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Characteristics of Roof Ground Subsidence While Applying a Continuous Excavation Continuous Backfill Method in Longwall Mining

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Abstract: Activities of traditional longwall mining will result in ground subsidence and therefore cause issues such as damages to buildings and farmlands, water pollution and loss, and potential ecological and environmental problems in the mining region. With advantages of the longwall backfill mining method, as well as the room and pillar mining method, a continuous excavation and continuous backfill (CECB) method in longwall mining is recommended to effectively control the ground subsidence. In this method, mining roadways (MRs) are initially planned in a panel, and then they are excavated and backfilled in several stages until the whole panel is mined out and backfilled. According to the geologic conditions of an underground coal mine, and the elastic foundation beam theory, a mechanical model was built to study the subsidence of the roof while using this new mining method. In addition, methods to calculate roof subsidence in various stages in CECB were also provided. The mechanical parameters of backfilling materials, which were used in the theoretical calculation and the numerical analysis for mutual check, were defined through analyzing the stability conditions of the coal pillars and the filling bodies. The control effect for the ground subsidence of using the newly proposed mining method was analyzed based on both simulation results and site monitoring results, including the ground subsidence, horizontal displacement, tilt, curvature and horizontal strain. This research could provide suggestions to effectively control ground subsidence for a mine site with similar geologic conditions.

Keywords: continuous excavation and continuous backfill; elastic foundation; foundation coefficient; stability of the coal pillar; ground subsidence

1. Introduction

The problem of ground subsidence induced by traditional underground coal mining could cause serious ecological and social issues [1–3]. Due to insufficient support, the overlying strata could sink into the goaf caused by their own weight during the mining process or in a period of time after mining [4]. This subsidence could extend up to the ground without any effective preventative measures, and therefore cause ground subsidence in an area larger than the mining range [5]. Since the ground subsidence due to underground mining could lead to buildings and farmlands being damaged [6–8], as well as water pollution and loss [9–11], it has been a worldwide problem and has generated widespread concerns in realms such as mining and environmental protection [5,12,13].

In order to mitigate the negative effects on surface buildings, farmlands and water bodies from coal mining, mining methods such as partial mining and backfill mining were used to control ground subsidence. The partial mining methods, which include strip mining and room and pillar mining, are

employed to mine out part of a coal seam and leave the rest as various pillars to support the roof. However, these methods may cause problems recovering coal pillars, and the coal mining rates of this kind of methods are typically less than 40% [14,15]. Besides, the failure of the coal pillars left during or after mining may cause security issues and ground subsidence [16–20]. The backfill mining method uses filling bodies to replace coal pillars to support the roof to control the movement and deformation of overlying strata, and therefore to reduce the ground subsidence [21–23]. Currently, the backfill mining method has been widely employed in longwall mining; however, it is still experiencing limits in applications. Since the overlying strata behind the working face would start collapsing and sinking into the goaf in a short time period, it is difficult for the filling bodies to get the designed strength. This could result in failing to support the roof on time, and therefore lead to the subsidence and deformation of the overlying strata [24–26]. In addition, coal mining and backfilling systems are arranged in one limited underground space, and it would cause the mutual effects of these two systems [27]. Thus, combining advantages of the longwall backfill mining method as well as the room and pillar mining method, a continuous excavation and continuous backfill (CECB) method in longwall mining is proposed [27,28]. As seen in Figure 1, this proposed method plans transportation roadways in a panel as with the traditional longwall mining, while achieves the coal mining objective in the manner of roadway excavation. The excavation of one mining roadway (MR) in the CECB method is equal to one cut of a shearer in conventional longwall mining, and the length and width of the MR is equal to the length of the mining face and the distance a shearer penetrates to the coal seam, respectively [28,29]. This mining method has smaller exposure area on the roadway roof; moreover, it has the coal wall and the filling body to support two ribs of the roadway. Therefore, using this method could reduce the movement and deformation of the roof, and efficiently control the subsidence of the ground. In addition, the excavated space is backfilled at the same time, so that the mining and backfilling can be effectively operated in parallel without mutual effects.

Figure 1. The comparison between traditional longwall backfill mining method and the continuous excavation and continuous backfill (CECB) method.

In order to reduce the cost of backfill, the solid wastes in the mining region, such as the gangue and the coal ash, were usually used as filling aggregates. Backfilling materials were formulated by using the recipe of mixing with these aggregates, cementing materials and water. In addition, with the requirements on the pipeline transportation, as well as the ground subsidence control in field applications, laboratory tests were conducted to measure properties of the backfilling material, such as the solidification rate, fluidity and strength [30]. In the CECB method, there are three different stages of supporting mechanisms, that of: changing from coal pillars support, to the combination of coal

