Cooperative Control Mechanism of Efficient Driving and Support in Deep-Buried Thick Top-Coal Roadway: A Case Study

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Abstract: For deep-buried thick top-coal roadways under high stress, there exists great difficulty in controlling the stability of the surrounding rock as well as in the necessity for low driving speeds. Taking the return air roadway 20201 (RAR 20201) of the Dahaize Coal Mine as the background, this paper presents a typical engineering case of a deep-buried thick top-coal roadway in a western mine. Through methods such as in situ investigation, theoretical analysis, numerical simulation and engineering practice, we studied the deformation and failure mechanisms of the surrounding rock in a deep-buried high-stress thick top-coal roadway, and revealed the driving speed effect. Results show that compared with shallow buried roadways, the deep-buried thick-roof coal roadway suffers a greater range of damage and failure. The roof damage is so deep that it exceeds the action range of bolts, resulting in the stress transferring to both sides, which affects the stability of the roadway surroundings. The curve of unloading disturbance stress produced by roadway head-on driving is in accordance with the “power exponential” composite function; that is, the faster the driving speed, the less unloading disturbance intensity that is exerted on the roof strata. This paper puts forward targeted cooperative control countermeasures of efficient driving and support in a deep-buried thick top-coal roadway. On one hand, the support efficiency of a single bolt is improved so as to reduce the overall support density; on the other hand, under low support density, the driving-supporting circulation efficiency is also accelerated so as to weaken the unloading disturbance and improve roadway formation speed. Engineering practice shows great control effect of the roadway surrounding rock, and the roadway formation speed is also greatly improved. This research can provide reference for efficient driving and support design in similar deep-buried thick top-coal roadways.

Keywords: high stress in deep-buried mine; thick top-coal roadway; bolt support; driving and support coordination; surrounding rock stability control

1. Introduction

Safe and efficient driving and support in deep-buried, high-stress thick top-coal roadway have become major technical problems in deep mining [1–3]. However, due to the high stress and low intensity of top-coal, large deformations [4–6] and support structure failures often occur during driving [7–9], making it more difficult to control the stability of the surrounding rock in deep-buried thick top-coal roadways [10,11]. The current support scheme of high-density bolt and cable combinations [9,12,13] not only fails to meet the requirement of effective stability control [14,15] but also restricts the roadway driving speed, binging great technical challenges in ensuring safe and efficient coal mining [16,17].
In view of the deformation mechanism and stability control of the surrounding rock in deep-buried thick top-coal roadways, many scholars have carried out detailed research. Li et al. studied the stress evolution law of the surrounding rock in deep, thick top-coal roadways through a large-scale geomechanical model test system and proposed that the influence of roadway driving on the surrounding rock stress gradually weakened with the increase in the surrounding rock depth; the surrounding rock in the shallow part was seriously loosened and damaged, and the stress drop of the shallow measuring point was 1.9 times that of the deep part [18,19]. Li et al. studied the deformation and failure characteristics of the surrounding rock in deep, thick top-coal roadways and proposed that the vertical stress release of the surrounding rock in the roof and floor of roadways was more intense than that of the two sides, which was contrary to the horizontal stress release. In addition, the surrounding rock in the middle of the roadway roof was the main part of the vertical stress release of the roof [20]. Based on the instability and failure mechanism of the surrounding rock in thick top-coal roadways, Shan et al. put forward an overall anchorage support system of the roof and two sides and clarified its action mechanism, which achieved successful practical application [21]. After analyzing the evolution of surrounding rock stress, the elastic properties of deep-buried thick top-coal roadways and the stress state of anchor cables, Xu et al. revealed the failure mechanism of anchor cables in thick top-coal roadways and proposed surrounding rock grouting technology, which successfully solved the problems of strong ground pressure behaviors such as anchor cable fracture and failure [22]. Yang et al. studied the deformation mechanism of the surrounding rock of a large section of thick top-coal roadway using comprehensive methods and put forward control countermeasures. It was considered that the tensile stress in the middle of the roof and the concentration of shear stress at both ends of the long-span roadway were the main causes of roof fracture and surrounding rock extrusion deformation. A comprehensive support system based on a “high strength, high resistance and high prestress anchor cable” was proposed and successfully applied at the project site [23]. Liu et al. studied the failure law of the surrounding rock of deep-buried thick top-coal roadway under rock burst and proposed that the surrounding rock has typical characteristics, such as fast deformation, long duration, asymmetric deformation and large loose broken area; the authors suggested surrounding rock control technology based on large-diameter drilling and deep hole blasting unloading, which effectively controlled the stability of the surrounding rock [24].

In terms of the research on efficient and rapid driving on deep-buried roadways, Ma et al. proposed new support technology and construction schemes through optimizing the driving and support technology of deep roadways and conducted industrial tests that showed great control effects of the surrounding rock stability and significant improvements in roadway formation speed [25]. Yang et al. studied the deformation and failure characteristics of the strata in empty roof zones during rapid roadway driving and proposed that when the distance between the roof empty zone was within 0~2.3 m, the bending deformation of the roof was slow; when the distance was greater than 2.3 m, the bending deformation of the roof accelerated, which provided theoretical guidance for reasonably controlling the distance of the empty roof zone during rapid head-on driving [26]. Liu et al. studied the influence law of driving speed on the stress field and displacement field of the surrounding rock in thick top-coal roadways through theoretical calculation and numerical simulation; they proposed a support system with non-row spacing of bolts between the roof and rib, which significantly improved the driving speed of the roadway [27].

