A time-independent energy model to determine the height of destressed zone above the mined panel in longwall coal mining

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ABSTRACT

Longwall mining is the most widely used method in underground coal extraction. After the coal seam extraction, the strata above the mined panel will be destressed and then the overburden weights will be redistributed and transferred to the front abutment and neighboring solid sections. In this research, the height of destressed zone (HDZ) is taken as equivalent to the combined height of caving and fracturing zones above the mined panel roof induced due to longwall mining. Accurate determination of this height is essential to proper estimate of transferred loads toward the gates and pillars. In this paper, a time-independent analytical model based on the strain energy balance in longwall mining is developed to determine the HDZ. Unlike the previous analytical and empirical models, the proposed energy model incorporates both possible influencing geometrical and geomechanical parameters in calculating of HDZ. The model proposed sensitivity analysis shows that depth of cover, extracted coal seam thickness and unit weight have the direct influence and elastic modulus, Poisson ratio, uniaxial compressive strength and bulking factor have the inverse influence on the HDZ. The results of the proposed model are compared with the results of in-situ measurements and existing, analytical, empirical and numerical models reported in literature. The comparison results show that the proposed model complies with the comparable models as well as with the in-situ measurements. Therefore, it can be concluded that the proposed energy model can be successfully used to determine the height of destressed zone above the mined panel in longwall mining.

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1. Introduction

Longwall mining is the most widely used underground mining method that involves the complete removal of large rectangular panels of coal seams. This method is suitable for relatively flat, thick and uniform coal beds. One of the most important advantages of this method is the automated form of underground coal mining which leads to characterized by high efficiency. The main objective of underground coal mining is to extract of coal from the ground safely and economically. However, the mining efficiency exclusively depends on the coal seams and roof rock strata overall conditions. In this method, due to the extraction of coal seams within a defined panel width, after advancing the hydraulic jacks, the immediate roof of the mined panel is unsupported and hence is allowed to collapse and cave in some distance behind the hydraulic jacks or in the goaf area. The downward movement of the roof strata then gradually extends upwards and will cause the disturbed roof strata to become destressed. Thus, the overburden pressure above the destressed zone will be redistributed toward the front abutment and the two adjacent neighboring solid sections where the main gates, the intervening pillar and the adjacent un-mined solid sections are located (Majdi et al., 2012a). Therefore, to evaluate the amount of stress transferred to the adjacent access tunnels, gates and intervening pillars, the combined height of caving and fracturing zones (the height of destressed zone) should be estimated.

Nowadays, the main concerns of many underground coal mining researchers are to find the suitable approaches to evaluate the panel roof strata behavior during and after the panel extraction. There are common approaches for studying the progressive fracturing and caving of panel roof rock strata such as in-situ measurement and physical, empirical, numerical and analytical modeling (Xie et al., 2009). Although in-situ measurement and physical modeling provide a reliable and useful results but they are time consuming and expensive due to inherent complexity of the method implementation. Empirical methods are based on the data extracted from a specific case study with the particular characteristics and cannot be qualified for other cases with different mechanical or physical properties (Gao et al., 2014). Numerical
modeling is widely used in industry as a means of assessing cavity stability. However, this method requires a large number of input variables, which depending on available data, may need to be estimated or assumed (Suchowerska et al., 2012). Among the aforementioned methods, analytical modeling is a simple and inexpensive method that includes the useful and reliable results.

The height of destressed zone (HDZ), in this paper, is considered as the combined height of caving and fracturing zones above the longwall panel. Beyond this height the overburden pressure will be transferred toward the adjacent solid sections. Therefore, accurate determination of HDZ plays a vital rule in the amount of transferable loads to the front abutment, the adjacent access tunnels, the barrier pillars and the panel rib-sides. Accordingly, the height of destressed zone should be estimated in order to properly evaluate the amount of additional loads induced due to longwall mining. The observations and results of previous investigations showed that there exist many effective parameters in determination the height of disturbed zones above the longwall panel (Majdi et al., 2012a). However, none of the researchers has taken all of the possible effective parameters into account in their investigations. In some cases, only the effect of extracted coal seam thickness and rock property constants are investigated (NCB, 1975; Chuen, 1979; Zhou, 1991), whereas in other cases, only the effect of extracted coal seam thickness and bulking factor are studied (Peng and Chiang, 1984; Majdi et al., 2012a,b). In addition, there are some investigations in which only the effect of geometrical parameters are considered in analysis of the height of caved and fractured zones (Singh and Kendorski, 1981; Follington and Isaac, 1990).

Unlike the past investigations, in this research, the effects of both geometrical and geomechanical parameters of mined panel and roof rock strata have been simultaneously considered in determining the height of destressed zone. In addition, the strain energy consideration is taken into account in which the strain energy release can be the most important factor of the roof disturbance in longwall mining. Accordingly, a new time-independent analytical model is developed in the present study to determine the height of destressed zone (HDZ) above the mined panel based on the strain energy balance in longwall coal mining.

2. Background

Behavior of the panel roof strata and process of gradual destruction and upward movement have been investigated by many researchers. According to Eavenson (1923), the inner-burden shear failure during multiple seam extraction in longwall mining extends to the ground surface (Fig. 1). Dinsdale (1935) showed that the height of destressed zone (dome) has a direct relation with the depth of cover and the panel width and it has got the inverse relation with the horizontal reaction. Denkhaus (1964) believed that for a dome with sufficient cohesion, the maximum height of destressed zone is equal to 50% of the depth of cover above the excava-

![Fig. 1. Massive innerburden shear failure during multiple-seam mining (Eavenson, 1923).](image-url)
Singh (2009) showed that the maximum height of caving zone in openings. Zhimin et al. (2010) used the in-situ measurements of horizontal fractures located 12.9–149.4 m above the underground measurements in Donetsk Coal Basin, Ukraine. He showed that the horizontal fractures within the fractured zone based on the field measurement. Xie et al. (2009) utilized the physical and numerical modeling to determine the height of water-flow fractures. They showed that the height of fractured zone is equal to 14.33–17.71 and 16.04 times the mining height.

