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**CONTIGUOUS COAL SEAM MINING
USING POWERED SYSTEMS
IN TERMS OF LVOV-VOLYN COAL FIELD MINES**

Monograph

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P65 **Contiguous** coal seam mining using powered systems in terms of Lvov-Volyn coal field mines: Monograph / V.I. Buzilo, Ya.M. Nalivaiko, A.G. Koshka et al; Ministry of Education and Science of Ukraine; National Mining University. – Dnipropetrovsk: NMU, 2017. – 126 p.

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The monograph is devoted to the problems of substantiation of the parameters for contiguous coal seam extraction using powered systems under complex mining and geological conditions. Rational parameters (strength and geometrical parameters of a powered support, allowable distance between stopes within contiguous seams, shearer feed rate, longwall advance rate) for stoping operations in Lvov region mines are determined. Recommendations as for the parameters for contiguous seam extraction in terms of both overworking and underworking are given.

The work is students, engineers and technicians, experts of higher educational institutions, scientific and research institutions and design offices of coal industry.

44 figures, 112 reference sources.

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INTRODUCTION

Tendencies of coal extraction and use in Ukraine show the increase in its importance within the fuel and energy complex. Energy strategy of Ukraine dating up to 2030 stipulates considerable increase of coal share in fuel and energy balance of Ukraine. It requires growing efficiency of coal enterprises in most cases being determined by the degree of operation reliability control of levels and areas under definite mining and geological conditions.

Development of Lvov region coal industry is connected with the development of a suite of five contiguous seams of 0.6...1.8 m operating capacity at 10...28 m parting thickness. Coal commercial reserves of ten productive mines of "Lvovugol" State Enterprise cover more than 180 mln t. Coal is extracted by means of 22 powered longwalls. The problem is not only in the fact that 2-3 seams are extracted simultaneously but also in the fact that mining and geological conditions have been deteriorated considerably due to incoordination in the operation of the enterprise mines: seams have turned to be underworked more than once, overworked, stoping faces have to operate within the zones of high rock pressure (HRP) due to the effect of pillars and selvages of contiguous seams.

According to the mine technical service, 81% of longwall faces have operated under (above) pillars and selvages of contiguous seams; as a result powered support load has grown, frequency and intensity of inrush formations of roof rock has increased. The available longwall sets of equipment has turned to be unadapted to operate under such conditions; advance rate has decreased by 40...60% comparing to the operation outside the zones of pillar and selvedge effect; injuries have become more frequent as well.

In this respect the question has arose considering the necessity to determine rational parameters for stoping operations within the areas of high rock pressure, in particular, values of support reactions of the complexes under use, advance rate, and allowed distance between the stopes of contiguous seams.

Solution of the problem being topical for mining science and practice requires certain determination and analysis of stress and strain state (SSS) of a layered heterogeneous mass near stoping as well as the estimation of the degree of pillars and selvages effect upon rock pressure manifestation under conditions of underworking and overworking, the issue having been considered in this monograph.

Members of the Department of Underground Mining of State Higher Educational Institution "National Mining University" as well as workers of "Vidrodzhennya", "Bendiuzka", "Lisova", and "Stepova" mines of "Lvovugol" State Enterprise took part in field study.

Members of the Department of Higher Mathematics of the National Mining University, Professor, Doctor of Engineering Novikova Lyudmila Vasilievna and Associate Professor Zaslavskaya Lyudmila Ivanovna took part in the development of computational procedure.

The authors highly appreciate their help.

CHAPTER 1. STATE-OF-THE-ART AND RESEARCH TASKS

1.1. Physical and mechanical properties of coal and enclosing rocks of Lvov region

Lvov-Volyn basin, which “Lvovugol” SE mines belong to, is characterized by specific features of geological structure connected with its formation and history of geological development. The suite under development includes n_7^H , n_7 , n_7^6 , n_8 , n_8^6 , and n_9 seams. Thicknesses of these seams vary within 0.6...0.8 m while parting thicknesses are from 7 to 28 m [109]. n_8^6 upper seam is one of the main commercial seams being worked intensively by almost all Velikomostovsk mines of Chervonograd geological and industrial area. Along with n_7^H one, this seam has been worked out within considerable areas. Current stoping is performed mostly within n_7^H , n_7^6 , n_8 , and n_8^6 seams at the depth of $H = 360 - 560$ m. Coal-bearing rocks are as follows: sandstones, argillites, and aleurolites. Geological study [1] contains their physical and mechanical properties. Let us have brief description of those ones being the initial data to solve the problems of stress and strain state estimation of the mass area under consideration.

Sandstones build up most part of main roof of working coal seams within Chervonograd geological and industrial area mine fields. Sandstones of the northern part of the area are stable with 70...100 MPa ultimate compression strength. The sandstones are quite stable in the roof of n_8 and n_8^6 seams while sometimes the floor of n_8^6 and n_7^H seams contains underclay sandstones with 30...50 MPa strength.

Comparing to sandstones aleurolites take more active part in forming immediate roof and floor of coal seams. They often occur in working sections and have worse strength characteristics.

Aleurolites are abundant in the southern part of Chervonograd area. In the main roof, they either alternate with sandstones forming separate layers of 1.5...4.0 m thick or make up homogeneous thickness up to 6.0...10.0 m. Their stability is moderate with ultimate compression strength from 24.3 MPa up to 70.4 MPa. In seam floor aleurolites have compact structure being characterized by poor stability. They have maximum strength within n_8 and n_8^6 seams and minimum strength within n_7^H and n_7^6 seams.

Comparing to other lithotypes of coal-bearing rocks argillites are the most abundant ones. They have low strength even minor rock pressure deforms them with the following inrush formation. It is stipulated by intraformational washout during

which rocks of the primary seam roof are replaced by brownish-grey argillite with unstratified texture. Strength of such rocks is within 16...50 MPa. They have the poorest strength in the immediate mine roof of n_7 and n_8 seams.

Mechanical strength of coal-bearing rocks of the basin decreases noticeably at their water saturation; in case of some of their kinds, it decreases even at their slight moistening. Laboratory tests of rock samples show that full water saturation results in the fact that average strength limit of argillites and aleurolites decreases by 1.5...2.3 times and average strength limit of layered sandstones and aleurolites decreases by 3.9 times. Thus, strength of coal-bearing rocks of the area under study varies greatly; besides, ultimate compression strength of each rock lithotype increases from the north to the south along with stratigraphic depth, and in successive lithologic series it increases from shale to sandstone and limestone (thickness covering coal deposits) as well as from the rocks being in the immediate contact with coal seam it increases up and down through the section. Strength properties of the rock decrease noticeably at their water saturation. Rocks of immediate roof and floor of coal seams are weakly and moderate stable in the north and moderate stable in the south of the basin.

As for coals, the humus ones belong to a middle class in their strength while sapropelic and high-ash humus ones belong to the strong ones. Their properties vary greatly as well as the ones of enclosing rocks.

Table 1.1 represents average ultimate tensile and compression strength values for coal and enclosing rocks as well as E elasticity moduli, ν Poisson's ratios and γ densities according to VNIMI (Scientific and Research Institute of Rock Mechanics and Mine Surveying) [2,3] and the Institute of Geotechnical Mechanics of the Academy of Science of Ukraine [4].

Table 1.1

Physical and mechanical characteristics of coal and enclosing rocks
of Lvov region

Parameter Rock	$E \cdot 10^{-4}$, MPa	ν	$\gamma \cdot 10^{-3}$, kg/m ³	σ_c , MPa	σ_p , MPa
Sandstone	1.4	0.25	2.67	80	8.0
Aleurolite	1.0	0.26	2.78	44	4.4
Argillite	0.63	0.20	2.70	30	3.0
Coal	0.28	0.30	1.69	20	2.0

Table 1.2 characterizes modes of occurrence for seams within the area under study.

Table 1.2

Number of longwalls within mines, seams, and mining conditions

Mine	Seams of workable thickness	Mining depth, m	Parting thickness	Interburden layers	Number of longwalls (1996-2000)	Including the ones working in terms of effect of pillars (selvedges) within the seams						Notes	
						n ^H	n ₇	n ₇ ^B	n ₈	n ₈ ^B	n ₉		total
1	2	3	4	5	6	7	8	9	10	11	12	13	14
Velykomostivska	n ₈ ^B	-	-		-	-	-	-	-	-	-	-	-
	n ₈	415-420	8,6		3	3	-	3	-	-	-	-	3
	n ₇ ^B	429-465	12-14,4		2	2	-	-	-	2	-	-	2
	n ₇ ^H	438-472	7,3-14,2		2	-	-	-	-	-	-	-	-
Benduzka	n ₈ ^B	358-391	-		2	-	-	2	2	-	-	-	2
	n ₈	360-410	8,8-16,4		10	-	-	9	-	8	-	-	9
	n ₇ ^B	380-425	10-19,1		-	-	-	-	-	-	-	-	-
	n ₇ ^H	-	-		-	-	-	-	-	-	-	-	-
Mezhyrichanska	n ₈ ^B	451-513	-		-	-	-	-	-	-	-	-	-
	n ₈	461-527	10,5-22,6	Aleurolite - 2 m	4	4	-	1	-	2	-	-	4
	n ₇ ^B	475-536	9,3-14,4	Sandst. - 2 m	6	3	-	-	4	5	-	-	5
	n ₇ ^H	490-553	13,5-25,2	Argillite - 2 m	1	-	-	-	1	1	-	-	1

1	2	3	4	5	6	7	8	9	10	11	12	13	14
Vidrozhennya	n ₈ ^B	451-513	-		5	4		1	1	-	-	4	
	n ₈	461-527	10,5-22,6		2	2		-	-	1	-	2	
	n ₇ ^B	476-537	9,4-13		2	2		-	2	2	-	2	
	n ₇ ^H	490-554	15,1-22,2		-	-		-	-	-	-	-	
VM No.5	n ₈ ^B	381-391	-	Argil.-3 m Aleur.-4 m	1	1	-	1	-	-	-	1	
	n ₈	390-402	5,3-15,8	Argil.-5 m Aleur.-7 m	1	1	-	-	-	1	-	1	
	n ₇ ^B	402-417	6-18,6	Sandst.-5m Aleur.-5 m	1	1	-	-	-	1	-	1	
	n ₇ ^H	427-433	15-20	Sandst.-6m	2	-		1	-	2	-	2	
Lisova	n ₈ ^B	455-506		Argil.1.2m Aleur.6.5m	5	-		1	-	-	-	5	
	n ₇ ^B	484-533	25-28,2	Sandst.-7m Sandst.12m	1	1		-	-	1	-	1	
	n ₇ ^H	498-545	12-16,5	Aleur.-8 m	2	2		2	-	2	-	2	
Zarichna	n ₉	-	-		-	-		-	-	-	-	-	
	n ₈ ^B	463-476	-		3	-		-	-	-	-	-	
	n ₈	475-494	12,7-17,3		-	-		-	-	-	-	-	
	n ₇ ^B	489-507	12,7-13,8		4	2		-	2	2	2	4	
n ₇ ^H	507-528	17-20,4		-	-		-	-	-	-	-		

1	2	3	4	5	6	7	8	9	10	11	12	13	14	
Vizeiska	n ^B	395-415	-		-	-	-	-	-	-	-	-	-	
	n ^B	416-436	23-26,2		4	1	-	-	-	4	-	-	-	
	n ₇	421-432	3,7-5,6		1	-	-	-	-	-	-	4	-	
	n ^H	436-448	14,5-15,6		4	-	4	-	-	2	-	4	-	
Nadiya	n ^B	376-384	-		-	-	-	-	-	-	-	-	-	
	n ₇	400-415	23-31		2	1	-	-	-	2	-	2	-	
	n ^H	418-432	14,3-15,6		6	1	5	-	-	6	-	6	-	
	n ^B	505-510	-		7	2	-	-	-	-	-	2	-	
Stepo va	n ^H	540-550	37,2-40,2		2	-	-	-	-	2	-	2	-	
	Cheronog radska	n ^B	470-490		4	-	-	-	-	1	-	-	1	-
		n ₈	480-500	8,3-12,3		5	-	-	-	-	4	-	4	-
		n ^B	500-520	17,1-28,8		2	-	-	-	2	2	-	2	-
Total within SHC	n ₉				-	-	-	-	-	-	-	-	Total	
	n ^B				27	11	-	5	4	11	-	15	-	
	n ₈				25	10	-	13	-	16	-	23	-	
	n ^B				22	10	-	-	10	19	2	21	41	
	n ₇				3	1	-	-	-	2	-	2	3	
	n ^H				19	-	9	3	1	15	-	17	28	
Total				96	32	9	21	15	52	78	2	131		

* - underworking (overworking) without pillars

As Table 1.2 shows, the mass structure within the area under study varies considerably. Such parameters as mining depth, bed and parting thicknesses can vary; rock replacement occurs in the roofs of mined seams, longwalls come into overworked or underworked zones as well as into areas of high rock pressure due to pillars or selvedges effect.

1.2. Characteristics of the conditions for contiguous seam mining of Lvov region

Panel development scheme is being used now in the mines of Lvov region. Deposits are worked by longwalls using back drive. KMK97M, KM98, KM87, and KM88 powered systems have been used since 1996.

Fig.1.1 shows typical stratigraphic section of coal-bearing thickness.

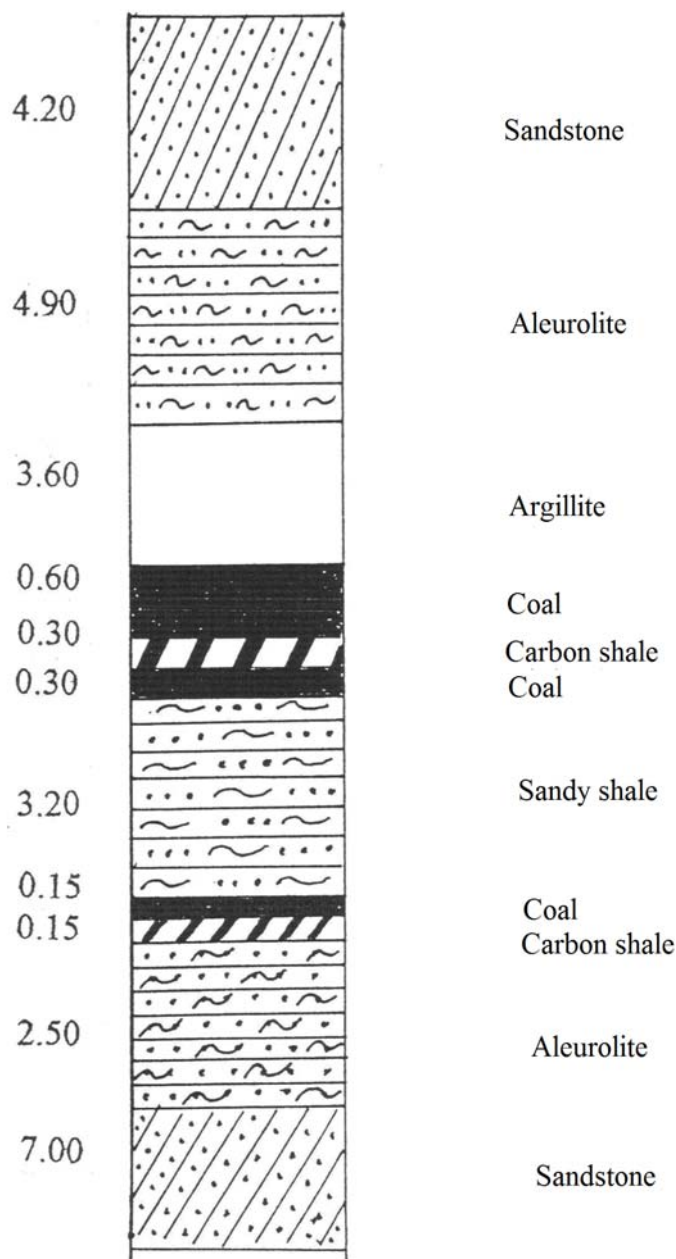


Fig. 1.1. - Typical stratigraphic section of coal-bearing thickness

Table 1.3 gives the data concerning distribution of commercial coal deposits throughout the seams of “Lvovugol” SE mines.

Table 1.3

Commercial coal deposits throughout “Lvovugol” SE productive mine fields

Mines	n_9	n_8^e	n_8	n_7^e	n_7	n_7^u	Total thous.t.
Velykomostivska	0	274	936	81	-	166	1457
Bendiuzka	-	308	638	359	-	333	1638
Mezhyrichanska	1213	865	1286	6284	2094	1191	12933
Vidrodzhennya	2471	989	-	3607	0	510	7577
Lisova	0	3224	0	5750	0	2245	11219
Zarichna	1499	751	0	2324	0	467	5041
Vezeiska	-	-	-	291	206	339	836
Nadiya	-	-	-	-	-	353	353
Stepova	9817	16331	-	7843	1121	8496	43608
Chervonogradska	7272	3257	1898	7574	3901	-	23902
Total	22272	25999	4758	34113	7322	14100	108564

This table shows that n_9 , n_8^e , and n_7^e seams contain maximum reserves. Specific character of their development means repeated underworking and overworking, available selvedges and pillars of contiguous seams resulting in high stress concentration. Period of 1996-2001 covers 47.7% of the area worked by means of 96 longwalls within high rock pressure zones.

More detailed data as for the conditions of contiguous seam mining given in Table 1.2 show that within the period of 1996-2001 81% of longwalls operated within the effect areas of selvedges and pillars, among them 56.4% operated within the areas of high rock pressure due to pillars being left within two-three seams simultaneously.

Under such conditions moderately stable roof becomes an unstable one, we can observe frequent rock outbursts, moreover powered complexes under the use show their inoperativeness.

It is obvious that to provide efficient operation of longwalls within high rock pressure zones it is necessary to determine adequate values of strength and geometrical parameters of powered support as well as to define the most rational advance rate for a stoping face under definite mining and geological conditions. It requires detailed analysis of stress and strain state of layered heterogeneous mass around a stope taking into account both underworking and overworking as well as the available pillars and selvedges within the area under study.

Hence, the next paragraph is dedicated to the survey of methods to study stress and strain state of rock mass in the context of the development of contiguous seams as well as to the analysis of the known results as for the solution of the corresponding problems.

1.3. Development and the current state of the methods to study stress and strain state of a mass in the context of suite mining

Most rock pressure hypotheses are the generalization of the results of field and laboratory studies [5-7, 8, 9]. As for stopes, rock pressure hypotheses can be divided into two main groups. Hypotheses of the first group are based on the concept that the properties of rock masses are described by means of models of elastic, plastic, loose, orthotropic and other media. Calculation methods for such media are mostly numerical. However, calculation schemes do not take into account definite mining and geological conditions that is why the results cannot be used in terms of other conditions [10-15].

Second group consists of the hypotheses and calculation methods that consider certain forms of rock breaking over a stoping zone and regularities of rock pressure manifestation depending on lithological composition and degree of roof rock disturbance. As a rule, these hypotheses tell about slight stope impact area covering mostly the area of broken rock interacting with the support in a stoping face. These are the hypothesis of virtual displacements by G.N. Kuznetsov [16-19], hypothesis of step-by-step rock fault by P.M. Tsymbarevich [19]. Calculation methods by S.T. Kuznetsov [20,21], O. Jakobi, [22-24], M.P. Zborshchik, N.N. Kasian, A.P. Kluev, R.I. Azmatov [25] and others belong to this tendency as well.

Among the known schematics of rock displacement and rock mass pressure under the influence of underground mining the ones belonging to VNIMI, the National Mineral Resources University (Saint Petersburg), and Donetsk National Technical University [26–30] reflect the displacement pattern to the fullest degree. According to these schematics the whole area affected by coal-face operation is divided into the specific zones, each having definite types of rock deformation. Works [27, 31] call them de-stressed zones, zones of uncontrolled breaking, zones of full displacements, abutment pressure zones, or zones of boundary stress and strain state. Determination of the limits of these zones is a very important task as for the practical operations which solution gives the data considering stress and strain rock state around a stope. These data are necessary to determine rational technological and geometrical parameters for a mining method as well as to determine parameters to support workings ensuring proper operation of both stoping faces and transportation and ventilation systems. For example, if we take Western Donbass mines (according to field studies) then we will see that working along an underworked seam results in its discontinuity and heaving. Thickness of the seams affected by heaving can be up to 6...7 m. Taking into account all these data, supports of new technical level have been adopted to be used in Western Donbass to prevent deformations of principal direction workings within stoping effect zones [32-34].

It is possible to single out front, back (relative to a face), and lateral abutment zones within the seam plane. Deformations within front and back zones differ quantitatively. Seam deformations within lateral zones reach their maximum at sufficient distance from a longwall [35].

Certain distribution of abutment zones are not stable depending on definite factors: working depth, processes of deformation and breaking not only of the

immediate and main roof but also of the whole covering thickness, seam properties, floor rocks etc.

During the face advance value of abutment pressure maximum change as well as the distance from it to a face within the certain limits either during the periods between main roof caving or during the periods between complete rock subsidence of the whole covering thickness to the surface. It is explained by the instability of mechanical characteristics of covering thickness rocks, changes in their thickness, fissility etc. Abutment pressure dynamics is stipulated by the dynamics of changes in overhang spans of covering thickness rocks as well as by the deformations and breaking of the selvedge of a stoping face and enclosing rocks (roof and floor); it is also stipulated by such technological parameters as: preparation method, development system, stoping rate, cutting depth, longwall length etc. Breaking probability depends on working depth and correlation between strength properties of coal and enclosing rocks. According to the paper [36], working with similar geometrical parameters of reservoir engineering in soft rocks in terms of strong coal is quite reliable down to the depth of 160...180 m (in case of solid rocks it is down to 500 m).

Width of the front abutment pressure zone varies within 20-250 m while the distance from a face to abutment pressure maximum varies from 0 to 15 m and more. According to some data, width of lateral abutment pressure zone (to the dip and to the rise) is 15...50 m [37].

Development of calculation methods for abutment pressure makes certain difficulties. M.M. Protodiakonov [38] and V.D. Slesarev [39] obtained first solutions for determining the width of lateral abutment pressure zone using the field observation data. Later, other scientists simulated a rock mass by means of homogeneous isotropic elastic semi-plane with horizontal rectangular split at unspecified depth to calculate abutment pressure [40, 41]. Theoretical works by A.M. Linkov and I.N. Petukhov [31,42] and experimental works by B.V. Vlasov, G.I. Gritsko, E. Aizakson, L.N. Gapanovich and others [43-45] are devoted to the study of underworked and overworked rocks in homogeneous and layered masses. They make up the basis for values of shift angles for horizontal and inclined seams being adopted in regulations [46, 47] used while determining floor loading and rock loading surrounding the working. Some papers consider the tasks for geomechanics for underworked and overworked mass areas in terms of studying preparatory stage of dynamic phenomena [48-50]. Solutions are performed on the basis of elastic medium model with the setting of boundary conditions in stresses obtained by in-situ measurements. The obtained results are of great interest. However, it is impossible to use them (as well as the very approach to the solution) while considering the conditions of Lvov region mines as there are no corresponding experimental data.

Papers by A.A. Borisov, N.N. Kaidalov, and V.G. Labazin [51] describe methodology to calculate abutment pressure during the period of roof deformation up to the first fall using Lagrange variation method. Elastic medium model and calculation scheme of multilayered support beam is used. Interlayer friction is assumed to be negligible. As a result epures of abutment pressure distribution depending on the length of stoping span are obtained. Here the length of lateral

abutment pressure zone is taken according to the results of underground measurements.

Studies [52, 53] by the same authors solve the problem of stress calculation in the mined seam floor under selvedges (in case when roof is controlled by smooth lowering) by the method of elasticity theory in a simplified statement. The simplification means that the actual loading (abutment pressure is stipulated by the deformation of layered thickness) is considered to be specified. Stress condition of the floor under a face and in the beginning of roof and floor convergence is determined. Floor rocks are simulated by means of homogeneous isotropic medium. Stress calculations mean solution of the known problem considering the effect upon elastic semi-plane being distributed equally over the definite load fragment. Diagrams of σ_y and τ_{xy} stresses are constructed according to calculation results. Analysis of calculation results has shown that if abutment pressure maximum is assumed as a unit then additional σ_y stress maximum at the depth of 5 m will be 0.5 while it will be only 0.14 at the depth of 25 m. Tangent stress maximum (τ_{xy}) is about $0.3(\sigma_y)_{max}$.

Other conclusions are drawn from the obtained analytic solution as well being interesting from the viewpoint of applications though they belong to the development of a separate seam.

V.G. Gmoshinskyi [54] studied formation of abutment rock pressure epure in front of a stoping face in time taking into account partial coal breaking near a mine working. The most general differential equation of rheological medium state was the initial one [55] written for the case of brittle coal breaking. The author have found both theoretically and experimentally than the length of coal breaking zone depends considerably upon coal strength, sizes of worked-out area, seam thickness, and initial pressure of overlying rocks. Solutions of a plane problem of elasticity theories in closed form for determining rock pressure upon a coal seam depending on stoping face advance rate given in paper [54] are of special interest. They can be used to determine powered support load but only for a stope in homogeneous isotropic mass. Here underworking and overworking issues are not considered having great influence upon the value of stress concentration coefficient within the zone of abutment pressure in front of a face, therefore influencing the load value on roof support canopy. If we determine the initial pressure effecting the area within the mined seam roof by means of solving geomechanical problem for the case of suite development under conditions of underworking and overworking as well as if we determine stress concentration coefficients taking into account available pillars and selvedges within the area under study and put them into analytical relation given in paper [54], then it will be possible to use them while calculating rational parameters of roof bolting in stoping faces in terms of mining and geological conditions under consideration.

Works [56-59] are devoted to the study of regularities of mechanical processes occurring in rock masses around fully-mechanized stoping faces. They contain quantitative estimation (performed by finite element method) of the influence of such factors as mass structure, physical and mechanical rock properties, mining depth, seam thickness, and longwall length upon rock pressure manifestation. However, these papers do not consider suites.

A.A. Borisov reveals physical essence of the phenomenon taken place in the mass while suite mining in his paper [60]. It contains configuration of high and low stress zones in underlying thickness under overworking conditions according to the results of 3-D model tests. Qualitative deformation figure of an overworking seam is given; attention is paid to the peculiarities of rock pressure manifestation under pillars and seam selvedges. Problems of underworking influence are considered including development workings on the upper seam.

Works by O.V. Kolokolov and N.A. Lubenets [61] substantiate permissible underworking boundaries of rather contiguous seams; they also determine rational location of a development working for the conditions of Western Donbass mines.

Works [62, 63] are devoted to mutual influence of contiguous steep seams at different depth; they generalize field observation data performed in mines of Prokopievsk-Kiseliovsk deposits of Kuzbass and Central Donbass. The data show the fact that the degree of unfavorable effect of abutment pressure during the development of contiguous seams increases along with the growth of mining depth; moreover, if at the depth of 100...200 m measurable influence of abutment pressure in case of stoping boundaries overlapping could be observed at the parting of 15...25 m, then at the depth of 300...400 m rock bursts in stoping and development workings even at parting thickness of 50...60 m were observed. It tells about the fact that in each definite case while substantiating parameters for contiguous suite development, stress and strain state of a rock mass around a stoping face should be determined taking into account overworking and (or) underworking for sure as well as taking into account possible stope entering into the area of pillars and seam selvedges being left. Here calculation algorithm should express necessarily specific features of the considered mining and geological conditions as well as mining and technical situations.

Figure of stress distribution around a stope depends for sure upon the distance between a stoping faces within contiguous seams as well. It will be different in terms of underworking and overworking. All these aspects should be bore in mind of researchers while substantiating the order of suite mining for definite mining and geological conditions. Here it is impossible to use only field observations. Corresponding calculation algorithms are required as well.

M.P. Zborshchik and V.V. Vishnevetskiy [64] have developed one of such algorithm. It allows determining stress distribution taking into account thickness of seams, distance between them, stiffness and strength properties of the rocks as well as relative position of mined-out areas and their mining procedure. To take into account all the influencing factors while calculating downfolds equally, the authors involved planning of mathematical experiment according to Latin square method. Each of four determining factors (elasticity modulus and shearing modulus of parting layer, parting thickness and worked-out area length) was specified using 5 values. Their measurement limits were substantiated beforehand taking into account definite mining and geological conditions. However, strength properties of the rock and parameters of the coal seams itself, either geometrical or physical and mechanical were not taken into consideration. Besides, calculation models of the studied mass areas did not contain pillars; influence of seam selvedges was not studied as well.

Influence of pillars and selvages of seams located above upon stress and strain state and stability of layered heterogeneous roof in the longwall of a mined seam allows evaluating calculation algorithm developed by V.V. Nazimko, Yu.B. Griadushchyi, and M.I. Bugara [65]. The authors used it to substantiate parameters of roof bolting within the area of possible inrush formation in case of suite development of contiguous seams in No.1 “Yuzhno-Donbasskaya” mine under overworking conditions. The algorithm involves determining weighted average roof strength in general taking into account not only thickness and strength of a separate layer but also its location (share of each layer strength is inversely proportional to the distance from its rock exposure). This is its difference from traditional method to calculate weighted average roof strength. Unfortunately, the paper does not contain any generalizing dependences; it only gives the data of one definite calculation.

The problem of the influence of pillars and selvages being left upon roof state in stoping faces of contiguous coal seams is not studied enough. The reason is the great number of mining and geological factors as well as mining and technical ones effecting the processes of formation of abutment pressure on pillars and selvages as well as stress transfer through parting onto stope roof. Guidelines for roof control in stoping faces [66-68] represent the influence of such factors as depth of the seam on which pillar or selvedge is left, its mining height, thickness of parting layer, and pillar width.

It is obvious that these factors are not all the determining ones at all. For example, suite can contain more than two contiguous seams; moreover, physical and mechanical properties of coal and enclosing rocks that are not taken into account in the papers can play important role etc. Besides, many conclusions in papers [66-68] are based on the data of field observations that is why they can be applied first of all to the conditions of Karaganda, Pechorsk, and Donetsk basins where such institutes as VNIMI, Donetsk Scientific and Research Coal Institute, KuznUGI, PechorNIiproekt, and KNIUI carried out their experimental studies at the proper time. Application of the given recommendations in Lvov region mines is considered to be irrelevant.

Works [69-71] also generalizes data of the observations in “Donetskugol” PA mines within the period of 1980—1981 as for rock pressure manifestation under pillars and selvages of the seam. Observation methods involved obtaining comparative parameters in stoping faces both within the assumed effect zone and outside it. Subsidence rate of roof, its hanging behind the support, inrush height, and coal spall were taken as such parameters. It is noticed that within high rock pressure zone roof subsidence rate increases sufficiently during the period when there is not operating influence comparing to the subsidence rate outside the zone. It is 1.5...2 times less within the supposed de-stressed zone neighbouring the effect zone. Moreover, number of inrush formations increase within the area of high rock pressure; the height of separate inrushes can reach up to 0.8 m while it is not more than 0.1...0.2 m outside the effect zone. Actual angle of pillar effect ($\sim 84^{\circ}$) is obtained. Maximal roof hanging behind the support (not taking into account end longwall sections) equal to one support advance increment made up 0.5...0.7m; the highest values of spall depth within de-stressed zone made up 0.4...0.5 m.

It is natural that the enumerated quantitative characteristics of the parameters under study as well as the graphs of rock fault rate and the test data of rock strength given in the quoted papers are related to the definite mining and geological as well as mining and technical conditions.

To test and specify the boundaries of high rock pressure zones determined on the basis of normative documents [72], field observations were carried out in mines of Makeievugol, Leninksugol, Chalyabinskugol associations in mines of – VNIMI, Ukrzapadugol – LGI, Yuzhkuzbassugol – KuznNII, Severokuzbassugol – VostNII associations. The work [73] generalized the conclusions considering the results of these observations. They stress that the approach rate of lateral seams increases by 2...5 times in stoping faces for roofs of any class within the areas of high rock pressure. Considerable stability decrease of lower roof layers in stoping faces operating within the effect area of abutment pressure of pillars and selvedges was observed mostly for roof classes characterized in their stability as unstable and extremely unstable. Here inrush height decreases by 3...10 times reaching the whole thickness of unstable rock layers (2...6 m and more), pillars and selvedges effect stable and moderate stable rocks of lower roof layers to a less extent.

Rate decrease of a stoping face advance by 50% and more is the characteristic of intensity of pillars and selvedges effect manifestation in stoping faces (partial or complete longwall gobs, mass choking, support deformation and damage, high inrushes) showing accident conditions of stoping faces in general.

While analyzing experimental data of surveying documentation and field studies it has been found out that in some cases pillars and selvedges effect upon stoping face operation under similar mining and geological conditions (within one mining extracted area) becomes apparent in different ways. Moreover, differences can be so considerable that in one case the effect can be intense while in other case it can be unnoticed. It is explained by the influence of definite factors complicating stoping operations within the areas of high rock pressure, such as support operation on the lower extension limit, poor technical state of powered support hydraulic system, lengthy downtime of a stoping face. These factors cannot only intensify dramatically pillars and selvedges effect but also enlarge the area of their influence [74].

Similar phenomena were observed in the mines of Shakhtiorskugol association as well while mining flat and inclined seams of Almaznaya, Kamennaya, and Smolyaninovskaya suites. Experimental studies has found out that the values and rates of lateral rock convergence within high rock pressure zones are by 1.5...2 times more and coal spall is by 1.7 times more then outside these zones. High rock pressure zone is formed in front of advancing stoping face and at its stationary boundaries; it expands to the roof and floor of a seam. While mining, stress and strain states of both mined and contiguous seams are subjected to repeated changes. Here structural changes (loosening or consolidation) take place. Seam loading within the area of abutment pressure under conditions of non-uniform three-axial compression is accompanied by the development of internal fissility. Intensity of fissure formation process is higher along with the increase of the difference between standard components of stress tensor. Curve zone between high rock pressure zone and de-

stressed zone is the most complicated. It is so-called zone of alternate stresses [75].

Results of stand tests as well as triaxial compression coal samples tests give clear idea of the character of structural changes, fissility development or solidification. The paper [76] finds out that the decrease of horizontal stress at the boundary of abutment pressure and de-stressed zone results in the failure of the initial coal structure at the expense of microdefects development. Coal and brittle rocks (sandstones, sand argillites, and aleurolites) at the boundary of abutment pressure and de-stressed zones tend to have dramatic (impact-like) breaking characteristics. Coal seams with higher humidity show some plasticity.

Thus, while predicting possible effect of pillars and selvedges upon the operation of stoping faces within contiguous coal seams, in any single case it is necessary to take into account peculiarities of mining and geological as well as mining and technical working conditions able to change considerable the character and degree of the effect. Hence, it is required to consider adopted effect zones described in normative documentation. First, it concerns algorithms under development to solve corresponding problems of geomechanics.

The available algorithms and methods to calculate parameters of development systems use mostly numerical method of finite elements (MFE). MFE application in mining geomechanics, while considering physical non-linear media as well, is described in detail in monographs by B.Z. Amusin, A.B.Fadeev [77, 78]. Zh.S. Yerzhanov, T.D. Karimbaev [79. 80]. Yu.M. Liberman [81-83] and others. The abovementioned papers by S.N. Komissarov also belong to this area of study.

O.V. Kolokolov and N.A. Lubenets in the paper [84] established advantages of ascending mining sequence of quite contiguous seams in terms of Western Donbass by finite element method using the model of elastic and plastic medium.

It should be noted that despite wide MFE possibilities as for consideration of structural and physical heterogeneity of the objects under study its use for analysis of stress and strain state of a layered media as applied to the development of approached coal suites cannot be considered as rational one, i.e. it is necessary to discretize the whole, sufficiently large mass area. Development of the initial equation system and its solution here require a lot of machine time. Hence, another numerical method – boundary element method (BEM) is becoming more and more popular now. It provides discretisation not of the whole area but only of its boundaries; owing to this principle time for input information preparation is reduced as well as the solution of the developed equation system. Thus, it is possible to solve the widest range of different and complex problems of geomechanics [85-93].

Boundary element method is more preferable in the thesis work as well.

As for problems of geomechanics, here it is very important not only to determine the pattern of stress and strain state within the mass area under study but also to analyze it using one or another rock strength criterion. It is most often considered that the actual maximum stresses should not exceed ultimate rock compression and tensile strength. Mohr's strength condition with straight and curved envelopes is also popular among the researchers [94]. Famous criterion by P.P. Balandin expressing to the full extent medium properties being resistant differently to extension and compression belong to this area of study [95, 111]. This is the specific

property of rocks; that is why the paper uses this very criterion. As a result, we can draw following conclusions:

1) Difficulty of obtaining reliable physical and mechanical characteristics of rocks, their dependence upon stress and strain state, compelled idealization of real complex mining and technical conditions does not allow obtaining efficient solution as for rock pressure manifestation in terms of suite development of contiguous coal seams by purely analytical methods;

2) Known results of field and laboratory experiments are inconsistent under different mining and geological as well as mining and technical conditions;

3) The available solutions obtained by numerical and experimental and analytical methods most often do not contain generalizing dependences; they express determined conditions; that is why they have limited area of application;

4) Substantiation of parameters for reservoir engineering of contiguous seam suite required thorough analysis of stress and strain state of layered heterogeneous mass with the areas of high and low rock pressure taking into account underworking and overworking; areas subjected to the influence of stoping face, pillars, and selvages of contiguous seams require special attention;

5) Calculation algorithms developed for definite stresses and displacements within the abovementioned areas should take into account structure and geometrical dimensions of the studied mass area including bearing elements and zones of overworked and (or) underworked rocks, physical and mechanical characteristics of both coal and partings, mining depth, time factor, and stope advance rate.

All these factors have predetermined statement of the defined research problems.

1.4. General methodology of the research

Theoretical, applied, and experimental studies of stress and strain state of heterogeneous layered mass in terms of suite development of contiguous coal seams under complex mining and geological conditions were carried out to complete the task and solve the defined problems. Fig. 1.2 represents structural scheme of research methodology.

