**Research Article**

**Numerical Simulation Study on the Influence of Mine Earthquake on the Bolt Stress**

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Mine earthquake, as an underground disaster that occurs frequently, has a great impact on coal mine roadway and support. The stability analysis of the bolt support in roadway under different mine earthquake magnitudes is a key issue to be solved urgently in mining fields. This paper attempted to simulate the occurrence state of mine earthquake with explosive blasting process and verified it with actual coal mine microseismic monitoring data. ANSYS/LSDYNA software was used to analyze the impact of magnitude and location of mine earthquake hypocenter on the stability of bolt support and the dynamic stress characteristics of bolt. The results showed that with the increase in source energy of mine earthquake, the damage location of bolt mainly appears in the front of bolt and the loading position has no obvious change, but there is stress wave superposition effect, which deepens the damage of bolt. The bolts in the middle of the lane and the middle of the roof are greatly affected, so the support strength should be strengthened in these places. In addition, this paper compared the safety factor of bolt and the supporting effect of different schemes from three aspects such as roof subsidence, axial stress of bolt, and safety factor of bolt and then put forward a more economical and effective supporting scheme.

**1. Introduction**

In the process of coal mine production and construction, bolt support is often used to maintain the stability and integrity of surrounding rock and prevent deformation and failure of surrounding rock. Bolt support has the advantages of high efficiency and low cost, and it is widely used in coal mine industry. However, the dynamic load produced by stope disturbance, blasting, mine car, and so on will disturb the roadway surrounding rock and bolt bearing static load, resulting in the combined action of dynamic and static loads, which greatly affects the effect of bolt support. Therefore, experts and scholars at home and abroad have done a lot of research on the influence of external dynamic load on roadway surrounding rock and bolt support.

Zang et al. [1] studied the deformation process of surrounding rock under different static loads and different disturbances through numerical simulation and put forward the optimizing scheme of combined support. Wang et al. [2] analyzed the deformation mechanism of the roadway and explained the importance of the support scheme and support form for the soft rock roadway. Li et al. [3] analyzed the deformation process of surrounding rock under dynamic load of blasting and earthquake and studied the rock failure process under dynamic load disturbance. Some researchers have studied the vibration velocity, amplitude, and frequency, and they found the attenuation law of stress waves in different geological conditions and clarified the relationship between the location of explosion source and charge [4–6]. Gao et al. [7] used the new UDEC Trigon method to carry out numerical simulation, studied the extrusion damage process of roadway under high stress, and verified it with a real field case. Singh [8] studied the impact failure characteristics of blasting on mines, monitored the stress-strain...
numerical changes of roof and bolt, and analyzed the vibration and deformation of coal roadway before and after blasting. The research mentioned above mainly studied the influence of static load or blasting dynamic load. The mine earthquake is induced by static load under different dynamic load. Therefore, the study of mine earthquake is indispensable. Shen et al. [9] carried out microseismic monitoring on the roof of a coal wall working face in Australia and revealed the influence of microseismic precursors on the instability characteristics of roadway surrounding rock. According to the source model and date of microseismic monitoring, Cai et al. [10] proposed a quantitative method to determine the formation of rock fractures. Driad-Lebeau et al. [11] analyzed a rock burst with a magnitude of 3.6 that occurred in the Melebach Mine in France and found that the mine earthquake is the precursor of rock burst. Meng et al. [12] developed a new software package by combining discrete element and finite element methods, which can be used to process complex microstructure materials and can be applied to the microscopic analysis of rock forces. Zhu et al. [13] used 3DEC software to develop a numerical model of the cable and analyzed the force characteristics of the cable during the slope deformation process. The new development model provides a new idea for subsequent slope failure and cable control. Zhu et al. [14] analyzed the collision characteristics of different rock types and different heights through orthogonal experiments and established the relationship between different shapes, different heights, and the thickness of the buffer layer. Wang et al. [15] proposed a high-strength bolt grouting method to automatically cut the roof into a roadway and made a comparative analysis of surrounding rock deformation using evaluation indicators such as roadway deformation and anchor cable strength utilization. Wang et al. [16] studied the fracture evolution characteristics of marble under fatigue cyclic loading and unloading of confining pressure, providing a theoretical basis for the prediction of rock dynamic disasters. Wang et al. [17] proposed an automatic forming roadway mining method without advance excavation, which can effectively improve the control of underground surrounding rock and reduce the risk of damage to the surrounding rock. Li et al. [18] proposed a new dynamic method to evaluate the stability of the sidewall through microseismic data and continuum modeling, which was verified by a case. Wu et al. [19, 20] studied the creep model of rocks and made continuous improvements to the model, which provided a reference for the long-term stability of deep rocks.