Monitoring study of the ground movement process associated with longwall coal mining was conducted by Kelly et al. (2002) as shown in Fig. 2. They believed that the failure extended to a height of about 120 m above the extracted coal seam. Palchik (2002) showed that the height of caved zone could reach 4.1–11.25 times the thickness of the underlying coal seam where overburden rocks are weak and porous. The fractured zones caused by longwall mining were further studied by Palchik (2003) using the vertical wells. He found that the maximum height of the zone of interconnected fractures and separate horizontal fractures may reach 19–41 and 53–92 times the extracted coal seam thickness, respectively. According to Rafiqul Islam et al. (2009), the upward fracture propagation would be about 22–37 m for single-slice (height 3.5 m) and 240 m for multi-slice (six slices) mining extraction. Xie et al. (2009) utilized the physical and numerical modeling to survey the roof rock strata in top coal caving face. Their modeling results showed that the stress arch above the face is 62 m high (11.5 times the mining height) and the height of the stress arch on the strike is 130 m (24.1 times the mining thickness). Singh and Singh (2009) showed that the maximum height of caving zone in the panel roof strata is equal to 15 times the mining height. Palchik (2010) investigated the apertures of mining-induced horizontal fractures within the fractured zone based on the field measurements in Donetsk Coal Basin, Ukraine. He showed that the horizontal fractures located 12.9–149.4 m above the underground openings. Zhimin et al. (2010) used the in-situ measurements and numerical modeling to determine the height of water-flow fractured zone during coal mining. They found that the height of fractured zone resulted from in-situ measurements and numerical modeling are equal to 14.33–17.71 and 16.04 times the mining thickness, respectively.

Shaojie et al. (2011) investigated the height of water flowing fractured zone in longwall coal mining by using the numerical modeling. They showed that the height of fractured zone is equal to 6.2 times the mining height. Zhang et al. (2011) investigations indicated that the height of the caved zone varies from 5 to 6 times the mining height, while the height of the fractured zone is about 10–11 times the mining height. Karekal et al. (2011) utilized a mesh-free continuum numerical method for simulation of the rock caving and fracturing processes. They concluded that the height of disturbed zone is equal to 4.5 times the mining height. Hai Feng et al. (2011) employed the field measurements as well as numerical, physical and empirical modeling to estimate the maximum height of caving and fracturing zones in overburden strata of a coal seam. The results of their investigations showed that the combined height of caving and fracturing zones is equal to 51.7 m, 51.98 m, 45.5 m and 42.8 m from the field measurement, numerical, physical and empirical modeling, respectively. The in-situ measurements conducted by Miao et al. (2011) showed that the height of caving and fracturing zones reach to 4.03 and 32.64 times the height of excavation, respectively.

Wenbing et al. (2012) studied the fractured zone height above the longwall panel and its effects on the overburden aquifers based on the field observations, proved that the height of the fractured zone is in the range of 72.7–85.3 m. Majdi et al. (2012a,b) developed five theoretical models to predict the height of destressed zone above the mined panel based on the coal seam thickness and expansion factor. Their findings indicated that the height of destressed zone for the short term and long term condition are equal to 6.5–24 and 11.5–46.5 times the extracted coal seam thickness, respectively. Shabanimashcool and Charlie (2012) used the numerical modeling to simulate the caving and fracturing process of the panel roof strata. They showed that the height of caving and fracturing zones from the panel floor are equal to 16 m and 110 m, respectively.

Zhang et al. (2013) investigated the overburden fracture evolution laws in mining of very thick coal seam under water-rich roof by using the numerical modeling. Their finding proved that the height of fractured zone in the overburden strata ranges from 18.66 to 47.66 times the extraction seam height. Shun et al. (2013) employed the in-situ measurements to investigate the time-domain characteristics of overlying strata failure under condition of longwall ascending mining. They found that the height of caving and fracturing zone reach to 2.05 and 13.37 times the extraction coal seam height, respectively. Gao et al. (2014) developed a numerical approach to simulate the progressive caving of strata above a longwall coal mining panel. Their results showed that fractures extend approximately 40 m into the roof.

According to the above referred works, there are some empirical and analytical equations to predict the height of caved/fractured/destressed zone. These equations are presented in Table 1. Also, the results of existing methods to calculate the height of disturbance zones in terms of the coefficient of extracted coal seam thickness (Hc) are shown in Table 2.

3. Energy considerations in longwall mining

In mining operations, the energy transfer takes place in the rock mass during a transition from one equilibrium state to another, which is assumed to be an elastic continuum. This process leads to energy release and causes the roof rock strata to be fractured, caved and destressed. Energy balance and relationships between the energy components in mining have been firstly investigated by Cook (1967) and later refined by Salamon (1974, 1984). Salamon (1984) presented a comprehensive investigation of each of the energy components and derived a set of basic mathematical definitions for each term. He showed that these definitions can be modified and used for any mining structure in an elastic condition.

In the longwall mining, as the extraction of coal seam from the panel is started, the energy balance within the system enclosing
the mine openings and the surrounding rocks is changed. In this research, state I is considered as the primary status before the coal seam extraction and state II is considered as the end of coal seam extraction inside a panel (Fig. 3). It is postulated that both in state I and in state II, the forces present in the system are in equilibrium.