pillars and filling bodies support, to the filling bodies support [31]. Thus, stability analyses on the coal pillars as well as the filling bodies should involve their load status and mutual effects in different stages. Former studies indicated that filling bodies with high strength could improve the stability of the coal pillars [32,33]. However, these studies mainly focused on the room and pillar method, and they failed to analyze the stability of the filling bodies. In the aspect of strata movement and ground subsidence control, Zhang JX et al. [34] developed reasonable mining and backfilling schemes according to the numerical simulation results of stress and plastic zone on the coal pillars in roadway backfill mining, and successfully applied them to control the ground subsidence in the mining area. Li HZ et al. [35] analyzed the characteristics of ground subsidence in super-high water backfilling strip mining and proposed correlations between the strip width and the maximum ground subsidence amount. Zhou N et al. [36] analyzed the supporting pressure during the process of roadway excavation and the law of ground subsidence based on the data monitored on the ground. Liu JW et al. [37] compared the movements and deformations of the overlying strata in intermittent cut-and-fill mining, longwall excavation and continuous filling, and revealed that the intermittent cut-and-fill mining could effectively reduce the damage and sink of the strata. Through studying the damage law of overlying strata in roadway backfill mining, Sun Q et al. [38] found that the filling body could successfully support the overlying load, and the strata remains stable after mining. Yu YH et al. [39] analyzed the stability of coal pillars in roadway backfill mining, and proposed subsidence equations of every stratum in mining. The above-mentioned various coal mining and backfilling methods are similar to the CECB method in the panel plan and mining and backfilling sequences. Thus, their research results can be used as references for the application of the CECB method for ground subsidence control induced by mining. However, most of these studies focused on strata movement and ground subsidence after mining, and had insufficient attention on the movement law of overlying strata while mining and backfilling. Due to the fact that multiple stages are applied in CECB to excavate and backfill a coal seam, the sinking amount of the roof caused from the earlier stage would decide the filling space and therefore impact the subsidence of the roof in the next stage. Thus, study on the roof sinking amount in every coal mining and backfilling stage would help to reveal the mechanism of roof sinking and therefore effectively control ground subsidence.

Thus, this research was conducted at the Wangtaipu coal mine at Jincheng in Shanxi, China, to analyze the subsidence characteristics of the roof and ground in different stages while using the CECB method. In addition, stability analyses were performed on the coal pillars as well as filling bodies. The effect of controlling the ground subsidence through using this coal mining method was also verified according to the results from numerical simulation and the data monitored on site.

2. The CECB Method

2.1. Geologic Conditions

The mining panel was located at the Wangtaipu coal mine at Jincheng in Shanxi, China. As shown in Figure 2, structures, such as buildings, facilities and chimneys, were on the ground above the mining panel. These buildings have complex structure types and large differences in their resistance to deformation. To guarantee the stability of all structures, with advantages of the longwall backfilling method and the room and pillar method, a CECB method is recommended to control the subsidence of the ground induced by mining. Due to the strict requirements of the structures such as high-rise buildings and chimneys on the ground deformation, the ground sinking and deformation above the test panel were monitored along strike and dip directions.

According to ‘the regulation of leaving coal pillar and mining coal of holding under the buildings, water bodies, railways and the main roadway’, the maximum deformation after mining activities must be less than the Building Class I damage rating indicator. It refers to the requirement that the tilt is not more than 3.0 mm/m, the curvature is not more than 0.2 mm/m and the horizontal strain is not more than 2.0 mm/m [40].

Figure 2. Buildings on the ground above the mining panel and arrangements of measuring points.

The dimension of the mining panel included a length of about 300 m, a width of about 160 m, an average thickness of the coal seam of 2.5 m, and an average bury depth of 220 m. There is no large geological tectonism, and the dip angle of the coal seam is $1\sim 2^\circ$, so the thickness of each stratum in the panel is assumed to be constant. According to the drilling data [27–29], the typical stratigraphic column in the mining area is shown in Figure 3. The direct roof of the coal seam was gray limestone which had a UCS value of 101.6 MPa and a BTS value of 4.0 MPa, which was categorized as a hard roof. The direct floor of the coal seam was light gray, aluminous mudstone, which had a UCS value of 28.3 MPa and a BTS value of 1.4 MPa, and which was classified as a soft strength layer. The mining panel had a relatively simple water condition: the main water source in the working face was a limestone aquifer in the roof. The water, which got into the working face primarily through cracks in the roof rock, generated minimal effects on the regular production cycle.

System	Series	Formation	Thickness /m	Columnar	Lithology	Remarks
Quaternary (Q)	Pleistocene (Q ₂)		32.0		Loess	Ground Surface
Neogene (N)	Pliocene (N ₂)		20.0		Red Clay	strata
Permian (P)	Late (P ₂)	Upper Shihezi (P ₂ s)	56.0		Medium Sandstone	
	Early (P ₁)	Lower Shihezi (P ₁ x)	44.0		Sandy Mudstone	
		Shanxi (P ₁ s)	14.0		Mudstone	
Carboniferous (C)	Late (C ₃)	Taiyuan (C ₃ t)	14.0		Sandy Mudstone	
			22.0		Fine Sandstone	
			4.0		Limestone	
			5.0		Sandy Mudstone	
			8.5		Limestone	Roof
			2.5		Coal	Coal Seam
	Middle (C ₂)	Benxi (C ₂ b)	10.0		Bauxitic Mudstone	Floor

Figure 3. The typical stratigraphic column in the mining area.

