An investigation on the effect of high energy storage anchor on surrounding rock conditions

Bowen Wu, Xiangyu Wang, Jianbiao Bai, Wenda Wu, Ningkang Meng and Huasheng Lin

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Note: Reports are unedited and appear as submitted by the referee. The review history appears in chronological order.

Review History
RSOS-201105.R0 (Original submission)

Review form: Reviewer 1 (Rudrajit Mitra)

Is the manuscript scientifically sound in its present form?
Yes

Are the interpretations and conclusions justified by the results?
Yes

Is the language acceptable?
No

Do you have any ethical concerns with this paper?
No

Have you any concerns about statistical analyses in this paper?
No

Recommendation?
Major revision is needed (please make suggestions in comments)
Comments to the Author(s)
Good study, however, it needs to have some major changes as specified in the annotated paper (Appendix A). Some of the points are not clear in the paper, which needs to be clarified.

Review form: Reviewer 2

Is the manuscript scientifically sound in its present form?
Yes

Are the interpretations and conclusions justified by the results?
Yes

Is the language acceptable?
Yes

Do you have any ethical concerns with this paper?
No

Have you any concerns about statistical analyses in this paper?
No

Recommendation?
Major revision is needed (please make suggestions in comments)

Comments to the Author(s)

This article investigates the influence of high pre-tension on anchor with various surrounding rock strengths under uniaxial compression, aiming to study the different strengthening behaviors of tension rock bolt under various conditions based on monitor of the energy evolution, the stress-strain relation as well as the fracture development in different rock strengths.

The subject of the study is interesting as the results provide a scientific basis for the rational stability control of coal roadway and an improvement in safety of coal mines. It is necessary however to make minor and major revisions before publication of the article.

For the minor revisions, please find attached (Appendix B) the reviewed manuscript with several corrections and comments.

Regarding the major revisions:

6.1 Mechanical properties and damage analysis
Table 8 presents the degree of damage for each rock type, with and without anchorage, at a particular strain level of 1.2%. It is not clear to the reader the reasons that the authors chose this level for damage degree comparison and not for example that one corresponds to the peak stress or any other strain level.
Please explain and clarify.

6.2 Analysis of the law of energy evolution
Page 7, lines 28-29:
Fig. 5 shows that the degree of damage increasing in Stage 1 (before the yielding strength). Please comment and clarify on this.
Page 7, lines 34-35:
This is in contrast to Fig. 5. Fig. 5 shows that the degree of damage is increasing at Stage 2. Please comment and clarify on this.

7 Conclusions
In this section there are some points that are mentioned extensively more than one time in Discussion section. For example, the point in page 8, lines 42-43 is repeated in page 7, lines 1-2 and 9-10 as well as in page 8, lines 16-17.
The authors should take care and reform properly the Discussion and the Conclusion section.

Regards

Decision letter (RSOS-201105.R0)

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Associate Editor Comments to Author (Professor Zach Agioutantis):
Associate Editor: 1
Comments to the Author:
Please document all changes to the original document.

Reviewer comments to Author:
Reviewer: 1

Comments to the Author(s)
Good study, however, it needs to have some major changes as specified in the annotated paper. Some of the points are not clear in the paper, which needs to be clarified.

Reviewer: 2

Comments to the Author(s)
Comments to the Editor - Authors

This article investigates the influence of high pre-tension on anchor with various surrounding rock strengths under uniaxial compression, aiming to study the different strengthening behaviors of tension rock bolt under various conditions based on monitor of the energy evolution, the stress-strain relation as well as the fracture development in different rock strengths. The subject of the study is interesting as the results provide a scientific basis for the rational stability control of coal roadway and an improvement in safety of coal mines. It is necessary however to make minor and major revisions before publication of the article.

For the minor revisions, please find attached the reviewed manuscript with several corrections and comments.

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Author’s Response to Decision Letter for (RSOS-201105.R0)

See Appendix C.

Decision letter (RSOS-201105.R1)

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openscience@royalsociety.org

on behalf of Professor Zach Agioutantis (Associate Editor) and R. Kerry Rowe (Subject Editor)
openscience@royalsociety.org

Associate Editor Comments to Author (Professor Zach Agioutantis):

The authors have revised the document according to the reviewers comments. However, there are still a number of typos and odd expressions in the document

a) please correct "targe" in Table 5
b) Is "tension" in Tables 2,3,4 "tensile strength"?
c) rock mass does not have plural form. Please correct rock masses to rock mass
d) the sentence "The installation of tension rock bolts was more suitable for medium hard rock (e.g. sandy mudstone), whereas it was not effective for hard rock (e.g. sandstone)" is repeated in the summary text twice.
e) and fracture development of anchor was monitored by FISH function. - why are you talking about fractures of the anchor?
f) What is a Coulomb-sliding model. That is not a standard term.

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-- An editable file of each table (.doc, .docx, .xls, .xlsx, or .csv).
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Author's Response to Decision Letter for (RSOS-201105.R1)

See Appendix D.

Decision letter (RSOS-201105.R2)

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Thank you for your fine contribution. On behalf of the Editors of Royal Society Open Science, we look forward to your continued contributions to the Journal.

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Andrew Dunn
Royal Society Open Science Editorial Office
Royal Society Open Science
openscience@royalsociety.org

on behalf of Professor Zach Agioutantis (Associate Editor) and R. Kerry Rowe (Subject Editor)
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### Appendix A

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An Investigation on the effect of high energy storage anchor on surrounding rock conditions

| Journal: | Royal Society Open Science |
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| Manuscript ID | RSOS-201105 |
| Article Type: | Research |
| Date Submitted by the Author: | 26-Jun-2020 |
| Complete List of Authors: | Wu, Bowen; China University of Mining and Technology; State Key Laboratory of Coal Resources and Safe Mining \ Wang, Xiang-yu; China University of Mining and Technology; State Key Laboratory of Coal Resources and Safe Mining \ Bai, Jianbiao; State Key Laboratory of Coal Resources and Safe Mining \ Wu, Wenda; China University of Mining and Technology \ Meng, Ningkang; China University of Mining and Technology, School of Mines \ Lin, Huasheng; University of New South Wales |
| Subject: | Energy < ENGINEERING AND TECHNOLOGY, Civil engineering < ENGINEERING AND TECHNOLOGY |
| Keywords: | Energy evolution, tension bolts, UDEC, differential analysis, rock |
| Subject Category: | Engineering |
**Author-supplied statements**

Relevant information will appear here if provided.

**Ethics**

*Does your article include research that required ethical approval or permits?:*

This article does not present research with ethical considerations

*Statement (if applicable):*

CUST_IF_YES_ETHICS :No data available.

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Our data are deposited at Dryad.

Review URL: https://datadryad.org/stash/share/Agfg6O9idvPjujUn9zqr8m79qBwHC6MF69F5n4fWk8

doi:10.5061/dryad.cvdncjt1w

**Conflict of interest**

I/We declare we have no competing interests

*Statement (if applicable):*

We have no competing interests.

**Authors’ contributions**

This paper has multiple authors and our individual contributions were as below

*Statement (if applicable):*

B.W. and X.W. conceived the article structure. B.W. and W.W. performed the rock mechanical experiments. J.B. developed energy balance criterion. B.W. numerical simulation. All the authors analysed the data. N.M. prepared the plots. H.L. language revision. All the authors prepared the initial draft and revised the paper.
An Investigation on the effect of high energy storage anchor on surrounding rock conditions

Bowen Wu 1,2, Xiangyu Wang 1,2 *, Jianbiao Bai 2, Wenda Wu 1,2, Ningkang Meng 1,2, Huasheng Lin 3

1 School of Mines, China University of Mining & Technology, Xuzhou 221116, China
2 State Key Laboratory of Coal Resources and Safe Mining, Xuzhou 221116, China
3 School of Minerals and Energy Resources Engineering, University of New South Wales, Sydney, NSW, 2052, Australia

Keywords: Energy evolution, tension bolts, UDEC, differential analysis, rock

1. Summary

High pre-tension bolt is an effective strata control technique and is the key to ensure the stability of anchorage and roadway. Based on the performances of high energy storage tension rock bolts in different rock properties, this study proposed a constitutive model to describe the energy balance of anchor under uniaxial compression. UDEC was used to simulate the behaviour of anchor in coal under uniaxial compression and results were analysed to study the rock mechanical properties, degree of damage and energy evolution. Simulation results showed that tension rock bolts can improve the mechanical properties and energy storage capacities of anchor. The energy evolution was divided into three stages: i) the external work was stored in the form of elastic strain energy ($U_e$) in the anchor prior to the yielding strength; ii) the elastic strain energy reached its maximum near the peak strength; iii) Energy was dissipated from fracture friction ($W_f$), plastic deformation ($W_p$) and acoustic emission ($U_r$) during post-peak stage. The installation of tension rock bolts resulted in the most improvement of surrounding rock condition for medium hard rock including sandy mudstone, whereas it had the least impact on hard rock, e.g. sandstone.

2. Introduction

Rock strength is a critical factor for underground structural design. To improve the mechanical properties of rock, rock and cable bolts are generally implemented. In underground coal mines, the rock strengthening measure is immediate bolt support after excavation to ensure the roadway stability and subsequent workplace safety [1]. Extensive practical work revealed that this is an effective strata control method, which has also been widely used in Chinese coal mines [2].

To determine the effect of rock bolt on anchor strengthening and improve the ability of anchorage, a significant body of researches have been carried out. Hou [3] theoretically analysed the effect of rock bolts on the peak and the residual strengths of rock around anchorage range. Based on the improvement of mechanical properties prior and after the peak strength, Hou [3] proposed a constitutive model for rock strengthening effect around the roadway with rock bolts. Kang et al. [4] described the mechanisms and technologies of rock bolt supporting system in China. The authors suggested that high strength-stiffness-reliability and low-density proactive rock bolts can improve the strength of surrounding rock. Wang et al. [5] derived the equations of shear strength prior and after anchorage. The analysis showed that once the grouting is completed, the shear strength of anchorage range was improved together with the mechanical properties of surrounding rock (elastic modulus, cohesion, friction angle and dilation angle). Wei et al. [6] carried out physical simulation tests to study the influence of pre-tension on anchor strength and deformation and found that high pre-tension can improve the anchorage strength. Based on the triaxial physical simulation tests,
Wang et al. [7] revealed that the rock bolt does not only improve the capacity of fracture surrounding rock, but also controls the discontinuous deformation of fractured surrounding rock. This will in turn improve the continuity and integrity of surrounding rock mass of deep roadway.

In numerical simulation, the bolt section in anchorage range can be represented by material model as well as structural elements. Both methods have been proven successful in simulating the mechanical properties of bolts and anchors under various loading conditions. Meng [8] imported the constitutive model of anchor in fractured rock to FLAC3D and subsequently simulated the effect of anchoring condition on ultimate bearing capacity. Results showed that the higher pre-tension and density can improve the ultimate bearing capacity. Zuo et al. [9] used elastic-isotropic model in FLAC3D to study the influence of the alignment between bolt and borehole diameters on anchorage numerical. Numerical simulation suggested a reasonable alignment (4 – 11 mm diameter difference) can improve the anchorage effect. D.-A. Ho et al. [10] used elasto-plastic model to study the fracture mechanism of rock bolt and anchoring agent under various horizontal stress conditions. Thereby, Grasselli [11], Aziz and Jalalifar [12] and Tatone et al. [13] also used material model to simulate various laboratory condition of rock bolt. However, the mesh size is an important factor which can influence the simulation results using material model. This constrains the preciseness of the simulation. On the other hand, structural element approach can overcome this problem. At present, the structural element can be divided into ‘cable element’ and ‘rockbolt element’. Zhang et al. [14] used cable element to study the parameters including length, anchorage length and rock anchor spacing via FLAC3D and proposed several suggestions to support bolt design. Wu et al. [15] attempted to simulate the supporting element (e.g. wooden cylinder) in supported roadway using Cable element and found that the rock bolts can improve the capacity of wooden cylinder and prevent the fracture development. Pull-test or underground excavation with rock bolt support were simulated by (Vardakos et al. [16], Malmgren and Nordlund [17], Li et al. [18], Gao et al. [19], Shreedharan and Kulatilake [20]). Ma et al. [21] used rockbolt element to simulate pull-test via FLAC2D and studied the relationship between rock bolt and rock as well as the mechanical properties of newly developed rock bolt and its preferable conditions.

