Evaluation on the Pressure Bump and Outburst Compound Dynamic Disaster

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Abstract. After coal mining into the deep, pressure bump and outburst coexist, which are mutual inducers and compounds that increase the mining risk and control difficulty. The hazard evaluation of compound dynamic disaster before mining can predict the dangerous area and take preventive measures to reduce casualties and property losses. This paper establishes a mechanical model of “stress surrounding rock gas support,” taking into account the stress environment of coal body, impact tendency of coal body, gas pressure, roadway structure, and support stress. Taking, for example, 22220 working face of a mine, according to the history of composite dynamic disaster in the same level coal seam, the composite dynamic disaster evaluation grade is divided. The minimum critical stress index of composite dynamic disaster is taken as the medium risk evaluation standard, while the maximum is taken as the strong risk evaluation standard. The critical stress index of each area of the working face is calculated, and the risk grade is divided. The results show that the strong dangerous area of working face is mainly located in the open cut and the distance between the track transport and the open cut is 20~250 m, and the medium dangerous area is mainly located in the air return roadway 40~250 m. Compared with the comprehensive index method, the critical stress index method can predict the relationship between support strength, gas concentration after drainage, and composite dynamic disaster. Moreover, the evaluation results tend to be quantitative and conservative.

1. Introduction
A total of 1044 outburst mines and 121 pressure bump mines coexist in China, as the coal mining goes into deeper. The dynamic disaster often reveals the properties in which compound pressure bump and outburst coexist. Until November 2020, 18 dynamic disaster mines in which compound pressure bump and outburst coexist have been accurately reported, such as the coexistence of outburst which was induced by the pressure bump in the 12011 working face of Xinyi coal mine and the pressure bump which was induced by coexistence of outburst in the 1330 working face of Yangou coal mine. Since two disasters are mutual inducers and compounds during the processes of breed, occurrence, development, and control, the hazard grade of those dynamic disaster mines is difficult to be evaluated.

The hazard evaluation of dynamic disaster is the base of prevention and prediction, and only if accurate risk evaluation is made can the reasonable prevention and control technologies be adopted and the relevant prediction system be established. The hazard evaluation of dynamic disaster also affects the mining distribution and causes problems of losing or leaving coal, resulting in waste coal resources. Currently, the major hazard grade evaluation methods of
pressure bump are based on the stress and surrounding rock classification \[5\], unascertained measurement model \[6\], blind number theory \[7\] and variable weight identification model \[8\]. These methods mostly choose mining and geological conditions as the index of method. There are also many evaluation methods of coal and gas outburst, such as the evaluation methods of coal and gas outburst base on rough set\[9\], multiparaterization\[10\], unascertained measure model\[11\], gray target decision model\[12\], fuzzy comprehensive evaluation\[13\], and GRNN model\[14\], among others. Most of the indexes in these methods include maximum gas pressure of coal seam \(P\), the damage forms of soft layer coal, initial speed of gas diffusion from coal \(\Delta p\), and the consistent coefficient of coal \(f\), which base on the rules in “regulations of coal and gas outburst prevention.” Few research exist about the compound dynamic disaster of pressure bump and coal and gas outburst. Some researchers such as OUYANG Zhenhua \[15\] considered the influence of gas on the risk of rock burst in a coalmine and brought up an evaluation method that included the improved comprehensive index of coal and gas outburst\[16\]. This method gave the parameter to gas properties, but it did not consider the coupled relation between the gas pressure and surrounding rock pressure. PAN Yishan used disturbance response instability theory to build the relations between disturbance quantity, response quantity, and control quantity in the circular roadway and proposed the critical index of compound dynamic disaster of pressure bump and coal and gas outburst in circular roadway. After comprehensively considering with the surrounding rock stress, gas pressure, and supporting stress, this paper evaluated the risk of compound dynamic disaster of pressure bump and coal and gas outburst, based on the critical index method of the 22220 working face and provided data for the hazard evaluation before the mining of composite dynamic disaster mine.

2. Critical stress index method

Critical stress index method is a hazard evaluation method of compound dynamic disaster that is based on the stress environment and impact tendency of coal body. It considers disturbance response instability theory after the rigorous dynamical derivation, the relations between the critical stress, the UCS (uniaxial compressive strength) of coal body, and the compound dynamic disaster index when the circular roadway pressure bump is derived. Figure 1 shows the model of compound dynamic disaster in circular roadway.

