Stability of roadways in coalmines alias rock mechanics in practice

Richard Šnupárek*, Petr Konečný
Institute of Geonics, Academy of Sciences of Czech Republic, Brno 60200, Czech Republic
Received 23 June 2010; received in revised form 28 August 2010; accepted 8 September 2010

Abstract: This paper describes the procedures for design of supports and stabilization measures in the roadways. The procedures are based on the system developed in Ostrava-Karviná coal basin in Czech Republic. The calculation of load bearing capacity of roadway supports contains the period of roadway construction and mining in the vicinities, based on the size of the natural rock arch. The loading of supports during mining comes from a stress wave in the rock mass in the forefront of coalface and the caving area of mined-out panel. The input data for the calculation method are deduced according to in-situ measurements of convergence and displacement in the roadways.

Keywords: coalmines; roadways; stability; support; bolting

1 Introduction

Many European coal deposits are exploited by the longwall mining method with controlled caving. Experiences show that stability of the roadways, which ensures transport and ventilation in coalfaces, affects their output, safety, and economy in coal production [1].

The problems of rock mechanics for stability of the roadways are especially urgent for deep mining. Recently, primary studies in the coal mining fields were published in countries with active deep mines [2–4]. For very hard rock layers with high geostress, a dynamic phenomenon—rockburst can be induced, which will greatly affect the safety of roadways. Rockburst problem has become an urgent issue in deep mining. Analysis of rockburst effects in Ostrava-Karviná Coalfield (OKR) has indicated a principal difference between effects of rockbursts on gates and those on longwall faces. From the analysis of rockbursts in OKR from 1993 to 2003, it is evident that among the total number of 64 already investigated cases, 61 rockburst events (or 95%) happened in the long mine workings, e.g. roadways and break-throughs [5].

Roadways are exposed to the effect of rock stress associated with the excavation of roadways and extraction of coal seams, which significantly affect the stability of the rock mass. Support design methods often use both empirical methods based on measurements and field observation in roadways and modeling methods. Recently, numerical models often prevail and physical modeling techniques are also used.

Due to the large number of roadways, the currently used roadway support planning follows empirical or combined empirical-analytical methods. In this paper, using the research achievements in the Czech part of the Upper Silesian Coal Basin, we will show that, in general, it is necessary to take into account all necessary aspects in the field of rock mechanics to obtain successful results.

2 Geological and mining conditions

The OKR is the Czech part of the Upper Silesian Coal Basin, and occupies a region of about 1 500 km². Geologically, the basin belongs to the Upper Carboniferous rock (Namurian A, B, C and Westphalian A), and is divided into the Ostrava and Karviná formations. The Ostrava formation consists of a large number of coal seams with varying thickness, predominantly with fine grained marine and paralic sediments in the overlying strata. The younger Karviná formation is composed exclusively of continental sediments (Fig.1).

The total thickness of the coal-bearing Upper Carboniferous layer is about 3 000 m with 415 coal seams, of which 141 can be mined. The thicknesses
of the seams vary considerably from an average value of 73 cm in Ostrava formation to 180 cm in Karvina formation. In the Karvina, the formation seams sporadically have the thicknesses between 12 and 15 m. The major rock types are sandstone, siltstone (sandy shale), mudstone and conglomerate. They have a mean uniaxial compressive strength of 90, 80, 60 and 20 MPa, respectively. There is a Germano-type tectonic structure in the Karviná part of the Basin, with a frequent occurrence of throws. Recently, the main extraction is underway in the east Karvina part with dominant sub-horizontal bedding and faults striking in the NS and WE directions.

The hard coal in OKR is mined underground. After 250 years of coal mining, it is found that the mean mining depth is about 800 m, and the maximum one reaches 1 300 m. The only employed method is longwall mining method with a longwall face connected to a gateroad at each side of the extracted panel (maingate and tailgate). The high-production longwall panels have a length from about 150 to 200 m (or more) and an excavated thickness from 2.5 to 4 m (or 6 m in special cases). The speed of roadway advancing depends strongly on the geological conditions and is usually 1.2–4 m per day, but often more.

