (N/m²) | $L$ (mm) | $L_g$ (mm) | $D$ (mm) | $F_t$ (N) |
|--------|-------------|-------------|-------------|----------|------------|----------|-----------|
| Roof bolt | 200       | 4.0e5       | 2.2e9       | 3100     | 1800       | 22       | 2.30e5    |
| Rib bolt  | 200       | 4.0e5       | 2.2e9       | 2400     | 1200       | 22       | 2.30e5    |
| Cable    | 200       | 4.0e5       | 2.0e9       | 8300     | 2400       | 21.8     | 5.20e5    |

Note: $E_g$ is the elastic modulus of the grout per unit length, $C_g$ is the cohesion of the grout per unit length, $K_g$ is the stiffness of the grout per unit length, $F_t$ is the tensile strength, $D$ is the diameter, $L$ is the length of the support elements, and $L_g$ is the length of the resin grout.
The deformation modulus $E_m$ can be calculated using the GSI:

$$E_m = \left(1 - \frac{D}{2}\right) \sqrt{\frac{\sigma_{ci}}{100}} \left[\frac{10^{(\text{GSI} - 30)}}{40}\right] \quad \sigma_c < 100 \text{MPa}$$

$$E_m = \left(1 - \frac{D}{2}\right) 10^{\left(\text{GSI} - 30\right)/40} \quad \sigma_c \geq 100 \text{MPa}$$

The shear modulus and bulk modulus can be calculated using the deformation modulus and Poisson’s ratio:

$$K = \frac{E}{3(1-2v)}$$

$$G = \frac{E}{2(1+v)}$$

where $K$ is the bulk modulus, $G$ is the shear modulus, and $v$ is the Poisson’s ratio.

Because the rock mass is largely disturbed during the mining process, the $D$ value is 0.8. The GSI and $m_i$ are obtained from a table provided by Hoek and Brown. The “tunnels” application was selected, and its depth is 415 m. The parameters of the rock mass obtained using the RocLab (10.0) software are shown in Table 3.

### 3.3 Double-yield model

#### 3.3.1 Double-yield model parameters

The caved gangue rocks in the gob support the overlying strata and reduce the abutment stresses on the coal pillars. Therefore, it is necessary to consider the support force of gangue rocks. At present, the numerical simulation of the mechanical properties of cave gangue rocks in the gob is mainly achieved by the double-yield constitutive model. Wang and Li pointed out the simulation results of double yield were consistent with the analytical model of the Salamon constitution and Terzaghi constitution, which well reflected the loading mechanism of the caved gangue rocks in the gob. Jiang et al. studied the compression characteristics of caved gangue rocks in the gob using double-yield constitution, and the plastic zone was compared by the simulation results and the limiting equilibrium theory to verify the reliability and advantage of double yield. Thus, the double-yield model can be used to realistically study the characteristics of the caved rocks in the gob.

The cap pressure presents the load-bearing capacity of the caved rock in the double-yield model, which is achieved by using the Table commands in the FLAC3D simulation software. The cap pressure can be expressed as follows:

$$\sigma = \frac{E_0\epsilon}{1-\left(\epsilon/\epsilon_{\text{max}}\right)}$$

where $\sigma$ is the cap pressure, $\epsilon$ is the bulk strain of the coal and rock mass, $\epsilon_{\text{max}}$ is the maximum bulk strain, $E_0$ is the initial elastic modulus, and $\epsilon_{\text{max}}$ and $E_0$ can be estimated as follows:

$$\epsilon_{\text{max}} = \frac{(b-1)}{b}$$

$$E_0 = 10.39\sigma_c^{0.042}/b^{7.7}$$

where $\sigma_c$ is the strength of the caved rocks, and $b$ is the expansion coefficient of the caved rocks in the gob, which can be calculated as follows:

$$b = \frac{(h_m+h_c)}{h_c}$$

where $h_m$ is the mining height, and $h_c$ is the caved height of the strata overlying the gob.

**FIGURE 12** Abutment stress and deformation monitoring
Based on the field investigations, the caved height of the strata overlying the gob is ~2-4 times the mining height. The mining height in the MDT coal mine is 14 m; thus, the caved height of the overlying strata is ~28.9 m, and the expansion coefficient of the caved rocks in the gob is 1.36. The uniaxial compression strength of the caved rocks in the gob obtained by laboratory tests is 17.3 MPa. The maximum bulk strain and initial elastic modulus were determined to be 0.265 and 18.9 MPa, respectively. The cap pressures used in the double‐yield model are shown in Table 4.