2.2. Arrangement of the Mining Panel

As shown in Figure 4, corresponding transportation roadways for coal source and filling materials were initially arranged along the strike of the coal seam, and MRs were arranged between two transportation roadways. The width of the MR, which typically was 4 to 6 m, was designed while involving the productivity of the continuous miner and roadway support. The design of its length should consider requirements for the transportation of the coal and the filling materials as well as the ventilation, and it typically should be no more than 150 m [27]. For the convenience of transporting the coal source and the filling materials, and to guarantee the compaction of the filling in the MR, the coal seam dip angle should be fully utilized for the upward oblique mining and the downward oblique filling. Therefore, the transportation roadway, which had a higher level, was used for filling materials' conveyance, while the lower one was used for coal conveyance.

Figure 4. The arrangement of the mining panel in the CECB method in longwall mining.

2.3. Mining Procedures

Figure 5 introduces an example of using a four-stage CECB method. As shown in Figure 5a, MRs in the mining panel were firstly marked from No. 1 to No. n ($n = 16$ in this example), and every four MRs were divided into one group. This first stage was to continuously excavate the first MR in each group; in the meantime, to seal and backfill the former excavated MR. The second stage was to dig the second MR in each group and backfill the former excavated MR simultaneously (see Figure 5b). Following the same procedure, the third (see Figure 5c) and fourth (see Figure 5d) MRs in each group could be mined out and backfilled in stage three and stage four, respectively.

Figure 5. Coal mining procedures of the CECB method. (a) The 1st stage; (b) The 2nd stage; (c) The 3rd stage; (d) The 4th stage.

3. Mechanical Mechanism of Roof Sinking

3.1. Mechanical Model

The shear stress between filling body and coal pillar, or between two filling bodies, is much lower than the bearing pressure, so it can be ignored in stress analysis. Therefore, after each mining and backfilling stage, coal pillars and filling bodies in a same group of MRs were considered as a unit to support the roof, and supports from all groups to the overlying strata were independent. As shown in Figure 6, the Winkler elastic foundation beam theory was employed to calculate the roof-sinking amount while using the CECB method. The coal pillars and filling bodies in each group of MRs could be regarded as a ‘spring’ model in the Winkler theory. Assuming $y_i(x)$ is the roof-sinking function, and Equation (1) demonstrates the roof subsidence formulates at the i^{th} stage:

$$\begin{cases} EI \frac{d^4 y_i}{dx^4} + k_i y_i = q, & (0 \leq x < l) \\ EI \frac{d^4 y_i}{dx^4} + k' y_i = q, & x \leq 0 \end{cases} \quad (1)$$

where EI is the bending stiffness of the roof ($N \cdot m^2$), k_i is the foundation coefficient of the mining panel at the i^{th} stage (N/m^3), k' is the foundation coefficient of the boundary coal pillar (N/m^3) and q is the pressure applied on the roof (N/m^2).

Figure 6. The mechanical model of roof sinking in the first stage of using the CECB method.

With the known roof-sinking function $y_i(x)$ at the i^{th} stage, the corresponding rotation angle, bending moment, and shear force of the roof are as follows:

$$\begin{cases} \theta_i(x) = \frac{dy_i}{dx} \\ M_i(x) = -EI \frac{d^2 y_i}{dx^2} \\ Q_i(x) = -EI \frac{d^3 y_i}{dx^3} \end{cases} \quad (2)$$

where θ_i is the rotation angle at the i^{th} stage (rad), M_i is the bending moment at the i^{th} stage ($N \cdot m$) and Q_i is the shear force at the i^{th} stage (N). At the center part of the mining panel, the rotation angle and shear force of the roof are zero, that is:

$$\theta_i(l) = 0, Q_i(l) = 0 \quad (3)$$

At infinity away from the mining panel, the roof subsidence is the compression of coal seam under the action of in situ rock stress, that is:

$$y_i(-\infty) = \frac{q}{k'} \quad (4)$$

At the boundary of the stope, the roof subsidence, rotation angle, bending moment and shear force calculated from Equations (1) and (2) are equal respectively, i.e.:

$$\begin{cases} y_i(0-) = y_i(0+) \\ \theta_i(0-) = \theta_i(0+) \\ M_i(0-) = M_i(0+) \\ Q_i(0-) = Q_i(0+) \end{cases} \quad (5)$$

Substituting the Equations (3)–(5) into the Equations (1) and (2), then the roof-sinking function $y_i(x)$ can be solved as follows:

$$y_i(x) = \begin{cases} e^{\beta_i x} (A_i \cos \beta_i x + B_i \sin \beta_i x) + e^{-\beta_i x} (C_i \cos \beta_i x + D_i \sin \beta_i x) + \frac{q}{k_i}, & (0 \leq x \leq l) \\ e^{\beta' x} (A' \cos \beta' x + B' \sin \beta' x) + \frac{q}{k'}, & (x \leq 0) \end{cases} \quad (6)$$