Most of the research outlined above focused on the deformation and damage laws of deep-buried thick top-coal roadways and the corresponding control countermeasures of driving support, which provide guidance for the further study of safe and efficient driving and support of such roadways. However, there is still a lack of research on the cooperative relationship between the damage range of surrounding rock and the thickness of the bolt anchorage layer as well as the driving speed effect. Compared with shallow buried low-stress roadways, the deep-buried thick top-coal roadway has a wider range
of damage and failure, in which the depth of roof damage may exceed the action range of bolts, resulting in poor bearing efficiency of single bolts. In the case of low support efficiency of single bolts, the current support scheme of the high-density bolt and cable combination shows poor group anchor effect, and the driving speed is slow, all of which lead to more severe damage to the rock mass and support system, caused by unloading disturbance during head-on driving.

Therefore, it is of great significance to study the damage range of surrounding rock and the speed effect of deep-buried thick top-coal roadways during head-on driving, which can provide engineering guidance for improving driving speed and weakening the head-on disturbance. Using the engineering background of the return air roadway 20201 (RAR 20201) of the Dahaize Coal Mine, a typical engineering case of deep-buried thick top-coal roadways in western mines is introduced. Through comprehensive methods such as in situ investigation, theoretical analysis, numerical simulation and engineering practice, this paper clarifies the stress distribution characteristics of the surrounding rock in deep-buried high-stress and thick top-coal roadways, revealing the transmission form of disturbance stress and determining the speed effect of driving, all of which are successfully applied, allowing for the realization of efficient driving-support coordination in such roadways.

2. Analysis of Engineering Geological Conditions

2.1. Engineering Geological Conditions of Roadway

2.1.1. Layout of Working Face and Roadway

The Dahaize Coal Mine in Yulin, Shanxi, China, with a designed production capacity of 15 million t/a and vertical shaft development, was studied. The working face 20201 is the first mining face in the mining area 202 of the Dahaize Coal Mine, designed with two mining roadways, including a return air roadway and transportation roadway. In this study, the RAR 20201 (Figure 1) was selected as the research object, with a total length of 3804 m. The roadway was driven along 2# Coal seam floor, with a rectangular section and a driving size of width × height = 5740 mm × 4550 mm.

![Figure 1. Layout of working face and location of test roadway.](image)

2.1.2. Lithology of the Coal Seam, Roof and Floor

Within the driving scope of the working face 20201, the average buried depth of 2# Coal seam is 650 m, and the average thickness is 6.8 m. The maximum vertical stress is 16 MPa, and the lateral pressure coefficient is 1.0, so the maximum horizontal stress is 16 MPa. The roof of the coal seam is followed with 16.1 m siltstone, 1 m fine grained sandstone, 9.9 m sandy mudstone and 9.4 m medium-grained sandstone; the floor of the coal seam is 10.8 m siltstone. The compressive strength of the main roof siltstone is 40.56 MPa, and that of floor siltstone is 36 MPa. The specific mechanical parameters of the rock strata are shown in Figure 2, and the corresponding rock occurrence and mechanical parameters are shown in Table 1.
Table 1. Rock occurrence and mechanical parameters.

<table>
<thead>
<tr>
<th>Number</th>
<th>Lithology</th>
<th>Thickness (m)</th>
<th>Compressive Strength (MPa)</th>
<th>Elastic Modulus (GPa)</th>
<th>Poisson Ratio</th>
<th>Tensile Strength (MPa)</th>
</tr>
</thead>
<tbody>
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<td>4</td>
<td>Medium Sandstone</td>
<td>9.4</td>
<td>52.4</td>
<td>22.9</td>
<td>0.3</td>
<td>8.8</td>
</tr>
<tr>
<td>3</td>
<td>Sandy Mudstone</td>
<td>9.9</td>
<td>32.1</td>
<td>11.5</td>
<td>0.2</td>
<td>4.0</td>
</tr>
<tr>
<td>2</td>
<td>Fine Sandstone</td>
<td>1.0</td>
<td>43.5</td>
<td>17.1</td>
<td>0.2</td>
<td>6.7</td>
</tr>
<tr>
<td>1</td>
<td>Siltstone</td>
<td>16.1</td>
<td>40.6</td>
<td>15.8</td>
<td>0.2</td>
<td>6.2</td>
</tr>
<tr>
<td>0</td>
<td>2# Coal</td>
<td>6.8</td>
<td>10.4</td>
<td>1.8</td>
<td>0.2</td>
<td>1.2</td>
</tr>
<tr>
<td>−1</td>
<td>Siltstone</td>
<td>10.8</td>
<td>40.6</td>
<td>15.8</td>
<td>0.2</td>
<td>6.2</td>
</tr>
</tbody>
</table>

2.2. Analysis of Existing Roadway Support Scheme and Maintenance Effect

2.2.1. Existing Support Scheme

(1) Roof support: As shown in Figure 3, the roof adopts the combination support of “anchor bolt + anchor cable”, and the foundation support adopts a Φ22 × 2400 mm left-handed spiral steel anchor support. Each row adopts six bolts with a spacing of 1000 mm and row spacing of 1100 mm. Among them, the anchor bolts at the shoulder angle are installed at an external inclination of 20°, and all other anchor bolts are installed perpendicular to the roof. Each anchor bolt is anchored with one MSK 2335 mm and one MSZ 2360 mm resin cartridge, and the pre-tightening force of the bolts is 40 kN.

