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Abstract	The underground mining method is the activity that leads to the surface impacts caused by subsidence movements. The mining Vietnam industry is developing, and step by step, applying advanced technologies to practical production in order to improve mining efficiency. Because of safety issues, optimal exploitation of mineral reserves, and reduction of losses are always the top priorities in mining companies of Vietnam. The paper assesses the ability to utilize the sources of ash and slag from the thermal power plant as backfilling material for underground mining activities in the Mao Khe coal mine, Vietnam. This technological solution helps to release a large amount of ash and slag discharged from the plants and minimizes negative impacts on the environment. Besides, the use of backfill methods also increases the stability of the safety bank and decrease coal losses during the exploitation process in underground coal mines in Vietnam. In this study, the authors performed theoretical and experimental researches with the hydraulic transport method using the materials with various ratios of water and ash in the field. The results were recorded and statistically analyzed to select the most suitable materials that are applicable at the southern wing area of The Mao Khe coal mine.

Keywords	Ash and slag - Backfilling - Safety pillar - Natural resource recovery - Loss reduction

Recycling Ash and Slag of the Thermal Power Plant to Replace Protective Pillars in Mao Khe Coal Mine, Vietnam

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Abstract. The underground mining method is the activity that leads to the surface impacts caused by subsidence movements. The mining Vietnam industry is developing, and step by step, applying advanced technologies to practical production in order to improve mining efficiency. Because of safety issues, optimal exploitation of mineral reserves, and reduction of losses are always the top priorities in mining companies of Vietnam. The paper assesses the ability to utilize the sources of ash and slag from the thermal power plant as backfilling material for underground mining activities in the Mao Khe coal mine, Vietnam. This technological solution helps to release a large amount of ash and slag discharged from the plants and minimizes negative impacts on the environment. Besides, the use of backfill methods also increases the stability of the safety bank and decrease coal losses during the exploitation process in underground coal mines in Vietnam.

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1 Introduction

Backfilling in underground coal mines in the world has been practiced for more than 100 years, and evidence anticipates the application of mine fill technology at an increasing rate during this decade. The evolution of backfill technology is closely related to the establishment of new mining methods. Hydraulically transported tailings and alluvial fills were introduced in the 1950s, thus permitting the adoption of cut and fill mining where the backfill was used as a working floor. In the early 1960s cemented hydraulic

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backfill was introduced, followed by the adoption of undercut and vertical retreat mining methods. The 1990s has been considered by many as the decade of high-density tailings fills and paste fills, during which several mines have successfully introduced paste fill into their operations [1]. Paste fill, relatively new technology in mining, is gaining importance because of its many perceived advantages. The reasons behind the conversion to paste fill include economic, environmental, geotechnical, and safe improvements. The use of backfill in underground mine openings and workings is increasing due to the need for systematic selection [2]. Backfill is defined as the material or materials that are utilized in void openings of underground mines for mining technical or mining safety purposes. Backfill is applied to prevent fires and explosions, to improve mine ventilation, to improve the stability of the rock, to reduce subsidence effects at the surface, as well as for economic and environmental factors. Materials are mainly from the mining industry and mining-related, such as fly ash, Flue Gas Desulfurization (FGD), gypsum, slag, infertile overburden, tailings, filter dust, residues from mineral processing, etc., and other industries (e.g., incineration ash, used building material, old bricks, used foundry sand, furnace blow-out, etc.). Mining with backfill technology helps mining companies achieve many of these goals. The backfilling technology enables a wide range of engineering solutions for particular mine sites and their unique sets of problems and opportunities. William Ross Wayment was well known to backfill excavated or mined out regions of an underground mine by transporting a slurry of sands or mill tailings having 40 to 70% solids, by weight, to the stope areas to be filled [3].

Further, backfill is not utilizable in the mining operations until it has been proved about consistency and strength to permit men and machines to support safely on the surface of the fill. Dimitre Antonov has presented mine backfill design concepts and procedures, physical and mechanical properties of backfill, and their measurement [4]. Concerning the backfilled stope stability, some new methods for minimizing amount of cement used in backfilled stopes have been presented. Manoon Masniyom investigated backfill materials and techniques suited for systematic selection and application of backfill in underground mines [5]. Laboratory tests were carried out on the physical, chemical, and mechanical properties of different backfill materials and mixtures, therefore. Special attention was paid to materials generated as by-products and other cheaply available materials, e.g., fly ash and FGD-gypsum from power plants, natural and synthetic anhydrite. The system selection, application, and placement of backfill in underground mines are truly multidisciplinary processes. One of the investigated material mixtures can be used as a technically and economically viable backfill for underground mines, is the crushed backfill material [5].

In recent years, in Vietnam, some coal mines are in the studying process to convert the backfilling form, especially the possible utilization of entire waste rocks and fly ashes as components of paste fill for environmental benefits [6]. Some studies have indicated that the backfill can enhance the support potential in underground mining operations [7, 8].

The tests at the underground coal mines in Cam Pha and Ha Long areas, Quangninh province in Vietnam, have had many positive results in decreasing the surface subsidence and increasing the coal recovery from the protective pillars in these mines.

According to the geological documents, down to the level of −400 m, the coal reserve at the southern wing area of the Mao Khe coal mine, under the protected surface works is relatively significant, about 23.2 million tonnes, which accounts for 33.2% of the total capacity of the whole region [9]. This reserve is mainly concentrated in the seams, namely V9A, V9, V9B, V8, and under constructions such as residential areas, power lines, streams and reservoirs, open pits, old underground. The coal capacity by the current mining technology of roof rock control will cause the surface displacement deformations, which can destroy civil and industrial constructions, and increase the water output or the risk of water burst from water-filled objects in old underground tunnels. Currently, to ensure the safety in mining activities, the Mao Khe coal mine has applied the technical solution of leaving the coal reserve as the safety banks to keep the stability of the surface works and mining facilities nearby the area. This solution can meet the protected requirements of surface works but wastes a large volume of coal resources. Thus, it is necessary to study and apply reasonable mining technology to exploit the reserves maximally. At the same time, the surface works are still protected as well as the mining operations must be safe. The mining technology of controlling a collapsed rock wall by insertion is a technological solution that ensures technical requirements.

Meanwhile, the Mao Khe Thermal Power Plant invested by Vietnam National Coal and Mineral Industries Holding Corporation Ltd. (VINACOMIN), has been officially put into commercial operation since January 18, 2013. Two thermal generation sets operate by using two circulating fluidized bed boilers CFB, burning with coal dust 6B. After seven years of operation, the plant has produced over 20 billion kWh of electricity for the national electricity system. Annually, it consumes about 1.6 million tonnes of coal from Dong Trieu - Uong Bi areas and emits about 650 thousand tonnes of ash and slag, of which bottom ash is from 30–40%, and fly ash is from 60–70% [10]. According to the analysis, the ash and slag from the Mao Khe Thermal Power Plant have no hazardous components so that they can be used for the production of unburnt construction materials and as cement additives. However, in recent years, the ash and slag from the plant have been partly consumed as additives for manufacturing cement and unburnt construction materials (accounting for nearly 17% of the total amount of ash and slag discharged by the plant). The rest cannot be consumed and still have to be transported to the waste dump of the plant. Hence, environmental issues, waste dump, and treatment costs have cared as significant problems.

This study aims to solve two simultaneous problems:

- The thorough exploitation of coal resources under the surface works that need to be protected;
- Releasing ash and slag from the thermal power plants by thoroughly using them as materials to fill in the mined areas in the underground mines to protect surface works.