According to the previous studies, it can be found that the focus was mainly on the anchorage technique, rock bolt, coordination effect and stress transfer. Although there is an overall agreement that high pre-tension can improve the mechanical properties of anchor, the influence of high pre-tension under various rock strength conditions on anchorage was not investigated. Majority of the research analysed the anchorage effect via elasto-plastic model and stress-strain relationship, while there were limited studies investigated the problem using energy conservation and the fracture evolution of anchorage is a process of energy dissipation and release. Hence, it is more reasonable to study the anchorage effect using energy law (Xie et al.[22]). Many numerical simulation use cable element to approach this topic; this is inappropriate considering the method is one-dimensional element which consists of two degrees of freedom. On the other hand, rockbolt element is more suitable for the analysis as it is a two-dimensional element with three degrees of freedom (Itasca Consulting Group Inc. [23]). The paper used rockbolt element through UDEC Trigon to study the influence of high pre-tension on anchor with various surrounding rock strengths under uniaxial compression. This aims to study the different strengthening behaviours of tension rock bolt under various conditions based on monitor of the energy evolution, the stress-strain relation as well as the fracture development in different rock strengths.

### 3. Energy balance and components

Nowadays, the study of energy evolution in rock mechanics has been refined to different depth of cover, loading conditions, in-situ stress magnitudes (Salamon [24], Napier [25], Onur Vardar et al. [26], Zhang et al. [27], Wang et al. [28]). In UDEC, the energy exchanged based on the work done on rock, joint and boundary. Without considering dynamic calculation, the energy balance in UDEC mainly involves work done on boundary, stored strain energy and dissipated energy (Itasca Consulting Group Inc. [23]). This study determined the constitutive law based on the previous studies as well as the simulation conditions.

In uniaxial compression, work done from external force is denoted by $W$, $U$ is used to represent the elastic strain energy due to elastic deformation of the anchor. The difference in work done between boundary and elastic strain energy is dissipated energy, $U^d$. Hence, total input energy can be written as:

$$ W = U^e + U^d \quad (1) $$

Energy is mainly dissipated in three ways. Firstly, energy is dissipated by friction due to the fracture development in rock; this is denoted as $W_f$. Second source is due to the plastic deformation of rock ($W_p$). Once the rock is undergone plastic deformation, energy dissipated from plastic work done. The rest of energy is dissipated via acoustic emission, denoted as $U$. Therefore, $U^d$ can be expressed as:
In UDEC, the incremental change of these energy components is cumulative and determined by timesteps. By combining Eq (1) and Eq (2), $W$ can be expressed as:

$$W = W_f + W_p + U' + U''$$  \hspace{1cm} (3)

Subsequently, this energy balance equation can be used to investigate the complex energy dissipation process in anchor.

### 4. Parameters calibration

#### 4.1. The UDEC Trigon approach

UDEC Trigon model was proposed by Gao and Stead [29], aiming at simulating brittle fracture of rock. In this model, a rock is represented by an assembly of triangular blocks bonded together via the grain contacts. Each block is made elastically by dividing them into triangular finite difference zones. In the direction normal to a contact, the stress-displacement relation is assumed to be linear and governed by the stiffness $k_n$, such that:

$$\Delta \sigma_n = -k_n \Delta u_n$$  \hspace{1cm} (4)

where $\Delta \sigma_n$ is the effective normal stress increment and $\Delta u_n$ is the normal displacement increment. A limiting tensile strength ($T$) is assumed for the contact. If this value is exceeded, then $\sigma_n = 0$.

In the shear direction, the response is governed by constant shear stiffness. The shear stress, $\tau_s$, is determined by a combination of contact properties, cohesion ($c$) and friction angle ($\phi$), where:

$$|\tau| \leq c + \sigma_n \tan \phi = \tau_{max}$$  \hspace{1cm} (5)

then

$$V\tau_{max} - k_s \Delta u_s$$  \hspace{1cm} (6)

however, if $|\tau| \geq \tau_{max}$, then:

$$\tau_s = \text{sign}(\Delta u_s) \tau_{max}$$  \hspace{1cm} (7)

where $\Delta u_s$ is the elastic component of the incremental shear displacement and $\Delta u_i$ is the total incremental shear displacement.

The proposed modeling approach has been implemented in UDEC (Itasca Consulting Group Inc. [23]).

#### 4.2 Mechanical parameters of coal and rock mass

Since blocks in UDEC Trigon model are considered as elastic materials, they cannot be plastically destroyed. However, the anchor breakage is plastic damage. To closely mimic the mechanical properties and energy evolution of anchor post peak strength, this study also used strain softening model in UDEC. The strain-softening model is based on the UDEC Mohr–Coulomb model with non-associated shear and associated tension flow rules.

This study selected coal (soft), sandy mudstone (medium) and sandstone (strong) to represent various rock strengths. The intact properties of the rock masses are listed in Table 1. These properties were obtained through laboratory compression tests and were provided by the Yuwu coal mine. The RQD values of the rock masses were evaluated from borehole televiewer images.

| Table 1 | Intact rock properties and scaled rock mass properties of coal measures from Yuwu coal mine |

The rock mass elastic modulus was calculated using the relationship between RQD and the elastic modulus ratio ($Z$), as shown in Eq. 8, where $E_m$ is the elastic modulus of the rock mass and $E_i$ is the elastic modulus of the rock sample.

$$E_m/E_i = 10^{0.0138\text{RQD}-1.91}$$  \hspace{1cm} (8)

The rock mass strength was then calculated using the relation between the unconfined compressive strength ratio $\sigma_{oc}/\sigma_c$ and the deformation modulus ratio $E_m/E_i$ (Singh and Seshagiri Rao [31]). The value of $q$ is 0.63 (Gao. et al, [32]):

$$\frac{\sigma_{oc}}{\sigma_c} = \left(\frac{E_m}{E_i}\right)^q$$  \hspace{1cm} (9)

To represent the coal and rock by using an assembly of triangular blocks, the properties of the blocks and contacts were calibrated against the coal measure and rock mass properties listed in Table 2 to 4. This was
achieved by simulating unconfined compression tests in a numerical model created using the Trigon logic. The size of the rock sample is 1 m (in width) × 2 m (in height) (Singh and Seshagiri Rao [31]) (Fig. 1). The bottom of the numerical model was fixed and a loading rate of 0.02 m/s was applied at the top. The calibrated properties of the UDEC model are illustrated in Table 5.

The uniaxial compressive strength and elastic modulus data derived by numerical simulation are similar to the data obtained from laboratory tests (within an error of 9%). Hence, the micromechanical parameters of the coal mass and the rock mass were properly calibrated.

Table 2 Calibrated mechanical parameters of blocks and joints of the coal measures

Table 3 Calibrated mechanical parameters of blocks and joints of sandy mudstone

Table 4 Calibrated mechanical parameters of blocks and joints of sandstone

Table 5 Calibrated microproperties in the UDEC Trigon model to represent the rock masses

Fig.1 Results of uniaxial compressive strength tests a numerical simulation of uniaxial compression, b stress–strain curve of coal, c stress–strain curve of sandy mudstone, d stress–strain curve of sandstone

3.3 Rockbolt element parameter

In UDEC, rockbolt element is different from cable element. Rockbolt element is a two-dimensional element which its two nodes have three degrees of freedom (two displacements and one rotation). It can resist bending and can yield along axial direction. Fig. 2 illustrates the components of rockbolt element. Rockbolts interact with UDEC via shear and normal coupling springs, which the springs are non-linear connectors. These are used to transfer force and motion between rockbolt element and the mesh nodes. Fig. 2 shows material behavior of shear and normal coupling spring for rockbolt elements. Shear coupling springs are used to simulate the shear behaviour of anchorage length and normal coupling spring are used to simulate the compression of surrounding rock.

Fig.2 Conceptual mechanical representation of the rockbolt element

Rockbolt element can be broken at the node. The bolt breakage can be simulated by defining the tensile failure strain limit ($t_{fstrain}$). Rock bolt will be deemed as failed if $\varepsilon_{pl} \geq t_{fstrain}$, where $t_{fstrain}$ has to be defined. $\varepsilon_{pl}$ is the total tensile yield strain of any element of anchor:

$$\varepsilon_{pl} = \frac{d}{2} \sum \theta_{pl} + \sum \varepsilon_{ax}$$

(10)

$\varepsilon_{ax}$ is the axial deformation; d is rock bolt diameter, $\theta_{pl}$ is the average rotation angle of the component.

This study used rockbolt element to simulate HRB335 threaded steel anchor at 22 mm diameter. Based on the laboratory pull-test results, the input parameters of rockbolt element can be obtained, as displayed in Table 6.

Table 6 Rockbolt and anchor parameters

5. Research background and model establishment

The common way to apply pre-tension is to install the nut on the tail thread of rock bolt. By applying torque to the nut, axial tension is acting onto the rock bolt (Li et al. [33]). However, high-pretension is difficult to be controlled in practice. For instance, when the torque achieves 300 – 400 Nm, the pre-tension can sometimes still be low. This low conversion efficiency is due to the torque at thread. To increase the pre-tension of rock bolt, a new type of anchor lock has been developed by the research team. This development changed the traditional pre-tension method, which is capable of applying high pre-tension without damaging rock bolt. The anchor lock is composed of an anchor ring and three clips. The anchor ring is a ring-shaped structure, in which the center hole is tapered. The clips are disposed between the inner wall of the tapered hole of the anchor ring and the anchor body, as displayed in Fig. 3.

Fig.3 Rockbolt barrel and wedge and assembly drawing

Fig. 4 shows the setup of UDEC Trigon simulation. The blocks and interfaces used strain softening and Coulomb-sliding models, respectively. The dimensions of the numerical model are 1 m × 2 m. Due to the
limitation of rockbolt element, the effect of vertically installed rock bolts was not significant. Hence, rock bolts were installed horizontally to the rock body at \( y = 0.4m, 1.0m \) and \( 1.6m \). Pre-tension was applied by applying load at the nodes of the rock bolts. In practice, the common torque applied to the rock bolt is approximately 300 – 400 Nm, which are 40 – 50 kN pre-tension correspondingly (Kang et al. [34]). To achieve high pre-tension in the numerical model, the study set the pre-tension as 70% of the yielding strength, i.e. 138.6 kN/0.365 MPa. The loading rate of 0.02 m/s was applied at the top boundary of the model while fixing the bottom boundary. During the simulation process, the stress-strain relationship and fracture development of anchor was monitored by FISH function. Boundary, frictional and plastic work were recorded using UDEC energy element. ‘SET energy on’ command was used to activate the energy monitoring function while turn off ‘mass-scaling’. As this study used non-viscous boundary which does not consider the dynamic calculation, damping was set as ‘auto’. The simulation procedures involved: i) model construction and calibration; ii) test simulation.

### Table 7 Numerical simulation scheme

| Fig.4 | Model overview and dimensions |

| 6. Discussion |

#### 6.1 Mechanical properties and damage analysis

To monitor the degree of damage of rock and anchor, the total fracture length as well as the shear and tensile fracture lengths during uniaxial compression were measured. The anchor and surrounding rock fracture are a complex process, which the shear and tensile failures can occur in a single fracture. Hence, only the initial (first) failure type was recorded for each fracture. The degree of damage \( (D) \) can be calculated as:

\[
D = \frac{L_s + L_t}{L_c} \times 100\% \tag{11}
\]

where \( L_c \) is the total fracture length, \( L_s \) and \( L_t \) are total shear and tensile fracture lengths, respectively.

Fig.5 The stress and degree of damage of rock with and without anchorage: a coal without anchorage; b coal with anchorage; c sandy mudstone without anchorage; d sandy mudstone with anchorage; e sandstone without anchorage; f sandstone with anchorage

Fig. 5 illustrates the stress and degree of damage of rock during the process with and without anchorage. Based on the comparison, the mechanical properties (peak strength, residual strength and elastic modulus) of rock were increased with the installation of tension rock bolts, while the degree of damage decreased. Under the same axial strain, the higher strength always led to higher degree of damage regardless of the anchorage condition. The installation of tension rock bolts did not have significant influence on the axial strain of rock during damage. There was increase in the degree of damage prior to the peak rock strength and the gradient became flatter when it was near the peak strength. The indication here is that installation of tension bolts can prevent the fracture development and coalescence, such that the degree of damage was reduced. To compare the effect of tension rock bolts under different rock conditions, Table 8 summaries the critical parameters from Fig. 5.