The reinforcement support of the roof comprises Φ22 × 6300 mm anchor cables. Each row has two cables with the spacing of 2000 mm, and the row spacing is 2200 mm. All anchor cables are installed perpendicular to the roof. Each anchor cable is anchored with one MSK 2335 mm and two MSZ 2360 mm resin cartridges, and the pre-tightening force of the cable is 250 kN.

(2) Coal rib support: The support parameters of the two coal sides are consistent, and the supporting foundation comprises Φ22 × 2400 mm left-handed spiral steel anchor supports. Each row has five bolts with spacing of 950 mm, and the row spacing is 1100 mm. All anchor bolts are installed perpendicular to the coal rib. Each anchor bolt is anchored with one MSZ 2360 mm resin cartridge, and the pre-tightening force of the bolts is 40 kN.
2.2.2. Maintenance Effect

The RAR 20201, with an average buried depth of 650 m, vertical stress of 16.3 MPa and top-coal thickness of 2.25 m, is a typical deep-buried high-stress and thick top-coal roadway. During roadway driving, the separation fractures outside the roof anchorage zone are developed; meanwhile, coal rib fall exists and tends to be serious (Figure 4), with a maximum rib fall exceeding 1.8 m. Moreover, the adopted support scheme of “high-density bolt and cable combination” results in longer supporting time during the driving-supporting circulation. Under the circumstances of a longer supporting time and severe coal rib fall, the driving speed is slowed down to only 390 m/month, which is far from the requirements of rapid working face mining.

![Figure 3. Existing support scheme of RAR 20201.](image)

**Figure 3.** Existing support scheme of RAR 20201.

2.3. Issues Proposed and Research Directions

2.3.1. Issues Proposed

(1) Under deep-buried high-stress environments, the roadway surrounding the rock suffers from a larger range of damage.

As previously analyzed, the RAR 20201 lies in a high-stress environment. After driving on the deep-buried high-stress roadway, the stress of the surrounding rock is rapidly adjusted, causing damage to the shallow part. Compared with shallow buried roadway, the rock mass in the deep-buried roadway experiences a larger range of damage under high stress, as outlined below.

![Figure 4. Maintenance effect of roadway: (a) rib fall; (b) separation fracture outside the roof bolt anchorage zone.](image)

**Figure 4.** Maintenance effect of roadway: (a) rib fall; (b) separation fracture outside the roof bolt anchorage zone.
Roof: The 2.4 m thickness of the anchorage layer may be insufficient, with weak roof control ability. The RAR 20201, with a top-coal thickness of 2.25 m and low intensity, lies in the deep-buried high-stress environment. Once the damage range of the roof exceeds 2.4 m, the overall 2.4 m bolts will fall in the damage range (Figure 5a), which reduces the bearing capacity of the anchor bolts. The development of a separation fracture outside the roof bolt anchorage zone fails to promote the continuous transfer of roof stress; therefore, the stress transfers to both sides, resulting in tensile fracture of the rock mass in the middle of the roof (Figure 6a).

Coal rib: The discontinuous transfer of roof stress leads to the stress concentration of the coal rib and a larger range of damage and failure. For the coal rib, the larger damage range along with the roof stress transfer to two sides further concentrates the coal rib stress and enlarges the damage range (Figure 6a).

(2) The support density is large, and the driving speed is difficult to improve. The overall bolt-cable support density of the roadway’s surrounding rock reaches 15.5 pieces/m, with the roof support density reaching 6.4 pieces/m. That is, a large amount of head-on support and a long supporting time in driving-supporting circulation significantly restrict the driving speed of the roadway.
(3) The slow driving speed increases the likelihood of rock stress, which results in more serious damage to the rock mass.
Most of the high-stress roadways adopt single-row circulation, that is, supporting one row after driving over it. Therefore, it is inevitable for each row of bolts to undergo disturbance from the heading face. The slower the driving speed, the longer the action time of the head-on cutting disturbance stress on the adjacent anchorage system, and the larger the damage to the anchorage rock mass and the support system.

If the driving-supporting speed is fast, that is, upon the completion the current row of driving support in the initial stage of disturbance stress diffusion adjustment, the next cycle of driving support follows rapidly and is supported in a timely fashion, the speed effect of the head-on unloading disturbance can be fully utilized to alleviate the action time of the disturbance stress, reduce the damage range of the surrounding rock and alleviate the damage degree of the anchorage rock mass and support system.

2.3.2. Research Directions

(1) Study of the damage range of the surrounding rock in deep-buried high-stress roadways. From the previous analysis, it can be concluded that the damage range of deep-buried high-stress roadways obviously differs from that of shallow buried roadways. Therefore, taking the shallow buried roadway as a comparison helped in the study of the damage range of the surrounding rock of deep-buried thick top-coal roadways for bolt support design.

