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BOUNDARY PARAMETERS OF MINING CONCENTRATION TO EXTRACT ADJACENT SEAMS IN THE WESTERN DONBASS

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Г 77 **Boundary** parameters of mining concentration to extract adjacent seams in the Western Donbass: Monograph / V.I. Buzilo, V.Ye. Vasyliev, O.O Yavorska, A.V. Yavorskyi, L.O. Tokar; Ministry of Education and Science of Ukraine; Dnipro university of technology. – Dnipro: НТУ «ДП», 2021. – 95 p.

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The monographs concerns problems substantiating parameters of concentration of extraction and development operations to mine two adjacent seams with the advanced overworking under the conditions of the Western Donbas mines.

Boundary-element method has been applied to evaluate effect of simultaneous extraction of two adjacent seams on the stress-strain state of rock mass in terms of the advanced overworking. Calculation results of the algorithm have been generalized and represented in the form of ratios characterizing regularities of rock pressure manifestation in the process of extraction of adjacent seams under the analyzed mining and geological conditions. The derived dependencies help identify safe distance between development mine workings and mining operations of the adjacent seams.

For students; engineers and technicians; staff members of higher educational institutions, research institutions, and developers of coal industry.

45 figures, 143 reference sources.

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INTRODUCTION

Bedding in Western Donbass is noted for the fact that predominantly suite being mined consists of three to six pairs of adjacent seams where interbed thickness is less than 12 m. Such seams contain almost 40% of balance reserves of mines. Maximum distances between adjacent seams are not more than 50m. Another feature of the region is the fact that almost the whole amount of the reserves is concentrated within thin and very thin coal seams.

Due to insignificant thickness of interburden layers, adjacent seams are mined in descending order. Until recently mining in terms of upper (overworked) seam was performed with advance of one or two extraction pillars at least. The abovementioned helps locate mine workings of a lower seam within the overworked area out of effect of upper layer mining operations.

According to stipulations of "Energy Strategy of Ukraine up to 2030", intensification of coal branch owing to increase in coal mining, decrease in its cost, and increase in profitability level of mines is one of the tendencies of economic and social development of Ukraine. In the context of Western Donbass it is impossible to solve the problems without intensification of seam mining. Thus a number of mines need maximum shortening of distances between secondary mining and first mining performed within adjacent seams; the case is in concentration of mining operations. In several, as for *Stepnaia* mine (*DTEK Pavlogradugol* PJSC) in the process of c_6 and $c_6^/$ adjacent seams stoping with 10 to 11 m parting thickness it has been decided to construct normally boundary entry of c_6 seam at the distance of 15m from $c_6^/$ seam selvedge. The decision was based on recommendations of current normative technique of UkrSRDI. As a result, the mine working turned out to be within affected area of $c_6^/$ seam overlying longwall. To clear it out floor rock lifting followed by lengthwise retimbering took place. In this context, actual cost to clear out of a running meter of mine working was UAH 1928.35 while expenses connected with a construction of a running meter of a new mine working were UAH 2039.43. It means that current normative techniques cannot provide required stability of mine workings and cannot be used to substantiate safe boundaries of mine operations concentration within adjacent seams in the context of Western Donbass mines.

Problems concerning substantiation of safe boundaries of mine operations concentration still remain in abeyance. For that matter, assessment of mine operation impact on stress-strain state of rock mass as well as determination of boundaries of safe and expedient location of stopes and development workings while mining within adjacent seams is actual scientific and practical task. The monograph deals with the problems.

Staff of the Department of Mining Engineering and Education of the Dnipro University of Technology and employees of Stepnaia mine (DTEK Pavlogradugol PJSC) took part in field studies. Computer procedures were developed in cooperation with L.V. Novikova, Professor of the Department of Higher Mathematics of the Dnipro University of Technology, and L.I. Zaslavskaia, Associate Professor of the Department of Higher Mathematics of the Dnipro University of Technology. We highly appreciate their contribution to the research.

CHAPTER 1. SPECIFIC CHARACTER OF ADJACENT SEAMS MINING IN THE CONTEXT OF WESTERN DONBASS MINES

1.1 Mining-and-geological characteristic of adjacent seams mining in Western Donbass

Productive series of Western Donbass consists of Paleozoic, Mesozoic, and Cainozoic sedimentary formations being deposited on rocks of Pre-Cambrian crystalline basement (Ukrainian crystalline shield).

Coal formation is represented by three carbon portions – lower, middle, and upper one. Total carbonic thickness varies from 3000-3500 m in the east to 150 m in the west. Commercial coal content within the area under development is associated with coal formation of Samara series C_1^3 of lower carbonic portion [1 - 4].

6 to 24 seams are commercial ones. $c_{10}^{в}$, c_9, $c_8^{в}$, $c_8^{н}$, $c_7^{н}$, c_6^1, c_6, c_5, c_4^1, $c_4^{в}$, c_2 and c_1 seams are among the most continuous formations; their dominant thickness is 0.7 to 1.2 m. Now they are mined by ten mines of SHC *Pavlogradugol* Ltd [5, 6, and 7]. Dip angles of coal seams are 0…6^0; stratification depth is 60 to 900 m.

As a rule, three to six pairs of adjacent seams are deposited in a series; in this context minimum parting thickness is 10 to 12 m and maximum one is 50 to 85 m.

Report [1] includes comprehensive geological and commercial outline of Western Donbass. Papers concerning substantiation of techniques of adjacent seams development and extraction also explain geological conditions of the region [8, 9, 10]. Lithologically coal-bearing mass consists of argillites, aleurites, and arenites which content is 32.5%; 41.5%, and 18.0% respectively. Coal content of the mass varies within 2.2…7.4% [2].

Strength properties of enclosing rocks greatly depend on their openness, cleavage, and moisture; moreover, they vary in a broad range. For example, strength for uniaxial compression of argillites, aleurites, and arenites is 12 to 33 MPa, 8 to 28 MPa, and 10 to 44 MPa respectively. Natural moisture of rocks is 2 to 10%; it declines with the depth. Table 1.1 shows averaged physical and mechanical characteristics of coal, and rocks of roof and floor for operative mines in Western Donbass according to data listed in papers [7, 11, and 12].

Data from Table 1.1 argue that in the context of mining and geological conditions being considered, enclosing rocks are of less strength to compare with coal. That is why seam mining is followed by heaving, and rock outsqueezing from walls and roofs into mine workings. Field studies have shown that thickness of seams involved in heaving may reach 6 to 7m; moreover, heaven of floor in development workings within zones of mining impact is 60 to 70% of convergence value [10, 13, and 14].

Table 1.1

Characteristic of mode of thin seams occurrence in mines of Western Donbass

Mine	Seam	Mining depth, m	Thickness, m	Physical and mechanical properties													
				Coal				Roof					Floor				
				$\sigma_{сж.}$, MPa	γ, t/m^3	E*10^{-3} MPa	ν	rock	$\sigma_{сж}$, MPa	γ, t/m^3	E*10^{-3} MPa	ν	rock	$\sigma_{сж.}$, MPa	γ, t/m^3	E*10^{-3} MPa	ν
1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Yubileinaia	C_6	130... 370	0.91	30	1.38	3.3	0.35	aleurites of mean stability	35	2.8	2.9	0.3	aleurites of mean stability	35	2.9	2.9	0.3
	C_6^1	120... 360	0.71… 0.73	30	1.44	3.1	0.33	argillites and aleurites of mean	30	2.9	2.9	0.3	aleurites of mean stability	30	2.8	2.9	0.3
Stepnaia	C_6	140... 200	0.9	30	1.40	3.5	0.36	arenites	40	3,1	2.8	0.3	aleurites	40	3.1	2.8	0.3
	C_6^1	130... 190	0.67	30	1.44	3.6	0.38	argillites of mean stability	30	2,8	2.9	0.3	aleurites, argillites	30	2.8	2.9	0.3
Pavlogradskaia	C_5	140	1.00… 1.55	25... 30	1.4… 1.5	3.4	0.39	unstable argillites	10	2,6	2.8	0.3	unstable argillites	10	2.6	2.8	0.3
	$C_2^в$	160	0.78	30	1.37	3.3	0.4	unstable argillites	10...15	2,6	2.7	0.3	argillites and aleurites of mean	10...15	2.6	2.7	0.3
Geroev Kosmosa	$C_{10}^в$	350	1.03	30	1.48	3.5	0.4	unstable argillites	8…12	2.9	3.0	0.3	unstable argillites	8…12	2.9	3.0	0.3
	C_{11}	370	1.03	30	1.5	3.6	0.4	unstable arenites	15... 20	3.0	3.0	0.3	unstable argillites	15... 20	3.0	3.0	0.3

1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Samarskaia	C_1	160... 185	0.89.. .1.2	40	1.32...1. 6	3.7	0.4	unstable argillites	10...15	2.8	2.9	0.35	unstable argillites	10...15	2.8	2.9	0.35
	C_4	100... 140	0.8... 0.83	40	1.39	3.5	0.4	unstable argillites	15...20	2.65	2.6	0.3	unstable argillites	15...20	2.6	2.	0.3
Blagodatnaia	$C_7^{\text{н}}$	90... 180	0.8... 1.3	35	1.48	3.5	0.4	unstable argillites	10	2.65	2.8	0.3	unstable argillites	10	2.65	2.8	0.3
	C_8	90... 140	1.08	40	1.48	3.3	0.38	unstable argillites	15	2.62	2.7	0.3	unstable argillites	15	2.62	2.7	0.3
	C_9	100... 130	1.10… 1.12	30... 40	1.44	3.6	0.36	unstable argillites	8...10	2.64	2.6	0.3	unstable argillites	8...1 0	2.64	2.6	0.3
Stashkov	$C_1^{\text{в}}$	225	1.05… 1.22	40	1.42… 1.6	3.3	0.35	unstable argillites	20	2.64	2.8	0.3	unstable argillites	20	2.64	2.8	0.3
Ternovskaia	C_5	175... 220	0.7... 0.85	30...40	1.37...1. 4	3.2	0.33	unstable argillites	20	2.65	2.8	0.3	unstable argillites	20	2.65	2.8	0.3
	$C_4^{\text{в}}$	240... 270	0.94... 0.95	30	1.39	3Д	0.33	unstable argillites	20	2.65	2.8	0.3	aleurites of mean stability	20	2.65	2.8	0.3
	$C_6^{\text{в}}$	170... 200	0.9...0.9 5	30...40	1.46	3.2	0.32	argillites of mean stability	20...25	2.65	2.7	0.3	unstable aleurites	20...25	2.65	2.7	0.3
Dneprovskaia	$C_8^{\text{в}}$	190... 280	1.12… 1.15	30	1.68… 1.9	3.5	0.4	unstable argillites	15... 20	2.8	2.5	0.3	unstable argillites	15... 20	2.8	2.5	0.3
	$C_{10}^{\text{в}}$	135... 235	1.10… 1.16	30	1.44	3.5	0.4	unstable aleurites	15...25	2.65	2.8	0.3	aleurites of mean	15... 25	2.65	2.8	0.3
Zapadno-Donbasskaia	$C_8^{\text{в}}$	480	0.91	30	1.25	3.6	0.4	unstable argillites	8…10	2.65	2.8	0.3	unstable argillites	8… 10	2.65	2.8	0.3
	$C_8^{\text{н}}$	480	1.28	30	1.42	3.6	0.38	unstable aleurites	15... 20	2.9	2.9	0.3	unstable argillites	15... 20	2.9	2.9	0.3

Rheological properties of enclosing rocks (especially argillites) also factor into a process of rock heaving and outsqueezing. According to data of indoor tests of samples, the properties become apparent even when load is only 20% of crashing one. According to data by VNIMI [12] creeping parameters δ, $c^{0.3}$ of arenite, aleurite, argillite, and coal are $3.28 \cdot 10^{-3}$, $5.54 \cdot 10^{-3}$, $1.17 \cdot 10^{-2}$, and $2.32 \cdot 10^{-3}$ respectively; nondimensional parameter α of all the rocks and coal is near 0.7. Values of creeping deformations are 50 to 100% of conditionally instantaneous, and nature of deformation is close to linear one [15, 16, 6, 17]. In this case, temporally rock deformations are relevantly explained with the help of a model of linear congenital creeping with Abel type kernel [18].

There are many examples of the model use in rock mechanics. Thus, papers [19, and 20] apply it to analyze durability of carnotite and potash pillars.

Papers [21, and 22] use it while substantiating operational parameters to mine sawn limestone; papers [23, and 24] use it to analyze thin manganese-ore seam augering; using it papers [25, and 26] determine rational parameters to mine pillars in Western Donbass mines. In each case the approach proves to be rather efficient and successful. For this reason the monograph imitates rock mass is simulated by means of linear and congenital medium with Abel creep kernel if adjacent seams are being mined.

1.2 Techniques to extract adjacent seams in mines of DTEK Pavlogradugol PJSC

Today reserves of "DTEK Pavlogradugol" PJSC mines are thin and very thin flat coal seams with 0.6 to 1.2m thickness. Almost 40% of balance reserves are concentrated in adjacent seams where parting is 7 to 14 m (*Stepnaia*, *Yubileinaia*, and *Zapadno-Donbasskaia* mines). In *Ternovskaia* mine, distance between seams being mined is 30 to 60 m; in *Pavlogradskaia* and *Samarskaia* mines as well as in Stashkov mine it is 37 to 40 m, and in *Blagodatnaia* mine parting thickness varies from 25 to 85 m. Mining depth is H = 160-400 m. Table 1.1 demonstrates specific values according to the region mines. Unstable argillites, aleurites, and free-caving arenites occur in floor and in roof (Table 1.1).

Stratigraphic column in Fig. 1.1 explains mode of adjacent seams occurence in Western Donbass mines.

Every mine applies horizon-oriented mine development with longwall seam mining.

Most of all, columns are mined to the rise with the help of double longwalls or single ones. Length of the columns is 900 to 2500 m. Longwall length varies within 160-200 m. Roof is controlled by means of complete caving.

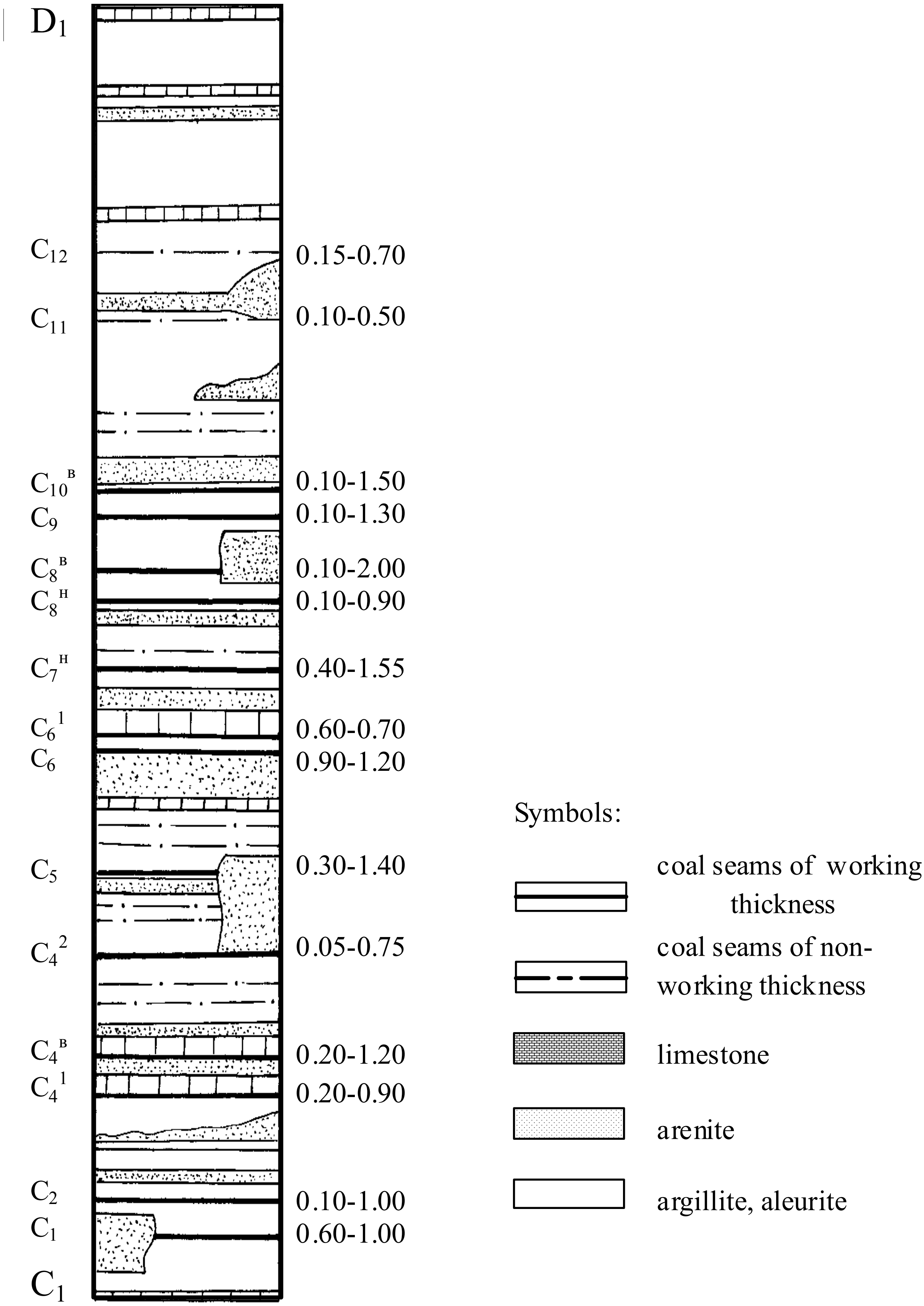


Fig. 1.1 Typical stratigraphic section of coal-bearing series in Western Donbass

Longwalls are equipped with powered systems КМК-97М with МК-67coal shearer, КД-80 with КА-80 or 1К101У coal shearer, КД-90 with КА-200 coal shearer, КД-99 with КА-200 coal shearer, and ДМ with 1ГШ-200 or 1 К101У coal shearer.

Several seams in series are mined simultaneously; as a rule it takes place in descending order. Alongside with that, norms specifying adjacent seams mining equally provide ascending order as well [27]. In this context, parting thickness is the restriction. Mining with complete roof rock caving needs parting thickness to be nor less than 12 m (m is seam thickness) in terms of ascending order, and nor less than 6m in terms of descending one.

However, the norms don't mention physical and mechanical properties of coal and enclosing rocks stipulating stress-strain state of roof and floor of adjacent seams being mined.

Papers [28, 29, and 30] involve ratios of bearable parting thickness and geometry of a seam being mined; however, they are a result of experimental observations and their application area is limited.

It is clear that in each specific case bearable thickness of parting should be determined using analysis of stress-strain state of coal and enclosing rocks in the neighbourhood of stopes within adjacent seams involving relevant mining and geological conditions. Moreover, the example is also effected by distance between stopes within adjacent seams. In this context, dimensions of zones of broken rocks and dimensions of zones of bearing pressure within analyzed area of rock mass are determinant.

The monograph considers each of the aspects while substantiating the parameters of operation schedules to mine adjacent seams in the context of mining and geological conditions being analyzed.

1.3 Conditions to construct and maintenance development workings while mining adjacent seams in the context of Western Donbass

While mining adjacent seams, elimination of mutual effect of second working and first working on mine working of adjacent seams to provide their operation stability is engineering problem of top priority. Process-oriented detalization of mining alternatives mainly depends on mine working siting [27, 31, and 32]. Existing practice and theoretical research by RIRMMS, NMU, DSRCI, and DPI go to show that to make section development workings stable they should be placed within de-stressed zones [29, 33, 34, 35, 36, and 37]. In this context, operation stability of workings is achieved if only relevant de-stressed zones are available. That can be done with the help of ribbings along a seam under mining as well as protective pillars along adjacent ones [37, 38]. A technique of repeated de-stress with the use of stowing masses is rather reliable [39].

As applied to situation in Western Donbass the techniques of development working protection can be better referred to main haulage roads and others located out of area of mining effect. However, when extraction workings are located within de-stress zones formed at the boundary of second workings under goaf of overlying seam they will subject to sharply increasing rock pressure with all ensuing consequences after its selvedge has been mined with the help of next longwall.

Currently locations of mine workings, selection of techniques and support parameters as well as protection of workings in mines are ruled by branch documents based upon RIRMMS instructions [6, 31, 40, 41]. Structure of rock mass as well as changes in stress state of enclosing rocks resulting from mining is taken into consideration with the help of empirically determined coefficients which are not sufficient for total reflection of definite mining and geological conditions and specific mining situations. That is why engineering decision being made often turn out to be ineffective. In Western Donbass mines extraction drifts are fastened either with the help of metal arch yielding support КМП-3А or КШПУ made of special СВП-17 – 27 shapes. Protection of reusable mine workings is performed with the help of: coal pillars; wood chocks; cutting-off supports (*Stepnaia*, *Yubileinaia*, and *Geroev Kosmosa* mines); packs (*Blagodatnaia* mine) and cast strips (*Samarskaia* mine). However, floor rock outsqueezing into roadway entry, support forcing into floor, and roof rock lamination take place within zones of increased rock pressure.

To bring into balance a technique and support parameters, and mine working protection and specific mining and geological conditions it is required to analyze stressed behaviour of rock mass area being studied and identify boundaries of de-stressed zone and zone of increased rock pressure.

For Western Donbass mines, paper [42] determines de-stress zones from filled development working by an experimental approach; they are used for recommendations concerning location of a development working within adjacent seam. However, the author has considered specific situation without any generalizing ratios.

Paper [43] involves more explicit evaluation of stoping effect on development workings of adjacent seams being mined using mathematic simulation but it concerns only Lviv-Volyn coal field.

The paper applies similar approach to demonstrate factors exercising significant effect on stress-strain state of roof and floor rocks to determine rational location for workings in Western-Donbass mines.

1.4 Analysis of research concerning problems of adjacent seams mining

Two main tendencies concerning analysis of stress-strain state (SSS) of rock mass in the context of strata series mining are available.

Tendency one is a picture of rock mass as a continuum being elastic, tight, and having rheological properties. While developing theoretical approaches, the results of field tests and laboratory tests including those which involve models of equivalent materials are used [44-53]. Generally, theoretical algorithms of the tendency cover numerical procedures of rock mechanics. The majority of the solutions concern specific mining and geological conditions rarely incorporate integrating data; that is why they cannot be applied for other conditions.

Tendency two involves research applying schemes of independently block systems and loose media, and basing upon various hypotheses such as hypothesis of virtual displacements or benched rock fault etc. [54-59].

While analyzing stress-strain state of rock mass with stopes, zones of load, zones of irregular caving, zones of boundary state of stress, and zones of bearing pressure are identified [58-65]. Determination of the zones boundaries is required to substantiate both technological and geometrical parameters of mining systems and to identify rational parameters of support and protection of development workings within area subjected to mining effect.