To sum up, in the studies on the influence of dynamic load on surrounding rock and bolt support, the main research method adopted by most scholars at home and abroad is use external force to impact the model to generate impact load and use numerical simulation software to simulate the influence of dynamic loads such as explosion load and transverse load on roadway surrounding rock and bolt. However, there is no basis to determine these simulated impact load values, and these data do not match with field examples [21–23]. There are two main methods used in mine earthquake simulation, one is to apply a sine wave and the other is to input measured seismic data. The former needs to take the incident angle of waves into consideration, and it can only apply incident waves to a single point or surface. The latter can only input the seismic waves according to filed measured data, which cannot be carried out when the field seismic monitoring data are insufficient. Both methods have their drawbacks.

Based on the geological survey of Baodian Coal Mine, this paper established a numerical model of explosive blasting, monitored the velocity-time history vibration curves around the roadway, transformed them into the spectrum by fast Fourier transform, and then compared them with the field measured microseismic curves. By these, the paper proved the feasibility of using explosive blasting to simulate mine earthquakes. Moreover, this paper analyzed the relationship between explosive quantity and mine earthquake magnitude (energy), simulated and analyzed the static stress characteristics of bolts under different source energy, explored the changed law of dynamic axial stress of bolts, analyzed the influence of bolts spacing on bolt safety factor, and gave the best bolt spacing value in the support scheme.

2. Engineering Background and Preparation for Numerical Simulation

2.1. Engineering Background. The main coal seam in Baodian Coal Mine is 3 layers of coal (3 upper and 3 lower). At present, the main mining face No.103 upper 02 is located in the northern part of the No. 10 mining area, which is the second section of the upper coal seam in the No. 10 mining area; the working face adjacent to the north and south of it has been mined. The mining work of the No. 103 upper 02 face has been carried out in an isolated island state where three sides were mined out. Under such a state, there is a high possibility of mine earthquake.

In the mining process of No.103 upper 02 working face, the frequency of mine earthquakes with energy greater than 10^5 J has exceeded 40 times, the influence range of mine earthquake has spread to buildings above the ground, and the most serious impact is the occurrence of underground rock pressure. Figure 1 is a partial single-channel microseismic waveform spectrum of a typical strong mine earthquake. The energy released by the mine earthquake is \( E = 5.32 \times 10^6 \) J, and it can be seen from this spectrum that the frequency is mainly in the range of 0–10 Hz, which is a typical low-frequency signal.

2.2. Preparatory Work for Numerical Simulation

2.2.1. Determination of Mine Earthquake Simulation Method. Up to now, there are two main methods to study mine earthquake simulation: the one is to input a sine wave to replace mine earthquake wave and the other is to make a model based on the specific data of three-dimensional velocity-time history recorded by measured microseismic records. The disadvantage of the first method is that the sine
waveform input by analog is regular, which is not the case in the actual microseismic monitoring waveform; in addition, the input of sine wave can only be for a single point or surface, so it is difficult to determine the incident angle of the wave in this process. The disadvantage of the second method is that the input of each seismic waveform requires field measured data as basis. If there is a lack of field monitoring data of mine earthquake, it is impossible to predict and simulate mine earthquake. Although the waveforms of blasting and mine earthquake are not exactly the same, they are very close under suitable conditions. Table 1 shows the comparison of some basic characteristics of blasting and mine earthquake. In the simulation process, we can make the blasting waveform infinitely close to the mine earthquake waveform by creating suitable conditions so that simulation results can be closer to the actual situation. In addition to these advantages, the amount of explosive required to reach the energy of explosion hypocenter can also be obtained according to the simulation calculation without field measurement so as to study the mine earthquakes with different energy levels.