The parameters used in the double‐yield model were obtained using trial and error. A cube rock model was generated in the FLAC3D software. A constant velocity was applied to the top boundary of the model, and the horizontal and vertical displacements of a bottom boundary were fixed. The parameters used in the double‐yield model were adjusted to fit the strain‐stress curves of Salamon’s model (Figure 13). The parameters obtained from the simulation are shown in Table 5.

### Table 3

| Rock strata | Lithology          | GSI | D  | K (GPa) | G (GPa) | φ (deg.) | c (MPa) | σt (MPa) | v   |
|-------------|--------------------|-----|----|---------|---------|----------|---------|----------|-----|
| Roof        | Medium‐fine sandstone | 66  | 0.75 | 14.3 | 9.53 | 2.52 | 0.75 | 14.3 | 0.46 |
|             | Coarse sandstone   | 70  | 0.75 | 19.8 | 12.22 | 2.00 | 0.98 | 22.0 | 0.36 |
|             | Mudstone           | 42  | 0.75 | 8.05 | 8.05 | 2.58 | 0.58 | 8.22 | 0.20 |
|             | Coal seam 2        | 44  | 0.75 | 2.75 | 2.75 | 1.04 | 0.03 | 22.0 | 0.03 |
|             | Coal seam 3        | 47  | 0.75 | 2.75 | 2.75 | 1.04 | 0.03 | 22.0 | 0.03 |
|             | Coal seam 5        | 43  | 0.75 | 2.75 | 2.75 | 1.04 | 0.03 | 22.0 | 0.03 |
| Mining seam | Coal seam 5        | 43  | 0.75 | 2.75 | 2.75 | 1.04 | 0.03 | 22.0 | 0.03 |
| Floor       | Medium‐fine sandstone | 66  | 0.75 | 14.3 | 9.53 | 2.52 | 0.75 | 14.3 | 0.46 |

Note: σt = tensile strength; c = cohesion; φ = friction angle.

### Table 4

| Strain | Stress (MPa) | Strain | Stress (MPa) |
|--------|--------------|--------|--------------|
| 0.02   | 0.41         | 0.14   | 5.61         |
| 0.04   | 0.89         | 0.16   | 7.63         |
| 0.06   | 1.46         | 0.18   | 10.60        |
| 0.08   | 2.16         | 0.2    | 15.41        |
| 0.10   | 3.03         | 0.22   | 24.48        |
| 0.12   | 4.14         | 0.24   | 48.08        |

### Figure 13

Stress‐strain curves based on the simulation and Salamon’s model

Based on the field investigations, the caved height of the strata overlying the gob is ~2-4 times the mining height. The mining height in the MDT coal mine is 14 m; thus, the caved height of the overlying strata is ~28.9 m, and the expansion coefficient of the caved rocks in the gob is 1.36. The uniaxial compression strength of the caved rocks in the gob obtained by laboratory tests is 17.3 MPa. The maximum bulk strain and initial elastic modulus were determined to be 0.265 and 18.9 MPa, respectively. The cap pressures used in the double‐yield model are shown in Table 4.

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### 3.3.2 Double‐yield model validation

To validate the rationality of the double‐yield model used in the gob, the vertical stresses in the middle section of the coal...
seam (z = 22 m) were obtained after the N5210 panel retreat. The results are shown in Figure 14.

The caved rocks filled the gob after the N5210 panel retreated. The caved rocks in the middle of the gob were compacted first. Based on extensive field practices and research, Campoli et al. reported that the vertical stresses increase from the edge to the center of the gob and are close to the initial in situ vertical stress. The vertical stress at the edge of the gob is close to 0 MPa, while it is close to the initial in situ vertical stress. The vertical stress increases from the edge to the center of the gob and are consistent with the results of other research and that the double‐yield model indicates that the double‐yield model is consistent with the results of other research and that the double‐yield model parameters listed in Tables 4 and 5 are reasonable and can be used for the gob.

### 3.4 Strain-softening model

#### 3.4.1 Strain-softening model construction

When the stresses reach the peak, with increasing strain, the uniaxial compression strength of the rock rapidly drops to the low value. This phenomenon is called “strain-softening.” The constitutive strain-softening model assumes that the properties of the rock mass (cohesion, friction angle, and dilation angle) change with the strain when the rock mass is damaged and remain constant after reaching the residual stage.