Table 8 Mechanical properties and degree of damage of rock prior and after anchorage

Table 8 suggested that there are differences in the strengthening effect of tension rock bolts on rock mechanical properties with various rock types. When the bolt was installed in coal, elastic modulus, peak strength and residual strength increased 4.1% (0.03 GPa), 5.9% (0.3 MPa) and 5.2% (0.2 MPa), while the degree of damage decreased 13.7%. In sandy mudstone, elastic modulus, peak strength and residual strength increased 20.7% (0.6 GPa), 3.6% (0.9 MPa) and 13.1% (1.7MPa), while the degree of damage decreased 11.9 %. For sandstone, elastic modulus, peak strength and residual strength increased 21.6% (0.1 MPa), 0.6% (0.2 MPa) and 0.5% (0.1 MPa), while the degree of damage decreased 5.3 %. Sandy mudstone had the most change in absolute magnitudes and sandstone had the least change. In terms of the relative magnitude difference, coal had the most change whereas sandstone had the least. The absolute magnitude change of sandy mudstone was the most while its degree of damage was only in the middle, indicating the mechanical properties of rock was improved by installation of rock bolts. However, this is not the primary reason for the reduction in degree of damage, in which the main reason was due to the prevention of fracture development and coalescence using rock bolts. Sandstone had the low change in terms of absolute and relative magnitudes. This suggests that high pre-tension rock bolts do not have significant influence on strong rock. This is because the rock strength was high and it required substantially higher stress to induce the rock failure. Under this condition, the effect of rock bolt is negligible comparing with the high strength. At post-peak of the loading process, the
fracture was more likely to be developed due to the low plasticity, which was difficult to be controlled using rock bolts. Overall, it can be concluded that tension rock bolts have insignificant influence on strong rock, such that the high pre-tension is not recommended for hard rock support design.

### 6.2 Analysis of the law of energy evolution

During the loading process, rock was continuously deforming and storing elastic strain energy. At the same time, the energy was dissipated due to internal damage. When the elastic strain energy reached the limit, it was released and led to rock failure. Hence, the failure process of rock is closely related to the energy evolution. Based on the analysis carried out in Section 5.1, tension rock bolts are not suitable for hard rock. Therefore, this section will focus on the energy evolution of coal and sandy mudstone, as displayed in Fig. 6 and 7.

In Fig. 6 and 7, the energy dissipation due to acoustic emission \( (U_{AE}) \) was illustrated solely. This is because the energy dissipation curve of acoustic emission was similar to the accumulative acoustic emission, which is a major indication of anchor breakage. Secondly, the energy dissipation due to acoustic emission was substantially lower than the other sources. A sole illustration can represent the energy evolution of acoustic emission more clearly. According to the results from Fig. 6 and 7, the energy evolution of rock was consistent regardless of the rock bolts. Based on the correspondence between stress curve and energy curve, the energy evolution of rock can be divided into three stages:

**Stage 1:** From the start of the load to the yielding strength. External work was fully absorbed by rock and stored in the form of elastic strain energy. The energy curve increased non-linearly during this stage, while the external work curve was well aligned with the elastic strain curve. Based on the stress and energy curves, the rock was under elastic deformation prior to the yielding strength. There was no observable internal damage, such that the internal energy dissipation was considered as none.

**Stage 2:** From the yielding strength to the peak strength. External work continuously increased and the total energy that the rock absorbed was also increasing. However, as the rock was in plastic condition and there was some degree of internal damage, the frictional work in-between cracks and plastic deformation started to occur and increased at a high rate. This indicates that the degree of damage was decreased although the acoustic emission level was still low. During this stage, the elastic strain energy was also increasing but the growth rate was flatter. Thereby, its alignment with the external work was getting worse and reached its maximum near the peak strength point.

**Stage 3:** Post-peak stage. External work continuously increased but the elastic energy in the rock was released at a high rate. There was considerable increase in the energy dissipation due to frictional work, plastic deformation and acoustic emission. The acoustic emission energy curve was growing rapidly, suggesting that the degree of damage was high. The external work was mainly transferred to the energy require for rock fracture towards the end of the simulation and the elastic strain energy was not stored anymore. When the stress dropped to the residual strength, the strain energy and energy dissipation curves became flatter, although the energy continuously dissipated.

Although the energy evolution of rock was consistent regardless of the existence of rock bolts, the degree of damage and other parameters changed noticeably with the installation of rock bolts. Firstly, there were increasing in stress and decreasing in strain at anchor damage with rock bolts. For coal, the stress increased from 4.4 MPa to 4.8 MPa and strain reduced from 0.57% to 0.5%. For sandy mudstone, stress increased from 18 MPa to 20.6 MPa and strain reduced from 0.6% to 0.49%. Secondly, the peak elastic energy increased from 29.2 kJ to 35 kJ and from 202.7 kJ to 240.7 kJ for coal and sandy mudstone, respectively. When strain reached 1.2%, the external work of coal and sandy mudstone increased 26.2 kJ and 33.7 kJ. For coal, the work done from fracture friction was 35.1 kJ without rock bolt, accounting for 50.1% of the total input energy; whereas the energy was 37.4 kJ and took account of 38.9% with rock bolt. On the other hand, for sandy mudstone, the work done from fracture friction was 230.3 kJ without rock bolt, accounting for 58.8% of the total input energy; whereas the energy was 158.7 kJ and took account of 37.2% with rock bolt. Based on aforementioned data, the installation of tension rock bolts can improve the strength of surrounding rock and its storage of elastic strain energy. At the same time, it can also prevent fracture development and postpone the elastic strain energy release at post-peak. Fig. 8 shows the comparison of each energy categories. From the figure, it can be seen that the performance of sandy mudstone was better than coal. Hence, the indication here is that the tension...
rock bolts can improve the mechanical properties and energy storage capacity of rock, which agrees with the conclusion from Section 6.1. Thereby, the most suitable rock type for the application is medium hard rock, e.g. sandy mudstone.

Fig. 8 The comparison between coal and sandy mudstone on different energy categories

Fig. 9 The stress and energy evolution of sandstone with and without anchorage; a stress-strain relationship curve of sandstone (without anchorage); b stress-strain relationship curve of sandstone (with anchorage); c energy evolution of sandstone (without anchorage); d energy evolution of sandy mudstone (with anchorage)

Fig. 9 depicts the stress and energy evolution of sandstone with and without anchorage. According to the figure, it can be found that the relationships of sandstone follow the same trend as those of coal. The difference between sandstone and coal was that the changes of frictional work and plastic deformation of sandstone were insignificant with rock bolts, indicating that the installation of rock bolts does not increase the plasticity of sandstone nor prevent fracture development. The plastic deformation curve was always at low level while the frictional work curve was always at high level. This phenomenon again supports that the fracture development was the main reason of rock failure. This agrees with the conclusion from stress and damage analysis; thereby proved that tension rock bolts are not effective for hard rock roadway support, e.g. sandstone.

7 Conclusion

Based on the UDEC simulation on the strengthening effect of tension rock bolts in different rock properties, a number of parameters were studied, including mechanical properties, degree of damage and energy evolution. The following conclusions were drawn from the numerical results:

(1) Tension rock bolts can improve the mechanical properties and energy storage capacity of anchor. Subsequently, it can reduce the degree of damage of the anchor while increasing the elastic strain energy storage and plastic deformation work. By suppressing energy dissipation due to fracture friction and delaying the post-peak elastic strain release, the stability of anchor can be increased.

(2) Based on the stress-strain and energy dissipation-strain relationships during uniaxial compression, the energy evolution of anchor was divided into three stages. In the first stage, the external work was fully absorbed by rock and was stored as elastic deformation energy. In the second stage, the axial stress surpasses the rock yielding strength, which work was done from fracture friction and plastic deformation. At the same time, the growth rate of elastic strain energy was reduced and the elastic strain energy reached the maximum value near the peak strength point. In the third stage, the energy stored in the rock was rapidly released from elastic strain energy. The work from fracture friction, plastic deformation and acoustic emission increased substantially. At the end of the process, the energy of external work was mainly transferred to rock failure, where the energy was dissipated via fracture friction, plastic deformation and acoustic emission. Among them, work from fracture friction and plastic deformation was the major contributing factor for energy dissipation.

(3) Based on stress and energy evolution analysis on different rock types, it can be seen that the tension rock bolts were not effective for hard rock. The elastic modulus, peak strength and residual strength increased 2.1% (0.1 GPa), 0.6% (0.2 MPa) and 0.5% (0.1 MPa), respectively. The implementation could not prevent the fracture development within the rock mass, such that it was not recommended for hard rock supporting design.

(4) The absolute magnitudes of mechanical properties of sandy mudstone increased more than that of coal. However, the relative magnitudes and reduction of degree of damage was lower than coal. This indicates that the improvement of mechanical properties of rock due to rock bolts was the secondary factor for degree of damage reduction, whereas its effect on preventing fracture development and coalescence was the main contributing factor. For energy evolution, the stress at initial damage increased 14.4% (2.6 MPa) and 9% (0.4 MPa) for sandy mudstone and coal. In addition, the peak elastic strain energy increased 18.7% (38 kJ) and 9.6% (5.8 kJ) for sandy mudstone and coal. The work done due to fracture friction decreased from 58.8% to 37.2% of the total energy for sandy mudstone; and it decreased from 50.1% to 38.9% for coal. Therefore, tension rock bolts is more suitable for medium hard rock (e.g. sandy mudstone), followed by soft rock (e.g. coal).

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Competing Interests
We have no competing interests.

Authors’ Contributions
B.W. and X.W. conceived the article structure. B.W. and W.W. performed the rock mechanical experiments. J.B. developed energy balance criterion. B.W. numerical simulation. All the authors analysed the data. N.M. prepared the plots. H.L. language revision. All the authors prepared the initial draft and revised the paper.

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Tables

| Lithology          | $E_r$ (GPa) | $b_r$ (MPa) | RQD | $E_m$ (GPa) | $b_{cm}$ (MPa) | $b_{tm}$ (MPa) |
|--------------------|-------------|-------------|-----|-------------|----------------|----------------|
| Coal               | 2.6         | 10.8        | 75% | 0.79        | 5.1            | 0.51           |
| Sandy mudstone     | 5.4         | 35.5        | 90% | 3.1         | 25             | 2.5            |
| Sandstone          | 9.7         | 53.5        | 88% | 5.2         | 36.1           | 3.6            |

Table 2 Calibrated mechanical parameters of blocks and joints of the coal measures

| Density (kg/m$^3$) | Young’s modulus (GPa) | Cohesion (MPa) | Friction ($\degree$) | Tension (MPa) |
|--------------------|------------------------|----------------|----------------------|---------------|
| Block              | 1400                   | 0.79           | 1.6 ($\varepsilon_p=0$) | 27            | 0.9           |

| Normal stiffness (GPa) | Tangential stiffness (GPa) | Cohesion (MPa) | Friction ($\degree$) | Tension (MPa) |
|------------------------|----------------------------|----------------|----------------------|---------------|
| 113                    | 45.2                      | 1.3            | 18                   | 0.4           |

Table 3 Calibrated mechanical parameters of blocks and joints of sandy mudstone

| Density (kg/m$^3$) | Young’s modulus (GPa) | Cohesion (MPa) | Friction ($\degree$) | Tension (MPa) |
|--------------------|------------------------|----------------|----------------------|---------------|
| Block              | 1800                   | 3.1            | 8.0 ($\varepsilon_p=0$) | 27            | 2.5           |

| Normal stiffness (GPa) | Tangential stiffness (GPa) | Cohesion (MPa) | Friction ($\degree$) | Tension (MPa) |
|------------------------|----------------------------|----------------|----------------------|---------------|
| 372                    | 149                       | 6.2            | 18                   | 1.8           |

Table 4 Calibrated mechanical parameters of blocks and joints of sandstone

| Density (kg/m$^3$) | Young’s modulus (GPa) | Cohesion (MPa) | Friction ($\degree$) | Tension (MPa) |
|--------------------|------------------------|----------------|----------------------|---------------|
| Block              | 2550                   | 5.2            | 13.0 ($\varepsilon_p=0$)| 30            | 2.5           |

| Normal stiffness (GPa) | Tangential stiffness (GPa) | Cohesion (MPa) | Friction ($\degree$) | Tension (MPa) |
|------------------------|----------------------------|----------------|----------------------|---------------|
| 665                    | 266                       | 11.2           | 18                   | 1.8           |

Table 5 Calibrated microproperties in the UDEC Trigon model to represent the rock masses

| Lithology          | Young’s modulus (GPa) | Compressive strength (MPa) |
|--------------------|-----------------------|----------------------------|
|                     | Target                | Calibrate | Error (%)   | Target | Calibrate | Error (%) |
| Coal                | 0.79                  | 0.73       | 7           | 5.1    | 5.1       | 0         |
| Sandy mudstone     | 3.1                   | 2.9        | 6           | 25     | 25.1      | 0.4       |
| Sandstone          | 5.2                   | 4.7        | 9           | 36.1   | 36.3      | 5         |
Table 6 Rockbolt and anchor parameters

|            | Cross-section area (m²) | Elastic modulus (GPa) | Tensile yield strength (kN) | Second moment of area(m⁴) | tf strain |
|------------|-------------------------|-----------------------|----------------------------|---------------------------|----------|
| Rockbolt   | 3.8 × 10⁻⁴              | 200                   | 198                        | 1.2 × 10⁻⁸                | 0.012    |

|            | Exposed perimeter (m)  | Cohesive strength of shear coupling spring (MPa) | Stiffness of shear coupling spring (GPa) | Frictional resistance of the shear coupling spring (°) | Cohesive strength of normal coupling spring (MPa) | Stiffness of normal coupling spring (GPa) | Frictional resistance of the normal coupling spring (°) |
|------------|-------------------------|--------------------------------------------------|------------------------------------------|--------------------------------------------------------|-----------------------------------------------|---------------------------------------------|--------------------------------------------------------|
| Anchor parameters | 0.07                | 1                                                | 8                                        | 45                                                     | 200                                           | 20                                          | 0                                                     |