Configuration of the zones and stress values within them primarily depend on physical and mechanical properties of floor and roof rocks as well as on rock mass structure. Mined seam thickness, mining depth, and location of mine working are important as well.

Early papers concerning calculations of bearing pressure were based either on data of field studies [66, 67] or on a model of elastic half-plane with horizontal rectangular slot at arbitrary depth [68, 69] and dealt with homogeneous medium. Later papers (both theoretical and experimental ones) analyzed stratified masses and stress-strain state in terms of overworking and underworking [70-74]. Definition of boundary conditions involved measurement results obtained in mines of Russia. That is why the results cannot be used while considering conditions of mines in Western Donbass.

Schwartz algorithm where stope is approximated with the help of a slot opens wider possibilities. Papers [75-78] apply the technique for the development of calculated algorithm to forecast stress-strain state of rock mass and formulate multicriteria approach for increased rock pressure zones and de-stressed zones while strata series mining. Author used superposition principle and considered several analytical models with all mutual-action stopes to understand redistribution of stresses in the process of sequential mining of each seam of the series. However, it is not quite relevant to use superposition principle at the stage of rock fragmentation. Moreover, the analytical models did not involve development workings.

With the help of finite-element method, papers [79-84] have analyzed effect of such factors as mining depth, seam thickness and longwall length on a value and stress distribution nature in the neighbourhood of powered stope. Different mining and geological conditions have been considered but in the context of single seam mining. However, in the context of Western Donbass mines and *DTEK Pavlogradugol* PJSC forecast of stress-strain state in terms of strata series mining is important.

Basing upon tests of three-dimensional models made of equivalent materials paper [85] developed locational pattern of increased and decreased stresses in the neighbourhood of overworking and underworking entries. However, it is known that physical simulation of geomechanical processes helps obtain only relative rather than absolute values of stresses; hence, it is referred only to qualitative situation with stress-strain state.

Available data of field tests carried out in the context of mines in Kuzbass, Central Donbass and in mines of *Shakhtiorskugol* while mining adjacent flat and sloping seams at various depths testify that in the process of adjacent seams mining increase in mining depth resulted in increase of bearing pressure level. Structural changes (i.e. loosening and solidification) of both mined seam and adjacent one took place, hollows in stopes and development workings were observed; moreover, the latter were involved even when parting was comparatively thick [86-88]. It means that each specific case should take into account unique mining and geological conditions.

Recommendations [40, 89] are applicable for mines in Western Donbass as they concern order of rather adjacent seams mining and location of development workings. Moreover, the recommendations were formulated on the basis of data of mine observations.

When authors of papers [90, 91] studied the problems as applied to overworking conditions they relied upon mathematical experiment on the Latin square method. At the same time of all physical and mechanical properties only elastic moduli of type one and two were in vision of authors; strength characteristics were not involved. Parameters of exploited seam were not varied either.

Monographs [92-97] explain science of the method and its numerous applications in the context of linear and non-linear solutions in rock mechanics. Except that need for discretization of the whole rock mass being analyzed factors into systems of higher-order linear algebraic equations. Thus, in increasing frequency researchers refer to boundary-element method (BEM). According to the method, finite elements approximate only boundaries of areas under analysis. Naturally it takes less time to prepare input information and considerably decreases an order of equation system being formed; in addition its solution lightens.

Mathematical background of BEM, its mechanical essence as well as different applications are explained in papers [98-100]; papers [101-106] by foreign authors are also devoted to the issue.

Papers [97, 99-101,107,108] concerning coal series mining discover multifold possibilities of BEM implementation in terms of the most diverse and complex geomechanical problems. Thus, efficient BEM-based calculation algorithm has been used to substantiate parameters of adjacent seams mining with the help of powered longwalls (papers [43, 109-111]). The writings involve generalizing correlation dependences to calculate stresses and displacements within interburden layers obtained for Lvov-Volyn coal field.

It is obvious that comparable dependences are required to substantiate parameters of operation schedules to mine coal series in the context of mines in Western Donbass. In this regard mathematical simulation of mining situations using boundary-element method in combination with correlation analysis of results obtained is the most rational approach to solve geomechanical problems.

Analysis of stress-strain state of the mass areas under study covers determination of boundaries of increased rock pressure zones and de-stressed zones. Their dimensions and location depend on rock strength criterion being used.

Within the area of rock mass compression, rock strength is frequently estimated on the hypothesis of maximum normal stress; within the area where tension stresses are available they are set as those equal to zero (the majority of rocks weakly withstand tension), that is rock mass is considered as “tensionless medium” [57, 94, 112, 113]. However, if it is required to take into account a degree of rock resistance to tensile, either hypothesis of the greatest shear stresses (Mohr criterion) [114, 115] or strength theory by P.P. Balandin [19, 110] is applied.

Taking into account the fact that theory by P.P. Balandin not only reflects rock property to resist differently stretch and tensile but also concerns each component of stress tensor, the paper will use this very theory.

Eventually, we come to following conclusions:

1) while substantiating parameters of operation schedules for adjacent seams mining being adequate to mining and geological conditions under consideration, rock mass structure, physical and mechanical properties of coal and enclosing rocks as well as specific mining conditions (overworking) are determining factors;

2) according to data of laboratory tests, values of creeping deformations of roof and floor rocks (argillite, aleurite, and arenite) within development workings and in terms of mining and geological conditions being analyzed are 50 to 100% of conditionally instantaneous and deformation nature is close to linear one, their stress-strain state can be adequately explained with the help of a model of linear congenital creeping with Abel type kernel;

3) analysis of available approaches to solve geomechanical problems means that boundary-element method is the most effective one to determine stress-strain state of structurally non-homogenous rock mass; and

4) theory by P.P. Balandin involving both all components of stress tensor and typical property of rocks for different resistance to strength and tensile is the most adequate criterion for enclosing rocks breaking in the context of Western Donbass mines.

1.5 The research tasks

Analysis of the current state of formulated technological task has determined starting position for the research.

Substantiation of boundary parameters for relationship of stopes and development workings in the context of mining operations concentration within adjacent seams of Western Donbass involves solution of following partial problems:

1. Estimation of synchronous mining of two adjacent coal seams effect on stress-strain state of rock mass in the context of advance overworking as the key factor being determinative for concentration of mining.

2. Use the analysis of determinative mining and geological factors to identify rational boundaries for concentration of mining operations supporting safe and economically feasible second workings and first workings within adjacent seams.

3. Implementation of mine investigations concerning rock pressure manifestations in stopes and development workings while adjacent seams mining to confirm results of theoretical investigations.

CHAPTER 2. SUBSTANTIATION OF BOUNDARY PARAMETERS FOR CONCENTRATION OF MINE OPERATIONS IN THE CONTEXT OF ADJACENT SEAMS MINING

2.1 General information

Applying analysis of strain-stress state of covered area of layer nonhomogeneous rock mass in the neighbourhood of stopes with the use of obtained correlation ratios, a distance between stopes in terms of which longwalls within adjacent seams can work safely, effectively, and reliably under the conditions of Western Donbass mines is determined.

According to regulatory guidelines [31] safe distance between mining operations within adjacent seams mainly depends on interbed thickness and shift angles of rocks. However, in the context of its determination, dimensions of zones of bearing pressure in front of longwalls and values of maximum stresses within the zones; dimensions of rock failure zones behind longwalls within roof and floor of each of adjacent seams; and maximum floor and roof approximation within stopes are determinant ones [127,128].

Minimum safe distances between longwalls have been substantiated using boundary-element method; to do that algorithm to estimate stress-strain state of rocks while mining two adjacent seams with advance overworking has been developed. Approved mathematical tool and developed calculation algorithm make it possible to identify both redistribution of stresses within rock mass area under consideration and rock shift within stopes and development workings in the context of their different location relative to each other.

2.2 Explanation of analytical model

Values and nature of distribution of stresses within rock strata in the context of coal series mining mainly depend on rock structure, physical and mechanical properties of coal and rocks, thickness of seams and interburden layer, mining depth and characteristics of mining and technological situations.

If possible, the factors should be in the picture of algorithm of geomechanical problems being solved and load distributions being used in the context.

Analysis of specifity of adjacent seams mining in Western Donbass mines confirms that consideration of stress-strain state of rock mass as part of analytical model shown in Fig. 2.1 is relevant for mining and geological conditions under study.

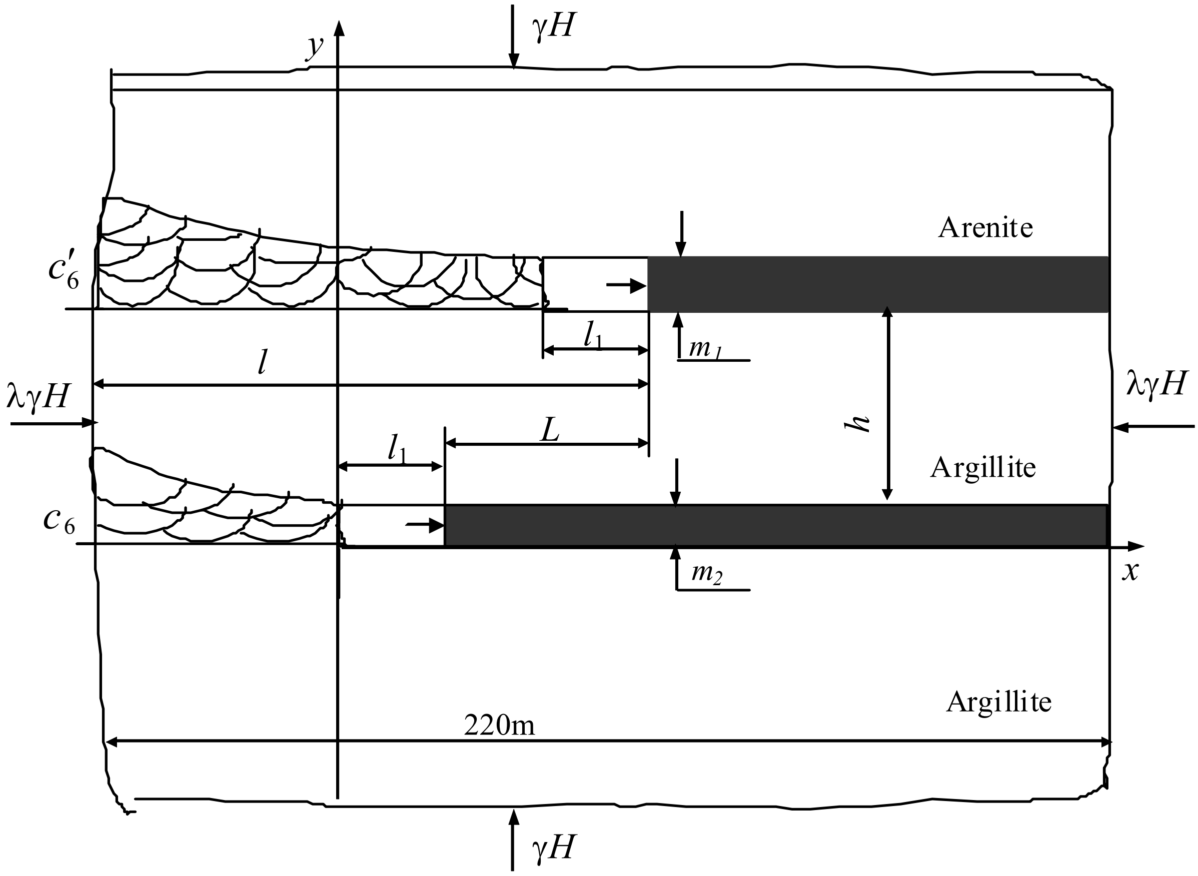


Fig. 2.1. As for determination of safe distance L between stopes when interbed thickness is h

Among other things it can help determine rules of redistribution of shifts in roof, floor, and partings of adjacent seams mined with advance overworking. Values of roof and floor convergence for each of adjacent seams will help determine adequate rock pressure manifestations under specific mining and geological conditions, acceptable values of interbed thickness as well as distances between stopes in the context of adjacent seams and within one seam area.

Overall dimensions of area under investigation are 220 × 30 m^2. Space portion of goaf ($l - l_1$) in Fig. 2.1 is packed with broken rocks. Parameter l_1 is caving step of immediate roof; according to data of mine investigations it is 4m in the context of conditions under consideration 4 m.

Depth mining *H*, distance between stopes *L* and interbed thickness *h* are running geometric parameters. Elastic moduli of rock and coal $E_п$ and E_y are running parameters as well. Real load is rock weight: it is γH in vertical direction at infinite, and $\lambda\gamma H$ in horizontal direction.

2.3 Formulation of the problem in terms of boundary-element method

Problems connected with stress-strain state determination of rock mass area under analysis according to above models are solved with the help of boundary-element method in terms of that its modification called pseudo-load method [108].

Analytical solution of known Kelvin problem on the action of normal forces and tangential forces uniformly distributed within a section, randomly oriented within infinite elastic medium, is basic one.

Following calculation stages are principal:

- Approximation of area under analysis with the help of finite elements and introduction of pseudo-normal P_n forces and tangential P_s forces into each of them.
- Establishment of boundary conditions in components of free contours and conditions of deformations in components of "roof-coal" and "coal-floor" adjacent contours.
- Development of analytical ratios for stresses and shifts in each of boundary (including adjacent) components expressed through unknown pseudo forces P_n and P_s. Basic solution is applied for the purpose.
- Calculation of stress coefficients and shift coefficients using the obtained ratio; formation of equation system to determine P_n and P_s forces in accordance with specified boundary conditions and deformation conditions.
- Obtained system of algebraic equations solving (Gauss method is applied for the algorithm); calculation of P_n and P_s forces providing fulfillments of specified boundary conditions and deformation conditions.
- Calculation of stresses and shifts within area under analysis from determined forces and specified load which provide implementation of specified criteria if taken together.

Ratios for shifts in the line of normally to i^{th} boundary component u_n^i and in the line of tangent to it u_s^i are:

$$\left.\begin{aligned} u_n^i &= \sum_{J=1}^{N} B_{ns}^{ij} \cdot P_s^j + \sum_{J=1}^{N} B_{nn}^{ij} \cdot P_n^j \\ u_s^i &= \sum_{J=1}^{N} B_{ss}^{ij} \cdot P_s^j + \sum_{J=1}^{N} B_{sn}^{ij} \cdot P_n^j \end{aligned}\right\}. \tag{2.1}$$

Ratios for correspondent stresses of i^{th} component are

$$\left.\begin{aligned} \sigma_n^i &= \sum_{J=1}^{N} A_{ns}^{ij} \cdot P_s^j + \sum_{J=1}^{N} A_{nn}^{ij} \cdot P_n^j \\ \sigma_s^i &= \sum_{J=1}^{N} A_{ss}^{ij} \cdot P_s^j + \sum_{J=1}^{N} A_{sn}^{ij} \cdot P_n^j \end{aligned}\right\}. \tag{2.2}$$

In (2.1) and (2.2) ratios, N is the number of boundary elements; B_{ss}^{ij}, B_{sn}^{ij}, B_{ns}^{ij}, and B_{nn}^{ij} are coefficients of shift effect; $A_{ss}^{ij}, A_{sn}^{ij}, A_{ns}^{ij}$, and A_{nn}^{ij} are coefficients of stress effect.

According to the analytical solution, effect coefficients are determined on [106] formulas

$$B_{ss}^{ij} = \frac{1}{2G}\left[(3-4\nu)cos\,\gamma\overline{F}_1 - \overline{y}\left(sin\,\gamma\overline{F}_2 - cos\,\gamma\overline{F}_3\right)\right],$$

$$B_{sn}^{ij} = \frac{1}{2G}\left[(3-4\nu)sin\,\gamma\overline{F}_1 - \overline{y}\left(cos\,\gamma\overline{F}_2 + sin\,\gamma\overline{F}_3\right)\right],$$

$$B_{ns}^{ij} = \frac{1}{2G}\left[-(3-4\nu)sin\,\gamma\overline{F}_1 - \overline{y}\left(cos\,\gamma\overline{F}_2 + sin\,\gamma\overline{F}_3\right)\right], \qquad (2.3)$$

$$B_{nn}^{ij} = \frac{1}{2G}\left[(3-4\nu)cos\,\gamma\overline{F}_1 + \overline{y}\left(sin\,\gamma\overline{F}_2 - cos\,\gamma\overline{F}_3\right)\right],$$

$$A_{ss}^{ij} = -2(1-\nu)\left(sin\,2\gamma\overline{F}_2 - cos\,2\gamma\overline{F}_3\right) - \overline{y}\left(sin\,2\gamma\overline{F}_4 + cos\,2\gamma\overline{F}_3\right),$$

$$A_{sn}^{ij} = (1-2\nu)\left(cos\,2\gamma\overline{F}_2 + sin\,2\gamma\overline{F}_3\right) - \overline{y}\left(cos\,2\gamma\overline{F}_4 - sin\,2\gamma\overline{F}_5\right), \qquad (2.4)$$

$$A_{ns}^{ij} = \overline{F}_2 - 2(1-\nu)\left(cos\,2\gamma\overline{F}_2 + sin\,2\gamma\overline{F}_3\right) - \overline{y}\left(cos\,2\gamma\overline{F}_4 - sin\,2\gamma\overline{F}_3\right),$$

$$A_{nn}^{ij} = \overline{F}_3 - (1-2\nu)\left(sin\,2\gamma\overline{F}_2 - cos\,2\gamma\overline{F}_3\right) + \overline{y}\left(sin\,2\gamma\overline{F}_4 + cos\,2\gamma\overline{F}_5\right)$$

In (2.3) and (2.4) formulas, G is rock shift module; $\overline{F}_1(\overline{x},\overline{y}) = f(\overline{x},\overline{y})$; $\overline{x}^i$, and $\overline{y}^i$ are local coordinated connected with ith component;

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

$$\left. -\left(\overline{x}-a^j\right)\cdot\ln\sqrt{\left(\overline{x}-a^j\right)^2+\overline{y}^2} + \left(\overline{x}-a^j\right)\ln\sqrt{\left(\overline{x}-a^j\right)^2+\overline{y}^2}\right]$$ is a function characterizing effect of pseudo loads applied to jth component in terms of stresses and shifts of ith component;

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

$\gamma = \beta^i - \beta^j$ is an angle identifying relative position of local coordinates $(\overline{x}^i, \overline{y}^i)$ of ith component and $(\overline{x}^j, \overline{y}^j)$ of jth component; and $2a^j$ is length of jth component.

$$\begin{aligned} \overline{x} &= (\overline{x}^i - \overline{x}^j)\cos\beta^j + (\overline{y}^i - \overline{y}^j)\sin\beta^j, \\ \overline{y} &= -(\overline{x}^i - \overline{x}^j)\sin\beta^j + (\overline{y}^i - \overline{y}^j)\cos\beta^j \end{aligned} \qquad (2.5)$$

(2.5) formulas determine local coordinates of i component center with reference to j component center.

Proprietary effect coefficients, characterizing effect of pseudo loads P_n^i and P_s^i of i^{th} component as for shifts and stresses of i^{th} component, result in accordance with expressions (2.3) and (2.4) if $\bar{x} = \bar{y} = \gamma = 0$. They are:

$$B_{sn}^{ij} = B_{ns}^{ij} = 0; \qquad B_{ss}^{ii} = B_{nn}^{ii} = -\frac{3-4\nu}{4\pi G(1-\nu)} a^i \ln a^i;$$

$$A_{sn}^{ij} = A_{ns}^{ij} = 0; \qquad A_{ss}^{ii} = A_{nn}^{ii} = \frac{1}{2} \tag{2.6}$$

$$A_{ts}^{ii} = A_{tn}^{ii} = 0; \qquad A_{tt}^{ii} = \frac{1}{2}\frac{\nu}{1-\nu}; \qquad \bar{y} = 0 \pm. \tag{2.7}$$

In general case resolving system of algebraic equations is:

$$\left.\begin{aligned} \sum_{J=1}^{N} C_{ss}^{ij} P_s^j + \sum_{j=1}^{N} C_{sn}^{ij} P_n^j = b_s^i \\ \sum_{J=1}^{N} C_{ns}^{ij} P_s^j + \sum_{j=1}^{N} C_{nn}^{ij} P_n^j = b_n^i \end{aligned}\right\}_{i=1,2,\ldots,N}, \tag{2.8}$$

where C_{ss}^{ij}, C_{sn}^{ij}, C_{ns}^{ij}, and C_{nn}^{ij} effect coefficients; depending upon conditions established within boundaries they are determined either on (2.3), and (2.6) formulas or on (2.4), and (2.7) formulas;

b_s^i and b_n^i are values determined with the help of conditions established at boundary surfaces and contact ones.

2.4 Determination of stresses and shifts within a zone of mining effect

Above calculation algorithm has been used to determine displacements within a zone of stope effect where adjacent seams were mined. Conditions of advance overworking were considered. Relevant analytical model in Fig. 2.1 demonstrates state of involved rock mass area after regular caving of main roof of mined seams when undisturbed rock mass part experiences elastic deformation.

Finite-element approximation of involved area boundaries was implemented according to paradigm shown in Fig. 2.2.