Theoretical studies included analysis of mining and geological as well as mining and technical conditions, analysis of the available approaches to the solutions of related problems, selection of method to determine and estimate stress and strain state of the studied mass area, and substantiation of calculation schemes. Calculation algorithm developed on the basis of boundary element method (BEM) and analytical solution by V.G. Gmshinskiy using the criterion by P.P. Balandin are the result of theoretical research.

In contrast to known ones, the developed algorithm allows determining powered support load during powered complexes operation within the zones of high rock pressure due to the effect of pillars (selvages) of contiguous seams under conditions of underworking and overworking. In this case, stress concentration within the abutment pressure area in front of a face and longwall advance rate are taken into account.

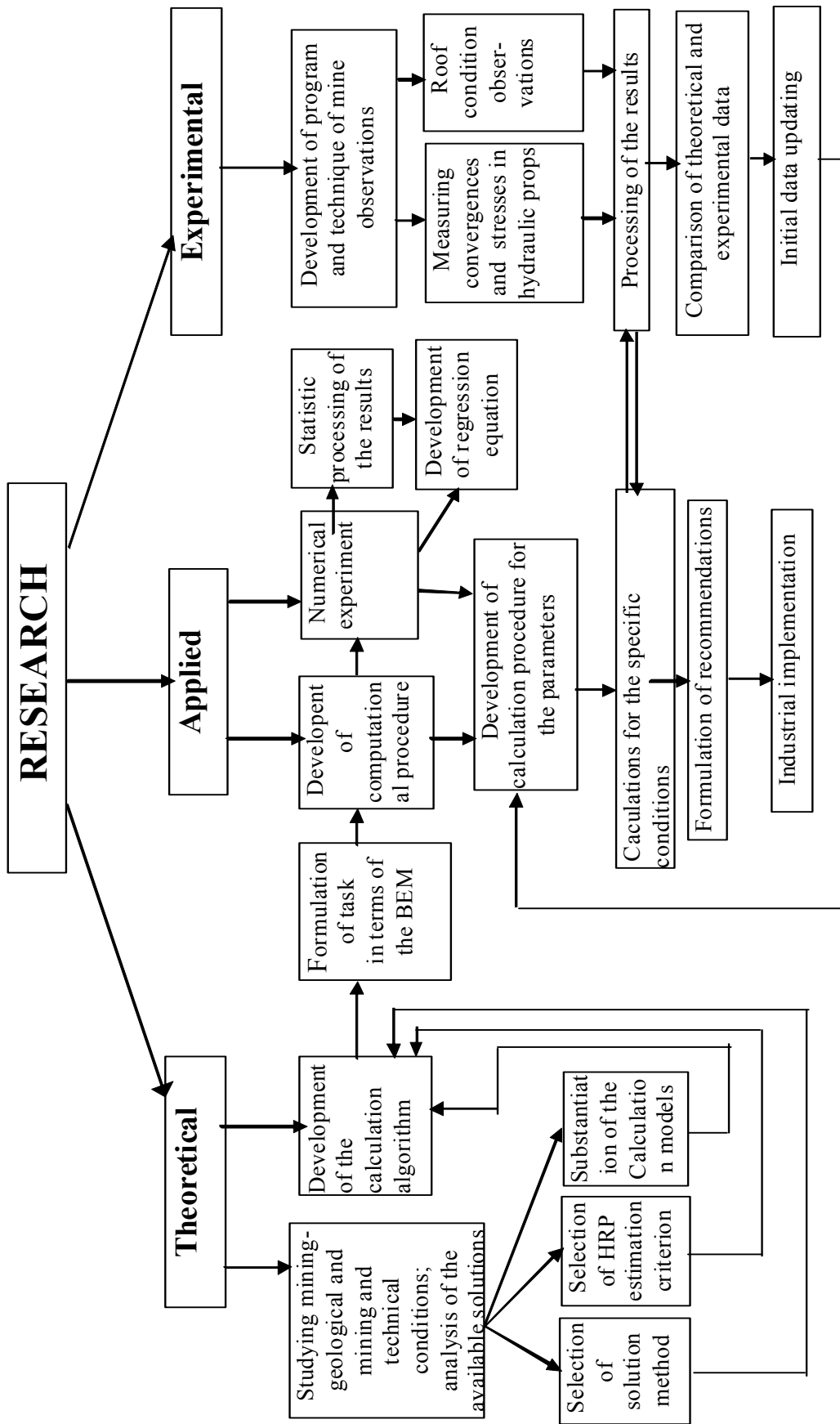


Fig. 1.2. – Structural scheme of research methodology

Applied research means developing calculation programs to implement the developed algorithm and perform numerical experiment. The essence of numerical experiment is in multiple repetitions of calculations at different values determining parameters and performing multiple correlation analysis of the results.

Finally, correlations to determine maximum coefficients of σ_{yy} stress concentrations in front of a stoping face relative to γH level and maximum convergence in a longwall within high rock pressure areas due to the effect of pillars (selvedges) of contiguous seams under overworking and underworking conditions.

The determined correlations have made the basis of the developed methodology to calculate rational parameters of stoping operation technology.

Experimental research includes the development of program and methodology to carry out mine observations.

According to the program, stresses in hydraulic props of powered support during operation of mechanized complexes within high rock pressure areas due to the effect of pillars (selvedges), convergence and rate of rock convergence in a longwall were determined; state of selvedges of a seam and roof rocks were observed; frequency, intensity and volume of inrush formation at different rates of shearer feed and longwall advance were determined.

Statistic processing of experimental data and their comparison with corresponding theoretical values were followed by clarification of the initial data for the calculation, in particular, calculation schemes for canopy during support operation within high rock pressure areas and outside them; corresponding corrections were introduced into the methodology of calculation for stoping operation parameters.

Rational parameters for conditions of Lvov region mines were determined according to the updated methodology: resistance of hydraulic props to powered support; shearer feed rate; longwall advance rate; admissible distance between a stoping face operating within contiguous seams.

The research was completed by the formulation of recommendations as for parameters for mining methods to develop contiguous seams in terms of Lvov-Volynsk basin and their implementation in “Lvovugol” SE mines.

CHAPTER 2. ALGORITHM TO CALCULATE STRESSES AND DISPLACEMENTS WITHIN A ROCK MASS IN TERMS OF THE DEVELOPMENT OF FLAT COAL SEAM SUITE

2.1. Coal seam roof pressure within homogeneous mass In terms of fixed stope

Stress changes in a coal seam roof in time for homogeneous seam is studied by V.G. Gmoshinskiy and described in work [54]. Introduce the analytical correlation from this work being used in the developed algorithm of numerical solution of the defined problems.

Following formula is used for normal σ_y stresses in terms of motionless неподвижном stope

$$\sigma_y(t) = Ce^{\frac{fx}{h\lambda}} + \left[\gamma H \left(1 + Ke^{-\frac{2x}{l_1}} \right) - Ce^{\frac{fx}{h\lambda}} \right] \cdot e^{-\frac{t}{T}}, \quad (2.1)$$

where C is coal adhesion, MPa;

$f = tg\varphi$ is the coefficient of coal-coal friction;

φ is the angle of coal internal friction;

$\lambda = \nu/(1-\nu)$ is the coefficient of lateral reaction;

ν is coal Poisson's ratio;

$h = m/2$ is a half of coal seam thickness, m;

x is the coordinate of the considered section measured from a stope line into the mass depth, m;

l_1 is the size of the worked-out area along Ox axis not filled with broken rock, m;

γ is roof rock density, t/m^3 ;

H is mining depth, m;

t is the time counted from the beginning of longwall operation, hours;

$T = 5...6$ hours is rheological coefficient characterizing time of stress relaxation in terms of coal breaking (it is determined experimentally by V.G. Gmoshinskiy under mine conditions);

$K = (K_y)_{max} - 1$;

$(K_y)_{max}$ is the maximum coefficient of stress concentration relative to γH level in front of a stope.

Paper [54] determines value of $(K_y)_{max}$ coefficient roughly depending on the correlation of elasticity moduli of rock and coal according to stress diagrams around rectangular single working within a ponderable homogeneous elastic medium [96,97].

The monograph defines coefficients of stress concentration being the result of numerical solution of geomechanics problem for a heterogeneous mass taking into account overworked and underworked rocks, pillars and selvages of contiguous coal seams being available within the area under study. The algorithm developed for this purpose as well as calculation model under use is described in the following paragraphs of this chapter.

Formula (2.1) is obtained resulting from the most general differential equation of rheological state [55]

$$T \frac{d\sigma_t}{dt} + \sigma_t = E_y \varepsilon + \tilde{m} \frac{d\varepsilon}{dt}, \quad (2.2)$$

where E_y is coal elasticity modulus;

ε is relative linear deformation;

\tilde{m} is coal toughness.

Stress σ_t , being the parts of (2.2) equation is determined by the congruence

$$\sigma_t = (\sigma_y - \sigma_{yp}) \cdot \psi(t) \quad (2.3)$$

Initial elastic stresses (σ_y) are specified by the correlation

$$\sigma_y = \gamma H \left(1 + Ke^{-\frac{2x}{l_1}} \right), \quad (2.4)$$

where σ_{yp} is stresses into which σ_y stresses are transformed at the expense of coal loosening (Fig. 2.1).

Formula to be used for their definition is as follows

$$\sigma_{yp} = Ce^{\frac{fx}{h\lambda}}. \quad (2.5)$$

Paper [54] designates σ_t stresses as rock pressure being “removed” from a seam due to its partial breaking; $\psi(t)$ function is designated as time function.

Formulas (2.4) and (2.5) for σ_y and σ_{yp} are obtained from the solution of a differential equation of limit equilibrium for dx surface element of a coal seam being at x distance from a face edge under condition that $x = 0$, $\sigma_{yp} = C$ (σ_{yp} stress on a seam edge is equal to coal adhesion).

$\psi(t)$ time function is obtained as follows.

If $T \neq 0$ and $\frac{d\varepsilon}{dt} = 0$ (brittle failure of the material) rheological state equation (2.2) will be

$$T \frac{d\sigma_t}{dt} + \sigma_t = E_y \varepsilon. \quad (2.6)$$

Assuming ε as total relative deformation corresponding to transformation of σ_y stresses into σ_{yp} ones we obtain such congruence as

$$E_y \varepsilon = \sigma_y - \sigma_{yp} = \sigma_0.$$

Then equation (2.6) will be:

$$T \frac{d\sigma_t}{dt} + \sigma_t = \sigma_0. \quad (2.7)$$

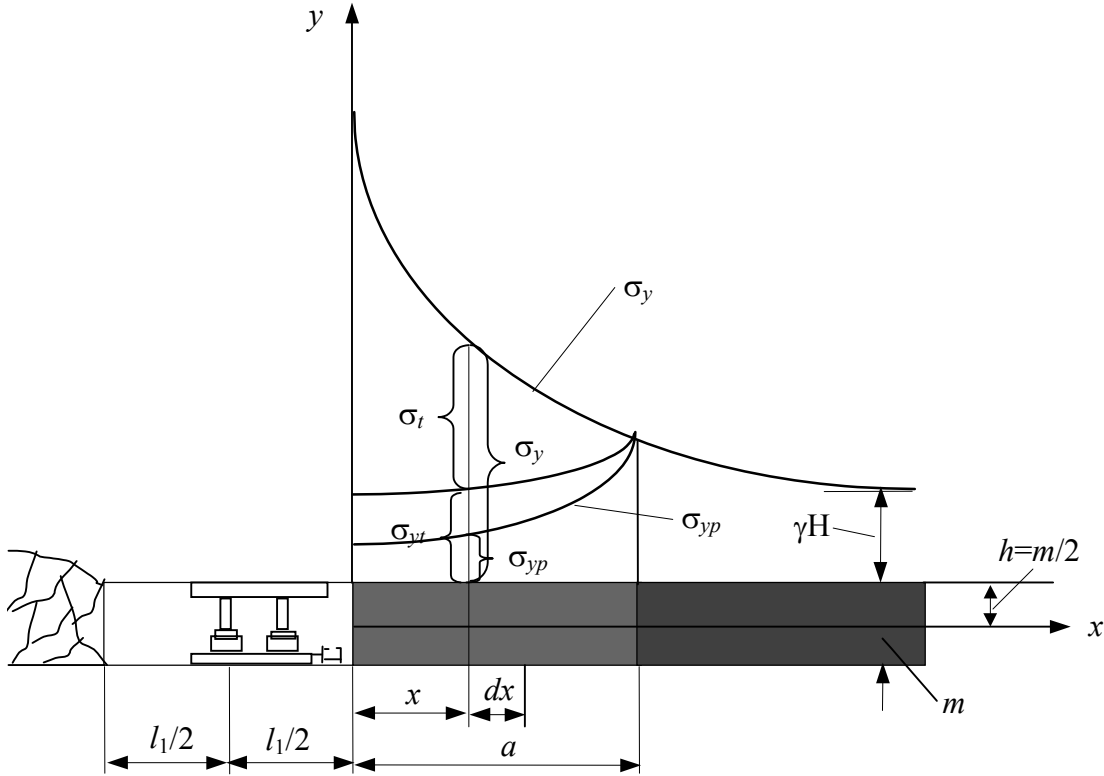


Fig. 2.1. – Curve of the abutment pressure in front of a stope.
 a is the length of a broken coal zone;
 l_1 is the size of a worked-out area along Ox axis .

General solution of differential equation (2.7) is as follows

$$\sigma_t = e^{-\frac{t}{T}} \left(\sigma_0 e^{\frac{t}{T}} - C_1 \right).$$

The initial condition $t = 0, \sigma_t = 0$ shows that arbitrary constant $C_1 = \sigma_0$.

Taking this fact into account, specific solution of differential equation (2.7) is

$$\sigma_t = \sigma_0 \left(1 - e^{-\frac{t}{T}} \right) = (\sigma_y - \sigma_{yp}) \left(1 - e^{-\frac{t}{T}} \right). \quad (2.8)$$

Comparing (2.3) and (2.8), we obtain the expression for time function

$$\psi(t) = 1 - e^{-\frac{t}{T}}. \quad (2.9)$$

Rock pressure measuring in both time and relation with partial coal breaking within a zone (Fig. 2.1) can be represented as

$$\sigma_y(t) = \sigma_y - \sigma_t$$

or in the following form taking into account expression (2.8)

$$\sigma_y(t) = \sigma_y - (\sigma_y - \sigma_{yp}) \left(1 - e^{-\frac{t}{T}} \right). \quad (2.10)$$

Putting σ_y and σ_{yp} expressions into this correlation (according to (2.4) and (2.5) formulas) we obtain formula (2.1) representing the whole curve spectrum (Fig.2.1) of rock pressure change in time beginning from a seam uncovering ($t = 0$) up to the complete pressure stabilization corresponding theoretically to $t = \infty$ time and in any $0 \leq t < \infty$ time interval within $0 \leq x \leq a$ interval.

Paper [54] obtain a size of the area of coal breaking (coal spall) roughly taking into account mining depths, coal seam thickness and its f , C , and v physical and mechanical characteristics using coefficient of concentration according to A.N. Dinnik and G.N. Savin. The thesis represents a as the distance being measured into the depth from a stope line to the maximum of the abutment pressure; it is determined as a result of solution of a plane problem of geomechanics for a suite of contiguous seams taking into account underworking and overworking conditions.

Putting expressions (2.4) and (2.5) instead of σ_y and σ_{yp} into equation (2.8) we will have:

$$\sigma_t = \left[\gamma H \left(1 + K e^{-\frac{2x}{l_1}} \right) - C e^{\frac{fx}{h\lambda}} \right] \cdot \left(1 - e^{-\frac{t}{T}} \right). \quad (2.11)$$

To obtain formula for determining $P(t)$ partial pressure being “removed” from the seams due to coal partial loss of its bearing capacity and being time function as well, we integrate correlation (2.11) on a variable x within the limits from 0 to a :

$$\begin{aligned} P(t) &= \int_0^a \sigma_t dx = \left(1 - e^{-t/T} \right) \int_0^a (\sigma_y - \sigma_{yp}) dx = \\ &= \left\{ \gamma H \left[\frac{l_1}{2} K e^{-2a/l_1} - \frac{l_1}{2} K - a \right] + \frac{h\lambda C}{f} \left(e^{fa/h\lambda} - 1 \right) \right\} \left(1 - e^{-t/T} \right). \end{aligned} \quad (2.12)$$

According to mine observations, most part of $P(t)$ pressure (about 97%) is taken up by a pillar and rock mass [107] while only $q(t) = 0.03P(t)$ is transferred onto powered support in the form of additional load.

Total powered support load is obtained by summing $q(t)$ value with P_0 pressure being produced by rocks contacting immediately with the support while

developing a stope in a rock mass. The latter is determined by boundary element method using the developed algorithm.

Separate paragraph of this chapter is devoted to the calculation technique of the actual loading and determination of the corresponding reactions of powered support hydraulic props.

2.2. Coal seam pressure in homogeneous mass in terms of displacing face

In case of displacing face, coal breaking and corresponding stress transformation occur simultaneously. Fig. 2.2 shows schematic of face line displacement.

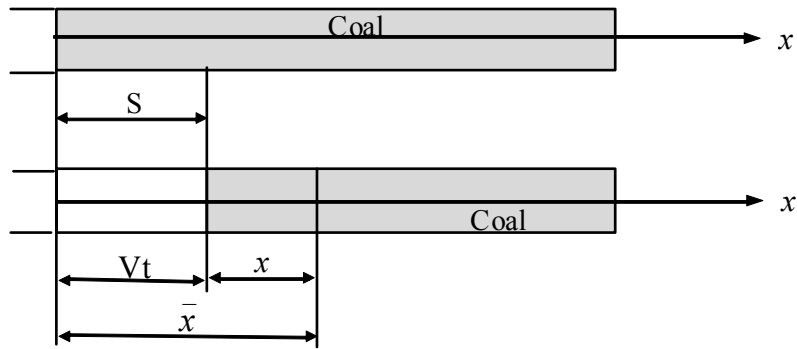


Fig. 2.2. – Schematic of stope displacement

This figure shows that

$$\bar{x} = Vt + x,$$

where \bar{x} is still the distance from face to the specified point;

V is face advance rate, m/hour;

t is time, hour.

This ratio gives
$$t = \frac{\bar{x} - x}{V}.$$

Mark that the difference $\bar{x} - x$ is a web mined per t time at V face advance rate. Define size of this web as S , then

$$t = \frac{S}{V}. \quad (2.13)$$

Hence, if longwall advances by x distance, then corresponding stress deformation will occur per t time. Replacing t with its value from (2.13) in expression (2.1) instead of t we obtain the formula characterizing rock pressure change in a roof depending on the rate of stope line displacement:

$$\sigma_y(V) = Ce^{\frac{fx}{h\lambda}} + \left[\gamma H \left(1 + Ke^{-\frac{2x}{l_1}} \right) - Ce^{\frac{fx}{h\lambda}} \right] \cdot e^{-\frac{S}{VT}}. \quad (2.14)$$

In particular, if $x = 0$ from (2.14) on a face edge, then we have

$$\sigma_y(V)\Big|_{x=0} = C + [\gamma H(1+K) - C] \cdot e^{-\frac{S}{VT}}. \quad (2.15)$$

Pressure being “removed” from a seam in terms of displacing face can be found from the expression (2.12), introducing t from the formula (2.13) into it:

$$P(V) = \left\{ \gamma H \left[\frac{Kl_1}{2} e^{-2a/l_1} - \frac{Kl_1}{2} - a \right] + \frac{h\lambda C}{f} \left(e^{fa/h\lambda} - 1 \right) \right\} \left(1 - e^{-S/VT} \right). \quad (2.16)$$

According to expressions (2.1) and (2.12), coal-breaking mechanism in time can be explained as follows. Working advance is accompanied by partial coal-breaking expressed in coal layer separation over cleavage plane and displacement towards a mine working. While mining next web, breaking process is quite fast with the following gradual attenuation. Rock pressure decreases in front of a face along with coal breaking.

Coal breaking results in irreversible deformation, so-called coal spall. Coal breaking, which means spall, will be maximum on a face surface, i.e. right where the difference of σ_y and σ_{yp} is the highest one (Fig. 2.1). There is almost no coal breaking in the center of abutment pressure zone (in section $x = a$) and the spall is equal to zero.

When the process of coal breaking finishes then the spall stops with simultaneous stress decrease within a breaking zone.

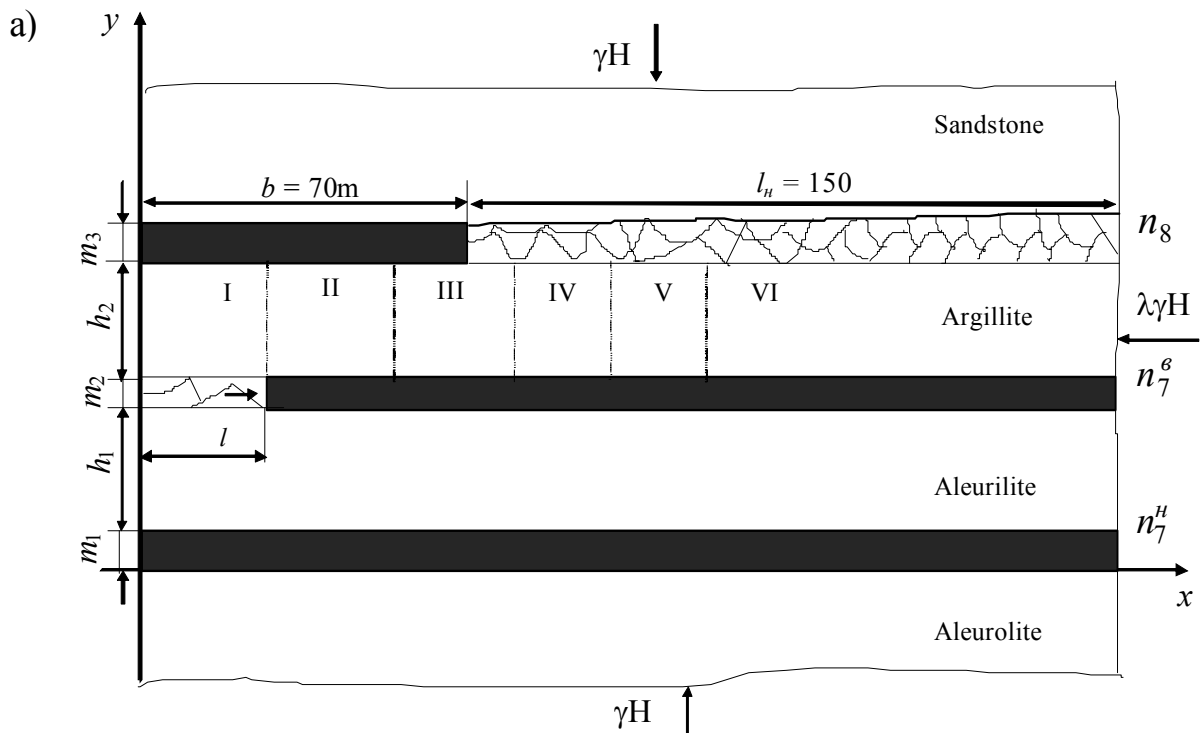
2.3. Calculation models to determine coefficients of stress concentrations in layered heterogeneous mass and powered support load

Value and character of stress distribution and displacements in a rock mass during suite extraction depend on a variety of factors. The most important factors among them are as follows: thickness of coal seams and partings, physical and mechanical properties of coal and enclosing rock, mining depth, available overworked or underworked rock, pillars, and seam selvages within the studied area. All these factors should be taken into account in the algorithm used to determine stress concentration coefficient, therefore, if possible, they should be represented in calculation models. Fig. 2.3 shows two models of such type being typical for “Lvovugol” SE mining conditions. They include n_7^h , n_7^g , and n_8 mines.

Stoping operations are carried out within n_7^g seam with $m = 0.6...0.7$ m thickness at $H = 520$ m depth. m_1 and m_3 thicknesses of n_7^h and n_8 seams are equal to 1.3...1.6 m and 1.4...1.6 m correspondingly; thickness of h_1 and h_2 partings varies within 12...16 m and 9...12 m. The models allow researching condition of n_7^g seam roof and floor when stoping approaches selvedge of n_8 seam located above

Calculations are supposed to be performed at different values of distances from a stope on the mined n_7^6 seam up to the selvedge of n_8 seam located above or up to the selvedge of n_7^H seam located under (positions I to VIII in Fig. 2.3 a and 2.3 b). Cases of longwall outbursts from under selvedges of contiguous seams were considered apart from the cases of longwall approaches to the selvedges of contiguous seams within the zones of these very selvedges to determine the most rational stopping direction (Fig. 2.4 a, b, positions I–VI of a stope).

Calculation models represented in Fig. 2.5 are aimed at evaluating pillar effect upon stress and strain state of roof and floor of the seam under development when pillar is over the seam (Fig. 2.5 a) and under the seam (Fig. 2.5 b).



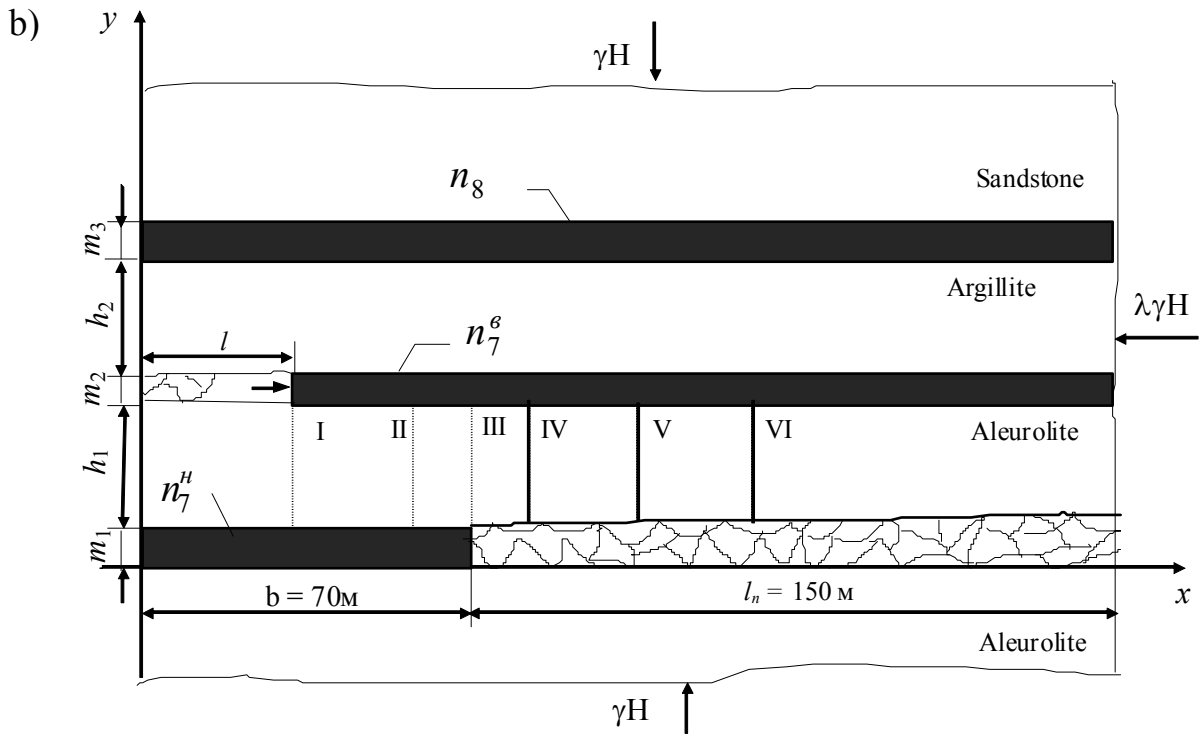


Fig. 2.4. – Calculation models at longwall outburst out of the zone of contiguous seam selvedge:

- a) operation within n_7^6 seam under overworking conditions;
b) operation within n_7^6 seam under underworking conditions.

The paper analyzes stress and strain state of the studied mass zone in terms of different mutual distances of a stope and pillar: within face-pillar boundary approach zone (position I) the face line coincides with pillar line (position II); the face line coincides with pillar midsection (position III); the face is under second pillar boundary (position IV); and the face is behind the pillar (position V).

Rock weigh is the actual loading in all the described calculation models: in vertical direction at infinity – γH , in horizontal direction – $\lambda \gamma H$. Roof and floor stability of the mined seams is estimated using the criterion by P.P. Balandin [95] that is as follows

$$\sigma_{\text{акв}} = \frac{(1-\psi)(\sigma_1 + \sigma_3) + \sqrt{(1-\psi)^2(\sigma_1 + \sigma_3)^2 + 4\psi(\sigma_1^2 + \sigma_3^2)}}{2} \leq [\sigma], \quad (2.17)$$

where σ_1 and σ_3 are the highest and lowest principal stresses, MPa;

$$\psi = \sigma_p / \sigma_c;$$

σ_p and σ_c are rock ultimate tensile and compression strength, MPa;

$[\sigma]$ is permissible stress equal to $\bar{\sigma}_p/\kappa_3$ within tensile zones and $\bar{\sigma}_c/\kappa_3$ within compression zones ($\bar{\sigma}_p$ and $\bar{\sigma}_c$ are average values of rock ultimate tensile and compression strength, the data are according to Table 1.1; κ_3 is a strength factor).

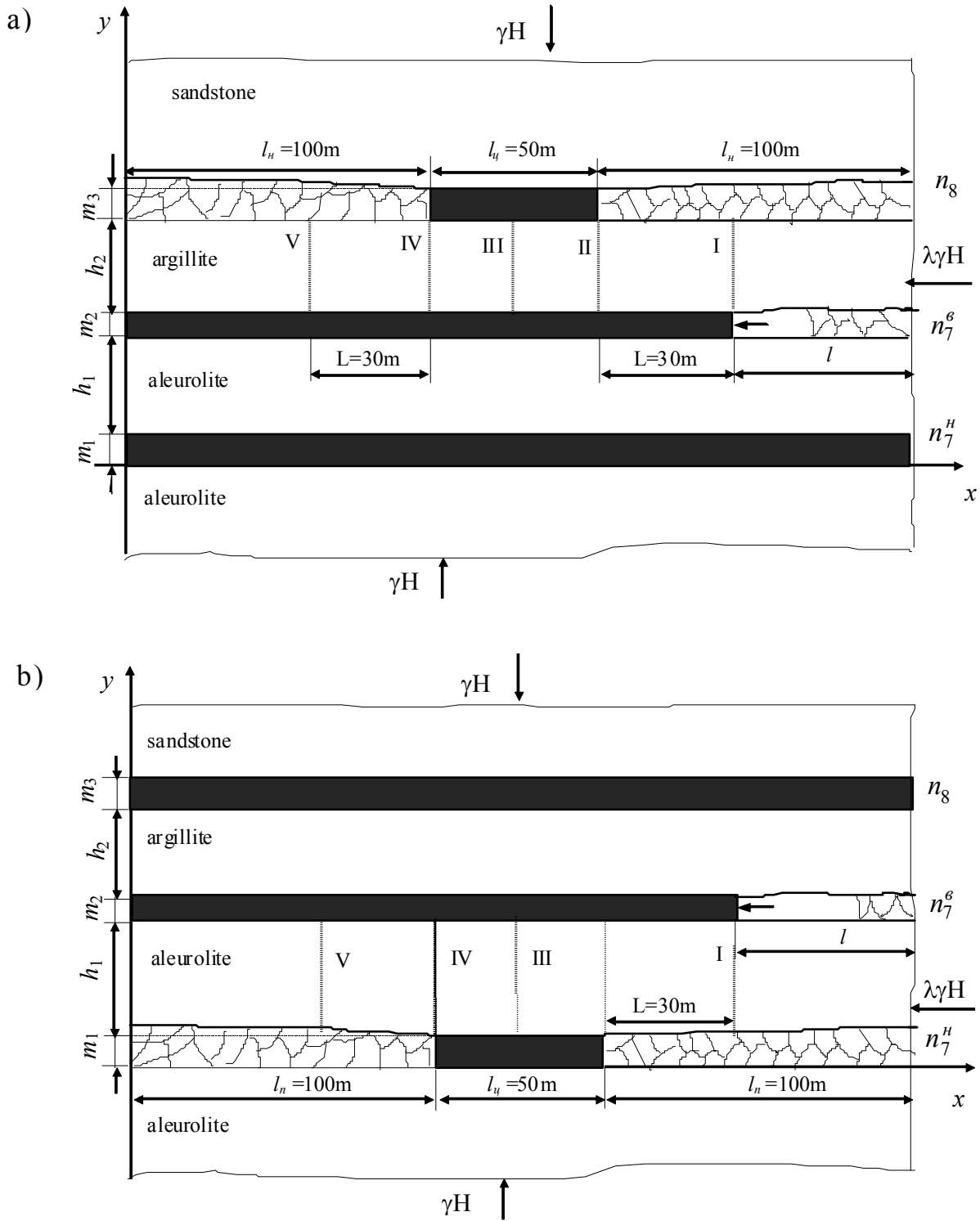


Fig. 2.5. – On pillar effect estimation:
a) pillar above a stope;
b) pillar under a stope.

In particular, criterion (2.17) is used to determine the boundaries of ultimate stress state under the longwall. Rock weight within this zone develops P_0 load upon powered support. As it is mentioned before it should be added up with $q(V)$ additional load equal to $0.03P(V)$ being determined taking into account time factor according to formula (2.16).

2.4. Problem statement for heterogeneous mass in terms of boundary element method

The problem of determination of stress and strain state of the mass area under study using models proposed in the previous paragraph is solved by boundary element method in its modification that is usually called pseudo-load method [89]. Analytic solution of the known Kelvin's problem is the basic one; it concerns the effect of normal and tangential forces being equally distributed within the interval oriented randomly in infinite elastic medium [98].

Following calculation types are basic ones:

- approximation of the boundaries of the area under study with the help of finite elements and introduction of fictitious P_n normal and P_s tangential forces into each of them;
- specification of boundary conditions of free edge elements and conditions of strain compatibility in the elements of adjacent “roof-pillars” and “pillars-floor” edges;
- obtaining of analytical correlations (using principal solution) for stresses and displacements in each of boundary elements (including adjacent ones) expressed by means of P_n and P_s fictitious forces;
- calculation using the obtained correlations of stress and displacement effect coefficients; development of equation system to determine P_n and P_s forces according to the specified boundary conditions and strain compatibility conditions;
- solution of the obtained system of algebraic equations (Gaussian method is used in the developed algorithm), calculation of P_n and P_s forces ensuring meeting the specified boundary conditions as well as strain compatibility conditions;
- calculation of stresses and displacements within the area under study due to the obtained forces and specified load which allow fulfilling the specified conditions.

Explain the technique of developing initial equation system by the example of the arbitrary shape working [89, p.66-70]. Correlation for the displacement towards a normal to i -th u_n^i boundary element as well as towards u_s^i tangential to it is expressed as

$$\left. \begin{aligned} u_n^i &= \sum_{j=1}^N B_{ns}^{ij} P_s^j + \sum_{j=1}^N B_{nn}^{ij} P_n^j \\ u_s^i &= \sum_{j=1}^N B_{ss}^{ij} P_s^j + \sum_{j=1}^N B_{sn}^{ij} P_n^j \end{aligned} \right\}. \quad (2.18)$$

Correlations for the corresponding stresses of i -th element are as follows

$$\left. \begin{aligned} \sigma_n^i &= \sum_{j=1}^N A_{ns}^{ij} P_s^j + \sum_{j=1}^N A_{nn}^{ij} P_n^j \\ \sigma_s^i &= \sum_{j=1}^N A_{ss}^{ij} P_s^j + \sum_{j=1}^N A_{sn}^{ij} P_n^j \end{aligned} \right\}. \quad (2.19)$$

In correlations (2.18) and (2.19) N is the number of boundary elements; B_{ss}^{ij} , B_{sn}^{ij} , B_{ns}^{ij} , B_{nn}^{ij} are coefficients of displacement effect; A_{ss}^{ij} , A_{sn}^{ij} , A_{ns}^{ij} , A_{nn}^{ij} are coefficients of stress effect.