Table 7 Numerical simulation scheme

| Rockbolt existence | Pre-tension (load) | Loading rate |
|--------------------|--------------------|--------------|
| Coal (soft)        | No                 | /            | 0.02m/s        |
| Sandy mudstone (medium) | Yes              | 0.365MPa     | 0.02m/s        |
| No                 | /                  |              | 0.02m/s        |
| Sandstone (strong) | Yes                | 0.365MPa     | 0.02m/s        |
| No                 | /                  |              | 0.02m/s        |

Table 8 Mechanical properties and degree of damage of rock prior and after anchorage

| Rock bolt existence | Elastic modulus (GPa) | Peak strength (MPa) | Residual strength (MPa) | Degree of damage (%) |
|---------------------|-----------------------|--------------------|------------------------|----------------------|
| Coal (soft)         | No                    | 0.73               | 5.1                    | 3.8                  | 35.1                |
| Sandy mudstone      | Yes                   | 0.76               | 5.4                    | 4.0                  | 30.3                |
| (medium)            | No                    | 2.9                | 25.1                   | 13                   | 46.2                |
| Sandstone (strong)  | Yes                   | 3.5                | 26                     | 14.7                 | 40.7                |
|                     | No                    | 4.7                | 36.3                   | 20.2                 | 50.8                |
|                     | Yes                   | 4.8                | 36.5                   | 20.3                 | 48.1                |

Figure and table captions

Table 1 Intact rock properties and scaled rock mass properties of coal measures from Yuwu coal mine
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Figure 9 The stress and energy evolution of sandstone with and without anchorage: 

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- **b** stress-strain relationship curve of sandstone (with anchorage); 
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- **d** energy evolution of sandy mudstone (with anchorage)
Fig. 1 Results of uniaxial compressive strength tests. 
(a) numerical simulation of uniaxial compression, (b) stress–strain curve of coal, (c) stress–strain curve of sandy mudstone, (d) stress–strain curve of sandstone.

Fig. 2 Conceptual mechanical representation of the rockbolt element.

Fig. 3 Rockbolt barrel and wedge and assembly drawing.
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Fig. 5 The stress and degree of damage of rock with and without anchorage: a coal without anchorage; b coal with anchorage; c sandy mudstone without anchorage; d sandy mudstone with anchorage; e sandstone without anchorage; f sandstone with anchorage.
Fig. 6 Stress and energy evolution with and without anchorage in coal

- **a**: stress-strain curve of coal (without anchorage);
- **b**: a stress-strain curve of coal (without anchorage);
- **c**: energy evolution of coal (without anchorage);
- **d**: energy evolution of coal (anchorage).

Fig. 7 Stress and energy evolution with and without anchorage in sandy mudstone

- **a**: stress-strain curve of sandy mudstone (without anchorage);
- **b**: a stress-strain curve of sandy mudstone (without anchorage);
- **c**: energy evolution of sandy mudstone (without anchorage);
- **d**: energy evolution of sandy mudstone (anchorage).
The magnitude of change and specific values

- A: Increase in initial damage stress (%)
- B: Increase in peak elastic strain energy (%)
- C: Increase in boundary work (kJ)
- D: Crack friction work ratio after installing rockbolt (%)

Fig. 8 The comparison between coal and sandy mudstone on different energy categories

Fig. 9 The stress and energy evolution of sandstone with and without anchorage:
- a) stress-strain relationship curve of sandstone (without anchorage);
- b) stress-strain relationship curve of sandstone (with anchorage);
- c) energy evolution of sandy mudstone (without anchorage);
- d) energy evolution of sandy mudstone (with anchorage)
### Table 1 Intact rock properties and scaled rock mass properties of coal measures from Yuwu coal mine

| Lithology         | Intact rock | RQD | Rock mass |
|-------------------|-------------|-----|-----------|
|                   | $E_r$ (GPa) | $\sigma_r$ (MPa) | $E_m$ (GPa) | $\sigma_m$ (MPa) | $\sigma_d$ (MPa) |
| Coal              | 2.6         | 10.8 | 75%       | 0.79           | 5.1           | 0.51           |
| Sandy mudstone    | 5.4         | 35.5 | 90%       | 3.1            | 25            | 2.5            |
| Sandstone         | 9.7         | 53.5 | 88%       | 5.2            | 36.1          | 3.6            |

### Table 2 Calibrated mechanical parameters of blocks and joints of the coal measures

| Density (kg/m$^3$) | Young’s modulus (GPa) | Cohesion (MPa) | Friction (°) | Tension (MPa) |
|--------------------|-----------------------|----------------|--------------|---------------|
| Block              | 1400                  | 0.79           | 1.1 ($\varepsilon_p=0.04$) | 27            | 0.9           |
| Joint              |                        |                |              |               |
| Normal stiffness   | 113                   | 45.2           | 1.3          | 18            | 0.4           |
| Tangential stiffness | 45.2               |                |              |               |               |

### Table 3 Calibrated mechanical parameters of blocks and joints of sandy mudstone

| Density (kg/m$^3$) | Young’s modulus (GPa) | Cohesion (MPa) | Friction (°) | Tension (MPa) |
|--------------------|-----------------------|----------------|--------------|---------------|
| Block              | 1800                  | 3.1            | 5.0 ($\varepsilon_p=0.06$) | 27            | 2.5           |
| Joint              |                        |                |              |               |
| Normal stiffness   | 372                   | 149            | 6.2          | 18            | 1.8           |
| Tangential stiffness | 149                  |                |              |               |               |

### Table 4 Calibrated mechanical parameters of blocks and joints of sandstone

| Density (kg/m$^3$) | Young’s modulus (GPa) | Cohesion (MPa) | Friction (°) | Tension (MPa) |
|--------------------|-----------------------|----------------|--------------|---------------|
| Block              | 2550                  | 5.2            | 7.0 ($\varepsilon_p=0.04$) | 30            | 2.5           |
| Joint              |                        |                |              |               |
| Normal stiffness   | 665                   | 266            | 11.2         | 18            | 1.8           |
| Tangential stiffness |                    |                |              |               |               |
Table 5 Calibrated microproperties in the UDEC Trigon model to represent the rock masses

| Lithology       | Young’s modulus (GPa) | Compressive strength (MPa) |
|-----------------|-----------------------|----------------------------|
|                 | Target                | Calibrated | Error (%) | Target | Calibrated | Error (%) |
| Coal            | 0.79                  | 0.73       | 7         | 5.1    | 5.1        | 0         |
| Sandy mudstone  | 3.1                   | 2.9        | 6         | 25     | 25.1       | 0.4       |
| Sandstone       | 5.2                   | 4.7        | 9         | 36.1   | 36.3       | 5         |

Table 6 Rockbolt and anchor parameters

| Cross-section area (m²) | Elastic modulus (GPa) | Tensile yield strength (kN) | Second moment of area (m⁴) | tfspring |
|-------------------------|-----------------------|----------------------------|---------------------------|---------|
| Rockbolt                | 3.8 × 10⁻⁴            | 200                        | 198                       | 1.2 × 10⁻⁸ | 0.012  |
| Exposed perimeter (m)   |                       |                            |                           |         |
| Cohesive strength of shear coupling spring (MPa) | | | | |
| Stiffness of shear coupling spring (GPa) | | | | |
| Frictional resistance of the shear coupling spring (°) | | | | |
| Cohesive strength of normal coupling spring (MPa) | | | | |
| Stiffness of normal coupling spring (GPa) | | | | |
| Frictional resistance of the normal coupling spring (°) | | | | |
| Anchor parameters       | 0.07                  | 1                          | 8                         | 45      | 200       | 20        | 0        |

Table 7 Numerical simulation scheme

| Rockbolt existence | Pre-tension (load) | Loading rate |
|--------------------|--------------------|--------------|
| Coal (soft)        | No                 | /            | 0.02m/s |
| Yes                | 0.365MPa           | 0.02m/s      |
| Sandy mudstone (medium) | No              | /            | 0.02m/s |
| Yes                | 0.365MPa           | 0.02m/s      |
| Sandstone (strong) | No                 | /            | 0.02m/s |
| Yes                | 0.365MPa           | 0.02m/s      |

Table 8 Mechanical properties and degree of damage of rock prior and after anchorage

| Rock bolt existence | Elastic modulus (GPa) | Peak strength (MPa) | Residual strength (MPa) | Degree of damage (%) |
|---------------------|-----------------------|---------------------|-------------------------|----------------------|
| Coal (soft)         | No                    | 0.73                | 5.1                     | 3.8                  | 35.1                |
| Yes                 | 0.76                  | 5.4                 | 4.0                     | 30.3                 |
| Sandy mudstone (medium) | No               | 2.9                 | 25.1                    | 13                   | 46.2                |
| Yes                 | 3.5                   | 26                  | 14.7                    | 40.7                 |
| Sandstone (strong)  | No                    | 4.7                 | 36.3                    | 20.2                 | 50.8                |
| Yes                 | 4.8                   | 36.5                | 20.3                    | 48.1                 |
Appendix B

An Investigation on the effect of high energy storage anchor on surrounding rock conditions

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Relevant information will appear here if provided.

**Ethics**

Does your article include research that required ethical approval or permits?:

This article does not present research with ethical considerations

Statement (if applicable):

CUST_IF_YES_ETHICS :No data available.

**Data**

It is a condition of publication that data, code and materials supporting your paper are made publicly available. Does your paper present new data?:

Yes

Statement (if applicable):

Our data are deposited at Dryad.

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**Conflict of interest**

I/We declare we have no competing interests

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We have no competing interests.

**Authors' contributions**

This paper has multiple authors and our individual contributions were as below

Statement (if applicable):

B.W. and X.W. conceived the article structure. B.W. and W.W. performed the rock mechanical experiments. J.B. developed energy balance criterion. B.W. numerical simulation. All the authors analysed the data. N.M. prepared the plots. H.L. language revision. All the authors prepared the initial draft and revised the paper.
An Investigation on the effect of high energy storage anchor on surrounding rock conditions

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Keywords: Energy evolution, tension bolts, UDEC, differential analysis, rock

1. Summary

High pre-tension bolt is an effective strata control technique and is the key to ensure the stability of anchorage and roadway. Based on the performances of high energy storage tension rock bolts in different rock properties, this study proposed a constitutive model to describe the energy balance of anchor under uniaxial compression. UDEC was used to simulate the behaviour of anchor in coal under uniaxial compression and results were analysed to study the rock mechanical properties, degree of damage and energy evolution. Simulation results showed that tension rock bolts can improve the mechanical properties and energy storage capacities of anchor. The energy evolution was divided into three stages: i) the external work was stored in the form of elastic strain energy ($U_e$) in the anchor prior to the yielding strength; ii) the elastic strain energy reached its maximum near the peak strength; iii) Energy was dissipated from fracture friction ($W_f$), plastic deformation ($W_p$) and acoustic emission ($U_r$) during post-peak stage. The installation of tension rock bolts resulted in the most improvement of surrounding rock condition for medium hard rock including sandy mudstone, whereas it had the least impact on hard rock, e.g. sandstone.

2. Introduction

Rock strength is a critical factor for underground structural design. To improve the mechanical properties of rock, rock and cable bolts are generally implemented. In underground coal mines, the rock strengthening measure is immediate bolt support after excavation to ensure the roadway stability and subsequent workplace safety [1]. Extensive practical work revealed that this is an effective strata control method, which has also been widely used in Chinese coal mines [2].