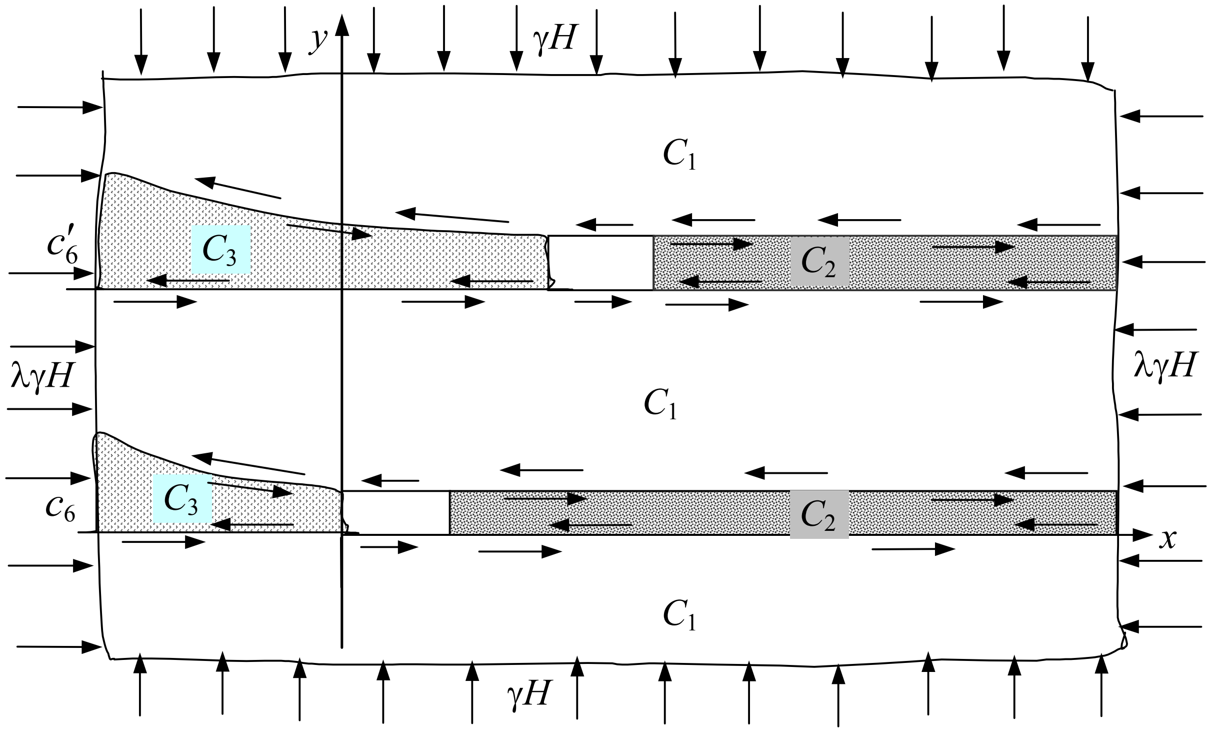


Fig. 2.2. Finite-element paradigm of involved area boundaries

The problem was solved in secondary stresses. Since composite stresses within free boundaries (C_1 is enclosing rock, C_2 is coal seams, and C_3 are caved rocks) are equal to zero, secondary stresses resulted from mine workings formation were set equal to γH if vertical direction, and λγH if horizontal one ($\lambda = \nu/(1-\nu)$, ν is Poisson ratio of rock).

Thus, right members in correspondent equations of initial system (2.8) acquire following form:

$$b_s^i = -\lambda\gamma H\,, \quad b_n^i = -\gamma H\,,$$

and effect coefficients are respectively:

$$C_{ss}^{ij} = \begin{cases} A_{ss}^{ij}, \text{if } j^{th} \text{ component belongs to } C_k \text{ area boundary}, \\ 0, \text{if } j^{th} \text{ component doesn't belongs to } C_k \text{ area boundary}, \end{cases}$$

$$C_{sn}^{ij} = \begin{cases} A_{sn}^{ij}, \text{if } j^{th} \text{ component belongs to } C_k \text{ area boundary}, \\ 0, \text{if } j^{th} \text{ component doesn't belongs to } C_k \text{ area boundary}, \end{cases}$$

(2.9)

$$C_{ns}^{ij} = \begin{cases} A_{ns}^{ij}, \textit{if } j^{th} \text{ component belongs to } C_k \text{ area boundary}, \\ 0, \text{if } j^{th} \text{ component doesn't belongs to } C_k \text{ area boundary}, \end{cases}$$

$$C_{nn}^{ij} = \begin{cases} A_{nn}^{ij}, \text{if } j^{th} \text{ component belongs to } C_k \text{ area boundary}, \\ 0, \text{if } j^{th} \text{ component doesn't belongs to } C_k \text{ area boundary}, \end{cases}$$

where *k* is the number of area having free boundaries.

Compatibility conditions were fulfilled at rock-coal contact surfaces. If i and i^* are the numbers of contacting elements belonging to different areas then while establishing conditions of displacement compatibility in stresses within adjacent contour of C_κ area and in displacements within adjacent contour of C_l area when $k > l$ we have respectively:

$$\sigma_s^i = \sigma_s^{i[k]} - \sigma_s^{i^*[l]} = 0, \quad \sigma_n^i = \sigma_n^{i[k]} - \sigma_n^{i^*[l]} = 0;$$

$$C_{ss}^{ij} = \begin{cases} A_{ss}^{ij[k]}, \text{if } j^{th} \text{ component belongs to } C_k \text{ area boundary,} \\ -A_{ss}^{i^*j[l]}, \text{if } j^{th} \text{ component belongs to } C_l \text{ area boundary} \end{cases} \tag{2.10}$$

and so on for C_{sn}^{ij}, C_{ns}^{ij}, and C_{nn}^{ij} components.

Similarly,

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

$$C_{ss}^{ij} = \begin{cases} B_{ss}^{i^*j[k]}, \text{if } j^{th} \text{ component belongs to } C_k \text{ area boundary,} \\ B_{ss}^{ij[l]}, \text{если } j^{th} \text{ component belongs to } C_l \text{ area boundary} \end{cases} \tag{2.11}$$

and so on for C_{sn}^{ij}, C_{ns}^{ij}, and C_{nn}^{ij} components.

Coefficients of stress effect A_{ss}^{ij}, A_{sn}^{ij}, A_{ns}^{ij}, and A_{nn}^{ij} in the context of (2.9) and (2.10) as well as coefficients of displacement effect in the context of (2.11) are calculated according to formulas of basic analytical solutions (2.3) and (2.4) respectively.

2.5 Calculation data and their analysis

Distribution pattern for stresses and displacements of involved rock mass area has been determined with the help of above algorithm. In the calculation process parameters L (distance between stopes of c_6' and c_6 seams) and thickness h of parting varied.

A system of 1852 linear algebraic equations developed using (2.10) and (2.11) conditions was solved with the help of Gaussian method for series of values of dimensionless key parameters. Their runs depended on possible variations of physical and mechanical parameters, and on structure of rock mass area under consideration.

Specific initial data were as follows:

$$E_n / E_y \in \{0.71;\ 0.94;\ 1.60\};$$

$$h/L \in \{0.09;\ 0.10;\ 0.12;\ 0.13;\ 0.15;\ 0.18;\ 0.20;\ 0.22;\ 0.24;\ 0.30;\ 0.40;\ 0.45;\ 0.60\};$$

$H/L \in \{4.3; 5.0; 5.29; 6.0; 6.17; 7.4; 7.5; 9.24; 10; 12.3; 15.0; 18.5\}$;

$h/H \in \{0.016; 0.020; 0.024; 0.030; 0.032; 0.040\}$;

$L/m_в \in \{28.6; 42.9; 51.1; 71.4; 85.7; 100\}$,

where E_n are E_y elastic moduli of enclosing rock and coal, MPa;
h is thickness of parting, m;
L is distance between stopes within adjacent seams, m;
H is mining depth, m;
$m_в$ and $m_н$ are thickness of upper seam and thickness of lower seam, m.

Elastic modulus E_p = 10 MPa and Poisson ratio ν_p = 0.499 for broken rocks were taken on operating schedules [81, p. 49] developed in ε_y and k_μ coordinates for different mining depth values (ε_y is maximum relative linear deformation of stope; $k_\mu = 1-2\nu^3/(1-\nu)$. Table 1.1 shows data on physical and mechanical characteristics of enclosing rocks and coal.

Stress state is estimated on P.P. Balandin criterion according to which equivalent stresses balanced against rock strength are determined as follows

$$\sigma_{eq} = \frac{(1-\psi)(\sigma_1+\sigma_3)}{2} + \frac{\sqrt{(1-\psi)^2(\sigma_1+\sigma_3)^2 + 4\psi(\sigma_1^2+\sigma_2^2-\sigma_1\sigma_3)}}{2}, \quad (2.12)$$

where σ_1, σ_2, and σ_3 are primary stresses, MPa; $\psi - \sigma_p/\sigma_c$; σ_p and σ_c are rock tensile strength and rock compression strength, MPa.

Fig. 2.3 demonstrates $\sigma_{экв}$ obtained on the criterion within considered area of rock mass. As it follows from the Figure, extension of maximum $\sigma_{экв}$ values zones within upper seam c_6' roof on x coordinate is almost 18m. Adequate dimension within floor of lower seam c_6 is 30m.

In the context of parting rocks, maximum stress values are available within area covering the whole its thickness including almost 30m in the line of Ox axis.

Fig. 2.4 explains calculated values of maximum concentration coefficients $(K_y)_{max}$ of σ_{yy} stresses relative to γH (H = 300 m) level within roof rocks of adjacent seams in zones of front abutment pressure in the context of different values of distance L and specified h = 15m value. It follows from the Figure that lower seam roof is more stresses; its $(K_y)_{max}$ is 1.36 times more to compare with upper seam roof.

Fig. 2.5 shows values of c_6 and c_6' seams floor and roof convergence at the distance of 4 m from stope towards goaf in terms of various distances L between stopes. It follows from the Figure that greater displacements take place within seam c_6 roof in the context of virtually the same curve shape.

Fig. 2.6 *a* and *b* describe obtained curves of equivalent stresses σ_{eq} within roof and floor of c_6 seam if parting thickness is h = 5 m. Curves 1 refer to a case when L = 30 m, and curves 2 refer to a case when L = 70 m.

Fig. 2.7 *a* and *b* describe curves of σ_{eq} stresses within roof and floor of c_6 seam if h = 15 m. Curves 1 in this Figure were obtained for L = 30 m, and curves 2 for L = 60 m.

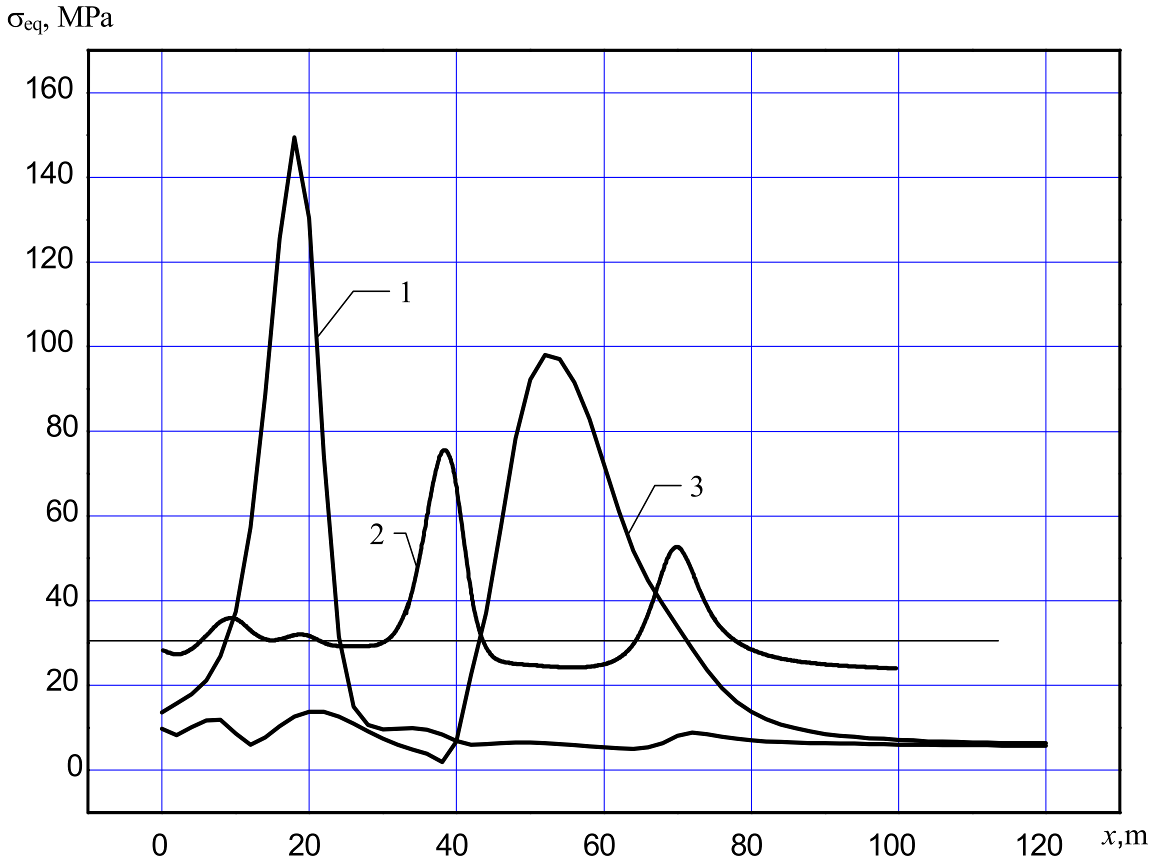


Fig. 2.3.Equivalent σ_{eq} stresses according to P.P. Balandin criterion
1 – in upper seam roof; 2 in parting; 3 in lower seam floor

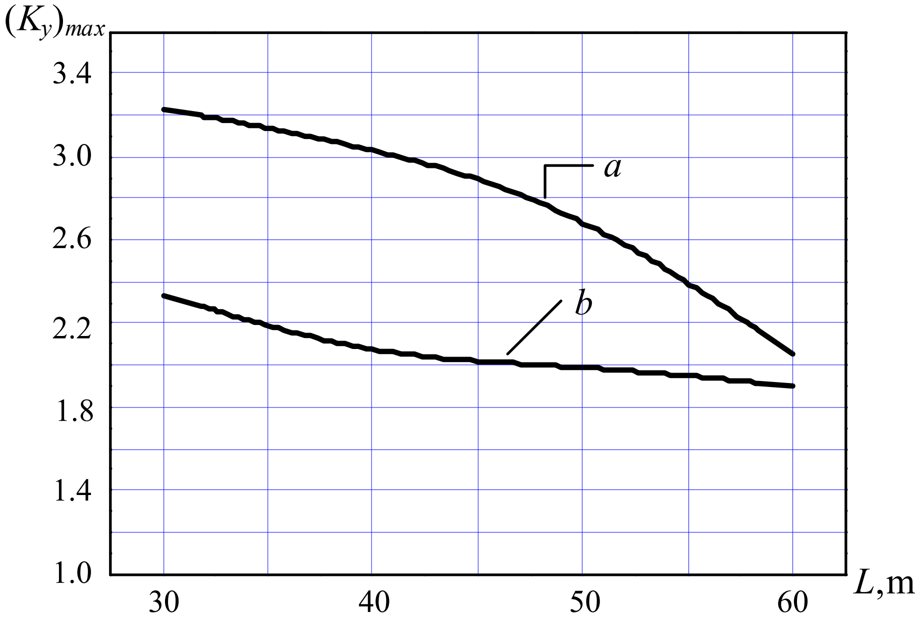


Fig. 2.4. Maximum concentration coefficients of σ_{yy} stresses in the roof of: a – lower c_6 seam; and b – upper c_6' seam.

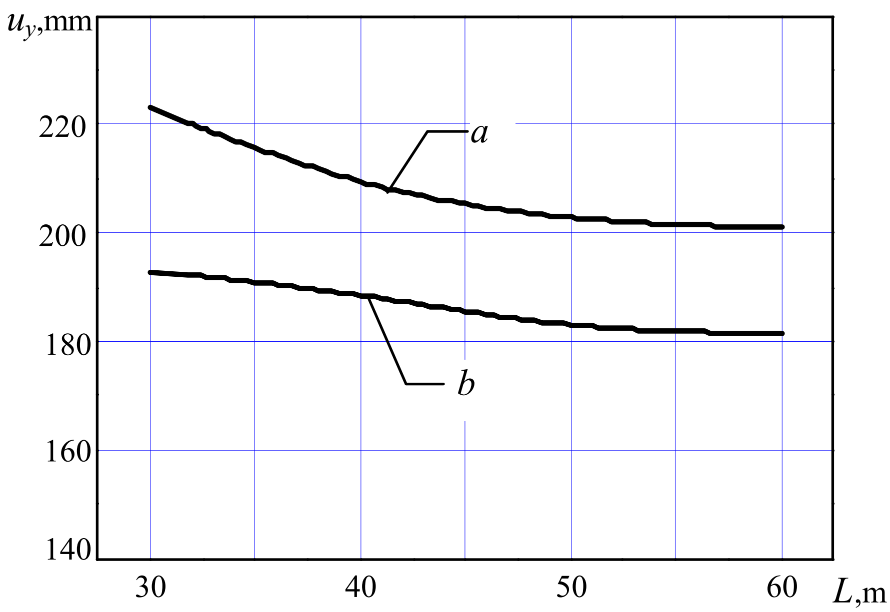


Fig. 2.5. Floor and roof convergence at the distance of 4m from stope: a – c_6 seam; b – c_6^1 seam.

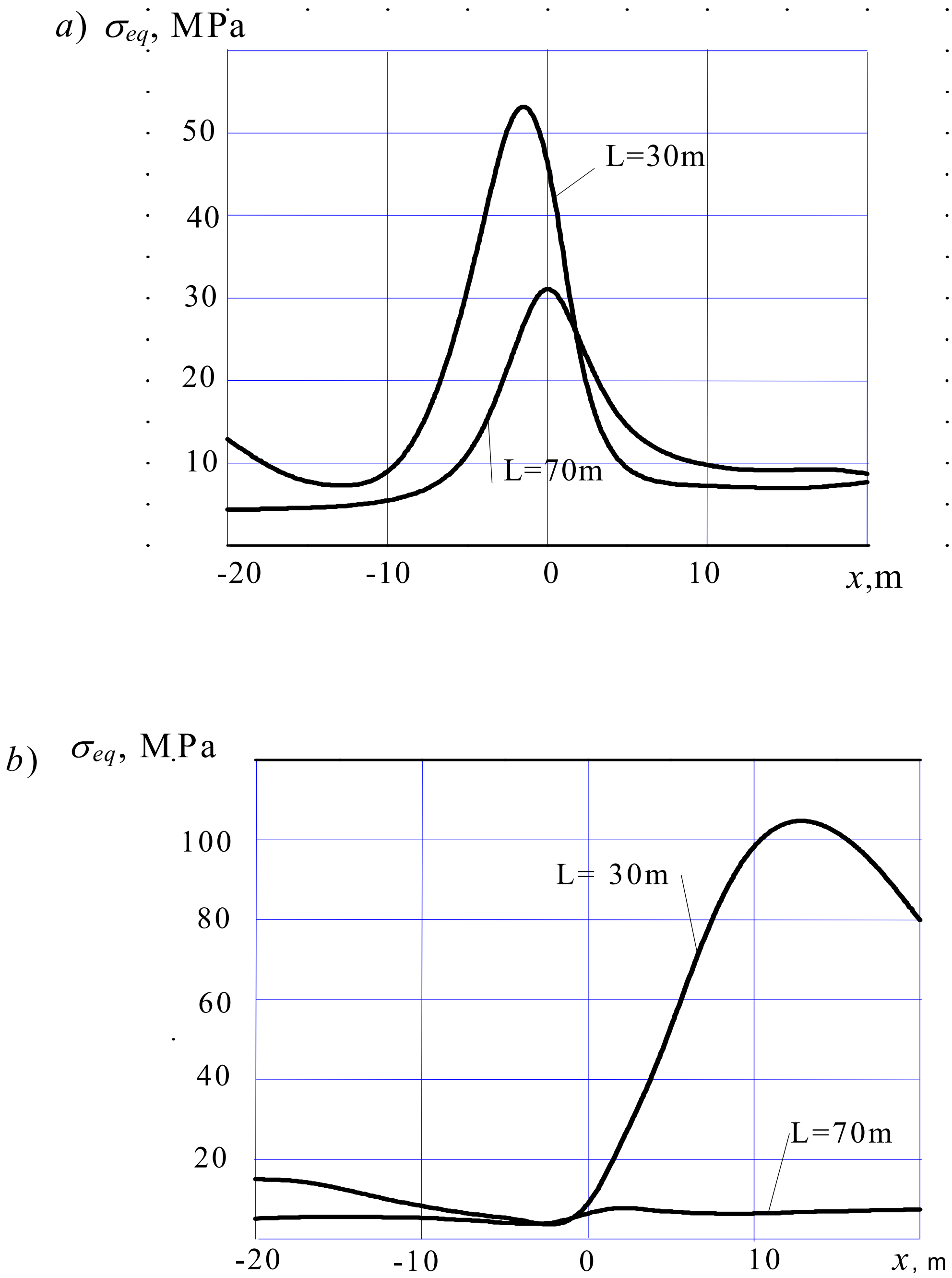
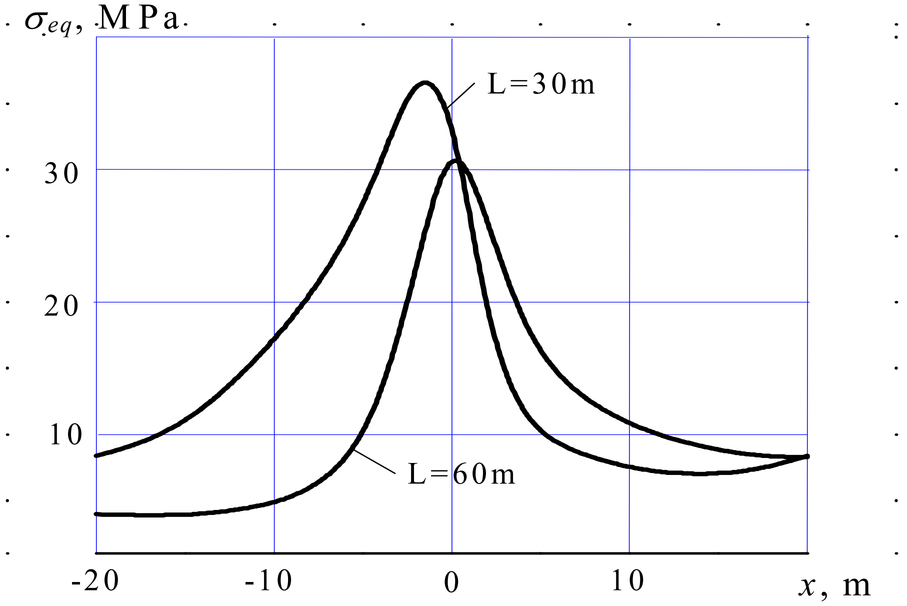


Fig. 2.6. $\sigma_{экв}$ *stresses according to P.P. Balandin criterion in a roof of* c_6 *seam (a) and in its floor (b) if parting thickness is* $h = 5m$.