In Fig. 3, \( L_m \) is the panel width, \( H \) is the depth of cover, \( H_d \) is the extracted coal seam thickness, \( H_c \) is the height of destressed zone, \( D \) is the distance of horizontal line across the centre of gravity of the panel to the ground surface, \( V_m \) is the volume of materials that should be extracted for transition from state I to state II (the volume of mined panel), \( S_m \) is the surface of mined panel. \( V \) is the volume of rock masses above a longwall panel that are affected and destressed due to mining (the volume of destressed zone above a panel) and \( S \) is the surface of destressed zone above a panel.

If the longwall panel and its surrounding rock mass are considered as a system, then the equilibrium between its energy components should be maintained. In the longwall mining, no permanent supports are used during the panel extraction. Therefore, due to the extraction of coal seams within a considerable panel width and after advancing the hydraulic jacks, the immediate roof of the mined panel is unsupported and hence is allowed to collapse and cave in some distance behind the hydraulic jacks or in the goaf area. This leads to release of the stored strain energy in the mined rock (coal) and transfer it to the panel roof strata. The released energy increases the strain of roof rock strata and causes the fracturing, caving and destressing of these layers. This process is continued until the full extraction of a longwall panel. At the end of panel extraction and after compression of caved materials, a defined zone with the height of \( H_d \) is formed above the longwall panel which is called the “destressed zone” (Fig. 3b). Although there is no general agreement regarding the exact shape of this zone, most studies show that it is in the form of a dome, a horse saddle, or a flat (Fig. 4). Since the main objective of this research is to determine the height of destressed zone (\( H_d \)), hence, shape of the upper part of destressed zone not affects on determination of its height. Considering the above descriptions, it can be concluded that the total stored strain energy in the mined rock (coal) is released and consumed in fracturing, caving and destressing the panel roof rock strata. Therefore, stored strain energy in the mined rock (coal) should be equal to the stored strain energy in caved materials within the destressed zone:

\[
U_m = U_d
\]

where \( U_m \) is the stored strain energy in the mined rock (coal) and \( U_d \) is the stored strain energy in the caved materials within the destressed zone.

4. Calculation of the strain energy components

With regard to the complex conditions of longwall coal mining, it is required to consider the general assumptions for simplification of the problem in calculating the aforementioned components of stored strain energy. For this purpose, the following fundamental assumptions are made in the modeling:

1. The horizontal components of displacements are taken to zero in state I.
2. Ground surface is the horizontal plane and there is no tectonic force.
3. Surrounding rock is homogenous and isotropic.
4. Vertical and horizontal normal stresses are the principle stresses.
5. Effect of body forces is ignored in modeling.

In this study, the two-dimensional analysis of strain energy components is introduced. Generally, discussions of a problem are algebraically simpler in two dimensions than in three. In most instances, no generality is lost by considering the two-dimensional case. Furthermore, many problems in rock mechanics are essentially two-dimensional, in the sense that the stresses do not vary

### Table 2
Results of the available methods to predict the height of caved/fractured/destressed zone in terms of the coefficient of extracted coal seam thickness.

<table>
<thead>
<tr>
<th>Height of caved zone ($h_h$)</th>
<th>Height of fractured or destressed zone ($h_f$)</th>
<th>Descriptions</th>
<th>Method of appraisal</th>
<th>Reference</th>
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<td>–</td>
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<td>–</td>
<td>Empirical</td>
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<td>–</td>
<td>13–31</td>
<td>For strong rock</td>
<td>Empirical</td>
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<td>–</td>
<td>–</td>
<td>Empirical</td>
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<tr>
<td>–</td>
<td>12</td>
<td>–</td>
<td>Empirical</td>
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<td>In-situ</td>
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<td>–</td>
<td>40</td>
<td>–</td>
<td>In-situ</td>
<td>Kelly et al. (2002)</td>
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<td>4.1–11.25</td>
<td>–</td>
<td>–</td>
<td>In-situ</td>
<td>Palchik (2002)</td>
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<td>–</td>
<td>6.28–10.57</td>
<td>–</td>
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<td>–</td>
<td>11.42</td>
<td>–</td>
<td>In-situ</td>
<td>Zhang et al. (2011)</td>
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<td>–</td>
<td>11.5</td>
<td>–</td>
<td>In-situ</td>
<td>Xie et al. (2009)</td>
</tr>
<tr>
<td>15</td>
<td>–</td>
<td>–</td>
<td>Numerical</td>
<td>Singh and Singh (2009)</td>
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<tr>
<td>–</td>
<td>14.3–17.9</td>
<td>–</td>
<td>In-situ</td>
<td>Zhimin et al. (2010)</td>
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<tr>
<td>–</td>
<td>16.04</td>
<td>–</td>
<td>Numerical</td>
<td>Zhang et al. (2011)</td>
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<tr>
<td>5–6</td>
<td>10–11</td>
<td>–</td>
<td>In-situ</td>
<td>Karekal et al. (2011)</td>
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<td>–</td>
<td>4.5</td>
<td>–</td>
<td>Analytical</td>
<td>Haifeng et al. (2011)</td>
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<tr>
<td>3.765</td>
<td>9.23</td>
<td>–</td>
<td>Numerical</td>
<td>Shaojie et al. (2011)</td>
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<tr>
<td>3.88</td>
<td>8.13</td>
<td>–</td>
<td>In-situ</td>
<td>Miao et al. (2011)</td>
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<tr>
<td>2.1</td>
<td>8.6</td>
<td>–</td>
<td>Empirical</td>
<td>Majdi et al. (2012a)</td>
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<td>–</td>
<td>5.8–6.7</td>
<td>–</td>
<td>Numerical</td>
<td>Majdi et al. (2012a)</td>
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<td>4.03</td>
<td>32.6</td>
<td>–</td>
<td>In-situ</td>
<td>Wenbing et al. (2012)</td>
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<td>–</td>
<td>40</td>
<td>Triangular model</td>
<td>Analytical</td>
<td>Shabanimashico and Charlie (2012)</td>
</tr>
<tr>
<td>–</td>
<td>30</td>
<td>Parabolic model</td>
<td>Analytical</td>
<td>Shun et al. (2013)</td>
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<tr>
<td>–</td>
<td>25.4</td>
<td>Elliptical mode</td>
<td>Analytical</td>
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<tr>
<td>–</td>
<td>22</td>
<td>Geometric model</td>
<td>Analytical</td>
<td>Gao et al. (2014)</td>
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<tr>
<td>–</td>
<td>29.8–35</td>
<td>–</td>
<td>In-situ</td>
<td>Shabanimashico and Charlie (2012)</td>
</tr>
<tr>
<td>4</td>
<td>27.5</td>
<td>–</td>
<td>Numerical</td>
<td>Shun et al. (2013)</td>
</tr>
<tr>
<td>2.05</td>
<td>13.3</td>
<td>–</td>
<td>In-situ</td>
<td>Zhang et al. (2013)</td>
</tr>
<tr>
<td>–</td>
<td>18.6–47.6</td>
<td>–</td>
<td>Numerical</td>
<td>Gao et al. (2014)</td>
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</tbody>
</table>