To determine the effect of rock bolt on anchor strengthening and improve the ability of anchorage, a significant body of researches have carried out. Hou [3] theoretically analysed the effect of rock bolts on the peak and the residual strengths of rock around anchorage range. Based on the improvement of mechanical properties prior and after the peak strength, Hou [3] proposed a constitutive model for rock strengthening effect around the roadway with rock bolts. Kang et al. [4] described the mechanisms and technologies of rock bolt supporting system in China. The authors suggested that high strength-stiffness-reliability and low-density proactive rock bolts can improve the strength of surrounding rock. Wang et al. [5] derived the equations of shear strength prior and after anchorage. The analysis showed that once the grouting is completed, the shear strength of anchorage range was improved together with the mechanical properties of surrounding rock (elastic modulus, cohesion, friction angle and dilation angle). Wei.et al. [6] carried out physical simulation tests to study the influence of pre-tension on anchor strength and deformation and found that high pre-tension can improve the anchorage strength. Based on the triaxial physical simulation tests,
Wang et al. [7] revealed that the rock bolt does not only improve the capacity of fracture surrounding rock, but also controls the discontinuous deformation of fractured surrounding rock. This will in turn improve the continuity and integrity of surrounding rock mass of deep roadway.

In numerical simulation, the bolt section in anchorage range can be represented by material model as well as structural elements. Both methods have been proven successful in simulating the mechanical properties of bolts and anchors under various loading conditions. Meng [8] imported the constitutive model of anchor in fractured rock to FLAC3D and subsequently simulated the effect of anchoring condition on ultimate bearing capacity. Results showed that the higher pre-tension and density can improve the ultimate bearing capacity. Zuo et al. [9] used elastic-isotropic model in FLAC3D to study the influence of the alignment between bolt and borehole diameters on anchorage effect. Numerical simulation suggested a reasonable alignment (4 – 11 mm diameter difference) can improve the anchorage effect. D.-A. Ho et al. [10] used elasto-plastic model to study the fracture mechanism of interface between rock bolt and anchoring agent under various horizontal stress conditions. Thereby, Grasselli [11], Aziz and Jalalifar [12] and Tatone et al. [13] also used material model to simulate various laboratory condition of rock bolt. However, the mesh size is an important factor which can influence the simulation results using material model. This constrains the preciseness of the simulation. On the other hand, structural element approach can overcome this problem. At present, the structural element can be divided into ‘cable element’ and ‘rockbolt element’. Zhang et al. [14] used cable element to study the parameters including length, anchorage length and rock anchor spacing via FLAC3D and proposed several suggestions to support bolt design. Wu et al. [15] attempted to simulate the supporting element (e.g. wooden cylinder) in supported roadway using Cable element and found that the rock bolts can improve the capacity of wooden cylinder and prevent the fracture development. Pull-test or underground excavation with rock bolt support were simulated by (Vardakos et al. [16], Malmgren and Nordlund [17], Li et al. [18], Gao et al. [19], Shreedharan and Kulatilake [20]). Ma et al. [21] used rockbolt element to simulate pull-test via FLAC2D and studied the relationship between rock bolt and rock as well as the mechanical properties of newly developed rock bolt and its preferable conditions.

According to the previous studies, it can be found that the focus was mainly on the anchorage technique, rock bolt, coordination effect and stress transfer. Although there is an overall agreement that high pre-tension can improve the mechanical properties of anchor, the influence of high pre-tension under various rock strength conditions on anchorage was not investigated. Majority of the research analysed the anchorage effect via elasto-plastic model and stress-strain relationship, while there were limited studies investigated the problem using energy conservation and the fracture evolution of anchorage is a process of energy dissipation and release. Hence, it is more reasonable to study the anchorage effect using energy law (Xie et al.[22]). Many numerical simulation use cable element to approach this topic; this is inappropriate considering the method is one-dimensional element which consists of two degrees of freedom. On the other hand, rockbolt element is more suitable for the analysis as it is a two-dimensional element with three degrees of freedom (Itasca Consulting Group Inc. [23]). The paper used rockbolt element through UDEC Trigon to study the influence of high pre-tension on anchor with various surrounding rock strengths under uniaxial compression. This aims to study the different strengthening behaviours of tension rock bolt under various conditions based on monitor of the energy evolution, the stress-strain relation as well as the fracture development in different rock strengths.

### 3. Energy balance and components

Nowadays, the study of energy evolution in rock mechanics has been refined to different depth of cover, loading conditions, in-situ stress magnitudes (Salamon [24], Napier [25], Onur Vardar et al. [26], Zhang et al. [27], Wang et al. [28]). In UDEC, the energy exchanged based on the work done on rock, joint and boundary. Without considering dynamic calculation, the energy balance in UDEC mainly involves work done on boundary, stored strain energy and dissipated energy (Itasca Consulting Group Inc. [23]). This study determined the constitutive law based on the previous studies as well as the simulation conditions.

In uniaxial compression, work done from external force is denoted by $W$, $U$ is used to represent the elastic strain energy due to elastic deformation of the anchor. The difference in work done between boundary and elastic strain energy is dissipated energy, $U^d$. Hence, total input energy can be written as:

$$W = U^e + U^d$$ \hspace{1cm} (1)

Energy is mainly dissipated in three ways. Firstly, energy is dissipated by friction due to the fracture development in rock; this is denoted as $W_f$. Second source is due to the plastic deformation of rock ($W_p$). Once the rock is undergone plastic deformation, energy dissipated from plastic work done. The rest of energy is dissipated via acoustic emission, denoted as $U_r$. Therefore, $U^d$ can be expressed as:
In UDEC, the incremental change of these energy components is cumulative and determined by timesteps 
Itasca Consulting Group Inc. [23]). By combining Eq (1) and Eq (2), W can be expressed as:

\[ W = W_f + W_p + U' + U'' \]  

(3)

Subsequently, this energy balance equation can be used to investigate the complex energy dissipation 
process in anchor.

4. Parameters calibration

4.1. The UDEC Trigon approach

UDEC Trigon model was proposed by Gao and Stead [29], aiming at simulating brittle fracture of rock. In this 
model, a rock is represented by an assembly of triangular blocks bonded together via the grain contacts. Each 
block is made elastically by dividing them into triangular finite difference zones. In the direction normal to a 
contact, the stress-displacement relation is assumed to be linear and governed by the stiffness \( k_n \), such that:

\[ \Delta \sigma_n = -k_n \Delta u_n \]  

(4)

where \( \Delta \sigma_n \) is the effective normal stress increment and \( \Delta u_n \) is the normal displacement increment. A 
limiting tensile strength \( (T) \) is assumed for the contact. If this value is exceeded, then \( \sigma_n = 0 \).

In the shear direction, the response is governed by constant shear stiffness. The shear stress, \( \tau_{ss} \), is 
determined by a combination of contact properties, cohesion \( (c) \) and friction angle \( (\phi) \), where:

\[ f_s \leq c + \tau_{ss} \tan \phi = \tau_{max} \]  

(5)

then

\[ \tau_{ss} = k_{ss} \Delta u_{ss} \]  

(6)

however, if \( f_s \geq \tau_{max} \), then:

\[ \tau_{ss} = \text{sign} (\Delta u_{ss}) \tau_{max} \]  

(7)

where \( \Delta u_{ss} \) is the elastic component of the incremental shear displacement and \( \Delta u_{ss} \) is the total incremental 
 shear displacement.

The proposed modeling approach has been implemented in UDEC (Itasca Consulting Group Inc.[23]).

4.2 Mechanical parameters of coal and rock mass

Since blocks in UDEC Trigon model are considered as elastic materials, they cannot be plastically destroyed. 
However, the anchor breakage is plastic damage. To closely mimic the mechanical properties and energy 
evolution of anchor post peak strength, this study also used strain softening model in UDEC. The strain-
softening model is based on the UDEC Mohr–Coulomb model with non-associated shear and associated 
tension flow rules.

This study selected coal (soft), sandy mudstone (medium) and sandstone (strong) to represent various 
rock strengths. The intact properties of the rock masses are listed in Table 1. These properties were obtained 
through laboratory compression tests and were provided by the Yuwu coal mine. The RQD values of the rock 
masses were evaluated from borehole televiewer images.

Table 1 Intact rock properties and scaled rock mass properties of coal measures from Yuwu coal mine

| Rock Type      | Intact Modulus Ratio | RQD Range | Scaled Modulus Ratio |
|----------------|----------------------|-----------|----------------------|
| Coal           | 0.7                  | 0-100     | 0.8                  |
| Mudstone       | 0.65                 | 100-200   | 0.75                 |
| Sandstone      | 0.55                 | 200-300   | 0.65                 |

The rock mass elastic modulus was calculated using the relationship between RQD and the elastic 
modulus ratio (Zhang and Einstein [30]), as shown in Eq. 8, where \( E_m \) is the elastic modulus of the rock mass 
and \( E_r \) is the elastic modulus of the rock sample.

\[ \frac{E_m}{E_r} = 10^{0.0136RQD - 1.91} \]  

(8)

The rock mass strength was then calculated using the relation between the unconfined compressive 
strength ratio \( \sigma_{um} / \sigma_c \) and the deformation modulus ratio \( E_m / E_r \) (Singh and Seshagiri Rao [31]). The value of 
q is 0.63 (Gao. et al, [32]):

\[ \frac{\sigma_{um}}{\sigma_c} = \left( \frac{E_m}{E_r} \right)^q \]  

(9)

To represent the coal and rock by using an assembly of triangular blocks, the properties of the blocks and 
contacts were calibrated against the coal measure and rock mass properties listed in Table 2 to 4. This was
achieved by simulating unconfined compression tests in a numerical model created using the Trigon logic. The size of the rock sample is 1 m (in width) × 2 m (in height) (Singh and Seshagiri Rao [31]) (Fig. 1). The bottom of the numerical model was fixed and a loading rate of 0.02 m/s was applied at the top. The calibrated properties of the UDEC model are illustrated in Table 5.

The uniaxial compressive strength and elastic modulus data derived by numerical simulation are similar to the data obtained from laboratory tests (within an error of 9%). Hence, the micromechanical parameters of the coal mass and the rock mass were properly calibrated.

3.3 Rockbolt element parameter

In UDEC, rockbolt element is different from cable element. Rockbolt element is a two-dimensional element which its two nodes have three degrees of freedom (two displacements and one rotation). It can resist bending and can yield along axial direction. Fig. 2 illustrates the components of rockbolt element. Rockbolts interact with UDEC via shear and normal coupling springs, which the springs are non-linear connectors. These are used to transfer force and motion between rockbolt element and the mesh nodes. Fig. 2 shows material behavior of shear and normal coupling spring for rockbolt elements. Shear coupling springs are used to simulate the shear behaviour of anchorage length and normal coupling spring are used to simulate the compression of surrounding rock.

Rockbolt element can be broken at the node. The bolt breakage can be simulated by defining the tensile failure strain limit ($t_fstrain$). Rock bolt will be deemed as failed if $\varepsilon_{pl} \geq t_fstrain$, where $t_fstrain$ has to be defined. $\varepsilon_{pl}$ is the total tensile yield strain of any element of anchor:

$$\varepsilon_{pl} = \sum \varepsilon_{pl}^{ax} + \sum \frac{d \theta_{pl}}{2L}$$  \hspace{1cm} (10)

$\varepsilon_{pl}^{ax}$ is the axial deformation; d is rock bolt diameter, $\theta_{pl}$ is the average rotation angle of the component.

This study used rockbolt element to simulate HRB335 threaded steel anchor at 22 mm diameter. Based on the laboratory pull-test results, the input parameters of rockbolt element can be obtained, as displayed in Table 6.

5. Research background and model establishment

The common way to apply pre-tension is to install the nut on the tail thread of rock bolt. By applying torque to the nut, axial tension is acting onto the rock bolt (Li et al. [33]). However, high-pretension is difficult to be controlled in practice. For instance, when the torque achieves 300 – 400 Nm, the pre-tension can sometimes still be low. This low conversion efficiency is due to the torque at thread. To increase the pre-tension of rock bolt, a new type of anchor lock has been developed by the research team. This development changed the traditional pre-tension method, which is capable of applying high pre-tension without damaging rock bolt. The anchor lock is composed of an anchor ring and three clips. The anchor ring is a ring-shaped structure, in which the center hole is tapered. The clips are disposed between the inner wall of the tapered hole of the anchor ring and the anchor body, as displayed in Fig. 3.

Fig. 4 shows the setup of UDEC Trigon simulation. The blocks and interfaces used strain softening and Coulomb-sliding models, respectively. The dimensions of the numerical model are 1 m × 2m. Due to the
limitation of rockbolt element, the effect of vertically installed rock bolts was not significant. Hence, rock bolts were installed horizontally to the rock body at y = 0.4m, 1.0m and 1.6m. Pre-tension was applied by applying load at the nodes of the rock bolts. In practice, the common torque applied to the rock bolt is approximately 300 – 400 Nm, which are 40 – 50 kN pre-tension correspondingly (Kang et al. [34]). To achieve high pre-tension in the numerical model, the study set the pre-tension as 70% of the yielding strength, i.e. 138.6 kN/0.365 MPa. The loading rate of 0.02 m/s was applied at the top boundary of the model while fixing the bottom boundary. During the simulation process, the stress-strain relationship and fracture development of anchor was monitored by FISH function. Boundary, frictional and plastic work were recorded using UDEC energy element. ‘SET energy on’ command was used to activate the energy monitoring function while turn off ‘mass-scaling’. As this study used non-viscous boundary which does not consider the dynamic calculation, damping was set as ‘auto’. The simulation procedures involved: i) model construction and calibration; ii) test simulation.