The plastic shear strain parameter $\varepsilon_p$ can be used to describe the plastic deformation of the model in the plastic stage:

$$\varepsilon_p = \frac{1}{6} \sqrt{\frac{2}{2}} \left( \varepsilon_1^p - \varepsilon_3^p \right)^2 + \left( \varepsilon_1^p + \varepsilon_3^p \right)^2 + \left( \varepsilon_1^p - \varepsilon_2^p \right)^2$$  \hspace{1cm} (14)

where $\varepsilon_1^p$ is the maximum plastic strain, and $\varepsilon_3^p$ is the minimum plastic strain.

The strain-softening model used in the FLAC3D software is based on the Mohr-Coulomb yield criterion:

$$\sigma_1 = \frac{1 + \sin \varphi (\varepsilon_p)}{1 - \cos \varphi (\varepsilon_p)} \sigma_3 + \frac{2c (\varepsilon_p) \cos \varphi (\varepsilon_p)}{1 - \sin \varphi (\varepsilon_p)}$$  \hspace{1cm} (15)

where $\sigma_1$ is the maximum principal stress; $\sigma_3$ is the minimum principal stress; and $\varphi (\varepsilon_p)$ and $c (\varepsilon_p)$ are the strain functions of the friction angle and cohesion after the plastic failure of rock, respectively, which can be expressed as follows:

$$c (\varepsilon_p) = \begin{cases} c_p & \varepsilon_p \leq \varepsilon_p^\text{ps} \\ \frac{\varepsilon_r - \varepsilon_p}{\varepsilon_r - \varepsilon_p^\text{ps}} (\varepsilon_p^\text{ps} - \varepsilon_p^\text{ps}) + c_p & \varepsilon_p^\text{ps} < \varepsilon_p < \varepsilon_r \end{cases}$$  \hspace{1cm} (16)

$$\varphi (\varepsilon_p) = \begin{cases} \varphi_p & \varepsilon_p \leq \varepsilon_p^\text{ps} \\ \frac{\varphi_r - \varphi_p}{\varphi_r - \varphi_p^\text{ps}} (\varepsilon_p^\text{ps} - \varepsilon_p^\text{ps}) + \varphi_p & \varepsilon_p^\text{ps} < \varepsilon_p < \varepsilon_r \end{cases}$$  \hspace{1cm} (17)

where $\varepsilon_p$ is the plastic strain at the peak stress of the coal and rock; $\varepsilon_p^\text{ps}$ is the plastic strain in the residual stage of the coal and rock; $c_p$ is the cohesion before the plastic failure of the coal and rock; $\varphi_p$ is the friction angle of the coal and rock before the plastic failure of the coal and rock; and $c_r$ and $\varphi_r$ are the cohesion and the friction angle of the coal and rock in the residual stage, respectively (Figure 15).

The strain ($\varepsilon_p$) at the peak stress of the rock and coal, plastic strain ($\varepsilon_p$) at the residual stress, and cohesion ($c_r$) and friction angle ($\varphi_r$) were determined by laboratory tests in this study (Table 6).

The plastic failure ranges based on the Mohr-Coulomb criterion and strain-softening were compared using the FLAC3D software. The results show that the plastic failure ranges based on the strain softening are greater than those based on the Mohr-Coulomb criterion (Figure 16). Thus, the strain-softening model more realistically describes the load-bearing capacity of coal and rock masses after failure.

| TABLE 5 Material parameters used in the double‐yield model |
|-------------|-------------|-------------|-------------|
| Parameter   | $\gamma$ (kg/m$^3$) | K (GPa)    | $\varphi$ (°) |
| Value       | 1100        | 5.35       | 28           |

Note: $\gamma$ = density, K = bulk modulus, $G$ = shear modulus, $\varphi$ = friction angle, $\sigma_1$ = tensile strength.
3.4.2 Validation of the strain-softening model

To validate the rationality of strain-softening model, a standard cylindrical numerical sample with a height of 100 mm and diameter of 50 mm was created and a constant velocity was applied to the top boundary of the sample. The stress-strain curve obtained from the simulation agrees well with that based on the laboratory tests (Figure 17). Simultaneously, numerical and laboratory specimens are both shear-failure along 45 degrees.

3.5 Global model validation

The surrounding rock convergence and roof subsidence data obtained from the field measurements and simulations (70 m from the longwall face of the N5209 panel) were compared to validate the correctness of the global model.