(2) Study of the fluctuation patterns of the head-on cutting disturbance stress and speed effect of cutting disturbance. The head-on cutting of roadways will result in unloading disturbance on the adjacent anchorage system. Therefore, it is of great significance to study the fluctuation patterns of the head-on cutting disturbance stress of roadways and the attenuation law of the unloading disturbance stress under different driving speeds, providing guidance for weakening the cutting disturbance by improving the driving speed.

3. Study on Stress Distribution Characteristics and Driving Speed Effect of Surrounding Rock in High-Stress Roadways

3.1. Stress Distribution Characteristics of Surrounding Rock in High-Stress Roadways

3.1.1. Establishment of Numerical Model

This section uses the finite element FLAC3D numerical software to establish two groups of roadways for numerical calculation. One group uses the numerical calculation model for deep-buried and high-stress roadway driving (with a roadway depth of 650 m), which is established based on the actual engineering geological conditions of RAR 20201 in the Dahaize Coal Mine. The control group uses the numerical calculation model for shallow buried and low-stress roadway driving (with a coal seam depth of 200 m). These two model groups will drive along the Coal 2# seam floor, with a roadway size of width $\times$ height $= 5.74 \, m \times 4.55 \, m$, a coal seam thickness of 6.8 m and a top-coal thickness of 2.25 m (consistent with the actual situation).

According to the size of the roadway section and the influence range of the working face, the model size is selected as width $(x) \times$ height $(z) \times$ thickness $(y) = 56 \, m \times 54 \, m \times 20 \, m$ (Figure 7). The distance between the coal seam and the upper boundary of the model is 36.4 m. The upper boundary of the model simulates the actual buried depth by applying the equivalent load. If the average density of the rock mass is 2500 $kg \cdot m^{-3}$, the vertical loads applied on the top boundary of the high-stress roadway model and the low-stress roadway model are 15.3 MPa and 4.1 MPa, respectively. The lateral pressure coefficient is selected as 1.0. The bottom boundary of the model is fixed vertically, and both the left and right boundaries are fixed horizontally to eliminate displacement. The coal seam, roof and floor strata in the model are consistent with the actual situation, and the mechanical parameters of the rock in each stratum can be seen in Table 1. The model adopts the Mohr-Coulomb yield criterion.
The mechanical parameters of each rock stratum in the numerical model are obtained by converting the mechanical parameters of the rock in Table 1. The specific conversion method is as follows: The elastic modulus $E_r$ of each rock in Table 1 is converted to obtain the elastic modulus $E_m$ of the corresponding rock mass through Formula (1). The $E_m$ and Poisson’s ratio $V$ are converted to obtain the bulk modulus $K$ and shear modulus $G$ of each rock through Formulas (2) and (3). The compressive strength $\sigma_c$ of the rock is converted to obtain the compressive strength $\sigma_{cm}$ of the rock mass using Formula (4), and the tensile strength $\sigma_{tm}$ of the rock mass is obtained using Formula (5). Then, the mechanical parameters of each rock mass obtained by conversion are verified until the macro mechanical parameters of the model are consistent with those of the target rock mass. The mechanical parameters of the rock mass given in the final numerical model are shown in Table 2.

$$E_m = 10^{0.0186RQD−1.91}$$  \hspace{2cm} (1)

### Table 2. Rock mass mechanical parameters of each rock stratum in the numerical model.

<table>
<thead>
<tr>
<th>Number</th>
<th>Lithology</th>
<th>Thickness (m)</th>
<th>Density (kg m$^{-3}$)</th>
<th>Bulk Modulus (GPa)</th>
<th>Shear Modulus (GPa)</th>
<th>Internal Friction Angle (°)</th>
<th>Cohesion (MPa)</th>
<th>Tensile Strength (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>4</td>
<td>Medium Sandstone</td>
<td>9.4</td>
<td>2630</td>
<td>8.6</td>
<td>4.5</td>
<td>27</td>
<td>5.4</td>
<td>3.6</td>
</tr>
<tr>
<td>3</td>
<td>Sandy Mudstone</td>
<td>9.9</td>
<td>2380</td>
<td>2.5</td>
<td>2.0</td>
<td>25</td>
<td>1.35</td>
<td>2.1</td>
</tr>
<tr>
<td>2</td>
<td>Fine Sandstone</td>
<td>1.0</td>
<td>2750</td>
<td>6.6</td>
<td>3.2</td>
<td>28</td>
<td>5.1</td>
<td>3.2</td>
</tr>
<tr>
<td>1</td>
<td>Siltstone</td>
<td>16.1</td>
<td>2510</td>
<td>6.0</td>
<td>3.8</td>
<td>26</td>
<td>8.1</td>
<td>3.1</td>
</tr>
<tr>
<td>0</td>
<td>2# Coal</td>
<td>6.8</td>
<td>1260</td>
<td>1.5</td>
<td>0.3</td>
<td>20</td>
<td>0.9</td>
<td>0.5</td>
</tr>
<tr>
<td>−1</td>
<td>Siltstone</td>
<td>10.8</td>
<td>2510</td>
<td>6.0</td>
<td>3.8</td>
<td>26</td>
<td>8.1</td>
<td>3.1</td>
</tr>
</tbody>
</table>

Among them, RQD is the rock quality index, which is obtained by observing the surrounding rock of RAR 20201.

$$K = \frac{E}{3(1−2v)}$$ \hspace{2cm} (2)

$$G = \frac{E}{2(1+v)}$$  \hspace{2cm} (3)
where \( j \) is generally taken as 0.56.