According to the used analytical solution, effect coefficients are determined using formulas [88, 110]

$$\begin{aligned} B_{ss}^{ij} &= \frac{1}{2G} [(3-4\nu)\cos\gamma\bar{F}_1 - \bar{y}(\sin\gamma\bar{F}_2 - \cos\gamma\bar{F}_3)], \\ B_{sn}^{ij} &= \frac{1}{2G} [(3-4\nu)\sin\gamma\bar{F}_1 - \bar{y}(\cos\gamma\bar{F}_2 + \sin\gamma\bar{F}_3)], \\ B_{ns}^{ij} &= \frac{1}{2G} [-(3-4\nu)\sin\gamma\bar{F}_1 - \bar{y}(\cos\gamma\bar{F}_2 + \sin\gamma\bar{F}_3)], \\ B_{nn}^{ij} &= \frac{1}{2G} [(3-4\nu)\cos\gamma\bar{F}_1 + \bar{y}(\sin\gamma\bar{F}_2 - \cos\gamma\bar{F}_3)], \\ A_{ss}^{ij} &= -2(1-\nu)(\sin 2\gamma\bar{F}_2 - \cos 2\gamma\bar{F}_3) - \bar{y}(\sin 2\gamma\bar{F}_4 + \cos 2\gamma\bar{F}_5), \\ A_{sn}^{ij} &= (1-2\nu)(\cos 2\gamma\bar{F}_2 + \sin 2\gamma\bar{F}_3) - \bar{y}(\cos 2\gamma\bar{F}_4 - \sin 2\gamma\bar{F}_5), \\ A_{ns}^{ij} &= \bar{F}_2 - 2(1-\nu)(\cos 2\gamma\bar{F}_2 + \sin 2\gamma\bar{F}_3) - \bar{y}(\cos 2\gamma\bar{F}_4 - \sin 2\gamma\bar{F}_5), \\ A_{nn}^{ij} &= \bar{F}_3 - (1-2\nu)(\sin 2\gamma\bar{F}_2 - \cos 2\gamma\bar{F}_3) + \bar{y}(\sin 2\gamma\bar{F}_4 + \cos 2\gamma\bar{F}_5) \end{aligned} \quad (2.20)$$

In formulas (2.19) and (2.20):
 G is a rock shearing modulus;

$$\bar{F}_1(\bar{x}, \bar{y}) = f(\bar{x}, \bar{y});$$

\bar{x}^i, \bar{y}^i are local coordinates connected with i -th element;

$$f(\bar{x}, \bar{y}) = -\frac{1}{4\pi(1-\nu)} \left[\bar{y} \left(\arctg \frac{\bar{y}}{\bar{x} - a^j} - \arctg \frac{\bar{y}}{\bar{x} + a^j} \right) - \right.$$

$\left. - (\bar{x} - a^j) \ln \sqrt{(\bar{x} - a^j)^2 + \bar{y}^2} + (\bar{x} + a^j) \ln \sqrt{(\bar{x} + a^j)^2 + \bar{y}^2} \right]$ is the function characterizing effect of fictitious loads applied to j -th element on stresses and i -th element displacements;

$$\bar{F}_2(\bar{x}, \bar{y}) = \frac{\partial f}{\partial \bar{x}}; \bar{F}_3(\bar{x}, \bar{y}) = \frac{\partial f}{\partial \bar{y}}; \bar{F}_4(\bar{x}, \bar{y}) = \frac{\partial^2 f}{\partial \bar{x} \partial \bar{y}}; \bar{F}_5(\bar{x}, \bar{y}) = \frac{\partial^2 f}{\partial \bar{x}^2} = -\frac{\partial^2 f}{\partial \bar{y}^2};$$

$\gamma = \beta^i - \beta^j$ is the angle determining mutual position of $\bar{x}^i \bar{y}^i$ local coordinate systems of i -th element and $\bar{x}^j \bar{y}^j$ of j -th element;

$2a^j$ is the length of j -th element.

$$\begin{aligned} \bar{x} &= (\bar{x}^i - \bar{x}^j) \cos \beta^j + (\bar{y}^i - \bar{y}^j) \sin \beta^j \\ \bar{y} &= -(\bar{x}^i - \bar{x}^j) \sin \beta^j + (\bar{y}^i - \bar{y}^j) \cos \beta^j \end{aligned} \quad (2.21)$$

Formulas (2.21) determine local coordinates of i -th element center relative to j -th element center.

Own influence coefficients characterizing effect of P_n^i and P_s^i fictitious loads of i -th element upon the displacement and stress of the very i -th element are obtained according to the expressions (2.19) and (2.20), if we introduce $\bar{x} = \bar{y} = \gamma = 0$ into them, and they will be as follows

$$\begin{aligned} B_{sn}^{ij} = B_{ns}^{ii} = 0; \quad B_{ss}^{ii} = B_{nn}^{ii} &= -\frac{3-4\nu}{4\pi G(1-\nu)} a^i \ln a^i; \\ A_{sn}^{ii} = A_{ns}^{ii} = 0; \quad A_{ss}^{ii} = A_{nn}^{ii} &= \frac{1}{2}; \\ A_{ts}^{ii} = A_{tn}^{ii} = 0; \quad A_{tt}^{ii} &= \frac{1}{2} \cdot \frac{\nu}{1-\nu}; \quad \bar{y} = 0 \pm. \end{aligned} \quad (2.22)$$

2.5. Problem statement for heterogeneous mass in terms of boundary element method Development of the initial equation system

Consider development of the initial equation system to calculate components of P_n^i and P_s^i “fictitious” loads by the example of calculation model expressed in Fig. 2.3 b. It includes six areas: C_1 is rock mass in the floor of n_7^H seam; C_2 is the selvedge of n_7^H coal seam; C_3 is the worked-out area filled with broken rock; C_4 is parting; C_5 is n_7^G coal seam; C_6 is the roof of n_7^G seam. Boundaries of C_κ ($\kappa = \overline{1, 6}$) areas are approximated using N_κ elements correspondingly. In our case $N_1 = 44$, $N_2 = 29$, $N_3 = 61$, $N_4 = 154$, $N_5 = 131$, and $N_6 = 110$. Then $N = \sum_{\kappa} N_\kappa = 529$.

Element numbers and pass-by directions of area boundaries for the considered model are given in Fig. 2.6. Contiguous contours in this figure are conventionally expanded.

In general, the initial system of algebraic equations with $2N$ unknown components of “fictitious” loads representing specified boundary conditions on free edges of the studied areas as well as conditions of strain compatibility on contiguous ones is as follows:

$$\left. \begin{aligned} \sum_{j=1}^N C_{ss}^{i,j} P_s^j + \sum_{j=1}^N C_{sn}^{i,j} P_n^j &= b_s^i \\ \sum_{j=1}^N C_{ns}^{i,j} P_s^j + \sum_{j=1}^N C_{nn}^{i,j} P_n^j &= b_n^i \end{aligned} \right\}_{i=1,2,\dots,N} \quad (2.23)$$

Boundary conditions b_s^i , b_n^i and coefficients C_{ss}^{ij} , C_{sn}^{ij} , C_{ns}^{ij} , C_{nn}^{ij} are determined in the following way.

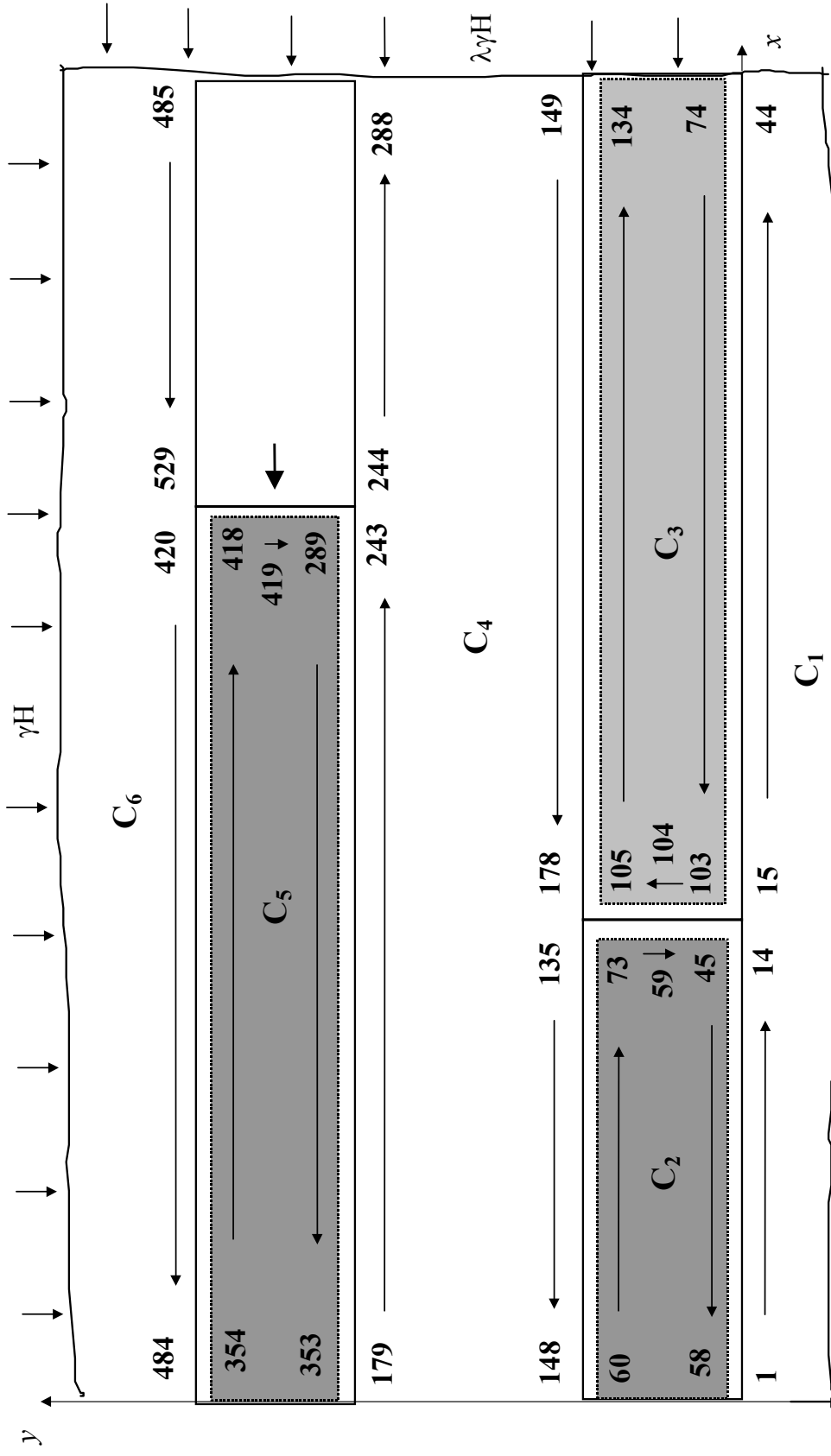


Fig. 2.6. - Finite element boundary approximation

a) If i -th element is on a free contour of C_κ ($\kappa = \overline{1, 6}$) areas, then

$$b_s^i = (\sigma_s^i)_0^\infty, \quad b_n^i = (\sigma_n^i)_0^\infty,$$

$$C_{ss}^{i,j} = \begin{cases} A_{ss}^{i,j}, & \text{if } j\text{-th element belongs to} \\ & \text{contour of } C_k \text{ area} \\ 0, & \text{if } j\text{-th element does not belong to} \\ & \text{contour of } C_k \text{ area} \end{cases}$$

$$C_{sn}^{i,j} = \begin{cases} A_{sn}^{i,j}, & \text{if } j\text{-th element belongs to} \\ & \text{contour of } C_k \text{ area} \\ 0, & \text{если if } j\text{-th element does not belong to} \\ & \text{contour of } C_k \text{ area} \end{cases}$$

(2.24)

$$C_{ns}^{i,j} = \begin{cases} A_{ns}^{i,j}, & \text{if } j\text{-th element belongs to} \\ & \text{contour of } C_k \text{ area} \\ 0, & \text{если if } j\text{-th element does not belong to} \\ & \text{contour of } C_k \text{ area} \end{cases}$$

$$C_{nn}^{i,j} = \begin{cases} A_{nn}^{i,j}, & \text{if } j\text{-th element belongs to} \\ & \text{contour of } C_k \text{ area} \\ 0, & \text{если if } j\text{-th element does not belong to} \\ & \text{contour of } C_k \text{ area} \end{cases}$$

b) If i and i^* are the numbers of contacting elements belonging to different areas, then taking into account strain compatibility conditions specified in stresses (contiguous contour of C_κ area) and in displacements (contiguous contour of C_l area) when $\kappa > l$ we have correspondingly:

$$b_s^i = \sigma_s^{i[\kappa]} - \sigma_s^{i^*[l]} = 0; \quad b_n^i = \sigma_n^{i[\kappa]} - \sigma_n^{i^*[l]} = 0;$$

$$C_{ss}^{i,j} = \begin{cases} A_{ss}^{i,j[k]}, & \text{if } j\text{-th element belongs to} \\ & \text{contour of } C_k \text{ area} \\ -A_{ss}^{i*,j[l]}, & \text{if } j\text{-th element belongs to} \\ & \text{contour of } C_l \text{ area} \end{cases}, \quad (2.25)$$

similarly for $C_{sn}^{i,j}$, $C_{ns}^{i,j}$, and $C_{mn}^{i,j}$ coefficients.

$$b_s^i = u_s^{i[l]} + u_s^{i*[k]} = 0; \quad b_n^i = u_n^{i[l]} + u_n^{i*[k]} = 0;$$

$$C_{ss}^{i,j} = \begin{cases} B_{ss}^{i*,j[k]}, & \text{if } j\text{-th element belongs to} \\ & \text{contour of } C_k \text{ area} \\ B_{ss}^{i,j[l]}, & \text{if } j\text{-th element belongs to} \\ & \text{contour of } C_l \text{ area} \end{cases} \quad (2.26)$$

etc. for $C_{sn}^{i,j}$, $C_{ns}^{i,j}$, and $C_{mn}^{i,j}$ coefficients.

For example, for C_1 and C_2 contours in Fig. 2.6 we obtain
 $1 \leq i \leq 14$, $i^* = 58, 57, \dots, 45$

$$b_s^i = \sigma_s^{i[1]} - \sigma_s^{i*[2]} = 0; \quad b_n^i = \sigma_n^{i[1]} - \sigma_n^{i*[2]} = 0;$$

$$C_{ss}^{i,j} = \begin{cases} A_{ss}^{i,j[1]}, & j \leq 14 \\ -A_{ss}^{i*,j[2]}, & 45 \leq j \leq 73 \end{cases}$$

etc. for $C_{sn}^{i,j}$, $C_{ns}^{i,j}$, and $C_{mn}^{i,j}$ coefficients.

For $45 \leq i \leq 58$, $i^* = 14, 13, \dots, 1$

$$b_s^i = u_s^{i[2]} + u_s^{i*[1]} = 0; \quad b_n^i = u_n^{i[2]} + u_n^{i*[1]} = 0;$$

$$C_{ss}^{i,j} = \begin{cases} B_{ss}^{i*,j[1]}, & j \leq 14 \\ B_{ss}^{i,j[2]}, & 45 \leq j \leq 73 \end{cases} \quad \text{etc.}$$

Solving the system of $2N$ linear algebraic equations (2.10) obtained like that, we determine P_s^i and P_n^i forces; then, according to the superposition principle, we determine displacements and stresses at any point of the studied mass area using formulas (2.18) and (2.19).

2.6. Coefficients of stress concentration around stope while suite developing

Level of underworking and overworking influence upon mass stress and strain state while extracting flat seam suite in terms of “Lvovugol” SE was estimated according to the value of K_y maximum concentration coefficient in the roof and floor of the developed seam within the effect zone of pillars and contiguous seam selvages.

Calculations were performed using the algorithm described above according to the models shown in Fig. 2.3 and 2.5 under the conditions of the following initial data: m_1 , m_2 , and m_3 thicknesses of n_7^H , n_7^6 , and n_8 seams make up 1.45 m, 0.65 m, and 1.5 m correspondingly; mining depth of n_7^6 seam under development is $H = 520$ m; parting thicknesses are $h_1 = 16$ m and $h_2 = 13$ m; physical and mechanical characteristics of coal and enclosing rocks were taken according to the data from Table 1.1.

In calculation models 2.3 a, b and 2.4 a, b dimension of seam selvedge within n_7^6 seam above (or under) a stope is $b = 70$ m; overworking and underworking zones have the same dimensions: $l_H = l_{II} = 150$ m.

Worked-out area length l behind a stope within n_7^6 seam took values of 50, 100, 120, 140, and 150 m in the calculations.

In such a case L length from a stope to the selvages of contiguous seams was 100 m, 50 m, 30 m, 10 m, and 0 m (face under the selvedge boundary), -10 m, -30 m, and -50 m (face under the seam selvedge). Part of a worked-out area ($l - l_1$) was filled with broken rock. l_1 parameter was represented by a caving step of the immediate mine roof determined from the condition

$$(\Delta u_y)_{max} \leq [\Delta u_y]. \quad (2.27)$$

In this condition $(\Delta u_y)_{max}$ is the maximum convergence of the stope roof and floor behind the longwall obtained by the calculation; $[\Delta u_y]$ is the permissible Δu_y value specified as for mine observation results taking into account technical possibilities of the used powered support (displacements of roof and floor rocks occurring in a longwall should not prevent from normal operation of the stoping equipment).

$E_p = 10$ MPa modulus of elasticity and $\nu_p = 0.499$ Poison's ratio for the broken rocks within $(l - l_1)$ area were taken according to operation schedules [31, p.49],

developed in ε_y and k_μ coordinates for different values of mining depth (ε_y is the maximum relative linear stope deformation of in the vertical direction; $k_\mu = 1 - \frac{2v^3}{1-v}$).

While calculating using 2.5 a and 2.5 b models, five different positions of face line relative to the boundaries of the pillar located above and below were singled out: when approaching it ($L = 30$ m, position I), right under (above) the boundary ($L = 0$, position II), under (above) pillar midsection ($L = 25$ m, position III), under (over) the second boundary ($L = 0$, position IV), and behind the pillar ($L = 30$ m, position V).

In calculations l_y pillar width was equal to 50 m and dimensions of l_H overworking and l_Π underworking zones on both sides of the pillar were 100 m.

Set of calculations to determine different face positions within n_7^6 seam allowed detecting changes of stress and strain state within the studied mass area along with stoping advance as well as determining boundaries of high rock pressure zones necessary to substantiate stoping parameters under considered mining conditions.

Typical diagrams of σ_y normal stress as well as diagrams of u_y vertical displacements in the roof and floor of the developed n_7^6 seam constructed according to the results of one of calculation variants on the basis of Fig.2.3 and in case if $L = 50$ m when approaching to the selvedge (overworking conditions) are shown in Fig. 2.7 a and b.

Analysis of this calculation variant has shown that high rock pressure zone (exceeding γH level) in the roof of the extracted n_7^6 seam in horizontal direction in front of a stope covers 12 m. In vertical direction, it is as long as 3.5 m in the roof and 5 m in the floor. In addition roof rocks behind a stope within l_1 area are in the boundary state of stress (strength condition according to (2.17) criterion is violated).

Pressure maximum (of σ_{yy} stress) within high rock pressure zone in front of a longwall falls on the section located at the distance of $a = 1$ m from a face along Ox axis; pressure maximum within high rock pressure zone behind a longwall falls on the section located at 3 m distance from a face along Ox axis.

P_0 load on a powered support unit being developed by the weigh of the rocks within boundary state of stress zone on l_1 length is 114.7 kN/m.

While underworking when a face is at the distance of $L = 50$ m from the edge of the underlying seam when approaching this edge, length of high rock pressure zone is 14 m along Ox axis and expands along Oy axis over 5 m in the roof (it is 31.2% from h_2) and over 5.5 m in the floor (it is 40.6% from h_1). In this case, P_0 load on a powered support unit is equal to 127.8 kN/m.

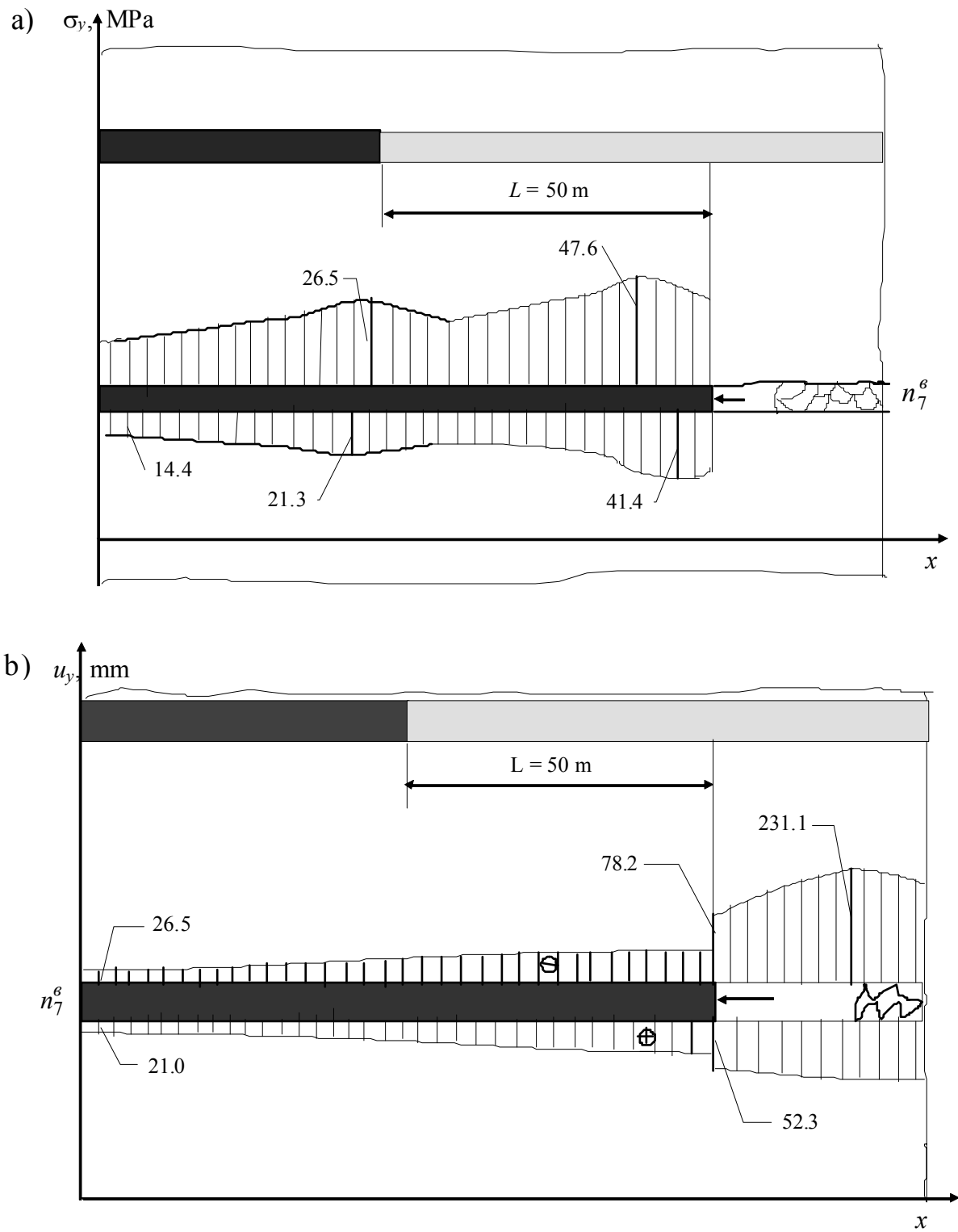


Fig. 2.7. – Diagrams of normal stress distribution and vertical displacements in the roof and floor of n_7^6 seam (overworking conditions)

Tables 2.1 and 2.2 represent the data of all calculation variants as for the estimation of the selvedge effect of contiguous seams upon stress and strain state of the roof and floor of the developed n_7^6 seam under overworking and underworking conditions. These are maximum values of the $(K_y)_{max}$ stress concentration coefficient within the abutment pressure zone in front of a stope. These are maximum roof and floor convergences in the longwall and values of P_0 load from rock weigh (being within the zone of boundary state of stress above the longwall) effecting powered support unit.

Table 2.1 contains the data concerning the case of longwall displacement towards selvedges; Table 2.2 contains the data concerning the case of stope outburst from under the selvedges.

Analysis of the results has shown that in terms of displacement towards the selvedges under overworking conditions maximum $(K_y)_{max}$, stress concentration coefficient equal to 4.35 occurs at the distance of $L = -30$ m (position VII, Fig. 2.3a and b). In this case P_0 powered support load is 135 kN/m. On the other side in position VI ($L = -10$ m) maximum belongs to $P_0 = 137.7$ kN/m support load with $(K_y)_{max}$ stress concentration coefficient equal to 4.23, i.e. less than in the first case.

Quite the same situation is in case of underworking just with the difference that the very values of $(K_y)_{max}$ stress concentration coefficients and P_0 power load during underworking are a bit higher. In this case, the most dangerous positions are IV and VI ones. Therefore, while determining rational strength parameters of powered support it is necessary to consider all these positions and select the most dangerous one. Table 2.1 represents corresponding data in bold.

Longwall displacement in the opposite direction is the reason for the most unfavourable conditions for a support; like in previous cases, the highest coefficients of concentration and stress upon powered support arise under underworking conditions (positions V and VI in Fig. 2.4 a and b) being represented in bold in Table 2.2. These are the positions to be taken into account while determining rational parameters for powered support operation within the effect zones of contiguous seam selvedges.

Table 2.3 gives the results of the analysis of stress and strain state (SSS) of the developed n_7^6 seam roof and floor under (above) a pillar.

As this table shows positions III ($L = 25$ m) and IV ($L = 0$ m) are the most unfavourable for powered support.

Calculation results have shown that under overworking conditions in this case the length of high rock pressure zone in horizontal direction (along Ox axis) in front of a stope is 25 m. It extends in the vertical direction (along Oy axis) 5.5 m upwards in the roof and 3 m downwards in the floor.

Table 2.1

Results of SSS analysis for the roof and floor of the developed n_6 seam under (above) selvedges of the contiguous seams
(in case of longwall displacement towards selvedges)

Stope position	Distance from the face to selvedge boundaries, L , m	Under overworking conditions			Under underworking conditions		
		Stress concentration coefficient in front of the face, $(K_y)_{max}$	Convergence in the longwall, $(u_y)_{max}$, mm	Load on powered support unit, P_0 , kN/m	Stress concentration coefficient in front of the face, $(K_y)_{max}$	Convergence in the longwall, $(u_y)_{max}$, mm	Load on powered support unit, P_0 , kN/m
when approaching to the selvedge	I	2.65	173.1	76.1	2.72	167.3	80.4
	II	3.40	231.5	114.7	3.26	253.3	127.8
	III	3.51	285.4	115.6	3.67	240.1	135.4
	IV	3.59	251.3	123.6	3.85	248.8	141.0
	V	3.86	238.0	126.7	4.13	248.2	137.2
	VI	4.23	237.0	137.7	4.33	245.5	111.4
	VII	4.35	246.6	135.0	4.35	240.9	81.3
	VIII	4.33	244.7	126.0	4.34	247.0	76.1
under (above) the selvedge							

Table 2.2

Results of SSS analysis for the roof and floor of the developed n_7^6 seam under (above) selvedges of the contiguous seams
(in case of longwall outburst from under selvedges)

Stope position	Distance from the face to selvedge boundaries, L , m	Under overworking conditions			Under underworking conditions		
		Stress concentration coefficient in front of the face, $(K_y)_{max}$	Convergence in the longwall, $(u_y)_{max}$, mm	Load on powered support unit, P_0 , kN/m	Stress concentration coefficient in front of the face, $(K_y)_{max}$	Convergence in the longwall, $(u_y)_{max}$, mm	Load on powered support unit, P_0 , kN/m
under (above) the selvedge	I	2.35	120.0	85.5	2.35	120.1	84.1
	II	2.96	159.0	106.2	2.98	159.8	108.1
	III	3.56	197.9	136.3	3.60	199.8	103.2
When leaving From under the selvedge	IV	3.55	212.1	150.0	3.55	207.8	182.0
	V	3.70	244.6	169.2	3.69	229.1	215.7
	VI	3.57	242.6	167.5	3.63	231.3	222.3

Table 2.3

Results of SSS analysis for the roof and floor of the developed n_7^6 seam under (above) pillars

Stope Position	Distance from the face to pillar boundaries, L , m	Under overworking conditions			Under underworking conditions		
		Stress concentration coefficient in front of the face, $(K_y)_{max}$	Convergence behind face, $(u_y)_{max}$, mm	Load on powered support unit, P_0 , kN/m	Stress concentration coefficient in front of the face, $(K_y)_{max}$	Convergence behind face, $(u_y)_{max}$, mm	Load on powered support unit, P_0 , kN/m
When approaching the pillar	I	2.81	195.9	72.4	2.88	190.2	108.6
	II	3.04	196.7	99.7	3.26	203.1	136.1
Under (above) the pillar	III	3.88	222.1	108.6	3.90	223.9	144.8
Behind the pillar	IV	4.35	277.2	106.4	4.34	268.5	142.0
	V	4.40	314.1	90.3	4.44	295.6	126.4

Roof rocks in front of a stope with the dimensions of 2...4 along Ox axis and with the dimensions of 1.5...2 m along Oy axis are in the boundary state of stress according to (2.17) criterion. Broken rock weigh above the longwall creates $P_0=168.4$ kN/m, powered support load and if $L = 0$ m then $P_0 = 165$ kN/m at $L = 25$ m distance.

Underworking (longwall is above the pillar) shows the following situation. Length of high rock pressure zone (dimension along Ox axis) in front of a stope is 35 m. Its vertical length (along Oy axis) in the roof is 6 m while in the floor it is 3 m. Powered support load at $L = 25$ m is 224.5 kN/m and if $L = 0$ then $P_0 = 220.4$ kN/m.

In all the considered cases a distance from a stope into the depth down to the section in which there is maximum concentration of σ_{yy} stresses, i.e. length of coal selvedge spall, is about 1 m, like in case of longwall operation near contiguous seam selvedges.

Thus, all the necessary values to calculate rational strength powered support parameters during its operation within high and low rock pressure zones as well as to determine longwall advance rate and permissible distance between the faces of contiguous seams have been obtained.

2.7. Stress and strain states of rocks around development working while suite mining

Ensuring operation conditions for the whole network of mine workings from the surface to the stopes with minimum expenses for their development and maintenance are the most important conditions of safe coal mining operation. Importance of ensuring development workings with operational stability under conditions of contiguous seam mining grows constantly due to the increase of the mining depth. Moreover, underworking and overworking have great influence upon rock pressure pattern. That is why when selecting the way of protection and fixation for mine workings, especially in case of soft enclosing rocks, it is necessary to take into account the analysis of mass state and strain state considering specific mining situations and conditions.

Layered heterogeneous medium in which stress and strain state of the roof and floor of the contiguous seams have various changes under the influence of mining operations is the typical one for Lvov-Volynsk basin mines. However, the branch documents regulating location, protection, and maintenance of mine workings are drawn up according to VNIMI regulations[67] in which this change is taken into account roughly using empiric coefficients specified mostly for mining conditions of Donetsk, Kuznetsk, and Pechorsk deposits. Similar approach is used while describing rock mass structure.

This paragraph uses abovementioned developed calculation model to evaluate the degree of underworking effect upon stress and strain state of the layered mass around in-seam development working in terms of Lvov-Volynsk basin.

to select support, is one of the reasons of development working deformation. Thus, calculation according to the developed algorithm improves reliability of design solutions as for ways and means of supporting in-seam development workings while suite mining.

Conclusions

1. Boundary element method is the basis for the developed efficient calculation algorithm to determine stress and strain state of heterogeneous mass in the context of the development of flat contiguous coal suites. Contrary to the known ones, the developed algorithm takes into account pillars and selvages available within the considered area under overworking or underworking conditions.

2. The developed algorithm for “Lvovugol” SE conditions has helped to do the following: to determine boundaries of high rock pressure zones in the roof and floor of the extracted seam; to define values of maximum coefficients of σ_y stress concentrations relative to γH level; and to find out the load effecting powered support unit due to the weigh of rock within boundary state of stress area in the roof under the longwall.

3. Formula (2.12) of V.G. Gmoshinskiy solution while considering $(K_y)_{max}$ maximum coefficients of stress concentration in front of a stope and boundaries of a coal spall area determined according to the developed algorithm helps determine analytically additional powered support load depending upon longwall displacement rate and being stipulated by partial coal breaking near seam selvedge.

4. The results of mathematical experiment has shown that under conditions of “Lvovugol” SE mines, the most dangerous period from the viewpoint of rock pressure manifestation is the one when longwall outbursts from under the selvedge of the seam located below while stoping within the areas of contiguous seam selvedge effect. In particular, powered support turns out to have maximum load when the face is at the distance of 5...10 m from the selvedge. That is why rational strength parameters of powered support should be determined according to the characteristics of stress and strain state right of this mass area.

5. It is determined that if the width of pillar being left is up to 50 m, support operation mode under underworking conditions is the most dangerous one when longwall is under pillar midsection approaching pillar selvedge (at the distance of $L = 25$ m).

The positions are the determining ones while calculating strength parameters of the powered support under use.

6. It is defined that when mining depth increases from 300 m to 600 m, u_y maximum displacements in the development working roof under underworking conditions increases by 1.79 times as well. The result should be taken into account while selecting support type.

CHAPTER 3. SUBSTANTIATION OF THE PARAMETERS FOR CONTIGUOUS SEAM MINING IN TERMS OF LVOV REGION MINES

3.1. Determination of high rock pressure boundaries in partings while stoping within the area of contiguous seam selvages

Stress and strain state of the studied mass area changes during stoping displacement. Depending on the dimensions of the worked-out space and the distance from contiguous seam selvages in the roof and floor of the developed seam, there occur areas of high rock pressure of more or less extended length with different stress degrees in them.

The most dangerous position of a stope line is the one when epures of abutment pressure due to the effect of a stope and selvages of the seam located above or below superimpose on one another in the roof and floor of the developed seam. As a rule, total stresses arising at that exceed permissible values [100]. If in this case quite long zone of boundary stress state appear above the worked-out area, then powered support is under high rock pressure. Situation is deteriorated by the fact that at high concentration of vertical normal stresses within the abutment pressure area $\sigma_y(V)$ stress in front of the advancing stope develops considerable additional load stipulated by coal spall ((2.12) and (2.16) formulas). As a result, there can be emergency situation.

Normal operation of powered support within high rock pressure zone due to the effect of contiguous seam selvedge requires such value of $(K_y)_{max}$ stress concentration coefficient in front of a stope and such longwall advance rate that specific resistance per 1 m² of the supported roof contact area as well as the resistance within the most loaded support will not exceed permissible values determined according to technical characteristics of the support being used. Moreover, maximum longwall displacements should not be more than the admissible support extension.

Tables 2.1 and 2.2 gives values of $(K_y)_{max}$ maximum stress coefficient concentration within the areas of high rock pressure in front of a stope, $(\Delta u)_{max}$ maximum convergence of the roof and floor in the worked-out area, and P_0 loads obtained using boundary element method on the basis of calculation models of Fig. 2.3 a,b and 2.4 a,b for different values of L distance between a stope and selvages of the contiguous seams.

Since physical and chemical characteristics of coal and bedding rocks vary within wide range except determined calculation which result is expressed in Tables 2.1 and 2.2, calculations were performed according to the developed algorithms in terms of different values of the determining parameters [101]; the obtained data were subjected to correlation analysis using the known technique [102].

On the basis of possible boundaries of variable value changes (see paragraph 1.1) $\gamma_n H / (\sigma_c)_y$ ratio in the calculation took [0.25; 0.27; 0.29; 0.31; 0.33; 0.35] values

and E_n/E_y ratio belonged to [1.8; 2.8; 4.0] area (E_n, E_y are moduli of elasticity of roof and floor rock; γ_n is roof rock density; $(\sigma_c)_y$ is coal ultimate compression strength).

Geometrical parameter of the studied l/l_n area (Fig. 2.3 a,b and 2.4 a,b) adopted [0.033; 0.166; 0.33; 0.66] values. Here the length of underworking zone was unchangeable making up $l_n = 150$ m and the dimension of the worked-out area behind a stope was $l \in [5; 25; 50; 100]$ m. Width of seam selvedge was equal to 70 m, h_2 parting thickness varied within the range of 5...25 m. Generally, 152 calculation variants have been considered for underworking conditions. Multivariate correlation analysis of the obtained data was the basis to determine following correlations for maximum values of roof and floor convergences in the longwalls in shares from seam thickness for the case of longwall displacement towards contiguous seam selvedge under overworking conditions.

$$\frac{\Delta u_y}{m} = 0,107 + 0,097 \frac{l}{l_n} - 0,065 \frac{E_n}{E_y} + 0,530 \frac{\gamma_n H}{(\sigma_c)_y} \quad (3.1)$$

And maximum relative stresses in the developed seam roof in front of the face is

$$(K_y)_{max} = \frac{(\sigma_{yy})_{max}}{\gamma_n H} = 2,42 + 1,81 \frac{l}{l_n} - 0,46 \frac{E_y}{E_n}. \quad (3.2)$$

(3.1) and (3.2) coefficients of dependence correlation make up 0.884 and 0.82 respectively.

Similar dependences used for the cases of underworking are as follows:

$$\frac{\Delta u_y}{m} = -0,311 + 0,115 \frac{l}{l_n} + 0,364 \frac{E_y}{E_n} + 0,541 \frac{\gamma_n H}{(\sigma_c)_y} \quad (3.3)$$

and

$$(K_y)_{max} = \frac{(\sigma_{yy})_{max}}{\gamma_n H} = 2,17 + 1,82 \frac{l}{l_n} + 0,001 \frac{E_y}{E_n}. \quad (3.4)$$

(3.3) and (3.4) coefficients of dependence correlation have similar values equal to 0.97.

Besides, following correlation dependences for the case of longwall outburst from under the selvedge are determined under overworking conditions:

$$\frac{\Delta u_y}{m} = -0,314 + 0,350 \frac{l}{l_h} + 0,310 \frac{E_y}{E_n} + 0,446 \frac{\gamma_n H}{(\sigma_c)_y}, \quad (3.5)$$

$$(K_y)_{max} = \frac{(\sigma_{yy})_{max}}{\gamma_n H} = 2,38 + 2,468 \frac{l}{l_h} - 0,196 \frac{E_y}{E_n}. \quad (3.6)$$

Following dependences were obtained for the case of longwall outburst from under contiguous seam selvedge under conditions of underworking:

$$\frac{\Delta u_y}{m} = -0,280 + 0,301 \frac{l}{l_n} + 0,285 \frac{E_y}{E_n} + 0,433 \frac{\gamma_n H}{(\sigma_c)_y}, \quad (3.7)$$

$$(K_y)_{max} = \frac{(\sigma_{yy})_{max}}{\gamma_n H} = 2,40 + 2,50 \frac{l}{l_n} - 0,25 \frac{E_y}{E_n}. \quad (3.8)$$

Correlation coefficients of (3.5), (3.6), (3.7) and (3.8) dependences are equal to 0.97, 0.88, 0.97, and 0.89 correspondingly.

Similar studies were carried out for the case when extraction is performed simultaneously under overworking and underworking conditions. Corresponding correlation ratios for $\Delta u/m$ и $(K_y)_{max}$ are represented in work [101].

To illustrate this Fig. 3.1 and 3.2 show diagrams of $(K_y)_{max}$ values changes; Fig. 3.3 and 3.4 represent diagrams of P_0 values depending upon L stope distance from the seam selvedge.

Fig. 3.5 and 3.6 represent calculation results of one of the variants ($L = 65$ m) in terms of longwall displacement towards the contiguous seam selvedge in case of underworking in the form of epures of σ_y normal stresses and u_y vertical displacements. In this case $(\Delta u_y)_{max}$ displacements in a longwall do not exceed the permissible ones for KM87 powered support.