2 Introduction of the Coal Reserve at the Southern Wing Area of the Mao Khe Coal Mine Under the Protected Surface Works

Based on the geological documents, the coal reserve at the southern wing area of the Mao Khe coal mine down to the level −400 m is 69,938.6 thousand tonnes [9]. In which, the reserve under the surface works that need to be protected, such as residential areas, power lines, reservoirs, and streams, is approximately 23,240.6 thousand tonnes, which accounts for 33.2% of the total coal capacity of the southern wing area to the depth of −400 m.

This coal reserve is saved to use as the safety banks for the protected surface works at the southern wing area of the Mao Khe coal mine. Most of them are mostly distributed in the coal seams, namely V10, V9, V9a, V9b, V8, V8a, V7 and V6 [10]. The coal reserve for the safety banks mostly concentrates at the following seams: the 9 seam (6,729.1 thousand tonnes, accounts for 29% of the total coal reserves), the 9a seam (5,853.5 thousand tonnes, 25.2% of the total), the 9b seam (3452.1 thousand tonnes, 14.9% of the total), the 8 seam (3,038.9 thousand tonnes, 13.1% of the total), and the 10 seam (3,009.6 thousand tonnes, 12.9% of the total). The coal reserve of the remained seams accounts for 1.1–2.2% the total capacity of the safety bank under the protected surface works at the southern wing area of the Mao Khe coal mine (Fig. 1).

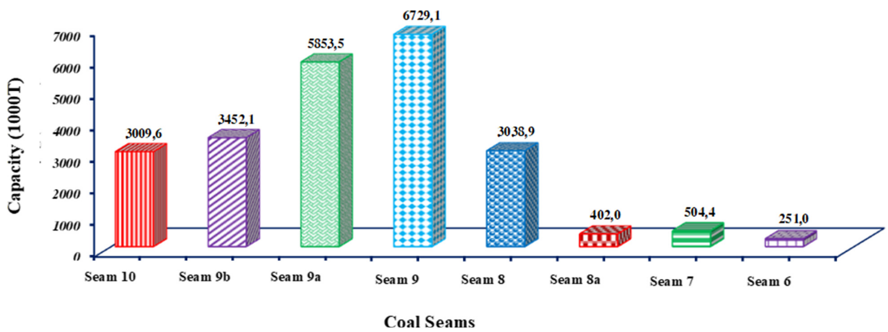

Fig. 1. Coal reserves for the safety bank at the southern wing area of the Mao Khe coal mine

The protected surface works at the southern wing of the Mao Khe coal mine is divided into four groups: (i) residential area, (ii) high voltage power lines (110 kV, 220 kV), (iii) streams, water reservoirs, and (iv) pit mine and old tunnels [9]. In which, the coal reserve under the area of high voltage power lines is the most significant, about 13,788.6 thousand tonnes (59.3% of the total) (Fig. 2).

According to the range of thickness and slope angle, the coal reserve in the safety bank area is mainly distributed at the seams having slope angle higher than 4.5°. The coal reserve of these seams is 20,947.3 thousand tonnes, which accounts for 90.1% of the total (Table 1).

Thus, the current coal reserve is maintained to protect the surface works at the southern wing area of the Mao Khe coal mine is over 23 million tonnes. With the current production is nearly 2 million tonnes/year, the coal reserve of this area is sufficient for the Mao Khe coal mine to exploit more than ten years. Therefore, it is necessary to find reasonably technical and technological solutions to exploit effectively and safely this reserve [7, 8].

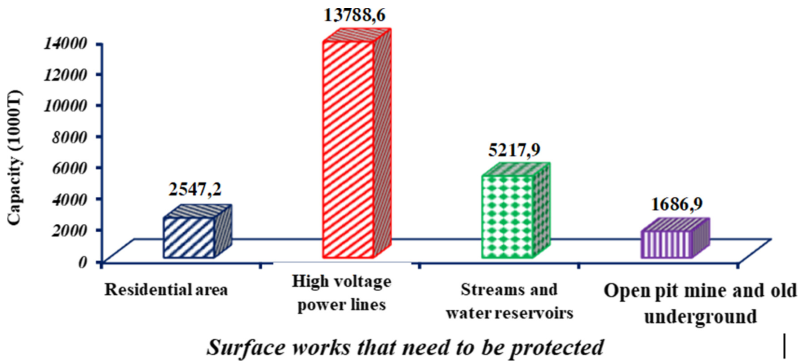

Fig. 2. Coal reserve for the safety bank at the southern wing area of the Mao Khe coal mine by the protected surface works

Table 1. Coal reserve in the safety bank at the southern wing area of the Mao Khe coal mine by the thickness and slope angle of coal seam

Coal reserve		Thickness of seam (m)			Total
		≤3.5	3.5÷5.0	>5.0	
Slope angle of seam (degree)	≤45	489.2	200.2	1,603.9	2,293.3
	>45	7,492.3	5,663.5	7,791.5	20,947.3
Total		7,981.5	5,863.7	9,395.4	23,240.6

3 Determination of Safe Mining Depth Under the Protected Surface Works

The safe mining depth is calculated based on the protected surface works. Each type of protected works is characterized by the permissible displacement parameters. They can be determined as follows:

3.1 Safe Mining Depth Under Residential Areas

For residential areas, the determination of the safe mining depth under the different conditions that the protected surface works are houses or civil constructions, and in the case of a coal seams, needs to take into account the slippage of soil and rock along its contact surface. The safe mining depth, in this case, is determined by the formula [11]:

$$H_a = \frac{0{,}9 \times m \times sin^2\alpha}{[D_c]} \tag{1}$$

where m is mining thickness (meters); [Dc] is the value of permissible horizontal deformation (deformation criteria) or the inclination of work, whose value is taken from Table 2; α is the slope angle of the coal seam (degree).

Table 2. Criteria of permissible ground deformation and limitation for the group of civil and industrial constructions on the mine surface

No.	Type of construction	Type of deformation	Value	
			Permission	Limitation
1	3-storey houses, brick wall 10 m × 15 m, atrophy 30%	[ε]	4×10^{-3}	5×10^{-3}
2	3-storey houses, frame structure 15 m × 20 m, atrophy 30%	[ε]	5×10^{-3}	6×10^{-3}
3	Brick barrier wall 0.2–0.3, 1.5 m high, 40–50 m long.	[ε]	10×10^{-3}	12×10^{-3}
4	School buildings, hospital, office, concrete frame structure, brick wall, 2–3-storey high, atrophy 30%	[ε]	3.5×10^{-3}	$4,5 \times 10^{-3}$
5	*Water pipelines*			
	a. Main pipeline on the ground: b. Main steel pipeline underground:	[ε] [ε]	10×10^{-3} 5×10^{-3}	15×10^{-3} 8×10^{-3}
6	Load of rail under 10 million tonnes/year, speed within 40 km < v < 80 km National highway 18	[i] [ε]	10×10^{-3} 8×10^{-3}	10×10^{-3} 8×10^{-3}
7	*Substation*			
	a. 110÷400 kV	[ε]		7×10^{-3}
		[i]		11×10^{-3}
	b. <110 kV	[ε]		10×10^{-3}
		[i]		14×10^{-3}
8	Depth of water leading construction with concrete structures	[ε]	5×10^{-3}	

3.2 Safe Mining Depth Under Power Lines

As the surface works that need to be protected as a power line, the safe mining depth is determined by the following formula [11]:

$$H_a = k_a \times m \tag{2}$$

where m is mining thickness (meters); k_a is the safety factor, shown in Table 3.

