DE GRUYTER
OPEN

Arch. Min. Sci., Vol. 59 (2014), No 3, p. 591–608

Electronic version (in color) of this paper is available: http://mining.archives.pl

DOI 10.2478/amsc-2014-0042

ANTONI TAJDUŚ*, JERZY CIEŚLIK*, KRZYSZTOF TAJDUŚ**

ROCKBURST HAZARD ASSESSMENT IN BEDDED ROCK MASS: LABORATORY TESTS OF ROCK SAMPLES AND NUMERICAL CALCULATIONS

ANALIZA SKŁONNOŚCI GÓROTWORU UWARSTWIONEGO DO TĄPAŃ NA PODSTAWIE WYNIKÓW BADAŃ LABORATORYJNYCH PRÓBEK SKALNYCH ORAZ OBLICZEŃ NUMERYCZNYCH

The paper presents the results of analyses on tendency of bedded rock mass towards rockburst, carried out on the basis of laboratory tests of rock samples and determined indices, as well as by means of numerical calculations. The first part of the paper discusses selected parameters evaluating the tendency of rocks towards burst, on the basis of which the analysis was carried out in order to assess rockburst hazard in selected complexes of strata in Upper Silesian Coal Basin. Rockburst hazard in bedded rock mass is also analyzed with the use of numerical methods and the results of calculations of rock damage development in the exploitation front during coal deposit extraction are presented. Burst hazard of rock mass is studied through the changes of intensiveness of damage process in the area of exploitation front.

Keywords: rockburst, rockburst hazard indices, numerical methods

W artykule zaprezentowano wyniki analiz skłonności górotworu uwarstwionego do tąpań przeprowadzone na podstawie badań laboratoryjnych próbek skalnych i wyznaczonych na tej podstawie wskaźników oraz obliczeń numerycznych. W pierwszej części artykułu zaprezentowano wybrane wskaźniki służące do oceny skłonności skał do tąpań. Na podstawie wybranych wskaźników przeprowadzono analizę skłonności do tąpań wybranych kompleksów warstw skalnych GZW. Skłonność górotworu uwarstwionego do tąpań, analizowano również wykorzystując do tego celu obliczenia numeryczne. Zaprezentowano wyniki obliczeń rozwoju zniszczenia skał w przodku eksploatacyjnym podczas eksploatacji pokładu węgla. Skłonność górotworu do tąpań była analizowana poprzez zmiany intensywności procesu zniszczenia w otoczeniu przodka eksploatacyjnego.

Słowa kluczowe: tąpania, wskaźniki skłonności do tąpań, obliczenia numeryczne

* AGH UNIVERSITY OF SCIENCE AND TECHNOLOGY, FACULTY OF MINING AND GEOENGINEERING, DEPARTMENT OF GEOMECHANICS, CIVIL ENGINEERING AND GEOTECHNICS, AL. A. MICKIEWICZA 30, 30-059 KRAKOW, POLAND

** AGH UNIVERSITY OF SCIENCE AND TECHNOLOGY, FACULTY OF DRILLING, OIL AND GAS, AL. A. MICKIEWICZA 30, 30-059 KRAKOW, POLAND

1. Introduction

Rockburst tendency of rock mass is the ability to accumulate energy in strata and its sudden release in the moment of change or damage of rock mass structure. Not only the tendency of coal or other mineral towards burst is decisive in this case but also the properties of surrounding rocks and the state of stress in rock mass.

The most frequently analyzed geomechanical properties of strata affecting the hazard with rockburst include:
- elastic modulus (Young's modulus),
- Poisson's ratio,
- strength to uniaxial compression,
- strength to uniaxial tension,
- post-damage modulus,
- residual strength.

The first four parameters mentioned above can be obtained by means of simple studies carried out with the use of traditional, the so-called soft strength machines, whereas obtaining the two other parameters calls for the tests using the so-called stiff machines. A series of indices describing the rockburst hazard of particular types of rock or rock complexes can be defined on the basis of these parameters determined for particular rock layers.

The first part of the paper constitutes a brief review of indices for the evaluation of rockburst hazard, which are applied for bedded rock mass, as well as their analysis for rock complexes occurring in several mines. In the second part of the paper the rockburst hazard of bedded rock mass was determined for the earlier-analyzed rock complexes with the use of numerical calculations. Also the calculations of development of strata damage in the exploitation front of the coal deposit were carried out. Burst tendency was treated in the analysis as the change of damage process intensiveness in the area of the exploitation front. Finite element method (FEM) and elastic-plastic theory were applied for the sake of calculations. The influence of elastic properties of roof strata, as well as elastic and plastic properties of the seam, on the intensiveness of damage process of the face was modeled with the assumption that coal was treated as an elastic-plastic medium with softening. The primary aim of the present paper is to compare the results of analyses on rockburst hazard of bedded rock mass, determined on the basis of laboratory tests of rock samples, with the results obtained by means of computer simulations.

2. Characteristics of selected methods of evaluating rockburst hazard for bedded rock mass

2.1. Elastic potential energy index of rock P_{ES}

According to J. Smółka and I. Bałaza (Smółka et al., 1978), the value of elastic strain potential energy, which can be accumulated in the unit of rock volume, constitutes a measure of burst hazard of rocks. The maximum level of accumulated energy occurs when the stress reaches the value of immediate compression strength of material:

$$\phi_{\max} = \frac{R_c^2}{2E_s} V \tag{1}$$

The authors of the method proposed that the rockburst hazard index of rocks (P_{ES} – elastic potential energy index) is the value of isolated elastic strain energy expressed with the following formula:

$$P_{ES} = \frac{1000 R_c^2}{2E_s} = 500 \frac{R_c^2}{E_s} \tag{2}$$

where:

E_s — elastic modulus in MPa,
R_c — compressive strength in MPa,
P_{ES} — elastic potential energy index in kJ.

It is assumed that the properties of particular roof strata with the thickness equal 10-times the thickness of exploited seam, however not less than 30 m, decide about the burst hazard of the roof. In such a case, the index P_{ES} is conveyed as a weighed mean of roof strata complex. Since rockburst hazard of the roof not only depends upon the value of elastic potential energy index of roof strata complex but also on its structure, the index P_{ES}, calculated as the weighed mean, is multiplied by a properly selected coefficient, whose value ranges from 1.3 to even 0.5, depending on the roof structure. As the hitherto executed experiments show, the occurrence of thick, homogeneous and strong layers in the roof of the exploited seam increases rockburst hazard, whereas heterogeneous roof structure decreases such hazard.