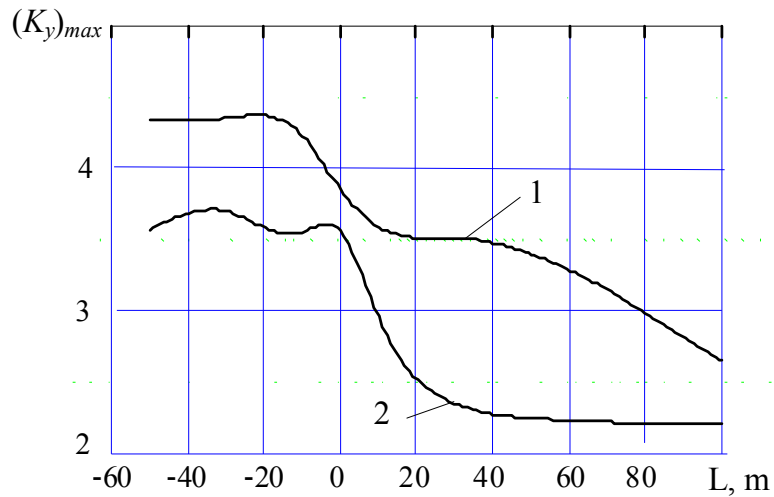


Fig. 3.1. – Change of maximum coefficient of σ_{yy} stress concentration due to the distance between the longwall and selvages of contiguous seams under overworking conditions:
 1 – in terms of longwall displacement towards contiguous seam selvedge;
 2 – in terms of longwall outburst from under contiguous seam selvedge.

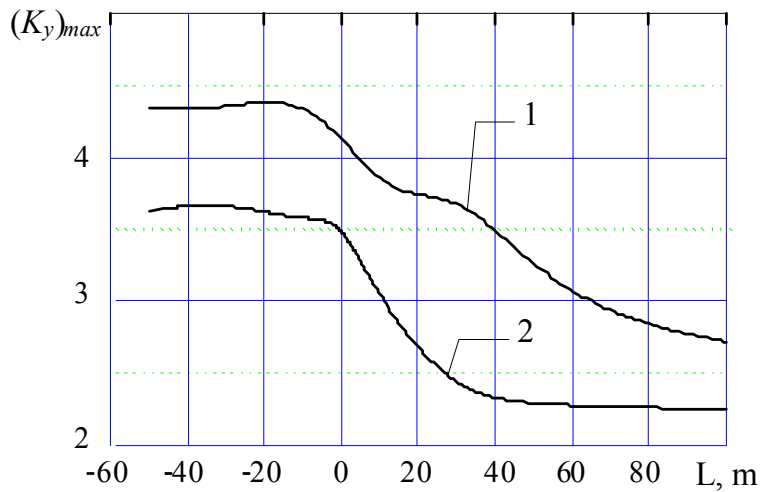


Fig. 3.2. – Change of maximum coefficient of σ_{yy} stress concentration due to the distance between the longwall and selvages of contiguous seams under conditions of underworking:
 1 – in terms of longwall displacement towards contiguous seam selvedge;
 2 – in terms of longwall outburst from under contiguous seam selvedge.

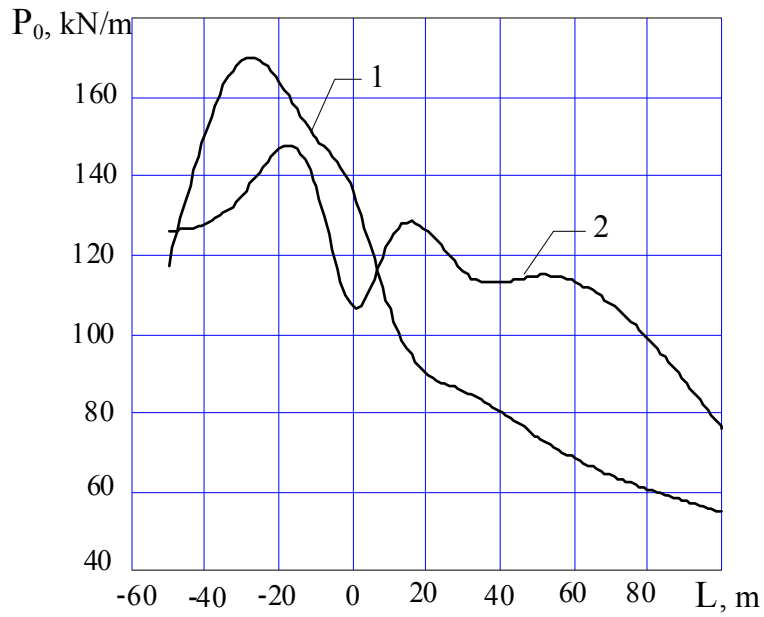


Fig. 3.3. – Load on powered support unit at different distance due to the longwall to selvedges of the contiguous seams under overworking conditions:
 1 – in terms of longwall displacement towards contiguous seam selvedge;
 2 – in terms of longwall outburst from under contiguous seam selvedge.

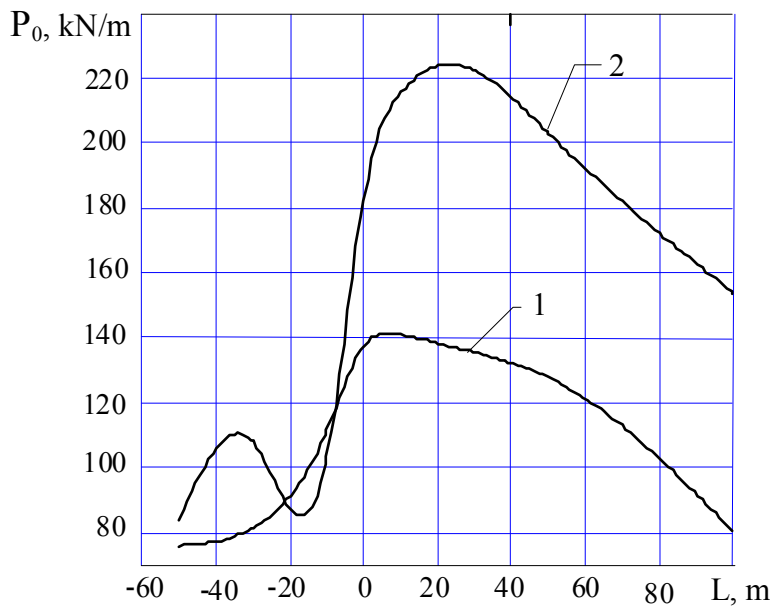


Fig. 3.4. – Load on powered support unit at different distance due to the longwall to selvedges of the contiguous seams under conditions of undermining :
 1 – in terms of longwall displacement towards contiguous seam selvedge;
 2 – in terms of longwall outburst from under contiguous seam selvedge.

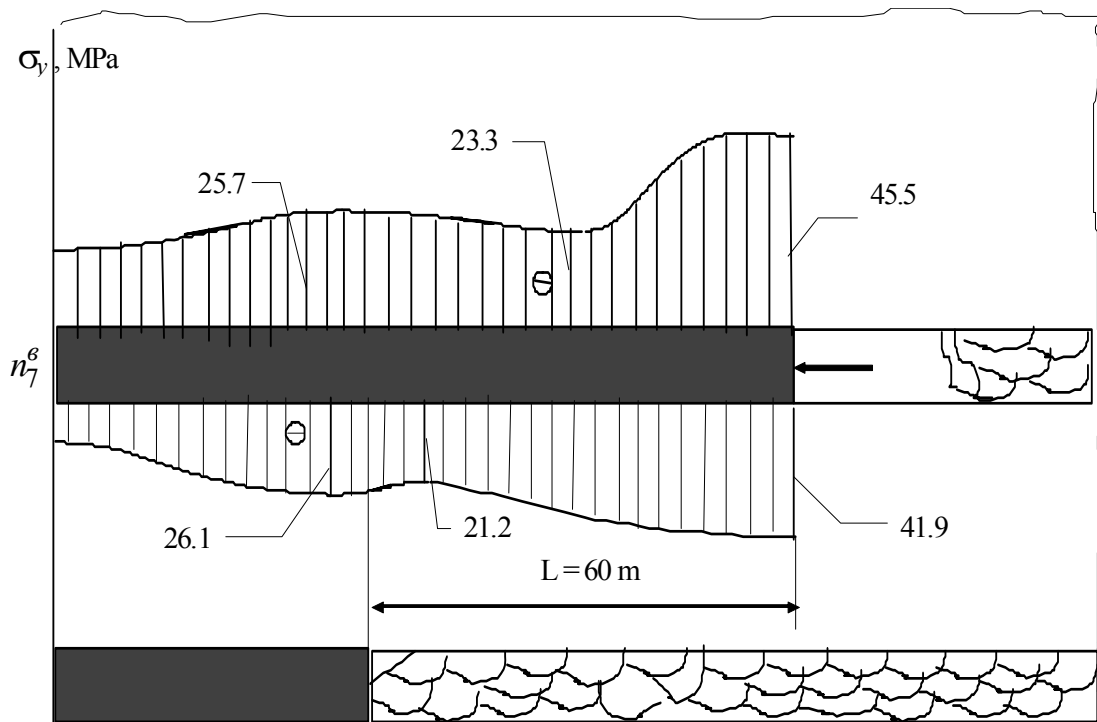


Fig. 3.5. – Epures of σ_y normal stresses in the roof and floor of n_7^6 seam (underworking, longwall displacement towards contiguous seam selvedge).

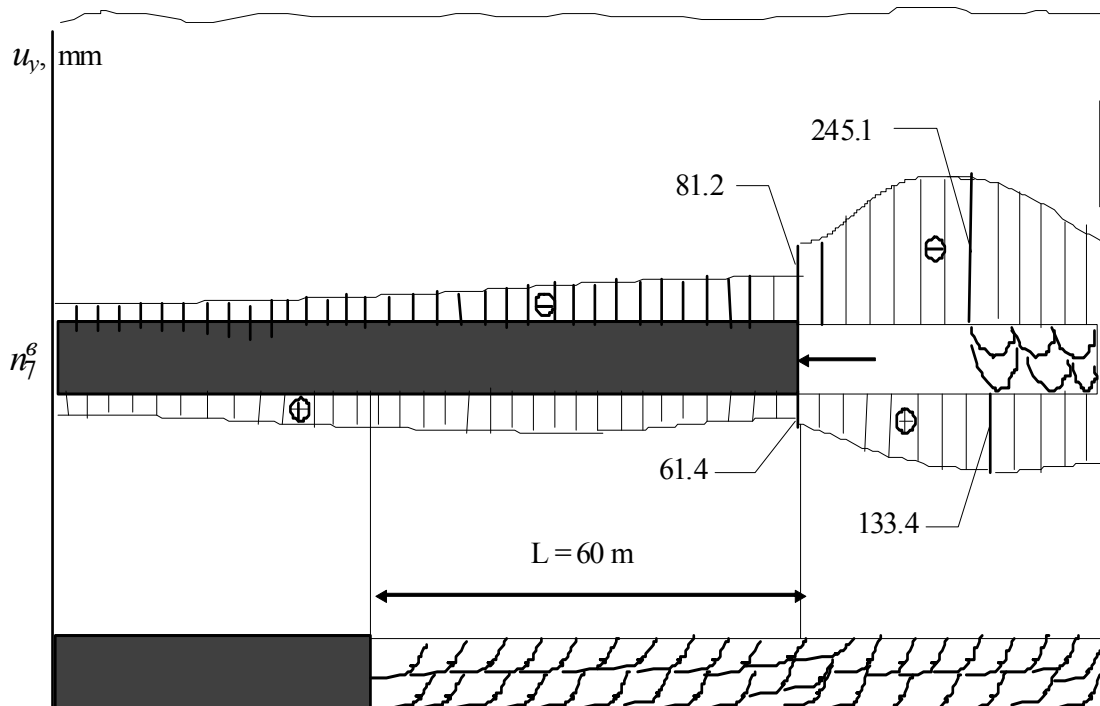


Fig. 3.6. – Epures of u_y vertical displacements in the roof and floor of n_7^6 seam (underworking, longwall displacement towards contiguous seam selvedge).

$L = 65$ m value is acceptable for underworking conditions.

Similar analysis of SSS situation in case of underworking has shown that according to (2.17) and (2.18) conditions $L = 60$ m is the admissible one.

Explain how $(K_y)_{max}$ stress concentration coefficient and V longwall advance rate effect $\sigma_y(V)$ stress value occurring in seam edge (in $x = 0$ section, Fig. 2.1) and developing additional powered support load.

Assume that longwall advance rate is $V = 2 \text{ m/day} = 0.083 \text{ m/hour}$. Then the $S = 1 \text{ m}$ web will be worked-out per $t = 6 \text{ hours}$. Value of $(K_y)_{max}$ stress concentration coefficient at $L = 65 \text{ m}$ under conditions of underworking is 3.6, consequently $K = (K_y)_{max} - 1 = 2.6$ (if $L < 65 \text{ m}$, then $(K_y)_{max} > 3.6$ and strength condition in the extracted seam roof within high rock pressure zone in front of the stope is violated). Introduce this value as well as $\gamma H = 14 \text{ MPa}$, $C = 1.2 \text{ MPa}$, and $T = 6 \text{ hour}$ values into (2.15) formula obtaining $\sigma_y(V) = 7,85 \text{ MPa}$. At $V = 4 \text{ m/day}$ rate $\sigma_y(V) = 19,4 \text{ MPa}$.

Corresponding calculations for overworking conditions when $L = 60 \text{ m}$ give $\sigma_y(V) = 7.56 \text{ MPa}$ if $V = 2 \text{ m/day}$ and $\sigma_y(V) = 16.63 \text{ MPa}$ if $V = 4 \text{ m/day}$.

Hence, along with the rate growth, stresses $\sigma_y(V)$ in $x = 0$ section increases; the higher σ_y stress concentration coefficient is, the faster this increase is.

However, in this case “removed” pressure decrease ((2.12) formula), consequently total $Q = q(V) + P_0$ powered support load (diagrams in Fig. 3.7) decreases along with the rate increase.

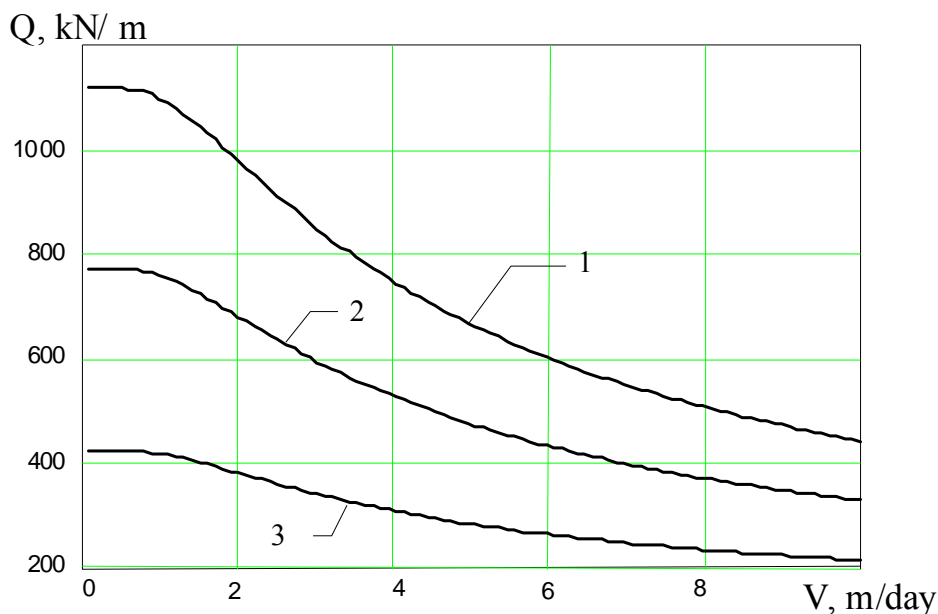


Fig. 3.7. – Dependence between total support load and longwall advance rate:

$$1 - (K_y)_{max} = 5; \quad 2 - (K_y)_{max} = 4; \quad 3 - (K_y)_{max} = 3.$$

Dependence of $q(V) = 0.03P(V)$ additional load value upon $(K_y)_{max}$ stress concentration coefficient is of practical importance. Such dependences for V values

equal to 1.5 m/day, 2.0 m/day, 3.0 m/day, and 4.2 m/day are obtained according to (2.16) formula and diagrams are developed represented in Fig. 3.8.

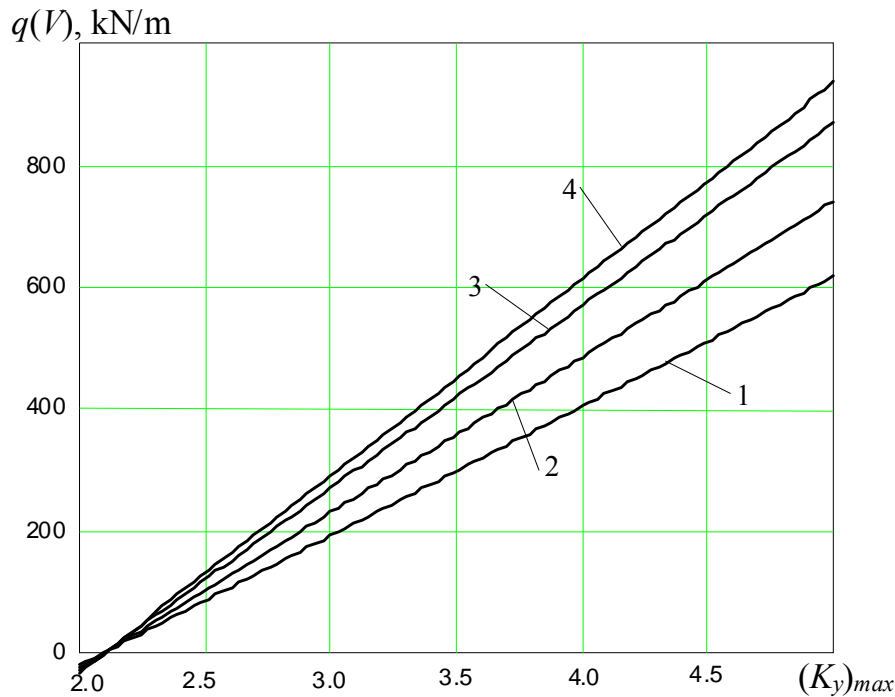


Fig. 3.8. – Dependence between additional support load and stress concentration coefficient:

1 – $V = 4.2$ m/day; 2 – $V = 3.0$ m/day;
 3 – $V = 2.0$ m/day; 4 – $V = 1.5$ m/day.

Thus, if we take L values according to the diagrams in Fig. 3.1 and 3.2, l/l_n , and $E_n/\gamma_n H$, values according to the ones of (3.2), (3.4), (3.6) or (3.8) formulas, then it is possible to determine corresponding value of $(K_y)_{max}$ maximum stress concentration coefficient in front of a stope and main P_0 powered support unit load due to the rock weigh within the zone of ultimate boundary stress state under the longwall using diagrams in Fig. 3.3 and 3.4 (depending on the direction of longwall displacement as well as considered development conditions). Next, determine $q(V) = 0.03P(V)$ additional powered support unit load at the specified longwall advance rate according to $(K_y)_{max}$ value obtained from the diagram in Fig.3.8 (or according to (2.16) formula).

Having determined actual support unit canopy load it is possible to determine reactions (resistance) of support hydraulic props.

Calculation models of 2.3 a, b and 2.4 a, b Figures used in the considered problem for one stope can be applied for the case of two stopes operating within the contiguous seams. Therefore, rational distance between longwalls within the contiguous seams can be determined according to the technique described in this paragraph using the same diagrams or formulas.

Note that according to the available [3,60,63,65] techniques of leaving longwall behind the overworked seam is determined according to the formula

$$L = hctg\delta + L_3, \quad (3.9)$$

here h is parting thickness;

δ is the angle of rock movement along the strike determined as average value for the whole rock thickness up to the surface;

$L_3 = 30\text{...}40$ m is the allowance for possible mismatch of face advance rates within contiguous seams, their emergency downtime etc.

The disadvantage of this approach is the fact that abutment pressure zone is considered only in front of a stope while stoping processes are influenced by high pressure zone behind a stope as well. Moreover, δ angle within parting thickness being the main factor to effect stress and strain state of the mass rock under consideration (stopping neighbourhood) can differ considerably from δ average value of the whole superincumbent rock. Finally, SSS situation for the roof and floor of the developed seams is also determined using stope advance rate that is not taken into account by (3.3) formula.

These aspects make the technique being developed and described in this paragraph to be different comparing to the known ones.

3.2. Determining high rock pressure zones in partings in terms of above (under) pillars mining

There are three zones according to danger level of rock pressure manifestation in stopes resulting from the effect of pillars left on contiguous seams: high risk zone (HRZ), dangerous zone (DZ), and potentially dangerous zone (PDZ).

Mode of rock pressure manifestation in stopes is the main criterion of danger level of high rock pressure zones.

High risk zone is characterized by the highest intensity of rock pressure manifestation. While stoping in this zone, dynamic manifestation of rock pressure expressed in instant breaking of roof bottom level or most part of rock mass around a stope up to the cut of the whole parting resulting in disastrous increase of support loads. Standard HRZ effect is the dramatic decrease of the immediate support stability. This zone is most often to have longwall gobs, firm blocking of powered support units, increased coal spall, and floor heaving.

While stoping the immediate support, stability decreases in DZ at the expense of increased fissility and rock foliation. There are possible cases of longwall gobs and firm blocking of powered support units but not so often as in HRZ cases. Increase of inrush formation processes is the most often rock pressure manifestation in DZ.

While stoping in PZ pillars can have no telling impact upon the character of rock pressure manifestation. Slight immediate support stability decrease is the most probable here.

Boundaries of high rock pressure zones are determined on the basis of the analysis of stress and strain state of the extracted seam roof and floor both in front and behind a stope above and under the pillar. In particular, maximum extension of the zones in which σ_y stresses exceeds γH level is determined for different stope positions relative to pillar boundaries (Fig. 3.5 a and b); ultimate stress state zones (in which equivalent stress strength conditions (2.17) are violated) are singled out inside these zones. Rock weigh loading powered support is calculated in the zone located under a longwall.

Stress and strain state of the roof and floor of the extracted n_7^g seam was analyzed in cases when a stope takes positions being marked as I ... V in Fig.2.5 a and b. Two stope positions when approaching a pillar (L distances between the longwall and pillar boundary were 30 m and 0 m correspondingly) were considered; one position is under (above) pillar midsection and two positions are behind pillar when a stope is under the second pillar boundary and at 30 m distance from it. Table 2.3 gives maximum values of $(K_y)_{max}$ stress concentration coefficients within the abutment pressure zone in front of a stoping face, values of maximum convergences of the roof and floor of the developed seam behind a face, and maximum values of P_0 powered support unit load from rock weigh being within ultimate stress state above a longwall for all the considered stope positions relative to pillars.

Diagrams in Fig. 3.9 and 3.10 give more descriptive model of $(K_y)_{max}$ and P_0 changes depending on L distance from a stope to the right pillar boundary.

Under overworking conditions (Fig. 2.5 a) in L = 60 m position, the pillar being superior to the developed seam has practically no effect on stress and strain state of the roof and floor near a stope. In this case high rock pressure zone extents over 11 meters horizontally into the mass depth covering 3 m roof thickness and 2 m floor thickness. Here (2.17) rock strength condition is violated only on the face edge.

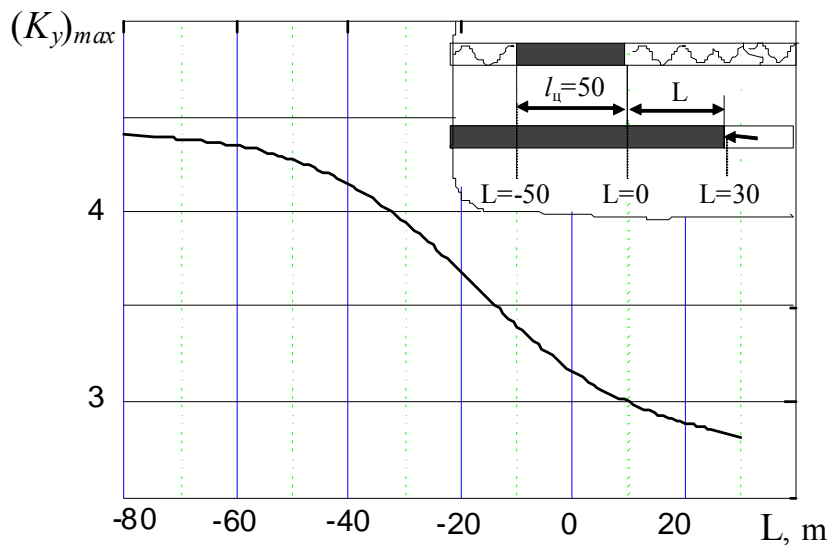


Fig. 3.9. – Stress concentration coefficients in front of a stope in terms of different distances from longwalls to the right boundary of a pillar (overworking conditions).

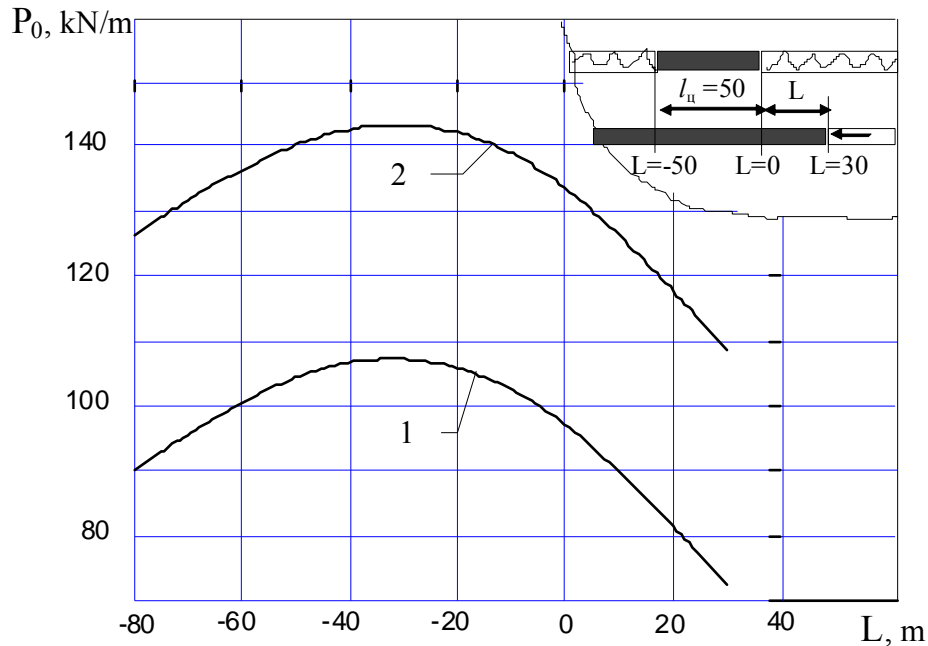


Fig. 3.10. – Powered support unit load at different longwall distance from the right boundary of a pillar:
1 – under the pillar; 2 – over the pillar.

In position I ($L = 30$ m) high rock pressure zone in front of a stope has 50 m length along Ox coordinate axis (horizontally); vertically it extends 5.5 m up into the roof and 3.5 m down into the floor along Oy axis.

In front of a stope zone of ultimate rock stress state occurs with the dimensions of 2...4 m along Ox axis and 2...3 m along Oy axis in both floor and roof. It tells about possible rock inrush at low face advance rate.

In position II ($L = 0$ m, face has approached the pillar boundary) high rock pressure zone extends over 55 m from a face down to the mass depth along Ox axis, covers 8 m of roof thickness and 5 m of floor thickness. Dimensions of ultimate stress state increase: its horizontal extension is 6 m (along Ox axis) and its vertical extension (along Oy axis) is 8 m in the roof and 5 m in the floor.

Almost the same situation is observed when a stope takes position III (it is under pillar midsection).

Stress and strain state of the floor and roof of the extracted n_7^6 seam becomes more dangerous when the face comes to the second pillar boundary (position IV). In this case high rock pressure zone has 58...60 m extension along Ox axis into the mass depth and 12 m into the roof and 6 m into the floor along Oy axis. Rocks within the area with the dimensions of 12 m along Ox axis and 6.5 m in the roof and 5 m in the floor along Oy axis are in ultimate stress state.

In position V when the face is at 30 m distance from the second pillar boundary (pillar is behind the face), the dimensions of high rock pressure zone in front of a stope decrease: its length is 18 m along Ox axis and 3.5 m in the roof and 6 m in the floor along Oy axis. However, almost all rocks of this mass are in ultimate stress

state. Consequently, there can be considerable intrushes in these positions.

Under conditions of underworking (pillar is under a stope, Fig. 2.4 b) in terms of general extension patterns of high rock pressure zones the situation is a bit different in a quantitative sense; mining conditions if a longwall become more complicated.

Fig. 3.11 and 3.12 represent high rock pressure zones in front of a longwall in terms of different longwall position as for pillars under overworking and underworking conditions.

Calculations for other stope positions relative to contiguous seam pillars were performed as well according to the developed algorithm. Corresponding stress diagrams in the roof and floor of the developed seam under conditions of overworking and underworking are represented in [103,104,112] papers.

The determined correlation dependence of the dimensions L_0 high rock pressure zone in front of the stope upon l_u pillar width and h_2 parting width that is the generalization of calculation results of stress and strain mass state in the context of the seam development under the pillar (position III); the correlation is as follows

$$\frac{L_0}{l_u} = 3,01 + 1,73 \frac{h_2}{l_u} - 0,30 \left(\frac{h_2}{l_u} \right)^2 - 0,16 \frac{l_u}{h_2} - 10,18 \frac{h_2 l_u}{(h_2 + l_u)^2}. \quad (3.10)$$

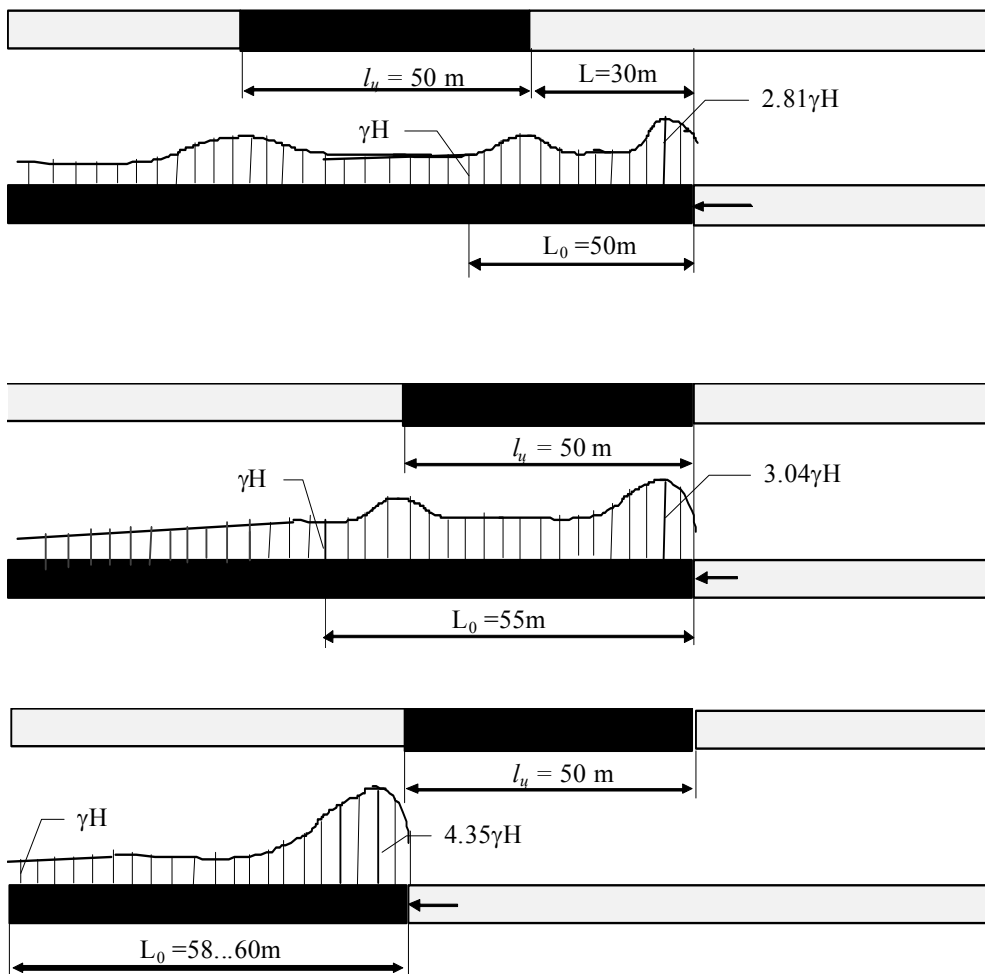


Fig. 3.11. – High rock pressure zones in front of the longwall in terms of different pillar-face positions (overworking conditions)

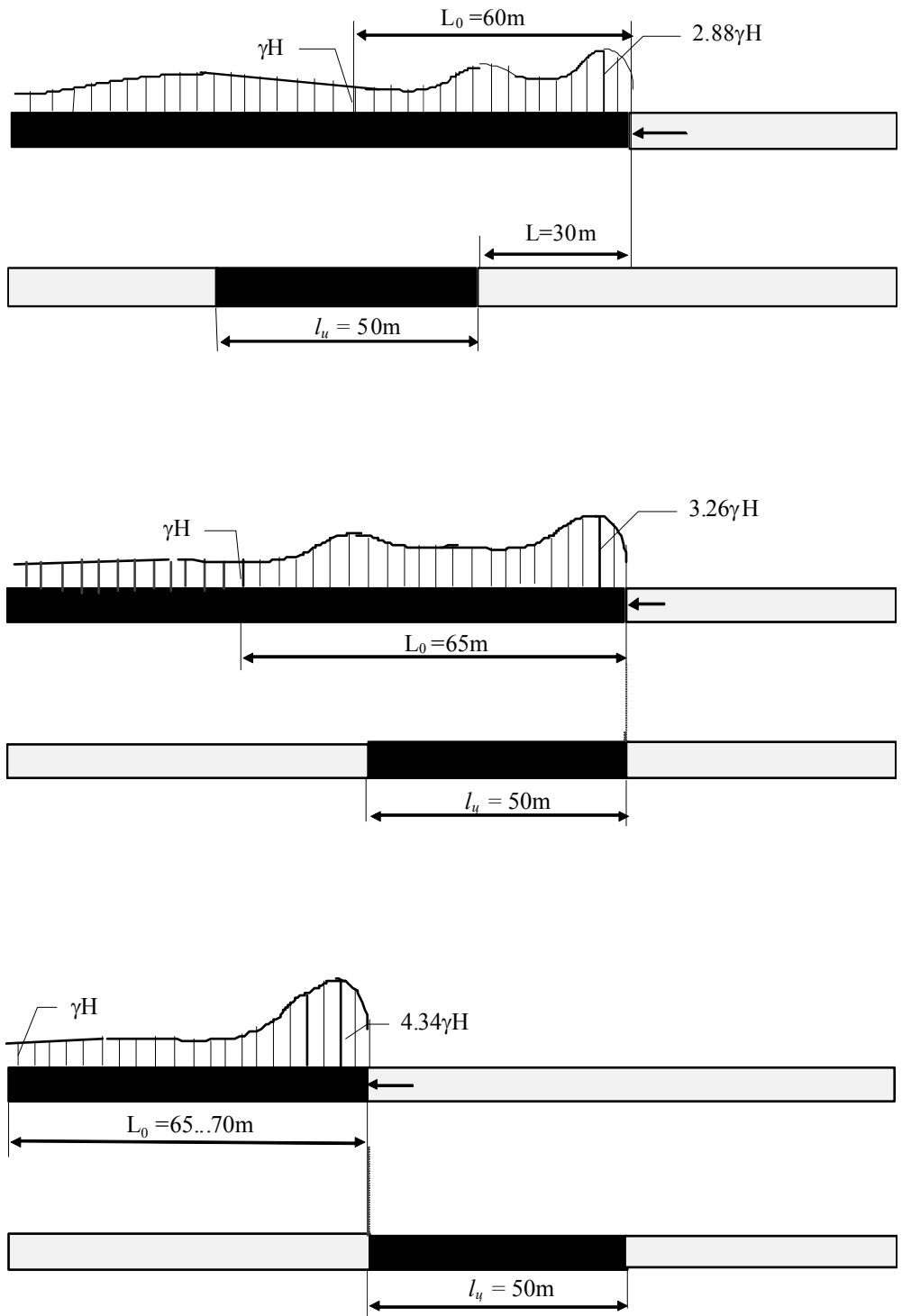


Fig. 3.12. – High rock pressure zones in front of the longwall in terms of different pillar-face positions (underworking conditions)

The dependence is obtained on the basis of 150 calculation results with different determining parameters. Value of the ratio of E_n rock elasticity modulus of the developed seam to γH varied within $(0.4...1.2) \cdot 10^3$ and $\gamma_n H / (\sigma_c)_y$ parameter belonged to $[0.25...0.35]$ interval. Thickness of h_2 parting varied within 5...20 m and l_u pillar width varied within 20...50 m.

We obtain $L_0 = 58$ m according to (3.10) formula if pillar width is 50 m and parting thickness if $h_2 = 12$ m that practically coincides with calculation results using the developed algorithm.

Dependence of $(\sigma_y)_{max} / (\sigma_c)_n$ relative maximum normal stress in the developed seam roof within high rock pressure zone in front of the longwall upon E_n / E_y , $\gamma_n H / (\sigma_c)_y$, h_2 / l_u parameters for the overworking conditions in case when the face is under pillar edge (position IV in Fig. 2.5 a) is determined as well; the dependence is as follows

$$\frac{(\sigma_{yy})_{max}}{(\sigma_c)_n} = 0,88 - 0,21 \frac{E_n}{E_y} + 1,59 \frac{\gamma_n H}{(\sigma_c)_y} + 0,91 \frac{h_2 l_u}{(h_2 + l_u)^2}, \quad (3.11)$$

where $(\sigma_c)_n$ is ultimate compression strength of the roof rocks.