It follows from Fig. 2.6 that in the context of 30 m L distance between stopes and parting thickness $h = 5$ m in terms of adjacent c_6 and c_6^1 seams, extension of potential failure area within lower seam roof in the neighbourhood of a stope is 10 m; it is 13 m in floor. If $L = 70$ m equivalent stresses don’t exceed rock strength of floor and roof.

a)



b)

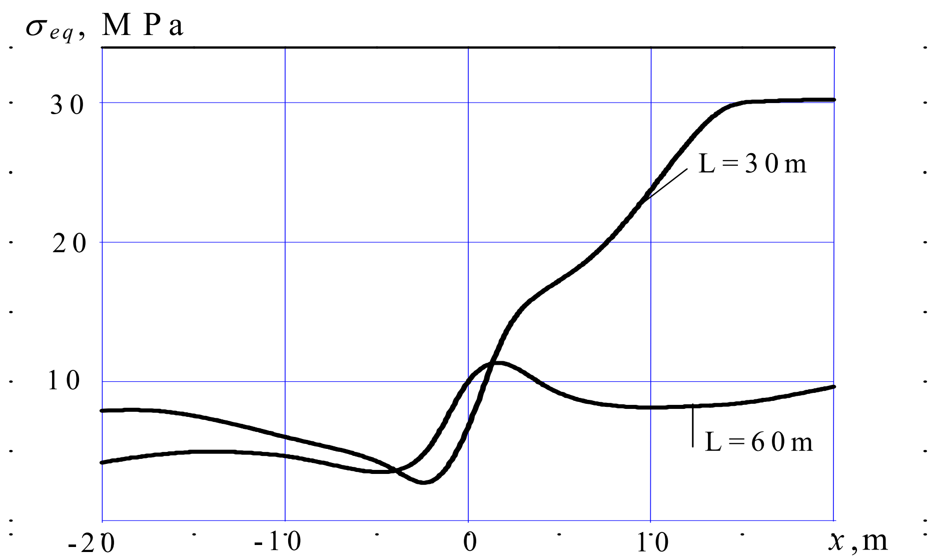


Fig. 2.7. σ_{eq} stresses on P.P. Balandin criterion
a) within c_6 seam roof c_6; b) within c_6 seam floor
if parting thickness is h = 15 m.

Comparable situation is available within c_6 seam roof in the context of L = 30 m and L = 60 m if only h = 15 m (Fig. 2.7 *a*, *b*).

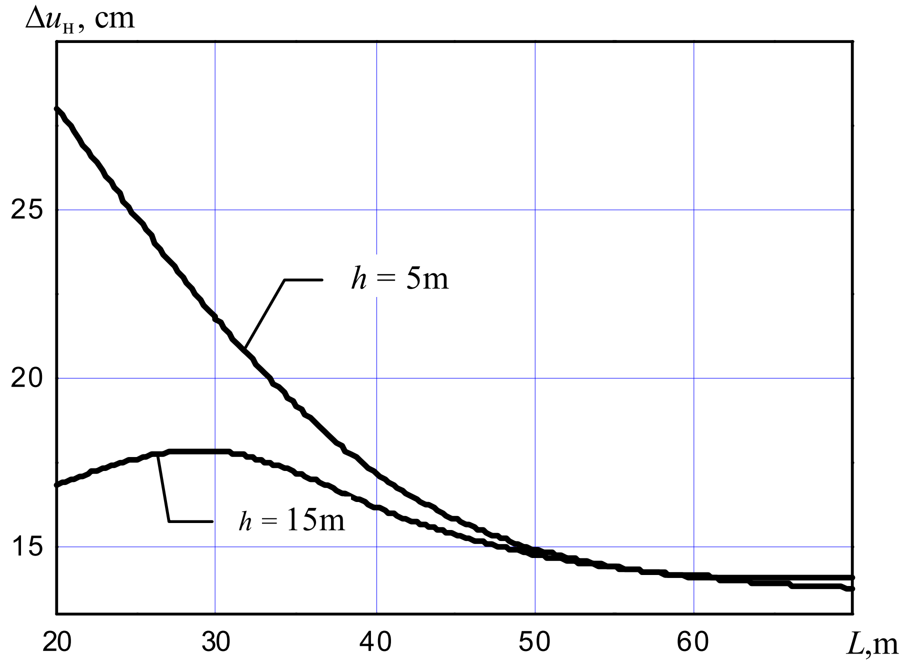
If so, stresses in a seam floor are not dangerous.

Hence, in terms of rock pressure when parting thickness is h = 5 m distance between stopes of adjacent c_6 and c_6' seams being 70 m is bearable for Western Donbass mines.

If parting thickness is h = 15 m, stopes of adjacent seams should be at a distance of $L \geq 60$ m from each other.

Figures 2.8, 2.9, and 2.5 illustrate certain results of calculations concerning displacements of enclosing roof and floor within stopes of adjacent seams in terms of different distances between them. The calculations have been performed for the involved practice when upper of two adjacent seams is mined.

a)



b)

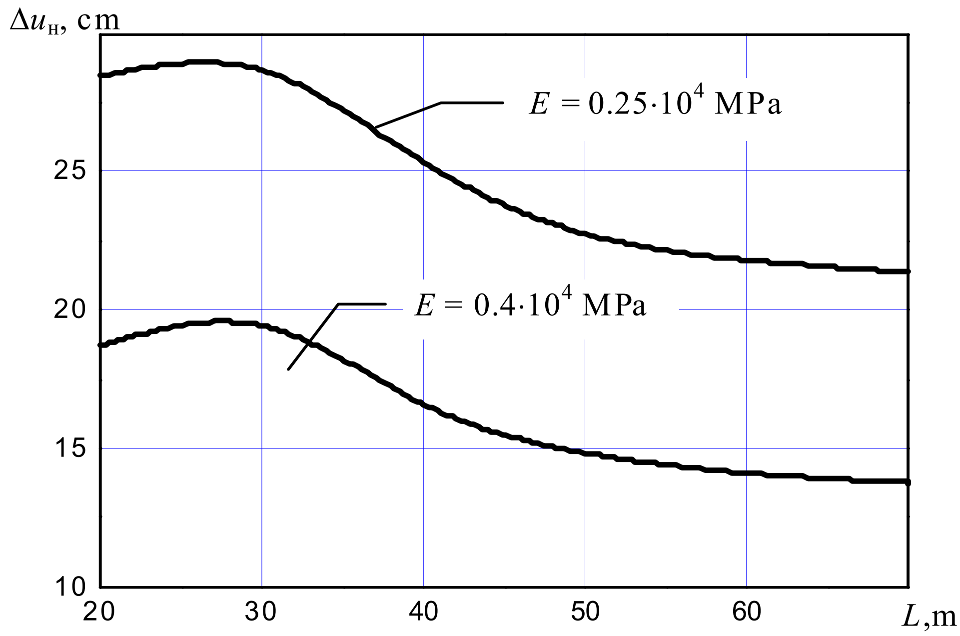


Fig. 2.8. Maximum floor and roof convergence of lower seam in the context of:
a) different values of parting thickness h;
б) different values of enclosing rock elasticity module E_n.

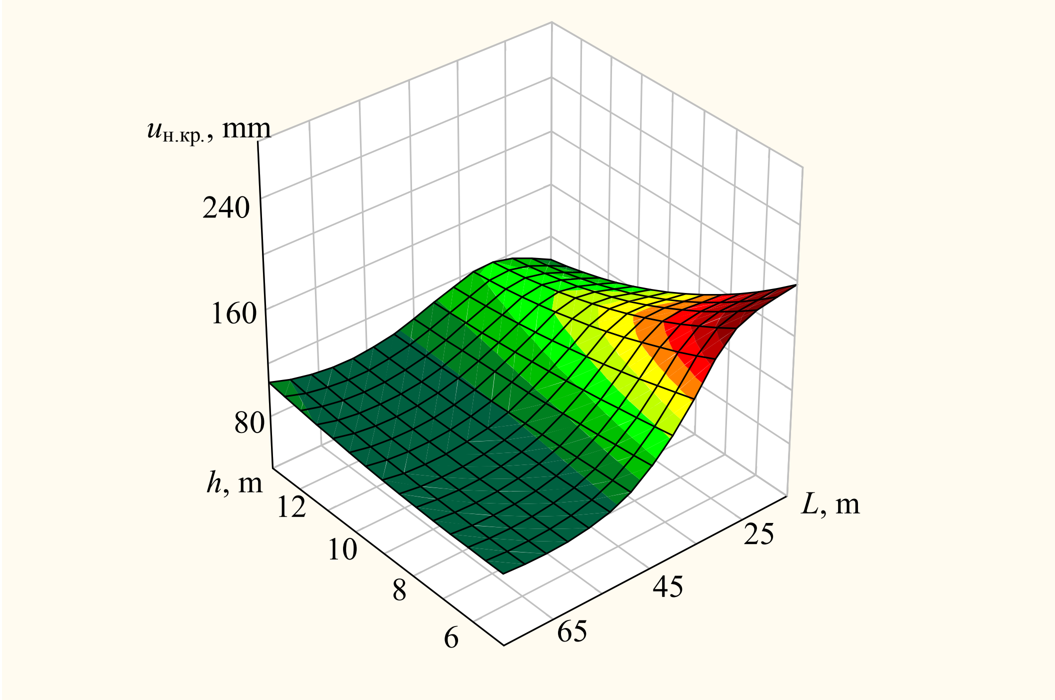


Fig. 2.9. Surface of maximum vertical displacements of lower seam

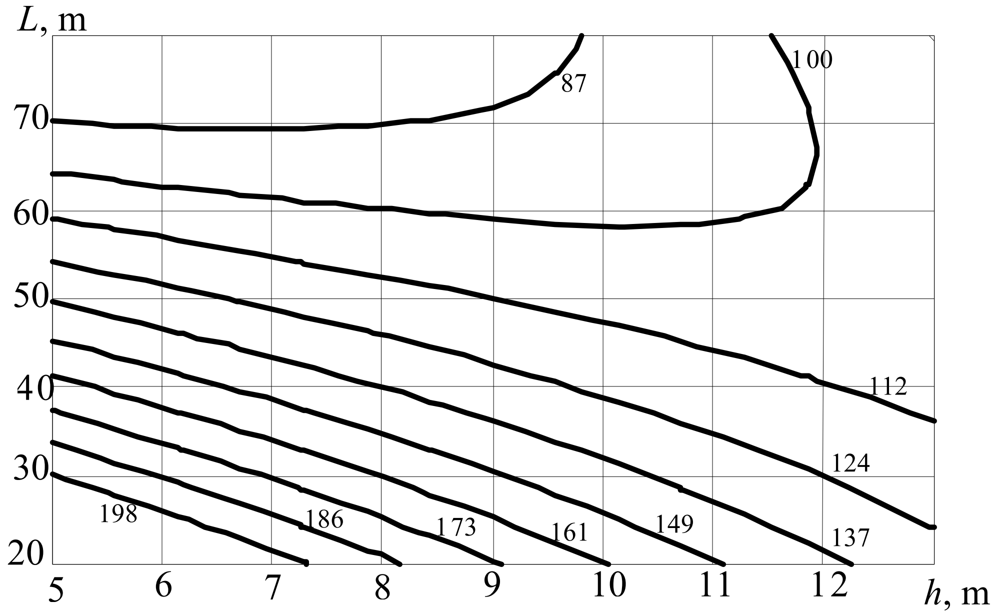


Fig. 2.10. Isolines of maximum vertical displacements of lower seam, mm

Problem is to determine bearable in terms of breaking factor enclosing roof and floor convergence in terms of advance stope and behind one which can not be more than 0.3 of seam thickness.

Developed mathematical apparatus makes it possible to determine displacements of enclosing roof and floor within stopes involving available mining and geological conditions of seam mining. Then results of displacement calculation for boundary values of interburden layer thickness are quoted; they are 5 and 15m. Besides the program helps determine the desired stresses and stresses and, finally allowable distances between stopes for any concerned conditions of Western Donbass mines.

Fig. 2.8 shows maximum convergence curves $\Delta u_н$ of lower seam floor and roof at a depth of H = 300 m depending upon L parameter for two values of interburden layer thickness h = 5 m and h = 15 m, and for two values of enclosing roof and floor elastic modulus E_n= 0.25·10^4 MPa and E_n = 0.4·10^4 MPa. It follows from Fig. 2.8 *a* that changes in h modify nature of convergence distribution $\Delta u_н$: when h = 15 m, maximum corresponding to distance L = 30 m manifests itself. Starting from L = 60 m values, roof and floor displacements are less than allowable values (0.3 m). In the context of permanent convergence distribution pattern, increase in E_n parameter results in decrease of $\Delta u_н$ value (Fig. 2.8, *b*).

For descriptive reasons Fig. 2.9 demonstrates graphic model of maximum vertical displacements of lower seam roof being adequate to different values of L and h parameters. Fig. 2.10 shows isolines of maximum vertical displacements within lower seam roof if H = 300 m, $m_н$ = 0.7m, E_y = 0.3·10^4 MPa, and E_n = 0.4·10^4 MPa. They are helpful to determine $(\Delta u_н)_{max}$ values of L and h parameters.

Results of statistical processing of data obtained as a consequence of mathematical experiment performed for involved mining and geological conditions [4, 5] help determine correlation ratios for maximum displacements of roofs of upper seam $(u_в)_{max}$ and lower seam $(u_н)_{max}$ and for maximum convergences $(\Delta u_в)_{max}$ and $(\Delta u_н)_{max}$ being as follows:

$$\frac{(u_в)_{\max}}{m_в} = 0.2593 - 0.1283\frac{E_n}{E_y} - 0.3005\frac{L}{H} - 0.1649\frac{h}{L} + 0.00503\frac{H}{L} + 0.6753\frac{h}{H};$$

$$\frac{(\Delta u_в)_{\max}}{m_в} = 0.5689 - 0.2405\frac{E_n}{E_y} - 1.5104\frac{L}{H} + 0.00171\frac{L}{h} - 0.1267\frac{h}{L} + 0.00241\frac{L}{m_в};$$

$$\frac{(u_н)_{\max}}{m_н} = 0.3255 - 0.1636\frac{E_n}{E_y} - 0.6\frac{L}{H} - 0.5031\frac{h}{L} + 0.0143\frac{H}{L} + 2.4638\frac{h}{H}; \qquad (2.13)$$

$$\frac{(\Delta u_н)_{\max}}{m_н} = 0.7358 - 0.3468\frac{E_n}{E_y} - 0.8630\frac{L}{H} + 0.0123\frac{H}{L} - 0.3441\frac{h}{L}.$$

For the dependences, correlation ratios R are 0.947, 0.960, 0.942, and 0.738, respectively.

(2.13) ratios are required to determine L distance between stopes of contiguous seams; when the thickness of interburden layer under considered conditions maximum displacements within stopes don't exceed acceptable values if advance overworking takes place. They have been obtained for Western Donbass mines; moreover, they can be used to analyze stress and strain state of layer nonhomogeneous mass inclosing mine workings within contiguous seams in the context of analogous mining and geological conditions.

Calculated with the help of (2.13) formulas, maximums of roof and floor rock convergences within lower seam (when distance between longwalls is 70 m) are $(\Delta u_н)_{max}$ = 131 mm. They are $(\Delta u_в)_{max}$ = 66.7 mm in terms of upper one. Proper values are $(\Delta u_н)_{max}$ = 123.7 mm and $(\Delta u_в)_{max}$ = 65.47 mm if L=60 m.

Hence, in both cases neither stresses nor displacements exceed acceptable values.

Conclusions

1. On the basis of boundary-element method, calculation algorithm to determine stress and strain state of nonhomogeneous mass has been developed as applied to contiguous coal seams mining in terms of overworking.

2. The calculation algorithm developed for mining and geological conditions of Western Donbass mines has helped determine the nature of changes in maximum floor and roof convergence of coal seam being mined in the context of overworking depending upon enclosing roof and floor elasticity modulus and distance between stopes within contiguous seams.

3. Quantative evaluation concerning degree of effect of interbed thickness and distance between stopes of contiguous seams on vertical displacements of coal seam roof in the context of overworking has been performed.

4. Results of statistical processing of data of mathematical experiment performed with the help of developed calculation algorithm have helped determine correlation ratios for maximum displacements of upper seam and lower seam roofs. The ratios take into consideration elasticity moduli of enclosing rocks, and coal, thickness of seams and interbed as well as distance between stopes within contiguous seams.

5. The correlation dependences are required to determine L distance between stopes of contiguous seams when L distance between stopes of contiguous seams; when the thickness of interburden layer under considered conditions maximum displacements within stopes don't exceed acceptable values if advance overworking takes place.

6. The calculations have helped formulate scientific postulate one: boundaries of concentration of mining operations within contiguous seams depend on a factor of their mutual effect characterized by allowable stresses and displacements of enclosing rocks within longwalls; their determination is implemented with the help of boundary-element method in the context of mining depth, interbed thickness (h), distance between longwalls as well as strength and deformation characteristics of enclosing rocks. In terms of considered conditions of Western Donbass mines, safe distances between stopes of contiguous seams are: no less than 70 m if h = 5 m and 60 m if h=15 m.

CHAPTER 3. SUBSTANTIATION OF DEVELOPMENT WORKING RATIONAL SITE IN THE CONTEXT OF CONCENTRATION OF MINING

3.1 General information

Operational stability is one of the basic conditions to mine development workings effectively. First of all, in the context of contiguous seams, stability of development workings depends on their site, priority of seam mining, interbed thickness, and physical and mechanical properties of enclosing rocks.

Siting location of extraction drift according to seams to be mined secondarily is of great importance. The matter is that they experience extra rock pressure resulting from mining operations taking place within upper seam. It stands for reason that mine workings will have the least loading if they are placed within de-stressed zone being a result of upper seam winning. However, within following after winning selvedge of the seam providing the distress zone, mine workings may be killed due to sharp increase in rock pressure. It has been determined that construction of mine workings within pressure zone is less unfavourable than effect of pressure rise on a mine working has already been driven [116].

A technique to protect mine workings within contiguous seams is very important as well. Both practice and scientific findings show that pillars left to protect inby development workings within any of them have an adverse effect on development working of next seam being mined. Hence, it is generally accepted that pillarless protection of extraction drifts within contiguous seams is useful [117, 118, 119].

As for the sequence of mining, for a variety of advantages, mines in Western Donbass like the majority of mines in Ukraine exercise descending sequence of seam mining including contiguous seams. Primary mining is individual. Contiguous seams are mined simultaneously with advance mining of overlying seam.

Standards [31] stipulate location of lower seam workings under worked-out area with displacement relatively to overlying seam mining. Distance between them depends upon mining depth and interbed thickness. In particular, if interbed thickness is 7…8 m and mining depth is 400...450 m then, according to Standards [31], distance between extraction drifts within contiguous seams should be 10…15 m.

If mining takes place in the context of lower seam following longwall within upper seam (300 m level) recommended distance should be no less than 200m. When this occurs, the mine working enters a zone of reset rock pressure. According to data by paper [120], active stage of overworked rock mass is completed at the distance equal to length of longwall being mined.

Primal problem is to determine such a site for development working which would exclude extra effect of mining operations performed in terms of overlying seam and provide sufficient possibility to plan and perform mine operations in the context of overworked lower seam. To do that, it is required to identify minimum (from the viewpoint of development working stability) distance from overworking boundary being edge of overlying seam.

Search for adequate parameters should obviously involve the fact that stability of mine workings has to be provided during rather long period. That is why calculation algorithm should take into consideration temporal changes in physical and mechanical properties of enclosing rocks. Moreover, calculation results also depend on rock structure and thickness of a seam being mined.

3.2 Description of structural model

Analysis of stress and strain state of rock mass area under study performed to determine rational location of development working in the context of overworking involves two-dimensional structural model. It goes without saying that the procedure can not consider effect of mining operations; that is why surcharging coefficients are introduced into the calculation algorithm. The coefficients are calculated on parameters of increased rock pressure zone. They are ratio between working stresses σ_{yy} acting within proper cross-sections of the zone and γH level. Surcharging coefficients are determined using structural model shown in Fig. 3.1.

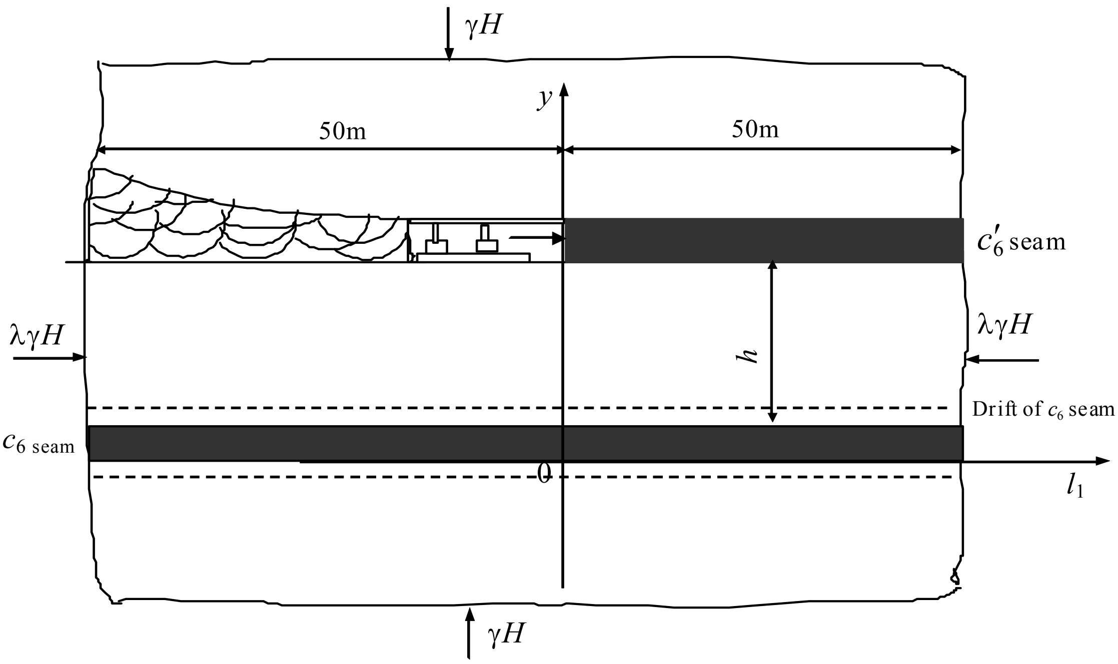


Fig. 3.1. On the determination of stress concentration coefficient σ_{yy} within a roof of development working (c_6 seam)

Fig. 3.2 demonstrates appropriate boundary and element diagram

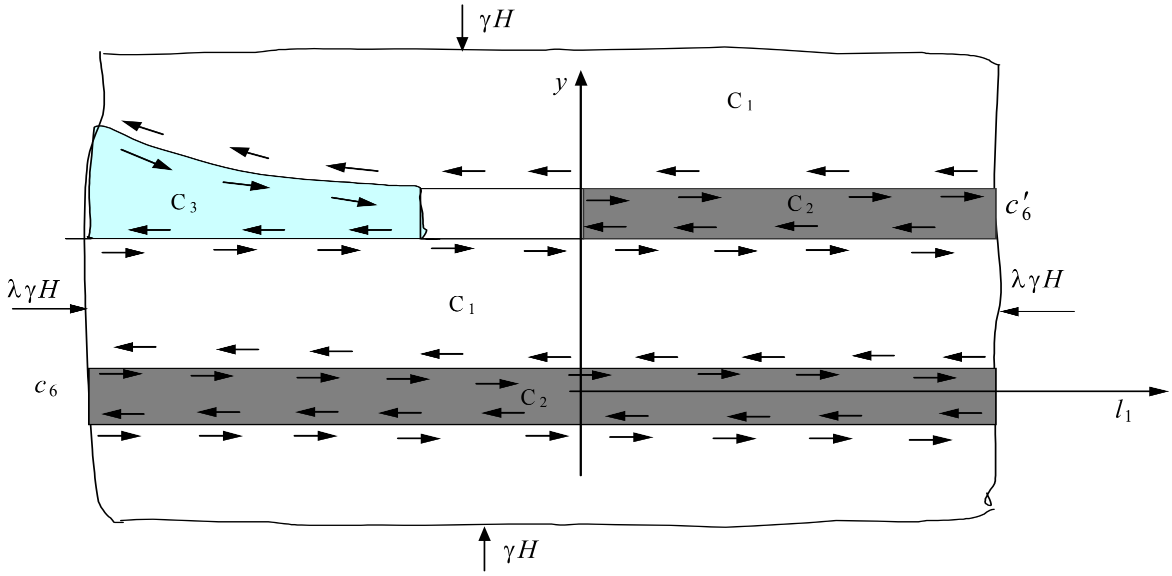


Fig.3.2. Discretization of boundaries of area under study

Resolving equation system was being formed depending upon boundary conditions (2.9) for elements of free boundaries and (2.10) for contacting elements. The calculation results help identify maximum values of surcharging coefficients within roof along axis of development working located in lower seam at different l_1 distances from a stope of overlying seam (Fig.3.3). The values are required for further calculations.