The major weakness of this index is not assuming the complete characteristics of stress – strain of rock material as, from the perspective of rockburst, the most important is the post-damage behavior of rock after exceeding its strength limit.

2.2. Post-damage modulus as a measure of rockburst hazard

A complete characteristics of stress – strain coveys numerous data about the behavior of material after exceeding compressive strength limit (Fig. 1). The parameters of full characteristics are also used for constructing burst hazard indices of rocks. The simplest such index

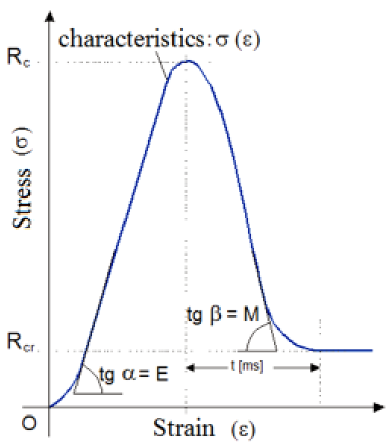


Fig. 1. Complete characteristics of normal stress – axial strain and corresponding parameters used in evaluation of burst hazard of rocks

is post-damage modulus (M), which allows for the isolation of two ways of rock behavior in post-damage areas: static course of damage – class I, and non-static course of damage – class II (Wawersik & Fairhurst, 1970).

The relation of post-damage modulus (M) and elastic modulus (E) is a modification of the index:

$$K_{pz} = \frac{M}{E} \tag{3}$$

where:

E — elastic modulus,
M — post-damage modulus (brittle-plastic modulus).

Both the indices used directly fail to convey sufficient information on burst tendency of rocks and describe it in isolation from the properties of surrounding rocks, hence they are rarely used separately as a measure of rockburst hazard. These indices are presented here due to the later application of particular parameters in the works of various authors.

2.3. Index proposed by Bukowska (2002)

In her works, Bukowska (2002) argues that the occurrence of dynamic damage requires the occurrence of a certain relation between the elastic modulus of surrounding rocks E_{sk} and the post-damage modulus of coal deposit M_p. She studies two cases of relationship between the elastic modulus of surrounding rocks and the post-damage modulus of the seam (the pillar).

Case 1: when the elastic modulus of surrounding rocks is significantly higher than the post-damage modulus of coal subject to the damage process ($E_{sk} > M_p$) but the seam has low or medium strength. Such conditions lead to static damage of the seam.

Case 2: when the elastic modulus of the roof E_{sk} is lower than the modulus M_p of coal ($E_{sk} < M_p$), then after exceeding the strength limit of coal, the dynamic effect occurs in the form of releasing energy accumulated in the roof. In such a case there is a dynamic model of the seam damage.

Following a wide-range analysis of various systems of 'roof – coal seam' and comparison of the results obtained with the conditions of rockburst occurring in coal mines in the years 1970-1999, the author proposed a new classification of rockburst hazard in the system 'surrounding rocks – coal seam', which is presented in Table 1.

TABLE 1

Assessment of rockburst hazard of rock mass (Bukowska, 2002)

Value of coefficient $T_B = \frac{M_p}{E_{sk}}$	**Burst hazard of the system 'surrounding rocks – coal seam'**
$\frac{M_p}{E_{sk}} < 1$	System not prone to rockburst

TABLE 1. Continued

$1 \le \frac{M_p}{E_{sk}} < 2$	System prone to rockburst
$\frac{M_p}{E_{sk}} \ge 2$	System not prone to rockburst

2.4. W_{ET} index for strata layout

Until recently, in Poland it was assumed that the so-called burst energy index was one of the most crucial indices allowing for the assessment of rockburst hazard (Szecówka & Ożana, 1973). The value of W_{ET} is determined from the graph of stress – strain, obtained for the pre-damage sample as a relation of the elastic strain potential energy to the energy dispersed as a result of irreversible deformations. Depending on the value of the index W_{ET}, the rocks are divided into: not prone to burst $W_{ET} < 2$, slightly prone to burst $2 \le W_{ET} < 5$, strongly prone to burst $5 \le W_{ET}$. Since the time, when the researchers and engineers began to assess the rockburst hazard on the basis of the complete characteristics of stress – strain, the importance of the index has seriously diminished. The above considerations referred to a homogeneous rock mass. However, in the case of bedded rock mass, the common, though controversial, practice in the analysis embraces either assuming extreme values of indices or determining average W_{ET} index as a weighed mean of values obtained for particular rock layers creating the bedded rock mass. The average W_{ET} index is calculated according to the following formula:

$$W_{ET} = \frac{\sum_{i=1}^{n} W_{ET_i} \cdot m_i}{\sum_{i=1}^{n} m_i} \tag{4}$$

where:

n — number of layers,
W_{ET} — energy index of i^{th} layer,
m_i — thickness of i^{th} layer.

Such an approach seems improper, which can be proven on a simple example. Let us assume that the seam consists of two layers of a similar thickness. The rocks have the same strength limit for uniaxial compression and elastic modulus. Fig. 2 presents the stress – strength characteristics of rock samples for the two layers (for clarity, the curves of characteristics were substituted with the straight sections).

The W_{ET} index of rockburst hazard determined for layer I (Fig. 2a) has the value of 4.0, whereas the W_{ET} index for layer II (Fig. 2b) is 0.25. The average rockburst hazard index $W_{ET(śr)}$ assumes the value of 2.125. If we determine the value of substitute W_{ET} index (according to the definition of this index), i.e. deformation characteristics of the sample consisting of two layers with the thicknesses corresponding to the thicknesses of particular layers creating the seam (Fig. 2c), its value is 1. Hence, we are having three values of W_{ET} index for bedded rock mass: maximum value = 4, average value = 2.125, and substitute value = 1. Which one is correct? In our opinion: the third one.