There were 108 variants of calculations to be performed. Initial data were specified taking into account the fact that either argillite ($(\sigma_c)_n = 30$ MPa, $E_n = 0.63 \cdot 10^4$ MPa, $\nu = 0.2$, $\gamma_n = 2.7$ t/m³) or aleurolite ($(\sigma_c)_n = 44$ MPa, $E_n = 1.0 \cdot 10^4$ MPa, $\nu = 0.26$, $\gamma_n = 2.78$ t/m³), or sandstone ($(\sigma_c)_n = 80$ MPa, $E_n = 1.4 \cdot 10^4$ MPa, $\nu = 0.25$, $\gamma_n = 2.60$ t/m³) can occur in parting.

Average values of physical and mechanical coal characteristics (if variation coefficient is 20 %) are: $(\sigma_c)_n = 15$ MPa, $E_n = 0.35 \cdot 10^4$ MPa, $\nu = 0,3$, $\gamma_n = 1.41$ t/m³. Parting thickness had such values as 10, 12, 15, and 20 m and l_u pillar width varies within 20...50 m.

Hence, determining parameters in the calculation belonged to the following intervals: $E_n / E_y \in [1.8; 2.8; 4.0]$, $\gamma_n H / (\sigma_c)_y \in [0.25; 0.35; 0.70]$, and $h_2 / l_u \in [0.2; 0.3; 0.4; 0.5; 0.6; 0.75; 1.0]$.

Using (3.11) formula for $l_u = 50$ m; $h_2 = 12$ m, $(\sigma_c)_n = 44$ MPa (aleurolite available in the roof) and $\gamma_n H = 14$ MPa we obtain $(\sigma_y)_{max} = 60.72$ MPa.

Consequently, in this case maximum stress concentration coefficient is

$$(K_y)_{max} = \frac{(\sigma_{yy})_{max}}{\gamma H} = 4,34,$$

that coincides accurate to 1% with the calculation result of the corresponding variant according to the developed algorithm (longwall outburst from under the pillar under conditions of underworking, position IV, Table 2.3).

The formula obtained for $(\sigma_y)_{max} / (\sigma_0)_n$ relative stress in case of underworking is represented in [100] paper. It differs from (3.11) just in the variable.

As for maximum convergences of the roof and floor in the longwall, according to the results of theoretical studies maximum convergence value can be calculated using (3.1) and (3.5) formulas in cases when longwall approaches pillar or moves away from it under overworking conditions as well as using (3.3) and (3.7) formulas respectively in terms of underworking.

While stoping within the determined high rock pressure zone under the longwall, tensile stresses developed in the roof being dangerous for rocks, equivalent stresses grow, powered support load increases along with the longwall approaching to the pillar edge.

Fig. 3.10 show that the support load is maximum when a stope is in position III (Fig. 2.3 a), in particular: $(P_0)_{max} = 108.6$ kN/m if operations are under the pillar in terms of undermining; $(P_0)_{max} = 144.8$ kN/m if operations are above the pillar in terms of undermining (see Table 2.3).

It should be added that high stress concentrations in front of a stope results in coal spall and increase of $q(V)$ powered support additional load. Hence, it is not improbable that the stope development project and the chart of roof and support control should take into account taking additional measures to control roof and longwall support while passing high rock pressure zones preventing or neutralizing dangerous rock pressure effects.

Such correlation values as 3.1...3.8, 3.10, and 3.11 represented in this chapter are the basis of the technique to determine rational powered support parameters. The latter, in particular, includes calculation of hydraulic props reactions within the area of high rock pressure; next paragraph describes the calculation sequence.

3.3. Technique to determine hydraulic props reaction of the powered support within high rock pressure zones

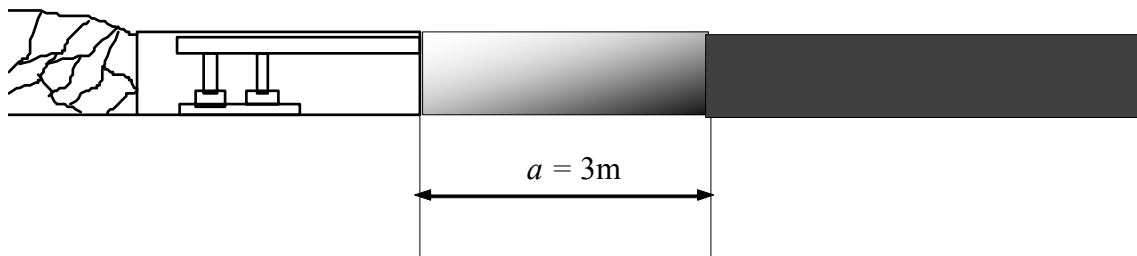
As it was mentioned before, total powered support load consists of the P_0 main load (weigh pressure of the rock of the boundary stress state under the longwall) and $q(V)$ additional one making up 3% from the pressure being “removed” from the pressure coal seam within a spall zone.

Support is calculated according to the beam schematic assuming that the load along the canopy length is distributed equally; moreover support is loaded by $q(V)$ pressure gradually. Loaded area length depends on the considered time moments well as longwall advance rate.

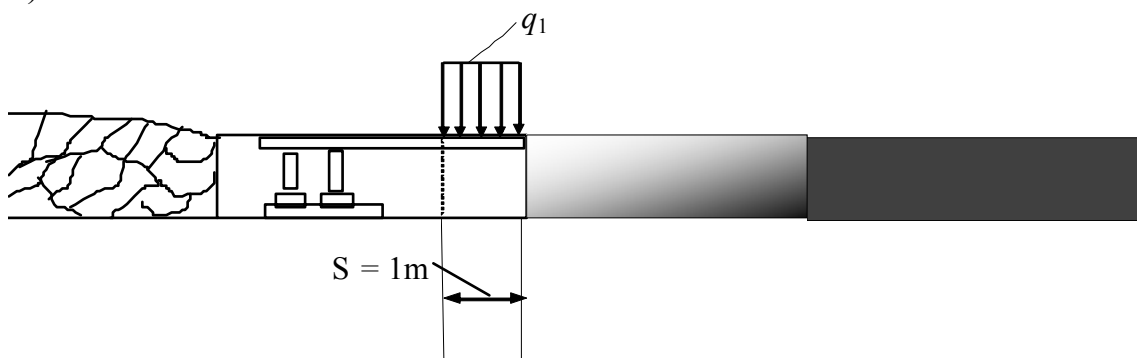
If we start t timing from the beginning of longwall operation, specify stoping displacement rate as $V = 4$ m/day = 0.17 m/hour, and consider t time moments equal to 6, 12, and 18 hours, then we can suppose that in 6 hours after the reference time $q(V)|_{t=6}$ pressure loads equally the canopy section of 1 m length beginning from the overhang free end. In the next 6 hours, at $t = 12$ hours time point, following section of 1 m length loaded with $q(V)|_{t=12}$ force will be added to this section etc. up to the loading of the whole canopy within the area of $P(V)$ pressure diagram

formation; after that the process repeats. Fig. 3.13 shows typical canopy loading diagrams for the case when $V = 4$ m/day. Case when the length of section where with $P(V)$ diagram formation is equal to 3 m is considered for generality.

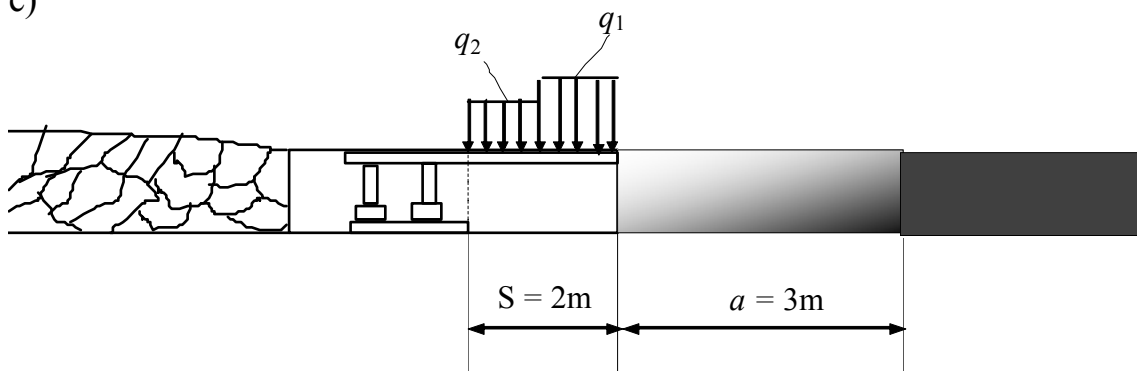
a)



b)



c)



d)

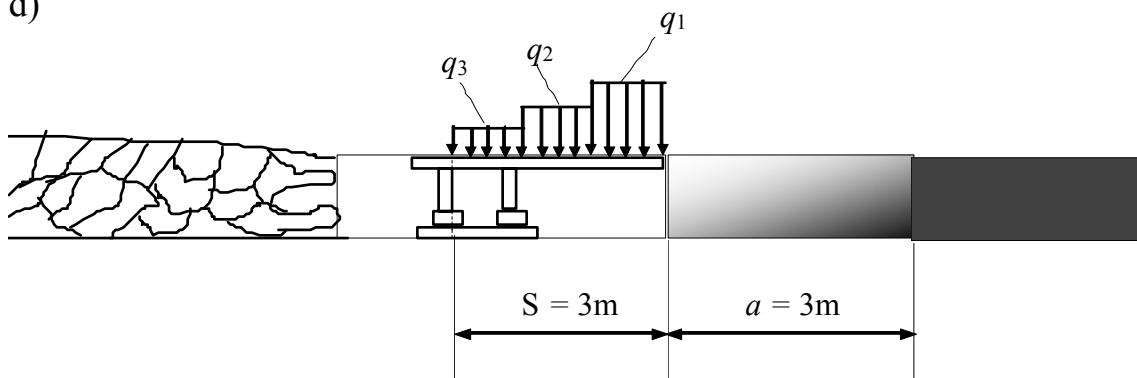


Fig. 3.13. – Powered support loading diagrams at different time points:

a) $t = 0$; б) $t = 6$ hours; в) $t = 12$ hours; г) $t = 18$ hours.

The calculation has following sequence:

1) value of main P_0 load effecting per unit of canopy length for the corresponding value of L parameter (face-pillar or face selvedge distance), mining conditions (underworking, overworking), and stoping displacement direction (within the selvedge, outbursts from under the selvedge) according to $P_0(L)$ diagrams (Fig. 3.3, 3.4 or 3.10);

2) R_0^j reactions of hydraulic props (j is hydraulic prop number) corresponding to this load are determined;

3) σ_{yy} maximum stress concentration coefficient in front of the stope is determined for the specified L parameter according to $(K_y)_{max}(L)$ diagrams (Fig. 3.1, 3.2, and 3.9);

4) we determine values of $q_i = 0,03P(V)|_{t=t_i}$ additional loads having taken V longwall advance rate taking into account the obtained $(K_y)_{max}$ coefficient using (2.16) formula for each of the calculation models of powered support canopy load at t_i time points;

5) $R_i(V)$ hydraulic props reactions corresponding the formed canopy pressure diagrams are calculated;

6) total hydraulic props reactions are determined: $P_j = R_j + R_j(V)$, kN;

7) specific resistance per 1 m² of the supported area is determined:

$$R = \frac{\sum_j P_j}{S}, \quad S = 2.7 \cdot 1 = 2.7 \text{ m}^2 \text{ is the roof contact area;}$$

8) $R \leq [R]$ condition is tested, where $[R]$ is the value of specific resistance per 1 m² of the supported area according to support technical specification;

9) $(P_j)_{max} \leq [P]$ condition is tested, where $[P]$ is the load-bearing capacity of a hydraulic prop according to the technical specification;

10) the fulfillment of $\Delta u_y \leq [\Delta u_y]$ condition is tested where Δu_y is the value of stope roof and floor contact determining for the specific values of m , E_n/E_y , $\gamma_n H/(\sigma_c)_y$, and l/l_n corresponding to 3.1, 3.3, 3.5 or 3.7 formula;

$[u_y] = H_{max} - H_{min} - d$ is the permissible support extension;

H_{max} and H_{min} are maximum and minimum structural support height;

d is the allowance of unloading hydraulic props extension (if $m < 1$ m then $d \geq 30$ mm values should be taken, and if $m > 1$ m $d \geq 50$ mm value should be taken [66]).

In case when the tested conditions are not fulfilled, support with higher load-bearing capacity should be used or specific measures should be taken, e.g. mounting additional props under canopy top of the capping or in the back rows of the units; coal extraction with coal-cutting with stone applied for roof and floor unstable layers to prevent blocking of support units etc.

3.4. Calculation of parameters for contiguous coal seam extraction in terms of “Lvovugol” SE mines according to the developed technique

3.4.1. Determining maximum reactions in support hydraulic props within the effect area of contiguous seam selvedges. While developing contiguous seams the most complicated conditions for powered complex can be observed in effect zones of selvedges and pillars of contiguous seams. The paragraph is devoted to clarification of the situation connected with loading of the support hydraulic props within selvedge effect area.

Since the direction of a longwall displaces is quite important and what the stoping conditions are (underworking or overworking), consider four calculation cases:

- 1) longwall displacement towards the selvedge in terms of overworking;
- 2) longwall displacement towards the selvedge in terms of underworking;
- 3) longwall outburst out of the selvedge effect zone in terms of overworking;
- 4) longwall outburst out of the selvedge effect zone in terms of underworking.

In the first calculation case maximum effort in hydraulic props can occur when a face is in positions VI ($L = -10$ m) or VII ($L = -30$ m). In the second calculation case - in positions IV ($L = 10$ m) or VI ($L = -10$ m). As for the third calculation case according to the data of Table 2.2 there is position V ($L = -30$ m) for a stop, and as for the fourth calculation case there are positions V ($L = -30$ m) and VI ($L = -50$ m). In the mentioned positions either $(K_y)_{max}$ stress concentration coefficient or P_0 powered support unit load take the highest values; or both these values are close to the maximum ones.

Talking about the effect of main P_0 load, it has the highest value equal to 222.3 kN/m when a stope is in position VI ($L = -50$ m) when leaving the selvedge zone and coal is extracted in terms of underworking (fourth calculation case).

In this case KM87 standard support is a two-point overhanging beam loaded equally over the whole canopy. The reactions of such beam, i.e. in the considered case the efforts in the front and back hydraulic props are correspondingly equal to

$$R_B = 4.84 P_0 = 1076 \text{ kN and } R_A = -1.04 P_0 = 231 \text{ kN,}$$

while permissible value for the efforts of KM87 support hydraulic props is 650 kN. Hence, front prop cannot bear the actual load.

It is easy to see that in terms of all other determined stope positions R_B reaction of the front prop exceeds the permissible value.

That is why the authors of the monograph recommend to use KM87 support with the overhang shortened by 0.33 m in terms of the considered mining conditions.

Using the developed technique, calculate such upgraded support to operate under conditions of effect zone of contiguous seam selvedges for two values of $V = 4$ m/day and $V = 2$ m/day longwall advance rates if there are following initial data: $\gamma H = 11.5$ MPa; seam thickness is $m = 0.7$ m; $h_2 = 13$ m; immediate roof is aleurolite ($E = 1 \cdot 10^4$ MPa, $\nu = 0.26$, $\gamma = 2.8$ t/m³, $(\sigma_c)_n = 44$ MPa); floor is argillite ($E = 0.63 \cdot 10^4$ MPa, $\nu = 0.2$, $\gamma = 2.3$ t/m³, $(\sigma_c)_n = 30$ MPa).

Physical and mechanical coal characteristics are:

Internal friction coefficient is $f = 0.7$;

Adhesion is $C = 1.2$ MPa;

Rheological constant is $T = 6$ hour.;

Ultimate compression strength is $(\sigma_c)_y = 20$ MPa;

Modulus of elasticity is $E_y = 0.28 \cdot 10^4$ MPa;

Poisson's ratio is $\nu = 0.3$;

Density is $\gamma = 1.69$ t/m³.

Geometrical parameters are:

Spall zone length is $a = 1$ m;

Width of the longwall and the part of a worked-out area not filled with the broken-down rock are $l_1 = 5$ m in case if the support has moderate stability and $l_1 = 2.7$ m in case of unstable support.

Technical specifications of KM 87 upgraded support.

Strength parameters:

Section resistance is $\sum P_j = 1560$ kN;

Specific resistance per 1 m² of the supported area is $R = 400$ kN/m²;

Maximum load-bearing capacity of a hydraulic prop is $P_j = 650$ kN.

Geometrical parameters:

Length of the shortened overhang is $l_\kappa = 1.9$ m;

Extension is $\Delta u = H_{max} - H_{min} - d = 0.85$ m.

1. Determining actual load and resistance of the hydraulic props.

a) According to Tables 2.1 and 2.2 determine $(K_y)_{max}$ values and P_0 loads corresponding to them for all the considered stope positions.

While determining hydraulic prop reaction take into account the determined character of roof breaking. According to the rock pressure observations under mining conditions [107], immediate roof within high rock pressure zones comes from the category of medium stability to the unstable one; immediate roof rocks are broken following the support relocation, in this context, only 1.1 m part of a front overhang is loaded.

This is the basis to consider two-point beam with general length of 2.7 m (back overhang is 0.5 m, distance between hydraulic props is 1.1 m).

Then R_A and R_B reactions in back and front props due to the P_0 load effect being distributed equally over the length of 2.7 m (Fig. 3.14 a) will be equal to $R_A = 0.6 P_0$, $R_B = 2.1 P_0$.

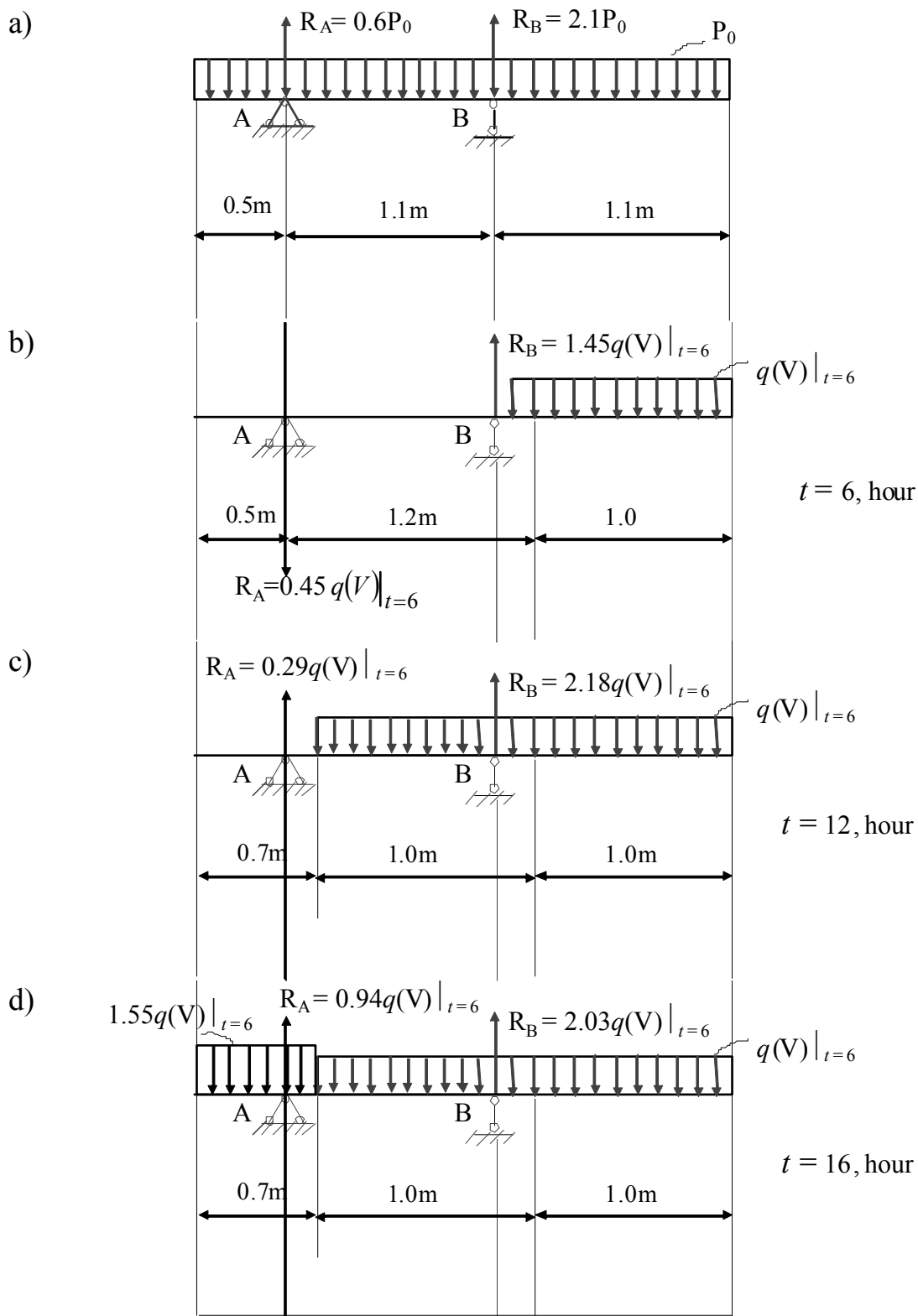


Fig. 3.14. – Calculation models for upgraded KM87 support:
 a) is main load effect;
 b), c), and d) are additional load effects

Table 3.1 contains the results of reaction calculations due to P_0 load in all the considered cases.

Table 3.1

Reactions in hydraulic props in terms of P_0 load effect

Calculation case	P_0 , kN/m	Direction of longwall displacement	Stope position	Mining conditions	R_A , kN	R_B , kN
1	137.7 135.0	To the selvedge	over working	VI (L = -10m)	82.6	289.2
				VII (L = -30m)	81.0	283.5
2	141 111		under working	IV (L = 10m)	84.5	296.1
				VI (L = -10m)	66.6	233.1
3	169.2 215.7 222.3	When leaving the selvedge effect zone	over working	V (L = -30m)	101.5	355.3
			under working	V (L = -30m)	129.4	453.0
				VI (L = -50m)	133.3	466.8

As for reactions due to $q(V) = 0.03P(V)$ additional load, the following things take place under the considered conditions. If longwall advance rate is $V = 4$ m/day, then S web equal to 1 m is extracted per 6 hours. Spall is over within this area per this time beginning within the next one. At that canopy part of 1 m length is loaded with $q(V)|_{t=6}$ pressure after 6 h from the beginning of spall process.

During the following 6 hours, the same pressure will load next part of 1 m length. The last part of 0.7 m length will be loaded within 4 hours. Actual load effecting it will be equal to $q(V)|_{t=4}$.

If longwall advance rate is $V = 2$ m/day, the process of canopy load formation is twice longer that deteriorates canopy operation conditions as $q(V)$ load increases along with the load decrease. It is shown by (2.16) formula (cofactor $(1 - e^{-t/T})$ grows) and Fig. 3.7.

Fig. 3.14 b, c, and d show corresponding calculation models.

Calculate additional load values using (2.16) formula. Parameters of (2.16) formula being similar for all the calculation cases have the values

$$e^{-2a/l} = e^{-0,74} = 0,48; \quad \frac{h\lambda C}{f} = 0,258;$$

$$e^{fa/h\lambda} = e^{4,65} = 104,6; \quad \frac{h\lambda C}{f} (e^{fa/h\lambda} - 1) = 26,73.$$

$e^{-t/T}$ value at the considered $t = 4$ hour and $t = 6$ hour time points adopts $e^{-4/6} = e^{-0,66} = 0.52$ and $e^{-1} = 0.368$ values.

Table 3.2 represents the rest of the parameters from (2.16) formula as well as calculation results of the additional loads for two of longwall rate values ($V = 4$ m/day and $V = 2$ m/day).

Tables 3.1 and 3.2 show that along with twofold increase of longwall advance rate (from 2 m/day up to 4 m/day) total support load decreases by 1.23...1.28 times.

Thus, according to the loading diagrams (Fig3.14) total values of hydraulic props reactions are:

$$\begin{aligned} \text{if } t = 6 \text{ hours} \quad R_A &= 0.6 P_0 - 0.45 q(V)|_{t=6}, \\ R_B &= 2.1 P_0 + 1.45 q(V)|_{t=6}; \end{aligned}$$

$$\begin{aligned} \text{if } t = 12 \text{ hours} \quad R_A &= 0.6 P_0 + 0.29 q(V)|_{t=6}, \\ R_B &= 2.1 P_0 + 2.18 q(V)|_{t=6}; \end{aligned}$$

$$\begin{aligned} \text{if } t = 16 \text{ hours} \quad R_A &= 0.6 P_0 + 0.94 q(V)|_{t=6}, \\ R_B &= 2.1 P_0 + 2.03 q(V)|_{t=6}. \end{aligned}$$

Table 3.3 shows their values for all the considered load cases within high rock pressure zones.

Table 3.2

Calculation of additional load upon KM87 upgraded support within the effect zones of contiguous seam selvedges

Calculation case	Direction of longwall advance	Mining conditions	Stope position	t , hour	$(K_y)_{max}$	$\frac{Kl_1}{2} e^{-\frac{2a}{l_1}}$ m	$P(V) \cdot 10^{-3}$, kN/m			$q(V)$, kN/m		
							V , 4m/day	V , 2m/day	V , 4m/day	V , 4m/day	V , 2m/day	V , 4m/day
1	To the selv-edge	over working	VI (L = -10m)	4	4.23	2.1	10.52	14.28	316.0	428.4		
				6	4.23	2.1	6.77	9.21	203.0	276.3		
				4	4.35	2.17	11.47	15.62	344.1	468.8		
2	To the selv-edge	under working	IV (L = -30m)	6	4.35	2.17	7.41	10.08	222.0	302.5		
				4	3.85	1.85	7.46	10.10	223.8	304.5		
				6	3.85	1.85	4.82	6.55	144.6	224.5		
3	When leaving the selvedge effect zone	over working	VI (L = -10m)	4	4.33	2.16	11.27	15.31	338.1	459.4		
				6	4.33	2.16	7.27	9.88	218.0	296.6		
				4	3.70	1.75	6.26	8.50	187.9	255.0		
3	When leaving the selvedge effect zone	under working	V (L = -30m)	6	3.70	1.75	4.04	5.49	121.2	164.7		
				4	3.69	1.74	6.26	8.51	187.8	255.3		
				6	3.69	1.74	4.04	5.50	121.2	165.0		
3	When leaving the selvedge effect zone	under working	VI (L = -50m)	4	3.63	1.70	5.86	7.96	175.8	238.9		
				6	3.63	1.70	3.78	5.15	113.6	154.4		

Table 3.3

Reactions in hydraulic props of KM87 upgraded support
within the effect zone of contiguous seam selvedges

Calculation case	Direction of longwall advance	Mining conditions	Stope position	t, hour	R _A , kN		R _B , kN	
					V ₄ , m/day	V ₂ , m/day	V ₄ , m/day	V ₂ , m/day
1	To the selvedge	over-working	VI (L = -10m)	6	-	-	58.4	69
				12	14	16	73	89
				16	27.3	34.2	70	85
			VII (L = -30m)	6	-	-	60	72.2
				12	14.5	16.9	76	94.3
				16	27.7	36.5	73.4	89.8
2		under-working	IV (L = 10m)	6	-	-	50.6	62
				12	12.6	64	15	78.5
				16	22	59	29.5	84.2
			VI (L = -10m)	6	-	-	55	66.3
				12	13	15.3	70.8	88
				16	27.2	34.5	67.5	83.5
3	When leaving The effect zone of selvedge	over-working	V (L = -30m)	6	-	-	53	59.4
				12	13.7	15	62	71.4
				16	21.5	25.6	60	69
		under-working	V (L = -30m)	6	-	-	62.8	69.2
				12	16.5	17.7	71.7	81.3
				16	24	28.5	70	78.8
			VI (L = -50m)	6	-	-	63	69
				12	16.7	17.8	71.4	80.3
				16	24	27.8	69.7	78

Check of condition meeting as for support reliable operation.

a) If specific resistance per 1 m² of the supported area in the most dangerous case (L = - 30 m, face advances towards the selvedge under overworking conditions) at V = 2 m/day $R = 412 \kappa H / M^2 > [R] = 400 \kappa H / M^2$ then the condition is not met;

at V = 4 m/day $R = 335 \kappa H / M^2 < [R] = 400 \kappa H / M^2$ the condition is met.

b) If the reaction in the most loaded prop at V = 2 m/day is

$(P_j)_{max} = 943 \kappa H > [P] = 650 \kappa H$ then the condition is not met.

It should be noted that in almost all considered cases the condition is not met. It is not met in some cases when V = 4m/day as well.

c) Determine roof and floor convergence using (3.1) formula for the most dangerous case

$$\Delta u_y = m \left[0,107 + 0,097 \frac{l_H - L}{l_H} - 0,065 \frac{E_n}{E_y} + 0,530 \frac{\gamma_n H}{(\sigma_c)_y} \right] =$$

$$= 0,259 \text{ м} < 0,85 \text{ м},$$

that is far less the permissible support extension.

Since the support has considerable allowance in its extension, in case of $(P_j)_{max}$ exceeding the permissible $R = 650 \text{ kN}$ value it will “escape” the load though so that there will be no hard caving жесткой посадки, $V_{\text{п}}$ longwall advance rate should be more than 4 m/day.

To have more reliable and efficient operation within the zones of high rock pressure due to pillars and contiguous seam selvages it is necessary to start using of new generation complexes.

3.4.2. D e t e r m i n i n g p e r m i s s i b l e d i s t a n c e b e t w e e n t h e s t o p e s o f c o n t i g u o u s s e a m s. L distance between the stopes operating within contiguous seams should be the one so that stress and state of the surrounding rocks will not reach boundary state. It means that within the zones with dangerous rock pressure (2.17) strength condition should be met and convergence both in longwalls and right near them behind the stopes should not be more that 30% from the extracted thickness, i.e. (2.27) condition should be met.

As it was mentioned in paragraph 3.2 it is true for the mines of the considered region if L values are equal to 60m and 65 m respectively in overworking and underworking cases.

Thus, according to (2.17) criterion by P.P. Balandin in the dangerous section in front of the longwall under overworking conditions ($L = 60 \text{ m}$) $(\sigma_{\text{экв}})_{max}$ equivalent stress has the value of 8.47 MPa. It is not more than $(\sigma_c)_y = 20 \text{ MPa}$ ultimate coal compressive strength. Similarly, under conditions of underworking ($L = 65 \text{ m}$) we have $(\sigma_{\text{экв}})_{max} = 12.45 \text{ MPa} < 20 \text{ MPa}$.

Maximum roof and floor convergences in the longwall have 246.6 and 248.8 mm values for overworking and underworking conditions respectively. In the first case they account for 35.2% of $m = 0.7 \text{ m}$ extracted thickness, in the second case it is 35.5%.

Determine forces in the hydraulic props of KM87 support with the shortened overhang.

According to calculations performed according to the developed algorithm, if $L = 65 \text{ m}$ under condition of underworking, then main prop load is $P_0 = 120 \text{ kN/m}$. The resulting reactions have the following values: $R_A = 0.6 P_0 = 72 \text{ kN}$ in the back prop and $R_B = 2.1 P_0 = 252 \text{ kN}$ in the front prop.

In case of overworking $P_0 = 112 \text{ kN/m}$ and $R_A = 67 \text{ kN}$, $R_B = 235 \text{ kN}$.

Additional load is determined similarly to the way used in the previous paragraph in terms of analysis of longwall operation conditions within the effect area of contiguous seam selvedges using the same initial data. Table 3.4 gives calculation results.

Table 3.4

Calculation of the additional load on KM87 updated powered support while longwall operating within contiguous seams

Mining conditions	Distance between the stopes, L, m	K_y	$\frac{Kl_1}{2} e^{-\frac{2a}{l_1}}$, m	$P(V) \cdot 10^{-3}$, kN/m		$q(V) _{t=6}$, kN/m	
				V , 4m/day	V , 4m/day	V , 4m/day	V , 4m/day
Overworking	60	3.50	1.62	5.84	9.24	17.50	27.70
Underworking	65	3.63	1.70	3.78	5.15	11.36	15.44

Table 3.5 contains calculation results for the total reactions of hydraulic props.

Table 3.5

Reactions in hydraulic props and additional canopy support within high rock pressure area

Mining conditions	t , hour	R_A , kN		R_B , kN	
		V , 4m/day	V , 2m/day	V , 4m/day	V , 2m/day
overworking	6	-	-	506	654
	12	123	152	634	856
	16	236	332	607	814
underworking	6	-	-	400	460
	12	100	112	483	572
	16	174	212	486	568

As Table 3.5 shows if $V = 2$ m/day, then $R_B = 856$ kN reaction of the back prop is more than $R = 650$ kN value, specified by technical characteristics. If $V = 4$ m/day then any of R_B maximum reaction values is not more than the admissible one. In all the cases back prop is underloaded.

Hence, in terms of "Lvovugol" SE mines, safe distances between contiguous seam longwalls are $L = 65$ m and $L = 60$ m as for the rock pressure factor. Moreover, longwalls should advance at $V \geq 4$ m/day rate.

3.4.3. Determining maximum reactions in hydraulic props of the support during longwall operations over (under) pillars.

While determining maximum reactions in hydraulic props of powered support we will be based on the results of calculation and analysis of stress and strain state of layered heterogeneous mass according to the scheme represented in Fig. 2.5.

$(K_y)_{max}$ stress concentration coefficients in front of the face as well as P_0 load values effecting roof support canopy are the determining ones. Table 2.3 contains corresponding data as applied to different stope positions relative to contiguous seam pillars. It shows that under conditions of overworking when approaching the pillar position III, when the longwall is under pillar midsection ($L = 25$ m), and position IV, when the longwall has come to the second boundary of the pillar, are the most dangerous ones. In the first case $(K_y)_{max} = 3.88$ but $P_0 = 168.4$ kN/m (the maximum value), in the second case $(K_y)_{max} = 4.35$ and $P_0 = 165.0$ kN/m (both values are close to the maximum ones).

The same longwall positions are dangerous in terms of underworking as well: in position III $(K_y)_{max} = 3.90$ and $P_0 = 224.5$ kN/m; in position IV – $(K_y)_{max} = 4.34$ and $P_0 = 220.4$ kN/m.

Use formula (2.16) to determine additional load upon powered support unit as we have done while studying effect zones of contiguous seam selvages in terms of the same mining depth and similar coal and enclosing rock characteristics.

Put the calculation results in Tables 3.6, 3.7, and 3.8.

Table 3.6

Hydraulic prop reactions in terms of P_0

Stope position		Mining condition	Distance from a face to the pillar boundaries L, m	$P_0, kN/m$	Reaction of hydraulic props	
					R_A, kN	R_B, kN
Under the pillar	III	over-working	25	108.6	101	354
Behind the pillar	IV		0	106.4	99	347
Under the pillar	III	under-wirking	25	144.8	135	471
Behind the pillar	IV		0	142.0	132	462

Table 3.7

Calculation of additional load upon KM87 upgraded support during longwall operation over (under) the pillar

Stope position	Mining conditions	Distance from the stope the pillar boundary L, m	$(K_y)_{max}$	$\frac{Kl_1}{2} e^{-\frac{2a}{l_1}}$, m	$P(V) \cdot 10^{-3}$, kN/m		$q(V) _{t=6}$, kN/m	
					$V, 4 m/s$	$V, 2 m/s$	$V, 4 m/s$	$V, 2 m/s$
Under the pillar behind the pillar	I	25	3.85	1.85	3.92	5.33	117.3	160.0
	IV	0	4.35	4.52	6.08	8.28	182.0	247.6
over the pillar behind the pillar	III	25	3.90	1.88	4.13	5.62	123.8	168.9
	IV	0	4.34	4.50	5.99	8.13	179.6	244.4

Table 3.8

Reactions in canopy props of KM87 upgraded support during longwall operation over (under) the pillars

Stope position	Mining conditions	Distance From the stope To the pillar boundary L , m	t , hour	R_A , kN		R_B , kN	
				V , 4 m/day	V , 2 m/day	V , 4 m/day	V , 2 m/day
under the pillar behind the pillar	over-working	25	6	-	-	398.2	460.0
			12	99.2	111.5	483.8	579.9
			16	175.5	215.5	466.2	552.9
over the pillar	over-working	0	6	-	-	487.3	582.4
			12	116.6	135.6	620.2	763.2
			16	234.9	296.6	592.9	726.0
over the pillar behind the pillar	under-working	25	6	-	-	483.6	549.0
			12	122.8	135.9	574.0	672.3
			16	203.3	245.7	555.4	647.0
over the pillar behind the pillar	under-working	0	6	-	-	558.6	652.6
			12	137.3	156.1	689.7	831.0
			16	254.0	314.9	662.8	794.3

Test of the fulfillment of the conditions as for reliable support operation.

a) Specific resistance per 1 m² of the supported area in the most dangerous case (L = 0, underworking):

if V = 2 m/day then $R = 365,6 \text{ kN} / \text{m}^2 < 400 \text{ kN} / \text{m}^2$ and condition is fulfilled;

if V = 4 m/day then $R = 306,3 \text{ kN} / \text{m}^2 < 400 \text{ kN} / \text{m}^2$ and condition is fulfilled.

b) Resistance in the prop loaded to the maximum:

if V = 2 m/day then $(P_j)_{max} = 831 \text{ kN} > [P] = 650 \text{ kN}$ and condition is not fulfilled.

if V = 4 m/day then $(P_j)_{max} = 689,7 \text{ kN}$. The value exceeds the permissible one by 1.06 times.

c) determine roof and floor convergence in the longwall according to (3.3) formula

$$\Delta u_y = m \left[-0,311 + 0,115 \frac{l_n - L}{l_n} + 0,364 \frac{E_y}{E_n} + 0,541 \frac{\gamma_n H}{(\sigma_c)_y} \right] =$$

= 0.220 m < $[\Delta u_y]$ the condition is fulfilled with ($[\Delta u_y] = 0.85 \text{ m}$) allowance.