The values of parameters A_i, B_i, C_i, D_i, A' and B' are as follows:

$$\begin{cases} A_i = F_i \left(-\frac{\beta_i - \beta'}{\beta_i + \beta'} e^{-2\beta_i l} - \sin 2\beta_i l + \frac{\beta_i + \beta'}{\beta_i - \beta'} \cos 2\beta_i l \right) \\ B_i = F_i \left(-e^{-2\beta_i l} + \frac{\beta_i + \beta'}{\beta_i - \beta'} \sin 2\beta_i l + \cos 2\beta_i l \right) \\ C_i = F_i \left(\frac{\beta_i + \beta'}{\beta_i - \beta'} e^{2\beta_i l} - \frac{\beta_i - \beta'}{\beta_i + \beta'} \cos 2\beta_i l - \sin 2\beta_i l \right) \\ D_i = F_i \left(-e^{2\beta_i l} - \frac{\beta_i - \beta'}{\beta_i + \beta'} \sin 2\beta_i l + \cos 2\beta_i l \right) \\ A' = F_i \left(\frac{\beta_i + \beta'}{\beta_i - \beta'} e^{2\beta_i l} - \frac{\beta_i - \beta'}{\beta_i + \beta'} e^{-2\beta_i l} - 2 \sin 2\beta_i l + \frac{4\beta_i \beta'}{\beta_i^2 - \beta'^2} \cos 2\beta_i l \right) + \frac{q}{k_i} - \frac{q}{k'} \\ B' = F_i \frac{\beta_i^2}{\beta'^2} \left(e^{2\beta_i l} - e^{-2\beta_i l} + 2 \frac{\beta_i^2 + \beta'^2}{\beta_i^2 - \beta'^2} \sin 2\beta_i l \right) \end{cases} \quad (7)$$

where,

$$F_i = \frac{\beta'^2 \left(\frac{q}{k_i} - \frac{q}{k'} \right)}{(\beta_i^2 + \beta'^2) \frac{\beta_i - \beta'}{\beta_i + \beta'} e^{-2\beta_i l} - 2(\beta_i^2 - \beta'^2) \sin 2\beta_i l + 4\beta_i \beta' \cos 2\beta_i l - (\beta_i^2 + \beta'^2) \frac{\beta_i + \beta'}{\beta_i - \beta'} e^{2\beta_i l}} \quad (8)$$

β_i and β' are characteristic coefficients, and $\beta_i = \sqrt[4]{\frac{k_i}{4EI}}, \beta' = \sqrt[4]{\frac{k'}{4EI}}$.

3.2. Elastic Foundation Coefficient

While using the CECB method, the load on the roof (q) and the foundation coefficient of boundary coal pillars (k') outside the mining panel are as follows:

$$q = \gamma H \quad (9)$$

$$k' = \frac{E_c}{h} \quad (10)$$

where, γ is the average bulk density of overlying strata (N/m^3), H is the vertical distance from the surface ground to the roof (m), E_c is the elastic modulus of the coal seam (MPa), h is the mining height (m). According to Equation (4), the roof subsidence function $y_i(x)$ involves the coal seam compression

amount before mining. Thus, with the Equations (8) and (9), the roof subsidence amount $U_i(x)$ at the i^{th} stage can be calculated as follows:

$$U_i(x) = y_i(x) - \frac{q}{k'} = y_i(x) - \frac{\gamma H h}{E_c} \quad (11)$$

Substituting foundation coefficients k_i of the mining panel in every stage into Equation (10) could calculate the corresponding roof-subsidence amount $U_i(x)$.

3.2.1. The Foundation Coefficient in the First Stage

While employing the Winkler elastic foundation beam theory to analyze the roof subsidence in the process of using the CECB, the coal pillar and the filling body in the same group were considered an integral unit to support the roof, just like a spring model in the elastic foundation beam theory. As shown in Figure 7, a group of MRs were numbered as a, b, c and d . In the first stage, the coal pillar in the MR a was excavated and its overlying load was transferred to coal pillars in the MR b, c and d . The roof started sinking and the subsidence amount was U_1 . Applying the backfilling to the MR a at this moment, the filling height was $h - U_1$. Since the roof subsidence had already occurred in the MR a before it was backfilled, the support from the filling body to the roof could be ignored comparing with the coal pillars. As shown in Figure 7a, the foundation consisted of coal pillars in the MR b, c and d . Thus, the foundation coefficient of the mining panel in the first stage was as follows:

$$k_1 = \frac{3E_c}{4h} \quad (12)$$

Figure 7. The compression amount of coal pillars and filling bodies in every stage. (a) The 1st stage; (b) The 2nd stage; (c) The 3rd stage; (d) The 4th stage.

3.2.2. The Foundation Coefficient in the Second Stage

As shown in Figure 7b, in the second stage, the subsidence of the roof was U_2 after the coal pillar in the MR b had been mined out, and the filling height in the MR b was $h - U_2$. Since the overlying load on the filling body in the MR b could be ignored, the foundation in this stage consisted of the filling body in the MR a as well as coal pillars in the MR c and d . Therefore, the following equation shows the foundation coefficient of the mining panel in the second stage:

$$k_2 = \frac{E_f}{4(h - U_1)} + \frac{E_c}{2h} \quad (13)$$

where, E_f is the elastic modulus of the filling body, MPa.