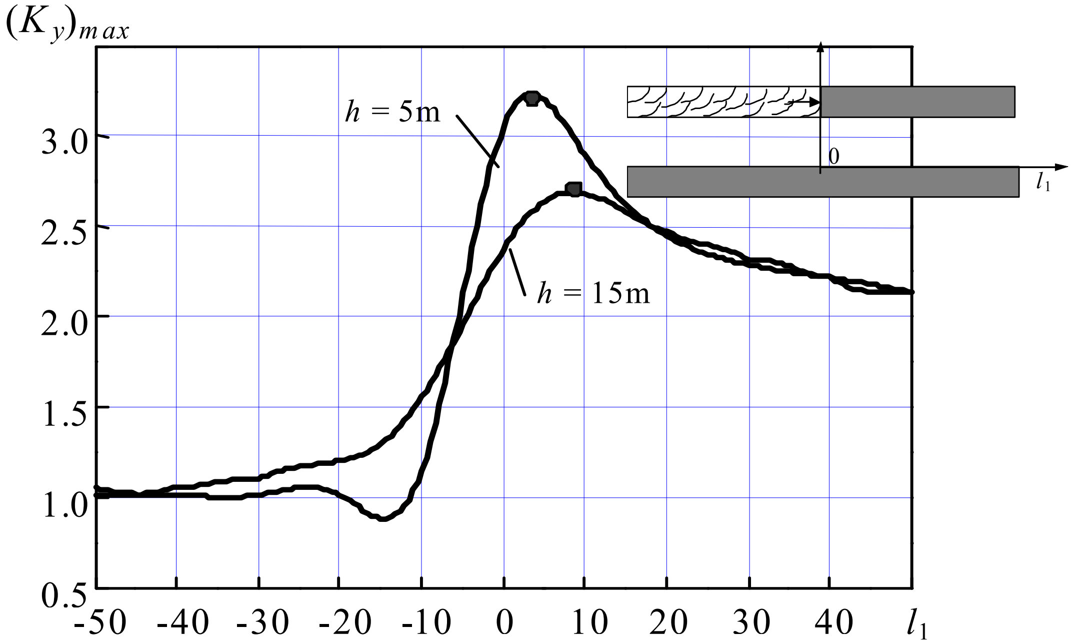


Fig. 3.3. Maximum surcharging coefficients within a roof of development working

To determine rational value of l distance from lower seam mine working contour to the edge of overlying seam, the monograph considers structural model shown in Fig. 3.4.

Fig. 3.5 demonstrates appropriate discretization diagram of boundaries of area under study.

The calculations concerned two boundary values of interbed thickness typical for Western Donbass: h = 5m and h = 15m. Formulation of boundary conditions used surcharging coefficients itemized with points in Fig. 3.3. Their values were $(K_y)_{max}$ = 3.2 if h = 5m and $(K_y)_{max}$ = 2.7 if h = 15 m; moreover, they were among the most dangerous cross-sections of mine working located at the distance of l_1 = 5 m and l_1 = 10 m respectively.

In the context of physical aspect of the problem under solution, a model of linear and congenital medium with Abel creep kernel has been set for the rock mass. According to the model, while calculating stresses and displacements at various time moments, elasticity moduli of roof and floor rocks are determined as follows:

$$E_t = \frac{E}{1+\Phi_t}, \tag{3.1}$$

where E is elasticity modulus according to data by Table 1.1;

$\Phi_t = \frac{\delta}{1-\delta} t^{1-\alpha}$ is creep function;

α and δ are creep parameters; and

t is time, s.

In the context of broken rock, values of elasticity modulus E_p and Poisson ratio ν_p were similar to those in section 2.3.

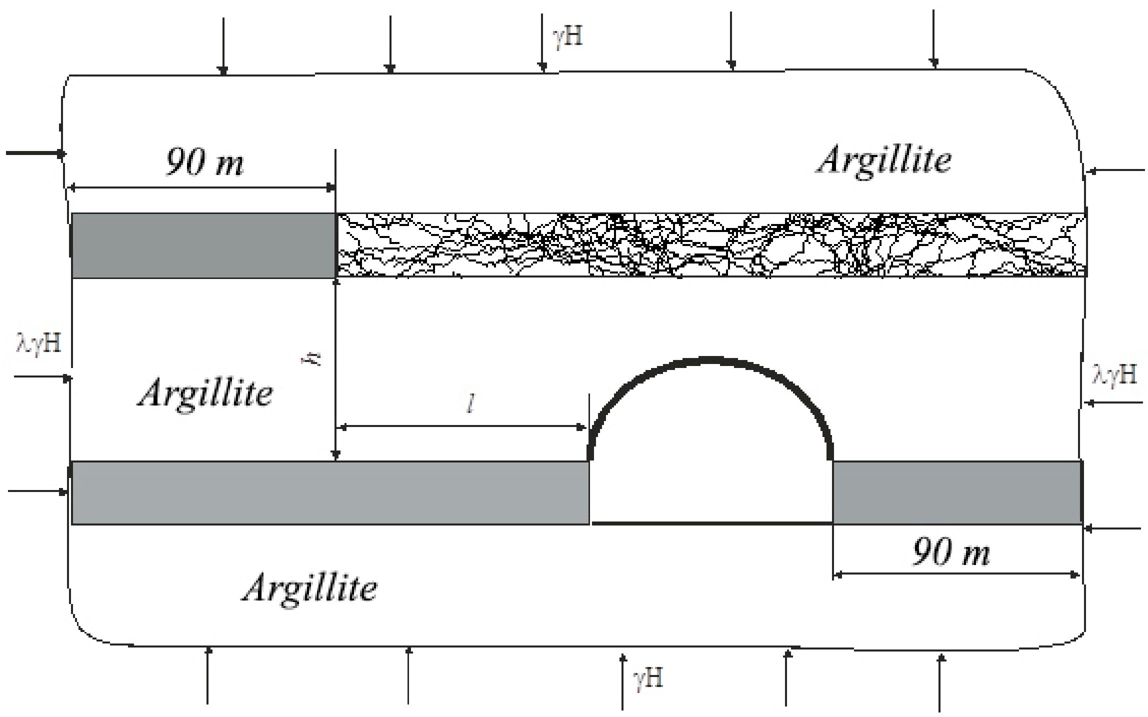


Fig. 3.4. On the determination of rational location of development working in the context of overworking

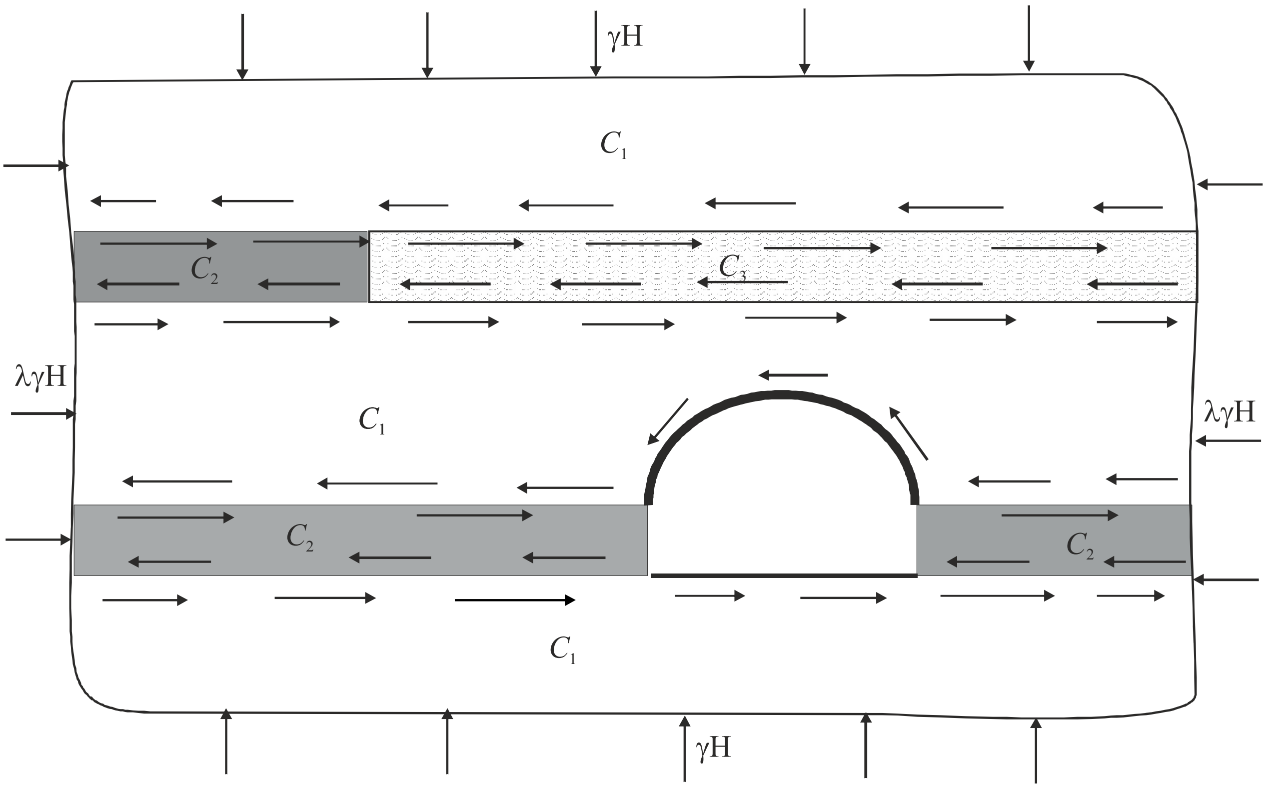


Fig. 3.5. Discretization of boundaries

3.3 Calculation results and their analysis

Below you can find calculation results concerning stresses in the neighbourhood of mine working and rock displacements in the context of contour; they are important for rational selection of its siting. Fig. 3.6 demonstrates diagrams of equivalent stresses within roof and floor of mine working if l = 0, and interbed thickness is h = 5 m. They are asymmetrical. Stress maximum falls within $\varphi \approx 120^0$ cross-section in roof being 250 MPa; that far exceeds rock strength.

All the stress diagrams to be discussed below have been described involving time for extraction pillars mining.

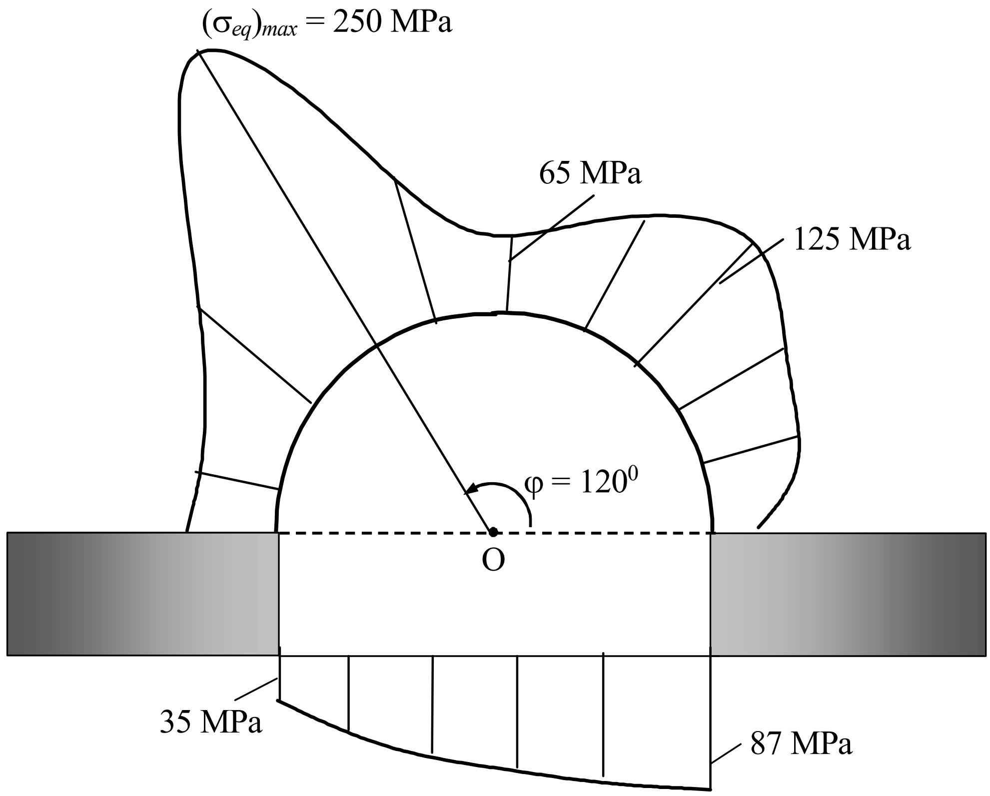


Fig. 3.6. Equivalent stress diagrams along contour of mine working if h = 5 m, and l = 0

Figures 3.7 and 3.8 illustrate nature of changes in equivalent stresses within roof of mine working along its contour.

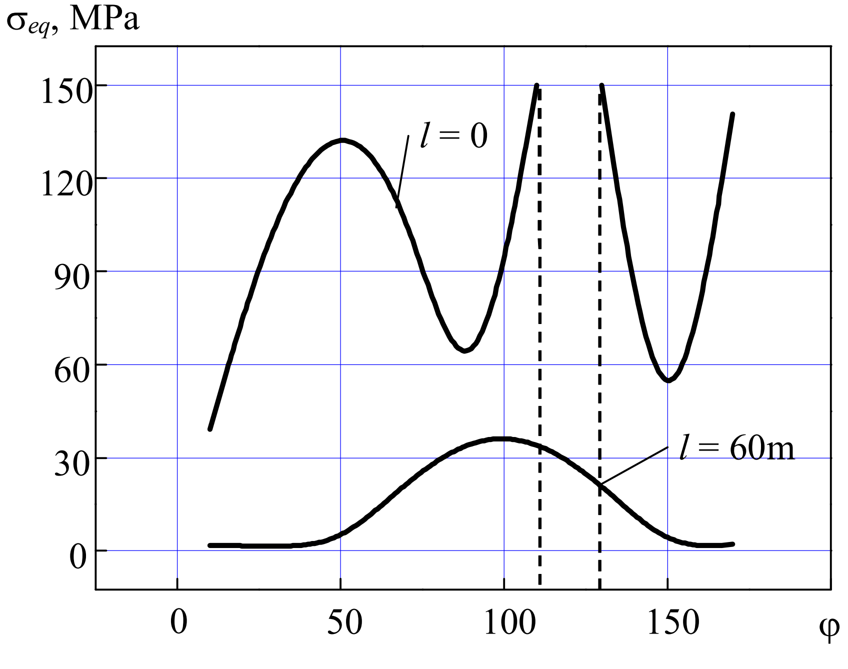


Fig. 3.7. σ_{eq} *stresses within roof of mine working along its contour if h = 5 m.*

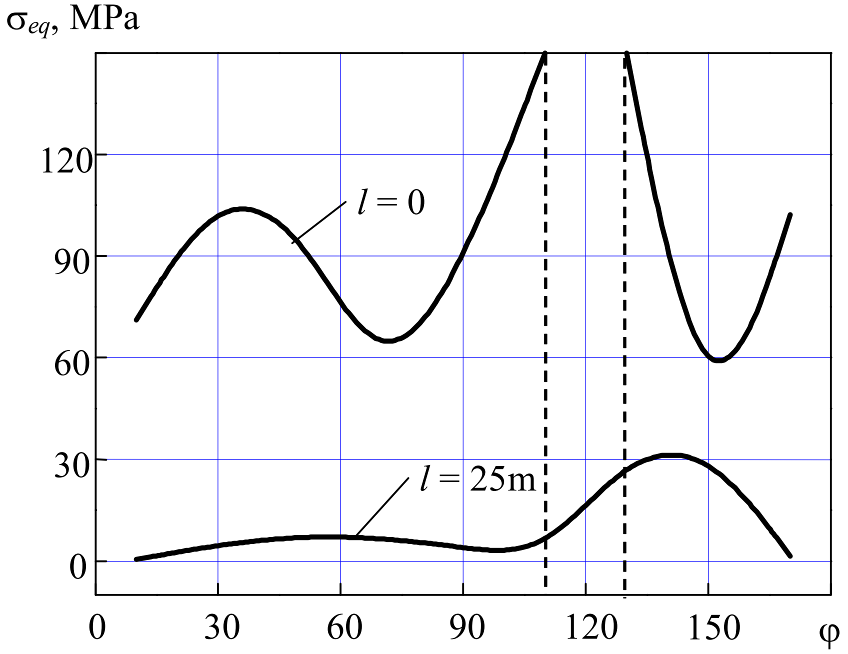


Fig. 3.8. σ_{eq} *stresses within roof of mine working along its contour if h = 15 m.*

According to stress diagrams in Fig 3.7, if interbed thickness is h = 5m and l = 0 (mine working is located under the selvage of overlying seam) then σ_{eq} stresses within roof are completely higher than rock compression strength σ_c = 30 MPa; in this context $110^0 \le \varphi \le 140^0$ area is the most stressed area. It goes without saying that such a location of development working is out-of tolerance. If however development working is located at l = 60 m distance, then working stresses are slightly higher than allowable ones.

If interbed thickness is h = 15 m and l = 0, then roof stresses σ_{eq} are higher than σ_c; in this context $110^0 \le \varphi \le 130^0$ area is the most dangerous. However, if l = 25 m, then stresses are not higher than rock compression strength.

To generalize results of the theoretical study, maximum equivalent stresses on (2.12) criterion were determined within dangerous cross-sections of mine workings for l parameter which values experienced 0 to 80 m change with 10 m increment.

As before cross-sections where surcharging coefficients were of the greatest values $\left(K_y\right)_{max}$ = 3.2 within l_1 = 5m cross-section if h = 5m, and $\left(K_y\right)_{max}$ = 2.7 within l_1 = 10m cross-section if h = 15 m) were considered as the most dangerous ones.

All the calculations relied on H = 300m depth; moreover, they were limited to t = 16 months.

The data were analyzed statistically. As a result following correlation dependences of maximum equivalent stresses $(\sigma_{eq})_{max}$ on l parameter have been determined:

<u>within floor</u>

if h = 5 m

$$\left(\sigma_{eq}\right)_{\max} = 63.675 - 3.075\sqrt{l} - 0.003l^2 \text{, MPa;} \quad (3.2)$$

if h = 15 m

$$\left(\sigma_{eq}\right)_{\max} = 25.163 - 2.126\sqrt{l} + 0.001l^2 \text{, MPa;} \quad (3.3)$$

<u>within roof</u>

if h = 5 m

$$\left(\sigma_{eq}\right)_{\max} = 177.974 - 21.662\sqrt{l} + 0.007l^2 \text{, MPa;} \quad (3.4)$$

if h = 15 m

$$\left(\sigma_{eq}\right)_{\max} = 128.010 - 15.966\sqrt{l} + 0.004l^2 \text{, MPa;} \quad (3.5)$$

Table 3.1 demonstrates relevant coefficients

Table 3.1

Indices of high correlation ratio between $(\sigma_{eq})_{max}$ and l

Correlation coefficient	Floor		Roof	
	h = 5m	h = 15m	h = 5m	h = 15m
R	0.83	0.97	0.93	0.96

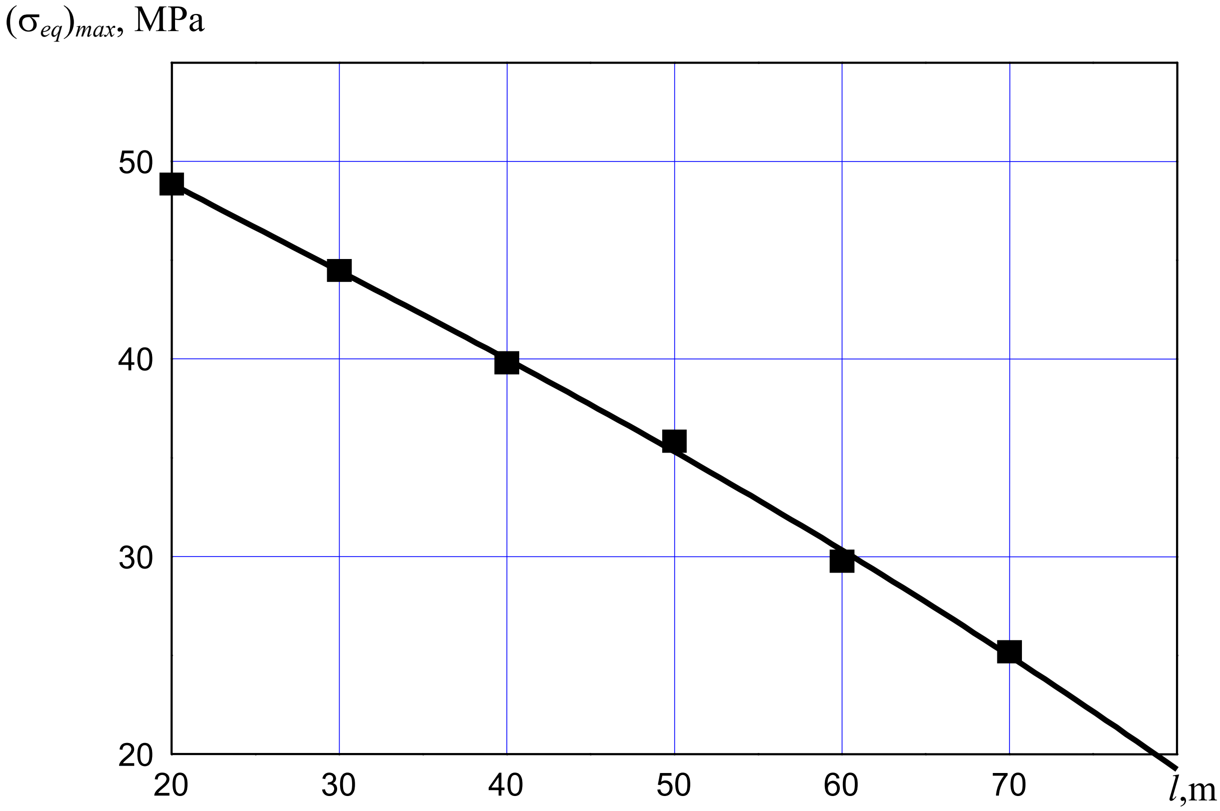
Points in Figures 3.9 and 3.10 show calculation values of each variation. Lines are curves plotted on (3.2), (3.3), (3.4), and (3.5) dependences.