2.2.2. Establishment of Numerical Simulation Model. The numerical simulation model in this paper took the geological mining conditions of Baodian Mine in Yanzhou No. 103 upper 02 working face as reference. According to the research purpose and based on the actual mining condition, the model size and rock layer distribution were simplified. The size of the model is $200\times326$ m, and the size of roadway is $4\times5$ m. The thickness of the coal seam in this model is 6 m, the floor stratum is 50 m, the immediate roof rock stratum is 70 m, and the super-thick stratum is 200 m. When this model was used for static analysis, a horizontal displacement constraint was imposed on its left and right boundaries and a full displacement constraint was imposed on its bottom boundary. In order to simulate the buried depth of the uppermost strata of 200 m, an equivalent vertical stress load of 5 MPa was applied to the top of the model, and the gravitational acceleration was $9.8 \text{ m/s}^2$. In explicit analysis, it is necessary to delete the displacement constraints imposed on the model in static analysis, re-impose horizontal and vertical displacement constraints on the explicit interface, and add nonreflective boundary
conditions to the boundary. The initial model is shown in Figure 2. The material parameters of rock used in the simulation are shown in Table 2. The bolt specification is \( \Phi 20 \times 2200 \text{mm} \), with 3 bolts placed in the roof and 5 bolts placed in the walls of roadway for each side. Bolt parameters are shown in Table 2.

2.2.3. Study on the Relationship between Explosive Quantity and Source Energy. The blasting energy released by 1 kg TNT explosive is 3900 – 4400 kJ/kg, and the average value is 4200 kJ/kg in this study. In the simulation, the density of the selected TNT explosive is 1180 kg/m\(^3\), the quantity of explosive is set to m and the unit is kg, and the explosive source energy \( E = 4200 \text{mJ} \). The explosive quantity was converted into the explosive unit volume in this model, and 1 kg explosive corresponds to the explosive unit volume as \( 0.85 \times 10^{-3} \text{m}^3 \). Because the explosive unit is too small, it is difficult to determine its value in the model, so the initial internal energy density of explosive parameters \( E_0 \) was appropriately modified in the K file to reduce its multiple and increase the volume of the selected explosive unit correspondingly. In this way, the formula for selecting explosive units in the model was as follows:

\[
V = \frac{m}{\rho} \times b. \tag{1}
\]

In this formula, \( b \) represents the ratio of the actual initial internal energy density to the initial internal energy density \( E_0 \) of the explosive parameter in the K file; \( V \) represents that explosive unit volume, and the unit is m\(^3\); \( m \) represents explosive quantity, and the unit is kg; and \( \rho \) represents the density of explosives, and the unit is kg/m\(^3\).

The relationship between explosive quantity and energy calculated from the blasting energy of 1 kg explosive in formula (1) is shown in Figure 3.

Based on the general situation of the No. 103 upper 02 working face, the hypocentral location was set at the lower part of the super-thick rock strata and 25 m above the coal seam. The linear distance between the hypocentral and the center of the roadway is about 101 m, as shown in Figure 2. According to Figure 3 and formula (1), the explosive quantity was selected to be about 1.27 kg, which was converted into a corresponding explosive unit volume of about 45 m\(^3\). The velocity-time history curve of the monitoring point obtained by simulation is shown in Figure 4.

The speed-time data in Figure 4 are imported into Origin, and the corresponding frequency-amplitude curve was obtained through fast Fourier transform, as shown in Figure 5.

Comparing Figures 4 and 5 with Figure 1, in the velocity-time history curve, the measured magnitude is \( 10^{-4} \text{m/s} \), the maximum vibration speed is about \( 6.3 \times 10^{-4} \text{m/s} \), and the vibration duration is about 4.5 s. In the spectrum, the measured frequency is mainly distributed in the range of 0–10 Hz and the maximum amplitude is in the range of 0.28–0.32. In the simulated speed-time history curve, the magnitude is also \( 10^{-4} \text{m/s} \), the maximum vibration speed is about \( 6.5 \times 10^{-4} \text{m/s} \), and the vibration duration is about 2 s. In the simulated spectrum, the frequency is mainly distributed in the range of 0–6 Hz and the maximum amplitude is about 0.31. This is because microseismic signals have longer duration and higher frequency than blasting signals.

The amplitude of microseismic signal is generally in the range of 5–300 mV, and the amplitude of blasting signal is generally in the range of 50–300 mV. Although the peak value of velocity and amplitude can be made as close as possible to the measured values by simulation, errors are inevitable. In the simulation, the error between the maximum vibration velocity and the measured value is about 3%. This paper mainly studied the influence of the maximum peak value on roadway and support, and numerical simulation can achieve this goal, which shows that the simulation results are similar to the engineering examples and they can meet the requirements of this paper. Formula (1) can be applied to numerical simulation.