Table 3. Determination of the safety factor k_a for the pillars of high voltage lines [11]

No.	Types of pillar	Voltage (kV)	k_a
1	Pillar has braces and braces in corners, in the intermediate corner, at the end	220–400	100
		6–110	75
2	Intermediate on the straight line	220–400	75
		6–110	60

3.3 Determination of Safe Mining Depth Under Water-Filled Objects (Streams, Ponds, Reservoirs or Old Tunnels, Open Pits)

According to the results of several research projects conducted by the Institute of Mining Science and Technology [6–8], the depth mining safety below the water-filled objects under geological conditions in Quang Ninh area is determined by the formula:

$$H_a = 50 \times m \tag{3}$$

where m is mining thickness (meters).

From the formulas for determining the safe mining depth, under the condition that the 9b seam is at the top of the southern wing coal seams, the authors calculated and determined that the average thickness and slope angle of this coal seam are 6.9 m and 65^o, respectively. The safe mining depths corresponding to each group of protected surface works are shown in Table 4.

The calculation results shown in Table 4, indicate that if only the 9b seam is exploited, the safe mining depth under the civil and industrial constructions is H_a = 510–1,547 m. In case the protected surface works are high-voltage power lines, the safe mining depth is H_a = 518–690 m. In case the protected surface works are water-filled subjects, the minimum safe mining depth is H_a = 345 m (for coal seam). Calculation results show that the coal reserves under the surface works in the assessment depth from the level −400 m to the seam road, can not be exploited by the traditional mining method of full collapse. Therefore, to mine coal in this area, it is necessary to consider the application of the insertion method, in which the insertion material is a factor that needs to be given priority.

4 Assessment of the Possibility of Using Ash and Slag from the Mao Khe Thermal Power Plants as Insertion Material

The Mao Khe Thermal Power Plant belonging to VINACOMIN, uses the circulating fluidized bed modern technology CFB. The output of the coal burning process from the plant is ash and slag, which are composed of unburnt inorganic substances and available in coal under the form of glass structure and amorphous. Ash and slag from the plant are divided into two types, including fly ash and bottom ash. Fly ash is a fine mineral remaining from the burning of coal with small particle size (0.5–100 μm). They float in the exhaust smoke and are collected by the electrostatic dust filter system whose main

Table 4. Calculation results to determine safe mining depth under the protected surface works

No.	Protected surface works	Permissible deformation criteria ε or safety factor k_a	Safe mining depth H_a (m)
I	*Civil and industrial constructions*		
1	1–3-storey houses, brick wall 10 m × 15 m, atrophy 30%	$\varepsilon = 4 \times 10^{-3}$	1,275
2	1–3-storey houses, frame structure 15 m × 20 m, atrophy 30%	$\varepsilon = 5 \times 10^{-3}$	1,020
3	Brick barrier wall 0.2–0.3, 1.5 m high, 40–50 m long.	$\varepsilon = 10 \times 10^{-3}$	510
4	School buildings, hospital, office, concrete frame structure, brick wall, 2–3-storey high, atrophy 30%	$\varepsilon = 3.5 \times 10^{-3}$	1,457
II	*High-voltage power lines*		
5	High-voltage power lines 110 kV	$k_a = 100$	690
6	High-voltage power lines 220 kV	$k_a = 75$	518
III	*Rivers, streams, reservoirs, and old tunnels and open pits*	–	345

components are SiO_2 and Al_2O_3. Bottom ash is coarser and bigger than fly ash with a size of 0.125–2.0 mm. Its structure is similar to sand and gravel. Fly ash is often used as an admixture for concrete or as unburnt lightweight bricks, while bottom ash is often used as a substitute for roadbed backfilling [12].

The survey results show that Mao Khe Thermal Power Plant is operating in the consumption power of about 5,000 tonnes of coal/day-night. It emitted about 2,000 tonnes of ash/day-night (including fly ash and bottom ash; the ratio of fly ash to bottom ash is 7:3), equivalent to about 3,000 m^3 of ash/day-night (approximately 900 m^3 of bottom ash/day-night). This amount of ash and slag is gathered at the waste dump of the plant (Fig. 3).

To conduct experiments for evaluating the properties of ash and slag from the plant, and study the possibility of using ash and slag as hydraulic backfill material, the authors realized that the uniform coefficient of fly ash and bottom ash from the Mao Khe Thermal Power Plant was significant. The particle size is less than 2.5 mm, and they do not contain clay and flammable particles. The fly ash of the plant has a high smoothness, porosity, small permeability coefficient, which makes it difficult to drain water, so it is

Fig. 3. Waste dump of ash and slag from the Mao Khe Thermal Power Plant

not suitable for the hydraulic insertion method. Otherwise, the bottom ash of the plant has characteristics similar to the sand in the river, which has good permeability, quick drainage, and high compaction. Therefore, it is very suitable for the hydraulic block insertion method. The results of the experiment (Figs. 4, 5 and 6) have confirmed that bottom ash discharged from the plants mostly meets the required criteria for hydraulic insertion materials such as particle composition, water permeability, ability to transport through pipelines, etc.

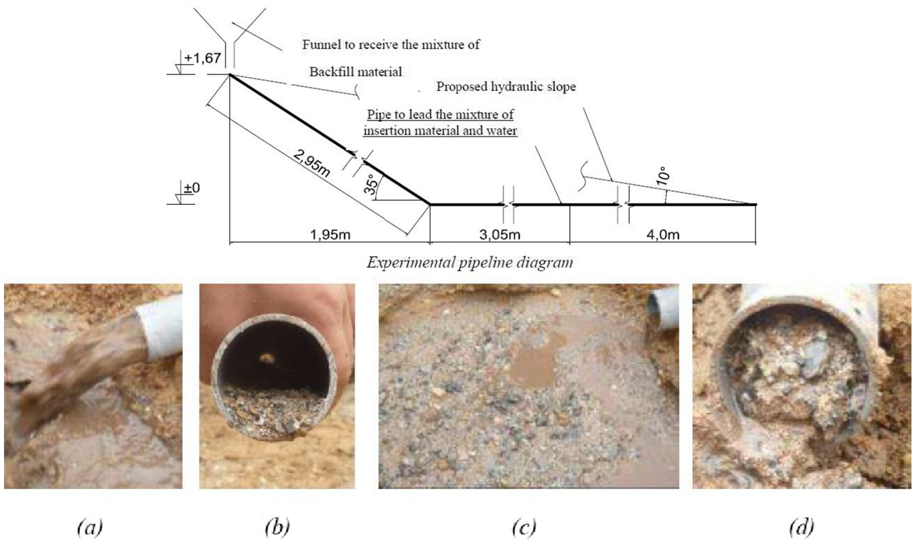

Fig. 4. Experimental results for determining the effect of the ratio of the mixture of insertion material and water to the pipe's transport capacity: (*a*) - Flow state of the mixture with the ratio of bottom ash to water is higher than 1:4; (*b*) - Condition of the pipeline after the experiment with the ratio of bottom ash to water greater than 1:4; (*c*) - State of the mixture after flowing out of the pipe (photo is taken after the experiment 1 min, the permeability of the mixture is large); (*d*) - Condition of the mixed pipeline with the ratio of bottom ash to water less than 1:4 (the pipe is clogged).