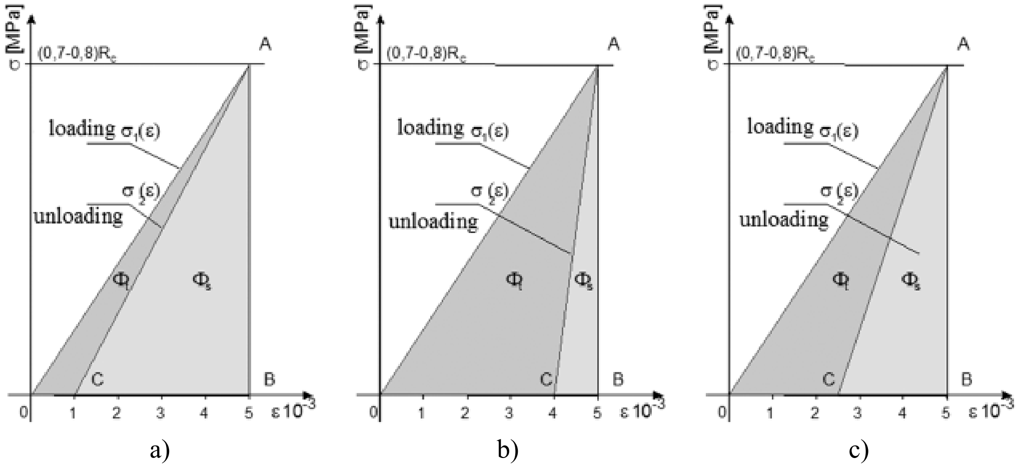


Fig. 2. Characteristic energies determined for defining W_{ET} index

2.5. Post-damage energy index of burst hazard for rocks W_{ET}^{pz}

Fig. 3 presents the proposal of a new index W_{ET}^{pz} (Tajduś et al. 2003), which seems to describe the measure of rockburst hazard better in comparison to a classic W_{ET}, as it is determined from the relation of post-damage work of elastic strain to pre-damage work (a similar proposal can be also found in the works of Pietuchow and Linkow 1974, 1976).

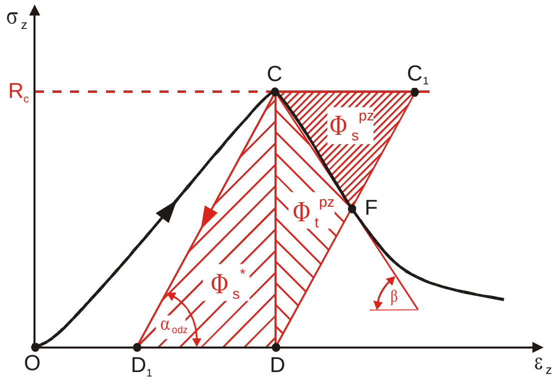


Fig. 3. Graphic representation of index W_{ET}^{pz}

The respective energies amount to:

$$\phi_s = \frac{R_c^2}{2E_{odz}}, \quad \phi_t^{pz} = \frac{R_c^2}{2(E_{odz} + M_s)}, \quad \phi_s^{pz} = \phi_s - \phi_t^{pz} \tag{5}$$

in which:

ϕ_s — elastic strain potential energy accumulated in the sample,

ϕ_t^{pz} — strain energy, lost for permanent deformations (brittle and plastic),

ϕ_s^{pz} — strain energy regained in the process of damage, which mostly changes into kinetic energy of damaged elements of the sample.

In this case, the index W_{ET}^{pz} is expressed in the following formula**:**

$$W_{ET}^{pz} = \frac{\phi_s^{pz}}{\phi_s}, \quad W_{ET}^{pz} = \frac{M}{E+M}, \quad W_{ET}^{pz} = \frac{1}{\frac{E}{M}+1} \quad \text{when} \quad |R_c| \le |\alpha_w \cdot p_z| \tag{6}$$

where:

E_{odz} — elastic modulus during unloading of the sample in pre-damage part,

M — post-damage modulus, which can be calculated twofold:

a) as a line tangent to the graph in post-damage part $M = \frac{\Delta\sigma}{\Delta\varepsilon}$,

b) as a relation $M = \frac{R_c - R_c^r}{\varepsilon_{pz}}$ (R_c^r – residual strength, ε_{pz} – post-damage strain,

α_w – coefficient of stress concentration in the area of the analyzed working (for the heading it can be assumed that α_w = 2.0-2.5, for the longwall α_w = 2.5-3.0).

The index should be used for the assessment of burst hazard in rocks constituting the seam or deposit planned for exploitation. The index is calculated only in the case when the compressive stress in the rocks around the studied working in the analyzed part of the seam exceeds the strength of rocks in the deposit R_c. If the compressive stress in rocks around the analyzed working is lower than the strength of the deposit, the rock is not damaged, hence no rockburst can occur.

If $W_{ET}^{pz} = 0$, only ideal plastic strain deformation occurs in the rock and such a rock is not prone to burst, at $W_{ET}^{pz} = 1$ ideal brittle damage occurs and the rock is highly prone to burst. A question arises: how to, depending on the value of the index W_{ET}^{pz}, assess rockburst hazard?

The experience gathered in the conditions of mines, partly verified in laboratory conditions, allows to propose the following division of rocks according to the value of W_{ET}^{pz}:

- rocks not prone or slightly prone to rockburst $0 \le W_{ET}^{pz} < 0.65$,
- rocks highly prone to rockburst $0.65 < W_{ET}^{pz} \le 1.0$

The studies carried out by Pietuchow (1972) suggest that the estimated initial velocity during rockburst ranges $3.0 \le v_0 \le 10.6$ m/s. The kinetic energy during rockburst can be determined from the formula $\Phi_k = \frac{1}{2}\rho v_0^2$. If we assume that the average velocity of damaged rocks (rock blocks) during rockburst equals v_0 = 6.8 m/s, we can determine the kinetic energy Φ_k, which describes the burst impact. If the strain energy regained in the process of damage Φ_s^{pz} is higher of equal to Φ_k ($\Phi_s^{pz} \ge \Phi_k$), then the rock can be described as prone to rockburst. Introducing the index $T_k = \frac{\Phi_s^{pz}}{\Phi_k}$ describing rockburst hazard, we have the following:

- $T_k \geq 1$ – rock prone to rockburst (**T** marking),
- $T_k \leq 1.0$ – rock not prompt to rockburst (**N** marking).

Using selected results of post-damage tests of several rock samples (Tajduś et al., 2003), average calculations of energy of the samples: Φ_s, Φ_s^{pz}, Φ_k, rockburst hazard index W_{ET}^{pz}, rock assessment in terms of burst hazard were carried out. The results are presented in Table 2.