As shown in Figure 18A, the surrounding rock convergences obtained by the field measurements and simulations show similar trends during the N5209 panel retreat. They decrease with increasing distance to the longwall face of the N5209 panel. The subsidence of the roof surface is 251 and 275 mm based on the simulations and field measurements, respectively. The simulation and field measurement results for the roof subsidence at a monitoring point 6 m from the roof surface are 126 and 134 mm, respectively (Figure 18B). Because FLAC3D software takes rock masses as the continuum medium, in actually, the deformation of surrounding rock could produce fissures and separations, so the deformations of surrounding rock measured in field are larger than those of numerical simulation. However, the numerical results are very close to the field measurement results, and the error between numerical results and in situ measurements is less than 2%. Therefore, the simulation results are rational and reliable, it could well study and guide field practice.

4 Discussion and analyses of the simulation results

4.1 Failure analyses of the coal masses in the extra-thick coal seam

Figure 19 shows the stress-strain curves of the coal mass obtained by simulations in the center (B monitoring point) and rib (A monitoring point) of the 30-m coal pillar. The vertical stress of the coal in the center and rib of the coal pillar reaches the maximum at a strain of 0.007. When the strain is greater than 0.007, the coal is damaged. The stress-strain relationship is similar to that obtained from the laboratory tests (Figure 17). The maximum load-bearing stress of the coal mass is 23.3 MPa in the coal pillar center and 12.4 MPa in the coal pillar rib. The reasons for the greater load-bearing capacity in the center of the coal pillar compared with that of the rib are illustrated in Figure 20.
As shown in Figure 20, the $\sigma_{v1}$, $\sigma_{v2}$, and $\sigma_{v3}$ present vertical stresses, and the $\sigma_{h1}$, $\sigma_{h2}$, and $\sigma_{h3}$ denote horizontal stresses applied to coal in different position. The coal mass A at the position of the virgin coal rib is only affected by vertical stress $\sigma_{v3}$, not by horizontal stress. The coal masses B and C deep in the coal seam are affected by both vertical and horizontal stresses. Wang et al.\(^{48}\) reported that the horizontal stress applied to the coal gradually increases with increasing distance from the virgin coal rib ($\sigma_{h1} > \sigma_{h2}$) and the vertical stress applied to the coal is abutment stress. Baumgarten\(^{49}\) showed that the triaxial compressive strength of the rock is greater than the uniaxial compressive strength and that the compressive strength of the rock increases with increasing horizontal confining pressure. With respect to the stress-strain principles in Figure 19, only vertical stress acts on the coal in the coal pillar rib. However, the coal in the coal pillar center is not only subjected to vertical stress but also to horizontal stress, and thus, its load-bearing capacity is greater. Although the abutment stress of the extra-thick coal seam has a greater peak value than that of the medium and normal coal seams, the failure ranges of coal were affected by vertical and horizontal stresses. Thus, it is necessary to consider the abutment stress and horizontal confining pressure to study the failure ranges of coal.

### 4.2 Abutment stress and coal failure range

To realize the mining pressure principles and determine the reasonable coal pillar width for extra-thick coal seams, three simulation models with a coal seam thickness of 5, 10, and 14 m were built using the FLAC3D software and three types of coal pillars (8, 20, and 30 m) were constructed for coal seams with different thicknesses. Furthermore, the abutment stresses and plastic failure ranges were studied 70 m from the longwall face during the N5209 tailgate retreat, as shown in Figure 21.

The maximum abutment stresses of the coal pillars with a width of 30 m (Figure 21A) in the 5-, 10-, and 14-m coal seams are 36.2, 31.9, and 31.2 MPa, respectively, which are larger than those of the virgin coal ribs (20.7, 19.8, and 23.8 MPa). In the 5-m coal seam, the abutment stress of the coal pillar has a “double-peak” shape. The maximum abutment stresses of the coal pillars with a width of 20 m (Figure 21B) in the 5-, 10-, and 14-m coal seams are still greater than those of the virgin coal ribs. However, when the width of pillar is 8 m (Figure 21C), the maximum abutment stresses of the coal pillars in 5-, 10-, and 14-m coal seams are 19.8, 18.4, and 16.6 MPa, that is, much lower than those of the virgin coal ribs (25.8, 29.8, and 30.9 MPa). Based on the...
above-mentioned data and analyses, the abutment stress peak increases with increasing coal seam thickness and gradually moves away from that of the virgin coal ribs, except for the coal pillar width of 30 m in the 5-m coal seam. Because the abutment stress of the coal pillar with a width of 30 m in the 5-m coal seam has a “double peak,” which is lower close to the GSE, the abutment stress of the virgin coal rib is greater than that of the 10-m coal seam. On the other hand, with decreasing coal pillar width, the abutment stresses of the coal pillars gradually decrease and shift from the coal pillars to the virgin coal ribs.