The value range of \( K \) is 0.05–0.1, and the value is taken as 0.1 in this formula.

3.1.2. Simulation Methods

The numerical calculation process is as follows: (1) Calculate the original rock stress to the initial equilibrium; (2) Drive the RARA 2021 along the floor of Coal 2# seam floor step by step. Each step comprises driving for 1 m until the calculation is balanced, followed by the next cycle of driving (simulating roadway head-on cyclic driving at the project site). A total of 12 operations (driving 12 m) are carried out.

3.1.3. Results Analysis

During roadway driving, the vertical stress distribution comparison of the surrounding rock in RAR 20201 (high-stress roadway) and the low-stress roadway are shown in Figure 8.

In the process of roadway driving, the surrounding rock of the roadway is gradually damaged from shallow to deep; meanwhile, the high stress is transferred to the deep part of the surrounding rock. As a result, a low-value stress area in the shallow part of the surrounding rock exists; this is the damaged zone.

(1) For low-stress roadways, when lagging 11 m behind the heading face, the low-value stress range of roof strata is 1–2 m, and the low-value stress range of the coal rib is 1 m. After the disturbance of low-stress roadway driving is stable, the stress yield ranges of the roof and coal rib are 1–2 m and 1 m, respectively.

(2) For high-stress roadways, when lagging 11 m behind the heading face, the low-value stress range of the roof strata is 3 m, and the low-value stress range of the coal rib is 2–3 m. Therefore, compared with the shallow buried low-stress roadway, the deep-buried high-stress roadway suffers a larger range of damaged depth and area.

Figure 8. Cont.
3.2. Disturbance Stress Distribution Characteristics of High-Stress Roadway in Head-On Cutting

3.2.1. Establishment of Numerical Model

The numerical calculation model selected in this section is consistent with that in Section 3.1; that is, we use two groups of head-on cyclic roadways for numerical calculation. One group uses the numerical model of head-on cyclic cutting of RAR 20201 (deeply buried high-stress roadways); the control group uses the model of shallow buried low-stress roadways. The parameters of the numerical model are consistent with the model in Section 3.1.

3.2.2. Simulation Methods

In order to simulate the unloading disturbance characteristics of the roadway head-on cyclic cutting on the roof rock mass behind the heading face [28], the RAR 20201 is first driven for 10 m, and then the cyclic cutting is carried out from the 11 m to simulate the stress disturbance characteristics on the roof rock mass within the range of 10 m behind. The specific numerical calculation process is as follows: (1) Calculate the original rock stress of the initial equilibrium; (2) drive the RAR 20201 directly to 10 m along the Coal 2# seam floor, then calculate the stress equilibrium; (3) starting at 11 m, drive 1 m per step and then calculate the stress equilibrium, followed by the next driving cycle, which comprises five steps in total (drive from 11 to 15 m).

During roadway driving, a vertical stress measuring line is arranged along the axial direction of the roadway in the middle of the roof at a depth of 2 m to monitor the stress law of roof rock stress. The stress propagation characteristics of the unloading disturbance of the head-on cyclic cutting are obtained after five steps of cyclic driving.

3.2.3. Results Analysis

The distribution laws of unloading disturbance stress (vertical stress difference of adjacent two driving roofs) in the roof rock mass during the five times of cyclic driving are shown in Figure 9a,c. The unloading disturbance stress values and the variation law generated by each cycle cutting are similar. For deep-buried high-stress roadways, according to the analysis of the unloading disturbance stress attenuation curve, the disturbance stress value is larger within the range of a 4 m lag behind the heading face (the maximum value is 2.25 MPa), followed by rapid attenuation. When the lag distance exceeds 4–4.5 m, the disturbance stress value is smaller, followed by slow attenuation.
For shallow buried low-stress roadways, the unloading disturbance range of head-on cyclic cutting is similar to that of high-stress roadways, but the disturbance intensity differs greatly (Figure 10). The maximum disturbance stress within the range of 4 m lag is 1.6 MPa, which is reduced by 40% when compared with the high-stress roadway (2.25 MPa).

There exists a certain functional relationship between the unloading disturbance stress generated by each cutting and the initial distance from the heading face. The unloading disturbance stress curve generated by five cyclic cuttings is fitted by function. Taking the fitting results of the unloading disturbance stress curve at 1 m driving as an example, the fitting results are shown in Figure 9b,d. It can be concluded that the unloading disturbance stress curve conforms to the “power exponential” composite function, and the expression is as follows:

$$\sigma = a - b \times c^x$$

(6)

where $\sigma$ is the unloading disturbance stress, MPa; $a$, $b$, $c$ are all constants; $x$ is the distance from one point of roof strata behind the heading face to the heading face, m.
3.3. Speed Effect Analysis of High-Stress Roadway in Head-On Cutting

3.3.1. Establishment of Numerical Model

The numerical calculation model selected in this section is consistent with that in Section 3.2; that is, we use two groups of head-on cyclic roadways for numerical calculation. One group uses the numerical model of head-on cyclic cutting of RAR 20201 (deeply buried high-stress roadway); the control group uses the model of shallow buried low-stress roadways. The parameters of the numerical model are consistent with the model in Section 3.2.