According to stress diagrams $(\sigma_{eq})_{max}$ within floor, in the context of P.P. Balandin criterion (2.12) and strength reserve K = 1.2 distance to mine working selvage from overlying seam selvage should be l = 70 m if h = 5 m; if however h = 15 m then stresses within any l are much less than compression strength; hence, they are not determinant.

On the assumption of rock stresses if h = 5 m then l should not be less than 80 m; if h = 15 m then allowable value is l = 25 m.

Thus, depending upon stress calculations, $l \geq 80$ m if h = 5 m and $l \geq 25$ m if h = 15m are reasonable values for mining and geological conditions under consideration. However, analysis of values of maximum displacements within a contour of concerned mine working should also be involved. The problem is discussed in paper [129].

a)



b)

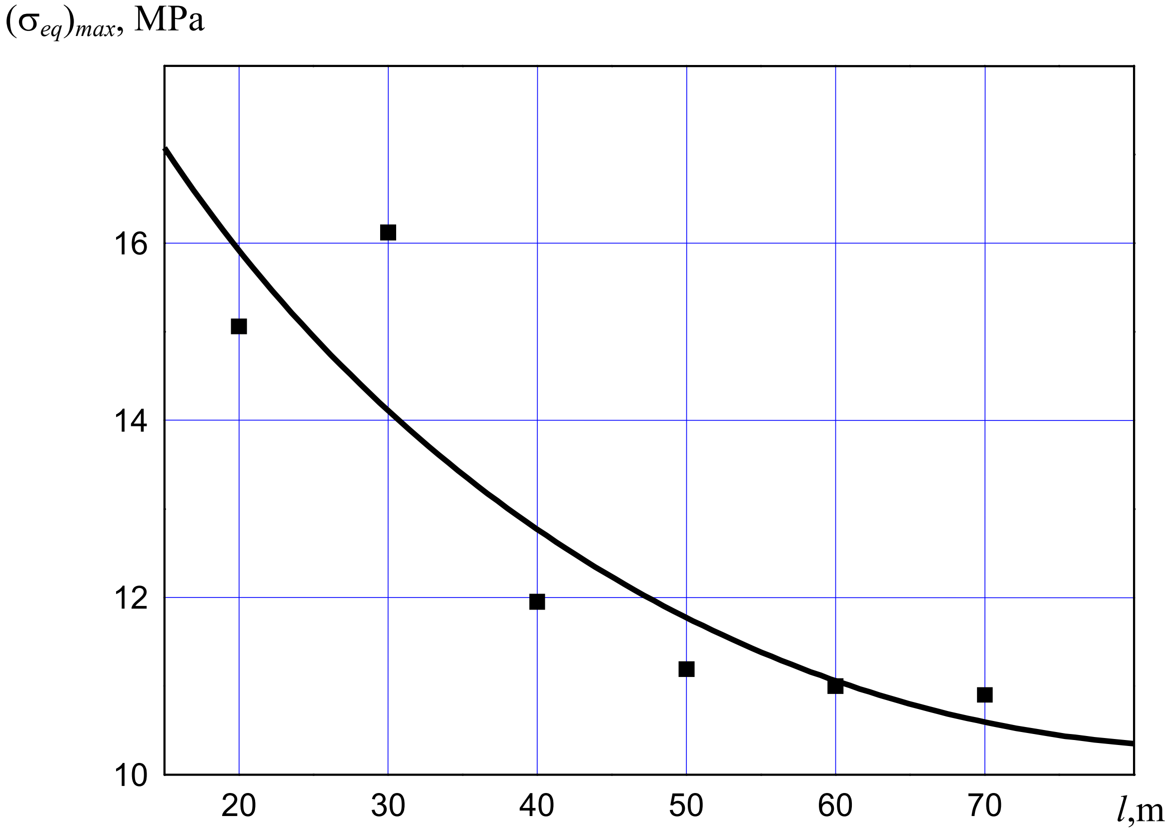
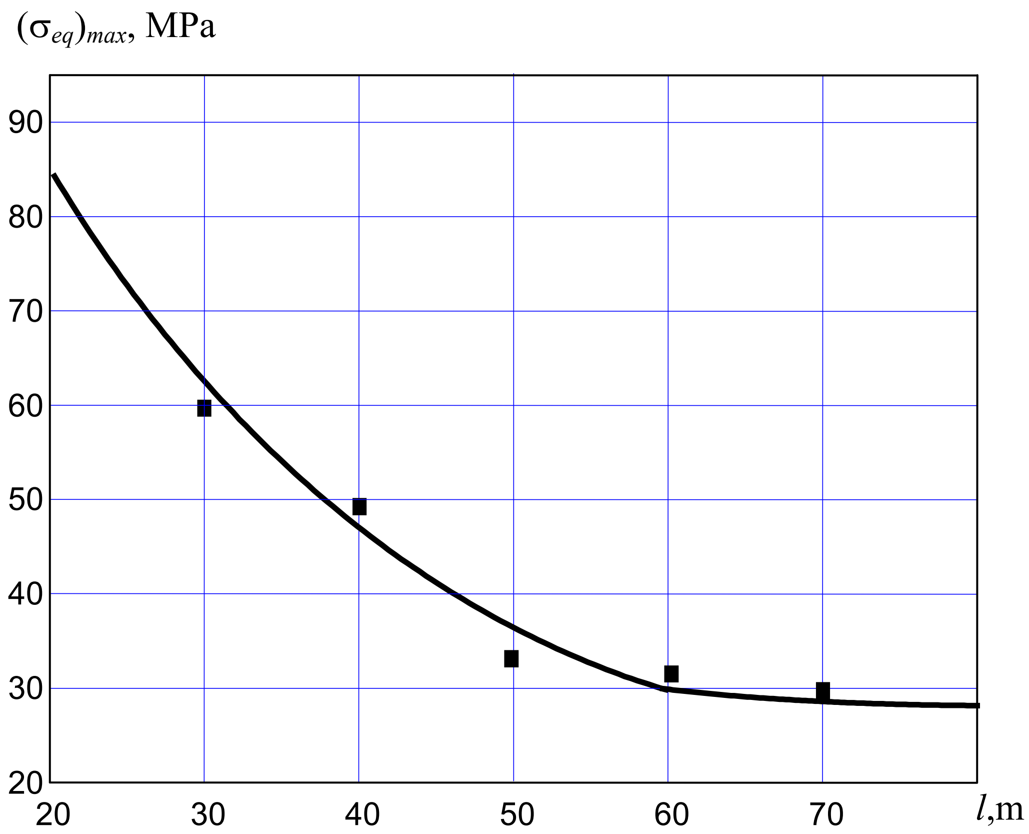


Fig. 3.9. Maximum equivalent stresses within floor
a) h = 5m; b) h = 15m.

a)



b)

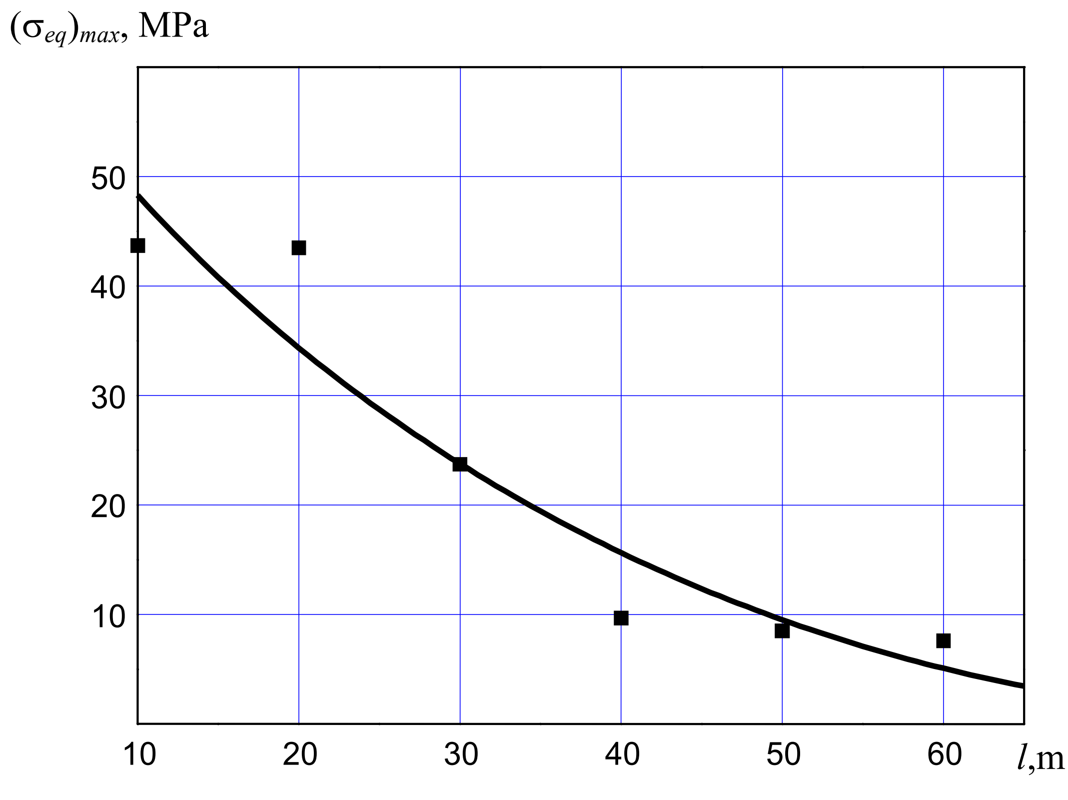


Fig. 3.10. Maximum equivalent stresses within roof
a) h = 5 m;   b) h = 15 m.

Displacement calculations within mine working contour have taken two stages. First they were performed for sizeable area covered with analytical model in Fig. 2.1. Then small part of rock mass located within neighbourhood of the mine working has been considered. Stresses, obtained in stage one, were specified as active load within its external boundaries. Dimensions of small neighbourhood were as follows: 45m horizontally and 7 m vertically. The mine working was placed symmetrically to vertical axis. Distance from floor to lower edge of the area was 2 m; distance to upper edge was 5 m.

Smaller increment of boundary segmentation to compare with solution concerning large area was applied. Calculations involved *l* values varying from 0 to 80 m with 10 m increment.

Table 3.2 demonstrates obtained values of roof and floor displacements for each calculation procedure.

Table 3.2.

Roof and floor displacements

h, m	l, m	$U_п$, mm	$U_к$, mm	Δu, mm	h, m	l, m	$u_в$, mm	$u_н$, mm	Δu, mm
5	0	174.8	248.5	423.3	15	0	74.3	105.6	179.9
	10	179.7	256.3	436.0		10	88.2	125.4	213.6
	20	137.1	194.9	332.0		20	65.3	92.8	158.1
	30	122.0	173.7	295.7		30	59.1	84.0	143.1
	40	115.5	164.1	279.5		40	55.8	79.3	135.1
	50	107.9	153.4	261.3		50	52.2	74.2	126.4
	60	94.6	134.5	229.1		60	51.1	69.0	120.1
	70	89.2	108.8	198.0		70	50.1	63.0	113.1
	80	87.1	105.2	192.3		80	49.8	62.9	112.7

The results involving statistic analysis made it possible to determine correlation dependences relating convergence of roof and floor $(\Delta u_в)_{max}$ with *l* distance. They are:

for $h = 5$ m

$$(\Delta u)_{\max} = 477.07 + 20.21\sqrt{l} - 8.81l + 0.05l^2 \text{, mm;} \tag{3.6}$$

for $h = 15$ m

$$(\Delta u)_{\max} = 182.61 + 20.39\sqrt{l} - 5.53l + 0.03l^2 \text{, mm.} \tag{3.7}$$

Correlation coefficients of the dependences are 0.91 and 0.96, respectively. Figures 3.11 and 3.12 show curves obtained on (3.6) and (3.7) dependences. In these very Figures, points specify results of relevant calculation variants.

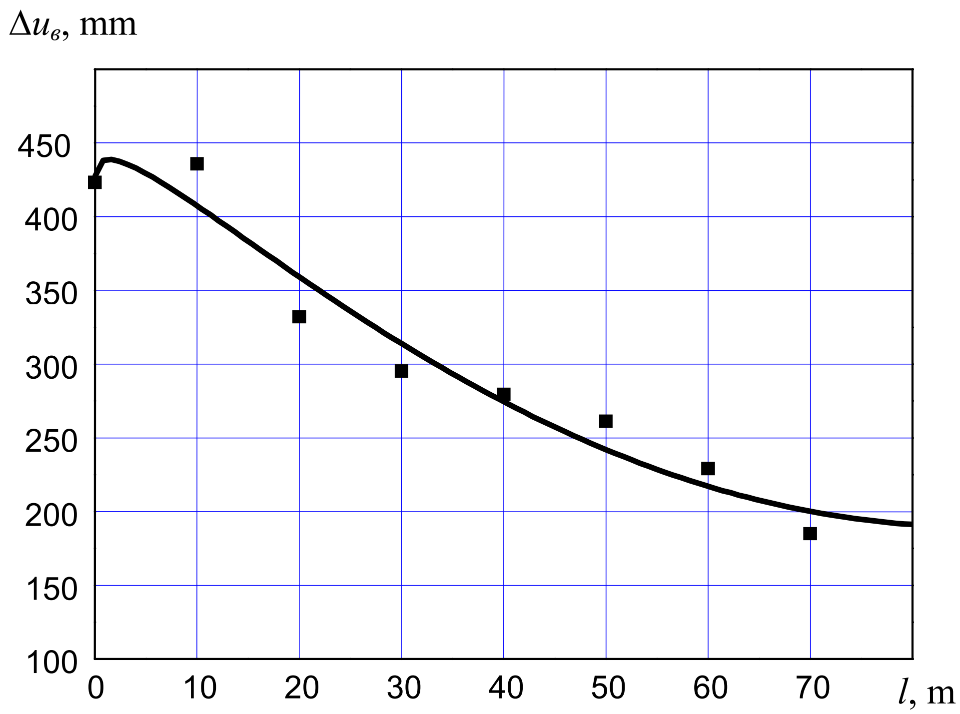


Fig. 3.11 Floor and roof convergences
H = 300 m; t = 16 months; l_1 = 5 m; h = 5 m.

It follows from Figures 3.11 and 3.12 that the greatest displacements take place when l = 5 m. If in this context interbed thickness is h = 5 m then convergence reaches 440 mm while allowable value is 300 mm. If l = 40 m and h = 5 m then $\Delta u_в$ is <300 m; that is in this case l should not be less than 40 m.

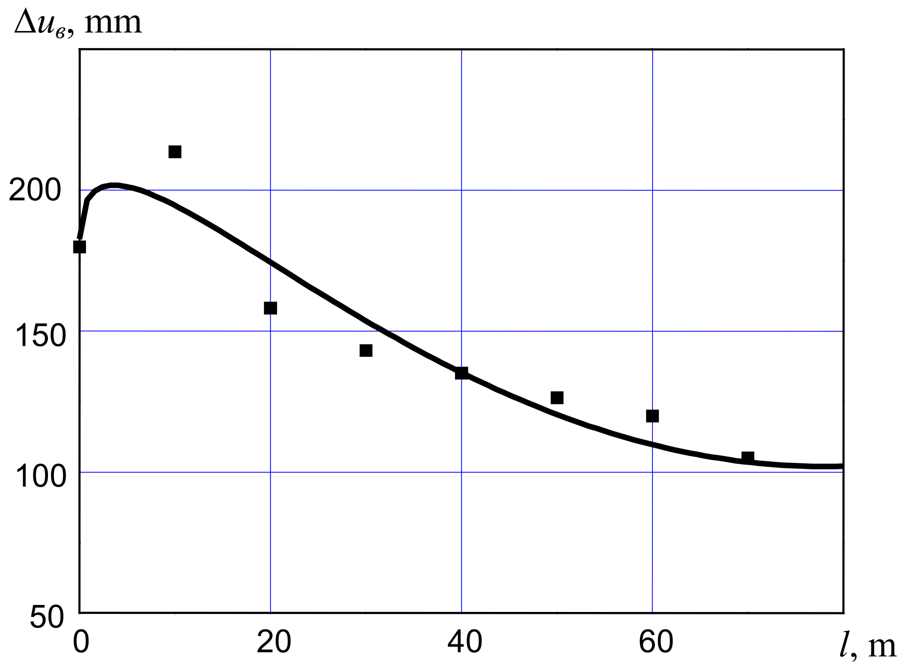


Fig. 3.12 Floor and roof convergence
H = 300m; t = 16 months; l_1 = 10m; h = 15m.

If interbed thickness is h = 15m then convergence within dangerous cross-section is not more than allowable value. Calculations on (3.5) and (3.6) formulas have identical results.

Therefore, in both cases (when h = 5 m and h = 15 m) stress calculation is determinant while determining reasonable distance. Accordingly, unknown distance l should not be less than 60 m if h = 5 m, and no less than 25 m if h = 15 m. In this context, floor and roof convergence values are not more than allowable ones being 230 m and 147 mm.

3.4 Recommended operation schedules

Substantiation of safe boundaries of mining operations makes it possible to develop two operation schedules to mine contiguous seams in the context of pillar mining technique and combined one. In terms of the schedules, boundary location of contours of stopes and development workings involves defined dynamics of rock pressure (Figures 3.13 and 3.14).

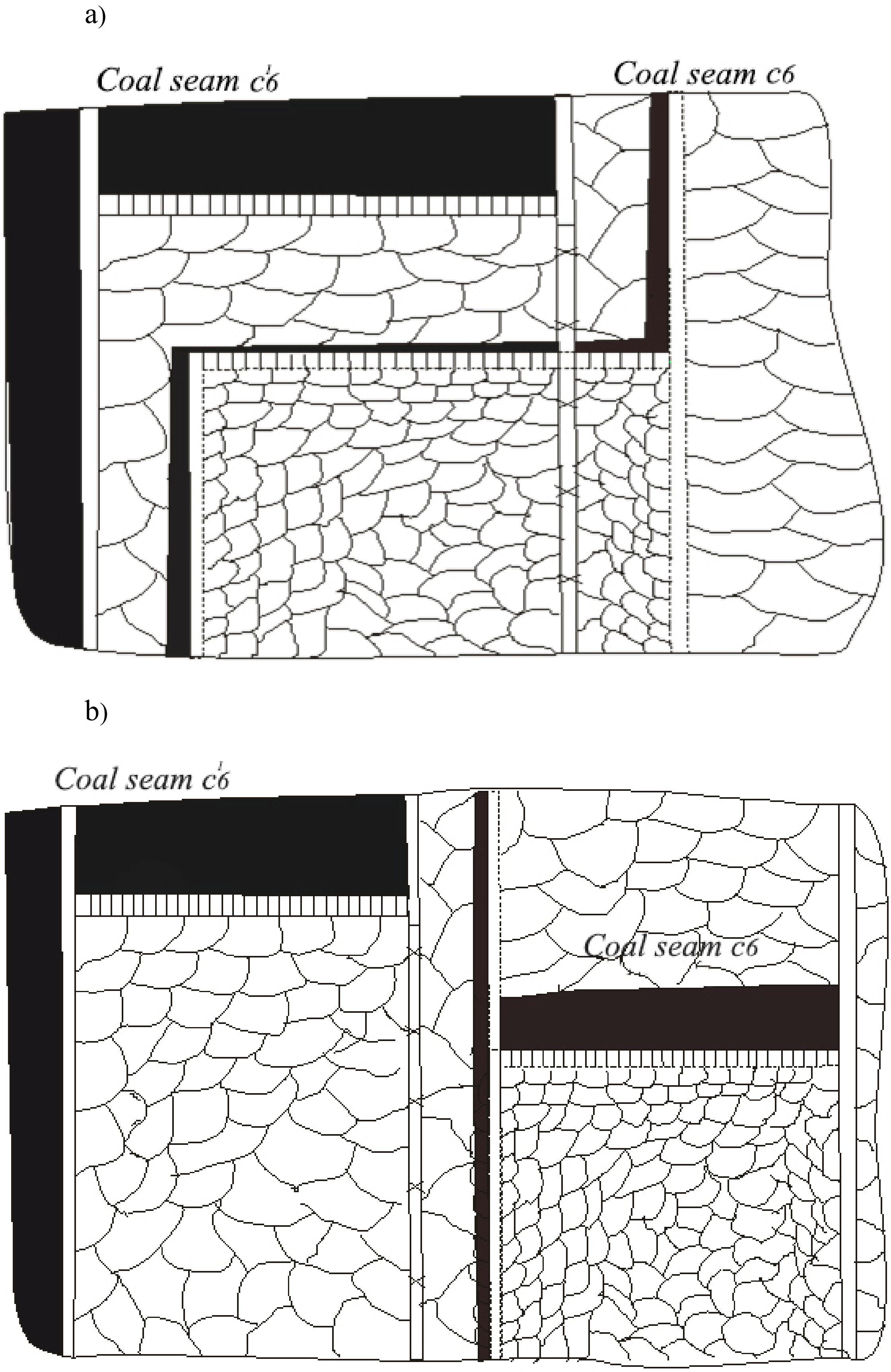


Fig. 3.13. Operation schedules to mine contiguous seams in terms of concentration of mining operations
a) combined mining technique;
b) pillar mining technique.

In this context l values being minimum allowable distances from a site lower seam are determined on abovementioned technique, and L being minimum allowable distance between stopes in the context of two contiguous seams mining is determined on a technique described in section two. Recommended lag of lower longwall face is:

$$L_p = L + \frac{l_n^m}{2},$$

where l_n^m is monthly advance of mining operations, m/month.

Conclusions

1. Numerical procedure of boundary elements has been applied for quantitative evaluation of overlying seam longwall effect on development working driven within lower seam. Dependence diagram of maximum surcharge roof coefficient in relation to γH level on distance between stope and cross-section of mine working has been developed. The determined surcharge coefficients are meant for their use as initial data while analyzing stability of long overworked mine working.

