J. Cent. South Univ. (2012) 19: 1086−1091
DOI: 10.1007/s11771-012-1113-3

Springer

Rock burst induced by roof breakage and its prevention

HE Jiang(何江), DOU Lin-ming(窦林名), CAO An-ye(曹安业), GONG Si-yuan(巩思园), LÜ Jian-wei(吕建为)

1. State Key Laboratory of Coal Resources and Safe Mining, China University of Mining and Technology, Xuzhou 221008, China;
2. School of Mines, China University of Mining and Technology, Xuzhou 221116, China

Abstract: Based on the research on rock burst phenomenon induced by the breakage of thick and hard roof around roadways and working faces in coal mines, a criterion of rock burst induced by roof breakage (RBRB) was proposed and the model was built. Through the model. a method calculating the varied stresses induced by roof breakage in support objects and coal body was proposed and a unified formula was derived for the calculation of stress increment on support objects and coal body under different breaking forms of roof. Whilst the formula for calculating dynamic load was derived by introducing dynamic index K_d. The formula was verified in Huating Mine by stress measurement. According to the formula for stress increment calculating, the sensitivities of dynamic load parameters were further studied. The results show that the thickness and breaking depth of roof, width of support object are the sensitive factors. Based on the discussion of the model, six associated effective methods for rock burst prevention are obtained.

Key words: roof breakage; dynamic load; stress increment; rock burst; mining engineering

1 Introduction

Rock burst is one of the typical dynamic hazards in coal mining, which extremely affects safety and production of mines [1]. Since the first recorded rock burst happened in England in 1738, the main mining countries like Germany, South Africa, Polish, Czech, Canada, Japan, France, and about more than 20 countries reported rock burst happening. Many researchers have studied rock burst from various aspects around the world [2−9]. In China, the first recorded rock burst happened in Fushun in 1933. As the mining depth increases (about 12−20 m per year), the tectonic stress of coal-rock mass around mining space gets larger and larger, which makes it easier to reach the peak strength of coal-rock mass. Under this condition, it will probably induce rock burst while the stress increases promptly.

Roof breakage can shock and vibrate the coal body, which is one of the main factors to increase stress promptly in coal-rock mass. Most rock bursts are mainly affected by thick and hard roof. Presently, many researchers have been researching rock burst induced by roof breakage (RBRB), for instance, BLAKE et al [10−11] studied the seismic phenomenon of roof breakage, and some Chinese researchers studied the component rock burst tendency of combined floor, coal seam and roof [12−13], and the energy cumulating and releasing effect of strata around working area was studied by some others [14−19]. The research works above made great progresses in learning mechanism of roof induced rock burst. However, energy is the necessary but inadequacy factor of rock burst, while the stress increasing in coal and rock mass directly causes the rock burst. Hence, the study of stress vibration of coal-rock mass can help to learn the essence of RBRB and has instructiveness in the control of rock bursts.

2 Mechanism of rock burst

In the mining progress, roof breakage will induce dynamic load of coal mass. If the sum of dynamic and primary load in coal mass is up to the peak strength, the rock burst will happen. The stress criterion of rock burst can be expressed as

$$\sigma_p + \sigma_d > [\sigma]_b \tag{1}$$

where σ_p is the primary load of coal mass; σ_d is the dynamic load induced by roof breakage; $[\sigma]_b$ is the peak strength of coal mass. Equation (1) represents that the interaction of σ_p and σ_d can lead to rock burst. Parameter σ_p is determined by geological and mining technique

Foundation item: Project(2010CB226805) supported by the National Basic Research Program of China; Projects(30370412, 30670558) supported by the National Natural Science Foundation of China; Project(SKLCRSM10X05) supported by the Self-research Program of the Key Laboratory of Coal Resources and Safe Mining, China
Received date: 2011−01−21; **Accepted date:** 2011−04−08
Corresponding author: HE Jiang, PhD candidate; Tel: +86−13852147725; E-mail: hejiang76@163.com

conditions of mine and usually uncontrollable, while σ_d is the factor that leads to the stress increasing suddenly up to the peak strength. Hence, it is significant to study σ_d induced by roof breakage.