### 3. Analysis on Numerical Simulation Results

Figures 6(a) and 6(b) are the comparison of vertical stress diagrams of roadways with and without support. Figure 6(c) is an axial stress diagram of bolt in static state. Figure 7 shows the amount of displacement before and after support. Figures 6(a) and 6(b) indicate that the vertical stress in the area around the roof and floor decreases after support, and in the area where the stress decrease is much larger than that without support, the stress is reduced by about 30%. At the four corners of the roadway, the stress level at the stress concentration and area where the stress concentration occurs is also reduced compared with that without support. The reduction is about 50%. Compared with unsupported roadway, bolt support obviously improves the stress state of surrounding rock. The improvement is manifested in a 10% increase in the stress of the roof and 20% increase in the stress of the walls of roadway, which shows that the reinforcement effect is obvious. As can be seen from Figure 6(c), the bolt undergoes slight bending deformation inside the roadway, but the deformation position is concentrated in the middle and upper part of the bolt, which indicates that the axial stress distribution on the bolt is uneven. The bolt near the roadway is stressed greatly, and with the distance from the roadway, the axial stress of the bolt becomes smaller and smaller. From Figure 7, it can be seen that with the addition of bolt support, the roof subsidence decreases and the failure and deformation trend of roadway surrounding rock weakens. Compared with the case of no support, the roof displacement decreases by about 47% on average, which shows that in this

| Event type       | Energy | As/Ap       | Period | Surface wave | Law of time | Damped wave |
|------------------|--------|-------------|--------|--------------|-------------|-------------|
| Blasting         | Weak   | Small       | Short  | Yes          | Many        | Quick       |
| Mine earthquake  | Weak   | Relatively small | Long   | Yes          | Yes         | Relatively quick |
simulation, bolt support plays a certain protective role in controlling the surrounding rock of roadway and inhibits the failure and deformation of surrounding rock.

The total number of bolts used in this paper is 13, which are numbered and divided into three units for analysis. The bolts placed at the walls of the roadway are numbered 1, 2, 3, 4, and 5 from top to bottom, and the bolt spacing is 1 m. The bolts placed at the roof of roadway are numbered 11, 12, and 13 from left to right, and the bolt spacing is 1 m. The sectional diagram of bolts is shown in Figure 8. In this

**Table 2: Material parameters of rock.**

| Lithology       | Thickness (m) | Density (kg/m³) | Modulus of elasticity ($10^{10}$ Pa) | Poisson’s ratio |
|-----------------|---------------|-----------------|--------------------------------------|-----------------|
| Fine sandstone  | 200           | 2700            | 3.34                                 | 0.23            |
| Medium-coarse   | 70            | 2600            | 3.6                                  | 0.26            |
| Coal seam       | 6             | 1300            | 0.35                                 | 0.32            |
| Fine sandstone  | 50            | 2487            | 1.35                                 | 0.3             |

| Component       | Modulus of elasticity ($10^{10}$ Pa) | Poisson’s ratio | Density (kg/m³) |
|-----------------|--------------------------------------|-----------------|-----------------|
| Bolts           | 23                                   | 0.26            | 5800            |

**Figure 3:** Relationship between explosive quantity and energy.

**Figure 4:** Velocity-time history curve of monitoring point in simulation.
Figure 5: Spectrum of monitoring point in simulation.

Figure 6: Cloud diagram of stress distribution and bolt force in surrounding rock before and after support: (a) vertical stress diagram without support; (b) vertical stress diagram under supporting condition; (c) axial stress diagram in static state.

Figure 7: Monitoring point at roof and subsidence of roof: (a) layout of roof monitoring points; (b) roof subsidence curve.
paper, the stress characteristics of bolt under different mine earthquake energy are analyzed.