The results show that if $V_{\text{pr}} = 4 \text{ m/day}$, then KM87 support with the shortened hangover operates in considered cases with minor resistance growth in the front hydraulic prop being loaded most of all; as for stiffness the support has considerable allowance.

Consequently it can be used quite efficiently within the zones of high rock pressure from contiguous seam pillars but under conditions that V_{pr} longwall advance rate will be not less than 4 m/day.

Conclusions

1. Multivariate correlation analysis of the calculated data for underworking and overworking conditions (taking into account longwall displacement direction relative to the contiguous seam selvedge) has been used to determine correlations for maximum values of stoping roof and floor convergence as well as to define maximum coefficients of σ_{yy} stress concentrations in the roof of the developed seam within the abutment pressure zone in front of a stope.

2. Nonlinear correlation dependence of the dimensions of high rock pressure zone in front of the stope upon the pillar width and parting thickness as applied to the seam development under the pillar.

3. Nonlinear correlation dependence of σ_y maximum stress in the floor of the developed seam within high rock pressure zone in front of the longwall upon the mining depth, physical and mechanical characteristics of coal and enclosing rocks as well as upon pillar width and parting thickness for overworking conditions when the stope is under pillar edge.

4. The determined correlation ratios has become the basis to develop the technique for calculation of rational powered support parameters in terms of operation within the zones of high rock pressure from the pillars and contiguous seam

selvedges. Contrary to the known techniques, the developed one determines the load effecting the canopy by means of calculation. Moreover, coal properties, stress concentration within the abutment pressure zone in front of the stope, and longwall displacement rate are taken into account.

5. The developed technique has been used to determine reactions (resistances) in hydraulic props of the support within the zones of high rock pressure from the pillars and contiguous seam selvedges; it has been shown that the efficient operation of the support requires canopy overhang to be shortened by 0.33 m as it increases support resistance near the stope, and hydraulic props are loaded more equally.

6. It is shown that according to the factor of rock pressure load upon the support longwall displacement rate should be not less than 4 m/day.

7. It has been determined that if stope displacement rate is not less than 4 m/day, then $L = 65$ m and $L = 60$ m distances between longwalls of contiguous seams are safe for underworking and overworking respectively according to rock pressure factor.

CHAPTER 4. EXPERIMENTAL STUDIES OF ROCK PRESSURE MANIFESTATION IN STOPES

Set of field studies has been carried out in the stopes of “Vidrodzhennya”, “Bendiuzka”, “Lisova”, “Stepova”, and “Velykomostivska” mines of “Lvovugol” SE to test basic abstract theorems, to specify the technique, and to obtain actual values of the considered parameters in terms of different variants of pillars and contiguous seam selvages effect upon rock pressure manifestation.

The studies were performed under the guidance and with the direct involvement of the authors. Actual convergence values of wall rocks in the longwalls, strength parameters of the support were determined; failure behavior of the roof and seam was studied [107].

The research was carried out according to the specially developed technique taking into account basic statements and requirements of the known branch techniques [72].

Moreover, to have confidence estimation, calculation data concerning stress and strain state of the rocks around in-seam working are given; the calculations are performed according to the developed algorithm for the conditions being simulated in VNIMI using equivalent materials [108]. The corresponding results are compared.

4.1. Characteristics of underground research conditions

The research are performed in the longwalls developing the seams being most characteristics for the region within the zones of high rock pressure from pillars and selvages in terms of overworking and underworking. For comparison, observations were performed outside the effect zones of pillars and contiguous seam selvages as well. Table 4.1 gives mining-geological and mining-technical longwall characteristics. Fig. 4.1 represents stratigraphic columns of seams and enclosing rocks. Table 4.1 shows that the extracted height of the mined seam varies within 1.4 - 1.6 m; amount of inclination is not more than 3° ; longwall length is 142-180 m.

Parting rocks are presented mostly by thin layers of argillites, aleurolites, and sandstone with the hardness $f = 3$ up to $f = 7$ respectively, coal hardness is $f = 1.5-2$ according to M.M. Protodiakonov’s scale. Parting thicknesses are from 8.8 up to 40.5 m.

Mining method is the one by longwalls along the strike. Longwalls are equipped with KM87 powered complexes with 1ГIII68 shearer. Fig. 4.2 – 4.5 represents excerpts from mining plans and layout of the effecting pillars relative to the selected longwalls.

Table 4.1

Mining-geological and mining-technical characteristics

No.	Mine, longwall, seam	Longwall length, m	Seam extracted height, m	Seam roof	Seam floor	Seam thickness	Available underworking or overworking	Longwall displacement relative to pillars (selvedges)
1	2	3	4	5	6	7	8	9
1	“Lisova”, 160, n_7^H	143	1.60	argillite $f=4-2\dots 8,5$ m aleurolite $f=6-2\dots 6$ m n_7 seam - 0,2 m aleurolite - 4 m sandstone - 2 m	sandstone - 3 m aleurolite - 6 m	n_7^e - 15.6 m n_8^e - 40.5 m	n_7^e seam is underworked in 1994; n_8^e seam is underworked in 1987	Under the selvedge of n_7^e and n_8^e seams
2	“Stepova”, 111, n_7^H	180	1.48	argillite $f=5-2$ m aleurolite $f=6-4$ m n_7 seam- 0,4 m aleurolite - 4 m sandstone $f=6-5$ m	aleurolite - 7 m	n_8^e - 40.2 m	n_8^e seam is overworked in 1984	Under the pillar n_8^e seam
3	“Bendiuzka” 440, n_8	142	1.60	argillite $f=4-3,2$ m aleurolite $f=6-5,6$ m	aleurolite - 2.4 m sandstone - 5.8 m aleurolite - 4.2 m argillite - 3.9 m	n_8^e - 8.8 m n_7^e - 16.3 m	n_8^e seam is overworked in 1998 n_7^e seam is underworked	Outburst from under the selvedge of n_8^e seam

1	2	3	4	5	6	7	8	9
4	“Vidrodzhen nya”, 421, n_8	175	1.60	argillite – 3.4 m aleurolite – 4 m sandstone – 5 m	argillite – 3.45 m sandstone – 6.1 m argillite – 6.2 m	n_8^6 - 11.8 m n_7^H - 29.4 m	n_8^6 seam is underworked in 1962 n_7^H seam is underworked in 1979	Displacement under perpendicular pillar
5	“Bendiuzka” 415, n_8	152	1.40	sandy clay shale $f=5$ – 2 m sandstone $f=7$ – 2,5 m sandy clay shale – 3 m sandstone – 1 m	sandstone – 4.5 m sandy clay shale – 5 m	n_7^6 - 13.0 m	n_7^6 seam is underworked 1974	Passing under pillars
6	“Velykomost ivska”, 409, n_8	165	1.43	argillite $f=4$ – 1.5 m aleurolite $f=6$ – 2.5 m sandstone $f=7$ – 4 m алевролит $f=6$ – 2 m	aleurolite – 1.5 m sandstone – 4 m	n_7^6 - 12.0 m n_7^H - 19.0 m	n_7^6 and n_7^H are underworked	Passing under two pillars Sand
7	«Velykomost ivska», 408, n_8	165	1.45	argillite $f=4$ – 1.5 m aleurolite $f=6$ – 2.5 m sandstone $f=7$ – 4 m aleurolite $f=6$ – 2 m	aleurolite – 1.5 m sandstone – 4 m	n_7^6 - 12.0 m n_7^H - 19.0 m	n_7^H is underworked	parallel pillar overworking of n_7^H seam
8	«Lisova», 536, n_8^6	150	1.30	argillite $f=4$ – 2...3 m sandstone $f=4$ – 8...15 m aleurolite $f=6$ – 2,5 m	aleurolite – 4 m sandstone – 10 m	n_7^6 - 28.2 m n_7^H - 24.1 m	n_7^H is underworked	parallel pillar overworking of n_7^H seam

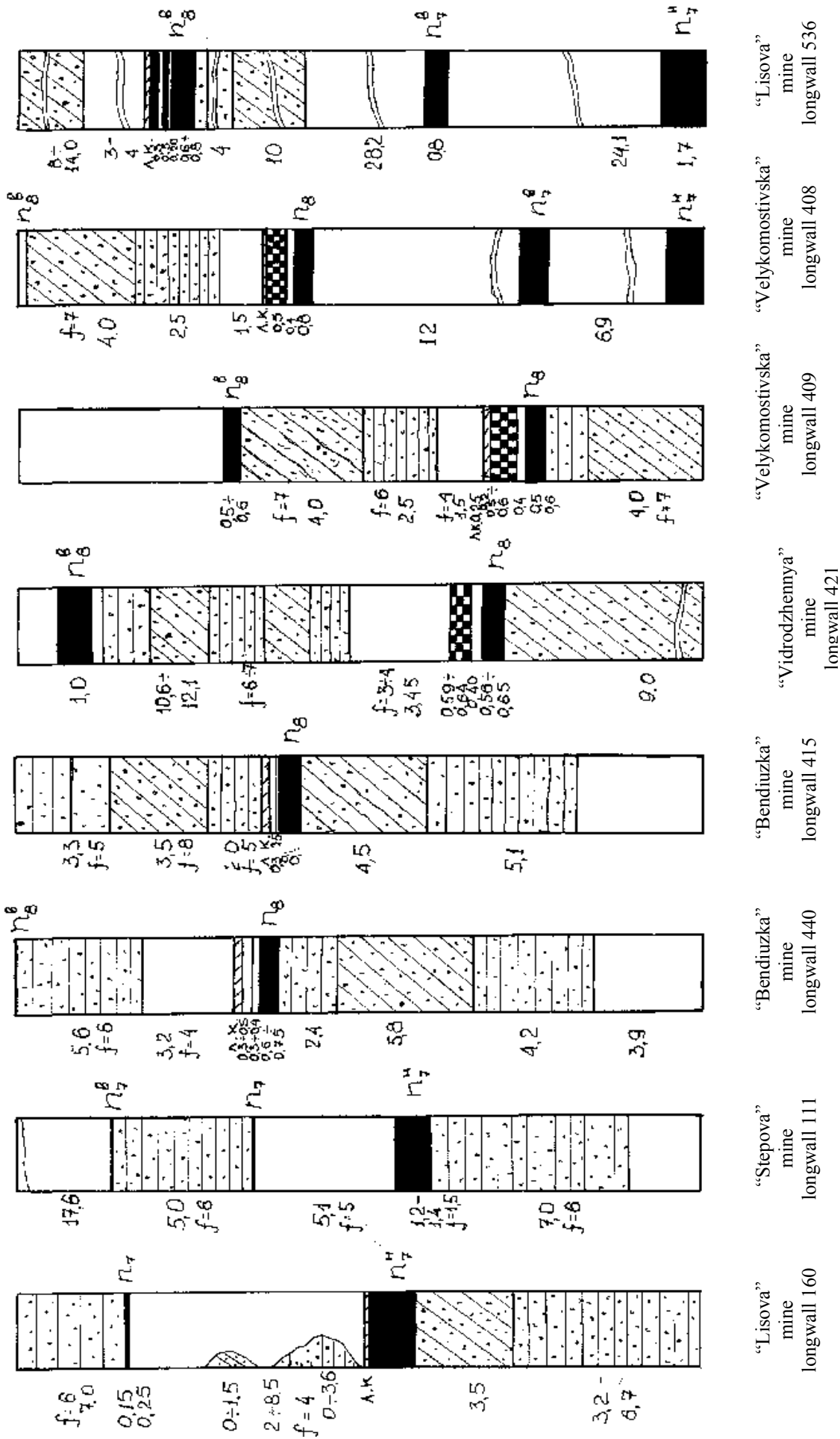


Fig. 4.1. - Stratigraphic columns of rocks and enclosing rocks

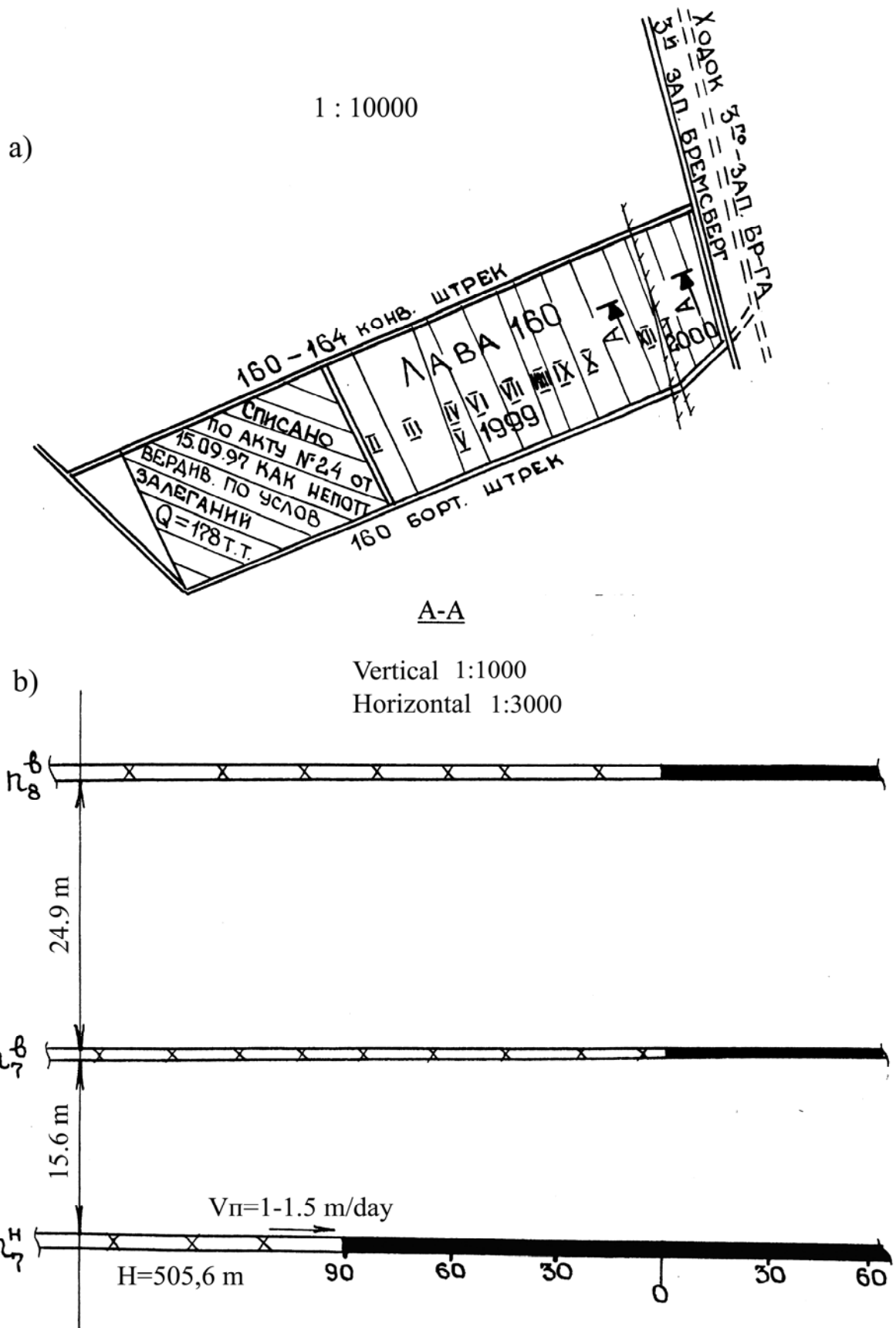


Fig. 4.2. - Excerpt from the mining plan (a) and longitudinal plan of the extraction pillar of longwall 160 (b)

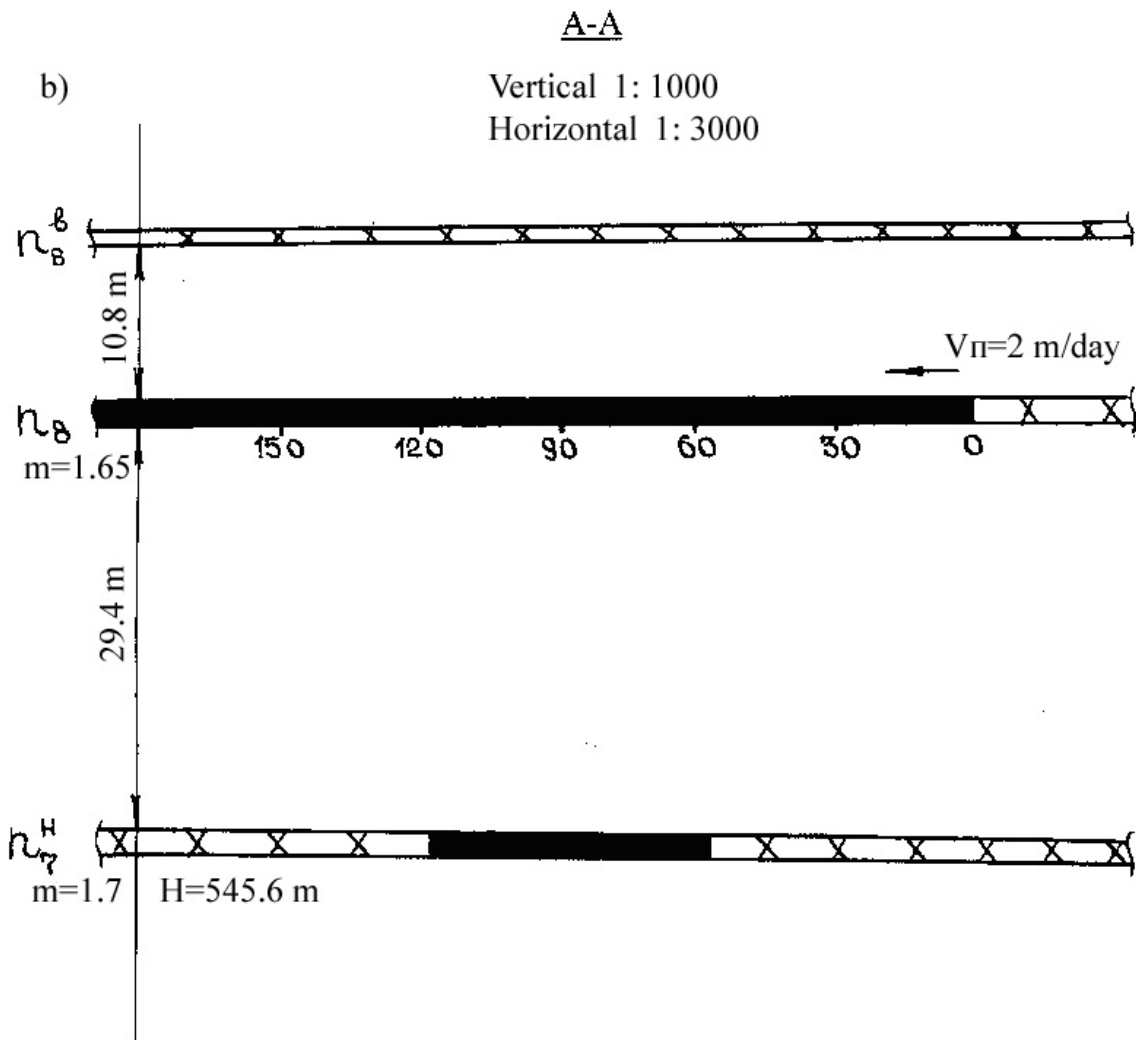
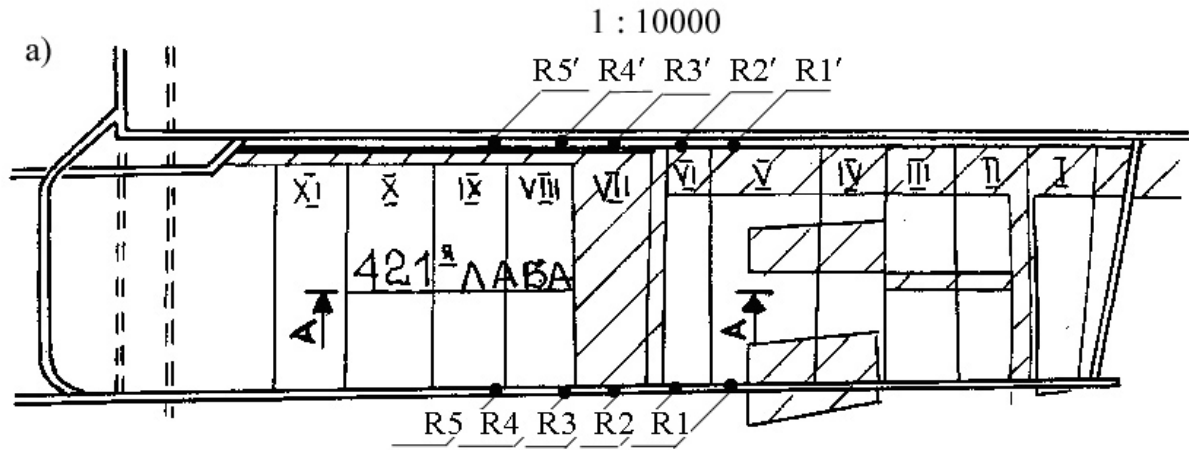


Fig. 4.3. - Excerpt from the mining plan (a) and longitudinal plan of the extraction pillar of longwall 421 (b)

a)

1:10000



b)

A-A

Vertical 1:1000

Horizontal 1:3000

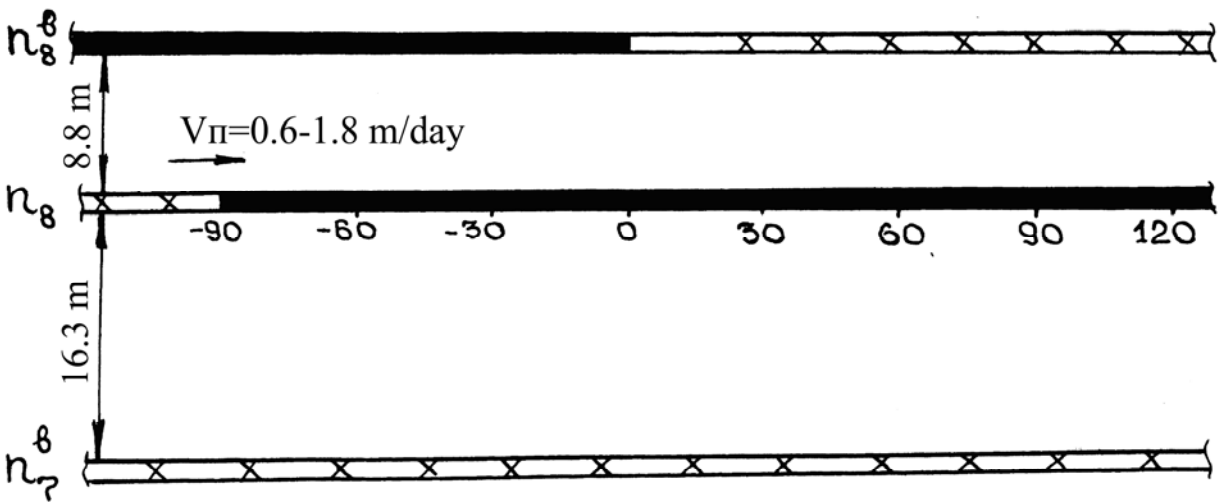


Fig. 4.5. - Excerpt from the mining plan (a) and longitudinal plan of the extraction pillar of longwall 440 (b)

4.2. Results of underground research

Studies of rock pressure manifestation within the longwall operating area included field observations as for the character of powered support interaction with wall rocks, rock inrushes of the immediate roof, change of values of wall rock displacements in the longwall operating area, and actual resistance of support hydraulic props.

4.2.1. Wall rock convergence

Values of wall rock convergence within the operating area of the eight studied stopes were measured with the use of SUI-II gage posts by means of measuring distances between mine survey plugs fixed in the rock of seam roof and floor at the distance of 0.8 m from each other across the longwall width. Fig. 4.6 shows installation diagram for gage posts. The character of wall rock convergence as well as stope displacement rate determined measuring frequency.

Fig. 4.7 explains the changes of wall rock convergence rate across the longwall width in terms of its various positions relative to pillars and contiguous seam selvages.

Curves 1 in these figures show convergence outside the pillar effect zone (the selvedge); curves 2 show convergence within high rock pressure zone, in terms of approaching the pillar (the selvedge); curves 3 show convergence during operation under (over) the pillar (the selvedge); and curves 4 show convergence within high rock pressure zone in terms of leaving the pillar (the selvedge). The figures show that during longwall operation within the pillar (selvedge) effect zone, roof rock fault near longwall face (within up to 1.6 length) increases by 2-6 times, from 8-15 mm/cycle up to 38/46 mm/cycle. At the same time rock convergence at the boundary of face area with the worked-out rocks decreases by 1.5-2 times, from 45-60 mm/cycle down to 27-42 mm/cycle. Total rock convergence is 180-304 mm per cycle.

Fig. 4.8 and 4.9 represent values of wall rock convergence rates along the length and width of longwall No.421 of "Vidrodzhennya" mine during operation within the pillar effect zone and outside it. Maximum convergence values within the pillar effect zone were observed near longwall face; they were confined to the process of seam extraction being 2.6 mm/min (curve 1 in Fig. 4.8). At 0.8 m distance from the face convergence while extracting was 1.7 mm/min while at 1.6-2.4 m distance (curves 3 and 4, Fig. 4.8) it was practically constant. Support advancing had minor effect upon convergence rate. Its maximum during technological operation was 1.7 mm/min being fixed right near longwall face within the zone of about 1.6 m width; further convergence rate in terms of longwall width was constant.

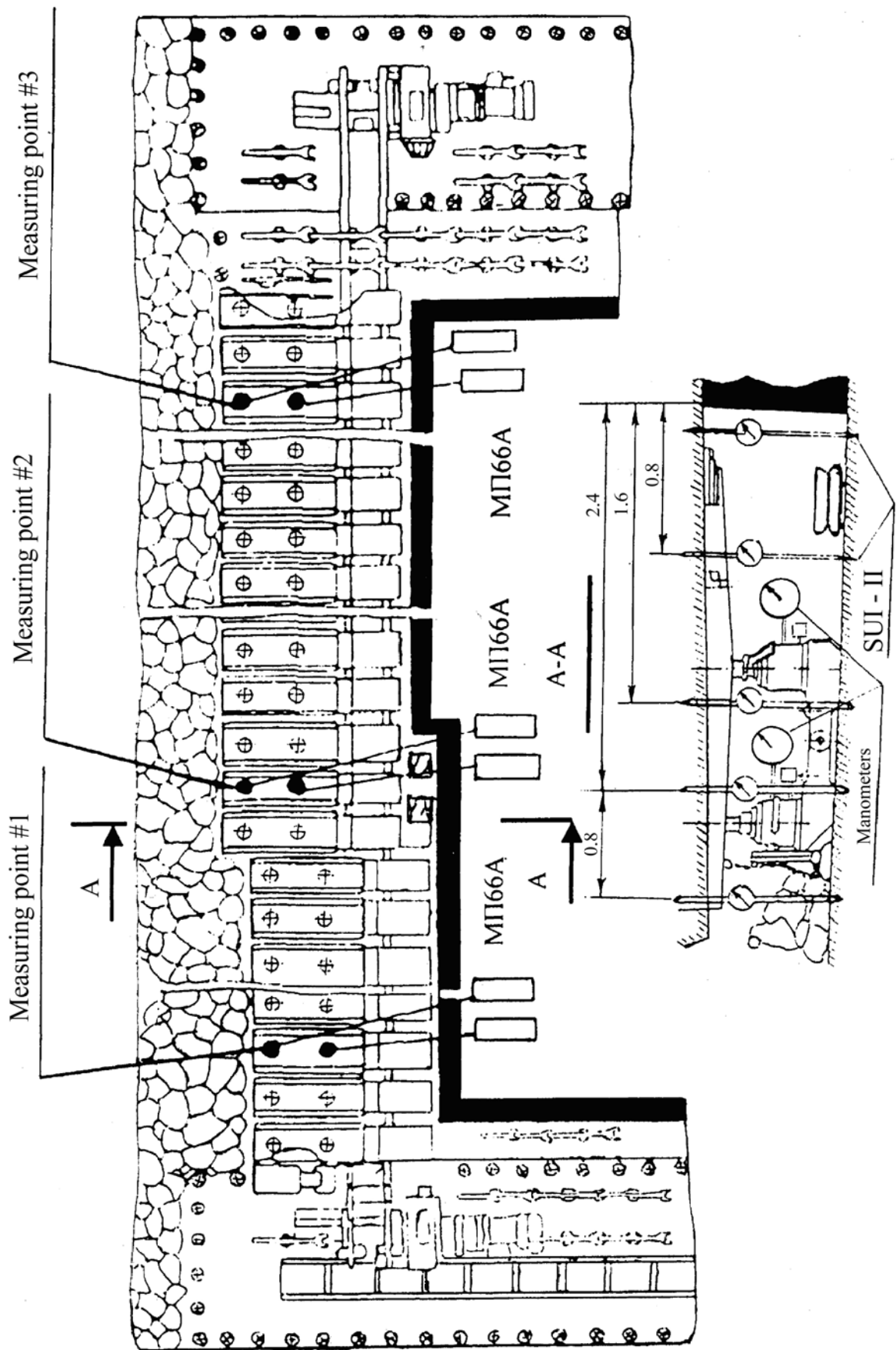


Fig. 4.6. – Layout of measuring points in the longwall

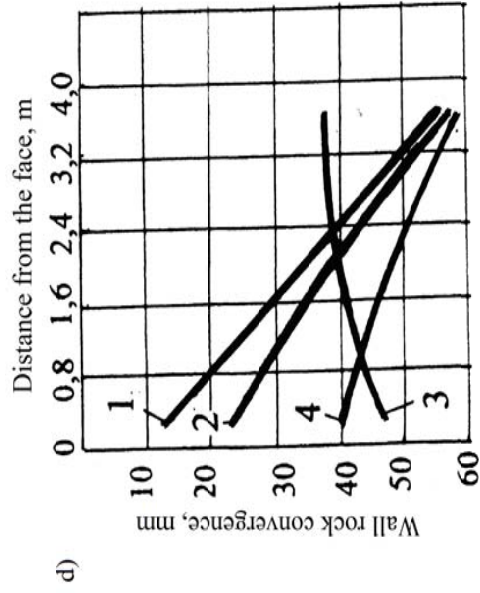
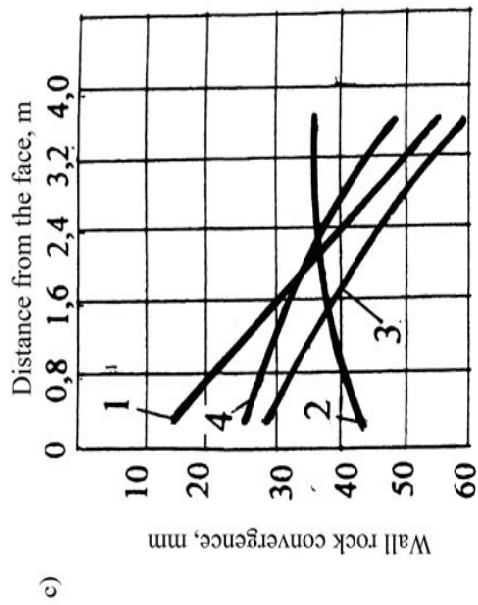
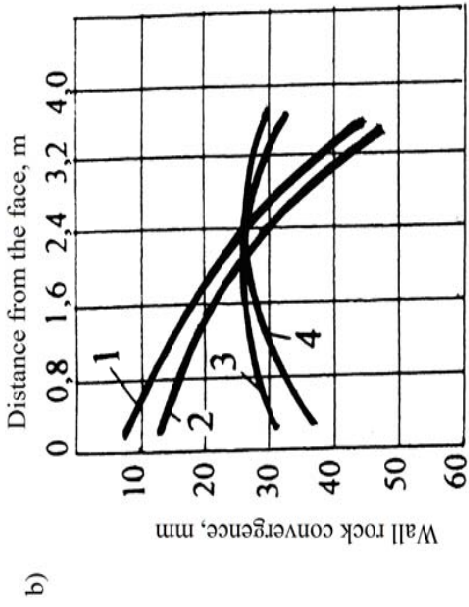
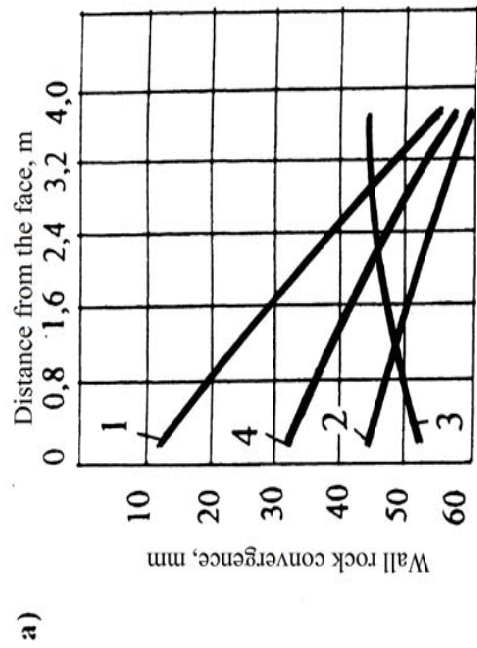


Fig. 4.7. – Convergence of roof and floor rocks according to longwall width: a) under the pillar; b) above the pillar; c) while displacing under the selvedge; d) when leaving from under the selvedge

1 – outside pillar (selvedge) effect; 2 – when approaching the pillar (selvedge); 3 – during the operation under (over) the pillar (selvedge); 4 – when leaving the pillar (selvedge).

The situation outside the pillar effect zone is quite different: effect of technological processes can be observed across the whole longwall width; besides, support relocation effects convergence rate in a greater degree (Fig. 4.9).

According to the observations, convergence rate near the face depends greatly upon the shearer feed rate. Maximum convergence values could be seen when the shearer feed rate was 4.2 m/min being 8 mm/min; minimum values were 2 mm/min at 1 m/min rate. In the first case the effect of extraction process spread in front of the cutter loader over 2 m; in the second case it spread over 21 m. It can be seen in Fig. 4.10.

All the studied longwalls showed increase of seam spall from 0.4 up to 1.0 m and deterioration of roof state in longwall face area when feed rate of a cutter loader increased from 2.4-3.2 m/min up to 3.8 – 4.5 m/min. It is explained by the fact that when the size of longwall area being effected by the stoping operations decreases, absolute convergence values increase dramatically resulting in intense fissure formation in soft roof rocks and in seam selvedge spall.

Operating mode at 3.0...3.2 m/min feed rate of a cutter loader was the most favourable one for powered complexes. It is possible to ensure high mining productivity at such rates by means of mechanism shutdown decrease and increase of machine operating time for cutter loaders. With this view, the stopes have been changed over five-shift operation since 2001 according to the authors' recommendations.

Mine observations in all the considered longwalls have shown that increase of V stope advance rate improves roof condition. In particular, it was noticed that V increase from 1.2 m/day up to 4.2 m/day reduces the area of longwall intrushes by 5-8 times other factors being equal.

Implementation of the authors' developed recommendations into production has shown that if there is five-shift longwall operation at $V_k = 3.0...3.2$ m/min shearer feed rate, stope displacement is possible at $V = 4.0...4.2$ m/day rate. In this case, output per face is 150 t/day.

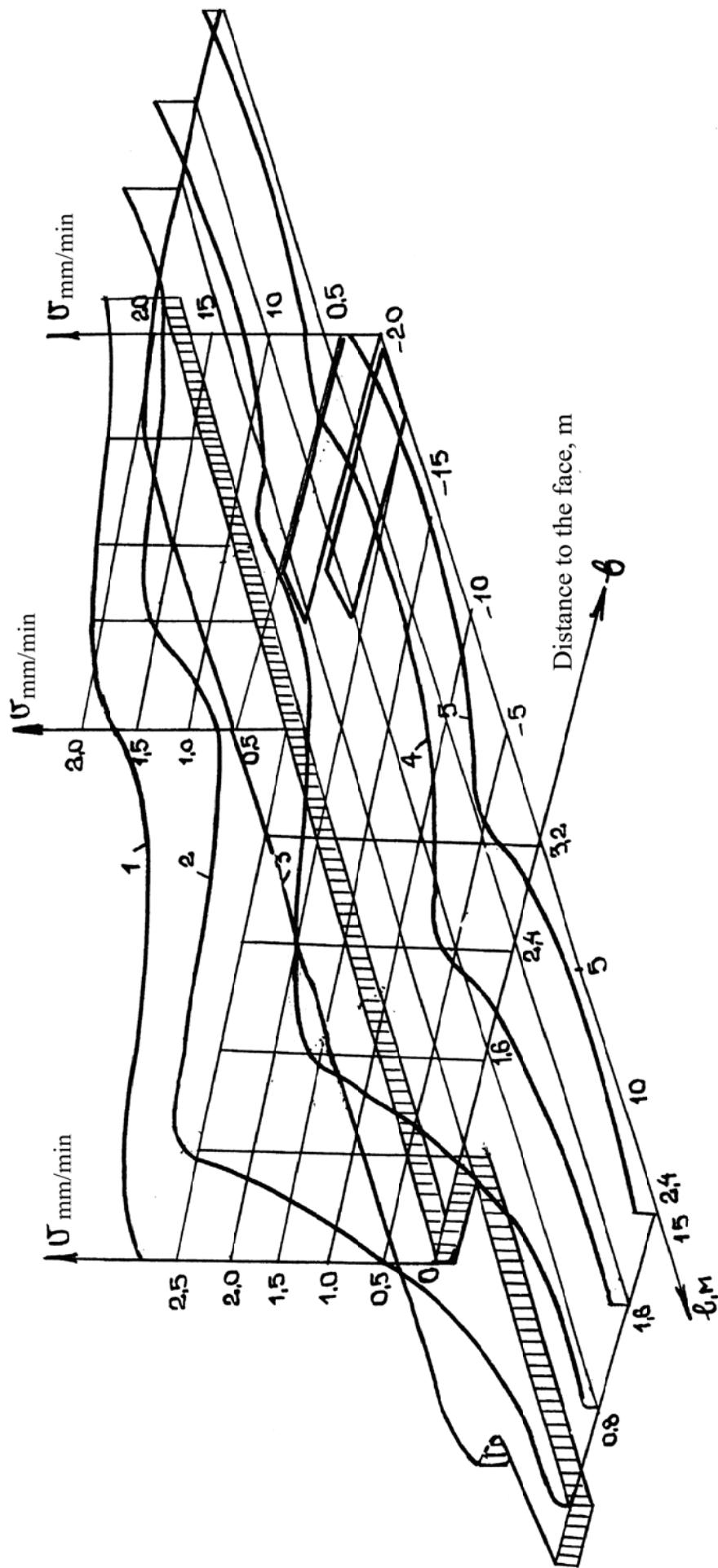


Fig. 4.8. —Change of wall rock convergence rate along the length and width of longwall No. 421 within the pillar effect zone
 1 — distance from the face is 0.2 m; 2 — 0.8 m; 3 — 1.6 m; 4 — 2.4 m; 5 — 3.2 m.