3 Analysis of process

3.1 Establishment of model

Under the disturbance stress, roof strata will break. The fracture line in the coal body is always parallel to the working face or road way. The fracture width along with the working face or road way is larger than the fracture depth in coal body, and the stress along the width aspect is under the same condition. So, it is possible to analyze the stress condition of roof by plane strain with the beam theory. The model of rock burst induced by roof breakage is shown in Fig. 1.

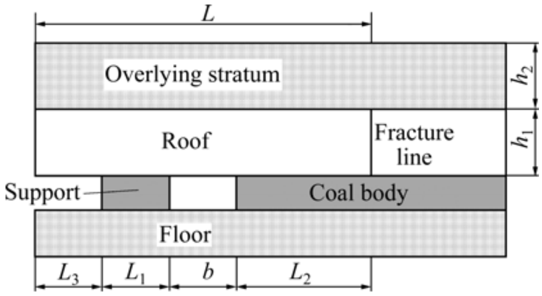

Fig. 1 Model of rock burst induced by roof breakage

In Fig. 1, L_1 is the width of support object; L_2 is the depth of roof breaking in coal body; L_3 is the length of roof hanging in goaf; b is the width of working space unsupported; h_1 is the thickness of roof; h_2 is the thickness of overlying strat, nm; ρ_1 is the density of roof; p is the stress loaded by overlying strata; $f(L_1)$ is the stress distribution function of support; $f(L_2)$ is the stress distribution function of coal body between the working space and fracture line.

Figure 2 shows the stress analysis of roof before breaking, where σ_T is the tensile stress of upper edge; τ is the shear stress of fracture face. Because the stress increment is mainly caused by the change of stress state on the fracture line, the stress on the left side of the model has been neglected for simplification. The mechanical equilibrium of roof before breaking is presented as

$$\begin{cases} \int_0^{h_1} (\sigma_T - 2\sigma_T \dfrac{h}{h_1})\mathrm{d}h = 0 \\ \int_0^{L_1} f(L_1)\mathrm{d}l_1 + \int_0^{L_2} f(L_2)\mathrm{d}l_2 + \tau h_1 - \rho_1 g h_1 L - pL = 0 \\ \int_0^{L_1} f(L_1)\cdot(L_1 + b + L_2 - l_1)\mathrm{d}l_1 + \int_0^{L_2} f(L_2)(L_2 - l_2)\mathrm{d}l_2 + \\ \quad 2\int_{h_1/2}^{h_1} (h - \dfrac{h_1}{2})^2 \dfrac{2\sigma_T}{h_1}\mathrm{d}h - \rho_1 g h_1 \dfrac{L^2}{2} - p\dfrac{L^2}{2} = 0 \end{cases} \tag{2}$$

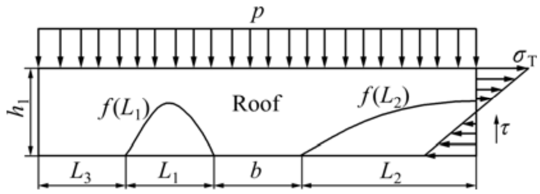

Fig. 2 Stress analysis of roof

Because of the change of stress state on fracture line after roof breaking, there will be stress increment in support and coal body so as to get new mechanical equilibrium. As the distribution of stress increment is determined by characteristics of coal and the structure of surrounding coal and rock mass, and the distribution of stress increment varies in different conditions, it is difficult to get exact distribution of stress increment. In order to evaluate the stress increment, dynamic load, and their influencing level on working space, the average stress increment $\Delta\sigma$ is used to describe the comprehensive mechanism effect of roof breakage. The new mechanical equilibrium of broken block after breaking can be presented as

$$\begin{cases} \int_0^{L_1} [f(L_1) + \Delta\sigma]\mathrm{d}l_1 + \int_0^{L_2} [f(L_2) + \Delta\sigma]\mathrm{d}l_2 + \\ \quad \rho_1 g L - pL = 0 \\ \int_0^{L_1} [f(L_1) + \Delta\sigma]\cdot(L_1 + b + L_2 - l_1)\mathrm{d}l_1 + \\ \quad \int_0^{L_2} [f(L_2) + \Delta\sigma](L_2 - l_2)\mathrm{d}l_2 - \rho_1 g h_1 \dfrac{L^2}{2} - p\dfrac{L^2}{2} = 0 \end{cases} \tag{3}$$