The gateways are driven in the seam, often with some stripping in the floor or on the roof and their full-size cross-section is 15–20 m² on average, with a mean advance of 8–10 m per day. The gateroads are mainly supported with yielding steel arch support (TH arches). Small parts of gateroads with a hard rock overburden are of a rectangular shape with roof bolting. The geomechanical classification parameter RMR, according to Bieniawski [6], is used for design of roof bolting.

3 Characterization of the problem

The methods for designing the roadway supports in OKR are based on the empirical findings and primarily on the database of measurements gained in-situ approximately during the last forty years (by the Coal Research Institute, Ostrava Radvanice, Institute of Geonics (former Mining Institute) Czech Academy of Sciences, DPB Paskov). This database includes extensive measurements of convergence in different stages of roadway construction and service. Based on the measurements, the algorithms were derived for convergence calculations depending on the depth, roadway dimensions, rock mass properties, coal seam thickness and mining methods. At the same time, zones of inelastic deformation around the mine workings can be measured by using extensometers and geophysical techniques, and the correlations between the values of convergence in mine workings and the dimensions of the inelastic deformation zones are formulated. This procedure makes it possible to establish the approaches for calculation of support loads caused by the weight of the loosened roof in the relevant phases of a roadway service life. The extensive laboratory and model test studies are used to investigate the properties of the supports and supporting elements, leading to the determination of their bearing capacities that depend on their design, dimensions, and materials used and conditions of installation. On this basis, the complex methodology for the estimation of the bearing capacity of supports in roadways in OKR is proposed.

4 Determination of parameters of conventional supports in roadways

The supports in roadways are dimensioned according to the loads that are applied during the roadway service life. During the first period of construction of a roadway, the minimum bearing capacity of the supports must be adequate for bearing the load of the loosened rock in its vicinity or a portion
of this load depending on the varying cases. The support therefore bears the loads, and at the same time makes the loosened rock have, to a certain degree, coherence by stopping continues rock loosening.

The area of rock loosening in the vicinity of a longwall mining and the relevant rock arch above this mine working was described by Pariseau [7]. For the geotechnical conditions in OKR, the Protodjakonov’s theory, which utilizes the values of the angle of internal friction ($\phi$) of rocks for the characterization of the rock mass, leads often to good results for rock arch descriptions.

According to the arch theory, a natural arch is formed above the roadway, and along this natural arch, the rocks may loosen and separate from the rock massif (Fig.2).

![Fig.2 Sketch of a rock arch above the mine working with an arch support where $\phi_k$ is friction angle.](image)

The arch height can be determined using the convergence measurement results and the loads on roadway supports. Under the conditions of OKR, based on the extensive measurements in galleries, it is deduced that the displacement of rocks against the mine workings, could be calculated according to the following relationship [8]:

$$u = 0.1B(1 - e^{-0.015t}) \left( e^{\frac{1.5H - q}{45\sigma_k}} - 1 \right)$$  \hspace{1cm} (1)

where $u$ is the displacement of rocks in a roadway in period of excavation, $H$ is the so-called effective depth, $B$ is the mine workings’ dimensions (usually the width), $t$ is time (day), $\sigma_k$ is the reduced compressive strength of rocks, and $q$ is the resistance (bearing capacity) of the support.

The reduced compressive strength of overlying rocks $\sigma_k$ is defined as

$$\sigma_k = \beta \sum \sigma_{\sigma_i} m_i$$  \hspace{1cm} (2)

where $\beta$ is the coefficient of stratification in Table 1, based on the results of laboratory and model research of the uniaxial compressive strength of carboniferous rocks with different numbers of layers; $\sigma_{\sigma_i}$ is the uniaxial compression strength of the $i$-th layer; $m_i$ is the thickness of the $i$-th layer. The overlying strata at the distance of $2B$ above the roof of the mine workings are assessed.