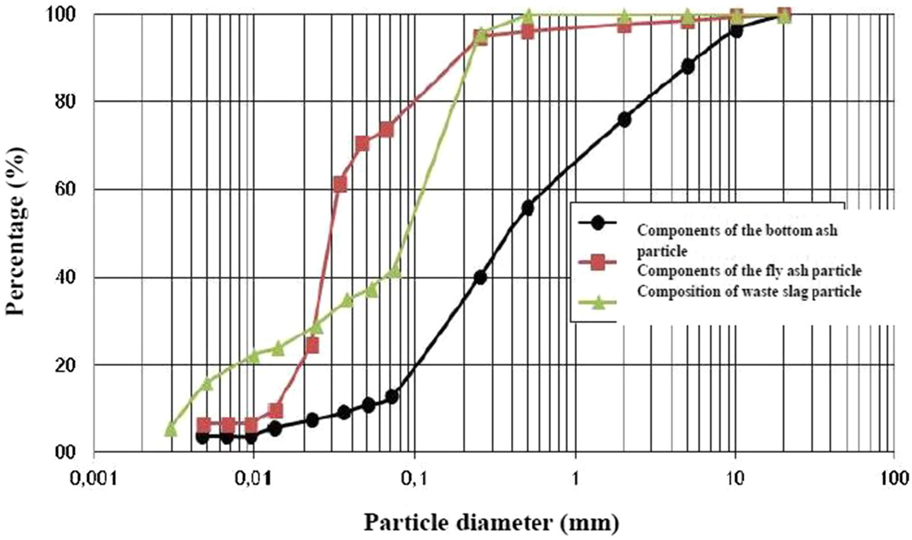

Fig. 5. Experimental results for determining the composition of the particle of the backfill materials

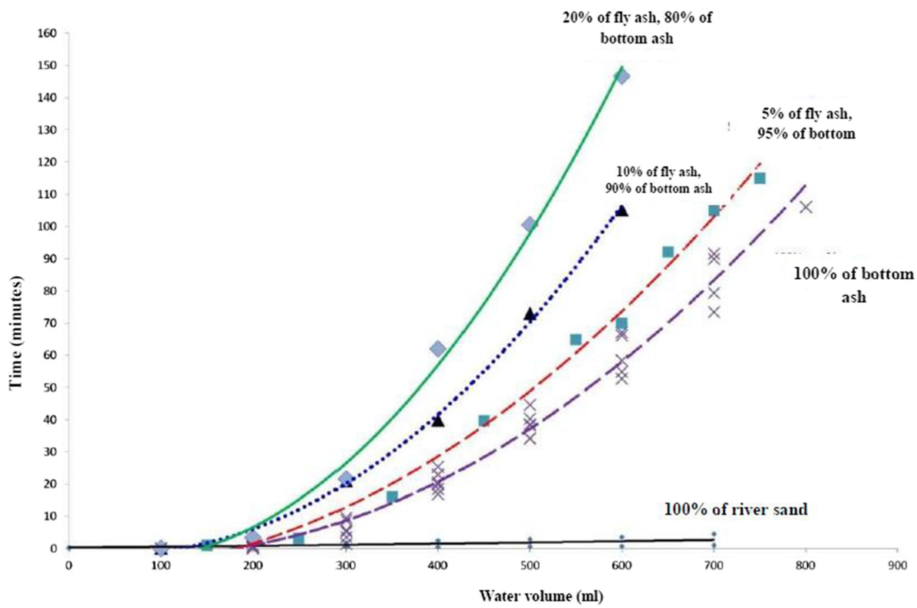

Fig. 6. Experimental results for determining the permeability of the insertion materials

Flow state of the mixture with the ratio of bottom ash to water is higher than 1:4, and with the component of fly ash is 20%, and bottom ash is 80% (Figs. 4 and 6) recorded is the best in here.

5 Proposal and Calculation of Technical Solutions to Apply for the Mining Area

5.1 Proposal for the Opening Solution

Based on the current situation of the mine site of the design area, the authors propose an introductory solution to open seams in accordance with the backfilling technology as follows:

Transportation at the level −80:

- Digging the crosscut No.3 at the level −80: The crosscut is dug from the drift at the level −80 on footwall to the coal seam. The length of the crosscut is 33 m. The digging area is 8.4 m^2; the support steel is type of CBΠ-22; the distance between two supports is 0.7 m.
- Digging the water storage at the level -80: The storage is dug from the crosscut TN1 at the level -80 m to the junction with the crosscut 9b with a distance of 15 m. The water storage is dug in rocks with a length of 30 m, slope of 15÷20° and its cross-section in a dome shape. The digging area is 8.4 m^2; the used area is 6.4 m^2, the support steel is CBΠ-22; the distance between two supports is 0.7 m.

Ventilation level:

- Digging a raise from the levels −80/+25: The position to open the raise is on the crosscut TN1 and is 15 m from the junction with the crosscut 9b. The raise is dug from the level −80 to the level +25 (terrain surface). The location of the collar is to the northwest of area I and 25 m far from the 9b seam. The slope angle of the raise is 35°; its length is 183 m; the section of support is in arched-shape; the dug area is 8.4 m^2; the used area is 6.4 m^2; the support steel type of CBΠ-22; the distance between two supports is 0.7 m.
- Digging a crosscut at the level +0: From the raise at the level ±0, digging a crosscut to the coal seam (until meeting the seam); the length of the crosscut is 46 m; the section of support is in arched-shape; the support steel is CBΠ-22; the distance between two supports is 0.7 m.

Diagram of the seam opening and preparation of longwall mining system based on the mining slope in order from bottom to top by the mining technology with horizontal and inclined slice division for the longwall of the 9b seam at the levels −80/+0, is shown in Figs. 7 and 8.

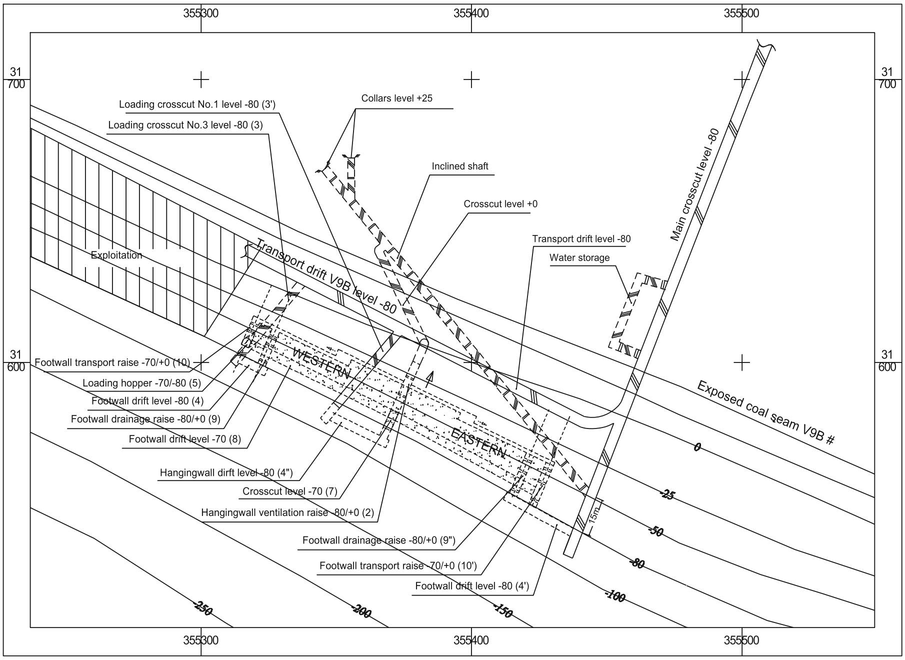

Fig. 7. Diagram of opening seams and preparing the application area

5.2 Technical Instructions to Keep Longwall Face

Mining operation by the drilling-blasting method is carried out in strips, the width of the range is from 0.8 m to 1.2 m depending on the hardness of the coal. In the longwalls supported by the wooden pillars, the technical instructions are as follows: the width of strips is 1.0 m, supported by wooden shorts with a length of 2.4 m, each beam supported by three wooden pillars. Actually, the width of the strip is selected as 1.0 m, depending on the actual conditions of the longwall with or without additional support by increasing the capacity for supporting beamlines (Fig. 9).