TABLE 2

Rockburst hazard assessment for several selected rocks

Type of rock	**Site of extraction**	R_c **[MPa]**	Φ_s **[J/m³]**	Φ_s^{pz} **[J/m³]**	Φ_k **[J/m³]**	W_{ET}^{pz}	**Rock assessment "T_k"**
Limestone	Mine "Lubin"	108	$6.53 \cdot 10^4$	$5.59 \cdot 10^4$	$4.86 \cdot 10^4$	0,86	**T** – prone to rockburst
Dolomite	Mine "Lubin"	58	$1.72 \cdot 10^4$	$0.11 \cdot 10^4$	$5.78 \cdot 10^4$	0,07	**N** – not prone to rockburst
Anhydrite	Mine "Lubin"	98	$8.99 \cdot 10^4$	$4.67 \cdot 10^4$	$6.70 \cdot 10^4$	0,52	**N** – not prone to rockburst
Arenaceous shale	Mine "Wesoła"	105	$25.82 \cdot 10^4$	$20.54 \cdot 10^4$	$5.78 \cdot 10^4$	0.80	**T** – prone to rockburst
Sandstone	Mine "Wesoła"	60	$12.72 \cdot 10^4$	$8.31 \cdot 10^4$	$5.55 \cdot 10^4$	0.65	**T** – prone to rockburst

Two indices defined in such a way: W_{ET}^{pz} and T_k may allow for an initial assessment of the state of rockburst hazard of the areas, where mining exploitation is planned. The observations carried out in the "Wesoła" hard-coal mine and the "Lubin" copper ore mine seem to prove the above thesis. Arenaceous shales and sandstones deposited in the roof of the seam 501 of the "Wesoła" mine constitute a source of medium- and high-energy tremors, which may lead to the occurrence of rockburst. The rocks, according to

W_{ET}^{pz} classification, belong to the ones highly prone to burst (arenaceous shale $W_{ET}^{pz} = 0.80$, sandstone $W_{ET}^{pz} = 0.65$), also the index $T \geq 1$ for these rocks. Therefore, it can be assumed that in relation to the roof strata of the seam 501 of the "Wesoła" mine, the proposed indices suggest a serious state of rockburst hazard. In the roof of the "Lubin" mine there are rocks of a dolomite-limestone and anhydrite series with a considerable variability and a thickness of up to 200 m. Limestones, according to the proposed indices, are highly prone to rockburst; in the case of anhydrites it is difficult to provide an unequivocal answer, since the obtained results display large variability. Dolomite can be classified as a rock not prone to rockburst. Table 2 (above) presents the initial results of the study. An unambiguous assessment of rocks in relation to rockburst hazard requires further post-damage studies of rock properties with the use of unified research methodology.

2.6. K_{tap} index of rockburst hazard in the system 'deposit – surrounding rocks'

The index was proposed on the basis of the works of Pietuchow and Linkow (1974, 1976) pertaining to the fatigue crack propagation in rock medium. It was assumed that the condition for the occurrence of unstable behavior of rock mass (or rock sample) has the following form:

$$-\Delta\Psi > \Delta W_0 \tag{7}$$

$\Delta\Psi$ — energy supplied from outside (e.g. by the pads of strength testing machine),
ΔW_0 — energy consumed during damage.

The above equation was used for the description of results obtained from the damage test of samples with fatigue cracks. In this case, the energy consumed for the unit of crack surface does not exceed a certain value of g depending on the properties of material. For such a case, the inequity has the following form:

$$-\Delta\Psi > 2g\Delta S \tag{8}$$

$2g$ — maximum energy for the unit of fatigue crack surface,
ΔS — increase of crack surface, which corresponds to the energy supply $-\Delta\Psi$.

While adjusting the given criterion to mine conditions, it was assumed that the behavior of the analyzed rock sample describes the damaged part of the seam, whereas roof and floor strata fulfill the role of compression pads in the strength testing machine. The value $-d\Psi_1/dS_1$ can be expressed by means of the coefficient of stress intensiveness k_I according to the following formula:

$$-\frac{d\Psi_1}{dS_1} = (1-\nu_{sk}^2)\frac{k_I^2}{E_{sk}} \tag{9}$$

where:
ν_{sk}, E_{sk} — average Poisson's ratio and elastic modulus of roof strata,
k_I — coefficient of stress intensiveness in the contour of the working.

By means of substitution of the equation (9) to the equation (8), the following condition is obtained:

$$\frac{1-\nu_{sk}}{E_{sk}}k_I^2 > g_p \tag{10}$$

Assuming that the value g_p for the seam can be estimated according to the formula:

$$g_p \approx \frac{\sigma_{sr}^2}{2M_p}m \tag{11}$$

where:
σ_{sr} — average stress perpendicular to bedding,
M_p — post-damage modulus of the seam,
m — thickness of the seam.

And assuming after Pietuchow and Linkow (1976) the relation for the critical value of intensiveness coefficient

$$\left|k_I\right| < \left|k_{I_0}\right|, \quad \left|k_{I_0}\right| = \eta R_{c_p}\sqrt{\frac{m}{2}} \tag{12}$$

where:

R_{c_p} — average compression strength of the seam,

the relation was obtained, describing a local instability of the system ‘surrounding rocks – seam’ in the form:

$$(1-\nu_{sk}^2)\frac{M_p}{E_{sk}}(\eta\frac{R_{c_p}}{\sigma_{sr}})^2 > 1 \tag{13}$$

Using this condition, an index called K_{tap} describing rockburst hazard was proposed in the following form:

$$K_{tap} = (1-\nu_{sk}^2)\frac{M_p}{E_{sk}}(\eta\frac{R_{c_{sz}}}{\alpha_w \cdot p_z})^2 \tag{14}$$

where:

$R_{c_{sp}}$ — compression strength of the seam,

p_z — primary vertical stress.

The value of the coefficient η can be estimated by means of the following formula:

$$\eta = 0.613 \times \left[\frac{M_p}{E_{sk}}\right]^{-1.126} \tag{15}$$

then

$$K_{tap} = 0.376 \times (1-\nu_{sk}^2)\left[\frac{E_{sk}}{M_p}\right]^{0.8} \times \left[\frac{R_{c_p}}{\alpha_w \cdot p_z}\right]^2 \tag{16}$$

Depending on the value of K_{tap}, we have the following:

- index $K_{tap} \leq 1$ there are no conditions for rockburst occurrence,
- index $K_{tap} > 1$ there are conditions for rockburst occurrence.