### Table 7 Numerical simulation scheme

| Table 7 Numerical simulation scheme |

| Fig.4 Model overview and dimensions |

### 6. Discussion

#### 6.1 Mechanical properties and damage analysis

To monitor the degree of damage of rock and anchor, the total fracture length as well as the shear and tensile fracture lengths during uniaxial compression were measured. The anchor and surrounding rock fracture are a complex process, which the shear and tensile failures can occur in a single fracture. Hence, only the initial (first) failure type was recorded for each fracture. The degree of damage \( D \) can be calculated as:

\[
D = \frac{L_s + L_t}{L_C} 
\]

(11)

where \( L_c \) is the total fracture length, \( L_s \) and \( L_t \) are total shear and tensile fracture lengths, respectively.

Fig. 5 illustrates the stress and degree of damage of rock during the process with and without anchorage. Based on the comparison, the mechanical properties (peak strength, residual strength and elastic modulus) of rock were increased with the installation of tension rock bolts, while the degree of damage decreased. Under the same axial strain, the higher strength always led to higher degree of damage regardless of the anchorage condition. The installation of tension rock bolts did not have significant influence on the axial strain of rock during damage. There was increase in the degree of damage prior to the peak rock strength and the gradient became flatter when it was near the peak strength. The indication here is that installation of tension bolts can prevent the fracture development and coalescence, such that the degree of damage was reduced. To compare the effect of tension rock bolts under different rock conditions, Table 8 summaries the critical parameters from Fig. 5.

Table 8 Mechanical properties and degree of damage of rock prior and after anchorage

| Table 8 Mechanical properties and degree of damage of rock prior and after anchorage |

Table 8 suggested that there are differences in the strengthening effect of tension rock bolts on rock mechanical properties with various rock types. When the bolt was installed in coal, elastic modulus, peak strength and residual strength increased 4.1% (0.03 GPa), 5.9% (0.3 MPa) and 5.2% (0.2 MPa), while the degree of damage decreased 13.7%. In sandy mudstone, elastic modulus, peak strength and residual strength increased 20.7% (0.6 GPa), 3.6% (0.9 MPa) and 13.1% (1.7 MPa), while the degree of damage decreased 11.9%. For sandstone, elastic modulus, peak strength and residual strength increased 21.7% (0.1 GPa), 0.6% (0.2 MPa) and 0.5% (0.1 MPa), while the degree of damage decreased 5.3%. Sandy mudstone had the most change in absolute magnitudes and sandstone had the least change. In terms of the relative magnitude difference, coal had the most change whereas sandstone had the least. The absolute magnitude change of sandy mudstone was the most while its degree of damage was only in the middle, indicating the mechanical properties of rock was improved by installation of rock bolts. However, this is not the primary reason for the reduction in degree of damage, in which the main reason was due to the prevention of fracture development and coalescence using rock bolts. Sandstone had the low change in terms of absolute and relative magnitudes. This suggests that high pre-tension rock bolts do not have significant influence on strong rock. This is because the rock strength was high and it required substantially higher stress to induce the rock failure. Under this condition, the effect of rock bolt is negligible comparing with the high strength. At post-peak of the loading process, the
fracture was more likely to be developed due to the low plasticity, which was difficult to be controlled using rock bolts. Overall, it can be concluded that tension rock bolts have insignificant influence on strong rock, such that the high pre-tension is not recommended for hard rock support design.

6.2 Analysis of the law of energy evolution

During the loading process, rock was continuously deforming and storing elastic strain energy. At the same time, the energy was dissipated due to internal damage. When the elastic strain energy reached the limit, it was released and led to rock failure. Hence, the failure process of rock is closely related to the energy evolution. Based on the analysis carried out in Section 5.1, tension rock bolts are not suitable for hard rock. Therefore, this section will focus on the energy evolution of coal and sandy mudstone, as displayed in Fig. 6 and 7.

In Fig. 6 and 7, the energy dissipation due to acoustic emission ($U'$) was illustrated solely. This is because the energy dissipation curve of acoustic emission was similar as the accumulative acoustic emission, which is a major indication of anchor breakage. Secondly, the energy dissipation due to acoustic emission was substantially lower than the other sources. A sole illustration can represent the energy evolution of acoustic emission more clearly. According to the results from Fig. 6 and 7, the energy evolution of rock was consistent regardless of the rock bolts. Based on the correspondence between stress curve and energy curve, the energy evolution of rock can be divided into three stages:

Stage 1: From the start of the load to the yielding strength. External work was fully absorbed by rock and stored in the form of elastic strain energy. The energy curve increased non-linearly during this stage, while the external work curve was well aligned with the elastic strain curve. Based on the stress and energy curves, the rock was under elastic deformation prior to the yielding strength. There was no observable internal damage, such that the internal energy dissipation was considered as none.

Stage 2: From the yielding strength to the peak strength. External work continuously increased and the total energy that the rock absorbed was also increasing. However, as the rock was in plastic condition and there was some degree of internal damage, the frictional work in-between cracks and plastic deformation started to occur and increased at a high rate. This indicates that the degree of damage was decreased although the acoustic emission level was still low. During this stage, the elastic strain energy was also increasing but the growth rate was flatter. Thereby, its alignment with the external work was getting worse and reached its maximum near the peak strength point.

Stage 3: Post-peak stage. External work continuously increased but the elastic energy in the rock was released at a high rate. There was considerable increase in the energy dissipation due to frictional work, plastic deformation and acoustic emission. The acoustic mission energy curve was growing rapidly, suggesting that the degree of damage was high. The external work was mainly transferred to the energy require for rock fracture towards the end of the simulation and the elastic strain energy was not stored anymore. When the stress dropped to the residual strength, the elastic strain energy and energy dissipation curves became flatter, although the energy continuously dissipated.

Although the energy evolution of rock was consistent regardless of the existence of rock bolts, the degree of damage and other parameters changed noticeably with the installation of rock bolts. Firstly, there were increasing in stress and decreasing in strain at anchor damage with rock bolts. For coal, the stress increased from 4.4 MPa to 4.8 MPa and strain reduced from 0.57% to 0.5%. For sandy mudstone, stress increased from 18 MPa to 20.6 MPa and strain reduced from 0.6% to 0.49%. Secondly, the peak elastic energy increased from 29.2 kJ to 35 kJ and from 202.7 kJ to 240.7 kJ for coal and sandy mudstone, respectively. When strain reached 1.2%, the external work of coal and sandy mudstone increased 26.2 kJ and 33.7 kJ. For coal, the work done from fracture friction was 35.1 kJ without rock bolt, accounting for 50.1% of the total input energy; whereas the energy was 37.4 kJ and took account of 38.9% with rock bolt. On the other hand, for sandy mudstone, the work done from fracture friction was 230.3 kJ without rock bolt, accounting for 58.8% of the total input energy; whereas the energy was 158.7 kJ and took account of 37.2% with rock bolt. Based on aforementioned data, the installation of tension rock bolts can improve the strength of surrounding rock and its storage of elastic strain energy. At the same time, it can also prevent fracture development and postpone the elastic strain energy release at post-peak. Fig. 8 shows the comparison of each energy categories. From the figure, it can be seen that the performance of sandy mudstone was better than coal. Hence, the indication here is that the tension
rock bolts can improve the mechanical properties and energy storage capacity of rock, which agrees with the conclusion from Section 6.1. Thereby, the most suitable rock type for the application is medium hard rock, e.g. sandy mudstone.

Fig. 8 The comparison between coal and sandy mudstone on different energy categories

Fig. 9 The stress and energy evolution of sandstone with and without anchorage: a stress-strain relationship curve of sandstone (without anchorage); b stress-strain relationship curve of sandstone (with anchorage); c energy evolution of sandy mudstone (without anchorage); d energy evolution of sandy mudstone (with anchorage).

Fig. 9 depicts the stress and energy evolution of sandstone with and without anchorage. According to the figure, it can be found that the relationships of sandstone follow the same trend as those of coal. The difference between sandstone and coal was that the changes of frictional work and plastic deformation of sandstone were insignificant with rock bolts, indicating that the installation of rock bolts does not increase the plasticity of sandstone nor prevent fracture development. The plastic deformation curve was always at low level while the frictional work curve was always at high level. This phenomenon again supports that the fracture development was the main reason of rock failure. This agrees with the conclusion from stress and damage analysis; thereby proved that tension rock bolts are not effective for hard rock roadway support, e.g. sandstone.

7 Conclusion

Based on the UDEC simulation on the strengthening effect of tension rock bolts in different rock properties, a number of parameters were studied, including mechanical properties, degree of damage and energy evolution. The following conclusions were drawn from the numerical results:

1) Tension rock bolts can improve the mechanical properties and energy storage capacity of anchor. Subsequently, it can reduce the degree of damage of the anchor while increasing the elastic strain energy storage and plastic deformation work. By suppressing energy dissipation due to fracture friction and delaying the post-peak elastic strain release, the stability of anchor can be increased.

2) Based on the stress-strain and energy dissipation-strain relationships during uniaxial compression, the energy evolution of anchor was divided into three stages. In the first stage, the external work was fully absorbed by rock and was stored as elastic deformation energy. In the second stage, the axial stress surpasses the rock yielding strength, which work was done from fracture friction and plastic deformation. At the same time, the growth rate of elastic strain energy was reduced and the elastic strain energy reached the maximum value near the peak strength point. In the third stage, the energy stored in the rock was rapidly released from elastic strain energy. The work from fracture friction, plastic deformation and acoustic emission increased substantially. At the end of the process, the energy of external work was mainly transferred to rock failure, where the energy was dissipated via fracture friction, plastic deformation and acoustic emission. Among them, work from fracture friction and plastic deformation was the major contributing factor for energy dissipation.

3) Based on stress and energy evolution analysis on different rock types, it can be seen that the tension rock bolts were not effective for hard rock. The elastic modulus, peak strength and residual strength increased 2.1% (0.1 GPa), 0.6% (0.2 MPa) and 0.5% (0.1 MPa), respectively. The implementation could not prevent the fracture development within the rock mass, such that it was not recommended for hard rock supporting design.

4) The absolute magnitudes of mechanical properties of sandy mudstone increased more than that of coal. However, the relative magnitudes and reduction of degree of damage was lower than coal. This indicates that the improvement of mechanical properties of rock due to rock bolts was the secondary factor for degree of damage reduction, whereas its effect on preventing fracture development and coalescence was the main contributing factor. For energy evolution, the stress at initial damage increased 14.4% (2.6 MPa) and 9% (0.4 MPa) for sandy mudstone and coal. In addition, the peak elastic strain energy increased 18.7% (38 kJ) and 9.6% (5.8 kJ) for sandy mudstone and coal. The work done due to fracture friction decreased from 58.8% to 37.2% of the total energy for sandy mudstone; and it decreased from 50.1% to 38.9% for coal. Therefore, tension rock bolts is more suitable for medium hard rock (e.g. sandy mudstone), followed by soft rock (e.g. coal).

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Our data are deposited at Dryad. Review URL: https://datadryad.org/stash/share/Afg-q609idvPjuiUn9zgr879kqBC6MFbF5n4fWk8
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**Competing Interests**
We have no competing interests.

**Authors’ Contributions**
B.W. and X.W. conceived the article structure. B.W. and W.W. performed the rock mechanical experiments. J.B. developed energy balance criterion. B.W. numerical simulation. All the authors analysed the data. N.M. prepared the plots. H.L. language revision. All the authors prepared the initial draft and revised the paper.