3.2.3. The Foundation Coefficient in the Third Stage

As shown in Figure 7c, the roof subsidence was U_3 in the third stage and the filling height in the MR c was $h - U_3$. The foundation in this stage consisted of the filling bodies in the MR a and b , and the coal pillar in the MR d , and the foundation coefficient was as follows:

$$k_3 = \frac{E_f}{4(h - U_1)} + \frac{E_f}{4(h - U_2)} + \frac{E_c}{4h} \quad (14)$$

3.2.4. The Foundation Coefficient in the Fourth Stage

As shown in Figure 7d, the roof subsidence in the fourth stage was U_4 , and the corresponding filling height was $h - U_4$. The foundation in this stage consisted of the filling bodies in the MR a , b and c . The following equation indicates the foundation coefficient in this stage:

$$k_4 = \frac{E_f}{4(h - U_1)} + \frac{E_f}{4(h - U_2)} + \frac{E_f}{4(h - U_3)} \quad (15)$$

Substituting Equations (11)–(14) into Equation (10), then the corresponding roof subsidence in the four stage U_1 , U_2 , U_3 and U_4 can be calculated.

4. Subsidence of the Roof and the Ground

The elastic modulus of filling materials has significant effects on the subsidence of the roof and the ground while applying the backfill [41]. In the CECB method, the filling body could not bear more loads if it has insufficient elastic modulus. It may cause more load on the coal pillar as well as the filling body, which is in the early stage, and therefore cause them to fail.

To guarantee the stability of the coal pillar and the filling body during the process of mining and backfilling, the strengths of the coal pillar and the filling body must be greater than their overlying loads [42]. Therefore, the value range of the elastic modulus and strength of the filling body can be determined by the stabilities conditions of the coal pillar and the filling body. Furthermore, the subsidence of the roof and the ground can be calculated based on the values of these parameters.

4.1. The Stability of the Coal Pillar

The total overlying load on coal pillars and filling bodies is constant, so the filling body with insufficient elastic modulus could not share more loads, which may cause the instability of the coal pillar for over-bearing loads. From Figure 7c, the coal pillar in the MR d had the peak compression amount U_3 when the third stage has been completed, so the maximum load on the coal pillar can be calculated as follows:

$$\sigma_m = \frac{U_3}{h} E_c \quad (16)$$

where, σ_m is the peak load applied on the coal pillar (MPa). To guarantee the stability of the coal pillar, the maximum stress load σ_m should be no more than its strength $[\sigma_c]$, which can be achieved from the following formula [42]:

$$[\sigma_c] = 6.88w^{0.50}/h^{0.70} \quad (17)$$

where, w is the width of the coal pillar (m).

4.2. The Stability of the Filling Body

The filling body in every stage of using the CECB method in longwall mining was under different load situations. From Figure 7d, the filling body in the MR a was under the maximum compression

which was $U_4 - U_1$ when the fourth stage has been completed; moreover, it was under the peak load as shown in the following equation:

$$\sigma'_m = \frac{U_4 - U_1}{h - U_1} E_f \quad (18)$$

where, σ'_m is the maximum load applied on the filling body (MPa). To maintain the stability of the filling body, the load applied on the filling body σ'_m should be no greater than its strength $[\sigma_f]$.

4.3. Parameters of Filling Materials

In order to optimize the recipe of various filling materials, uniaxial compressive strength (UCS) tests were performed on specimens from seven filling materials which had varied water–solid ratios. Properties of these seven materials, including the strength and the elastic modulus, were estimated. The specimens were cast by pouring the mixed paste filling-material into a standard specimen mold ($\varnothing 50 \text{ mm} \times 100 \text{ mm}$) for adequate compaction until they became solid for UCS tests [30]. As shown in Figure 8, in the first 7 days, the strength of the sample showed a rapid increase with the growth of the solidification time, while the increase rate of the strength revealed significant decrease from the time period that specimens had been cured from 7 days to 14 days. In addition, the increase rate of the strength became constant after 14 days of curing, and their mechanical properties approached stable. The elastic modulus of the filling material could be achieved through analyzing the slope of the stress–strain curve at the elastic deformation stage (typically using the tangent slope at 50% of the peak load) [43]. Table 1 demonstrates the elastic modulus of the filling materials with different water–solid ratios after cured 14 days. Table 2 shows geological and mining parameters of the test panel.

Figure 8. Strengths of filling material specimens with different water–solid ratios and curing times. Reprint with permission [30]; Copyright 2016, Springer.

Table 1. The elastic modulus of the filling materials with different water–solid ratios.

Water–solid ratios	0.7:1	0.8:1	0.9:1	1.0:1	1.1:1	1.2:1	1.3:1
Elastic modulus of filling materials (E_f)/GPa	0.29	0.25	0.24	0.18	0.15	0.13	0.12

Table 2. Geological and mining parameters to calculate roof sinking amount in CECB [27,30].