In the index K_{tap} there appears both the relation $\frac{E_{sk}}{M_p}$ and the relation $\frac{R_{c_p}}{p_z}$, according to the study of Hoek (1999), responsible for the stability of rocks around mining workings.

Table 3 presents the indices of rockburst hazard in coal seam for various roof rocks. For coal, it was assumed that: $E_p = 2018$ MPa, $v = 0.3$, $R_c = 20$ MPa, $M_p = 7291$ MPa.

The analysis of Table 3 suggests that rockburst hazard occurs in the seam, in which fine-grained sandstone is deposited in the roof. The stronger the fine-grained sandstone, the higher the rockburst hazard. The index $\frac{M_p}{E_{sk}}$ provides rather unreliable results. The obtained results are similar to mine observations.

TABLE 3

Rockburst hazard indices for the system ‘seam – surrounding rocks’

Roof rocks	E_{sk} [MPa]	R_c [MPa]	v_{sk} [-]	H = 800 m	α_w	$\frac{M_p}{E_{sk}}$	K_{tap}
Coarse sandstone	4864	46.9	0.24	p_z = 20 MPa	2	1.50	0.35
Medium-grained sandstone	7208	54.4	0.22	p_z = 20 MPa	2	1.01	0.66
Fine-grained sandstone	8690	68	0.19	p_z = 20 MPa	2	0.84	1.20!
Strong fine-grained sandstone	12000	90	0.19	p_z = 20 MPa	2	0.61	2.73!
Mudstone	7638	62.6	0.18	p_z = 20 MPa	2	0.96	0.93
Siltstone	5642	44.1	0.19	p_z = 20 MPa	2	1.29	0.36

3. Numerical analysis of rockburst hazard in bedded rock mass

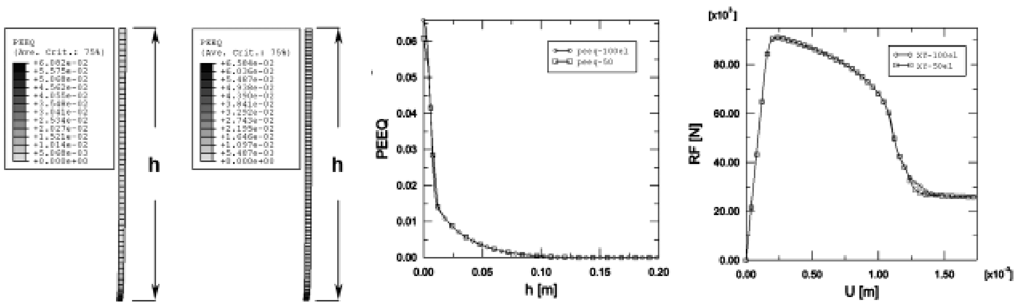
In this part of the paper, rockburst hazard of bedded rock mass is analyzed with the use of numerical calculations. For this purpose, computer simulations of damage development in exploitation face of excavated coal seam were carried out. Rockburst hazard was treated as the changes of intensiveness of the damage process in the area of mining face for various cases of geological structure of the working’s roof and floor. The influence of elastic properties of roof strata, as well as elastic and plastic properties with the weakening of coal deposit, on the course intensiveness of the damage process of the face, treating coal as an elastic-plastic medium with weakening.

3.1. Numerical calculations of media with softening

Modeling the damage of rocks as media, which during the damage process become weaker, brings about the consequences related to the assumption of the form of rock medium damage, as well as the numerical consequences directly related to the course of damage in the medium with softening.

Rock damage can occur in various forms, depending on rock properties and loading condition. In uniaxial state of stress, the rocks are subject to brittle damage in the form of individual cracks, often separated, and in the case of complex (biaxial and triaxial) loading in the form of shear cracks, shear bands or ductile deformation, typically plastic (Gustkiewicz, 1989; Gustkiewicz & Nowakowski, 2004; Wawersik et al., 1970; Mogi, 1971). The character of damage largely depends upon the loading conditions. In the case of the face, the state of stress occurs directly in the face, which is approximate to the biaxial one with the brittle fracture of coal observed in the *in situ* conditions. However, if we go deeper into the face, there appear conditions for damage in the form of shear or shear bands as a result of complex state of loading, under the influence of shear stress. In the calculation presented in this paper, it was assumed that the character of post-damage behavior of coal in the face is unstable and weakened, and in the conditions of triaxial loading, the damage occurs along the privileged shear planes. Such assumptions allow for the use of elastic-plastic model for the sake of the calculations, whereas in another case, the models based on fracture or damage mechanics should be applied.

A second consequence of modeling a medium with softening is numerical problems related to the change of type of differential equations with the appearance of softening (instability) and with pathological sensitivity of the obtained results to the discretization, which results of non unique solution. The sensitivity manifests itself in the form of plastic deformation concentration in the zones with the thickness depending on the size of FEM elements; the process of non-elastic deformation of the whole construction occurs practically in single elements in a certain direction of plastic deformation. The sensitivity of calculation results to the discretization becomes stronger alongside with the increase of density of finite elements assumed for the simulation. In the case of the calculations of coal deposit model discussed here, the material instability is caused by the plastic nonassiociated flow rule and plastic softening of the rock. Such problems can be solved by means of applying viscoplasticity (Sluys & Wang, 1998; De Borst et al., 1993; Cieślik, 2003; Cieślik & Tajduś, 2004), gradient models (Pamin, 1994; De Borst & Muhlhaus, 1991), Cosserat medium constitutive model (Muhlhaus & Vardoulakis, 1987), which introduce an additional scale parameter, the so-called "length scale", allowing to define the thickness of zones, in which plastic deformations concentrate in the conditions of softening. In this paper, the viscoplastic model of Perzyna (1966) was applied for the sake of calculating the medium with softening. However, in quasi-static issues, the model loses its physical interpretation (Diez et al., 2000; Wang & Sluys, 1997), and the parameters of viscoplasticity and stress excess function are selected by means of elimination of the effect of results' sensitivity to discretization and the proper description of the weakening zone thickness (Fig. 4; Cieślik & Tajduś, 2004).



a) b)

Fig. 4 a) Range of plastic strain band for viscoplastic model (from the left, the network consists of 25 elements, and of 50 elements), b) unambiguous band balance path determined for the model (Cieślik & Tajduś, 2004)

3.2. Model assumptions for the conditions of coal seam exploitation

Numerical calculations were carried out on an elastic model for the roof and floor, and on an elastic-plastic model with softening for the coal seam. Similarly to the analysis of rockburst hazard on the basis of laboratory tests, presented in the previous section of this paper, the geometry of numerical models embraced a simplified system 'seam – roof – floor' (Fig. 5). The coal seam with the thickness of 2 m and with the same thickness of roof and floor, i.e. 30 m, was assumed for the calculations. Mixed boundary conditions of solution were assumed, i.e. in the upper and lower edge, the section of rock mass was loaded with vertical, equally distributed loading resulting from the depth of deposition, for which p_z = 25 MPa, whereas the possibility of horizontal displacement was blocked for the vertical edges of the model.