![Fig. 3. Longwall mining configuration, (a) state I: before panel extraction (b) state II: after final extraction of longwall panel.](image-url)
along one Cartesian coordinate. The most common examples of such problems are stresses around boreholes, around long tunnels and longwall panels. Many other problems are idealized as being two-dimensional so as to take advantage of the relative ease of solving two-dimensional elasticity problems as compared to three-dimensional problems (Jaeger and Cook, 1979).

Actually, a two-dimensional model can be used for the analysis of stresses and displacements in the rock surrounding a tunnel, shaft or borehole, where the length of the opening is much larger than its cross-sectional dimensions. The stresses and displacements in a plane, normal to the axis of the opening, are not influenced by the ends of the opening, provided that these ends are far enough away. However, most aspects of the theory of stress (and strain) can be developed within a two-dimensional context, and extensions to three dimensions are in most cases straightforward. Indeed, the theory of stresses in three dimensions is in general a straightforward extension of two-dimensional theory. So, it is important to study the two-dimensional analysis of a problem in the first stage. Since the stresses do not vary along the panel length, thus two-dimensional modeling is sufficient for stress analysis in this research. Therefore, two-dimensional analysis would be sufficient for accurate determination of strain energy components in longwall mining in order to estimate the height of destressed zone (HDZ) in the roof rock strata above a longwall panel.

4.1. Calculation of $Um$

To calculate the stored strain energy in the extracted coal layer ($Um$), the following equation is used (Salamon, 1984):

$$Um = \frac{1}{2} \int_{Sm} T^{(p)} u^{(p)} dV - \int_{V_m} X u^{(p)} dV$$  \hspace{1cm} (2)

where $T^{(p)}$ is the stress or traction vector acting on a surface in state I, $u^{(p)}$ is the component of the displacement vector in state I, $X$ is the body force per unit volume in the rock; and $Sm$ and $V_m$ are the surface and volume of the extracted coal seam, respectively. It can be noted that the notations superscript $p$ stands for “primitive” or state I. According to the assumptions made in Section 4.2, the effect of body forces is ignored in modeling ($X = 0$). Therefore, Eq. (2) is modified as:

$$Um = \frac{1}{2} \int_{Sm} T^{(p)} u^{(p)} dS$$  \hspace{1cm} (3)

Based on the considered assumptions and the Jaeger and Cook (1979) investigations, the principal stresses in state I can be calculated using the following equations:

$$\sigma_{xx}^{(p)} = \sigma_{yy}^{(p)} = \gamma H$$  \hspace{1cm} (4)

where $\sigma_{xx}$, $\sigma_{yy}$ and $\sigma_{zz}$ are the principle stresses in directions of $x$, $y$ and $z$ respectively, $\gamma$ is the rock mass unit weight and $v$ is the rock mass Poisson ratio.

The components of strain and displacement tensor can be calculated by

$$e_y = \frac{1}{2} \left( \frac{\partial u_y}{\partial x} + \frac{\partial u_x}{\partial y} \right)$$  \hspace{1cm} (5)

The only non-zero strain and displacement components follow from Eqs. (4) and (5) and Hooke’s law as follows:

$$e_y^{(p)} = \frac{(1+v)(1-2v)\gamma H}{(1-v)E}, \quad u_y^{(p)} = \frac{(1+v)(1-2v)\gamma H^2}{2(1-v)E}$$  \hspace{1cm} (6)

where $E$ is the rock mass elastic modulus, $e_y^{(p)}$ and $u_y^{(p)}$ are the vertical strain and displacement components in state I, respectively.

Substitution the first part of Eq. (4) and the second part of Eqs. (6) into (3) yield

$$Um = \int_{Sm} \frac{(1+v)(1-2v)\gamma H^3}{2(1-v)E} dS = \frac{(1+v)(1-2v)\gamma H^2}{2(1-v)E} \int_{Sm} H^2 dA$$  \hspace{1cm} (7)

In Eq. (7), the term of $I = \int_{Sm} H^2 dA$ is the moment of inertia of surface $Sm$ with respect to the plane of ground surface (see Fig. 3b).