5.3 Insertion Operation and Roof Control

In controlling rock-wall by hydraulic insertion method, the insertion materials such as ash and slag are taken from the Mao Khe Thermal Power Plant. The materials are transported by dump trucks from the plant to area in front of the tunnel, then fed into an underground bunker, and mixed with other materials at the level +25 of the tunnel door by a ripper. At the underground mixing bunker location, a high-pressure water gun was available. Workers control the gun to shoot dry material at the foot of the bunker. The mixture of water and insertion material flows down the navel to collect water at the bottom of the bunker and flows into the insertion material transporter at rock-above tunnel at the levels −80/+25. In the next step, the mixture of insertion materials and water under the kinetic energy of the flow is transported through a pipeline system located at the tunnel lines: rock-above tunnel at the levels −80/+25, rock-curb through seams at the level ±0, ventilated upper tunnel, to the insertion position. The formwork is available at the location of the tunnel. Workers control a pipeline of mixed materials to flow in and fill the exploited space.

The insertion step is often selected from 6–8 m. When the thickness of the seam increases and the strength of the coal decreases, the insertion step is from 4–6 m. In the seams where the thickness is not thick, and the solid coal mass is stable, the insertion step may be up to 12 m or thicker. In case the coal seams are 6.9 m thick and relatively soft as the condition of the 9b seam in the southwest region, the insertion step is designed in the range of 4–6 m. In fact, it depends on the status and stability of the longwall, and the length of the insertion can be increased to enhance the insertion speed as well as to reduce the cost of wood as a partition between the insertion strips. The technical instruction of the tunnel is shown in Fig. 10.

At the stage of insertion, the face supporting operation should be stopped to conduct the insertion. The technological process of the insertion includes the following stages: cable cleanup, transport scraper conveyers, formwork to make space for pumping materials operating pumps to insert materials into newly created spaces, installing electric cable systems, industrial cleaning of furnace lines.

5.4 Calculation of the Degree of Surface Displacement Using the Insertion Method

Based on the calculation and prediction methods of dynamic surface displacement in mining operation and the control of roof rock by the hydraulic fills [1–8, 13–17], the authors have calculated to simulate the results using the input geological parameters related to the seam 9b, and the area expected for the pilot application.

Typical parameters related to the topographic surface displacement (Fig. 11) include settlement η, inclination I_x, horizontal deformation ε_z, and curvature K_x determined by [18].

Settlement

The formula determines vertical deformation as below:

$$\eta_z = \eta_m \times S(z) \tag{4}$$

where η_m is maximum subsidence (mm); η_z is subsidence at z point on the surface (mm); S(z) is functional determination subsidence at z point on the surface (the value of S(z) in Table 5); z is a relative coordinate of z point, $z = x/L_{1,2}$ with x is the distance between z points and maximum subsidence η_m, while $L_{1,2}$ is the size of supercritical (Fig. 12).

Inclination

The inclination of the placement in the subsidence pot follow a formula:

$$i_z = \pm \frac{\eta_m}{L_{1,2}} \times S'(z) \tag{5}$$

Tilt of the Displacement in the Horizontal Direction

The tilt of the displacement in the horizontal direction of the points at the coal wall area corresponding with $L_{\mathrm{B}1}$, $L_{\mathrm{B}2}$ (see Fig. 11) is:

$$\xi_z = k_0 \times \eta_m \times F(z) \tag{6}$$

The tilt of the displacement in the horizontal direction of the points at the coal pillar area corresponding with $L_{\text{Л}1}$, $L_{\text{Л}2}$ (see Fig. 11) is:

$$\xi_z = -k_l \times k_0 \times \eta_m \times F(z) \tag{7}$$

where F(z) is functional determination subsidence follows the horizontal direction of z point (see Table 6); k_1, k_0 are coefficients dependent at depth H_{B} and space exploitation D_1 (see Table 7).

Horizontal Deformation

The horizontal deformation of any point in the subsidence is determined as follows:

– At $L_{\mathrm{B}1}$ area:

$$\varepsilon_z = \frac{\eta_m}{L_{B1}} \times S'(z) \tag{8}$$

– At $L_{\mathrm{B}2}$ and $L_{\text{Л}1}$ area:

$$\varepsilon_z = \frac{k_0 \times \eta_m \times (1 + k_\pi)}{L_{B2} + L_{\pi 1}} \times F'(z) \tag{9}$$

– At $L_{\text{Л}2}$ area:

$$\varepsilon_z = \frac{k_0 \times \eta_m \times k_\pi}{L_{\pi 2}} \times S'(z) \tag{10}$$

The curvature

The curvature of any point in the subsidence pot the following formula:

$$K_m = \frac{i_m - i_{m-1}}{l_{cp}} \tag{11}$$

where i_{m}, $i_{\mathrm{m}-1}$ are degrees of inclination at m point and $m - 1$ point; l_{cp} is average distance length.

Based on the selected calculation method, the prediction of surface displacement parameters was calculated when controlling the roof rock by hydraulic backfilling (see Figs. 13, 14, 15 and 16).

The summary of calculated terrain deformation parameters for the proposed pilot application area is shown in Table 8. From this result, the surface displacement area corresponding to cutting lines on the topographic map of the mine can be specified (Figs. 17 and 18).

6 Conclusion

The above-calculated results show that when exploiting and backfilling by hydraulic method at the levels −80/+0 of the long wall with a horizontal length of exploitation of 210 m, a surface displacement tank with horizontal dimensions will be formed a length (about 380 m) and a width (about 300 m). However, the parameters of displacement and surface deformation are not significant: maximum settlement $\eta_m = 42$ mm, maximum inclination $I_z = 0{,}64.10^{-3}$, maximum horizontal deformation $\varepsilon_z = 0{,}52.10^{-3}$. The predicted values of displacement and deformation parameters are all within the permissible limits for civil and industrial constructions on the surface in Table 4. Therefore, the technology of exploiting and controlling cliff walls using hydraulic furnace inserts meets the requirements of protecting surface works. Flow state of the mixture with the ratio of bottom ash to water is greater than 1:4, and with the component of fly ash is 20%, and bottom ash is 80% is the best chosen.

Research results of the calculation and application of waste ash and slag from the Mao Khe Thermal Power Plant as backfilling materials for the underground mining works at the southern wing area of the Mao Khe coal mine are feasible. The proposed technical solution meets not only the technical requirements (mining safety) but also the criteria to maximally utilize coal mineral resources without causing significant impacts on surface works that need to be protected. Besides, the use of fly ash and bottom ash will contribute to the release of a large amount of production output from the Mao Khe Thermal Power Plant, which helps to reduce the area of the waste dump, reduce environmental impacts, and increases economic benefits.

Conflict of interest. The authors declare that there is no conflict of interest.