The void openings of the exploitation (Fig. 5, grey color) were modeled as a zone of damaged rocks with equivalent parameters corresponding to the broken rocks (Table 4), whereas the working space was mapped without any support. These simplified calculation conditions corresponded to the assumptions of the analysis of rockburst hazard based on laboratory tests and they ignore many effects observed in real conditions of exploitation.

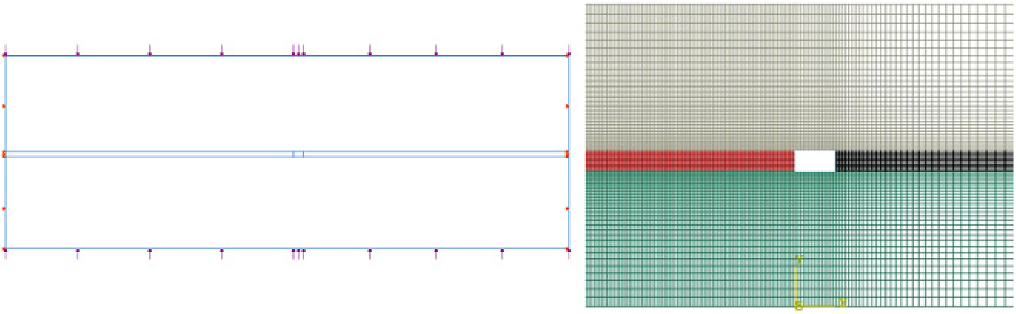
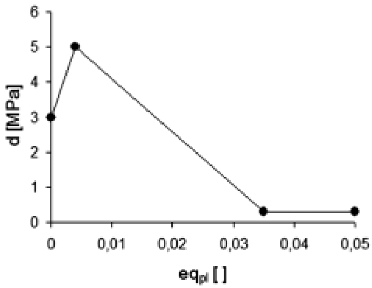


Fig. 5. Geometry, boundary conditions and discretization of calculation model of the system 'coal seam – roof – floor'

TABLE 4

Physical parameters of material models assumed in numerical calculations of the systems 'seam – roof – floor'

Parameters of constitutive model for coal	
E = 2500 MPa, ν = 0.3, β = 45 [°], ψ = 20 [°] D = 0.03 [s^{-1}]  	E – elastic modulus, ν – Poisson's ratio, β – friction angle, ψ – dilation angle, d – cohesion, ε_q^{pl} – intensity of plastic strain, $d(\varepsilon_q^{pl})$ – hardening and softening function D – the constant determining viscosity, $D = 1/T_m$, where T_m – relaxation time
Linear-elastic model of roof and floor strata	
Calculation variant I	E = 2.5 GPa, ν = 0.2
Calculation variant II	E = 5 GPa, ν = 0.2
Calculation variant III	E = 10 GPa, ν = 0.2
Calculation variant IV	E = 20 GPa, ν = 0.2
Roof fall	E = 0.15 GPa, ν = 0.4

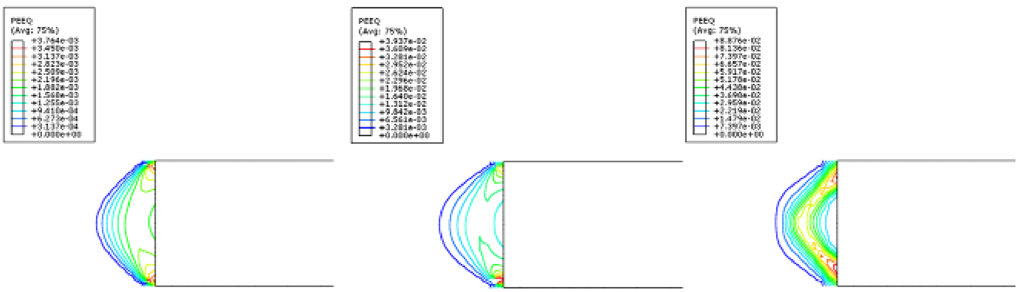
The parameters of particular rock mass strata used in the calculations are presented in Table 4. The parameters for the elastic-viscoplastic model with Drucker-Prager yield criterion were assumed on the basis of data taken from literature, pertaining to laboratory tests of coal in uniaxial (Bukowska & Gawryś, 2010; Klisowski & Iwulski, 2006) and triaxial state of stress (Krzysztoń & Sanetra, 2003).

The calculations were carried out in three calculation steps: the first step provided the initial state of stress, the second step – mapping of the exploitation front in the middle of the model (by means of modeling the abandoned workings, which occurred as a result of roof strata caving), and the third step – mapping of exploitation front movement by the advance resulting from the applied excavation technology. The incremental method of solving FEM algebraic equations was applied for the sake of calculations, accompanied with Newton-Raphson iterative algorithm (Kleiber, 1995).

The main aim of the calculations was the analysis of behavior and development of the face damage for changing roof and floor conditions, i.e. with constant parameters of physical model for the coal seam, the roof and floor deformation properties were changed. Such conditions assumed for the sake of calculation corresponded, though not directly, to the analyzed variants of the relation M_p/E_{sk} assumed in the analyses of rockburst hazard based on the results of laboratory tests presented in the works of Tajduś et al. (2003) and Bukowska (2002). The range of roof and floor deformation parameters assumed for the numerical calculations corresponded to the real values of parameters for rocks occurring in the conditions of Upper Silesian Coal Basin.