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Tables

Table 1 Intact rock properties and scaled rock mass properties of coal measures from Yuwu coal mine

| Lithology         | $E_r$ (GPa) | $\sigma_r$ (MPa) | RQD | $E_m$ (GPa) | $\sigma_m$ (MPa) | $\sigma_{tm}$ (MPa) |
|-------------------|-------------|------------------|-----|-------------|------------------|--------------------|
| Coal              | 2.6         | 10.8             | 75% | 0.79        | 5.1              | 0.51               |
| Sandy mudstone    | 5.4         | 35.5             | 90% | 3.1         | 25               | 2.5                |
| Sandstone         | 9.7         | 53.5             | 88% | 5.2         | 36.1             | 3.6                |

Table 2 Calibrated mechanical parameters of blocks and joints of the coal measures

| Density (kg/m$^3$) | Young's modulus (GPa) | Cohesion (MPa) | Friction ($\theta$) | Tension (MPa) |
|--------------------|------------------------|----------------|---------------------|---------------|
| Block              | 1400                   | 0.79           | 1.6 ($\varepsilon_p=0$) | 27            | 0.93               |
| Joint              | Normal stiffness (GPa) | Tangential stiffness (GPa) | Cohesion (MPa) | Friction ($\theta$) | Tension (MPa) |
| 113                | 45.2                   | 1.3            | 18                  | 0.4           |

Table 3 Calibrated mechanical parameters of blocks and joints of sandy mudstone

| Density (kg/m$^3$) | Young's modulus (GPa) | Cohesion (MPa) | Friction ($\theta$) | Tension (MPa) |
|--------------------|------------------------|----------------|---------------------|---------------|
| Block              | 1800                   | 3.1            | 8.0 ($\varepsilon_p=0$) | 27            | 2.5                |
| Joint              | Normal stiffness (GPa) | Tangential stiffness (GPa) | Cohesion (MPa) | Friction ($\theta$) | Tension (MPa) |
| 372                | 149                    | 6.2            | 18                  | 1.8           |

Table 4 Calibrated mechanical parameters of blocks and joints of sandstone

| Density (kg/m$^3$) | Young's modulus (GPa) | Cohesion (MPa) | Friction ($\theta$) | Tension (MPa) |
|--------------------|------------------------|----------------|---------------------|---------------|
| Block              | 2550                   | 5.2            | 13.0 ($\varepsilon_p=0$) | 30            | 2.5                |
| Joint              | Normal stiffness (GPa) | Tangential stiffness (GPa) | Cohesion (MPa) | Friction ($\theta$) | Tension (MPa) |
| 665                | 266                    | 11.2           | 18                  | 1.8           |

Table 5 Calibrated microproperties in the UDEC Trigon model to represent the rock masses

| Lithology    | Young's modulus (GPa) | Compressive strength (MPa) |
|--------------|------------------------|----------------------------|
|              | Target                 | Calibrated               | Error (%) | Target | Calibrated | Error (%) |
| Coal         | 0.79                   | 0.73                      | 7         | 5.1    | 5.1        | 0         |
| Sandy mudstone | 3.1                    | 2.9                       | 6         | 25     | 25.1       | 0.4       |
| Sandstone    | 5.2                    | 4.7                       | 9         | 36.1   | 36.3       | 5         |
Table 6 Rockbolt and anchor parameters

|                | Cross-section area (m²) | Elastic modulus (GPa) | Tensile yield strength (kN) | Second moment of area (m⁴) | tfstrain |
|----------------|-------------------------|-----------------------|-----------------------------|---------------------------|----------|
| Rockbolt       | 3.8 × 10⁻⁴              | 200                   | 198                         | 1.2 × 10⁻⁴                | 0.012    |
|                |                         |                       |                             |                           |          |
| Exposed perimeter (m) |               | Cohesive strength of shear coupling spring (MPa) | Stiffness of shear coupling spring (GPa) | Frictional resistance of the shear coupling spring (°) | Cohesive strength of normal coupling spring (MPa) | Stiffness of normal coupling spring (GPa) | Frictional resistance of the normal coupling spring (°) |
| Anchor parameters | 0.07                  |                       |                             |                           |          |

Table 7 Numerical simulation scheme

| Rockbolt existence | Pre-tension (load) | Loading rate |
|--------------------|-------------------|--------------|
| Coal (soft)        | No                | /            |
|                    | Yes               | 0.365MPa     | 0.02m/s                   |
| Sandy mudstone (medium) | No              | /            | 0.02m/s                   |
| Sandstone (strong)  | No                | /            | 0.02m/s                   |
|                    | Yes               | 0.365MPa     | 0.02m/s                   |

Table 8 Mechanical properties and degree of damage of rock prior and after anchorage

| Rock bolt existence | Elastic modulus (GPa) | Peak strength (MPa) | Residual strength (MPa) | Degree of damage (%) |
|---------------------|-----------------------|---------------------|-------------------------|----------------------|
| Coal (soft)         | No                    | 0.73                | 5.1                     | 3.8                  | 35.1      |
| Sandy mudstone      | No                    | 2.9                 | 25.1                    | 13                   | 46.2      |
| Sandstone (strong)  | Yes                   | 3.5                 | 36.3                    | 20.2                 | 50.8      |
|                     | Yes                   | 4.8                 | 36.5                    | 20.3                 | 48.1      |

Figure and table captions

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Table.2 Calibrated mechanical parameters of blocks and joints of the coal measures
Table.3 Calibrated mechanical parameters of blocks and joints of sandy mudstone
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Figure.1 Results of uniaxial compressive strength tests a numerical simulation of uniaxial compression, b stress – strain curve of coal, c stress – strain curve of sandy mudstone, d stress – strain curve of sandstone
Figure.2 Conceptual mechanical representation of the rockbolt element
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Figure 9 The stress and energy evolution of sandstone with and without anchorage: 

- a stress-strain relationship curve of sandstone (without anchorage);
- b stress-strain relationship curve of sandstone (with anchorage);
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Fig. 4 Model overview and dimensions

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Fig. 6 Stress and energy evolution with and without anchorage in coal: (a) stress-strain curve of coal (without anchorage); (b) a stress-strain curve of coal (without anchorage); (c) energy evolution of coal (without anchorage); (d) energy evolution of coal (anchorage).

Fig. 7 Stress and energy evolution with and without anchorage in sandy mudstone: (a) stress-strain curve of sandy mudstone (without anchorage); (b) a stress-strain curve of sandy mudstone (without anchorage); (c) energy evolution of sandy mudstone (without anchorage); (d) energy evolution of sandy mudstone (anchorage).
The magnitude of change and specific values

- A: Increase in initial damage stress (%)
- B: Increase in peak elastic strain energy (%)
- C: Increase in boundary work (kJ)
- D: Crack friction work ratio after installing rockbolt (%)

Fig. 8 The comparison between coal and sandy mudstone on different energy categories

Fig. 9 The stress and energy evolution of sandstone with and without anchorage:
- (a) stress-strain relationship curve of sandstone (without anchorage);
- (b) stress-strain relationship curve of sandstone (with anchorage);
- (c) energy evolution of sandy mudstone (without anchorage);
- (d) energy evolution of sandy mudstone (with anchorage)
Table 1 Intact rock properties and scaled rock mass properties of coal measures from Yuwu coal mine

| Lithology          | Intact rock | RQD | Rock mass |
|-------------------|-------------|-----|-----------|
|                   | $E_r$ (GPa) | $b_r$ (MPa) | $E_m$ (GPa) | $b_m$ (MPa) |
| Coal              | 2.6         | 10.8 | 0.79      | 5.1         |
| Sandy mudstone    | 5.4         | 35.5 | 3.1       | 25          |
| Sandstone         | 9.7         | 53.5 | 5.2       | 36.1        |

Table 2 Calibrated mechanical parameters of blocks and joints of the coal measures

| Density (kg/m³) | Young’s modulus (GPa) | Cohesion (MPa) | Friction (°) | Tension (MPa) |
|-----------------|-----------------------|---------------|--------------|---------------|
| Block           | 1400                  | 0.79          | 1.6 ($\varepsilon_p=0$) | 27 | 0.9 |
| Joint normal    |                       |               | 1.1 ($\varepsilon_p=0.04$) |  |  |
| Joint tangential|                       |               | 0.6 ($\varepsilon_p=0.15$) |  |  |
| Joint stiffness |                       |               | 45.2         | 1.3           | 18 | 0.4 |

Table 3 Calibrated mechanical parameters of blocks and joints of sandy mudstone

| Density (kg/m³) | Young’s modulus (GPa) | Cohesion (MPa) | Friction (°) | Tension (MPa) |
|-----------------|-----------------------|---------------|--------------|---------------|
| Block           | 1800                  | 3.1           | 8.0 ($\varepsilon_p=0$) | 27 | 2.5 |
| Joint normal    |                       |               | 5.0 ($\varepsilon_p=0.06$) |  |  |
| Joint tangential|                       |               | 2.0 ($\varepsilon_p=0.15$) |  |  |
| Joint stiffness |                       |               | 149          | 6.2           | 18 | 1.8 |

Table 4 Calibrated mechanical parameters of blocks and joints of sandstone

| Density (kg/m³) | Young’s modulus (GPa) | Cohesion (MPa) | Friction (°) | Tension (MPa) |
|-----------------|-----------------------|---------------|--------------|---------------|
| Block           | 2550                  | 5.2           | 13.0 ($\varepsilon_p=0$) | 30 | 2.5 |
| Joint normal    |                       |               | 7.0 ($\varepsilon_p=0.04$) |  |  |
| Joint tangential|                       |               | 4.0 ($\varepsilon_p=0.10$) |  |  |
| Joint stiffness |                       |               | 266          | 11.2          | 18 | 1.8 |
Table 5 Calibrated microproperties in the UDEC Trigon model to represent the rock masses

| Lithology           | Young’s modulus(GPa) | Compressive strength(MPa) |
|---------------------|-----------------------|---------------------------|
|                     | Target                | Calibrated                | Error(%) | Target | Calibrated | Error(%) |
| Coal                | 0.79                  | 0.73                      | 7        | 5.1    | 5.1        | 0        |
| Sandy mudstone      | 3.1                   | 2.9                       | 6        | 25     | 25.1       | 0.4      |
| Sandstone           | 5.2                   | 4.7                       | 9        | 36.1   | 36.3       | 5        |

Table 6 Rockbolt and anchor parameters

| Cross-section area (m²) | Elastic modulus (GPa) | Tensile yield strength (kN) | Second moment of area(m⁴) | tfstrain |
|-------------------------|-----------------------|-----------------------------|---------------------------|----------|
| Rockbolt                | $3.8 \times 10^4$     | 200                         | 198                       | $1.2 \times 10^8$ | 0.012    |

| Exposed perimeter (m) | Cohesive strength of shear coupling spring (MPa) | Stiffness of shear coupling spring (GPa) | Frictional resistance of the shear coupling spring (°) | Cohesive strength of normal coupling spring (MPa) | Stiffness of normal coupling spring (GPa) | Frictional resistance of the normal coupling spring (°) |
|-----------------------|------------------------------------------------|----------------------------------------|------------------------------------------------------|---------------------------------|----------------------------------------|------------------------------------------------------|
| Anchor parameters     | 0.07                                             | 1                                      | 8                                                     | 45                              | 200                                    | 20                                                   | 0                                                    |

Table 7 Numerical simulation scheme

| Rockbolt existence | Pre-tension (load) | Loading rate |
|--------------------|--------------------|--------------|
| Coal (soft)        | No                 | /            | 0.02m/s    |
|                     | Yes                | 0.365MPa     | 0.02m/s    |
| Sandy mudstone (medium) | No         | /            | 0.02m/s    |
|                     | Yes                | 0.365MPa     | 0.02m/s    |
| Sandstone (strong) | No                 | /            | 0.02m/s    |
|                     | Yes                | 0.365MPa     | 0.02m/s    |

Table 8 Mechanical properties and degree of damage of rock prior and after anchorage

| Rock bolt existence | Elastic modulus (GPa) | Peak strength (MPa) | Residual strength (MPa) | Degree of damage (%) |
|---------------------|-----------------------|---------------------|-------------------------|----------------------|
| Coal (soft)         | No                    | 0.73                | 5.1                     | 3.8                  | 35.1                  |
|                     | Yes                   | 0.76                | 5.4                     | 4.0                  | 30.3                  |
| Sandy mudstone (medium) | No         | 2.9                 | 25.1                    | 13                   | 46.2                  |
|                     | Yes                   | 3.5                 | 26                      | 14.7                 | 40.7                  |
| Sandstone (strong)  | No                    | 4.7                 | 36.3                    | 20.2                 | 50.8                  |
|                     | Yes                   | 4.8                 | 36.5                    | 20.3                 | 48.1                  |
Response to Referees

The authors would like to appreciate the time and efforts of editor and reviewers on this manuscript. The authors have carefully considered all the points and suggestions and have incorporated major revisions into this manuscript. Revisions in the manuscript are marked in red and listed below are the authors’ point-to-point responses to reviewers’ comments:

Response to Reviewer#1 Comments

1 Page 3 line 50 ‘In UDEC, the energy exchanged based on the work done on rock, joint and boundary’ is not clear.

The sentence has been revised for clearer understanding. This sentence comes from the UDEC user manual (udc521, Energy Calculation, pp 3-1). We have made some changes in the sentence pattern when quoting it, resulting in unclear expression. Now we replace this sentence with the original sentence ‘Energy changes determined in UDEC are performed for the intact rock, the joints and for the work done on boundaries.’