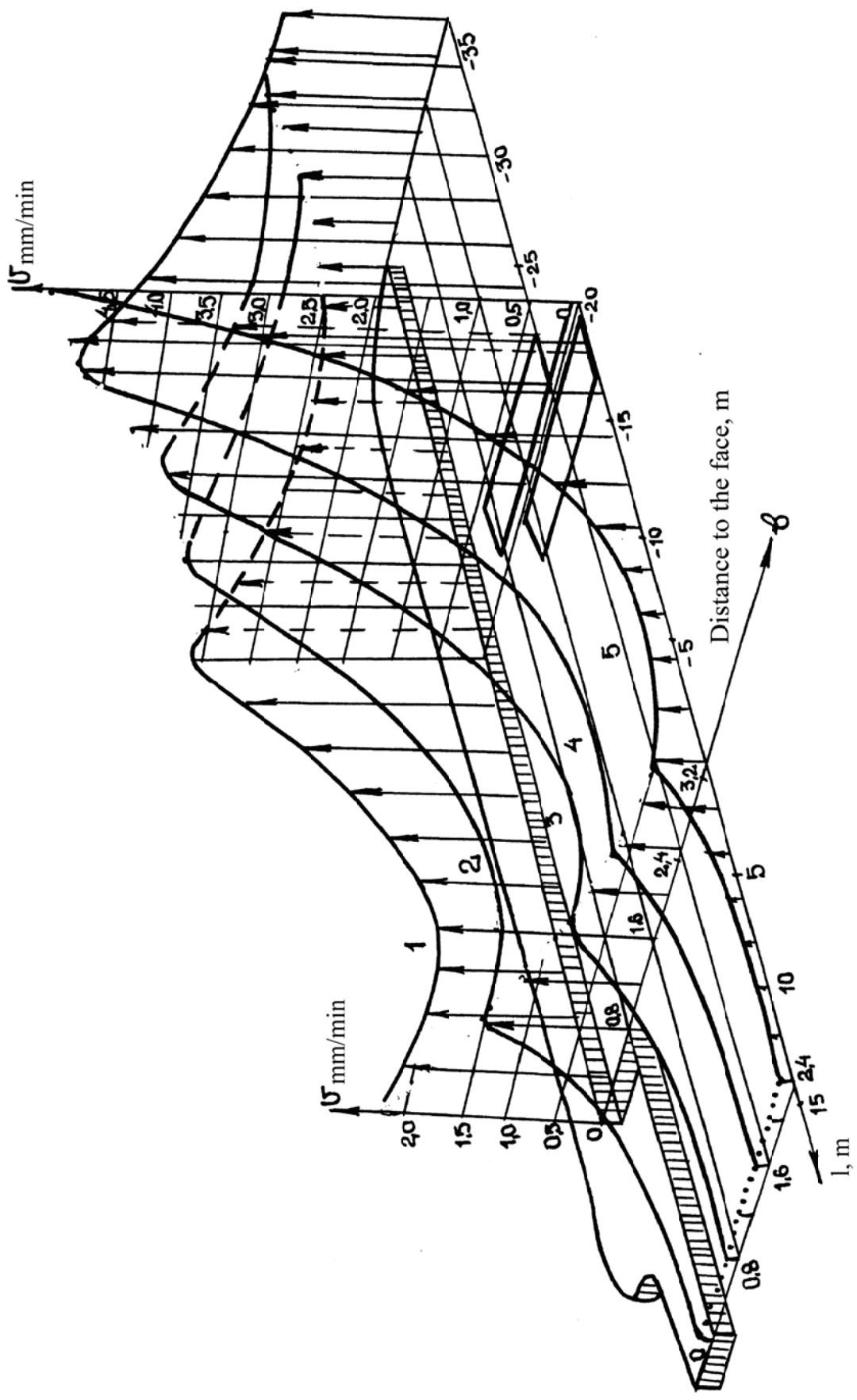


Fig. 4.9. - Change of wall rock convergence rate along the length and width of longwall No. 421 outside the pillar effect zone

1 - distance from the face is 0.2 m; 2 - 0.8 m; 3 - 1.6 m; 4 - 2.4 m; 5 - 3.2 m.

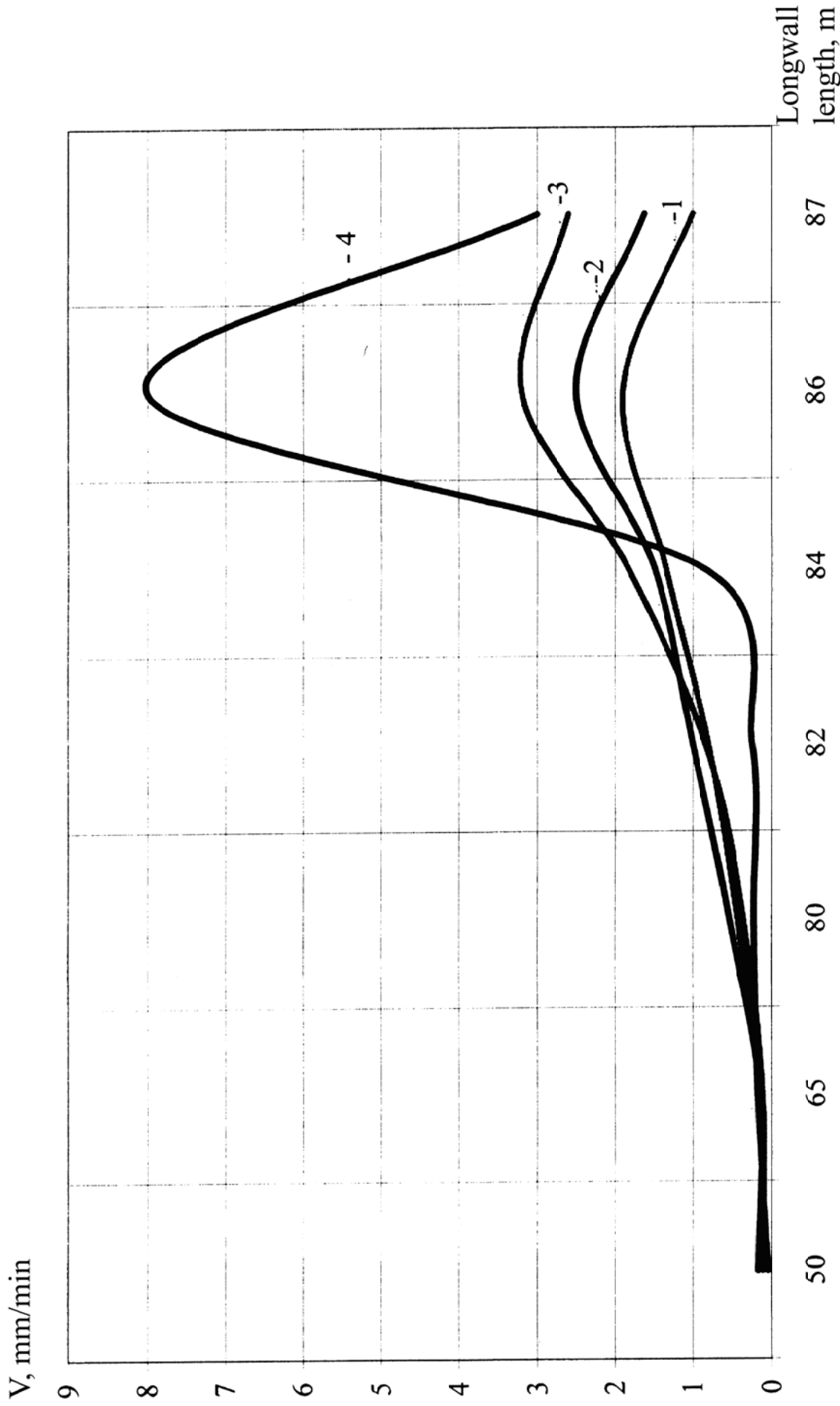


Fig. 4.10. - Change of wall rock convergence rate near the face along longwall length in terms of different rates of shearer feed rate: 1 - V=1 m/min; 2 - V=2 m/min; 3 - V=3 m/min; 4 - V=4 m/min.

4.2.2. Strength parameters of the support

Actual resistance of hydraulic props of KM 87 powered support units was measured by M66A registering manometers and clock-type manometers being connected into working spaces of front and back props. Measuring points were mounted on three sections in the middle longwall part and at the distance from 15-20 m from entries. Fig. 4.6 illustrates mounting of measuring points. Measuring range of each longwall was not less than 100 cycles or 60 m stoping advance.

Observations of rock pressure manifestation in the selected longwalls of “Lvovugol” SE mines have shown that support load varies during winning cycle; the load is formed in a different way depending on the condition and properties of roof rocks. Two main loading types are singled out.

First type (Fig. 4.11) is characteristic in the fact that hydraulic props operate in the mode of progressive resistance during one winning cycle. After relocation and setting of units, one could observe load increase with its following stabilization or slight growth during the whole winning cycle. While winning next web and relocating the neighbouring units, the load increases dramatically, reaching maximum values before unit relocation. Such load character is most often to be seen outside the effect zones of pillars and contiguous seam selvedges. Under such conditions values of initial setting of front and back props were 280 and 360 kN being 43.1% and 55.4% respectively from the load-bearing capacity of a hydraulic prop.

Average maximum load values (at $V = 1.2 - 2.0$ m/day stope advance value) were: 61.5% (in front props), 67.7% (in back props) from load-bearing capacity of the prop being equal to 370 - 450 kN and 390 - 485 kN respectively.

While operating outside the effect zones of pillars (selvedges), average values of maximum loads upon back props are higher than upon front ones in all the studied longwalls due to following factors:

- higher values of the initial setting specified by the back props;
- higher value of rock fault at the level of back props;
- available overhang of hanging rocks of the immediate roof which weigh is imparted mostly to the back props.

According to observation data, overhang length was 1 - 1.5 m (Fig. 4.11). Relief valve action was observed only in 5% cases (in back props) and in 2% cases (in front props).

All the cases of valve actions were due to lengthy downtime of stopes.

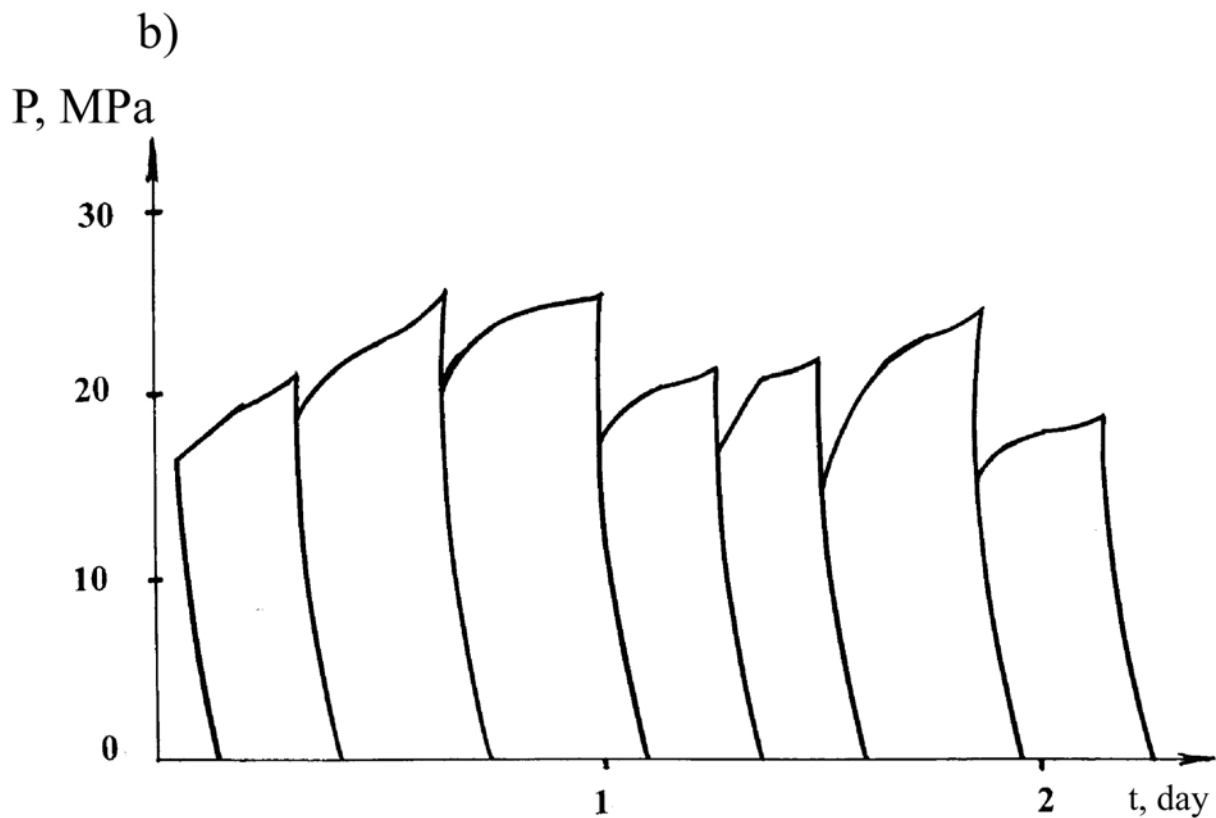
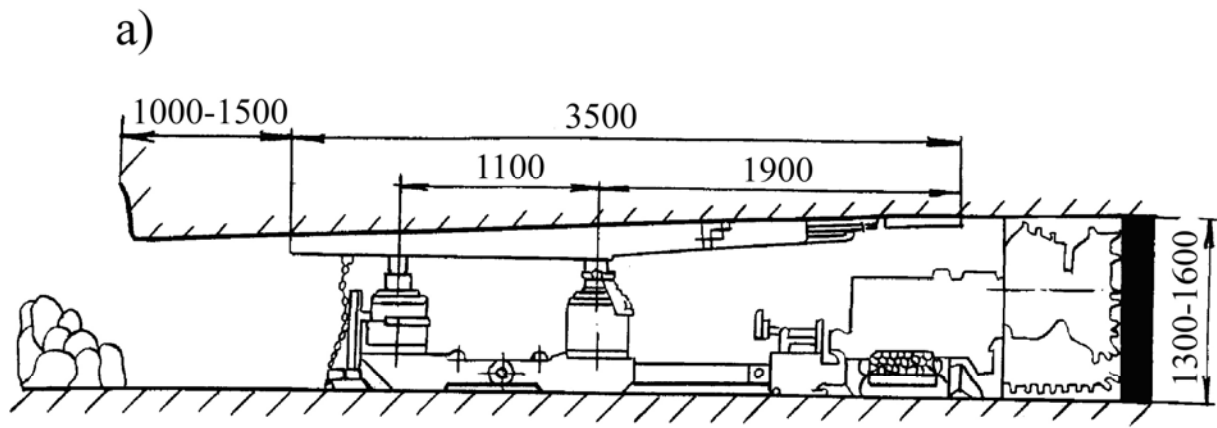


Fig. 4.11. – Roof rocks condition (a) and loading condition of KM87 support hydraulic props (b) while operating outside the effect zone of high rock pressure

Second type of support load (Fig. 4.12) is specific in the fact that hydraulic props operate in different modes, i.e. one prop operates with progressive resistance another one operates with regressive resistance.

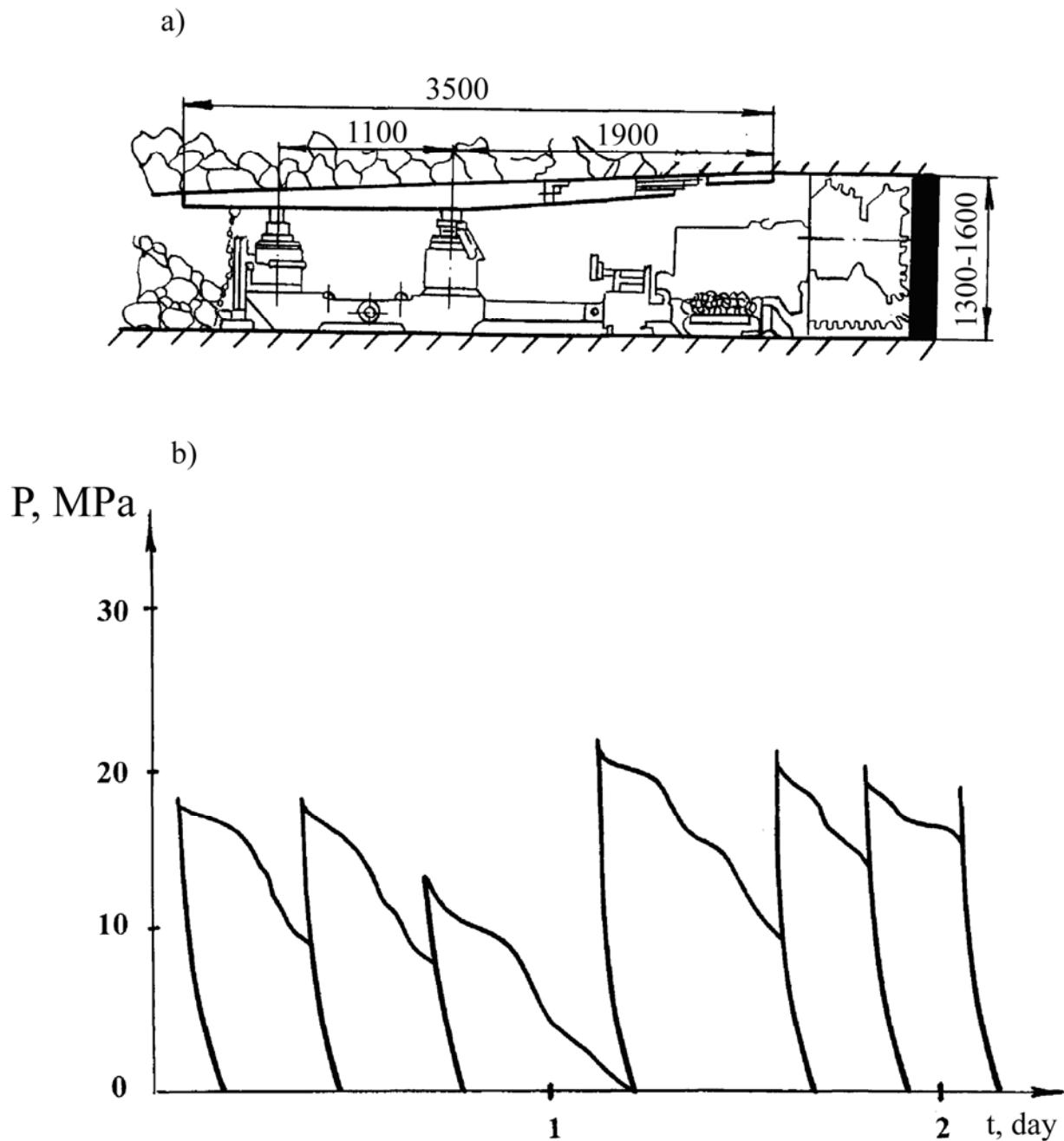


Fig. 4.12 - Roof rocks condition (a) and loading condition of KM87 support hydraulic props (b) while operating within the effect zone of high rock pressure

After relocation and setting of units stress in back prop usually decreases and stress in front prop usually increases. Such cases are seen mostly when longwall operates within the effect zone of pillars and selvedges, especially when approaching or leaving them. Thus, when 111 longwall approached the pillar being on the

overlying seam, some cases of reaction increase by 120-180 kN in front props of the support and reaction decrease by 20-50 kN in back props were observed. Such character of support load formation can be explained as follows. Rock “cushions” are formed on support canopies within high rock pressure area due to the excessive fissuring of the immediate roof. In this case, first rock contact with the canopy usually occurs at the distance of 0.6 – 1.0 m from the face. Canopy almost loses contact with the roof above back prop. In general, support loads within this area were close to maximum ones falling on front prop.

When leaving the pillar, the situation differed; support operated according to the first loading type.

When longwall 421 approached to the pillar being on underlying seam, loads on back support props increased (from 340 up to 420 kN). While coming to the pillar edge and displacing from it, front prop loads increased dramatically (from 400 up to 600 kN), several relief valve actions were noticed (27% cases). The same situation was observed in longwall 415; however, load values were lower by 20 - 60% that can be explained by high stope advance rate (3.6 – 4.8 m/day).

When longwalls operated within the zone of immediate roof rock inrushes there was a situation when both hydraulic props operated in the mode of decreasing resistance. Relocation and setting of units were followed by load decrease, their stabilization or slight decrease during the whole winning cycle. During the winning the following web and relocation of neighbouring units, loads changed (increase) slightly. In addition, roof rock inrushes at considerable height were recorded. Such situation was observed at low stope advance rates within the effect zones of pillars and contiguous seam selvedges.

Table 4.2 gives average measuring results characterizing support operation within different zones of high rock pressure due to pillars (selvedges) as well as outside these zones.

The analysis of the results of support strength parameters study shows that when longwall operates over (under) pillar edge or displaces from it, load on support front prop increases considerably due to intense rock fault within face area and seam selvedge spall. Moreover, back prop does not practically operate (this circumstances is considered in the first scientific statement). It indicates the necessity to correct support structure to increase support resistance near the face.

Shortening of front overhang of KM87 support by 0.33 m has become the first step in this direction. This proposition of the authors was implemented in “Velykomostivska” mine No.4 (now it is “Vidrodzhennya” mine).

This technical solution has allowed improving support condition in longwall face area, to decrease probability of “stiff” support caving, increase operation safety.

Table 4.2

Resistance of support hydraulic props in various HRP zones

Parameters Mine, Longwall	Direction of longwall displacement	Mining condition	Face position	Longwall advance rate, m/day	Resistance of hydraulic props, kN											
					front			back								
					Initial setting	working resistance		Initial setting	working resistance							
"Lisova" mine, Longwall No.160	under the selvedge	over working	L = -10 m	1.2	310-340	520-600	240-340	220-280								
											L = -30 m	3.6	340-360	520-600	340-360	230-250
"Stepova" mine, Longwall No.111	under the pillar	over working	L = 10 m	0.6	320-360	460-520	280-340	440-500								
											"Bendiuzka" mine, longwall No. 440	from under the selvedge	over working	L = -90 m	1.2	300-340
"Vidrodzhennya" mine, Longwall No. 421	above the pillar	under working	L = 25 m	1.8	320-330	460-480	270-280	360-410								
											L = -30 m	1.8	340-350	530-580	240-275	205-235
	outside the effect zone	under working		1.2-2.0	280-360	350-450	290-360	380-485								

"

4.2.3. Estimation of the stope roof condition.

Stope roof condition was estimated during longwall operation within the zones of high rock pressure due to pillars and selvages as well as outside these zones.

Studies were carried out in longwalls No. 421 of “Vidrodzhennya” mine, No.160, No.556 of “Lisova” mine, No.411 of “Stepova” mine, No.408 and No.409 of “Velykomostivska” mine, and in longwalls No.415 and 440 of “Bendiuzka” mine.

The research involved measuring geometrical dimensions (length, width, and height) of rushed-in rock domes in longwall face area per last winning cycle.

Observations were carried out 2-3 times a week; if roof condition deteriorated or when longwall operated within the effect zone of contiguous seam selvages observations were carried out every day.

Average inrush height on each powered support unit was taken as the first criterion to evaluate stope roof condition. This parameter was obtained by means of summing up inrush height according to each measurement with following division of the obtained sum by the number of measurements, i.e.:

$$H_{cp,i} = \frac{\sum_{j=1}^n H_j}{n},$$

where $H_{cp,i}$ is the average inrush height on i -th powered support unit, m;

H_j is inrush height on i -th powered support unit according to one of the measurements, m;

n is the number of measurements.

The criterion allows determining roof inrush distribution along the longwall length, defining areas with the most intensive inrush formation as well as the areas being free of inrush or with minor inrush cases.

Fig. 4.13 shows roof inrush distribution throughout the height in terms of different longwall position relative to the effecting pillar or selvedge as well as stope advance rate.

The distribution shows that if there are any inrushes outside HRP zones, they have minor height being equal to 0.5 m on average. Speaking about the effect zones of pillars and selvages especially when leaving them, here inrush height can be as high as 3.8 m.

Specific parameter of roof disturbance with inrushes (K_ϵ) is taken as the second criterion of roof condition:

$$K_\epsilon = \frac{\sum S_\epsilon}{S_0} \cdot 100\%,$$

where S_ϵ is inrush area, m²;

S_0 is the area of observation zone, m².

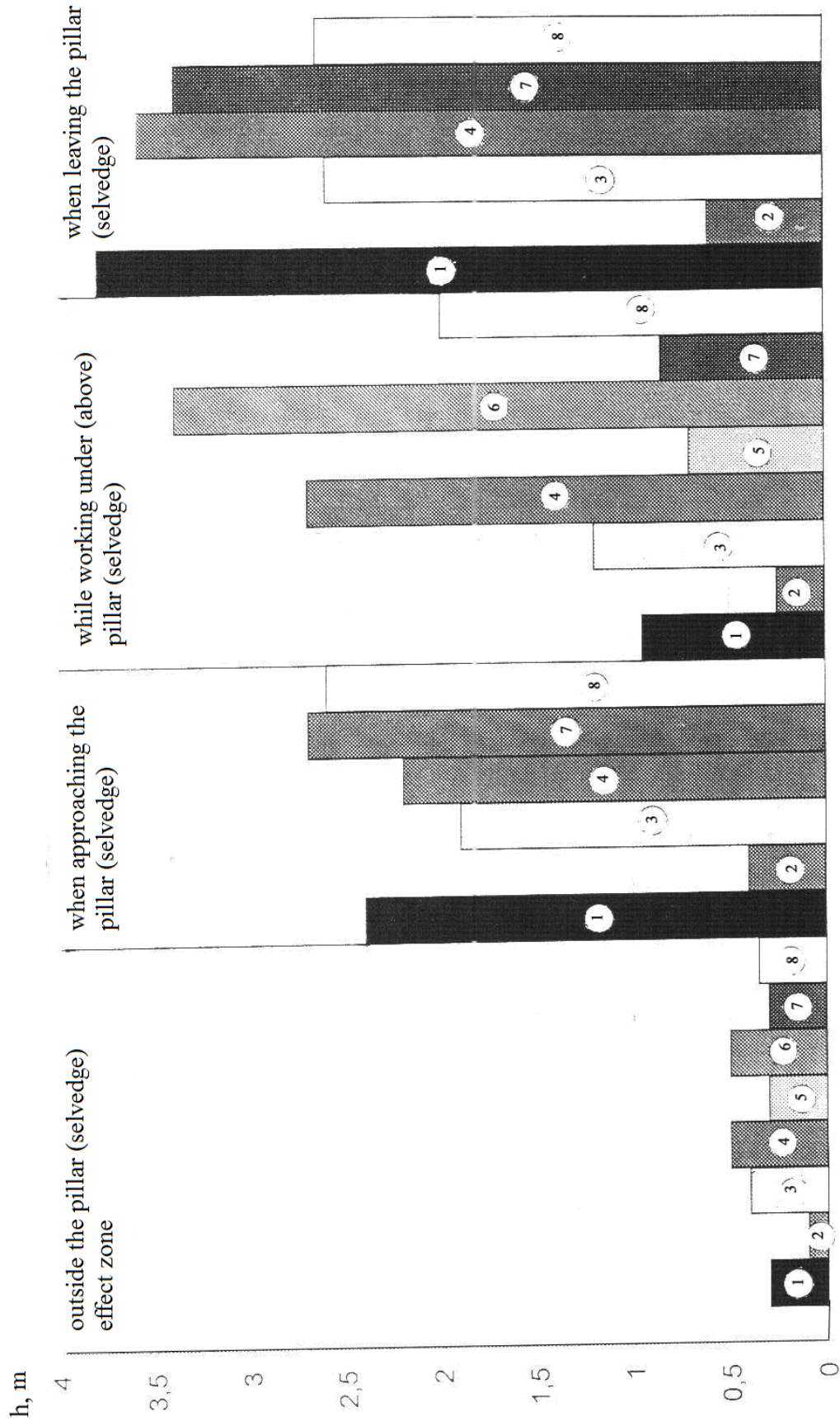


Fig. 4.13 – Roof inrush distribution according to the effect zones of pillars (selvedges) in longwalls:
 1 – 421; 2 – 415; 3 – 160; 4 – 440; 5 – 536; 6 – 408; 7 – 409; 8 – 111.

Fig. 4.14 represents the change of specific inrush area depending on the distance between the support canopy contact point and the face. Solid lines belong to the case when longwall displaces at 1.8 m/day rate; dot lines belong to the case when $V = 4.2$ m/day. Specific inrush area in front of the support increases along with the distance growth between the face and the first contact point of the cap and roof. The figure shows that stope advance rate effects considerably the inrush area. Thus, stope advance rate increase from 1.8 m/day up to 4.2 m/day reduces inrush area in the longwall by 5 – 8 times from 30...65% to 5...8% other factors being equal.

Specific inrush formation of roof rocks is the third criterion that makes it possible to compare quantitatively roof rock condition.

The essence of this criterion is in the fact that the general inrush volume per last winning cycle (within the definite longwall area) is divided by the length of this area. In other words, average inrush volume relative to 1 linear meter of a stope is (V_{cp}) specific inrush formation. This parameter can be determined according to the formula:

$$V_{cp} = \frac{\sum_{i=1}^n V_{cp,i}}{l_{yq}},$$

where $V_{cp,i}$ is average inrush volume on i -th powered support unit, m^3 ;

l_{yq} is the considered area length, m.

Analysis of the obtained data (Fig. 4.15) allows stating that most inrush formation occurs not above (under) the pillars and selvedges of the influencing seams but when approaching /leaving them. Zones of intense inrush formation were at the distance from -5 up to +40 m from the pillar boundaries; the width of these zones depended upon pillar thickness, its position relative to the developed longwall, thickness and structure of the parting. If there were strong and thick parting layers or if parting thickness increased, then inrush formation zones reached its maximum values at the distance from 40..45 m from the pillar (selvedge).

While studying, clear dependence of inrush formation intensity upon stope advance rate was recorded. Thus, specific inrush formation in longwall No. 415 due to keeping high stope advance rate (3.6 – 4.8 m/day) was by 5-7 times less comparing to longwall No. 421 where stope advance rate within the selvedge effect zone was not more than 1.5 – 2 m/day.

In all the studied longwalls, growth of shearer feed rate from 3.0 up to 4.5 m/min when approaching the pillar and when leaving from under contiguous seam selvedge increased specific inrush formation from 10% up to 70%.

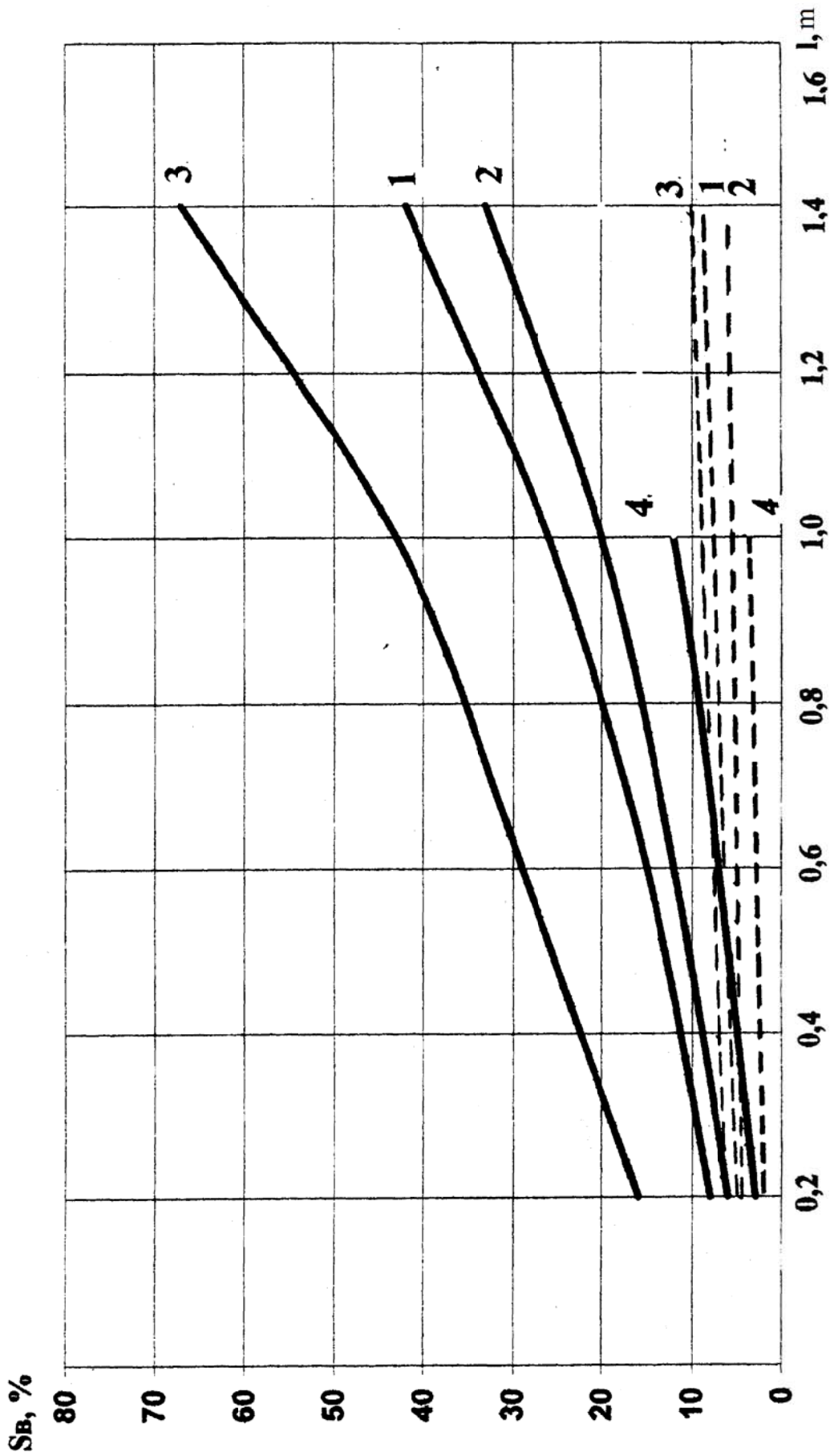


Fig 4.14. – Dependence of average specific inrush area in front of S_B support upon l distance between the first contact point of cap and roof and the face.

1 – when approaching the pillar; 2 – above the pillar; 3 – when leaving the pillar; 4 – outside pillar effect zone;
 Longwall displacement rate is — 1.8 m/day; ----4.2 m/day.

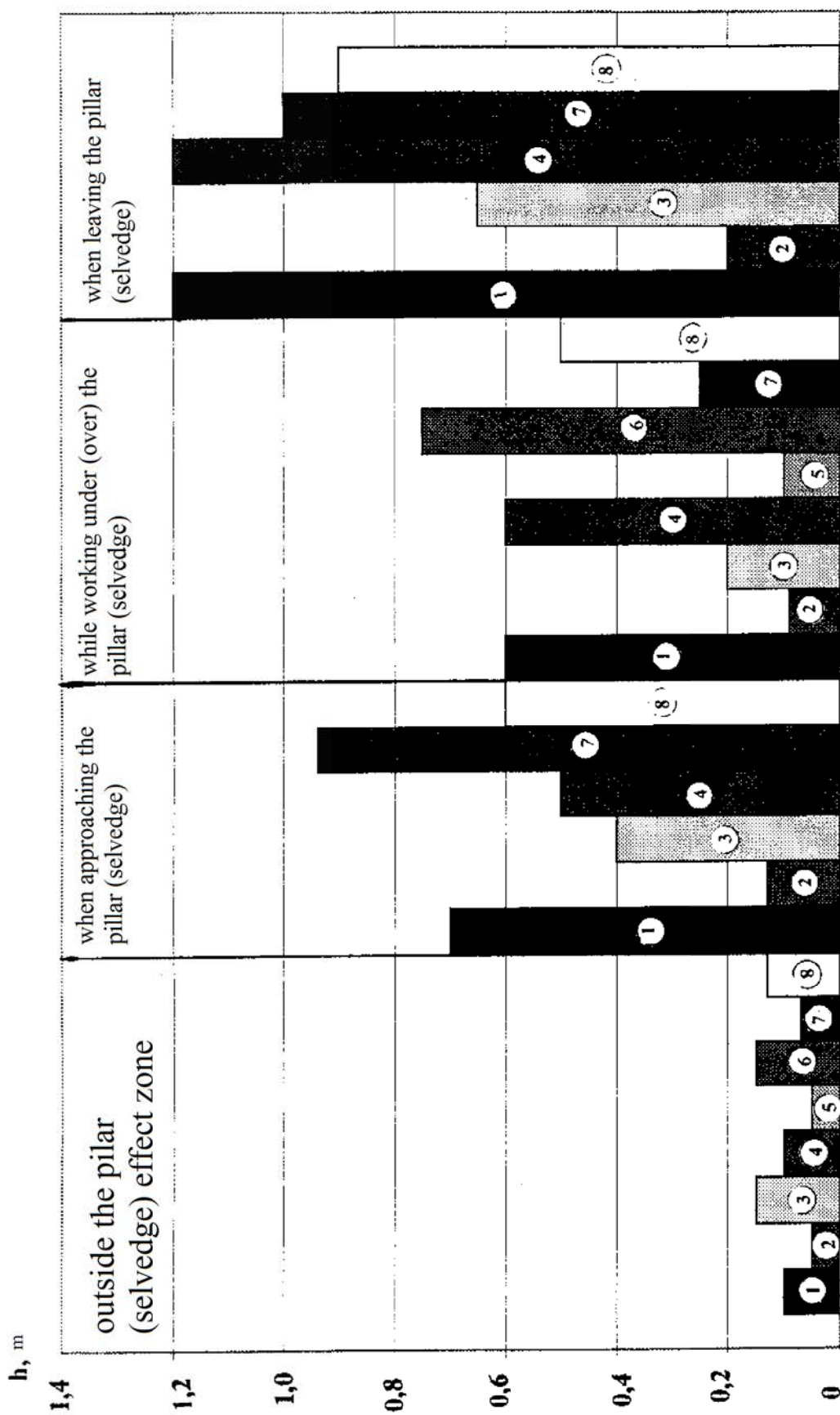


Fig. 4.15. – Specific inrush formation in longwalls while operating within different zones of pillar (selvedges) effect in longwalls: 1 – 421; 2 – 415; 3 – 160; 4 – 440; 5 – 536; 6 – 408; 7 – 409; 8 – 111.