![Figure 1. Model of compound dynamic disaster in circular roadway](image)

According to the effective stress principle, the effective radial stress component and effective circumferential stress component are \(\bar{\sigma}_r = \sigma_r - \Delta p\), \(\bar{\sigma}_\theta = \sigma_\theta - \Delta p\), respectively. As a result, the equation of equilibrium of effective stress is

\[
\frac{d\bar{\sigma}_r}{dr} + \frac{d(\Delta p)}{dr} - \frac{\bar{\sigma}_\theta - \bar{\sigma}_r}{r} = 0 \quad (2.1)
\]
where $\alpha$ is the effective stress coefficient and $p$ is the gas pressure. Both $\alpha$ and $p$ are functions of $r$. If $f_1(r) = -\frac{d(\alpha p)}{dr}$, then

$$\frac{d\sigma_r}{dr} - \frac{\sigma_\theta - \sigma_r}{r} = f_1(r) \quad (2.2)$$

The strain–displacement equations are as follows:

$$\left\{ \begin{array}{l} \varepsilon_r = \frac{du}{dr} \\ \varepsilon_\theta = \frac{u}{r} \\ \gamma_{r\theta} = 0 \end{array} \right. \quad (2.3)$$

The constitutive relations in elastic area can be shown as

$$\left\{ \begin{array}{l} \sigma_r = \bar{E}(\varepsilon_r + \nu\varepsilon_\theta) \\ \sigma_\theta = \bar{E}(\varepsilon_\theta + \nu\varepsilon_r) \end{array} \right. \quad (2.4)$$

where $\varepsilon_r$ is the radial strain, $\varepsilon_\theta$ is the circumferential strain, $\gamma_{r\theta}$ is the shearing strain, and $u$ is the radial displacement. $\bar{E} = \frac{E(1-\nu)}{(1+\nu)(1-2\nu)}$, where $E$ is the elastic modulus of coal and $\nu$ is Poisson’s ratio.

Plastic softening area satisfies the following:

$$\frac{\sigma_\theta}{1-D} = m \frac{\sigma_r}{1-D} + \sigma_c \quad (2.5)$$

where $D$ is the damage variable. $m = \frac{1+\sin \varphi}{1-\sin \varphi}$, where $\varphi$ is the internal friction angle which is assumed as $30^\circ$.

$$P^* = \frac{\sigma_c}{2} \left( 1 + \frac{1}{K} \right) \eta + \beta p_g \quad (2.6)$$

where $P^*$ is the critical stress when compound dynamic disaster occurs; $\sigma_c$ is the UCS of coal body; $K = \frac{\lambda}{E}$, where $K$ is the index of impact tendency of coal body (equal numerically to the burst energy index); and $\eta = 1 + \frac{4p_g}{\sigma_c}$, where, $\eta$ is correction factor, $p_g$ is support stress, and $\beta$ is the equivalent porosity of coal seam.

$$K_e = \frac{P^*}{P} \quad (2.7)$$

where $K_e$ is the critical stress index, $\omega$ is the stress concentration index, and $P^*$ is the approximate stress.

As the equation shows, when the approximate stress is large and the critical stress is small, then the hazard of compound dynamic disaster will be large and vice versa.

3. Determination of the critical stress index evaluation standard

The 22220 working face is located in the west of second district of second mining level. The effective stoping area starts in the east from the protective coal pillar of the second district downhill and ends in the west from the boundary of the second district mining area. The south of working face is the 22201 working face which had finished stoping, and the north of working face is the raw coal. According to the geological exploration information and the status of exposing coal seam, the thickness of coal seam in the designated area of the working face is between 4.20 and 6.60 m, about 5.24 m on average, and the dip angle of coal seam is between 3 and 12°, about 6.4° on average. The designed mining height of 22220 working face is 5.24 m, the average length of mining section along the strike is 1036 m, the average inclined length is 195.6 m, and the exploitable reserve is 1.3036 million tons. The 22220 working face used the longwall mining as the methods of coal mining and the hydraulic support as the support. Figure 2 shows the layout of roadway in the working face.
The original gas content in the coal seam, which was measured within the 700-m inner stoping area of 22220 working faces, accounts to 4.19~5.62 m$^3$/t. The original gas pressure was 0.21~0.56 MPa, for the 336-m outer stoping area. The original gas content in coal seam was 2.62~6.44 m$^3$/t, and the original gas pressure was 0.11~0.50 MPa. Four faults affect stoping, namely, F1, F2, F3, and F4 (details are summarized in Table 1).