![Table 1 The coefficients of stratification $\beta$.](image)

<table>
<thead>
<tr>
<th>Number of layer</th>
<th>$\beta$</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>1.00</td>
</tr>
<tr>
<td>2</td>
<td>0.95</td>
</tr>
<tr>
<td>3</td>
<td>0.90</td>
</tr>
<tr>
<td>4</td>
<td>0.86</td>
</tr>
<tr>
<td>5</td>
<td>0.82</td>
</tr>
<tr>
<td>6</td>
<td>0.79</td>
</tr>
<tr>
<td>7</td>
<td>0.76</td>
</tr>
<tr>
<td>8</td>
<td>0.73</td>
</tr>
<tr>
<td>9</td>
<td>0.71</td>
</tr>
<tr>
<td>10</td>
<td>0.70</td>
</tr>
</tbody>
</table>

The zone of inelastic deformation is used for specification of the load on the supports and corresponds to the height of the arch. In the past, it was verified with extensometers and geophysical techniques (Figs.3 and 4) [9]. In Fig.3, symbols ①–⑤ represent different rocks.

![Fig.3 Influenced zones in the vicinity of a roadway according to the velocity of seismic waves](image)

Radial boreholes are drilled from roadways and their changes in wave velocity are measured at an interval of 10 cm in several periods: from 2 weeks to 6 months after starting the roadway driving. Without influence of mining, the natural arch is stable in 2–3 months. In Fig.3, Zone (a) in Fig.3 answers the value of $n_B$ of natural arch around a roadway, as defined below.

The height of the natural arch could be determined from the results of convergence measurements according to the following equation [8]:

$$B_n = K_n B^{0.4} u^{-0.6}$$  \hspace{1cm} (3)

where $K_n$ is the coefficient characterizing the relationship between the zone of inelastic deformation of
the mine workings with a width of $B$ and the displacement $u$, and it is evaluated in the past on the basis of in-situ measurements [10].

After modification and for $t \to \infty$, we can get

$$B_n = 2B \left( e^{0.03 \frac{H}{B}} - 1 \right)^{0.6}$$

(4)

For the conventional supports in roadways, we assume the load of rock layers in the overburden. From the equation of equilibrium of an infinitesimal layer $dz$ for a column of rocks in the roof of mine workings of a width of $B$ and a unit length, the following equation is derived:

$$dQ + B\sigma_z - B(\sigma + dz) - 2\tau dz = 0$$

where $dQ = B\gamma dz$ is the weight of infinitesimal rock layer, $\sigma_z$ is the vertical stress, and $\gamma$ is the specific weight of rocks (Fig.5).

![Fig.5 The scheme of loading.](image)

The horizontal stress $\sigma$ and shear stress $\tau$ can be expressed respectively as

$$\sigma = \lambda \sigma_z, \quad \tau = \sigma \tan \varphi + K$$

(6)

where $\lambda$ is the coefficient of lateral pressure, and $K$ is the cohesion of rocks.

The solution to Eq.(5) leads to the assumed load on the support:

$$p = \frac{e^\frac{B}{2} \gamma - K}{\lambda \tan \varphi} \left( 1 - e^{-\frac{\lambda \tan \varphi}{\lambda}} \right)$$

(7)

If we assume that: (1) the height of the column of rock layers is equal to the height of the zone of inelastic deformations, (2) the values of the specific weight of rock is 25 kN/m$^3$ and the angle of internal friction is $45^\circ$ (a conservative assumption), and (3) the roof arch is of a parabolic shape then after modifications, the expression for determination of the standard load $q$ can be obtained:

$$q = 48B \left( 1 - \frac{0.08K}{B} \right) \left[ 1 - e^{-\frac{0.035}{\sqrt{\frac{H}{B}}}} \right]$$

(8)

where $q$ represents a minimum necessary bearing capacity of the conventional supports in roadways.

The effective depth $H$ is given by

$$H = H_G k$$

(9)

where $H_G$ is the geometric depth under the surface (m); $k$ is the coefficient of additional load on the roadway due to extraction operations. It is the ratio of actual stress in the particular point to the stress corresponding to the depth below the surface. It is determined individually from the idealized graphs of stress conditions or from the results of mathematical modelling, etc..

Based on the measurement of the load applied to the supports, the values of cohesion are determined for three general types of overburden according to the classification of the roof caving categories applied, i.e. for the collapsible roof, regularly caving in roof and roof with delayed caving (Table 2).

To use the above-mentioned procedure, a monograph is plotted to determine the required bearing capacity of roadway supports (Fig.6) [11] (the light profile in the figure means the inner profile inside roadway arch support).