The plastic failure depths of the virgin coal rib, coal pillar, and roof of the coal pillar with a width of 30 m (Figure 21A) are 4, 5, and 7 m in the 5-m coal seam, and the width of rectangular elastic zone in the coal pillar center is 20 m. The plastic failure depths of the virgin coal rib and roof are 5 and 8 m in the 10-m coal seam and 5 and 10 m in the 14-m coal seam, respectively. There are trapezoidal and triangular elastic zones at the bottom of the coal pillars in the 10- and 14-m coal seams, respectively. When the width of the coal pillar is 20 m (Figure 21B), the plastic failure depths of the virgin coal rib, coal pillar, and roof are 5, 7, and 8 m in the 5-m coal seam, respectively, and the width of the rectangular elastic zone in the coal pillar center is 6 m. The plastic failure zone in the 10-m coal seam is the same as that in the 14-m coal seam. When the width of the coal pillar is 8 m (Figure 21C), the failure ranges of the virgin coal rib are the same for the three types of coal seams. The coal pillars are completely damaged. However, the failure depth of the roof in the 14-m coal seam is 11 m, that is, greater than that in the 5-m coal seam. Based on these analyses and the stress-strain mechanisms of the coal masses in Section 4.1 (Figures 19 and 20), it can be concluded that the load-bearing capacities of the coal masses at the depths of the coal seams are great, although the abutment stress peaks of the virgin coal rib in the extra-thick coal seams are greater and shift to the depth of the coal seam. Therefore, the influence of high abutment stress on the failure range of virgin coal ribs is smaller. The horizontal confining pressure of the coal masses in the center of the coal pillar and the resistance to deformation and failure increase with increasing coal pillar width. The size of the elastic zone of the coal pillar therefore increases with increasing coal pillar width. Because the roof of the GSE is prone to subsidence and separation damage under the high abutment pressure, the plastic failure ranges of the roof are greater than those of the virgin coal rib and coal pillar.

4.3 | Deformation of the surrounding rock

In 14-m coal seam, the convergence of surrounding rock is compared and studied at different widths of the coal pillar (8, 20, and 30 m) at the location 70 m from the longwall face (Figure 22). When the coal pillar width is 8 m, the vertical displacements of the roof and floor and the horizontal convergence of the coal pillar are 150, 22, and 147 mm, respectively, that is, smaller than those of the 20- and 30-m coal pillars. The virgin coal rib convergences of the three types of coal pillars are similar. They are relatively smaller than the convergences of the coal pillar ribs. When the width of coal pillar is 20 and 30 m, the abutment stresses on the coal pillar are greater than those on the virgin coal ribs. The coal masses in the coal pillars are too weak to resist the deformation under the high abutment stresses; therefore, the convergences of the coal pillar are greater than those of the virgin coal rib (Figure 22C,D). The wide pillar causes the GSE to be in the high-stress zone in extra-thick coal seams; thus, the convergence of the surrounding rock is greater than that of the 8-m coal pillar.
4.4 | Determination of an optimal coal pillar width

The cognition that increasing the pillar width is beneficial to the stability of the GSE is one-sided. Even if the coal pillars are wide enough, their load-bearing capacity is much smaller than that of the virgin coal. In addition, the wide pillar will inevitably lead to a large amount of coal resources waste.50

The pressure principles of the extra-thick coal seam (14 m) in the MDT coal mine include wide influence ranges and high-peak stresses. When the pillar widths are 20 and 30 m, the GSE is still located in the high-stress zone; the deformation and failure of the roadway are greater. When the pillar width is 8 m, the coal pillar has a load-bearing capacity, although it experienced complete plastic failure. It is known that the abutment stress of the coal pillar decreases with increasing coal seam thickness (Section 4.2). Therefore, narrow-pillars are more suitable for extra-thick coal seams. When a narrow-pillar is used in the GSE, the coal pillar and GSE are both in the stress-relaxed zone and the high abutment stress shift to the depths of the virgin coal rib. In addition, the coal masses at the depths of the virgin coal rib are under multidirectional forces and their load-bearing capacity and deformation resistance are great (Section 4.1). Thus, the 8-m coal pillar guarantees the GSE stability.