Table 1: Information of the faults affecting stoping of 22220 working face

| Serial number | Direction | Angle of inclination (°) | Direction of inclined | Property | Difference of height (m) | Range of influence (m) | Degree of influence |
|---------------|-----------|--------------------------|-----------------------|----------|-------------------------|------------------------|-------------------|
| F1            | N340°W    | 55                       | NE70°                 | Positive | 4.7                     | 50                     | 70                | Serious           |
| F2            | NW308°~352° | 50                       | SW194°~218°           | Positive | 2.2                     | 0                      | 0                 | None              |
| F3            | NE27°     | 25                       | SE117°                | Positive | 0.5                     | 3                      | 20                | Less              |
| F4            | S252°W    | 45~65                    | SE162°                | Positive | 3.80~3.40               | 65                     | 90                | Larger            |

There has been 11 pressure bump events in the second mining level mining area, seven of which occurred in the track transport, air return roadway, and open cut of three working faces (details are shown presented in Table 2), while the remaining four events occurred in the rock layer.

Table 2: Second mining level mining area dynamic disaster statistics

| Serial number | Occurred Time  | Occurred site                | Vertical Depth (m) | Approximate stress (MPa) |
|---------------|----------------|------------------------------|--------------------|--------------------------|
| 1             | 2002.06.16     | 22120 air return roadway    | 837                | 29.30                    |
| 2             | 2005.01.03     | 22120 air return roadway    | 852                | 29.82                    |
| 3             | 2007.08.06     | 22062 track transport       | 800                | 30.00                    |
| 4             | 2008.04.23     | 22062 air return roadway    | 771                | 28.91                    |
| 5             | 2009.12.17     | 22122 open cut              | 835                | 31.31                    |
| 6             | 2010.01.15     | 22122 open cut              | 842                | 31.58                    |
| 7             | 2010.02.19     | 22122 open cut              | 853                | 31.99                    |

Taking the UCS as 13.99 MPa and the burst energy index as 4.23, the designed column and row distances between supporting bolts in working face were 750 mm × 750 mm, and the cables were arranged in a row of four with 1500-mm row distances (the column distance between two middle cables was 750 mm, and the column distance between two cables on two sides was 1125 mm) and
support with the cables which were arranged in a row of three with $1125 \times 1500$ mm column and row distances. The total support stress was 0.402 MPa.

As presented in Table 3, the critical stress index of the area pressure bump that occurred in the second mining level is between 2.47 and 2.71 (details presented in Table 4). The value 2.47 (2.50 for safety) was taken as the medium dangerous evaluation standard and the largest amount 2.71 (2.75 for safety) as the strong dangerous evaluation standard of critical stress index.

4. Evaluation result

4.1. Stress concentration and gravity stress of overlying rock

The burial depth of the open cut in 22220 working face was 928.34–955.7 m, and the gravity stress was 23.2085–23.8925 MPa. The burial depth of track transport was 955.7–983.95 m, and the gravity stress was 23.8925–24.5988 MPa. The burial depth of the air return roadway was 928.34–957.02 m, and the gravity stress was 23.2085–23.9255 MPa. The stress concentration areas mainly included following four kinds.

(1) The height difference of four faults was less than 5 m, taking the stress concentration factor as 1.2 and unilateral influence area as 100 m.
(2) Since the overlying rocks gradually fell down during the stoping of working face, the stress concentration factor of main roof weighting in the initial drivage and mining processes from 0 to 160m of the working face is set as 1.6.

(3) The open cut of working face broke the balance of stress. The area of the hanging roof of the overlying rocks above the open cut was large and stayed a long time, which easily caused crushing when the drivage started at the open cut, therefore taking the stress concentration factor as 1.7.

(4) The first square was at 200 m, and the second square was 350 m, taking the influence area as 50 m, which means that the stress concentration factors from 150 to 250 m and 300 to 400 m were 1.6.

After the calculation with stress concentration, the relations diagram between the distance of the track transport, the air return roadway from the open cut, and approximate stress are shown in Figures 4 and 5. The highest approximate stress area is the open cut, and the other high approximate stress areas are located in places in which track transport and air return roadway are within 250 m and between 300 m and 400 m from the open cut.

![Figure 4. Distribution diagram of track transport and approximate stress](image1)

![Figure 5. Distribution diagram of return air roadways and approximate stress](image2)
4.2. Evaluation result
Calculating the critical stress of each area in the working face and the critical stress index using Equations (4.8) and (4.9), the results of critical stress index are shown in Figures 6 and 7, and the area segmentation is shown in Figure 8. As the drivage of working face, the hazard of compound dynamic disaster is declining, there are three strong dangerous areas which are located at the open cut and 20–250 m from the open cut to track transport. Three medium dangerous areas are mainly located at 0–20 m from the open cut to track transport and 250–300 m, 400–530 m, and 710–990 m from open cut to air return roadway. Four weak dangerous areas are located at 550–805 m from the open cut to track transport and 250–300 m, 400–530 m, and 710–990 m from the open cut to air return roadway. The other areas are no hazard. The risk grade of track transport is slightly stronger than air return roadway due to the deeper burial depth.