From Eqs. (2) and (3), the following equations can be obtained as

$$\Delta\sigma = \frac{h_1^2}{3(L_1^2 + L_2^2) + 6(bL_1 + L_1 L_2)}\sigma_T \tag{4}$$

or

$$\Delta\sigma = \frac{h_1}{L_1 + L_2}\tau \tag{5}$$

The relationship between the tensile stress σ_T and the shear stress τ can be expressed as

$$\tau = \frac{h_1(L_1 + L_2)}{[3(L_1^2 + L_2^2) + 6(bL_1 + L_1 L_2)]}\sigma_T \tag{6}$$

3.2 Discussion of model

From Eqs. (4) and (5), the calculation of $\Delta\sigma$ is determined by the roof breaking form.

As Eq. (6) shows, the form of roof breakage can be determined by the parameters: $[\sigma_T]$, $[\tau]$, L_1, L_2, b and h_1, where $[\sigma_T]$ is the tensile strength; $[\tau]$ is the shear strength of roof. The parameters in Eq. (6) can be obtained by practical measurement for the working space. Taking $[\sigma_T]$

instead of σ_T in Eq. (6), a shear stress τ can be obtained. If $\tau<[\tau]$, when the tensile stress σ_T is up to the tensile strength $[\sigma_T]$ and the roof begins to break by tension, the shear stress on the fracture line is lower than $[\tau]$. The breaking form of roof is tensile fracturing. Conversely, if $\tau>[\tau]$, the breaking form of roof should be shear fracturing.

If the breaking form is tensile fracturing, the stress increment can be expressed as Eq. (7), while the stress increment can be written as Eq. (8) under condition of shear failure:

$$\Delta\sigma = \frac{h_1^2}{3(L_1^2+L_2^2)+6(bL_1+L_1L_2)}[\sigma_T] \tag{7}$$

$$\Delta\sigma = \frac{h_1}{L_1+L_2}[\tau] \tag{8}$$

According to the key strata theory, the movement of strata above coordinates to the key stratum after the key stratum breaks. The breaking of key stratum will lead all or partial strata above it to move together with itself [20]. The key stratum controls the movement of the strata above it. Because of the different breaking forms of roof including the strata above, the expressions for the stress increment of roof strata are not unique. But, the expression of stress increment of roof can be written by introducing parameters of C_T and C_τ as

$$\Delta\sigma = \frac{\sum C_{Ti}h_i^2[\sigma_T]_i}{3(L_1^2+L_2^2)+6(bL_1+L_1L_2)} + \frac{\sum C_{\tau i}h_i[\tau]_i}{L_1+L_2} \tag{9}$$

where h_i is the height of stratum i; $[\sigma_T]_i$ is the tensile strength of stratum i; $[\tau]_i$ is the shear strength of stratum i; C_{Ti} and $C_{\tau i}$ are two introduced parameters of stratum i. When the breaking form of stratum i is tensile failure, $C_{Ti}=1$ and $C_{\tau i}=0$, while in condition of shear fracturing $C_{Ti}=0$ and $C_{\tau i}=1$.

The stress measurement in mines reveals that the process of roof fracturing is abrupt and causes propagation of seismic wave validated by seismic observation. When the roof strata break, the stress increment load on the support object and coal body is equivalent to the isometric stress of balance force of fracture line before roof breaking. The stress load on the elastic coal body will cause dynamic load. If the deformation is x and the coefficient of elasticity of coal is K, according to the law of conservation of energy, the equation can be written as

$$\Delta\sigma x = \frac{1}{2}Kx^2 \tag{10}$$

So, the maximum and dynamic load σ_d can be written as

$$\sigma_d = Kx = 2\Delta\sigma \tag{11}$$

According to Eq. (11), $K_d=2$. Because of the different fracturing speeds and non complete elasticity of coal body, K_d can change between 1 and 2. So, the expression of average dynamic load can be written as

$$\sigma_d = \frac{K_d\sum C_{Ti}h_i^2[\sigma_T]_i}{3(L_1^2+L_2^2)+6(bL_1+L_1L_2)} + \frac{K_d\sum C_{\tau i}h_i[\tau]_i}{L_1+L_2} \tag{12}$$