3.1. Stress Analysis of Bolts When Mine Earthquake Energy Is $10^6 J$. Figure 9 shows the stress variation curve of some bolts as the mine earthquake energy is $10^6 J$. Figure 10 shows the propagation process of stress wave around the roadway. As can be seen from the figure, the axial stress value of the bolts fluctuates up and down, which indicates that the bolt is under the action of tension most of the time. Most of the stress concentration occurs in the A and B sections of bolts. After the occurrence of mine earthquake, the A section of the bolt near the roadway is in an obvious tensile state, and the bolt body is stretched and elongated to varying degrees. The stress borne by the bolt body is mainly the tensile stress caused by the reflection of vibration stress wave through roadway, and both the stress value and strain value increase after mine earthquake. The total stress of bolt is the result of superposition of static axial stress and additional stress caused by deformation of surrounding rock under dynamic load. The bolt is always affected by tensile stress and may be damaged or even broken under the impact of mine earthquake with high energy. Figure 11 is used to observe the axial stress and strain changes of the bolt after the mine earthquake. It can be seen that the tensile deformation of No. 2 and No. 4 bolts at roadway's side is relatively serious, and the deformation mainly occurs in section A of the bolt. The stress and strain value of No. 12 bolt in the middle of the roof is also greater than that of the other two bolts located there, which indicates that these bolts are greatly affected by mine earthquakes.

3.2. Stress Analysis of Bolts When Mine Earthquake Energy Is $10^8 J$. Figure 12 shows the axial stress increment curve of each bolt on the walls and roof of the roadway and the strain change curve of some bolts. As can be seen from the figure, source energy of mine earthquake increases from $10^6$ J to $10^8$ J. With the increase in energy, the position where the stress concentration occurs gradually moves to B section in bolt and the deformation degree of the bolt increases. According to the strain curve, there appear two strain fluctuation peaks, and the stress cumulative effect of bolt is more serious. The No. 2 and No. 4 bolts at roadway's side are under severe stress. With the increase in energy, the stress degree of No. 3 bolt also increases, which deepens the deformation degree of this bolt. The increase in stress and strain in the middle area of roadway' side is significantly higher than that in other areas. The deformation degree of bolt located in the middle area is also greater, which needs to be treated. The stress in B section of No. 12 bolt increases obviously, and it indicates that the middle part of roof will bear more stress concentration effect after the energy increases, which aggravates the damage of bolt.

3.3. Comparison of the Advantages and Disadvantages of Bolt Support Scheme under the Action of Mine Earthquake Action. According to $\log E = 1.8 + 1.9M_L$ (Polish empirical formula), mine earthquake energy can be converted into Richter scale value, wherein $10^4 J$ is converted into ML = 1.15, $10^6 J$ is converted into ML = 2.2, and $10^8 J$ is converted into ML = 3.2. From the above analysis, it can be seen that no matter in the roadway's side or the roof of roadway, the stress of the bolt is concentrated in the A and B sections of the bolt. Therefore, based on this research, this paper proposed a modification scheme for the parameters of bolt. This section mainly discussed the changes in axial stress of bolt and displacement of surrounding rock when the bolt spacing is 1 m, 1.2 m, and 1.5 m, respectively, and analyzed the safety factor of bolt in different supporting schemes. When the bolt spacing is set to 1 m, the corresponding support scheme requires 5 bolts to be set at the roadway's, which are numbered 1, 2, 3, 4, and 5 from top to bottom, and 3 bolts to be set on the roof, which are numbered 11, 12, and 13 from left to right. When the bolt spacing is set at 1.2 m, the corresponding support scheme requires to set up 4 bolts numbered 1, 2, 3, and 4 from top to bottom at the roadway's side and 3 bolts numbered 5, 6, and 7 from left to right at the roof; when the bolts spacing is set at 1.5 m, the corresponding support scheme requires 3 bolts numbered 1, 2, and 3 from top to bottom at the roadway's side and 3 bolts numbered 4, 5, and 6 from left to right at the roof.