3.3.2. Simulation Methods

In order to simulate the speed effect of the high-stress roadway head-on cutting, the RAR 20201 is first driven for 10 m, and then the cyclic cutting is carried out with different driving speeds from 11 m to simulate the stress disturbance characteristics on the roof rock mass within the range of 10 m behind the heading face. The specific numerical calculation process is: (1) Calculate the original rock stress of the initial equilibrium; (2) drive the RAR 20201 directly to 10 m along the Coal 2# seam floor and then calculate for the stress equilibrium; (3) starting at 11 m, 1 m per step is first driven and calculated for the stress equilibrium, followed by the next driving cycle, which results in driving five times in total (drive from 11 to 15 m).

The influence of driving speed is mainly reflected in the different release speeds of original rock stress after roadway driving [29]: a faster driving speed corresponds to a slower stress release speed. Therefore, five groups of different stress release speeds ($R = 0.1, 0.2, 0.3, 0.4, 0.5$) were set to simulate different driving speeds.

During roadway driving, a vertical stress measuring line is arranged along the axial direction of the roadway in the middle of the roof at a depth of 2 m to monitor the unloading stress law of roof rock stress. The speed effect of disturbance in roadway head-on cutting can be obtained by comparing and analyzing the variation law of unloading stress under five different driving speeds.

3.3.3. Results Analysis

The variation laws of unloading disturbance stress under five different driving speeds is shown in Figure 11a, all of which show the attenuation trend as the increase of the distance to heading face. The curves of unloading disturbance stress are fitted respectively, and the fitting results are shown in Table 3. It can be seen that the unloading stress curves of all the five different driving speeds show “exponential type coincidence function” attenuation, which verifies the above analysis. Among them, the fitting curve of $R = 0.1$ is shown in Figure 11b.
Figure 11. The unloading disturbance stress distribution law of the roadway heading under different driving speeds: (a) the unloading disturbance stress distribution curve of the roadway heading under different driving speeds; (b) the fitting function and curve of unloading disturbance stress of the roadway heading when $R = 0.1$.

Table 3. The fitting function of unloading disturbance stress of the roadway head-on cutting under different driving speeds.

<table>
<thead>
<tr>
<th>Driving Speed</th>
<th>Fitting Function</th>
<th>R-Square</th>
</tr>
</thead>
<tbody>
<tr>
<td>$R = 0.1$</td>
<td>$\sigma = 1.321 + 0.305 \times 0.724^x$</td>
<td>0.9935</td>
</tr>
<tr>
<td>$R = 0.2$</td>
<td>$\sigma = 2.361 + 0.195 \times 0.636^x$</td>
<td>0.9956</td>
</tr>
<tr>
<td>$R = 0.3$</td>
<td>$\sigma = 3.087 + 0.160 \times 0.729^x$</td>
<td>0.9971</td>
</tr>
<tr>
<td>$R = 0.4$</td>
<td>$\sigma = 3.530 + 0.215 \times 0.664^x$</td>
<td>0.9921</td>
</tr>
<tr>
<td>$R = 0.5$</td>
<td>$\sigma = 3.682 + 1.656 \times 0.407^x$</td>
<td>0.9944</td>
</tr>
</tbody>
</table>

When $R = 0.1$, the driving speed is faster and the rock stress release speed is lower; the maximum unloading disturbance stress of cutting on the roof strata is 1.55 MPa. When $R = 0.2, 0.3, 0.4$ and 0.5, the maximum unloading disturbance stress are 2.48 MPa, 3.20 MPa, 3.70 MPa and 4.3 MPa, respectively. Therefore, the following conclusion is drawn: with the increase of driving speed, the disturbance intensity of head-on cutting on the roof strata decreases. The faster driving speed means a shorter single cycle disturbance period and insufficient stress adjustment, which results in less damage to the rock mass. On the contrary, under slower driving speeds, the longer single-cycle disturbance period allows the stress to be fully adjusted, which results in more serious damage to the rock mass.

4. Cooperative Control Countermeasures of Efficient Driving and Support in Deep-Buried Thick Top-Coal Roadways

Through the issue analysis of the existing support scheme and the results of the numerical simulation, it can be concluded that the thick top-coal roadway suffers severe damage under the conditions of deep burial and high stress. Among them, the damage depth of the roof rock mass is up to 3 m, which exceeds the thickness of the bolt anchorage layer in the existing support scheme. Overall, the bolts are in the damaged zone, resulting in the low efficiency of a single bolt support. Under poor support efficiency, the high support density does not control the roof effectively and leads to a longer supporting time in driving-supporting circulation, thereby restricting the driving speed.

Therefore, the roadway driving-supporting technology can be optimized in two aspects. On one hand, we can optimize the support scheme and improve the support efficiency of a single bolt to reduce the overall support density; on the other, we can thicken the bolt anchorage layer to make it pass through the fracture layer in the shallow part of roof into the deep rock layer, which has good stability [30–32]. Moreover, we can eliminate
the separation fracture (Figure 5b) outside the anchorage end to promote the roof stress continuously transferring downward rather than towards both sides, which can preclude the tensile failure of the shallow rock mass and the stress concentration of coal ribs (Figure 6b).