3.3. Analysis of calculation results

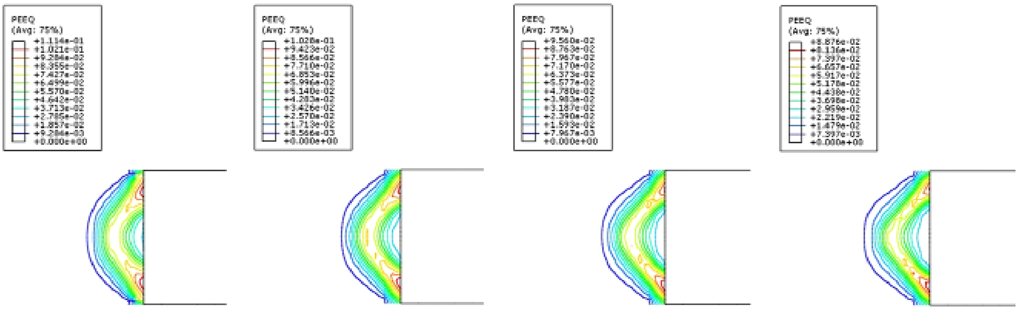
Damage development in the coal face with the evolving process of establishing secondary force balance in the rock mass after the advance, in the form of equivalent plastic strains in variant IV of the calculations, was presented in Figs. 6a-c.



a) b) c)

Fig. 6. Plastic strain development in the face, corresponding to coal damage process.
a), b), c) respectively: initial, transitory and final phase of establishing the force balance in the coal face

In the initial phase of the damage process of coal in the face, after completing the advance, permanent strain is concentrated directly in the face (Fig. 5a), which in real conditions may correspond to numerous cracks and face yielding. In the advanced level of strain at the depth of approximately half the height of the wall, the concentration of permanent strain can occur in the form of a certain zone (Fig. 5c). The width of the zone depends upon the post-damage properties of coal. Such a state in real conditions of exploitation may be achieved at low bearing capacity of power support. On the basis of analysis of the results obtained for all the four variants (Figs. 7a-d), the conclusion can be drawn that the increase of elastic modulus value of roof and floor strata (at invariable parameters of coal deposit) does not seriously affect the range and dimensions of damage zone (Fig. 8).



a) variant I a) variant II a) variant III a) variant IV

Fig. 7. Maps of permanent equivalent strain obtained in particular calculation variants

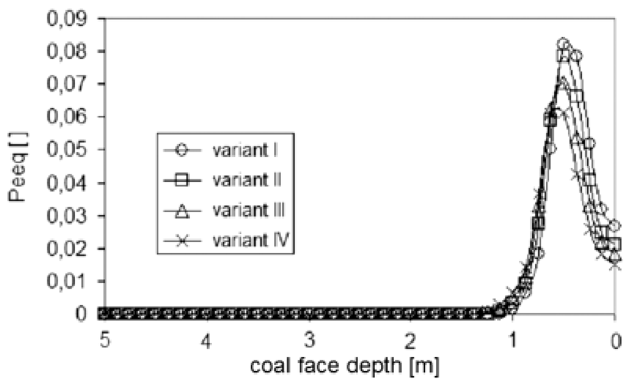


Fig. 8. Range of permanent equivalent strain accompanying the face damage in various calculation variants

However, the value of permanent strain obtained in particular variants of calculation changes (Fig. 8). In the case of the roof and floor with the lowest elastic modulus (calculation variant I), the values of obtained strain in the damage zone were largest, whereas in the variant with the highest value of roof and floor elastic modulus, they were lowest. These results show the intensiveness of the damage process, depending on the roof and floor properties, and they correlate with the rockburst index M_p/E_{sk}. In the calculated variants, the rockburst index M_p/E_{sk} has the highest value in variant I (the lowest value of roof and floor elastic modulus E_{sk}, at a constant value of M_p for the seam) and in this variant there is a highest intensiveness of the damage process. In the subsequent calculated variants the value M_p/E_{sk} decreases and, simultaneously, also the intensiveness of the damage process decreases (the value of permanent strain decreases as a result of damage work).

What also provides an interesting insight into the analyzed problem are the graphs of total elastic strain energy density A_c prepared in the middle of the wall's height and along its longwall panel (Fig. 8a), as well as the value of energy dissipated into permanent strain (marked as ALLPD), calculated as the sum of total volume of the seam for particular calculation variants (Fig. 8b).

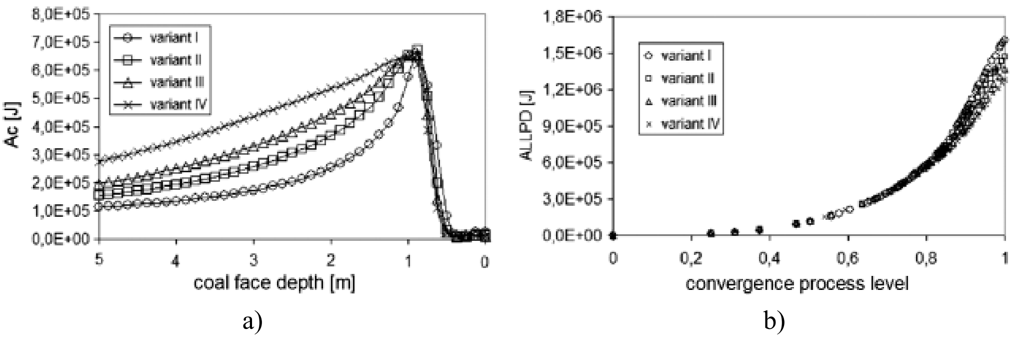


Fig. 8. Graph of total elastic strain energy density a), and value of energy dissipated into permanent strain b)

In the unstressed zone, which is a result of coal damage (Fig. 8a), the elastic strain energy density directly in the face (in all cases the zone has approx. 0.5m) is minimal, whereas while approaching the seam damage boundary it increases dramatically. After exceeding the zone, where permanent strain is concentrated (approx. 1m), the elastic energy density changes to the value resulting from the initial state of stress, however for the roofs more prone to deformation, it reaches the initial state of stress much faster than in the case of the roofs less prone to deformation. Such a distribution of elastic energy concentration depends directly on the roof and floor strain parameters, as well as on the process of energy dissipation in damage zones. In the graph of the energy dissipated into permanent strain (Fig. 8b), the intensiveness of the damage process can be observed, which in the case of the roof with the highest deformation tendency (variant I) is highest, whereas in the case of the roof with lowest deformation tendency (variant IV), it is also lowest.