Figure 13(a) shows the roof subsidence curve of roadway surrounding rock corresponding to different support schemes when the magnitude of mine earthquake is 2.2. The location of the monitoring point is shown in Figure 7(a). As can be seen from the figure, with the increase in bolt spacing, the roof subsidence of roadway gradually increases either. Compared with the bolt spacing of 1 m, when the bolt spacing is 1.2 m, the roof subsidence increases by 11.2 mm on average, and when the bolt spacing is 1.5 m, the roof subsidence increases by 58.4 mm on average. According to the above analysis results, this paper selected the most affected A section to further analyze about the stress state of the bolt. Figures 13(b) and 13(c) show an axial stress time history diagram of the bolt under different magnitudes of the mine earthquake. From this figure, it can be seen that under a mine earthquake with the magnitude of 2.2, with the increase in bolt spacing, the peak value of axial stress borne by bolt A section increases either. When bolt spacing is 1.2 m, the peak axial stress of bolt A section reaches 60 kN, and when the spacing is 1.5 m, the peak axial stress of bolt A section reaches 80 kN; as the magnitude of mine earthquake increases to 3.2, with the change of bolt spacing, the peak axial stress of bolt A section changes more significantly. When the bolt spacing is 1.2 m, the peak axial stress of bolt A section reaches 90 kN, and when the spacing is 1.5 m, the peak
Figure 10: Transmission process diagram of reflected stress wave around roadway: (a) 1 s; (b) 1.5 s; (c) 2 s.

Figure 9: Axial stress-strain curve of partial bolt: (a) anchor axial stress time history curve of bolt No. 1; (b) anchor axial stress time history curve of bolt No. 3; (c) axial strain time history curve of bolt No. 1; (d) Axial strain time history curve of bolt No. 3.
axial stress of bolt A section reaches 115 kN. The bearing limit of a bolt is 150 kN, and the calculation formula of safety factor of bolt is as follows:

\[ \frac{R_K}{N_K} \geq K_t. \]  \hspace{1cm} (2)

In this formula, \( N_K \) is the measured axial stress value of the bolt, and the unit is kN; \( R_K \) is the ultimate value of bolt bearing capacity, and the unit is kN; and \( K_t \) is the tensile safety factor of bolt. When \( K_t > 1 \), the bolt can play a reinforcement role, indicating that the scheme is safe. When \( K_t < 1 \), the axial stress of bolt in rock mass will exceed its ultimate bearing capacity and the bolt will be broken, so the roadway support will be at risk of failure. When \( K_t = 1 \), the bolt is in a critical state and is in danger of being pulled off at any time.

From the above analysis results, when the magnitude of the mine earthquake is 2.2, the safety factor of the bolt in the support scheme with the bolt spacing of 1 m is 2.7, the safety factor of the bolt in the support scheme with the bolt spacing of 1.2 m is 2.5, and the safety factor of the bolt in the support scheme with the bolt spacing of 1.5 m is 1.875. When the magnitude of the mine earthquake is 3.2, the safety factor of the bolt in the support scheme with the bolt spacing of 1 m is 2.2, the safety factor of the bolt in the support scheme with the bolt spacing of 1.2 m is 1.6, and the safety factor of the bolt in the support scheme with the bolt spacing of 1.5 m is 1.3. Taking into account factors such as technology,
Figure 12: Stress increment curve of each bolt and strain time history curve of some bolts: (a) axial stress increment curve of each bolt in roadway side; (b) axial stress increment curve of each bolt of roof; (c) time history curve of axial strain of No. 1 bolt; (d) time history curve of axial strain of No. 3 bolt.

Figure 13: Continued.
4. Conclusion

(1) Based on the geological conditions of Baodian Coal Mine and the measured data of mine earthquake, a corresponding numerical model was established by using the numerical simulation software ANSYS/LSDYNA, and the hypocenter of mine earthquake and other factors was studied by numerical simulation. By comparing the simulation results with field data, the feasibility of the simulation is proved.

(2) Numerical simulation software was used to study the axial stress state of bolt and the bolt support effect under the influence of static load and source energy of different magnitudes. The results showed that with the increase in source energy of mine earthquake, the stress concentration on the bolt gradually changes from only appearing in A section to simultaneously appearing in A section and B section. The bolts which are greatly disturbed by mine earthquake are located in the middle of roadway’s side and roof, so it is necessary to strengthen prevention and control in these areas.

(3) From three aspects of roof subsidence, axial stress of bolt, and safety factor of bolt, the support effect of different schemes and safety factor of bolt were compared (the bolt spacing was 1 m, 1.2 m, and 1.5 m). And the factors of technology, economy, and construction were comprehensively taken into consideration. This paper proposed that it is reasonable and feasible to arrange bolts at a spacing of 1.2 m.

Data Availability

The data used in this article are available through e-mail from the corresponding author.

Conflicts of Interest

The authors declare that they have no conflicts of interest.

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