2. In the context of involved mining and geological conditions, P.P. Balandin criterion has been applied on the basis of linear creep model with Abel kernel to determine correlation ratios between maximum equivalent stresses within roof and floor of overworked mine working and distance from cross section of the latter to overlying seam selvage. In the process of the dependences obtaining, effect of mining operations in the context of overlying seam was involved with the help of surcharging coefficients.

3. Correlation ratios have been identified between maximum roof and floor convergence of mine working located within overworked zone and distance from its cross section to overlying seam selvage.

4. It has been determined that concentration of mining operations within contiguous seams should involve equivalent stresses within rock mass as well as maximum allowable rock displacements in terms of contour of mine working.

In the context of concerned conditions, boundaries of safe and expedient construction of development workings in terms of lower overworked mine workings distances to overlying seam selvage are: 60m if interbed thickness is h = 5 m, and 25 m if h = 15 m; decrease of the distances down to zero results in total loss of cross-sections of mine workings with fivefold stress excess in roof over rock strength in the context of uniaxial compression.

CHAPTER 4. RESULTS OF UNDERGROUND INVESTIGATIONS OF ROCK PRESSURE IN THE CONTEXT OF CONTIGOUOS SEAMS MINING

To verify basic theoretical provisions and obtain actual values of involved parameters of rock pressure manifestations within stopes and development workings in the context of contiguous seams a number of field studies have been carried out.

The field studies took place from 2000 through 2004 in *Stepnaia* mine (DTEK *Pavlogradugol*) under the supervision of the authors and with their direct participation. The studies have resulted in determination of actual values of enclosing roof and floor convergence within longwalls and adjoining gate roads and power parameters of support; nature of enclosing roof and floor has been analyzed; parameters of index zones while mining as well as their effect on contiguous mine workings in the process of contiguous seams mining have been identified.

The underground investigations were carried out in accordance with specific technique being self-standing document involving principles of established industrial procedures [116, 121,122].

4.1 Characteristic of the research conditions

The research was carried out within stopes and development workings in the context of contiguous seams mining under typical for the region mining and geological conditions. Tables 4.1 and 4.2 contain characteristic of the research site. Fig. 4.1 demonstrates summary lithological column on contiguous seams C_6 and $C_6^{1.}$ Table 4.3 includes characteristics, and physical and mechanical properties of lithological members.

Data in the Figure and Tables are specified by mining operations in the process of extraction pillars mining.

In the context of *Stepnaia* mine, C_6 and C_6^1 seams are extracted by means of double longwalls equipped with КМК-97М powered system consisting of powered support МК-98, coal shearer МК-67, and conveyor СП-250. Level-oriented development technique is applied. Coal is mined with the help of longwalls (mainly double) to the rise. Figures 4.2 and 4.3 demonstrate excerptions from the plans of mine workings in terms of C_6 and C_6^1 seams; Fig. 4.4 illustrates integrated operating plan.

Coal seams are mined in descending order with advance C_6^1 seam mining. Mining depth is 220 to 380m. Boundary entries are reused. To do that they are supported after longwall pass; they are retimbered if required.

4.2. Results of underground investigations within longwalls

4.2.1. Roof rock state

Estimation of rock state within longwalls involved: processes of fissure formation, inrush formation, and roof rock breaking. Availability of goaf, overworking, as well as interbed thickness was taken into consideration.

Fissure formation was studied visually; opening width of fissures was measured with the help of a ruler. Table 4.4 contains the research results.

Table 4.1

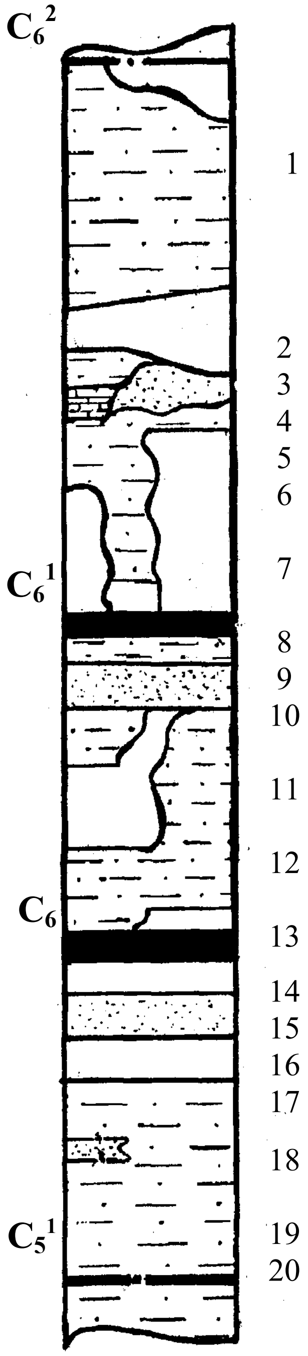
Characteristic of the research area. Stopes.

No	Mine, longwall, seam	Longwall length, m	Mined seam thickness, m	Powering type	Availability of overworking	Roof	Floor
1	*Stepnaia*; 215; c_6^1	200	0.87	МК – 67, МК – 98, СП - 250	Nonavailable	Layered argillite with 1.1 to 1.5 m thickness	Undulated and layered aleurite with 1.1 to 1.3 m thickness
2	*Stepnaia* ; 213; c_6^1	200	0.87	МК – 67, МК – 98, СП - 250			
3	*Stepnaia* ; 211; c_6^1	200	0.9	МК – 67, МК – 98, СП - 250			
4	*Stepnaia*; 217-bis; c_6^1	200	0.9	МК – 67, МК – 98, СП - 250			
5	*Stepnaia*; 215-bis; c_6^1	193	0.89	МК – 67, МК – 98, СП - 250			
6	*Stepnaia*; 213-bis;, c_6^1	200	0.88	МК – 67, МК – 98, СП - 250			
7	*Stepnaia* ; 119; c_6	200	0.95	МК – 67, МК – 98, СП - 250	Overworked	Undulated and layered aleurite with 1.1 to 1.3 m thickness	Lump-structure argillite with 1.1 to 1.3 m thickness
8	*Stepnaia* ; 117; c_6	175	0.97	МК – 67, МК – 98, СП - 250	Overworked		
9	*Stepnaia*, 117-bis, c_6	175	0.95	МК – 67, МК – 98, СП - 250	Overworked		
10	*Stepnaia* ; 106; c_6	200	0.83	МК – 67, МК – 98, СП - 250	Nonavailable		
11	*Stepnaia* ; 110; c_6	200	0.93	МК – 67, МК – 98, СП - 250	Nonavailable		

Table 4.2

Characteristic of the research area. Development workings.

<table>
<tr><th>No.</th><th>Mine, longwall, seam</th><th>Length of mine working, m</th><th>Cross-section of mine working, m^2</th><th>Period</th><th>Site</th><th>Effect of mining operations</th><th>Floor</th><th>Roof</th></tr>
<tr><td>1</td><td>Stepnaia; 215-bis;.c_6^1</td><td>1030</td><td>9.5</td><td></td><td>–</td><td></td><td rowspan="4">Undulated and layered aleurite with 1.1 to 1.3m thickness</td><td rowspan="4">Layered argillite with 1.1 to 1.5m thickness</td></tr>
<tr><td>2</td><td>Stepnaia; 215; c_6^1</td><td>1360</td><td>9.5</td><td></td><td>–</td><td></td></tr>
<tr><td>3</td><td>Stepnaia; 211; c_6^1</td><td>1340</td><td>9.5</td><td>03.1995 to 02.1996</td><td>–</td><td></td></tr>
<tr><td>4</td><td>Stepnaia; 219; c_6^1</td><td>1275</td><td>9.5</td><td></td><td>–</td><td></td></tr>
<tr><td>5</td><td>Stepnaia; 119; c_6</td><td>940</td><td>9.5</td><td></td><td>Under 219</td><td></td><td rowspan="6">Lumpy-structure argillite with 1.1 to 1.3m thickness</td><td rowspan="6">Undulated and layered aleurite with 1.1 to 1.3m thickness</td></tr>
<tr><td>6</td><td>Stepnaia; 117; c_6</td><td>1305</td><td>9.5</td><td>09.2000 to 04.2002</td><td>Under 217</td><td></td></tr>
<tr><td>7</td><td>Stepnaia;115; c_6</td><td>1365</td><td>9.5</td><td>06.2001 to 04.2002</td><td>15m under worked-out area</td><td>Effect of mining operations of longwall 215, c_6^1 seam</td></tr>
<tr><td>8</td><td>Stepnaia; 111; c_6</td><td>1220</td><td>9.5</td><td>08.2002 to 04.2003</td><td>40m under worked-out area.</td><td>Effect of mining operations of longwall 211, c_6^1 seam</td></tr>
<tr><td>9</td><td>Stepnaia; 115-bis; c_6</td><td>1015</td><td>9.5</td><td></td><td>25m under worked-out area.</td><td>Effect of mining operations of longwall 215-bis, c_6^1 seam</td></tr>
<tr><td>10</td><td>Stepnaia;111-bis; c_6</td><td>1095</td><td>9.5</td><td></td><td>40m under worked-out area.</td><td>Effect of mining operations of longwall 211-bis, c_6^1 seam</td></tr>
</table>



1. Horizontally layered aleurite, m=8.2–8.6m; f = 3–4

2. Argillite, m=1.7–2.6m; f=2–3
3. Aleurite, m=0–2.0m; f=3–4
4. Arenite, m=0.9–1,9m; f=5–6
5. Limestone, m=0–0.9m; f=7–8
6. Undulated and layered aleurite, m=0–4.9m; f=3

7. Argillite m=1.1–6.1m; f = 2–2.5

8. Coal seam C_6^1 m=0.55–0.69m; f=3.5
9. Nonlayered, lumped aleurite m=1.1–1.3m; f=3–4
10 Quartz arenite m=1.3–4.0m; f=5–6
11 Aleurite m=0–1.9m; f=3.0–3.5

12. Horizontally layered argillite m=0–5.3m; f=2.0–2.25
13. Undulated and layered aleurite m=2.5–3.7m; f=3–4

14. Argillite m=2.6–4.0m; f=2–2.5
15. Coal seam C_6 m=0.74–1.05m; f=3.5
16. Aleurolitic argillite m=1.1–1.3m; f=2.5–3
17. Quartz arenite m=1–3.2m; f=6–7
18. Horizontally layered argillite m=1.5–1.1м; f=2–3
19. Aleurolite m=2.5–4.5м; f=3–4
20. Arenite m=0–3.2м; f=5–6

21. Coal seam C_5^1 of non-operational thickness m=0.74–1.05m;

22. Aleurite m=2.2–3.0m; f=3–3.5

Symbols:

Fig. 4.1 Summary lithological column on C_6 and C_6^1 coal seams mined by Stepnaia mine

Table 4.3

Physical and mechanical properties of rocks enclosing C_6 and C_6^1 coal seams

No, of layer in Fig. 4.1	Standard seam thickness, m	Description of rocks	Caving, stability	Physical and mechanical properties							
				$\sigma_{сж}$, kg/cm^2	$\sigma_{раст}$, kg/cm^2	W, %	Q, g/cm^3	Fragmentation degree	Soaking	Fissuring, f/m	Tendency to heaving
1	2	3	4	5	6	7	8	9	10	11	12
		Rocks of main roof of C_6^1 seam									
1	8.2-8.6	Gray micaceous aleurite foliated owing to bands of arenite (argillite rarely)	A_2	230	12	–	–	–	–	–	–
2	1.7-2.6	Aleurite layered horizontally owing to aleurite partings	A_2	240	20	–	–	–	–	–	–
3	0-2.0	Aleurite layered owing to aleurite partings and arenite lenticules	A_2	220	12	–	–	–	–	–	–
4	0.9-1.9	Quartzy finely grained quar on calcareous and siliceous cement	A_3	450-650	50	5.6	2,70	1.8	No	1-2	–
5	0-0.9	Cryptocrystalline limestone with reliquiae	A_3	500-750	–	–	–	–	–	–	–
6	0-4.9	Undulated and layered aleurite owing to commonly occurring lenticules of quartzy arenite (argillite rarely)	A_2	220	12	3.0	2.72	1.8	During 24 hours	1-2	–
7	1.1-6.1	Argillite layered horizontally owing to aleurite partings (argillaceous siderite rarely)	A_2	230	20	19	2.67	1.7	During 12 hours	2-4	–

7	1.1-1.5	<u>Rocks of immediate roof of C_6^1 seam</u> Layered argillite. Contact with coal seam is sharp; adhesion is either weak or nonavailable. Within zones of tectonic faults and nonelastic strains it is heavy fractured; false roof	$А^2$, $Б^2$ $А_1$, $Б_1$	200 50	20 2	2.25 2.5	2.67 2.67	1.7 1.7	During 12 hours	4-5 5-10	– –
8	0.5-0.69	C_6^1 seam is hard finely banded coal of two-patch structure with sandstone band and slipping plane; fractured. It has been mined out over extraction pillars of longwalls 117 and 119		350	3.68	3.68	1.24	1.8	–	10-15	–
9	1.1-1.3	<u>Rocks of C_6^1seam direct roof and C_6 seam main roof</u>. Within roof, aleurite is of lumpy composition; below it is undulated and layered owing to sandstone bands. It is fractured; prone to heaving and soaking	$П_2$	200	20	2.42	2.70	1.8	During 12 hours	4-5	Heaving
10	1.3-1.5	<u>Rocks of C_6^1 seam main floor and C_6 seam main roof.</u> Quartzy layered fine-grained arenite; its properties depend on interlayering with aleurite on sandy-clayed cement.	$П_2$	350-550	45	4.0	2.71	1.8	During 48 hours	1-2	Heaving
11	0-1.9	Aleurite undulated and layered owing interlayers of arenite (argillite rarely); adhesion is weak.	$П_2$	230	20	2.0	2.70	1.8	During 24 hours	1-2	Heaving
12	0-5.3	Layered fractured argillite with interlayers of micaceous aleurite. Contact is sharp; adhesion is weak.	$П_2$	240	25	2.75	2.70	1.7	During 12 hours	1-2	Heavily heaving
13	0-3.7	<u>Rocks of C_6 immediate roof.</u> Undulated and layered aleurite with arenite interlayers; here and there it is very hard. It is very stratified, fractured, and rather unstable within nonelastic strain zones; adhesion inside layers is not available.	$П_2$	230	20	2.0	2.70	1.8	During 24 hours	1-2	Heaving

14	0-0.9	Horizontally layered fractured argillite; foliation is in places. Within nonelastic strain zones and tectonic deformations it is heavily foliated, intensively fractured, rather unstable; intralayer adhesion is not available.	$П_2$	240-200	24	2.56	2.64	1.7	During 12 hours	1-2	Heavily heaving
15	0.74-1.05	Coal seam C_6 is finely banded of simple one-patch structure; argillite bends with up to 0.05m thickness and up to 3-5m extension are rare. It is watered and hard with pyrite particles. Contact with roof rocks is sharp; adhesion is not available.		350		5.9	1.24	1.8	–	10-15	–
16	1.1-1.3	Rocks of C_6 seam immediate roof Aleurolitic argillite of lumpy composition with gliding planes and phytoleims. It is intensively heaving; it swells on wetting down to clay state with complete bearing-capacity failure.	$П_2$	150-200	17	2.28	2.68	1.7	During 12 hours	4-5	Intensively heaving
17	1.0-3.2	Rocks of C_6 main floor Quartzy fine-grained horizontally layered aleurite; its features depend on interlayering with micaceous aleurite. It is compact and hard, and is placed on siliceous and clayish cement.	$П_2$	450-650	45	–	–	–	–	–	–
18	1-1.5	Horizontally layered argillite	$П_2$	230	20	–	–	–	–	–	–
19	2-3	Micaceous aleurite	$П_2$	250	23	–	–	–	–	–	–
20	0-1.2	Quartzy fine-grained arenite	$П_2$	600	40	–	–	–	–	–	–
21	0-0.6	Coal seam C_5^1		350	–	–	–	–	–	–	–
22	2.5-4	Micaceous aleurite	–	250	25	–	–	–	–	–	–

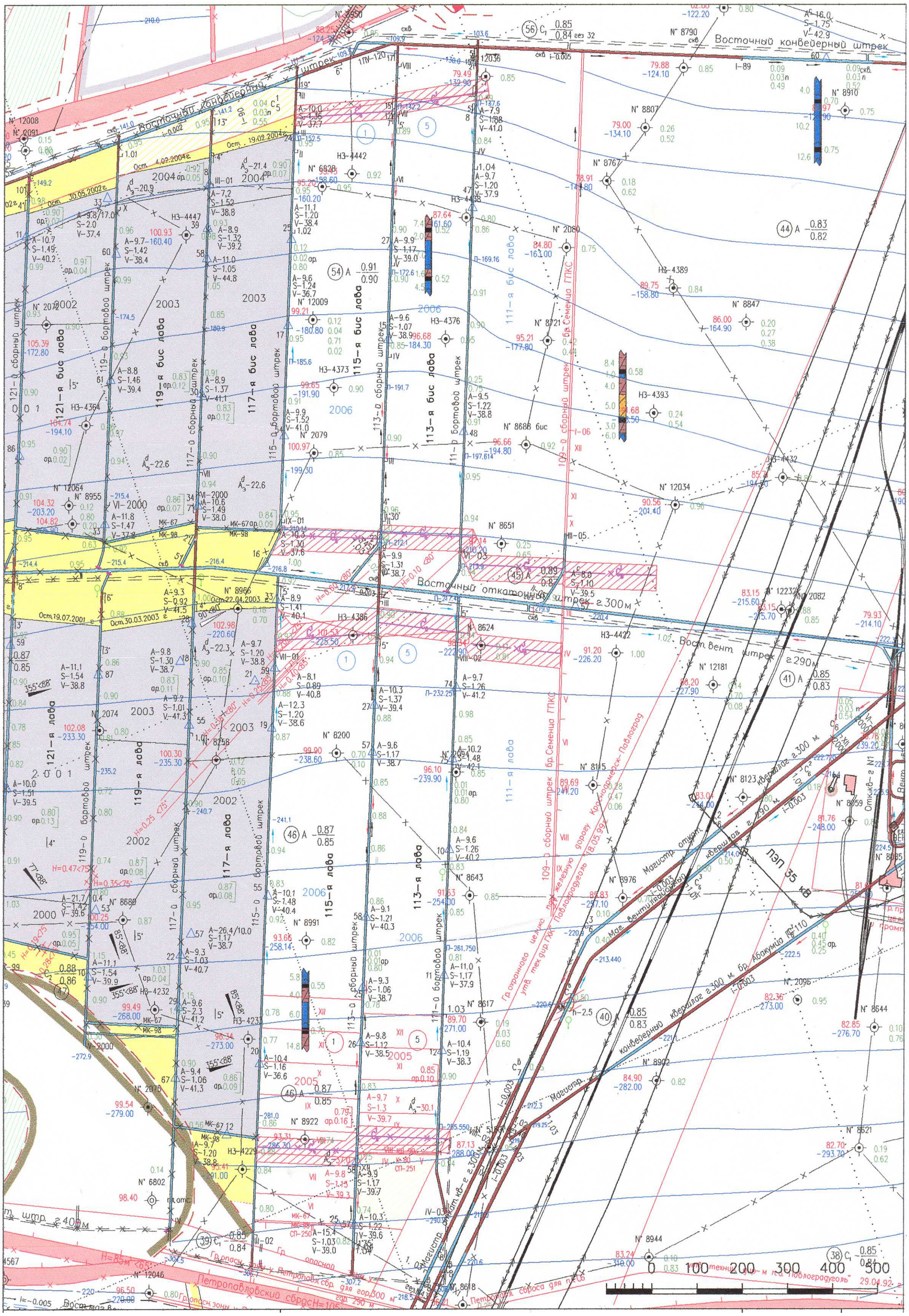


Fig. 4.2 Excerption from mining plan in the context of C_6 seam

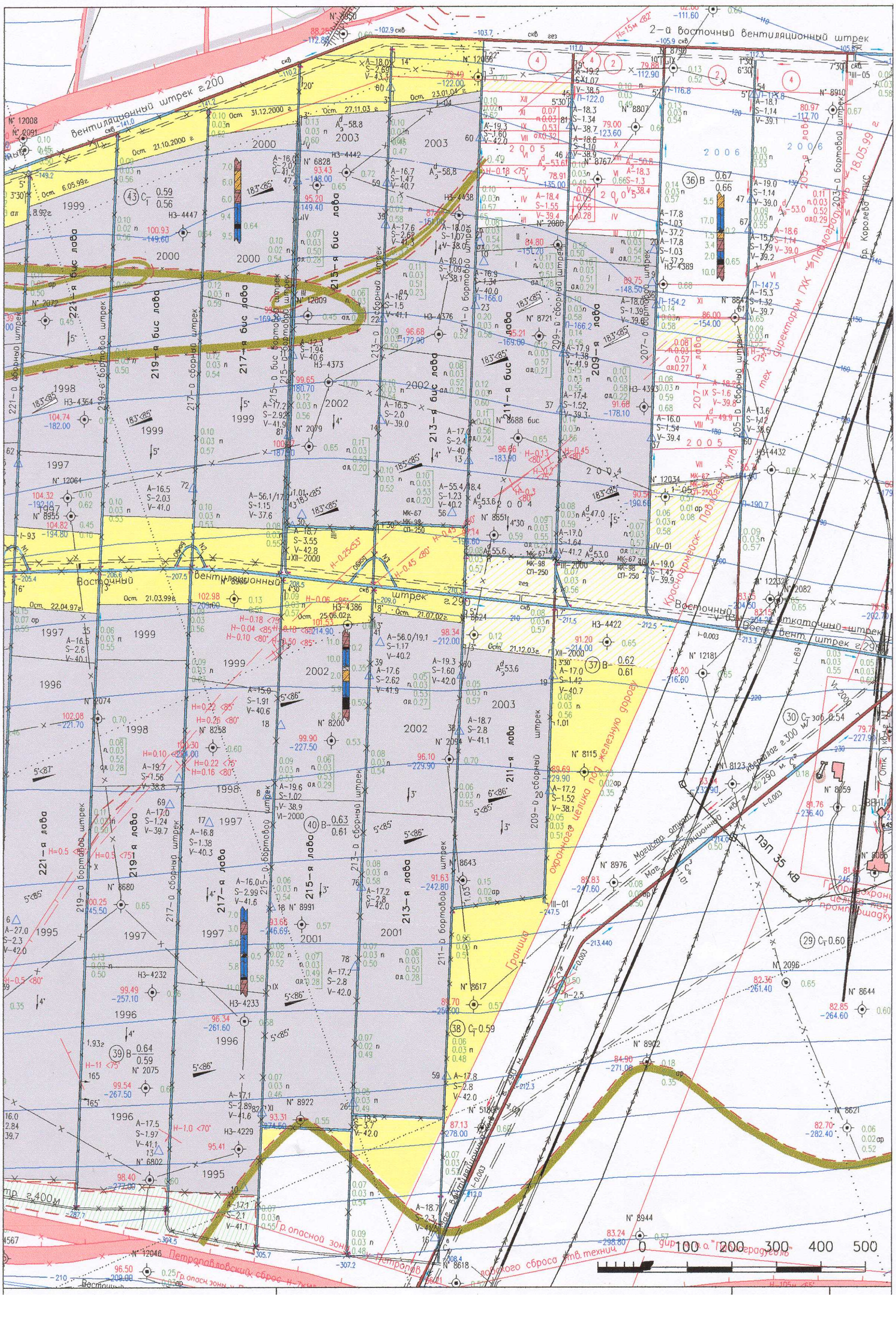


Fig. 4.3 Excerption from mining plan in the context of C_6^1 seam

Fig. 4.4 Integrated mining plan in the context of C_6 and $C_6^{\,1}$ seams

Table 4.4.