![Figure 6. Critical stress index of track transport](image1)

![Figure 7. Critical stress index of air return roadway](image2)
4.3. Validation of evaluation results
Due to the prediction of the dangerous areas in the working face, the coal mine did dynamic disaster prevention and control, and they drilled borehole down to pre-extract the coal bed gas to prevent the pressure bump within the 700-m inner stoping area and which drilled in a row along both track and air return roadway (two rows in the area which are 20 m from structure belt in both directions). The gaps of drills were 4 m, and the radius of drills are 94 mm. The drilling depth of track and air return roadway was not less than 115 m and 95 m separately, and the intermediate cross also was not less than 10 m. They used network gas drainage immediately after the drilling was finished. In the outer 336-m area, since the original content of the gas was higher than $6\text{m}^3/\text{t}$ and the influence of faults, the coal mine drilled borehole down and cross the seam to pre-extract gas from the coal bed, faults, and area with abnormal coal thickness for the prevention. There was no pressure bump or coal and gas outbursts and any other dynamic disaster during whole stoping process.

From June 25 to August 10, 2020, the coal mine used the measuring and orientation technology with charge meter in the working face. It started at the track transport of 40 m from the working face, and they arranged a measure point 10 m each with a total of ten measure points and labeled from 1 to 10, which was used to monitor the charge in high stress concentration areas. They also arranged two measure points 10 m from each other in the stopping line and labeled as 11 and 12, which were used to comparatively analyze with the measure points 1 to 10 in the high stress concentration areas. For each measure point, the depth of drill was 2 m, and the drill radius was 42 mm. The measuring drill was 1.2 m from bottom, and monitoring time was between 5 and 10 min at each point. The monitoring results revealed that with the drivage of working face, the further from the working face, the less charge was monitored. There was an unusual data in the track transport measure points on June 28 when the working face was 560 m from open cut and near the F4 faults, which was a weak dangerous area. At the same time, the data in the air return roadway measure points was normal, which was not a dangerous area.

According to the analysis of the geological conditions in working face, the major influence factors are the buried depth of coal seam, bursting liability of coal, tectonic stress, characteristics of roof rocks, gas pressure, and so on. By using the comprehensive index method with the influence of geological factors above, the hazard index $W_{11} = 0.5$ accounts to which has weak coal burst danger. In the analysis of mining technical requirement, the major influence factors are the effects from the faults of working

Figure 8. Dangerous areas segmentation
face and the left bottom coal during the drivage. Under the influence factors of mining technology, the hazard index $W_2=0. 125$ accounts to no coal burst danger. As the mining goes deeper, pressure bump and coal and gas outbursts may occur, decreasing the threshold value. When the pressure bump or coal and gas outbursts occur, the risk grades will be higher, easily causing sudden and major disaster accidents. Compared with the comprehensive index method, the critical stress index method evaluates the hazard grades of compound dynamic disaster higher, and the results are more conserved. The critical stress index method can also predict the relationship between support strength, gas concentration after drainage, and composite dynamic disaster, and the evaluation results tend to be more quantitative and conservative.

5. Conclusion
By using the critical stress index method to evaluate the risk grades of compound dynamic disaster in 22220 working face, the following results are gained: (1) The minimum critical stress index of composite dynamic disaster which had happened in the same level of coal seam and mining area before is taken as the medium risk evaluation standard, and the maximum is taken as the strong risk evaluation standard. The critical stress index of each area of the working face is calculated, and the risk grade is divided. The area whose critical stress index is less than 2.5 is no hazard, 2.25~2.50 is weak hazard, 2.50~2.75 is medium dangerous, and above 2.75 is strongly dangerous. (2) According to the evaluation standard, the 22220 working face was divided into several dangerous-grade areas, which included three strong dangerous areas, three medium dangerous areas, four weak dangerous areas, and the remaining areas were no dangerous areas. (3) Compared with the comprehensive index method, the critical stress index method can predict the relationship between support strength, gas concentration after drainage, and composite dynamic disaster, and the evaluation results tend to be more quantitative and conservative.

6. References
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