Based on the parallel-axis theorem in statics, one can write:

$$I = I_0 + A_m D^2$$  \hspace{1cm} (8)

where $I_0$ is the moment of inertia of the cross section of mined panel with respect to a horizontal line across its centre of gravity (point $O$ in Fig. 3b), $A_m$ is the cross section of the mined panel and; $H$ and $D$ are defined previously. With regard to the rectangular cross section of the extracted panel (see Figs. 3b and 4), $A_m$ and $I_0$ are calculated as follows:

$$A_m = L_m \times h_s$$ \hspace{1cm} (9)

$$I_0 = \frac{L_m h_s^3}{12}$$ \hspace{1cm} (10)

Now, the substitution of Eqs. (10) and (9) into (8) leads to

$$I = A_m \frac{h_s^2}{12} + (A_m D^2)$$ \hspace{1cm} (11)

Then, when Eq. (11) substituted into Eq. (7), results in

$$Um = \frac{(1+v)(1-2v)\gamma A_m h_s^2}{2(1-v)E} \left[ \frac{h_s^2}{3} + H^2 + H h_s \right]$$ \hspace{1cm} (12)

According to Fig. 3b:

$$D = H + \frac{h_s}{2}$$ \hspace{1cm} (13)

Finally, when Eq. (13) substituted into the Eq. (12) and rearranged it, the ultimate equation of $Um$ in longwall coal mining can be obtained as follows:

$$Um = \frac{(1+v)(1-2v)\gamma A_m \sigma_v}{2(1-v)E} \left[ \frac{h_s^2}{3} + H^2 + H h_s \right]$$ \hspace{1cm} (14)

4.2. Calculation of $U_d$

In general, total stored strain energy in the rock mass within the destressed zone ($U_d$) is composed of the elastic strain energy ($U_e$) and the viscoplastic strain energy ($U_p$). In short time condition, the mechanical properties of caved materials such as elastic modulus and coefficient of viscosity is constant and do not change under pressure of the roof. Therefore, the viscoplastic strain energy
of caved materials can be ignored in this condition \((U_d = 0)\). Consequently, the total stored strain energy of the caved materials system in short time (time-independent) condition is equal to the stored elastic strain energy of the system \((U_d = U_d)\).

If the destressed zone above a longwall panel is considered as a separate system (surface with height of \(H_d\) in Figs. 3b and 4), its stored strain energy can be calculated using the following equation:

\[
U_d = \frac{1}{2} \int \sigma dV = \frac{1}{2} \int_0^b \sigma_v A dh
\]  

(15)

where \(\sigma\) is the uniaxial applied stress on the caved materials which is here considered as the uniaxial compressive strength of caved materials \((\sigma_v)\), \(\nu\) is the strain occurring under the applied stress, \(h\) is the height of the caved materials within the destressed zone and \(A\) is the unit surface of the system. In this research, \(A_d\) is considered as the unit surface of destressed zone which is equal to panel width multiplied by the unit length of goaf area \((A_d = W \times 1\text{m})\). After the caving process is completed and the caved materials are through compressed, the total height of caved materials is equivalent to the height of destressed zone \((h = H_d)\).

By applying the above simplifications and integration of Eq. (15), we can write:

\[
U_d = \frac{1}{2} \sigma_v A_{d} H_d
\]  

(16)

To describe the stress–strain behavior of the caved materials, the following equation can be used (Salamon, 1990):

\[
\sigma_c = \frac{E \varepsilon}{1 - \nu / \nu_m}
\]  

(17)

where \(\varepsilon_m\) is the maximum possible strain of the caved materials, \(E\) is the initial elastic modulus which is considered same elastic modulus of rock mass \((E)\) due to time independent condition of this study and other variables are defined previously.

From Eq. (17), one can write:

\[
\varepsilon = \frac{\sigma_c}{E + \frac{\nu_c}{\nu_m}}
\]  

(18)

According to Yavuz (2004), the maximum possible strain \((\varepsilon_m)\) merely depends on the bulking factor and it can be determined as follows:

\[
\varepsilon_m = \frac{b - 1}{b}
\]  

(19)

where \(b\) is the bulking factor of caved materials or the ratio between the volumes of rock after caving to its initial volume \((l)\).

Now, the substitution of Eqs. (19) into (18) leads to

\[
\varepsilon = \frac{\sigma_c}{E + \frac{\nu_c}{\nu_m} \frac{b - 1}{b}}
\]  

(20)

Finally, when Eqs. (20) substituted into (16), the stored strain energy in the rock mass within the destressed zone \((U_d)\) in time independent condition can be obtained as follows:

\[
U_d = \frac{\sigma_c^2 A_{d} H_d}{2 \left[ E + \frac{\nu_c}{\nu_m} \frac{b - 1}{b} \right]}
\]  

(21)

5. Time-independent model to calculate the height of destressed zone

As mentioned previously, the stored strain energy in the mined rock (coal) releases during the longwall panel extraction. This released energy absorbs in the panel roof strata and causes the fracturing, caving and destressing of these layers. Finally, this process leads to form a destressed zone above the longwall panel. Accordingly, it can be concluded that the stored strain energy in the mined rock (coal) is equal to the stored strain energy in the caved materials within the destressed zone, as shown in Eq. (1). Therefore, the substitution of Eqs. (21) and (14) into (1) leads to calculate the height of destressed zone \((H_d)\) in time-independent condition as follows:

\[
H_d = \frac{(1 + \nu)(1 - 2\nu) \frac{A_m \sigma_v}{E} \left[ \frac{h_r^2}{3} + H^2 + H_{s} \right]}{E + \frac{b \sigma_v}{(b - 1)}}
\]  