The derivation process of the model presents a method to calculate the stress increment and dynamic load after roof strata break. If the stress distribution of support object and coal body can be confirmed, the stress increment and dynamic load can be exactly derived according to the derivation process of the model. Equation (12) can be used to evaluate the influencing level of roof breakage to support object and coal body. If the safety factor (n) is introduced in Eq. (1), the rock burst hazard criterion can be written as

$$\sigma_j + n\sigma_d > [\sigma]_b \tag{13}$$

3.3 Verification in practice

The mining depth of 2501 mining areas of Huating Mine in Gansu Province of China is 720 m underground. According to the borehole YB605, the main roof is fine sandstone of 19 m thick and 13 m above it are two separate strata of siltstone with the thickness of 10 m and 14 m. As shown in Fig. 3, the borehole stress meter shows that the stress 12 m besides the roadway increases abruptly when the distances are 6.8 m and 4.8 m to the working face respectively. It can be judged that the 10 m and 14 m thick siltstones break simultaneously after the 19 m thick sandstone fractures. According to the measurements, the tensile strength $[\sigma_T]$ of sandstone is 1.91 MPa and that of siltstone is 1.67 MPa, while the shear strength $[\tau]$ of both is 4.6 MPa. The canopy of hydraulic support is 4 m and unsupported working space is 0.2 m. By substituting the related data into Eq. (5), the shear stress of roof strata can be obtained. The shear stresses of strata from bottom to up are 1.10, 0.62 and 0.87 MPa, respectively. They are all lower than the shear strength, so the breaking forms of roof strata are tensile failures. Making $K_d=1$ and 2, the stress increment and dynamic load can be obtained, as listed in Table 1.

Table 1 Calculated results of stress variation

Breaking stratum	Stress increment/MPa	Dynamic load/MPa
19 m sandstone	1.94	3.89
14 m and 10 m siltstones	2.08	4.17

The YBS-40 borehole stress meter and KBJ-16 data acquisition instruments were applied to measuring the stress in coal body. Figure 3 shows the arrangement of instrument and Fig. 4 represents the stress measurement results. Figure 4 shows the stress of 12 m (elastic area with maximum varied stress) beside the roadway in coal body increases abruptly when the distances are 6.8 m and 4.8 m to the working face. The sudden increment is nearly twice the residual increment. Hence, the sudden increment can be recognized as dynamic stress. The stress values of variation observed are shown in Table 2.

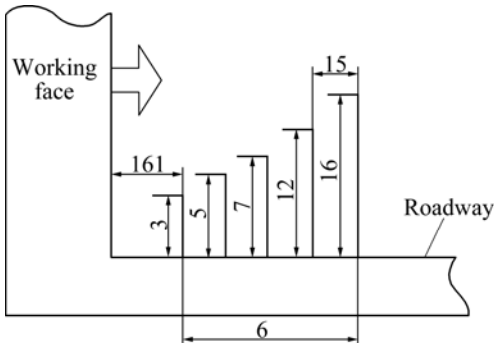

Fig. 3 Borehole stress meters collocation (unit: m)

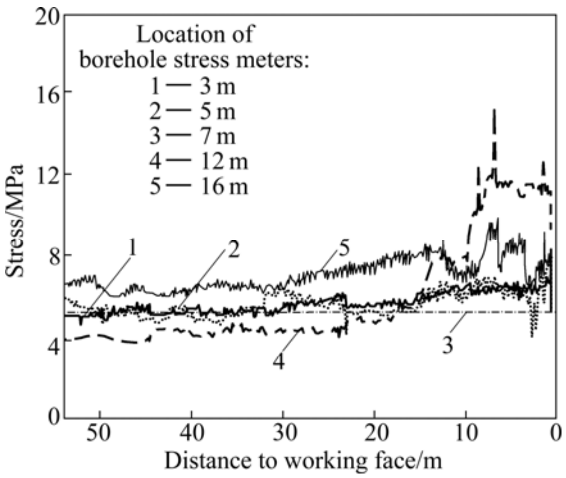

Fig. 4 Measurement results of borehole stress meters

Table 2 Measured results of stress variation

Breaking stratum	Stress increment/MPa	Dynamic stress/MPa
19 m sandstone	2.20	4.37
14 m and 10 m siltstone	2.08	4.97

According to Tables 1 and 2, the calculated stresses are slightly lower than the measured stresses. The calculated stresses are 90.3% of the measured ones on average. From Table 2, the practical dynamic index of roof strata can be calculated and the results are 1.99 and 2.39 which are approximately equal to the theoretical dynamic index, thus the calculated results of the model are reasonable by considering the measurement error.