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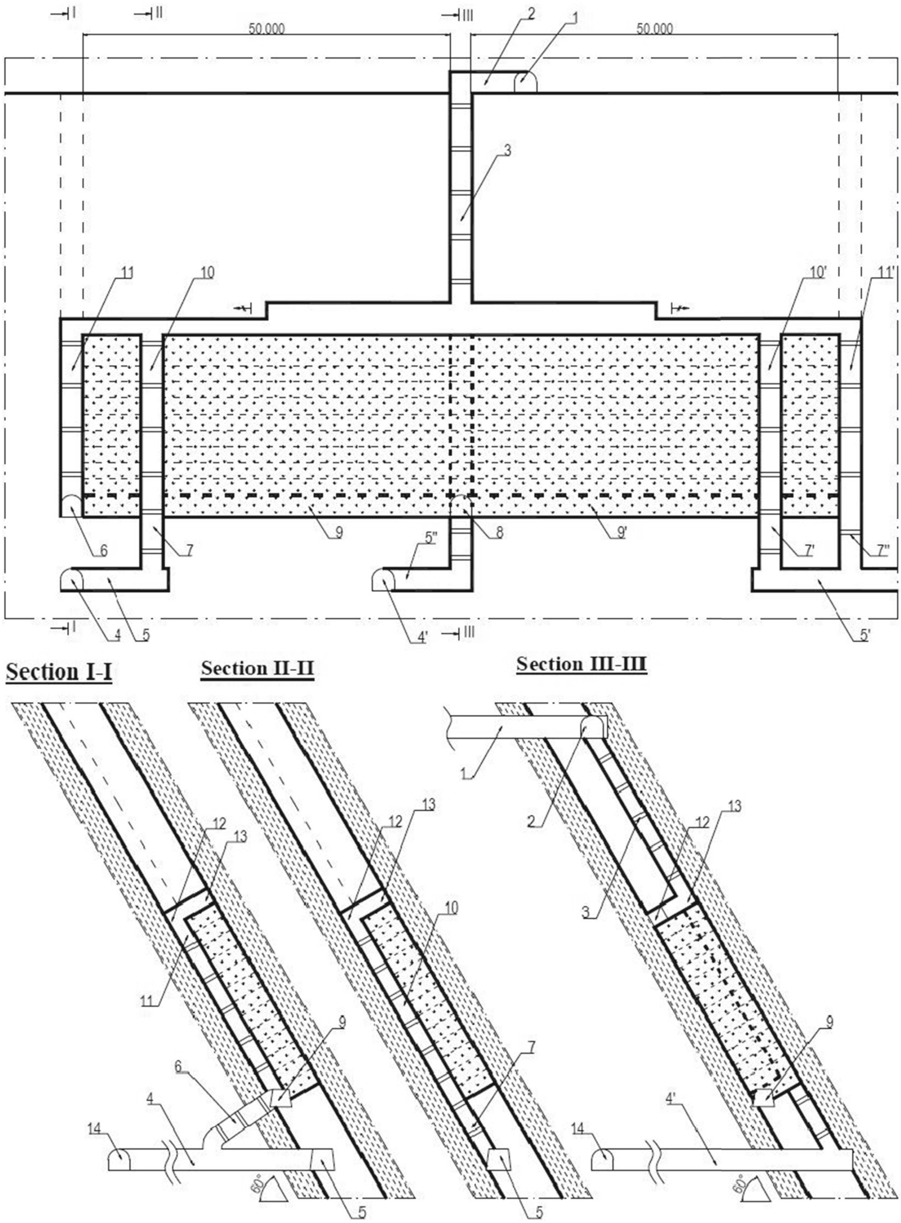

Fig. 8. Diagram of the preparation of the long pillar mining system based on the mining slope in order from bottom to top by the horizontal slicing exploitation technology of the 9b seam: *1* - Crosscut, level +0; *2* - Hangingwall ventilation drift, level +0; *3* - Hangingwall raise, levels −80/+0; *4, 4′* - Crosscuts, level −80 (western and eastern wing); *5, 5′* - Footwall drifts, level -80; *5″* - Hangingwall drift, level −80; *6* - Loading hopper, levels −80/−70; *7, 7′* - Footwall drainage raise, levels −80/−70; *7″* - Transport raise, levels −80/−70 in the eastern wing; *8* - Crosscut; *9* - Footwall drift, level −70; *10, 10′* - Drainage raises (formed in backfill block); *11, 11′* - Transport raises (formed in backfill block); *12* - Footwall transport drift (formed in mining process); *13* - Footwall ventilation drift (formed in mining process); *14* - Drift in the rock, level −80