2 Page 5 line 5-6 ‘The uniaxial compressive strength and elastic modulus data derived by numerical simulation are similar to the data obtained from laboratory tests (within an error of 9%).’ Will be good to include this figure.

Figure 1 has been revised accordingly, including the laboratory test curves.

3 Why is the first paragraph of Chapter 5 listed separately? Is the development of new type of anchor lock a part of this paper?

The traditional way of bolt preload is applying torque, and it is difficult to apply high preload. Therefore, a lock for the bolt has been developed, which uses tension method to apply pre tension force to the rockbolt. This kind of rockbolt lock is relatively rare, so a paragraph was written to briefly introduce it. The lock and assembly mode are shown in Fig. 3 to let readers have a simple concept of this kind of lock.

The development process of lock is not the research content of this paper, but the research of this paper is based on the rockbolt installed with lock. So a brief introduction is made.

4 Page 6 line 40-41. ‘The indication here is that installation of tension bolts can prevent the fracture development and coalescence, such that the degree of damage was reduced.’ Not clear how you are saying this.

Fig. 5 illustrates the stress and degree of damage of rock during the process with and without anchorage. It can be seen from the figure that the mechanical properties (peak strength, residual strength, elastic modulus) of the rock mass before and after anchoring are not greatly improved, but the damage degree decreases obviously. In this paper, the damage degree is calculated based
on the crack length, the smaller the crack length, the smaller the damage degree. Therefore, the
decrease of damage degree after the installation of tension rockbolt is mainly because the rockbolt
inhibits the development of cracks. The follow-up discussion of the article also illustrates this
point, but this conclusion is a bit abrupt here, so we deleted this sentence.

The degree of damage ($D$) can be calculated as:

$$D = \frac{L_s + L_t}{L_c} \times 100\%$$

where $L_c$ is the total fracture length, $L_s$ and $L_t$ are total shear and tensile fracture lengths, respectively.

5 Page 7 line 1-2 ‘Overall, it can be concluded that tension rock bolts have insignificant
influence on strong rock’ This conclusion is based on limited research, this expression is too
absolute, it is suggest to rephrase.

The study of the whole paper shows that compared with the other two kinds of rock, the
strengthening effect of tensile rockbolt on hard rock is relatively weak. However, the sentence
pointed out by experts was too absolute at the beginning of the discussion, so we deleted it and
described it in a relatively mild way in the conclusion.

6 The arrow mark in Figure 4 is not clear. The order of annotation in the title of Figure 6 is inconsistent with that in the figure.

The figures have been revised accordingly.

Response to Reviewer#2 Comments

1. 6.1 Mechanical properties and damage analysis. Table 8 presents the degree of
damage for each rock type, with and without anchorage, at a particular strain level of 1.2%.
It is not clear to the reader the reasons that the authors chose this level for damage degree
comparison and not for example that one corresponds to the peak stress or any other strain
level.

The greater the rock strength, the smaller the strain corresponding to damage. Therefore,
when coal enters the post-peak stage, under the same strain, sandy mudstone and sandstone must
also enter the post-peak stage. Therefore, using coal as the standard, select the appropriate strain
level in the post-peak stage. The simulation found that when the strain is 1.2%, the growth of the
coal damage curve tends to be stable, and the change after the strain 1.2% is small, so the strain
1.2% is selected

2. 6.2 Analysis of the law of energy evolution Page 7, lines 28-29: Fig. 5 shows that the
degree of damage increasing in Stage 1 (before the yielding strength). Please comment and
clarify on this. Page 7, lines 34-35: This is in contrast to Fig. 5. Fig. 5 shows that the degree of damage is increasing at Stage 2. Please comment and clarify on this.

Thanks to the experts for pointing out the error, the increase in damage before the yield strength is indeed wrong. The reason for this error is that we used two FISH languages to monitor the damage, one of which did not work well. We uploaded this monitoring result by mistake.

The difference between these two FISH languages is the method of judging the threshold of crack generation. The wrong FISH language uses joint stiffness to determine whether cracks are generated, and the improved FISH language uses joint strength to determine. The stiffness of the joint changes with the increase of the normal stress [1] [2] [3]. Therefore, we want to judge whether the crack is generated by the change of stiffness. When the stiffness in the normal direction is greater than the normal stiffness, it is considered that a tensile crack is generated. If the stiffness in the shear direction is greater than the tangential stiffness, it is considered that a shear crack occurs. However, the disadvantage of this method is that the joint stiffness is too sensitive to changes in force (Figure 1) [3], which makes the monitoring results show that the damage increases before the yield strength. The strength determination method can avoid this error. The determination basis of this method is that when the force in the normal direction is greater than the tensile strength, it is determined as a tensile crack, and when the force in the shear direction is greater than the shear strength, it is determined that shear cracks were generated. Gao [4] used this method to correctly monitor the damage curve of the rock mass during uniaxial compression. We have listed the monitoring codes in the table below. The re-monitored damage curve has been uploaded, and the damage starts to rise near the yield strength, which is also consistent with the subsequent energy curve.

The description on Page 7, lines 34-35 is a translation error. We checked the original article. The original article describes the increase in damage in the second stage, not the decrease.

Figure 1 Relationship between crack stiffness and normal stress [3]
Fracture monitoring code

```python
def set_ini:
    ic = contact_head ; start of contact
    list
    loop while ic # 0 : loop through all blocks
        c_extra(ic) = 1
        ic = c_next(ic)
    endloop
    end

def _slipnum:
    whilestepping
        ic=contact_head
        loop while ic # 0
            if c_extra(ic) = 1 then
                cont_coh = fmem(c_jex(ic)+$ac_coh)
                cont_fri = mem(c_jex(ic)+$ac_phi)
                cont_ten = fmem(c_jex(ic)+$ac_ten)
                nor_stress = c_nforce(ic)/c_length(ic)
                if nor_stress <= -0.98*cont_ten then
                    c_extra(ic) = 2
                    shear_stress = abs(c_sforce(ic)/c_length(ic))
                    tmax = cont_coh + nor_stress*tan(cont_fri*3.14/180.0)
                    if shear_stress >= tmax*0.98 then
                        c_extra(ic) = 3
                    endif
                endif
            endif
            ic = c_next(ic)
        endloop
        _tens = 0
        _shears = 0
        _add = 0
        _crack = 0
        ic = contact_head
        loop while ic # 0
            _add = fmem(ic+$SKCL)
            if c_extra(ic) = 2 then
                _tens = _tens + _add
            endif
            if c_extra(ic) = 3 then
                _shears = _shears + _add
            endif
            _crack= _crack+c_length(ic)
            ic = c_next(ic)
        endloop
        _damage=(_tens + _shears )/_crack
        end

hist(_damage)
```

The various data blocks, and the offsets of items within the blocks, are contained in a series of files supplied with UDEC. These files have the extension ".FIN" (for Fish INclude file); they provide symbolic names for offsets and current numerical values. The “.FIN” files serve two purposes: first, they document the meanings of the various data items; second, the files may be CALLED from a data file – they automatically execute and define appropriate symbols. The specific code of “contact. FIN” and “jmat. FIN” in pages 4-12 and 4-15 of the UDEC Manual UDC518.

[1] Qiao Y, Hang C, Zhang L et al. 2019 Numerical simulation of fluid-solid coupling of fractured rock mass considering changes in fracture stiffness. Energy Science & Engineering. doi:10.1002/ese3.518

[2] Yao C, Jiang Q, Shao J, Zhou C. 2015 Modelling of hydro-mechanical coupling and transport in densely fractured rock mass. Eur J Environ Civ En. 19,521 - 538.

[3] Zhou J, Zhang LQ, Qi SW et al. 2020 Empirical ratio of dynamic to static stiffness for propped artificial fractures under variable normal stress. Eng Geo, 2020, 273.(https://doi.org/10.1016/j.enggeo.2020.105683)

[4] Gao FQ, Stead D. 2014 The application of a modified voronoi logic to brittle fracture modelling at the laboratory and field scale. Int J Rock Mech Min Sci. 68,1–14. (https://doi.org/10.1016/j.ijrmms.2014.02.003)

3. 7 Conclusions In this section there are some points that are mentioned extensively more than one time in Discussion section. For example, the point in page 8, lines 42-43 is
repeated in page 7, lines 1-2 and 9-10 as well as in page 8, lines 16-17. The authors should take care and reform properly the Discussion and the Conclusion section.

The authors have carefully considered the comment of the reviewer and revised the sections accordingly.

4. Is (are) there any Reference(s) for the values presented in Table 1?

The data in Table 1 comes from laboratory tests, mine geological data, borehole televiewer images, no other references.

5. If the bottom is fixed the model is wrong. Please comment and clarify. Use ‘displacement rate 0.02m/s’ instead of ‘loading rate 0.02m/s’

In UDEC, there are two loading methods for rock parameter calibration by uniaxial compression. One is fixed at the bottom and loaded at the top. Gao[1], Gao[2] and Wu[3] adopt this loading method. Another loading method is to apply loads at the top and bottom at the same time. Wu[4] adopts this loading method. Therefore, it is feasible to establish the model with fixed bottom. Fixed bottom refers to setting the bottom Y direction velocity to zero. These two loading methods have little influence on the simulation results, which can be eliminated by adjusting the loading speed.

Examples. Taking the sandy mudstone without anchorage as an example, the two loading modes are compared. In this paper, the bottom is fixed and the top loading speed is 0.02m/s. Now apply the load at the same time at the top and bottom of the model at a speed of 0.01m/s. The simulation curve comparison is shown in the figure below.

\[ \text{Axial stress (MPa)} \]
\[ \text{Axial strain (%)} \]

0.0 0.2 0.4 0.6 0.8 1.0 1.2
0 5 10 15 20 25

0.02m/s Top
0.01m/s Top and bottom

[1] Gao FQ, Stead D. 2014 The application of a modified voronoi logic to brittle fracture modelling at the laboratory and field scale. Int J Rock Mech Min Sci. 68,1–14. (https://doi.org/10.1016/j.ijrmms.2014.02.003)

[2] Gao F, Stead D, Kang H. 2015 Numerical simulation of squeezing failure in a coal mine roadway due to mining-induced stresses. Rock Mech Rock Eng. 48,1635- 1645. (https://doi.org/10.1007/s00603-014-0653-2)

[3] Wu BW, Wang XY, Bai JB et al. 2019 Study on crack evolution mechanism of roadside
backfill body in gob-side entry retaining based on UDEC Trigon model. Rock Mech Rock Eng. 52, 3385–3399. (https://doi.org/10.1007/s00603-019-01789-6)

[4] Wu WD, Bai JB, Wang XY et al. 2019. Numerical study of failure mechanisms and control techniques for a gob-side yield pillar in the Sijiazhuang coal mine, China. Rock Mech Rock Eng 52, 1231–1245. (https://doi.org/10.1007/s00603-018-1654-3)

‘Displacement rate 0.02m/s’ has been used in the article to replace ‘loading rate 0.02m/s’.
Response to Referees

The authors would like to appreciate the time and efforts of editor and reviewers on this manuscript. The authors have carefully considered all the points and suggestions and have incorporated revisions into this manuscript. Revisions in the manuscript are marked in red and listed below are the authors’ point-to-point responses to reviewers’ comments:

Response to Associate Editor Comments

1 Please correct "targe" in Table 5

Table 5 has been revised accordingly. “targe” has been corrected to “target”.

2 Is "tension" in Tables 2, 3, 4 "tensile strength"?

Yes. The “tension” in Tables 2, 3, 4 has been changed to “tensile strength”.

3 Rock mass does not have plural form. Please correct rock masses to rock mass

We have made corrections and marked red in the manuscript.

4 The sentence "The installation of tension rock bolts was more suitable for medium hard rock (e.g. sandy mudstone), whereas it was not effective for hard rock (e.g. sandstone)" is repeated in the summary text twice.

The repeated sentence has been deleted.

5 Fracture development of anchor was monitored by FISH function. -> why are you talking about fractures of the anchor?

The failure process of an anchor under load is not only the process of the generation, expansion, and penetration of fractures, but also the process of energy evolution. There is a certain correlation between them, and they are basically consistent at some key inflection points. In this article, monitoring the fracture length through FISH language defines the damage degree of the anchors, and assists in evaluating the supporting effects of different lithological anchors. Therefore, it is necessary to monitor fractures.

6 What is a Coulomb-sliding model. That is not a standard term.

In UDEC, the Coulomb slip model is a joint behavior model. The Coulomb slip models are described in Section 1.2.4 in Theory and Background (UDEC Manual).

We have corrected Coulomb-sliding model to Coulomb slip model. Coulomb slip model is a standard term.