Consequently, the dynamic load and stress increment can be calculated comparatively accurately by the method proposed by the model.

4 Application in rock burst prevention

The model of rock burst induced by roof breakage can be applied to calculating the stress increment and dynamic load on the coal body under the fractured roof block and the support on the working face when the roof breaks in the coal body, and to calculating the stress increment and dynamic load on the coal body and coal pillar under fractured roof block as the roof breaks when roadways heading beside goaf. Additionally, it can be applied to calculating the abnormal stress when roadway or working face heads towards structures like fault as well.

The deformation and losing stability theory of rock burst pointed out that the stress of coal-rock mass will be soon up to the peak strength when rock burst happens. Under the condition, the anti-deformation ability of the coal-rock mass has declined and became unstable. If the stress vibrates, the rock burst will happen. When the roof breaks, the coal seam is loaded by the front abutment or side abutment, the bearing capacity has declined to a low level, and the breakage coefficient D has been up to a high level, so the rock burst is easy to happen. The possibility of rock burst happening depends on the stress state of coal-rock mass and the intensities of external stress disturbance. As the calculating results show, the dynamic load is up to 3.89 MPa when the thickness of roof is 19 m. The stress increment and dynamic load are of the second proportional relationship with the thickness of roof as Eq. (12) presents, so the breaking of the hard and thick roof directly causes rock burst. When the static stress of coal mass and the peak strength are measured and the dynamic load of roof breaking is calculated as well, the hazardous of rock burst can be calculated by Eq. (13). If the calculated stress of coal mass is larger than the peak strength $[\sigma]_b$, the suitable methods for rock burst controlling should be taken to release the rock burst hazards.

According to Eq. (13), take the 19 m sandstone in thickness of Huating Mine for example, make only one parameter changeable, and the variation law of dynamic load with other parameters can be obtained. The variation laws are shown in Fig. 5.

Figure 5(a) shows while the roof thickness is lower than 98.3 m and the breaking form of roof is tensile fracturing, the dynamic load is in the second proportional relationship with the thickness of roof. While the roof thickness is larger than 98.3 m, the relationship between dynamic load and roof thickness changes into linear relation. The dynamic load rises quickly while the thickness of roof is larger than 5 m. Figures 5(b) and (c) show that the dynamic load increases rapidly while the depth of roof breaking in coal body or the width of