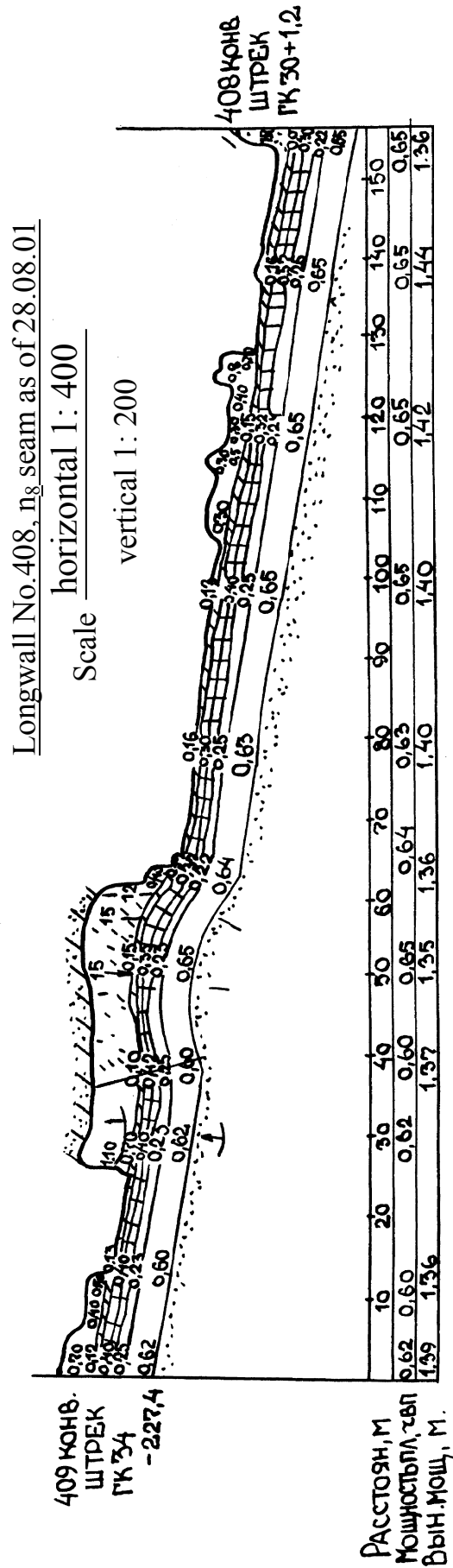
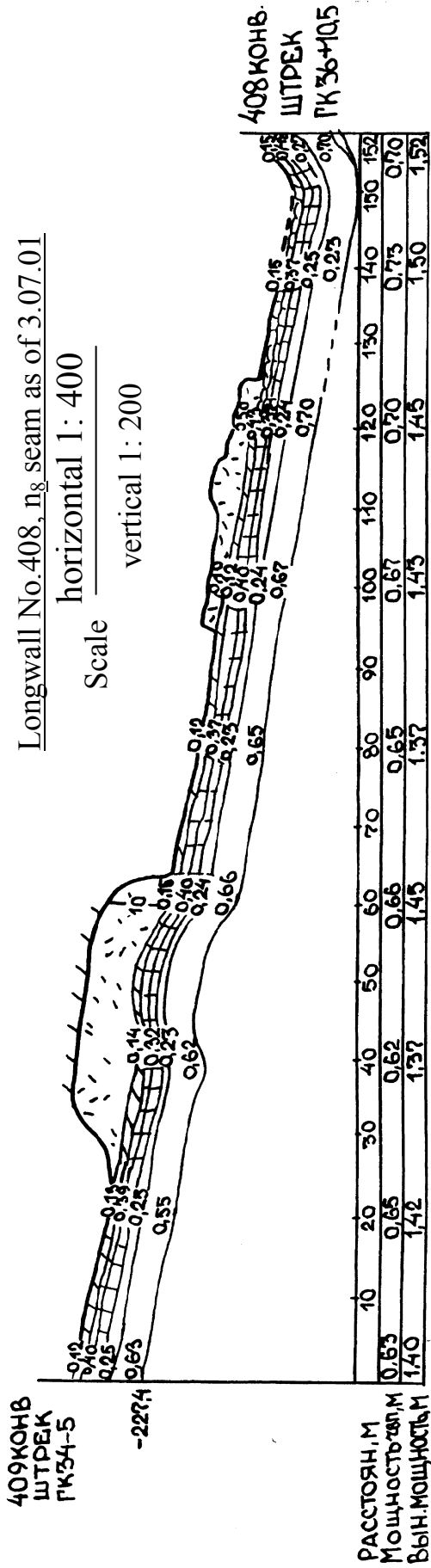


Fig. 4.16. — Sections within longwall 408 while operating over parallel pillar

Pillar effect upon inrush formation is most clear to see using the example of longwall 408 (Fig. 4.16) where roof rock inrushes occurred regularly over parallel chain pillar of n_7^H seam.

Hence, the results of mine studies confirm the second developed scientific statement as for the necessity to increase stope advance rate up to 4 m/day minimum within the effect zones of pillars and influencing seams.

4.3. Comparison of experimental and theoretical data

4.3.1. Comparison of calculation results and in-mine measurements.

Calculation of stress and strain rock state near a stope in terms of “Lvovugol” SE mining conditions according to the developed algorithm was performed to evaluate the reliability of the results of theoretical studies. The developed technique was used to obtain calculated resistance values of hydraulic props of the upgraded KM87 powered support.

Cases when longwall operates when approaching the contiguous seam selvedge and when leaving this zone were considered as well as the cases of longwall operation over the pillar in terms of undermining. Fig. 4.17 a, b, and c gives calculation models of the studied area for these cases.

Calculations were performed for two values of stope advance rate, $V = 1.2$ m/day and $V = 4.0$ m/day in terms of following initial data:

1) Actual load is $\gamma H = 11.5$ MPa.

2) Physical and mechanical properties are:

coal – $E = 0.28 \cdot 10^4$ MPa; $\nu = 0.3$; $\gamma = 1.69$ t/m³; $\sigma_c = 20$ MPa;

roof rocks (argillite) – $E = 0.63 \cdot 10^4$ MPa; $\nu = 0.2$; $\gamma = 2.6$ t/m³;
 $\sigma_c = 30$ MPa;

3) Geometrical parameters:

– extracting seam thickness, $m = 1.6$ m,

– size of worked-out longwall area not filled with broken rocks, $l = 2.7$ m,

– size of worked-out area along underlying seams, $l_n = 100$ m,

– pillar width, $l_y = 60$ m,

– parting thickness, $h_2 = 29.4$ m.

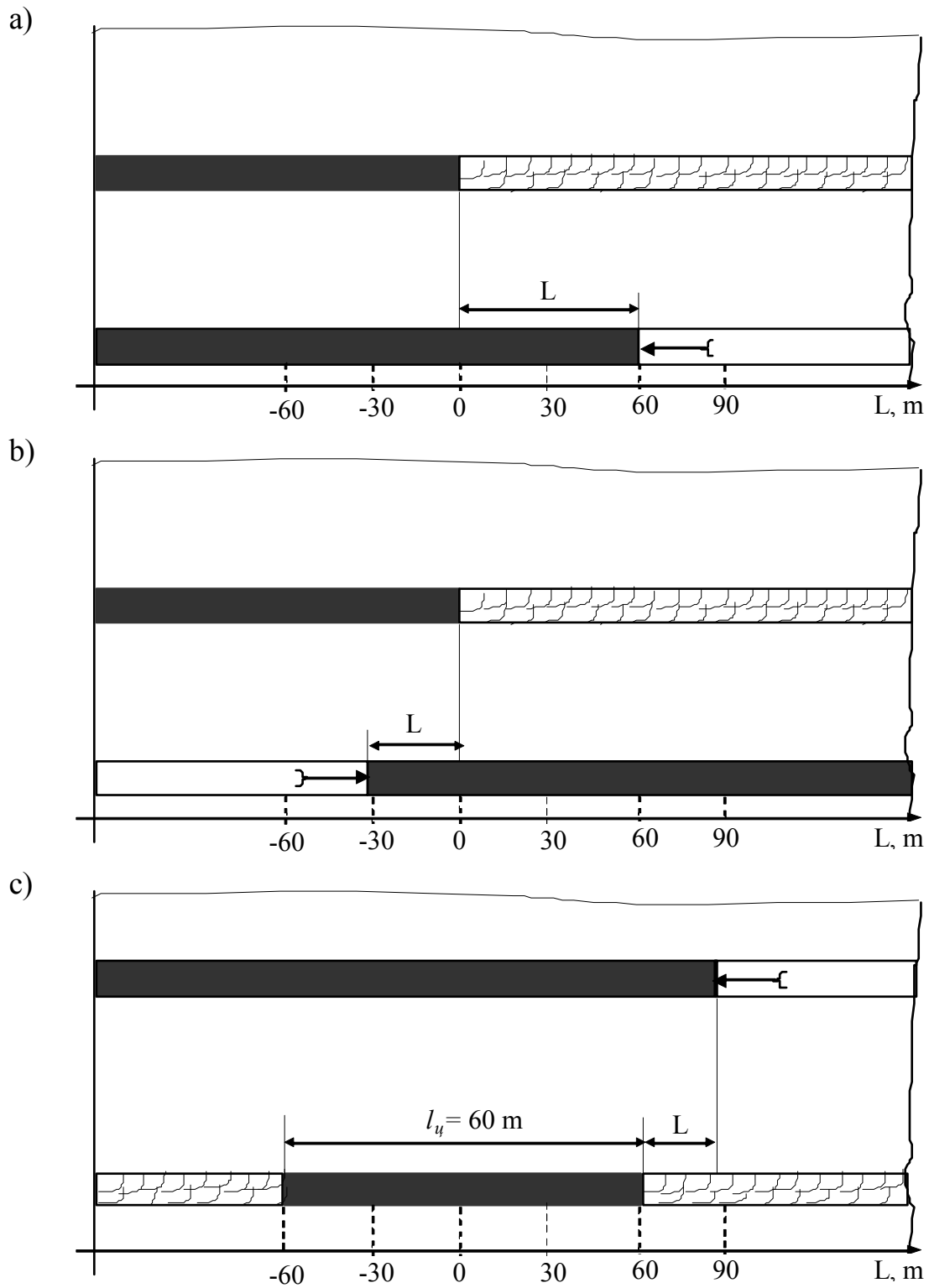


Fig. 4.17. – Calculation models:

- a) longwall when approaching contiguous seam selvedge;
- b) longwall when leaving from under contiguous seam selvedge;
- c) stope operation over the pillar.

Fig. 4.18 a, b, and c show calculation results of maximum roof and floor convergences in the longwall at various distances of the face from pillar boundaries and contiguous seam selvedges. To compare, points define the results of in-situ measurements. As it can be seen, difference is not more than 16 %.

Reactions in upgraded powered support hydraulic props within zones of high rock pressure due to the pillar (selvedge) were calculated according to the models represented in Fig. 4.19.

Model a) in Fig. 4.19 belongs to the case when support operates within the zone of high rock pressure due to the pillar (selvedge) of the contiguous seam. According to the in-situ observations (Fig. 4.12) in this zones contact of rock layer and canopy occurs at the distance of 1.1 m from the front prop; only 2.7 m part of the canopy is loaded with the broken rocks.

Support operation outside the zone of pillar (selvedge) effect corresponds to calculation model b), Fig. 4.19. It represents support loading being observed under field conditions and shown in Fig. 4.11. In this case, load is imparted to the canopy through undisturbed roof layer forming overhang of 1.5 m long from the side of the worked-out space; the canopy contacts with the roof over the whole area.

According to the models, support is meant for both P_0 main and $q(V)$ additional load effect. $q(V)$ load is determined according to the developed technique taking into account the fact that a spall zone covers 1 m and it takes 16 hours to develop rock pressure upon support canopy. Table 4.3 contains the results of calculation.

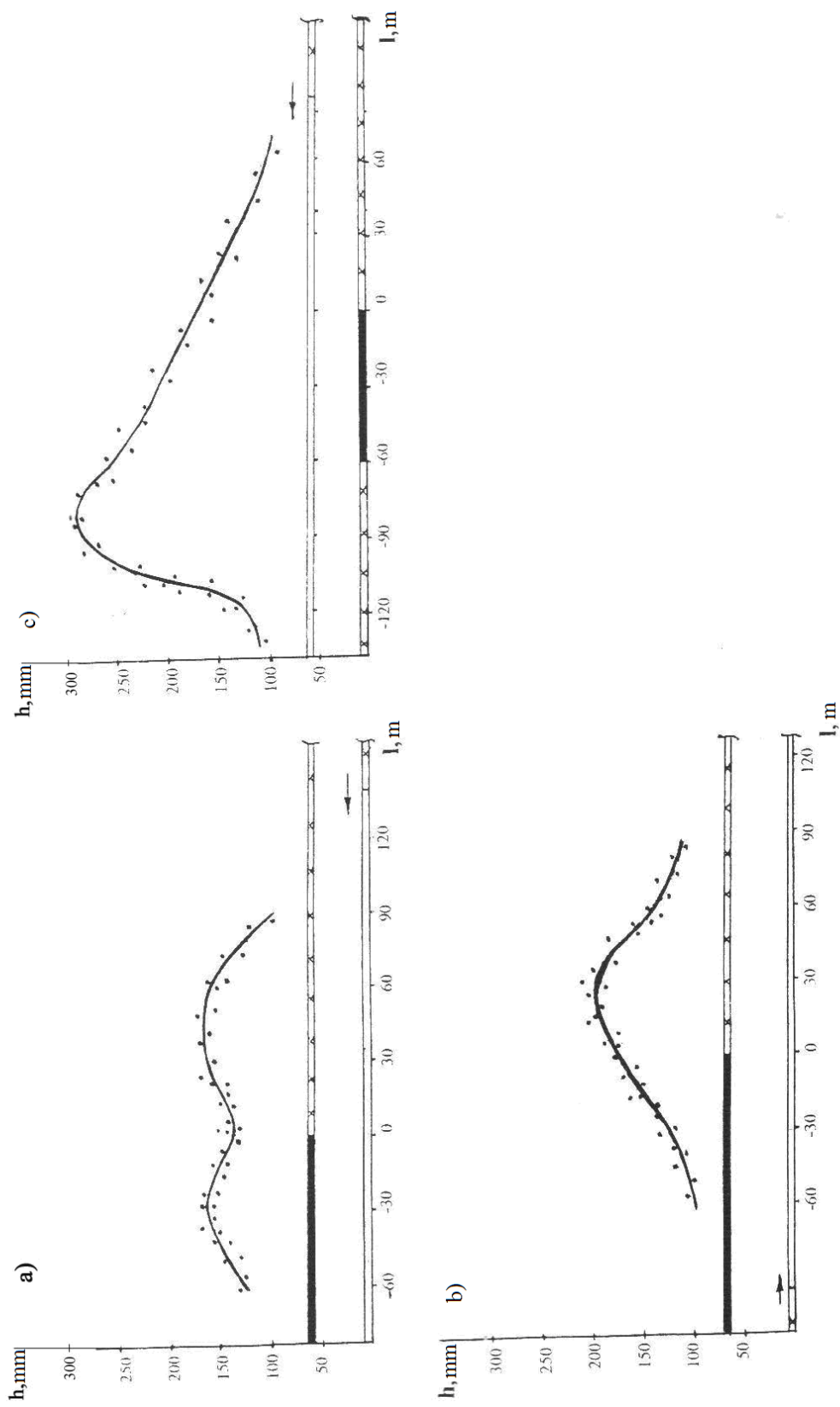


Fig. 4.18. – Values of maximum wall rock convergences at different longwall distances from pillar boundaries and selvedge of the contiguous seams: a) when getting under the selvedge; b) while operating under the pillar; c) while leaving from under the selvedge.

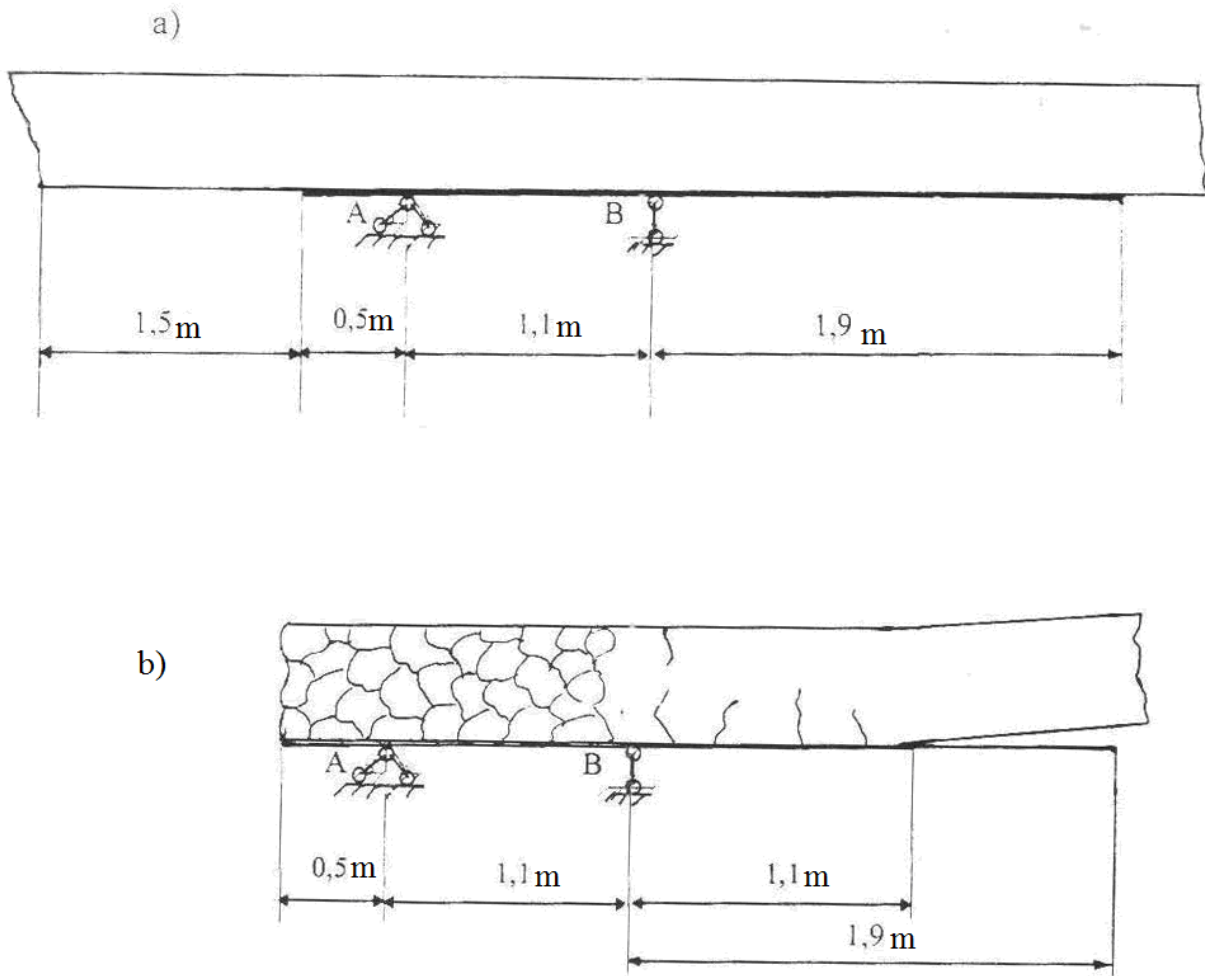


Fig. 4.19. – KM87 calculation models in terms of various loading condition:

- a) outside pillars (selvedges) effect zones
- b) within pillars (selvedges) effect zone

Table 4.3

Resistance of hydraulic props of KM87 upgraded powered support in various HRP zones

Parameters	Direction of longwall displacement	Mining condition	Face position	Longwall advance rate, m/day	Resistance of hydraulic props, kN							
					front		Convergence, %	back		Convergence, %		
					calculated	experiment		calculated	experiment			
Mine, Longwall												
“Lisova” mine, Longwall No.160	Under the selvedge	over-working	L = -30 m	1.2	746	600	19.6	213	250	17.4		
				4	734	600	18.2	277	240	13.3		
“Bendiuzka” mine, Longwall No.440	From under the selvedge	over-working	L = -30 m	1.2	700	600	14.3	280	330	18.0		
				4	607	-	-	174	-	-		
“Vidrodzhe nnya” mine, Longwall No.421	Over the pillar	Under working	L = -90 m	1.2	770	600	24.3	300	260	13.3		
				4	640	580	9.4	250	235	6.0		

The table data show that under considered conditions the support operates normally within the effect zones of pillars and selvages of contiguous seams if stope advance rate is not less than 4 m/day.

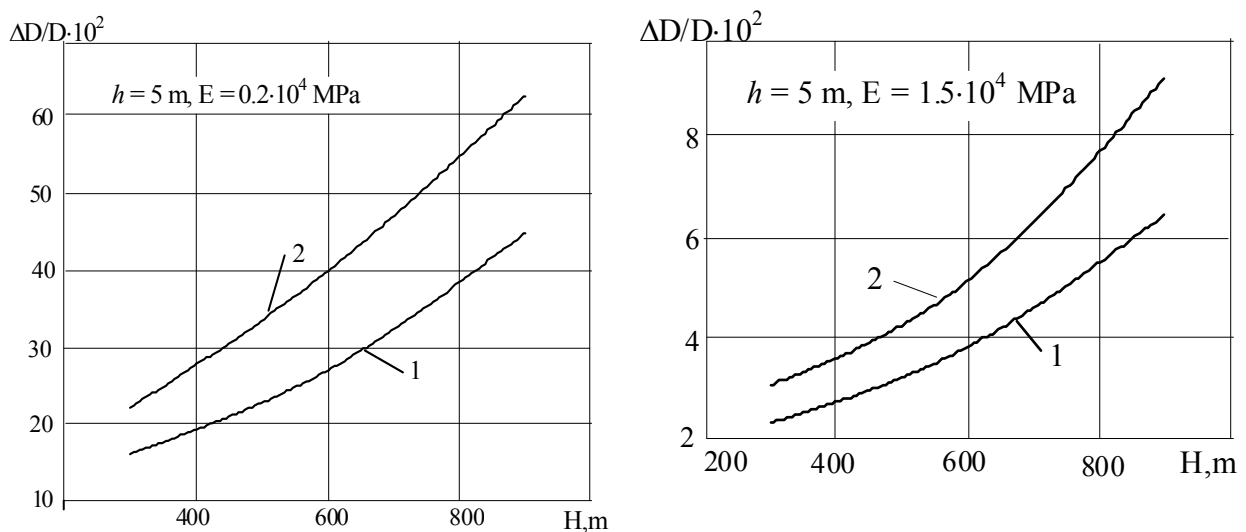
As for the resistance of hydraulic props, the table shows that theoretical values differ from the measured ones by not more than 18%.

4.3.2. Comparison of calculation results with the data of simulation performed using equivalent materials.

Apart from the comparative analysis described above, the developed algorithm has been used to estimate overworking and underworking effect upon the deformation of in-seam working in layered mass in terms of the conditions being simulated with the help of equivalent materials in VNIMI [106]. The models were used to simulate suite of three flat-lying seams which actual mining height was equal to 3 m. Conditions were developed being equivalent to mining support with the help of arch support made of AKII-9 special-purpose sections of 3 m in diameter and 3.2 m in span like in “Lvovugol” SE mines. Mining depth in conversion to the real object varied from 300 to 1200 m ($H = 500 \dots 520$ m in Lvov-Volyn basin mines). Material strength of the models corresponded to the strength of the enclosing rocks being from 20 to 100 MPa, i.e. mining and geological conditions close to the ones of Lvov-Volyn basin were studied (Table 1.1).

Ratio of elasticity moduli of parting and coal in the calculations as well as in the models adopted E_n/E_y , values belonging to [0.6...4.3] interval.

The calculations resulted in diagrams of ΔD roof displacement while overworking and underworking of in-seam working in its shares from its D diameters depending on H mining depth, h parting thickness, and E_n/E_y parameter (Fig. 4.10).



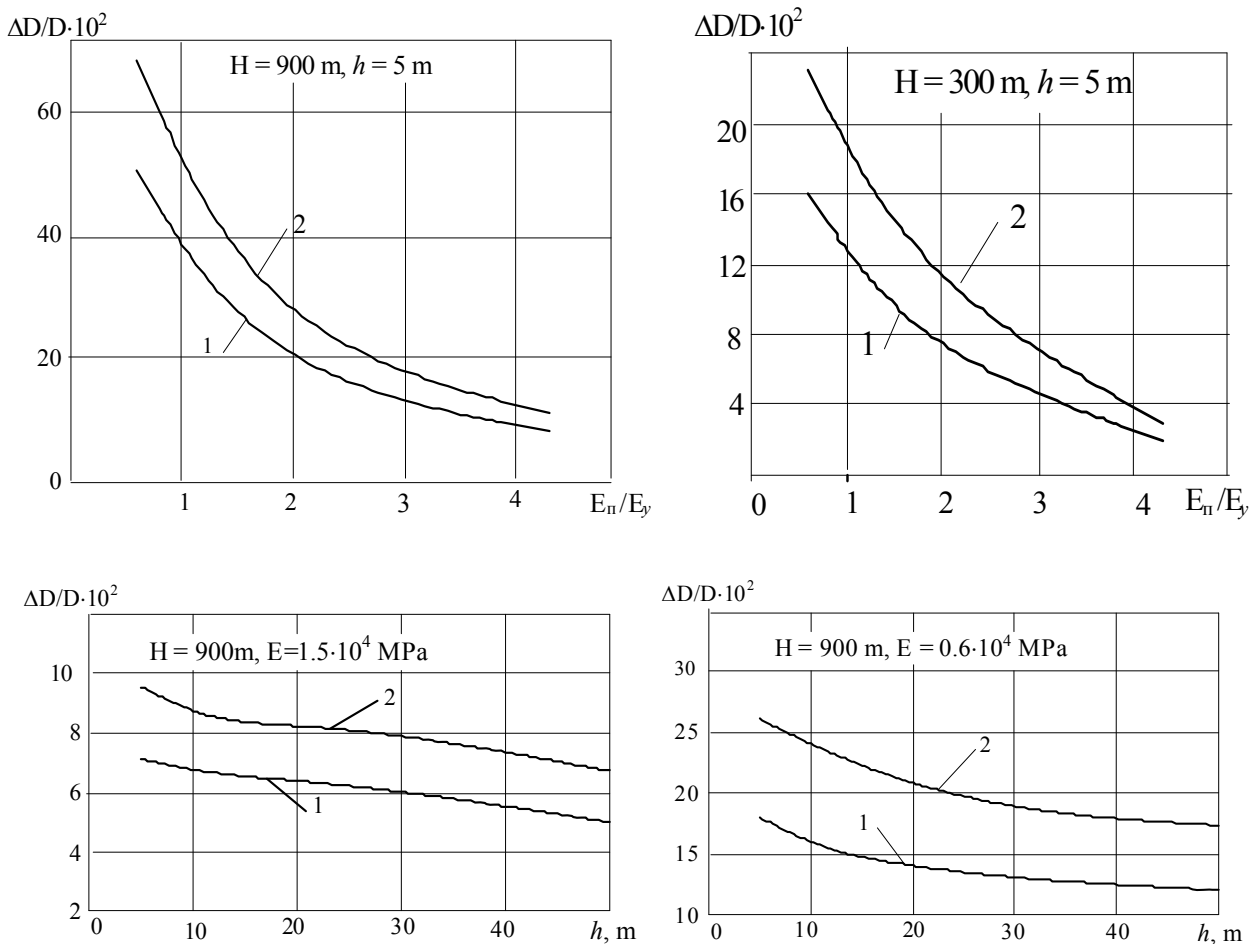


Fig. 4.20. – Relative displacements of in-seam mining roof depending on changes of basin mining and geological factors:
 1 – while overworking; 2 – while underworking.

The obtained curves coincide qualitatively with the experimental ones. They allow determining effect degree of one or another factor (other factors being equal) upon displacement value in the underworked and overworked mine working.

As an example, the work gives quantitative estimation of relative displacements of in-seam working roof depending on its H location depth for the following conditions: ultimate compression strength of roof rock is $\sigma_c = 40 \text{ MPa}$; ratio of ultimate compression strength of coal and roof rock is $\sigma_y/\sigma_{\pi} = 0.3$; parting thickness is $h_{\text{под}} = h_{\text{над}} = 20 \text{ m}$; E_{π}/E_{γ} ratio is 0.3, D working diameter is 3 m, thickness of coal seams is $m = 3 \text{ m}$.

As for overworked working being at the depth of $H = 300 \text{ m}$, experimental value of relative roof displacements is $0.04D$ while the displacements for the underworked working at the same depth is by 1.5 times more. Practically the same values are obtained by boundary element method (differences do not exceed 5 %).

It tells about sufficient flexibility and efficiency of the developed algorithm as well as about reliability of the results and conclusions obtained on its basis.

Conclusions

1. In terms of Lvov region conditions total rock divergences per stoping cycle in the zones of high rock pressure due to pillars and contiguous seam selvages are 180...300 mm. Maximum convergence is confined to the extraction process.

Outside the pillar (selvedge) effect zone total convergences per the same period are 100...115 mm; maximum convergence is observed in the process of support displacement.

2. As for effect zone due to pillar (selvedge) of the contiguous seams, here maximum convergence value near longwall face increases by 2-6 times while near worked-out area it increases by 1.5-2 times comparing to the operation outside this zone; moreover, length of the area with the increasing convergence is 1.6 across longwall depth.

3. Support hydraulic props operate in the modes of decreasing or alternating resistance within the zones of high rock pressure due to pillars and selvages of the contiguous seams; in addition practically all the load is taken up by the front prop. It shows the necessity to update the structure of KM87 support.

4. Most intense inrush formation is observed not above (under) the pillars and selvages of the contiguous seams but when approaching or leaving them. Length of inrush formation zone reaches up to 40...45 m; it depends on pillar thickness, its position relative to the developed longwall, and thickness and structure of partings.

5. Roof condition of coal seam within HRP zone depends considerably upon the feed rate of a mining machine as well as stope advance rate. Growth of feed rate of the mining machine results in the increase of spall zone in the seam selvedge deteriorating roof condition in the face area; on the contrary, increase of stope advance rate has positive effect as it decreases the volume of inrush formation. Feed rate of the mining machine being $V_k \in [3.0...3.2]$ m/min and stope advance rate not less than 4 m/day are considered to be rational ones for the studied mining and geological conditions.

6. Measured convergence values for longwall wall rocks differ from the calculated ones by not more than 16%; as for the stress in hydraulic props, the measured values differ from the calculated ones by not more than 18%.

7. Theoretical deformation values of in-seam working boundary in terms of underworking (overworking) coincide with the corresponding VNIMI data obtained with the help of models of the equivalent materials with the accuracy to 5%.

8. Following measured are recommended and implemented in "Lvovugol" SE mines to improve stoping efficiency on the basis of the performed mine studies:

- structure of KM87 support has been upgraded (hangover is shortened by 0.33 m);
- the holding company has organized five-shift operation in the mines;
- in 2002 the decision was taken to equip the stopes with the complexes of new МКД90 technological level.

OVERALL CONCLUSIONS

The monograph is the important scientific and research work in which topical scientific and technical problem concerning substantiation of technological parameters of the development of contiguous seams using powered complexes in terms of Lvov-Volyn deposit has been solved.

Main results, generalizing conclusions, and practical recommendations are as follows.

1. Calculation algorithm to determine and analyze stress and strain state of heterogeneous layered mass in terms of extraction of contiguous seams under conditions of overworking (underworking) has been developed on the basis of boundary element method and known analytic solution by V.G. Gmshinskiy using the criterion by P.P. Balandin.

Contrary to the known ones, the developed algorithm takes into account the stress concentration due to the effect of contiguous seam pillars (selvedges) as well as longwall advance rate.

2. Calculation results according to the developed algorithm are generalized and represented in the form of ratios characterizing regularities of rock pressure manifestation while extracting contiguous seams, that is:

– correlation ratios for maximum convergence in the longwall and maximum coefficients of σ_{yy} stress concentrations within the abutment zone in front of a stope have been obtained taking into account the direction of longwall displacement relative to contiguous seam selvedge and mining conditions (overworking or underworking);

– nonlinear correlation dependence of HRP zone size in front of a stope upon pillar width and parting thickness for overworking conditions have been obtained;

– nonlinear correlation dependence of σ_{yy} maximum stresses in the developed seam roof within the abutment rock pressure zone upon mining depth, physical and mechanical characteristics of coal and roof rocks, pillar width, and parting thickness are determined for overworking conditions for the cases when the face is under the pillar edge;

3. As a result of mine observations following specific rock pressure manifestations under the considered mining and geological conditions have been recorded:

– total rock convergences per Δu_y stoping cycle are 180...300 m within the zones of high rock pressure due to the effect of pillars and contiguous seam selvedges; maximum convergence is confined to the extraction process;

– outside HRP zones $\Delta u = 100...115$ mm; maximum convergence occurs during support relocation;

– when powered support operates outside the zones of high rock pressure due to contiguous seam pillars (selvedges), convergence rate near goaf longwall area is two times more than near the face; while operating in HRP zones it is four times more than near the goaf;

– most intense inrush formation is observed not above (under) the pillars and selvages of contiguous seams but when approaching and leaving them; inrush formation zone expands up to 40...45 m, it depends on pillar thickness, its position relative to the developed longwall, thickness and structure of the parting;

– increase of V_K cutter load feed rate from 3 m/min up to 4.5 m/min in terms of powered complexes operation within the zones of high rock pressure due to pillars and selvages of the contiguous seams results in the growth of specific inrush formation of roof rocks in face area from 10% up to 70%.

At the same time if V_{π} stope advance rate is increased from 1.2 m/day up to 4.2 m/day, inrush area decreases by 5...8 times and support load is reduced by 1.32 times as well.

4. “Technique and recommendations to determine rational parameters of powered support in terms of operation within the effect zones of pillars and selvages of the contiguous seams” has been developed on the basis of the obtained correlation ratios and mine observation results.

This technique is used to show that from the viewpoint of rock pressure manifestation, the period of longwall outburst from the effect zone of underlying seam selvedge is the most dangerous one; if longwall operates within pillar effect zone, underworking operation mode is the most dangerous one for the powered support when the face is at the distance of 20...25 m from the pillar edge (if pillar width is up to 50 m).

These statements are the determining ones while calculating strength parameters of the support under use.

5. Recommendations as for stoping parameters within high rock pressure zone due to the effect of pillars and selvages of the contiguous seams in terms of Lvov region mines have been developed:

– to increase the resistance of the used KM87 powered support near the face by means of shortening canopy overhang by 0.33 m;

– to organize five-shift longwall operation mode at shearer feed rate being equal to $V_K = 3.0...3.3$ m/min and to provide $V_{\pi} \geq 4$ m/day stope advance rate;

– to keep $L = 60$ m and $L = 65$ m distance between faces in terms of overworking and underworking respectively while longwall operation within contiguous seams.

Owing to the use of these recommendations in “Lvovugol” SE mines more than 5 mln tons of coal were mined within the period of 1997-2001 in terms of 1500 t/day output per face. Annual economic effect of the implementation of the recommended technology for contiguous seam mining using KM87 (KM88) powered complexes with new structural parameters in longwall 421 of “Vidrodzhennya” mine was 1,894,98 UAH.

6. Calculated results of reaction values in powered support hydraulic props differ from the corresponding measured ones in no more than 18%; the same of roof and floor convergence in longwalls is not more than 16%.

Theoretical boundary deformation values of in-seam working in terms of underworking (overworking) coincide with the corresponding data obtained by VNIMI with the help of the models of equivalent materials with the accuracy to 5%.

Multiple correlation ratios of the determined dependences have 0.90...0.98 values; values of Fisher's ratio test exceed considerably one-percent divergence limits. It indicates the available close correlation of the studied random values with the variable parameters as well as statistic importance of the determined relations.

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МЕХАНІЗОВАНИМИ КОМПЛЕКСАМИ В УМОВАХ ШАХТ
ЛЬВІВСЬКО-ВОЛИНСЬКОГО БАСЕЙНУ**

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