On the basis of comparison of the calculated standard load $q_n$ with the nominal bearing capacity of supporting arches, a necessary spacing $d$ of arches for the conventional support is determined from the following relationship:

$$d \leq \frac{q_{vl}}{q_n}$$

(10)

where $q_{vl}$ is the rated bearing capacity of one arch of the applied support (kN).
Table 2 Values of coefficients for calculation of bearing capacity of face supports.

<table>
<thead>
<tr>
<th>Roof category</th>
<th>Core lumpiness (cm)</th>
<th>Caving cushion composition</th>
<th>Cohesion (kPa)</th>
<th>Lithological type of roof</th>
</tr>
</thead>
<tbody>
<tr>
<td>Collapsible</td>
<td>&lt; 7.5</td>
<td>Chaotically arranged pieces of rock with roughly the same dimensions</td>
<td>11.8</td>
<td>Thicker attitudes of mudstones, thin-bedded stones, failure zones, roof of lower benches (with upper bench mined out)</td>
</tr>
<tr>
<td>Regularly caving</td>
<td>7.5–10.5</td>
<td>Chaotically arranged pieces of rock in lower part, coulisse-like fault above</td>
<td>19.0</td>
<td>Frequent interlaminating of mudstones, siltstones and sandstones with fine up to medium size of grain. Directly above the supports, well collapsing, hard rocks above</td>
</tr>
<tr>
<td>Hard rock with frequently delayed caving</td>
<td>&gt;10.5</td>
<td>Blocks of rocks arranged roughly in same way as in original attitude</td>
<td>23.7</td>
<td>Thick-bedded siltstones. Interlaminating layers with medium up to coarse grain sandstones and conglomerates</td>
</tr>
</tbody>
</table>

Fig.6 Normal load for regularly caving overburden [11].

5 Additional load on roadway support due to advancing of coalface and interconnection of caved areas

Afterwards, when the coalface advances from the starting of a connecting gate, the additional load is applied to a coal pillar due to the redistribution of the stress field. A stress wave propagates in the forefront of the coalface. It also aggravates the roadways. The example of stress distribution in the vicinity of a coalface calculated using a finite element model is shown in Fig.7. The extent and intensity of this wave depend mainly on the depth of coal extraction, the exploited thickness and geomechanical and geological conditions.

When taking into consideration the general assumptions, regarding the interconnection of the area below the arch in the roof of long mine workings with the caved space above the advancing coalface, the additional load on the roadway support during the exploitation of coalface can be deduced from a schematic concept of the mechanism, causing the additional load according to Fig.8.

During the mining at the coalface, a caving cushion with the thickness of $M_z$ is formed above the waste area. According to experiences, we can assume that this caving cushion will interconnect with the arch space above the mine workings. The zone of the loosened rock in the space above the long mine workings will increase and the load of roadway support will increase correspondingly. Therefore, the
additional load in the cross-section can be expressed approximately by means of the difference between the triangle area $ABC$ and the area of a parabolic arch above the mine workings, i.e. above the abscissa $AB$ (Fig.9).

![Fig.9 Geometric chart of surcharge of roadway supports.](image)

From these geometric relationships (Fig.9), it is apparent that the dimensions of the roadways, the angle of internal friction, the thickness of the caving cushion above the coalface and the angle of roof caving are the parameters affecting the additional load applied on roadway supports.

For simple orientations of mining engineering, we plot the monographs for the usually applied widths of mine workings; the monograph can be used to determine the additional load on the conventional support in the roadway, which is induced by the advancing coalface. The example of this monograph for a width of 5 m of the mine workings is shown in Fig.10.

![Fig.10 Surcharge of 5 m wide roadway supports under different compressive strengths of overlying rock.](image)

6 Reinforcement and stabilization of roadway supports

The usual methods of reinforcement used until today consist of the installation of middle props. They are installed on wooden sills and are used as the supports under the roof arches of steel arch supports.

At present, roadways are successfully stabilized by using cable bolting with steel arch supports [12]. The essence of this stabilization is to support the roadway arches with the beams bolted into the main roof using long cable bolts. This reinforcement of supports should replace the so far used middle props (Fig.11).

![Fig.11 Chart of a long cable bolting.](image)

The supporting beam is suspended on flexible components (ropes or tendons from the high-strength steel). The general difference of high roof bolting in comparison with roof bolting in roadways can be characterized as follows:

1. High-performance anchors cannot create the reinforced area in the vicinity of roadway, but they transfer forces into a remote massif, affecting the roadway support.