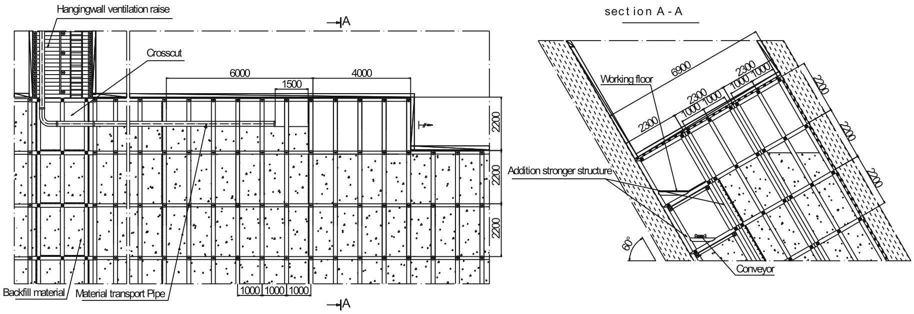

Fig. 9. Technical instructions of the longwall's face keeping

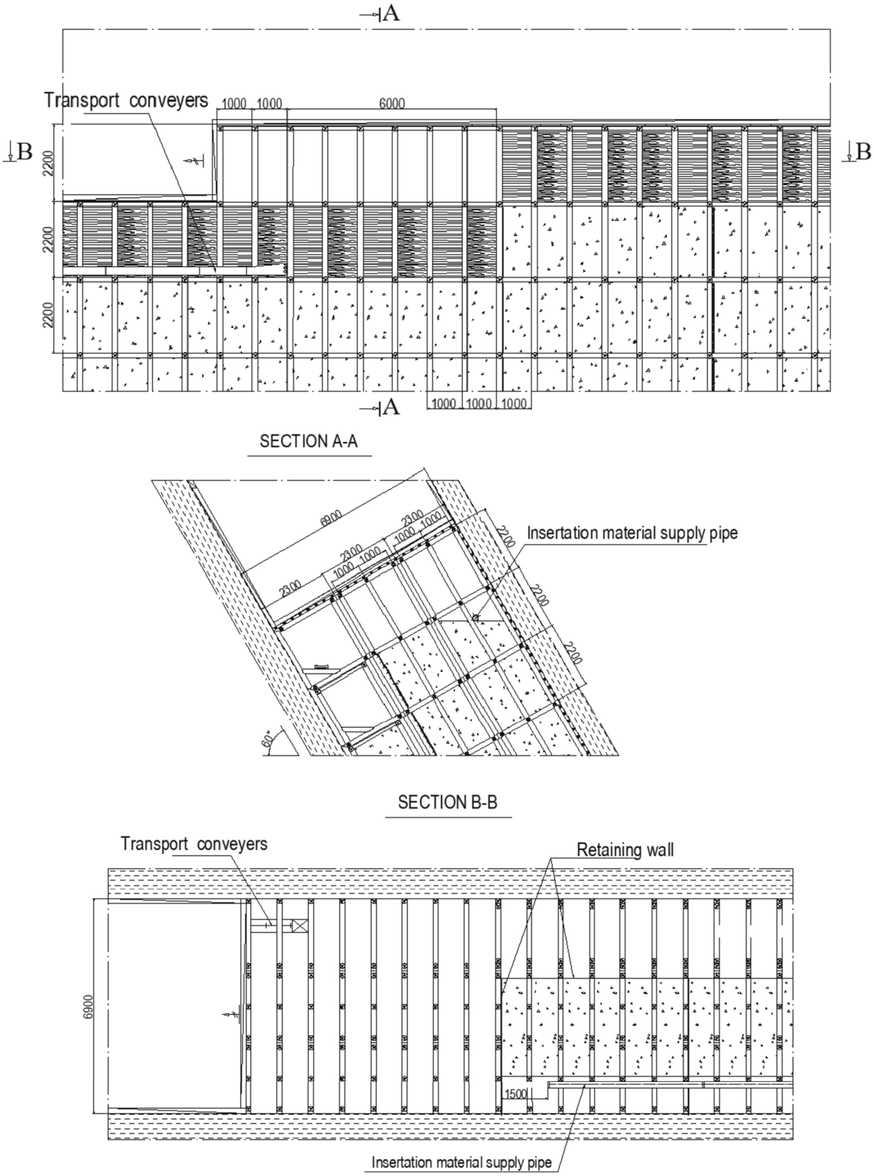

Fig. 10. Technical instructions of construction by hydraulic insertion method

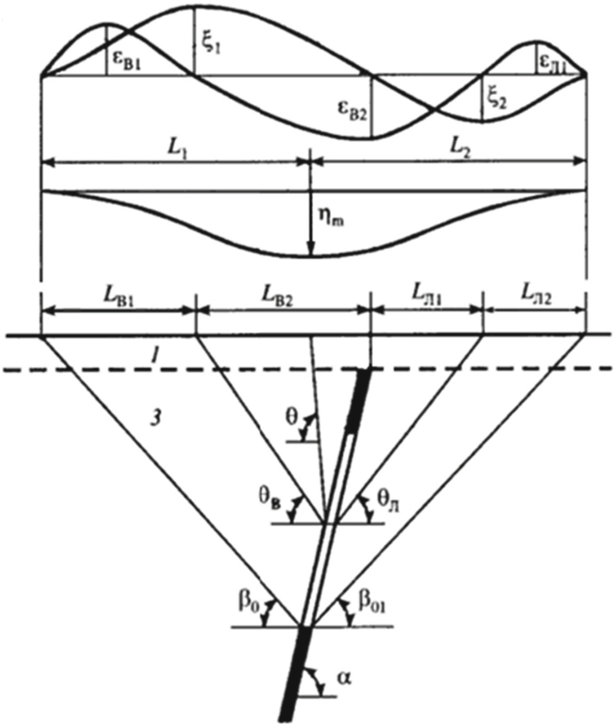

Fig. 11. Description of subsidence over longwall mining, face advance direction is perpendicular to the plane of the page

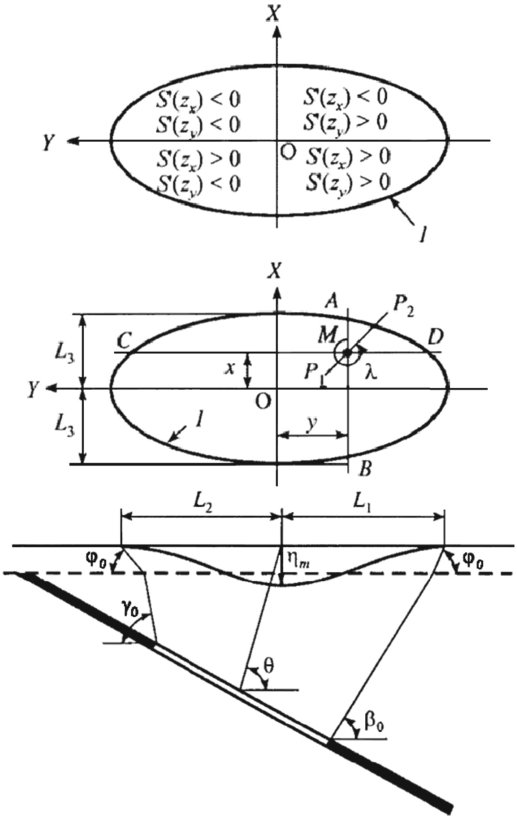

Fig. 12. The scheme of the coordinate axes and signs of functions when calculating the displacements and deformations of subsidence

Table 5. The value S(z)

z	0	0.1	0,2	0.3	0.4	0.5	0.6	0.7	0.8	0.9	1.0
S(z)	1.00	0.98	0.90	0.77	0.58	0.39	0.22	0.10	0.04	0.01	0.00

Table 6. The value F(z)

z	0	0.1	0,2	0.3	0.4	0.5	0.6	0.7	0.8	0.9	1.0
F(z)	1.00	0.97	0.93	0.88	0.81	0.71	0.60	0.47	0.32	0.17	0.00

Table 7. The value k_1 and k_0

Coefficient		Coal seam dip angle α, degrees		
		To 70°	To 80°	To 90°
k_1	when $H_B/D_1 \leq 0.1$	1,4	1,2	1,0
	when $H_B/D_1 > 0.3$	0,7	0,4	0,4
k_0		0,15	0,7	1,0