On the other hand, as the numerical simulation results show, the disturbance intensity on the roof strata decreases as the driving speed increases; that is, when the driving speed is slow, the longer single-cycle disturbance period allows the stress to fully adjust, which results in more serious damage to the roof rock mass; when the driving speed is fast, the shorter single-cycle disturbance period and insufficient stress adjustment will result in less damage to roof rock mass (Figure 12). Therefore, under low support density, the head-on cyclic driving and support should be rapidly completed to improve driving speed, which can weaken the unloading disturbance on the roof strata.

![Figure 12](image-url)

**Figure 12.** The disturbance influence of unloading stress on the roof strata under different driving speeds.

5. Industrial Testing

5.1. Optimization of Driving and Support

5.1.1. Optimization of Support Scheme

1. Roof support: As shown in Figure 13, the roof adopts $\Phi 21.8 \times 4500$ mm anchor cable supports. Each row has four cables with a spacing of 1500 mm and a row spacing of 1500 mm. All the anchor cables are installed perpendicular to the roof. Each cable is anchored with one MSCK 2360 mm and one MSK2360 mm resin cartridge, and the pre-tightening force of the cable is 200 kN. The yield load of the anchor cable is 706 kN.

2. Coal rib support: The support parameters of the two coal sides are consistent, and the foundation supporting adopts a $\Phi 22 \times 2600$ mm left-handed spiral steel anchor support. Each row adopts five bolts with the spacing of 1500 mm. Among them, the bolts at the shoulder angle are installed at an external inclination of 15°, and all other anchor bolts are installed perpendicular to the coal rib. Each anchor bolt is anchored with one MSCK 2335 mm and one MSK2360 mm resin cartridge, and the pre-tightening force of the bolt is 60 kN.
Compared with the original support scheme, the thickness of the optimized roadway roof anchorage layer reaches 4.2 m, with an increase of 68%, which greatly improves the stability of the roadway roof strata. In addition, the overall bolt-cable support density of the roadway surrounding the rock reached 9.3 pieces/m, with a decrease of 40%; the roof support density reached 2.7 pieces/m, with a decrease of 57.8%. On the basis of improving the support efficiency of the anchor bolt and cable, the overall support density is significantly reduced.

5.1.2. Improvement of Driving and Support Efficiency

Under the overall low support density, reduce the supporting time in the single row driving-supporting circulation as well as improving the single-row driving speed can weaken the unloading disturbance intensity and disturbance circulation on the roof rock strata.

5.2. Effect Analysis

5.2.1. Control Effect of Roadway Surroundings

(1) Roadway surface displacement

In order to obtain the surface displacement law of the surrounding rock in RAR 20201 under the new driving and support technology, a surface displacement test of the surrounding rock is arranged during roadway excavation. The cross-section measurement method is used to monitor the deformation amount and deformation speed of the surrounding rock, and the results are shown in Figure 14.

As shown in Figure 14a, the roof subsidence increases when the distance from the monitoring station to the heading face increases. When the distance to the heading face is less than 80–100 m, the subsidence basically shows a linear growth trend; when the lag distance is more than 80–100 m, the subsidence increases slowly at first and then tends to be stable, with a total roof subsidence of 21 mm after stability. As the distance to the heading face increases, the driving disturbance is weakened, as is the roof subsidence speed. According to the different deformation speeds, the roof subsidence can be divided into a severe deformation period and a slow deformation stabilization period, whose maximum subsidence speeds are 5 mm/d and 0–2 mm/d, respectively.
Figure 14. Monitoring of roadway surrounding rock deformation and deformation speed: (a) roof subsidence; (b) coal rib convergence.

For the coal rib convergence (Figure 14b), the variation law is similar to the roof subsidence. The coal side convergence is 26 mm after stability; the maximum displacement speed is 7 mm/d during the severe deformation period; the displacement speed is 1–2 mm/d during the slow deformation stabilization period.

Both the roof subsidence and the coal rib convergence are controlled within 30 mm, without obvious deformation (Figure 15). That is, the stability of the surrounding rock is effectively controlled, and the problem of coal rib fall is successfully solved.

Figure 15. Photos of roadway support effect.

(2) Working load of roof anchor cable
The working load of the roof anchor cable, which reflects the bearing state, is one of the important indexes to evaluate the supporting efficiency. In order to obtain the variation law of the roof cable load, a monitoring station for the working cable load is arranged; the installation and monitoring methods are shown in Figure 16a. The monitoring results in Figure 16b show that the cable load increases gradually when the distance to the heading face increases and ultimately tends to be stable.
Figure 16. Roof anchor cable working load monitoring: (a) monitoring stations and methods (reproduced from [30]); (b) monitoring results.

After stability, the cable load value is 249 kN, which is significantly higher than that of the initial pre-tension of 200 kN. It is in a good elastic bearing state (not reaching its yield state), indicating that the deep strata of the roof can be mobilized to participate in the bearing after increasing the thickness of the roof anchorage layer. When the rock mass in the anchorage zone deforms, the anchor cable can quickly increase in resistance to limit the deformation of the rock mass and give better support efficiency.

(3) Separation fracture of roof rock mass

The monitoring results of the roof separation fractures show that there is no obvious separation fracture development within 2.75–5.63 m (Figure 17). The coal body is located at a depth of 2.25 m from the roof, indicating that the stability of the rock above the 2.25 m top-coal is well controlled, and the separation fractures within and outside the anchorage zone are successfully eliminated, which creates a chance for the continuous transmission of roof stress.