(22)

In the above equation, all quantities of force and distance are in Newton and Meter, respectively. However, \(\sigma_v\) is the initial stress due to overburden \((N/m^2)\) which is equal to \(\gamma H\), \(H\) is the depth of cover \((m)\), \(\gamma\) is the panel roof rock’s average unit weight \((N/m^3)\), \(h\) is the extracted coal seam thickness \((m)\), \(\sigma_v\) is the average uniaxial compressive strength of caved materials \((N/m^2)\), \(A_m\) is the cross section of extracted panel \((m^2)\), \(A_s\) is the unit surface of destressed zone \((m^2)\), \(v\) is the panel roof rock’s average Poisson ratio, \(E\) is the panel roof rock’s average elastic modulus \((N/m^2)\) and \(b\) is the bulking factor of caved materials \((l)\). It should be noted that the panel width \((L_m)\) exists in the numerator and denominator (incorporated in \(A_m\) and \(A_s\)). Therefore, although it can be omitted from the equation but was kept in order to maintain the dimensional balance.

5.1. Verification of the model

In order to verify the proposed model, the obtained results are compared with the results of in-situ measurements reported by other researchers. These measurements and their corresponding results along with the results of Energy model are shown in Table 3. As it can be seen from this table, the results of suggested model are in a close agreement with the results of in-situ measurements. However, the obtained results are slightly lower than the results of the in-situ measurements in all cases. This is due to short time condition considered in this study where the caved materials are not compressed completely. In short time condition, it is assumed that the caved materials are not compressed completely and then, only the elastic energy of caved materials is considered in modeling. Therefore, the results of proposed model can be reasonable.

5.2. Sensitivity analysis

To evaluate the effect of incorporated parameters on the HDZ, the actual dataset 3 in Table 3 is used as the basis data. Variations of the height of destressed zone versus the extracted coal seam thickness and the average bulking factor of caved materials are illustrated in Figs. 5 and 6, respectively. In the above figures, variations of the extracted coal seam thickness and the bulking factor are considered between 0–5 m and 1.05–1.8 (expansion factor 5–80%), respectively. The rest of the involved parameters are kept constant according to the actual data. In addition, variations of the height of destressed zone versus the extracted coal seam thickness for different values of depth of cover, unit weight, elastic modulus, Poisson ratio and uniaxial compressive strength are shown in Figs. 7–11, respectively. In these figures, one of the parameters (chart target parameter) is changed and the rest are kept constant.

Figs 5 and 6 show that the height of destressed zone has direct and inverse relations with extracted coal seam thickness and bulking factor of the caved materials, respectively. In Fig. 6, the values of 1.05 and 1.8 are considered as the lower and upper limits of the bulking factor, respectively. However, it can be concluded from this figure that variations of the height of destressed zone for the bul-
king factors less than 1.1 (expansion factor less than 10%) is rela-

tively high compared to the bulking factors more than 1.1. This

result is in agreement with the results reported by Majdi et al.

(2012a) in which the average expansion factor of 5% and 10% are

considered to represent the long-term and short-term conditions

of the caved materials, respectively.

Fig. 7 shows that the height of destressed zone is in direct relation

with depth of cover. It can be seen from Fig. 8 that the height of destressed zone has a direct relation with panel roof rock’s average unit weight.

Fig. 9 indicates that the higher average elastic modulus of panel roof rock strata leads to the lower height of destressed zone. It can be concluded

from Fig. 10 that the height of destressed zone has a negative relation with panel roof rock’s average Poisson ratio. Fig. 11 represents that the height of destressed zone has an inverse relation with uni-

axial compressive strength of the caved materials.

As the extracted coal seam thickness changes from 0.5 m to 5 m (Fig. 5), the ratio between the height of destressed zone and the extracted coal seam thickness \( \frac{H_d}{h_s} \) varies from 6.5 to 6.6. Similarly, it can be concluded from Figs. 6–11 that the ratio of \( \frac{H_d}{h_s} \) varies from 6.5 to 8.2, 2.1 to 57.8, 2.02 to 8.1, 6.3 to 8.2, 3.5 to 7.3 and 6.5 to 17.9, respectively. According to the above analysis, Energy model represents the height of destressed zone in the range of 2.02 to 57.8 times the extracted coal seam thickness (Table 4).

Table 3

Comparison of the results of suggested model with the in-situ measurements (bulking factor is assumed 1.5).

<table>
<thead>
<tr>
<th>Dataset</th>
<th>Reference</th>
<th>( H ) (m)</th>
<th>( L_w ) (m)</th>
<th>( \gamma ) (N/m(^3))</th>
<th>( E ) (GPa)</th>
<th>( v )</th>
<th>( \sigma_t ) (MPa)</th>
<th>( b )</th>
<th>( h_s ) (m)</th>
<th>( H_d ) (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Haifeng et al. (2011)</td>
<td>285</td>
<td>223</td>
<td>24,660</td>
<td>2.32</td>
<td>0.2775</td>
<td>31.3</td>
<td>1.5</td>
<td>4</td>
<td>51.7</td>
</tr>
<tr>
<td>2</td>
<td>Zhimin et al. (2010)</td>
<td>253</td>
<td>200</td>
<td>26,627</td>
<td>25.2</td>
<td>0.174</td>
<td>12.07</td>
<td>1.5</td>
<td>3.9</td>
<td>64.5</td>
</tr>
<tr>
<td>3</td>
<td>Rafiqul Islam et al. (2009)</td>
<td>290</td>
<td>120</td>
<td>27,000</td>
<td>3.5</td>
<td>0.22</td>
<td>50</td>
<td>1.5</td>
<td>3.5</td>
<td>29.5</td>
</tr>
</tbody>
</table>

Fig. 5. Variations of the height of destressed zone versus the extracted coal seam thickness.