Mining Height (h)/m	Bury Depth of Coal Seam (H)/m	Width of the MR (w)/m	Length of the Panel ($2l$)/m	Elastic Modulus of the Roof (E)/GPa	Average Bulk Density of Strata (γ)/kN·m ^{−3}	Elastic Modulus of the Coal (E_c)/GPa
2.5	219.5	6	288	13.2	20.3	2.0

According to the Equations (15) and (17) as well as the parameters listed in the Tables 1 and 2, the maximum loads on the coal pillars and the filling bodies could be estimated. Figure 9a reveals the maximum loads and the strengths of the coal pillar while using filling materials with different water–solid ratios. With the increase of the water–solid ratio, the elastic modulus of the filling material was decreased, and this therefore caused higher stress load on the coal pillar. From Equation (16), the width and height of the coal pillar decided its strength, which was not related to the filling materials. Thus, its strength shows constant in Figure 9a. While the water–solid ratio of the filling material was 0.7:1, 0.8:1 and 0.9:1, the peak loads on the coal pillar were less than its strength, and the coal pillar could stay stable. Figure 9b shows the maximum loads and the strengths of the filling body with different water–solid ratios. The strength of the filling body was falling, but the bearing load was growing while increasing the water–solid ratio. For the water–solid ratios at 0.7:1, 0.8:1 and 0.9:1, the peak loads on the filling body were less than its strength, and the filling body could stay stable.

Figure 9. Strengths of coal pillars (a) and filling materials (b) and loads applied on them while using filling materials with different water–solid ratios.

Overall, both the coal pillar and the filling body could remain stable when the water–solid ratios of the filling material were 0.7:1, 0.8:1 and 0.9:1. However, from the cost of backfilling, a higher water–solid ratio could cause less usages of the filling aggregate and reduce the cost. Therefore, with comprehensive considerations in many factors, such as the stability of the coal pillar and the filling body as well as the cost of backfilling, the water–solid ratio of the filling material was finalized at 0.9:1 [30].

4.4. The Roof and Ground Subsidence in Every Stage

In the study of the ground subsidence induced by mining, the ground subsidence coefficient is commonly used to evaluate the relationship between the mining height and the subsidence amount [44]. In the horizontal coal seam, the ground subsidence coefficient can be calculated from the following formula:

$$k_s = U_{\max} / m_e \quad (19)$$

where, k_s is the ground subsidence coefficient, $U_{\max}$ is the maximum ground subsidence amount induced by mining (mm) and m_e is the equivalent mining height (mm). In mining with backfill, the equivalent mining height equals to the sum of the compression amount of the filling body and the unfilled height [45]. In this paper, the equivalent mining height m_e was approximately equal to the

ground subsidence amount at the fourth stage U_4 . Therefore, according to the Equation (18), the maximum ground subsidence amount was as follows:

$$U_{\max} = k_s U_4$$

The overlying strata of the coal seam at the Wangtaipu coal mine mainly consists of medium to hard rock such as sandstone and limestone, so the ground subsidence coefficient k_s is typically located in the range from 0.55 to 0.84 [46]. With Equations (10) and (19) as well as parameters included in Tables 2 and 3, the peak subsidence of the roof and the ground when using the filling material with the water–solid ratio of 0.9:1 can be achieved, as shown in Figure 10. The subsidence of the roof and the ground had an increased trend in all stages, and the corresponding subsidence took 2.47%, 6.57%, 16.49% and 74.47% of the total subsidence. Due to the elastic modulus of the filling body being less than the coal pillar, most of the overlying load was applied on the coal pillar in the second and third stage in which the coal pillar and backfilling body bore load jointly. In the fourth stage, all the load from the overburden was transferred onto the filling bodies, which caused the filling bodies to have a relatively large compression amount, and the roof had the peak subsidence in this stage which was 43.02 mm. The ground subsidence was 46.42 mm with the ground subsidence coefficient of 0.84, which means the ground subsidence was effectively controlled [47].

Table 3. Mechanics properties of strata and filling materials in the numerical model.

Lithology	Bulk Modulus/GPa	Shear Modulus/GPa	Cohesion/MPa	Internal Friction Angle/°	Tensile Strength/MPa	Bulk Density /kN·m ³
Loess	0.06	0.04	0.12	24	0.08	14.7
Red clay	0.08	0.05	0.33	27	0.12	19.7
Medium sandstone	3.2	2.0	1.6	31	1.4	20.6
Sandy mudstone	2.8	1.7	1	22	1	19.7
Mudstone	2.9	2.1	1.4	19	1.1	28.6
Sandy mudstone	3.0	1.8	3.0	1.8	3.0	22.4
Fine sandstone	3.8	2.6	1.6	31	1.4	20.6
Limestone	7.2	5.5	2	40	4.0	24.0
Sandy mudstone	3.0	1.8	1	20	1	19.8
Limestone	7.2	5.5	2	40	4.0	24.0
Coal/filling body	2.2/0.14	0.76/0.1	1/0.24	20/36	1/0.19	18.7/18.0
Bauxitic mudstone	4.2	2.8	1.4	34	1.4	28.6

Figure 10. Subsidence of roof and ground in every stage.