2. High bolts must be anchored into the massif, which is not disturbed by mining operations and therefore, the length of high-performance anchors exceeds significantly the length of usual anchors.

3. Requirements for the strength of individual high-performance anchors are considerably stricter in comparison with usual bolts.

4. The above-mentioned requirements lead to a different design of long anchors, which consist of root (anchoring) part and free length, and utilize primarily the wire and strand elements.

A standard length of anchors, which utilizes fully the tensile resistance of anchors glued with polymers, is usually 2 m in length. In order to determine simply the total length of high-performance anchors according to the described principles, we plot a monograph for the determination of the minimum length of anchors to be suspended on the conventional roadway supports (Fig.12).
When some geological features of weakness of rocks occur in the estimated area (faults, weak rock layers such as coal seams, water-bearing fractures), tendons must be prolonged.

The dimensions of high-strength bolting based on the assumed load are determined depending on the expected additional load applied in accordance with Fig. 9, considering choices of anchors and their number and spacing.

A specific case is the reinforcement of roadways designed for two neighboring panels. These roadways are reinforced with the combined supports, i.e. with roof bolting and conventional steel arch supports. Roof bolting is a proactive stabilization element of a rock massif. In order to operate properly, roof bolting should be established as promptly as possible after the advancing of the face of the mine workings when the compactness (alteration) of rock massif has not been disturbed yet due to the effect of mining-induced rock stress changes. The methodology for the calculation of a separate roof bolting can be used for the calculation of roof bolting parameters for the load during the period of roadway excavation.

The conventional steel arch supports are the passive supports and begin to function when the load is applied from the adjacent massive, i.e. with the development of convergence of mine workings. The conventional steel arch supports are installed usually as a part of the combined supports at appropriate intervals behind the roadhead. However, the largest distance is approximately 40 m away from the roadhead. In the phase, when rock bolting has a restricted function with the alteration and disturbance of rock massif, the roadway stability is ensured mainly by using the conventional steel supports. Therefore, the parameters of steel arch supports are calculated for the full bearing capacity of the separated supports. With regard to the fact that the relevant roadway is supported simultaneously with two types of supports (roof bolting and steel arch supports), the factor of safety of 1 is usually considered while each support is dimensioned to the full bearing capacity. The sufficient safety of the whole structure is ensured with the simultaneous effect of both supports used.

Based on the principles described, the standard schemes have been prepared for the characteristic conditions in OKR. Figure 13 illustrates a support scheme for the roadway with a cross-section of 16 m² in the coal seam with a thickness of 2.5 m, situated at a depth of 800 m.

**7 Conclusions**

1. The empirical methods of roadway support design, based on measurement and observation in roadways, are used in OKR. The methods contain different operational periods of roadway life: driving in front of advancing longwall faces and after the passage of longwall faces.

2. Equations for calculating the required bearing capacities of roadway supports in the period of driving were derived based on the natural arch (zone of inelastic deformation) around a roadway. Three categories of overlying rocks with different values of residual cohesion were considered.

3. Determination of support load in the period of advancing working faces is based on the patterns of induced vertical stress in front of longwall faces. The patterns were obtained from mathematical models.

4. The additional support load in the period after the passage of longwall faces was estimated based on the interconnection of the natural arches around...
roadways with the caved space above the advancing coalface.

(5) At present, roadways are successfully stabilized by using the technology of cable bolting with the steel arch supports and the combined supports, i.e. with roof bolting and conventional steel arch supports. Methods of design and dimensions, and the principles of executing these supports are based on the same assumption and measurement.

(6) It is confirmed that for the design of roadways, it is necessary to use many procedures of rock mechanics, including the classification of rock mass, laboratory tests of strength and deformation properties of rocks, ascertaining the character of disturbance of rock massif, measurement of stress in a rock massif with geophysical methods, monitoring deformation phenomena around underground mine workings, as well as the procedures of mathematical modelling of geo-mechanical tasks.

Acknowledgements

The work is elaborated within the framework of the research plan: physical and environmental processes in lithosphere induced by anthropogenic activities (AV0Z30860518).

References