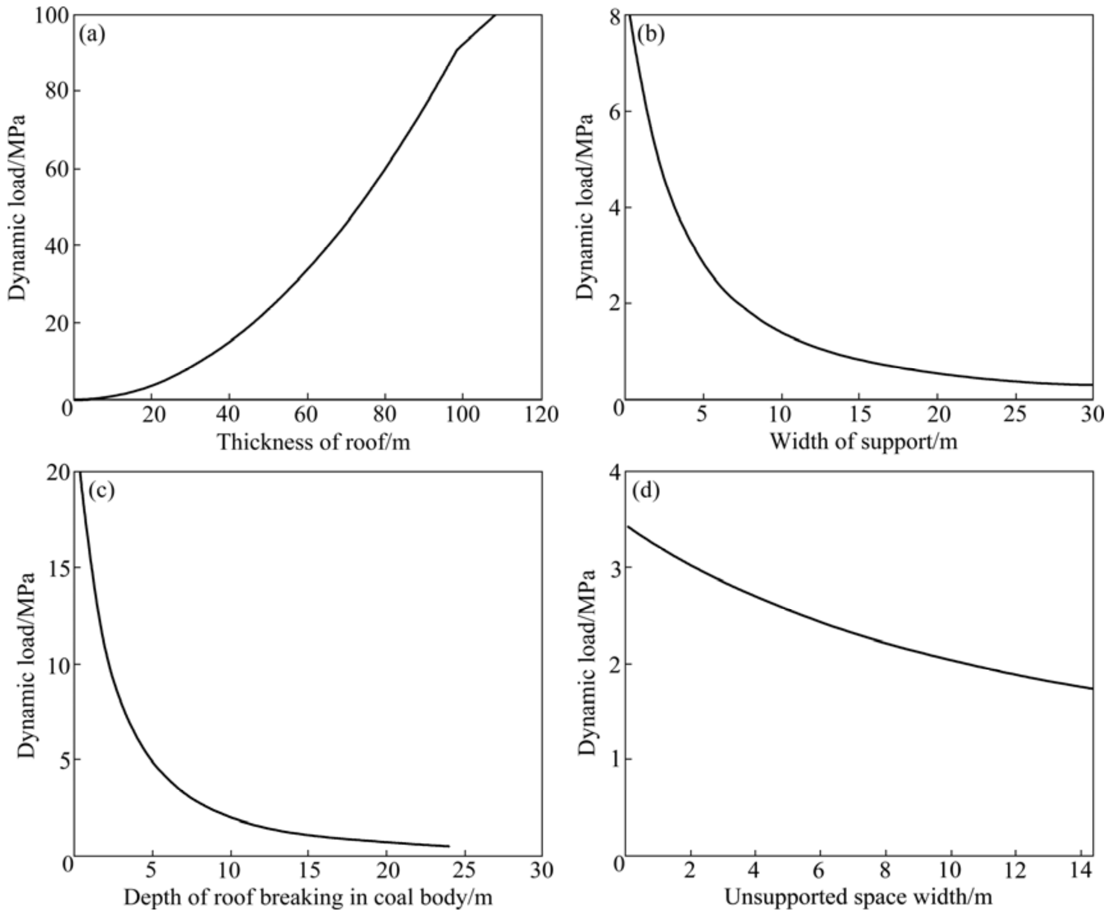

Fig. 5 Laws of dynamic load with parameters: (a) Relations between dynamic load and thickness of main roof; (b) Relations between dynamic load and width of support object; (c) Relations between dynamic load and broken depth of main roof; (d) Relations between dynamic load and width of excavation

support gets larger than 5 m. Figure 5(d) shows the dynamic load decreases slightly while the width of unsupported area of working space increases. Meanwhile, the unsupported area cannot enlarge too much in coal mining, so it’s not reasonable to decrease the dynamic load by increasing the width of unsupported area.

The results show that the thickness and breaking depth of the roof stratum, and the width of support object are the sensitive factors determining the dynamic load.

Based on the results, six rock burst controlling methods can be obtained:

1) Enhancing the strength of support. In order to make the roof stratum break behind the support object, the dynamic load can be reduced and the stress increment can be minus.

2) Enlarging the width of support. Make it not lower than 5 m for better, so as to reduce the dynamic load and stress increment.

3) Breaking the coal body so as to make the deformation of roof and coal compatible and the breaking depth of the roof stratum get larger. As a result, the dynamic load can be reduced and the possibility of rock burst can get much lower.

4) Presplitting the roof, so the stress in coal body will increase gradually and the dynamic load can be avoided. The position of presplitting should be more than 10 m ahead of mining face.

5) Weakening the intensity of roof, so the dynamic stress and stress increment can be reduced.

6) Breaking roofs in goaf so as to prevent roofs breaking in coal body to prevent rock burst caused by dynamic load.

5 Conclusions

1) The analysis in this work promotes a method for calculating the stress variation caused by roof breakage, deduces a function for calculating the average dynamic load and stress increment, and creates a criterion of rock burst induced by roof breaking.

2) The function for judging fracture form of the roof shows that the tensile fracture form of the roof is more than the shear failure, for the thickness of roof is much higher with shear failure than that of tensile fracture.

3) The thickness and breaking depth of the roof stratum, and the width of support object are the sensitive

factors of dynamic load.

4) When the stress state is high or there is rock burst hazard shown by the model calculating, suitable controlling methods promoted should be taken to prevent rock burst.

References

[1] DOU Lin-ming, HE Xue-qiu. Theory and technology of rock burst prevention [M]. Xuzhou: China University of Mining and Technology Press, 2001: 1−270. (in Chinese)

[2] ORTLEPP W D, STACEY T R. Rock burst mechanisms in tunnels and shafts [J]. Tunneling and Underground Space Technology, 1994, 59(6): 59−65.