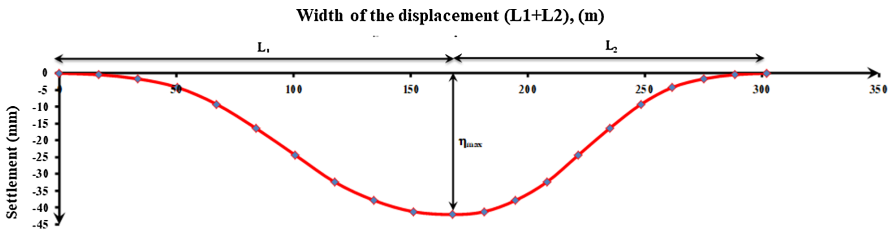

Fig. 13. Prediction of the variation in surface subsidence η

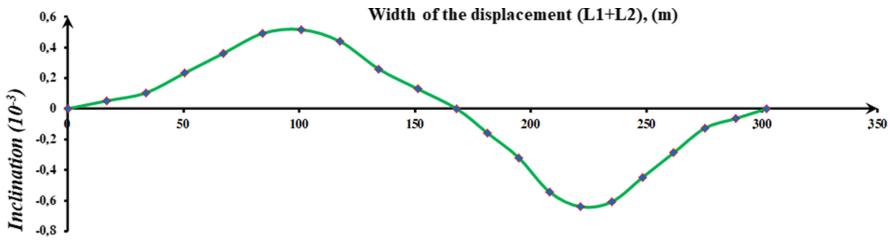

Fig. 14. Prediction of variation in inclination of topographic surface I_x

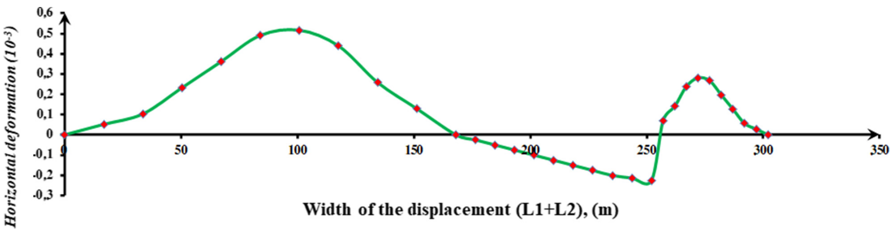

Fig. 15. Prediction of variation in horizontal deformation ε_z

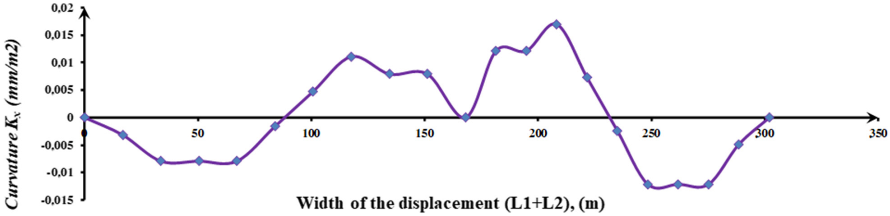

Fig. 16. Prediction of variation in curvature K_X

Table 8. Results of calculated ground deformation parameters in the test area

Relative coordinates z	Settlement η_z, mm	Inclination i_z ($\times 10^{-3}$)		The displacement in the horizontal direction ξ_z (mm)			Horizontal deformation ε_z ($\times 10^{-3}$)			Curvature K_Z (mm/m^2)	
		L_2 side	L_1 side	Wall side	L_1 side	L_2 side	Wall side	Areas L_{B2} and L_{B1}	Areas L_{J1} and L_{J2}	L_2 side	L_1 side
0	42.0	0.00	0.00	6.31	6.00	0.00	0.00	0.00	0.00	0.000	0.000
0.1	41.0	0.16	0.13	6.12	5.82	3.00	0.13	0.03	0.07	0.012	0.008
0.2	38.0	0.32	0.26	5.87	5.58	6.00	0.26	0.05	0.14	0.012	0.008
0.3	32.0	0.54	0.44	5.56	5.28	10.2	0.44	0.08	0.24	0.017	0.011
0.4	24.0	0.64	0.52	5.11	4.86	12.0	0.52	0.10	0.28	0.007	0.005
0.5	16.0	0.61	0.49	4.48	4.26	11.4	0.49	0.13	0.27	−0.002	−0.002
0.6	9.00	0.45	0.36	3.97	3.6	8.40	0.36	0.15	0.20	−0.012	−0.008
0.7	4.00	0.29	0.23	2.97	2.82	5.40	0.23	0.18	0.13	−0.012	−0.008
0.8	2.00	0.13	0.10	2.02	1.92	2.40	0.10	0.20	0.06	−0.012	−0.008
0.9	0.00	0.06	0.05	1.07	1.02	1.20	0.05	0.21	0.03	−0.005	−0.003
1	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.23	0.00	−0.005	−0.003

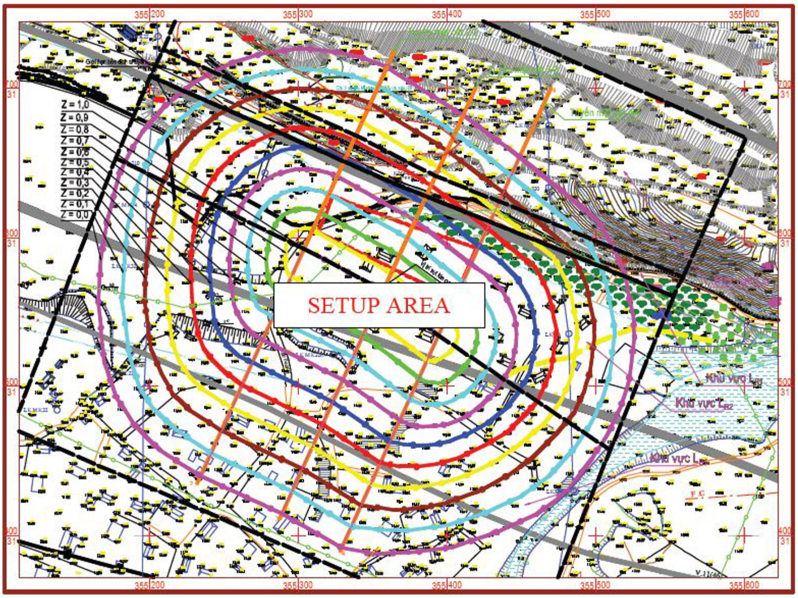

Fig. 17. Estimated displacement tank on top of the surface

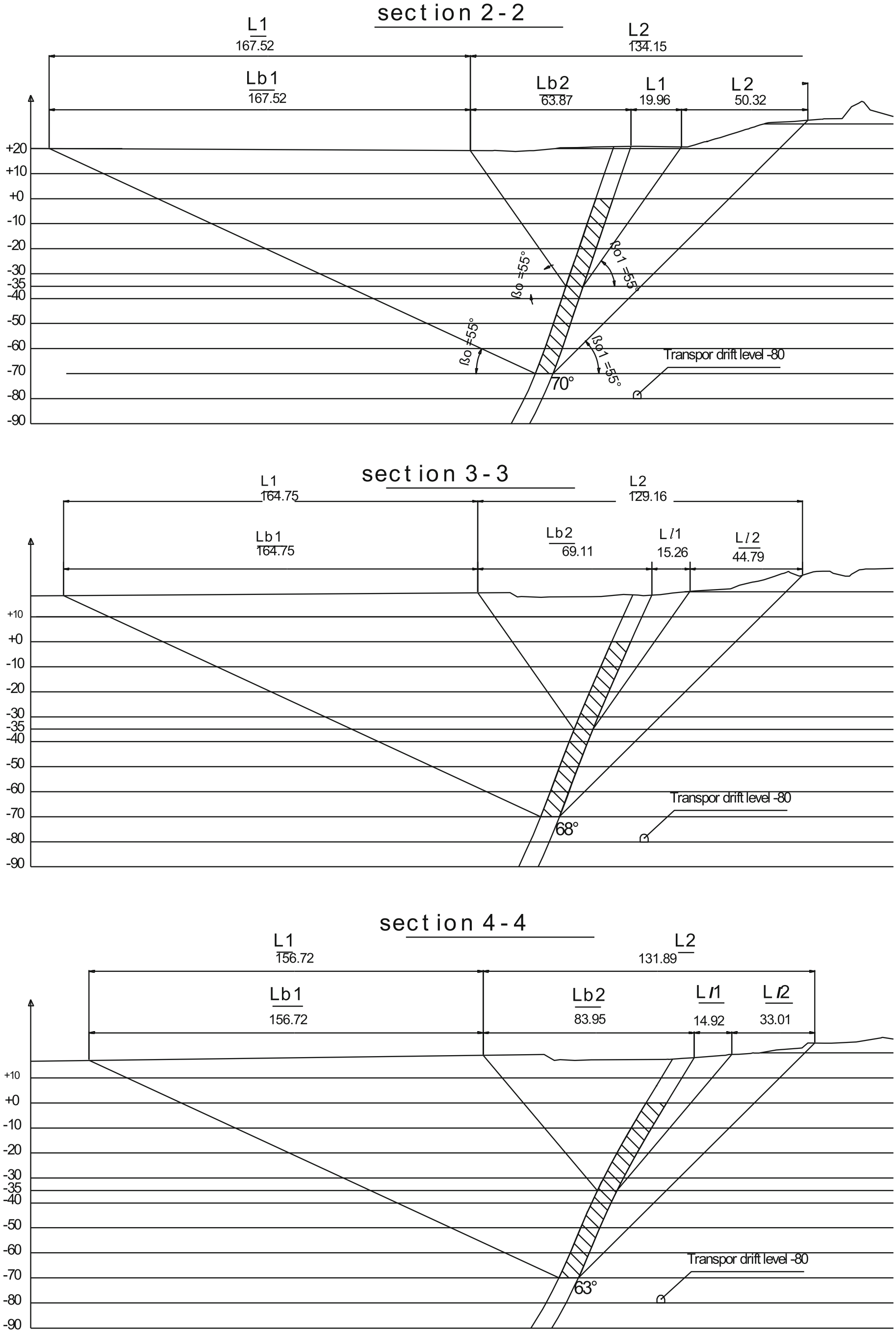

Fig. 18. Some cross-sections of the proposed displacement tank

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