4. Conclusions

The results of rockburst hazard assessment analysis of bedded rock mass, presented in the first part of the paper, indicated that in the case of the seam, which is potentially able to accumulate more energy than necessary for its damage, and in certain special conditions of elastic properties of the roof and post-critical properties of coal deposit, there is a potential rockburst hazard, i.e. hazard with unstable behavior of rock mass. Qualitatively, such a state causes the situation, in which energy supplied from the surrounding strata during the seam damage is higher than the maximum energy that can be accumulated.

A similar phenomenon can be observed during the analysis of numerical calculation results. Changes of roof and floor elastic modulus result in serious changes of intensiveness of energy dissipation process in shear bands. The larger the roof and floor tendency to deformation, the more intensive and violent character of the process. With the approximately constant width of shear band, the dissipated elastic energy tends to increase, which can be seen in the graph in the form of rising values of permanent strain. In terms of the analyzed roof and floor strain parameters, the loss of stability condition, which in numerical calculations would have to finish with the divergence of the incremental-iterative process, was not obtained. The laboratory test results on rockburst hazard of bedded rock mass qualitatively correspond to the results of numerical calculations. However, due to the differences arising from the analysis methodology, it is difficult to assume a common quantitative criterion of rock mass stability loss.

The research and present publication are part of the statutory research carried out in the Department of Geomechanics, Civil Engineering and Geotechnics of AGH University of Science and Technology – No. 11.11.100.277

References

Bukowska M., 2002. *Geomechanical properties of rocks from the rockburst hazard viewpoint*. Arch. Min. Sci., Vol. 47, No 2.

Bukowska M., Gawryś J., 2010. *Właściwości fizyczne węgli GZW w aspekcie wyrzutów gazów i skał*. Górnictwo i Geoinżynieria, 34, 2.

Cieślik J., 2003. *Optymalizacja geometrii filarów w komorowo filarowym systemie eksploatacji rud*. Praca doktorska - niepublikowana, Kraków.

Cieślik J. Tajduś A., 2004. *Pewne możliwości zastosowania teorii sprężysto-lepkoplastyczności do numerycznej analizy lokalizacji odkształceń plastycznych w filarze skalnym w warunkach quasistatycznego obciążenia*. XVI Konferencja Naukowa Metody Komputerowe w Projektowaniu i Analizie Konstrukcji Hydrotechnicznych, Kraków-Korbielów, 2004.

De Borst R., Muhlhaus H.B., 1991. *Gradient-dependent plasticity: formulation and algorithmic aspect*. Int. J. Num. Meth. Eng. 117, 733-75.3

Diez P., Arroyo M., Huerta A., 2000. *Adaptivity based on error estimation for viscoplastic softening materials*. Mech. of Cohes.-Frict. Mater., 5, 87-112.

Gustkiewicz J., 1989. *Synoptic view of mechanical behavior of rock under triaxial compression*. Rock at Great Depth, Maury&Fourmaintraux (eds.), Balema, Rotterdam, 3-10.

Gustkiewicz J., Nowakowski A., 2004. *Deformacje i pękanie skał w warunkach laboratoryjnych*. Arch. Min. Sci., Vol. 49, Special Issue, p. 9-50.

Hoek E., 1999. *Suport for very weak rock associated with faults and shear" Rock Support and Reinforcement Practice in Mining*, Villaescusa, Windsor & Thompson (eds), Balkema, Rotterdam, 19-32.

Kleiber M., 1995. *Komputerowe Metody Mechaniki Ciał Stałych*. Seria: Mechanika Techniczna, t. XI, PWN.

Klisowski R., Iwulski Z., 2006. *Badania przed- i pokrytycznych właściwości próbek węgla kamiennego*. Warsztaty 2006 z cyklu: Zagrożenia naturalne w górnictwie, str. 511-522 Kraków-Tomaszowice, 12-14 czerwca 2006.

Krzysztoń D., Sanetra U., 2003. *Badania trójosiowego ściskania skał przy wartości ciśnienia okólnego w zakresie 0-70 MPa*. Arch. Min. Sci., Vol. 48, No 2.

Mogi K., 1971. *Effect of the triaxial stress system on the failure of dolomite and limestone*. Tectonophysics, 11, 111-127.

Muhlhaus H.B., Vardoulakis I., 1987. *The thickness of shear bands in granular materials*. Geotechnique, 37, 271-283.

Pamin J., 1994. *Gradient Dependent Plasticity in Numerical Simulation of Localization Phenomena*. Praca Doktorska, Delft University Press, Delft, Netherlands.

Pietuchow I.M., Linkow A.M. 1974: *Ob energeticzeskom kriterii ustrojcziwosti w teorii gornych udarow*. Trudy Wsiesojuznowo Nauczno-Isledawatielskowo Instytuta Gornoj Geomechaniki i Markszejderskowo Dieła (WNIMI), sb. 91.

Pietuchow I.M., Linkow A.M. i drugije, 1976. *Teorija zaszczitnych płastow*. Izd. NIEDRA.

Perzyna P., 1966. *Teoria lepkoplastyczności*. PWN, Warszawa.

Smółka J., Bałazy I. i in., 1978. *Metoda określania skłonności do tąpań zwięzłych piaskowców i łupków piaszczystych (mułowców) otaczających pokłady węglowe*. Dokumentacja pracy GIG 01.4.01.48, Katowice.

Szecówka Z., Domżał J., Ożana P., 1973. *Energetyczny wskaźnik skłonności naturalnej węgla do tąpań*. Prace GIG, Komunikat nr 594, Katowice.

Tajduś A., Flisiak D., Klisowski R., 2003. *Propozycja wskaźnika dla oceny skłonności do tąpań układu warstwy stropowe--pokład*. Geotechnika w budownictwie i górnictwie, Oficyna Wydawnicza Politechniki Wrocławskiej, s. 254-263.

Tajduś A., Szumiński A., Olesiak S., 2003. *Klasyfikacja skał tąpiących na podstawie wskaźnika wykorzystującego wyniki badań pozniszczeniowych, Geotechnika w budownictwie i górnictwie*. Oficyna Wydawnicza Politechniki Wrocławskiej, s. 263-271

Wang W.M., Sluys L.J., De Borst R., 1997. *Viscoplasticity for instabilities due to strain softening and strain-rate softening*. Int. J. Numer. Meth. Engng., 40, 3839-3864

Wawersik W.R., Fairhurst C., 1970. *A study of brittle rock fracture in laboratory compression experiments*. International Journal of Rock Mechanics & Mining Sciences 7, 561-575.

Received: 07 May 2014