Figure 17. Peep chart of separation fractures in roof strata.

5.2.2. Roadway Driving Speed

After adopting the optimized driving and support technology, the support time in each driving-supporting circulation is shortened by about 15 min. The driving speed increased from 390 m/month to 660 m/month, with an increase of 69%. In conclusion, the roadway formation speed was significantly improved, which created a prerequisite for the rapid mining in the working face.
6. Conclusions

In this study, based on the typical engineering case of a deep-buried thick top-coal roadway, the main elements affecting the control effect of the surrounding rock and driving speed are analyzed. The stress distribution characteristics of the surrounding rock in the deep-buried high-stress thick top-coal roadway and the speed effect of the head-on cyclic cutting disturbance are found. The cooperative control countermeasures of efficient driving and support in a deep-buried thick top-coal roadway are proposed and are verified by industrial tests. The results show that the stability of the roadway surrounding rock is effectively controlled and the driving speed is also greatly improved, resulting in high-efficiency driving and supporting collaboration. Conclusions are drawn as follows:

(1) The maintenance effect and existing problems of deep-buried thick top-coal roadways are analyzed, and the research directions are determined as the damaged zone of the roadway surrounding the rock and the driving speed. During driving on the deep-buried thick top-coal roadway, there exist the problems of separation fracture development outside the roof anchorage zone, coal rib fall and slow driving speed. It is believed that under the condition of deep burial, high stress and thick top-coal, the damage range of the roof may exceed the range of the bolt anchorage. Moreover, the development of separation fracture outside the anchorage zone promotes the roof stress transfer to both sides, thereby affecting the stability of the roadway roof and ribs.

(2) The stress distribution characteristics of the surrounding rock in the deep-buried high-stress roadway are studied; furthermore, the fluctuation form and speed effect of the disturbance stress in head-on cutting are revealed. After roadway driving, for a deep-buried high-stress roadway, the low-value stress range of the roof strata and coal rib is 3 m and 2–3 m, which enlarge the damage range compared with shallow buried roadway. The fluctuation curve of unloading disturbance stress produced by head-on cutting conforms to the “power exponential” composite function. Compared with the shallow buried low-stress roadway, the head-on cutting of the deep-buried high-stress roadway causes stronger unloading disturbance intensity, which will gradually decrease with the increase of the driving speed.

(3) The cooperative control countermeasures of efficient driving and support in deep-buried thick top-coal roadway are proposed. On one hand, we improve the support efficiency of the single bolt to reduce the overall support density; meanwhile, we thicken the bolt anchorage layer to allow it to pass through the fracture layer in the shallow part of the roof into the deep rock layer with good stability. Moreover, we eliminate the separation fracture outside the anchorage end to allow the roof stress to continuously transfer downward rather than towards both sides, which helps to control the stability of the roof and rib. On the other hand, under overall reduced support density, the rapid completion of head-on driving-supporting circulation as well as improved driving speed can lessen the unloading disturbance of head-on cutting on the roof strata.

(4) The industrial tests of the cooperative control countermeasures of efficient driving and support in deep-buried thick top-coal roadways are carried out. The results show that the subsidence of the roadway roof and the convergence of the coal ribs are controlled within 30 mm, showing effective deformation control. The working load of the roof anchor cable is 249 kN after stability, with good bearing performance. There is no obvious separation fracture development found in the depth of 2.75–5.63 m of the roof, or for the zone outside the anchorage, which creates a favorable chance for the continuous transmission of roof stress. In addition, the driving speed increases from 390 m/month to 660 m/month, which speeds up the roadway formation.

This paper reveals the deformation and failure mechanism of the surrounding rock of the deep-buried high-stress thick top-coal roadway in terms of insufficient thickness of the roof anchorage layer. The results show that the transmission of the disturbance stress.
produced by the roadway head-on driving is in accordance with the “power exponential” composite function curve. Furthermore, the driving speed effect is also clarified, which solves the problems of poor control effect of the surrounding rock in the deep-buried thick top-coal roadway and low driving speed.

Author Contributions: Data curation, C.H. (Changliang Han); formal analysis, H.Y., C.H. (Changliang Han) and L.W.; funding acquisition, C.H. (Changliang Han), H.Y. and B.Z.; project administration, C.H. (Chengjun Hu); writing—original draft preparation—H.Y. and C.H. (Changliang Han); writing—review and editing, H.Y., C.H. (Chengjun Hu), L.W. and B.Z. All authors have read and agreed to the published version of the manuscript.

Funding: This work was supported by the 65th batch of China Post-doctoral Science Fun (2019M652019), the funder is China Postdoctoral Science Foundation; And this work was also supported by National Natural Science Foundation of China (No. 51404251), the funder is National Natural Science Foundation of China.

Institutional Review Board Statement: Not applicable.

Informed Consent Statement: Not applicable.

Data Availability Statement: Data are contained within the article.

Acknowledgments: The authors are grateful to the reviewers for carefully reading the manuscript and providing valuable suggestions.

Conflicts of Interest: The authors declare no conflict of interest.

Abbreviations

RAR 20201 return air roadway 20201

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