[3] KIRZHNER F, ROSENHOUSE G. Numerical analysis of tunnel dynamic response to earth motions [J]. Tunneling and Underground Space Technology, 2000, 15(3): 249−258.

[4] USZKO M. Monitoring of methane and rockburst hazards as a condition of safe coal exploitation in the mines of Kompania Węglowa SA [J]. Procedia Earth and Planetary Science 2009, 1(1): 54−59.

[5] MILEV A M, SPOTTISWOODE S M, RORKE A J, FINNIE G J. Seismic monitoring of a simulated rockburst on a wall of an underground tunnel [J]. Journal of South African Institute of Mining and Metallurgy, 2001, 8: 253−260.

[6] PATYNSKA R, KABIESZ J. Scale of seismic and rock burst hazard in the Silesian companies in Poland [J]. Mining Science and Technology, 2009, 19(5): 604−608.

[7] KABIESZ J, MAKÓWKA J. Selected elements of rock burst state assessment in case studies from the Silesian hard coal mines [J]. Mining Science and Technology, 2009, 19(5): 660−667.

[8] DOU Lin-ming, LU Cai-ping, MU Zong-long, GAO Ming-shi. Prevention and forecasting of rock burst hazards in coal mines [J]. Mining Science and Technology, 2009, 19(5): 585−591.

[9] HOLUB K, PETRS V. Some parameters of rockbursts derived from underground seismological measurements [J]. Tectonophysics, 2008, 456(1): 67−73.

[10] BLAKE W, LEIGHTON F, DUVALL W I. Micro seismic techniques for monitoring the behavior of rock structures [R]. Washington, USA; Burea of Mines, 1974.

[11] BRADY B, LEIGHTON F. Seismcity anomaly prior to a moderate rock burst: A case study [J]. International Journal of Rock Mechanics and Mining Sciences and Geomechanics Abstracts, 1977, 14(3): 127−132.

[12] LU Cai-ping, DOU Lin-ming, WU Xing-rong. Eexperimental research on rules of rock burst tendency evolution and acoustic-electromagnetic effects of compound coal-rock samples [J]. Chinese Journal of Rock Mechanics and Engineering, 2007, 26(12): 2549−2555. (in Chinese)

[13] LI Ji-qing, QI Qing-xin, MAO De-bing. Discussion on evaluation method of bursting liability with composite model of coal and rock [J]. Chinese Journal of Rock Mechanics and Engineering, 2005, 24(S1): 4805−4810. (in Chinese)

[14] ZUO Yu-jun, LI Xi-bing, ZHOU Zi-long. Determination of ejection velocity of rock fragments during rock burst in consideration of damage [J]. Journal of Central South University of Technology, 2005, 12(5): 618−622.

[15] MU Zong-long, DOU Lin-ming, ZHANG Guang-wen. Study of prevention methods of rock burst disaster caused by hard rock roof [J]. Journal of China University of Mining & Technology, 2006, 35(6): 737−741. (in Chinese)

[16] LU Cai-ping, DOU Lin-ming, WU Xing-rong. Controlled weakening mechanism of dynamic catastrophe of coal and rock and its practice [J]. Journal of China University of Mining & Technology, 2006, 35(5): 301−305. (in Chinese)

[17] WANG Wen-xing, PAN Chang-liang, FENG Tao. Fountain rockburst and inductive rockburst [J]. Journal of Central South University of Technology, 2000, 7(3): 129−132.

[18] XIE He-ping, PENG Rui-dong, JU Yang. Study on energy analysis of rock failure [J]. Chinese Journal of Rock Mechanics and Engineering, 2005, 24(15): 2603−2608. (in Chinese)

[19] LI Xin-yuan, MA Nian-jie, ZHONG Ya-ping. Storage and release regular of elastic energy distribution in tight roof fracturing [J]. Chinese Journal of Rock Mechanics and Engineering, 2007, 26(S1): 2786−2793. (in Chinese)

[20] QIAN Ming-gao, MIAO Xie-xing, XU Jia-ling. Theory of key stratum in ground control [M]. Xuzhou: China University of Mining and Technology Press, 2003: 1−234. (in Chinese)

(Edited by DENG Lü-xiang)