Fig. 6. Variations of the height of destressed zone versus the average bulking factor of caved materials.

Fig. 7. Relationship between the height of destressed zone and the extracted coal seam thickness for the different values of depth of cover.

Fig. 8. Relationship between the height of destressed zone and the extracted coal seam thickness for the different values of panel roof rock’s average unit weight.
6. Comparative analysis

In this section, obtained results from the proposed Energy model are compared with those of the comparable formulas given by Peng and Chiang (1984) and Majdi et al. (2012a). For this purpose, variations of the height of destressed zone versus the extracted coal seam thickness for the lower (1.05) and the upper (1.8) limits of the caved materials bulking factor are shown in Figs. 12 and 13, respectively. Also, Figs. 14 and 15 present the variations of the height of destressed zone versus the caved materials bulking factor for the lower (0.5 m) and the upper (5 m) limits of the extracted coal seam thickness, respectively. The available formulas to estimate the height of caved/fractured/destressed zone (Peng and Chiang; 1984, Majdi et al., 2012a) contain only two effective parameters namely extracted coal seam thickness and bulking (expansion) factor of the caved materials. Accordingly, all other incorporated parameters in the Energy model except these two mentioned parameters are kept constant in this comparison. Therefore, the extracted coal seam thickness and the average bulking factor of caved materials are changed and the rest are kept constant according to actual dataset 3 showed in Table 3.

As it can be seen from Figs. 12 and 13, Triangular, Parabolic, Elliptical models as well as Peng and Chiang’s model illustrate a linear relation between HDZ and extracted coal seam thickness. Therefore, in these models, the ratio between the height of destressed zone and the extracted coal seam thickness \( \frac{H_{d}}{h_{s}} \) is always constant. On the contrary, Arithmetic and Energy models represent a nonlinear relation between the HDZ and the extracted coal seam thickness. Hence, the ratio of \( H_{d}/h_{s} \) is not constant in the case of Arithmetic and Energy models; rather it is a lin-
ear function of both the bulking factor and the extracted coal seam thickness. As shown in Figs. 12–15, the average bulking factor of the panel roof rock strata, in all, is considered in the range of 1.05–1.8 for the extracted coal seam thickness ranging from 0.5 m to 5 m. According to Energy model, the minimum height of de-stressed zone is equal to $6.44h_s$ for a 0.5 m of extracted coal seam thickness and a 1.8 of bulking factor. On the other hand, the maximum height of de-stressed zone is equal to $8.24h_s$ for a 5 m of extracted coal seam thickness and a bulking factor of 1.05.

In the case of the Arithmetic model, however, this ratio varies from 1.81 to 51.5 for the different values of bulking factor and extracted coal seam thickness. As it is observed, the variations of $H_d/h_s$ in the Energy model are considerably lower than those of the Arithmetic model.

According to Fig. 12, Arithmetic and Energy models show the same value of HDZ for extracted coal seam thickness of 0.65 m. Greater than this value of extracted coal seam thickness, Energy model provides the lowest values of HDZ compared to the other models. By increasing the bulking factor to its upper limit (Fig. 13), the HDZ decreases in all models but this process occurs slowly in the Energy model. This demonstrates the high sensitivity of previous models to the bulking factor, especially in the lower values of bulking factor. As can be seen from Figs. 14 and 15, variations of the HDZ in previous models for the bulking factor values between 1 and 1.1 are very high compared to the other intervals. As an example, when the bulking factor decreases from 1.1 to 1.01 in Peng and Chiang model, the HDZ will be 10 times, whereas by reducing it from 1.2 to 1.1, this height will be 2 times. This high sensitivity in above mentioned interval (bulking factor of 1.01–1.1) is true about the Majdi et al. (2012a) models, too. Unlike these models, variations of the height of de-stressed zone for the different values of bulking factor are regular in Energy model and follow a certain order, as it is clearly observed from Fig. 6.

Fig. 14 shows that the Arithmetic model predicts the lower values of the height of de-stressed zone than the other models for the bulking factor values less than 1.5. In addition, for the bulking factor greater than 1.302, Energy model predict the highest values of HDZ compared to the other models. By increasing the extracted coal seam thickness to its upper limit (Fig. 15), height of de-stressed zone increases exponentially in Arithmetic model and reaches to the maximum value among the models. Generally, by increasing the bulking factor, height of de-stressed zone decreases with high gradient in all models except the Energy model. This shows that the effect of bulking factor in Energy model is lower than the other models. In addition, it can be concluded that the previous models show a very high sensitivity to the bulking factor. On the contrary, the Energy model shows a normal sensitivity to the bulking factor parameter in predicting of HDZ. However, the main result derived from Figs 12–15 is that the highest height of de-stressed zone is attained when the bulking factor is the least. It is obvious that the least bulking factor is attainable with time as the caved materials are compressed completely.