5. Numerical Simulation on the Ground Subsidence Induced by Mining

5.1. Numerical Model

FLAC^{3D}, a three-dimensional finite difference program developed by the ITASCA company, is one of the most widely used numerical simulation software in geotechnical engineering such as underground mining [38]. In this paper, FLAC^{3D} was employed to simulate the subsidence and deformation of the roof and the ground in CECB longwall working face. A Mohr–Coulomb constitutive model with the dimensions 648 m × 540 m × 232 m was built. The normal displacement is constrained at the four sides and the bottom surface of the model, and the top surface is unconstrained. 96 MRs, with a length of 80 m and width of 6 m, were arranged on both sides of the panel, and then they were excavated and backfilled in four stages. In order to confirm the mechanical parameters of each stratum, rock uniaxial compressive simulation tests were performed based on the Mohr–Coulomb criterion. Mechanical parameters of every stratum in the model could be obtained by continuously adjusting the simulation parameters until the simulation results were consistent with the stress–strain characteristics obtained from the actual uniaxial compression test. Mechanical tests on specimens of filling materials with the water–solid ratio of 0.9:1 and cured for 14 days were performed to collect mechanical parameters of the filling body. Figure 11 shows the numerical model and Table 3 highlights all of the parameters for this model.

Figure 11. Numerical model and panel arrangement.

5.2. Simulation Process

MRs in the mining panel from the starting cutting to the stop line were numbered 1, 2, ..., 48, and every number included two MRs on two sides. In every step in the simulation, MRs which were under mining and backfilling, and had been backfilled for less than 14 days (filling body cured for less than 14 days), were considered as having failed to get effective support. If it took three days to complete the mining and backfilling works in the MR, there were seven MRs without effective support, and they were set to the “null” model. The rest of the MRs which had been backfilled for more than 14 days were set to the Mohr–Coulomb model and assigned and calculated according to mechanical parameters of the filling materials to complete the numerical simulation of this step.

5.3. Roof Subsidence

Considering the property of symmetry, as shown in Figure 12, we took the left half of the roof subsidence curve for analysis. In the horizontal axis of Figure 12, -180 to 0 m referred to the boundary coal pillar in the simulation, 0 to 144 m represented the half length of the stope. In the first three stages, the roof subsidence curves obtained from numerical simulation fluctuated with a period of 24 m (the sum of the widths of four MRs), and the volatility was basically stable in the range from 96 to 144 m. Thus, the average of the roof subsidence in this range was considered as the numerical simulation result and then was compared with theoretical analysis result, and the differences were 14.58% , -9.73% and -13.13% in the first three stages.

Figure 12. The subsidence amount of the roof center line along the strike direction.

For the fourth stage, due to the subsidence and deformation of the strata, the overlying load moved from the goaf boundary to the top of the coal pillar. However, the theoretical analysis assumed that the overlying load was a uniform load. This caused the theoretical analysis results to be larger than the simulation results on the roof subsidence on the goaf boundary, but the opposite situation on the top of the coal pillar. At 144 m in the middle part of the stope, the peak subsidence of the roof from numerical simulation and theoretical analysis were 52.43 mm and 55.26 mm, respectively, and they had a 5.12% difference. Therefore, the numerical simulation results were in good agreement with the theoretical analysis results.

5.4. Ground Subsidence and Deformation

Figure 13 indicates the ground subsidence after each stage, and the peak subsidence of every stage is 1.38 mm, 4.68 mm, 9.32 mm and 26.72 mm, respectively. In the process of mining in every stage, the amount of ground subsidence increases in turn and takes 3% , 11% , 22% and 64% of the totally subsidence accordingly. The influence range induced by mining can be evaluated by the advance distance of influence, which is defined as the horizontal distance between the boundary of the goaf to the point at which the ground has sunk 10 mm [48]. The ratio of the coal seam depth and the advance distance of influence is the tangent of the subsidence angle. According the results of the numerical simulation, the advance distance of influence is 48 m and the subsidence angle is 78° , as seen in Figure 13. In addition to the ground subsidence, other indices are also used to characterize

effects on the ground by mining, including horizontal displacement, tilt, curvature and horizontal strain [49,50]. According to the displacement of the nodes in the x, y and z directions in the numerical simulation, these indices are also calculated and shown in Figure 14. In addition, the extreme values of horizontal displacement, tilt, curvature and horizontal strain in the direction of dip are 14.50 mm, 0.23 mm/m, 0.00336 mm/m² and 0.19 mm/m, respectively, which are greater than their extreme values in the strike direction, which are 14.28 mm, 0.21 mm/m, 0.00232 mm/m² and 0.15 mm/m.

Figure 13. The subsidence amount of the surface center line along the strike direction after mining.

Figure 14. Cont.

Figure 14. Numerical simulation results on the horizontal displacement, tilt, curvature and horizontal strain in the strike and dip directions.