The mining length of the N5209 panel is 1200 m, and the bulk density of the coal is 1.35 t/m³. If the existing 30-m coal pillar is changed to 8 m, 498.96 kt of coal resources will be recovered. Thus, the 8-m coal pillar is a reasonable, economical, and safe choice, which could not only isolate the caved gangue and harmful gases in the gob but also ensure the stability of the GSE and thus satisfy the ventilation, transportation, and safe production requirements.
5 | FIELD TESTS

To validate the rationality and feasibility of the coal pillar of 8 m for LMTCCM in extra-thick coal seams, field tests were carried out on the tailgate of the N5211 panel, which had the same mining and geological conditions as the N5209 panel. The longwall face and strike length of the N5211 panel are 150 and 1000 m, respectively. The N5211 panel is north of the N5210 panel (Figure 3).

Forty monitoring stations were installed to measure the deformation of the surrounding rock and failure of the N5211 tailgate every 2 days during the tailgate development and retreat of the N5211 panel.

As shown in Figure 23, the N5211 tailgate stabilizes after 80 days during development. The maximum vertical and horizontal convergences of the N5211 tailgate are 215 and 181 mm, respectively, and the average convergence rates of the N5211 tailgate (average vertical convergence rate of 2.7 mm/d; average horizontal convergence rate of 2.3 mm/d) are smaller than those of the N5209 tailgate (average vertical convergence rate of 4.3 mm/d; average horizontal convergence rate of 4.1 mm/d). When the N5211 panel retreats, the closer the longwall face is, the greater is the deformation. The maximum convergences of the N5211 tailgate (maximum horizontal convergence of 359 mm; maximum vertical convergence of 461 mm) are smaller than those of the N5209 tailgate (maximum horizontal convergence of 758 mm; maximum vertical convergence of 872 mm). Figure 24 shows the profiles of the N5211 tailgate with an 8-m coal pillar during the N5211 panel retreat. The deformation of the surrounding rocks is small, and the stability of the N5211 tailgate is good, indicating that the 8-m coal pillar is suitable for LMTCCM in extra-thick coal seams.
6 | CONCLUSION

This study provides a field and simulation method to obtain the rational coal pillar width in the context of LMTCCM in extra-thick coal seams. There are three original aspects compared to previous studies: (a) Mining pressures in coal seams with different thicknesses were numerically compared and studied to realize the regularities between the coal seam thicknesses and the mining pressures; (b) the strain-soften and double-yield constitutions were applied to the coal seams and gobs, respectively, and the simulation results were verified against the in situ measurements; (c) the failure mechanisms of the virgin coal and coal pillar ribs were analyzed, and a reasonable width of the coal pillar for LMTCCM in extra-thick coal seam mining was determined. According to the main research results of this paper, the following conclusions were obtained.

The field investigations show the serious deformation and damage of the N5209 tailgate with 30-m coal pillar during the N5209 panel retreat, which indicate that the coal pillar width of 30 m does not meet the GSE stability requirements of LMTCCM in extra-thick coal seams.

The results of the simulations show that the abutment stresses increase with increasing thickness of the coal seams, and the influence ranges of the extra-thick coal seam are wider than those of medium and normal coal seams. When the coal pillar width is 8 m in 14 m extra-thick coal seam, the narrow coal pillar is located at low pressure environment and has enough bearing capacity to keep GSE stability.
Figure 22: Deformation of the N5209 tailgate at the location 70 m from the longwall face of the N5209 panel: (A) vertical displacement of the roof subsidence; (B) vertical displacement of the floor heave; (C) convergence of the virgin coal rib; and (D) convergence of the coal pillar.

Figure 23: Deformation of the N5211 tailgate: (A) deformation of the N5211 tailgate during the development; and (B) deformation of the N5211 tailgate during the panel retreat.
The field tests demonstrate that the 8 m narrow coal pillar can guarantee the ventilation and transportation and is conducive to the safe and efficient production. The ideas and methods used in this study provide effective references for the determination of the coal pillar width in LMTCCM in extra-thick coal seams.

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CONFLICT OF INTEREST

The authors confirm that these article contents have no conflict of interest.

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