The results of four series of information collected from the literature (indicated in Table 2) in addition to the Energy model results for the calculation of HDZ in terms of the coefficient of extracted coal seam thickness ($h_s$) summaries in Table 4. It can be concluded from this comparison that the lower limit of Energy model results is in a close agreement with the lower limit of empirical model and in-situ measurements results. In contrast, its upper limit is in a close agreement with the upper limit of analytical and numerical models. At the same time, the upper limit of Energy model results is lower than the upper limit of in-situ measurements. It seems that this difference is due to considering the short-time or time-independent condition in the Energy model. It is expected that this difference will be reduced in the long-term condition.
7. Discussion and results

In general, the results of the proposed model in this paper are in a close agreement with the in-situ measurements and with those proposed analytically, empirically and numerically by other researchers as well. Furthermore, the most important advantage of the proposed Energy model compared to the other analytical and empirical models is that it incorporates further effective parameters in predicting of the HDZ. In the previous analytical models only two parameters namely extracted coal seam thickness and average bulking (expansion) factor of the caved materials are utilized to calculate the height of caving/fracturing zone (Peng and Chiang, 1984, Majdi et al., 2012a). In addition to the above mentioned parameters, the proposed Energy model incorporates five other effective parameters including depth of cover, unit weight, elastic modulus, Poisson ratio and uniaxial compressive strength of caved materials. Discussions on the above geomechanics related factors are given in Section 5.2. Furthermore, one of the main advantages of the proposed model compared to the previous analytical methods is that the influences of initial vertical and horizontal stresses are considered in analysis, as shown in Eq. (4). Indeed, the influence of the initial stress field on the height of destressed zone is considered in this research. It can be concluded from Eq. (22) that the initial vertical stress (\(\sigma_v\)) has a direct influence on the height of destressed zone. The above mentioned superiors help the energy model to determine the HDZ with higher accuracy compared to the previous models.

As the proposed model incorporates the most effective parameters and its results are in agreement with the results of comparable models, it can be successfully used to determine the height of destressed zone. Beyond this height the overburden pressure will be transferred to the front abutment and the two adjacent neighboring solid sections where the gate roads (access tunnels serving the coal face), the intervening pillars and the adjacent un-mined solid sections are located. Hence, the complete overburden pressure will not be inserted on the caved materials located some distance behind the powered roof supports in the gob-side. In fact, the difference between the longwall depth (\(H\)) and the height of destressed zone (\(H_d\)) must be considered for the calculation of the stress transfer toward the adjacent access tunnels and the intervening pillars.

As previously mentioned, this study is the time-independent analysis of the height of destressed zone. Time-independent condition of the height of destressed zone is easier to model than the time-dependent. Under the pressure of roof in long-term condition, the mechanical properties of caved materials can change, so elastic modulus and coefficient of viscosity of caved materials also change over time. Indeed, the viscoplastic strain energy in the time-dependent condition cannot be ignored in calculations and required considering the rheological properties of the caved materials in modeling. This makes it difficult to model and required to measure of material coefficients related to temperature. In contrast, the above mentioned strain energy component can be ignored in time-independent modeling, as discussed in Section 4.2. However, despite of the simplicity, the time-independent model yield very handy and reliable result that would be applicable to determine the maximum mining-induced stress as briefly discussed below.

It is obvious that the goaf materials are steadily compressed during the time due to bulking characteristics of it. This process causes the further caving and fracturing of the panel roof rock strata. Accordingly, the authors strongly believe that the long-term height of the destressed zone could be much more than those considered for the short-term condition. Therefore, the difference between the longwall depth (\(H\)) and the height of destressed zone (\(H_d\)) in time-independent model is greater than that of time-dependent model. Accordingly, it can be concluded that the transferred stress toward the roadways and adjacent pillars in time-independent model is larger than that of time-dependent model. As a result, the time-independent analysis of the height of destressed zone is important to determine the maximum transferred stress toward the surrounding gateroads and intervening pillars as well as to safe design of these structures. On the other hand, time-dependent model will be important to calculate the maximum surface subsidence due to longwall mining which is out of the scope of this paper.

8. Conclusion

In this paper, a new time-independent analytical model was proposed to predict the height of destressed zone (HDZ) above the mined panel. The new model is based on the strain energy balance in longwall coal mining. To verify the validity of the proposed model, the obtained results were analyzed and compared with the results of in-situ measurements. Also, a sensitivity analysis of the model was conducted in order to evaluate the effect of involved parameters on the HDZ. Furthermore, the results of proposed model were further compared with the results of available analytical, empirical and numerical models.

The results of sensitivity analysis showed that depth of cover, extracted coal seam thickness and panel roof rock’s average unit weight have direct influence on the HDZ. On the contrary, panel roof rock’s average elastic modulus and Poisson ratio along with average uniaxial compressive strength and bulking factor of the caved materials have negative influence on the HDZ. Also, the comparative analysis proved that the lower limit of Energy model is in a close agreement with the lower limit of all comparable methods. On the other hand, the upper limit of Energy model is close to the upper limit of analytical and numerical models but it is lower than the upper limit of empirical model and in-situ measurements. However, it can be concluded that the results of proposed model are in a close agreement with the results of available methods for HDZ prediction.

The main advantage of the present analytical model compared to previous analytical and empirical models is that it incorporates the effects of both geometric parameters and roof rock strata properties in predicting of the HDZ. Also, the further influencing parameters are considered in HDZ determination which helps accuracy of the proposed model and thus, the obtained results will be closer to reality. In addition, the strain energy consideration is taken into account which is the main important factor of the roof disturbance in longwall mining.

Since beyond the calculated HDZ the overburden weight will be transferred to the gateroads, the intervening pillars and the adjacent un-mined section, the results of this study can be used to calculate the induced stress due to longwall mining. In fact, the difference between the longwall depth and the calculated height of destressed zone can be considered to determine the transferred loads toward the front abutment and the two adjacent rib-sides.

References


Eavenson, H., 1923. Mining an upper bituminous seam after a lower seam has been extracted. Trans. SME, 69, 398–405.