6. Field Practice

The mining rate of the test panel at Wangtaipu coal mine is 96.8%, and as shown in Figure 15, the filling bodies in MRs fully connect to the roof, and a relatively good backfilling effect was obtained [27]. Two measuring lines with length of 680 m and 480 m were arranged on the ground along the strike and dip directions, including 29 measuring points on the strike direction and 15 measuring points on the dip direction (see Figure 2). The data of ground movement induced by mining was collected once a month for five months after the end of mining and backfilling.

Figure 15. Situations of filling bodies connected to the roof in mining roadways (MRs). Reprint with permission [24]; Copyright 2018, Journal of China coal society.

Figure 16 represents maximum values of various indices in terms of ground movements along the strike and dip directions for each of the five data collections. The monitored peak values of

ground subsidence, horizontal displacement, tilt, curvature and horizontal strain were 28 mm, 15 mm, 0.8 mm/m, 0.09 mm/m² and 0.63 mm/m, respectively, and all these values were larger than the simulated results. Results of the ground subsidence and the horizontal displacement, which were collected from site monitoring, were close to the simulation. The site monitoring data of the tilt, curvature and horizontal strain had relatively large differences from their simulated values. This may be due to the anisotropy of the overlying strata and the presence of primary fractures in this strata. Therefore, the ground deformation is uneven, and the deformation is large in the places where the rigidity is small or there is a primary crack. The monitored values of the ground tilt, curvature and horizontal strain were all less than the building Class I damage rating indicator.

Figure 16. Indices of ground movements along the measuring line in strike and dip direction after mining.

Monitoring results collected from the points which were arranged at the constructions on the ground showed that the maximum sinking and horizontal displacement values of the building were 10 mm and 9 mm, respectively. The maximum sinking and horizontal displacement of the chimney were 9 mm and 7 mm, respectively. In addition, no cracks appeared on the surface of constructions.

Considering the measurement error, it can be assumed that the ground buildings did not have significant subsidence and deformation, and a relatively good effect has been achieved on the ground subsidence control.

7. Discussion

Compared with the traditional longwall backfilling coal mining, the coal mining system and backfilling system were arranged at different MRs and operated in parallel in the CECB method. It could guarantee the working efficiency and avoid the mutual effects of mining operations and backfilling efforts. The MR had less exposed area, and two ribs of it were always supported by the coal wall or the filling body. The roof had small sinking and deformation amount before performing backfilling in the MR; therefore, enough filling space was available, and the filling body could effectively control overlying strata movement and ground subsidence.

For the CECB method in longwall mining which includes four stages, after completing mining activities in the third stage, the roadway-side coal pillar (the coal pillar in the MR in the fourth stage) had the peak compression amount which was the sum of the roof sinking amount occurring in the first, second and the third stage. Thus, the support of the roadway rib should be strengthened to prevent the collapse of the coal wall.

The filling body in the first stage had the maximum compression amount after completing the fourth stage, and it was the sum of roof sinking amount that appeared in the second, third and fourth stage. Therefore, the strength and filling effect of the filling body in the first stage should be evaluated before starting the fourth stage's coal mining. The main objective of the filling body in the fourth was to restrain the lateral slip of the two sides of the filling body, and to reduce the secondary subsidence after mining in the mining area. For the scenarios that the coal seams have relatively good conditions; namely, the filling body without lateral slip hazard, the filling procedures can be omitted from the fourth stage to reduce the cost of the newly suggested coal mining method, and therefore to improve coal mine economic benefits.

In the test panel at Wangtaipu coal mine, an average of three days was required to mine and backfill the mining roadway (MR). In addition, the total number of active MRs at any point in operation, which included filling bodies with a solidification time of less than 14 days, was within five. This number is negligible, relative to the entire of 96 MRs that were planned in the test panel. Thus, the elastic modulus of specimens with 14 days' solidification could be used to represent the elastic modulus of all filling bodies. After completing the fourth stage, the filling body in the first stage was bearing the peak load; however, the solidification time of it was the entire time to complete the other three stages (216 days), which was far greater than the full solidification time of the filling body (63 days). Therefore, while performing the stability analysis on the filling body, it was reasonable to consider the strength of the filling body in the first stage according to its final strength after 63 days of solidification.

8. Conclusions

Based on the Winkler elastic foundation beam theory, a mechanical model to determine the roof sinking while using the CECB method in longwall mining was built. Then the foundation coefficients of the test panel after the end of any stage were analyzed, and calculation methods to estimate the roof sinking amounts of various stages were provided.

While applying the CECB method in longwall mining, the filling bodies in the first stage as well as the coal pillars in the last stage were most prone to instability. Based on the stability of the coal pillar and the filling body, the parameters of filling materials were determined, and the maximum sinking amount of the ground was calculated at 46.42 mm.

The differences of the maximum sinking amounts of the roof which were obtained from theoretical analysis and the numerical simulation for four stages were all within 15%, and the difference of the final sinking amount of the roof was 5.12%. The calculation results of these two methods have a relatively good consistency. Indices of ground movements, collected from the numerical simulation and the field measurement, including the ground subsidence, horizontal displacement, tilt, curvature and horizontal strain, were all less than the level I damage rating of building in China, which means that the ground subsidence had been efficiently controlled.

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