Results of research concerning roof rock braking within longwalls

No.	Mine, longwall, seam	Research period	Availability of overworking	Interbed thickness, m	of fractures per running meter of longwall	Fracture opening (mm) at the distance from a stope, m					Distance from a stope to area of roof rock breaking, m	Thickness of rock cushion on a floor of support, cm
						0	0.8	1.6	2.4	3.2		
1	*Stepnaia*; 215; c_6^1	08.09.01-29.10.01	nonavailable	11	2-3	–	1-2	2-3	5-6	7-8	4	–
2	*Stepnaia*; 213; c_6^1	14.07.01-27.08.01	nonavailable	10	2-3	–	1-2	2-3	6-7	8-9	3.8	2
3	*Stepnaia*; 211; c_6^1	06.08.03-26.09.03	nonavailable	11	2-3	–	1-2	2-3	5-6	7-8	3.9	–
4	*Stepnaia*; 217-bis; c_6^1	20.06.00-24.08.00	nonavailable	11	3-4	–	1-2	2-3	5-6	7-8	4.1	3
5	*Stepnaia*; 215-bis; c_6^1	12.03.03-04.05.03	nonavailable	11	3-4	–	1-2	2-3	6-7	8-9	3.9	1
6	*Stepnaia*; 213-bis; c_6^1	11.05.03-03.07.03	nonavailable	11	2-3	–	1-2	2-3	5-6	8-9	3.8	–
7	*Stepnaia*; 119; c_6	15.07.02-28.08.02	overworked	11	11-14	2-3	5-6	12-15	broken	broken	1.9	8
8	*Stepnaia*; 117; c_6	10.09.02-22.10.02	overworked	11	10-12	2-3	5-6	12-15	broken	broken	2.0	7
9	*Stepnaia*; 117-bis; c_6	02.10.03-17.11.03	overworked	11	12-14	2-3	5-6	12-15	broken	broken	1.9	5
10	*Stepnaia*; 106; c_6	23.10.02-18.12.02	nonavailable	10	3-4	–	1-2	2-3	5-6	7-8	3.5	-
11	*Stepnaia*; 110; c_6	15.03.04-03.05.04	nonavailable	11	3-4	–	1-2	2-3	5-6	8-9	3.7	2

It follows from the Table data that state of roof rocks within C_6^1 seam longwalls is much better to compare with C_6^1 seam roof rocks. Excessive fracturing within C_6^1 seam roof rocks was registered within areas of mining-and-geological disturbances. However, intensity of fracture formation was largely two or three fractures per running meter of longwall. Operation fractures formed mainly in the process of seam mining and strongly marked within C_6 seam longwalls demonstrate minor openings within C_6^1 seam roof.

In this context no roof displacements were observed along the seam edge. Fig. 4.5 explains nature of roof break within longwalls of the considered seams.

a)

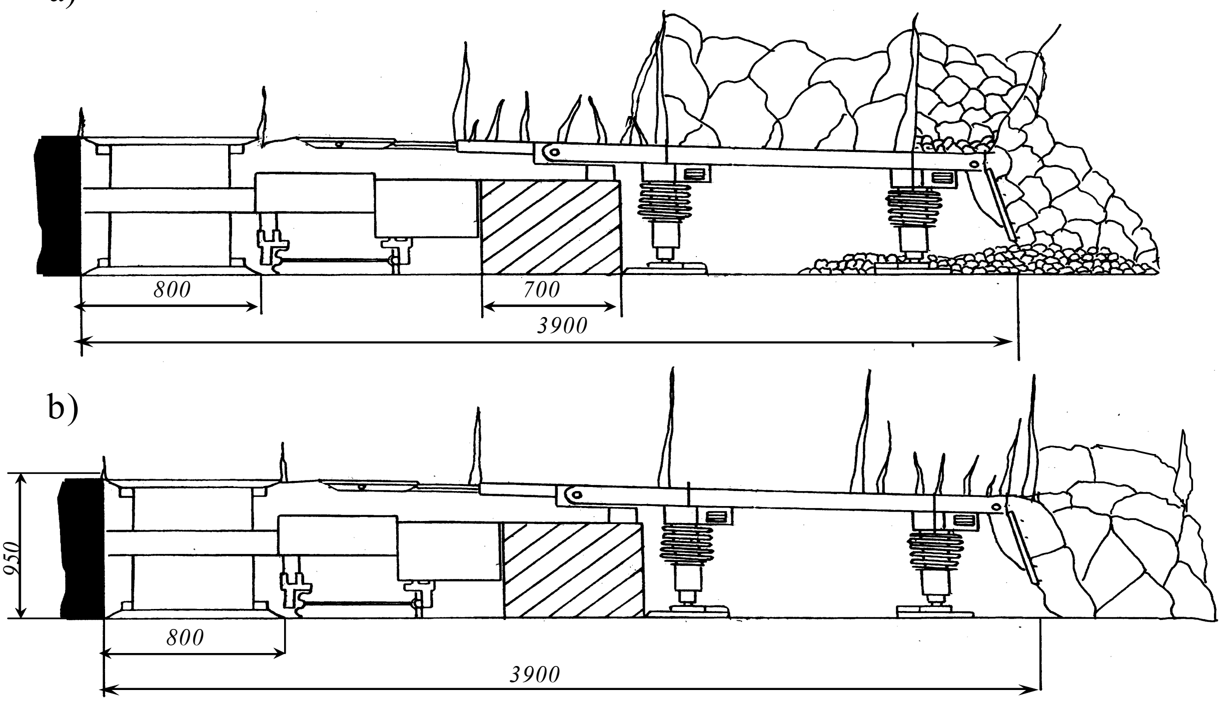


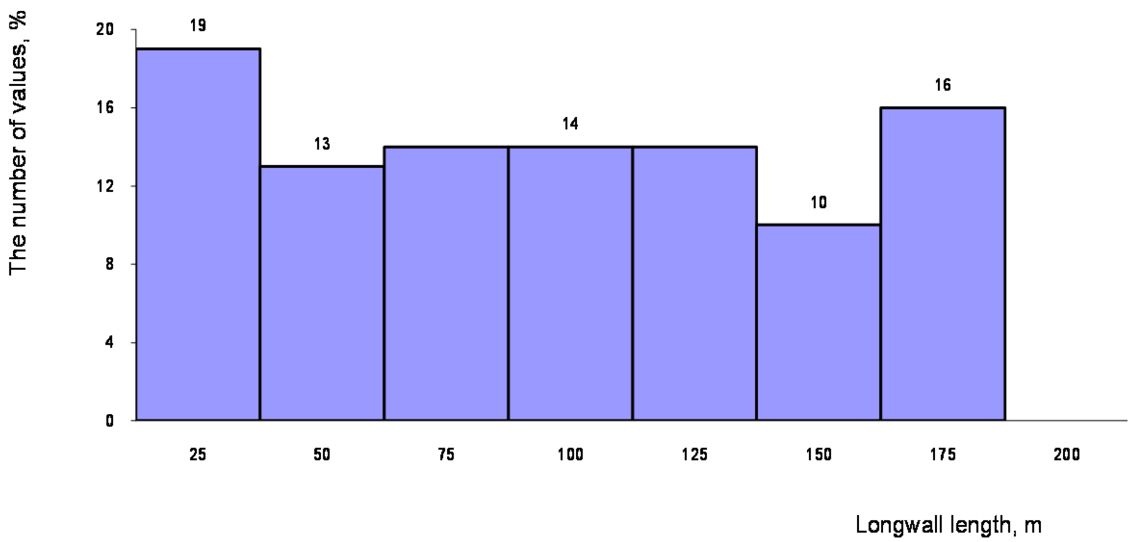
Fig. 4.5 Fracturing nature within longwalls of C_6 (a) and C_6^1 (b) seams

Roof rock breaking within C_6 seam longwalls took place either right after its daylighting or above floors of support components when in terms of overlying seam roof was mainly broken up within worked-out area of the longwall. That can be explained by the fact that both rocks enclosing C_6 seam and coal seam itself experienced high rock pressure in the process of overlying C_6^1seam. As a result, both rocks and C_6 seam itself were characterized by excessive fracturing, and had two cleavage systems instead of one (in contrast to C_6^1 seam). Fracturing intensity was six to twelve fractures per running meter; sometimes it was fifteen to eighteen fractures. Operation fractures spaced in parallel to longwall stope were formed right after coal strip extraction; distance between them was 0.8m (web width of final controlling element of coal shearer). Width of the fractures varied as follows: three-five millimeters in the neighbourhood of a stope to ten-fifteen millimeters at the distance of 1.6m from stope; after that rock broke and roof stopped to be continuous. In consequence of rock break, rock cushions with five to ten centimeters occurred on a support floor; cascading into working area took place. The process hampered advance of miners within longwall thus decreasing limited operating space.

Table 4.5 demonstrates results of inrush processes within longwalls where contiguous seams are mined.

According to the results of registration of the number of inrushes and measurement of their sizes within longwalls 117 in terms of C_6 seam (overworked) and 215 in terms of C_6^1 seam of *Stepnaia* mine for the whole period of their operation, histograms concerning distribution of percent-denominated amount of inrushes in the context of length of working area of longwalls have been constructed (Fig. 4.6).

a)



b)

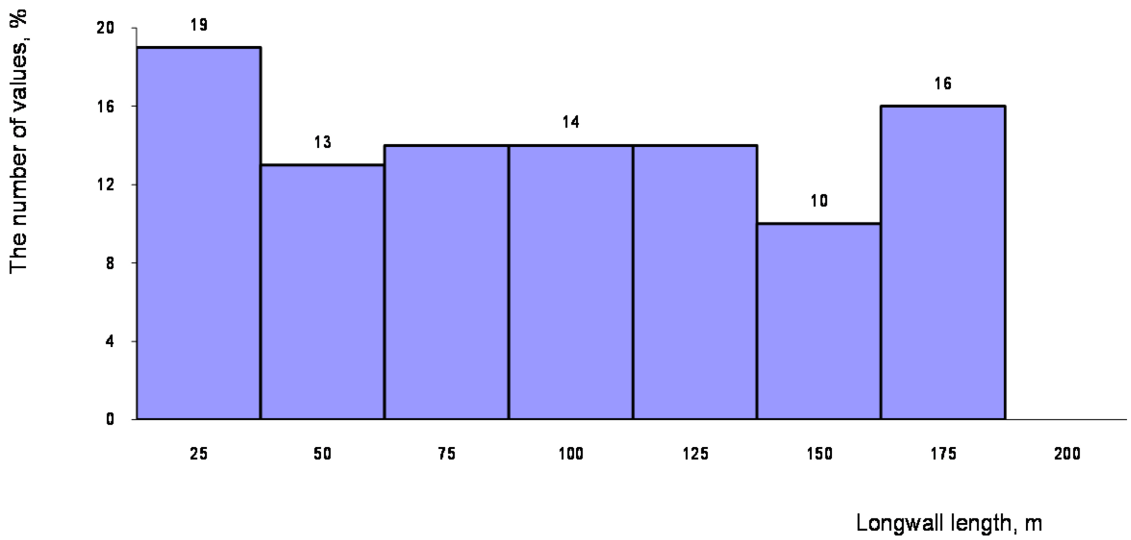


Fig. 4.6. Histogram of inrush distribution in terms of longwalls 117 (a) and 215 (b) length (C_6 and C_6^1seams)

Table 4.5

Results of inrush processes analysis within longwalls

No.	Mine, longwall, seam	Research period	Availability of overworking	Interbed thickness, m	Inrush height, $\frac{\text{min}-\text{max}}{average}$,m			Specific inrush, m^3/m		
					In the neighbourhood of boundary entry	In the central part of longwall	In the neighbourhood of conveyor entry	In the neighbourhood of boundary entry	In the central part of longwall	In the neighbourhood of conveyor entry
1	*Stepnaia*; 215; c_6^1	08.09.01-29.10.01	nonavailable	11	–	–	$\frac{0,1\text{-}0,3}{0.2}$	–	–	0.01
2	*Stepnaia*; 213; c_6^1	14.07.01-27.08.01	nonavailable	10	$\frac{0.2\text{-}0.3}{0.25}$	–	–	0.012	–	–
3	*Stepnaia*; 211; c_6^1	06.08.03-26.09.03	nonavailable	11	–	–	$\frac{0.1\text{-}0.4}{0.25}$	–	–	0.012
4	*Stepnaia*; 217-bis; c_6^1	20.06.00-24.08.00	nonavailable	11	$\frac{0.1\text{-}0.3}{0.2}$	–	–	0.01	–	–
5	*Stepnaia*; 215-bis; c_6^1	12.03.03-04.05.03	nonavailable	11	$\frac{0.1\text{-}0.6}{0.35}$	–	–	0.018	–	–
6	*Stepnaia*; 213-bis; c_6^1	11.05.03-03.07.03	nonavailable	11	$\frac{0.3\text{-}0.5}{0.4}$	–	–	0.02	–	–
7	*Stepnaia*; 119; c_6	15.07.02-28.08.02	overworked	11	$\frac{0.5\text{-}1.2}{0.85}$	$\frac{0.3\text{-}0.8}{0.55}$	$\frac{0.4\text{-}1.1}{0.75}$	0.042	0.027	0.037
8	*Stepnaia*; 117; c_6	10.09.02-22.10.02	overworked	11	$\frac{0.3\text{-}1.9}{1.1}$	$\frac{0.2\text{-}1.0}{0.6}$	$\frac{0.3\text{-}1.8}{1.05}$	0.062	0.034	0.058
9	*Stepnaia*; 117-bis; c_6	02.10.03-17.11.03	overworked	11	$\frac{0.6\text{-}1.9}{1.25}$	$\frac{0.3\text{-}1.2}{0.75}$	$\frac{0.4\text{-}1.2}{0.8}$	0.071	0.042	0.045
10	*Stepnaia*; 106; c_6	23.10.02-18.12.02	nonavailable	10	$\frac{0.1\text{-}0.7}{0.4}$	$\frac{0.15\text{-}0.55}{0.35}$	–	0.02	0.018	–
11	*Stepnaia*; 110; c_6	15.03.04-03.05.04	nonavailable	11	$\frac{0.2\text{-}0.6}{0.4}$	–	$\frac{0.15\text{-}0.5}{0.3}$	0.02	–	0.015

148 cases of 0.2 to 1.8 m roof rock inrush were recorded in longwall 117. Moreover, the majority of the cases took part within end sections of the longwall: 35 cases in the neighbourhood of conveyor entry and 39 cases in the neighbourhood of boundary entry. 63 inrushes were recorded in longwall 215 with 0.1 to 0.5 m. Upon the average, inrush height was 0.1 m in terms of C_6^1 seam and 0.4m in terms of C_6 seam. Thus it has been determined that inrushes in longwalls of C_6 were 2 – 2.5 times more intensive to compare with C_6^1 seam.

It should also be noted that within longwalls of C_6 seam mining it out of overworking zones, both values of specific inrushes and inrush height are close to proper values within longwalls of C_6^1 seam. It helps suppose that deterioration in roof conditions within longwalls engaged in mining of rather contiguous seams depend on their development overworking. In both cases the majority of inrushes happen in the neighbourhood of entries. That can be explained by the fact that within overworking zones both coal seam and enclosing rocks experienced extra stresses while mining in terms of overlying seam C_6^1.

4.2.2 Enclosing roof and floor convergence

Enclosing roof and floor convergence in the process of C_6 and C_6^1 seams mining were analyzed in accordance with provisions of operation technique at measuring points (Fig. 4.7.) located in central parts of longwalls and in end sections (10-15 m from entry). Gage posts СУИ -II with mounting attachments and clock-type indicators ИЧ-10 were used as measuring instruments.

Gage post one was mounted at the distance of 0.1 m from entry between ceilings of support sections; gage post two was mounted behind longwall conveyor at the distance of 1.2 m from the entry between the same sections. Then gage posts were mounted at the distance of 0.8 m from each other which depended on web width and advance increment of support.

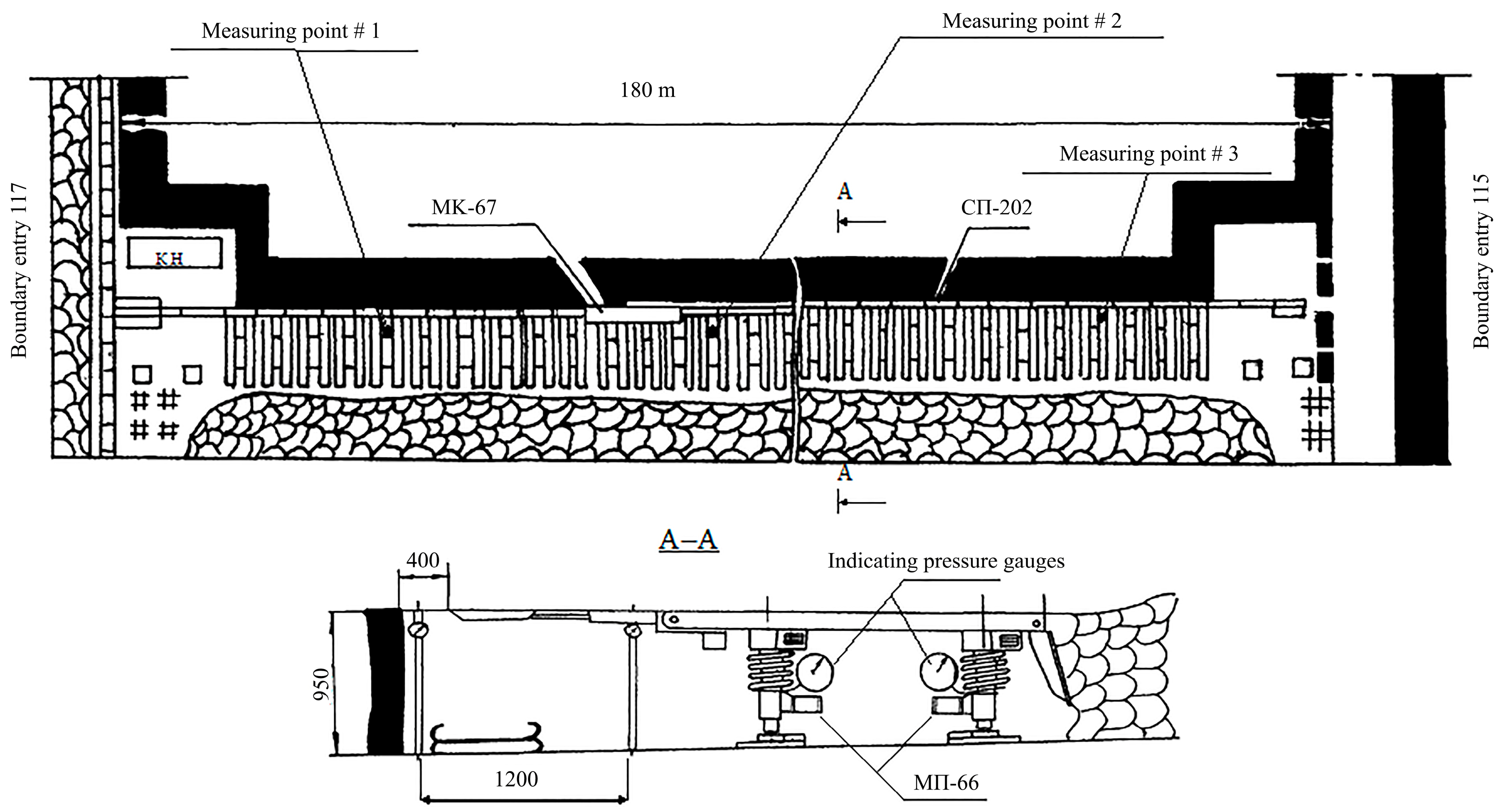


Fig.4.6. Allocation scheme for measuring points in longwall

When coal shearer approached a support pillar located in the neighbourhood of a stope, mining operations stopped, the support pillar was displaced behind the coal shearer and measurements continued. Arrangement of allocation of the rest of gage posts helped overwatch enclosing roof and floor convergence when coal shearer was passing and displacement of support took place. Thus, measuring process of enclosing roof and floor convergence was not interrupted during the whole period of winning cycle; moreover, distance to a coal shearer and support sections being displaced was controlled.

Table 4.6 shows results of the research. The results helped determine that convergence velocity of roof and floor rocks until coal shearer is at the distance of up to 3 m stays constant (background) being 0.02…0.06 mm/min. If the distance is less than 3 m, convergence velocity starts its increase; thus, when support is displaced the velocity is 0.8 mm/min. After support has been displaced and coal shearer has continued its movement, convergence rate goes on its acceleration; at the distance of 8 to 10 m behind a coal shearer (that is in a zone of support displacement) it is 1.6 to 1.8 mm/min. When coal shearer moves away from a gage post, convergence rate drops; at the distance of 25 to 30 m it becomes almost initial one (0.06 to 0.08 mm/min).

Analysis of the results shows that the process of enclosing rocks displacement within the area where seam is being mined with the help of coal shearer in terms of longwalls of C_6 and C_6^1 seams is of different velocities. Convergence velocity within longwalls mined C_6 seam in a zone of overworking was 10-15% higher to compare with similar conditions (out of overworking zone or within longwalls of C_6^1 seam); moreover, its attenuation was slower. That can be explained by different roof conditions within overworking zone and out of it. Thus, in the majority of cases, longwalls of C_6 seam demonstrated loss of immediate roof rock continuity just at the distance of 2 to 2.5 from a stope. Integrally, comparison of measuring results explains that average value of convergence of rocks within face area (at the distance of 2 m from face line) in overworking area is 15 to 20% more then seam is mined within rock mass (Fig. 4.8.).

Table 4.6.

Results of enclosing rocks convergence within longwalls of contiguous seams

No.	Mine, longwall, seam	Research period	Longwall advance, m/day	Mined seam thickness, m	Value	Value of enclosing rocks convergence					Total convergence values, mm
						Distance from measuring point to a stope, m					
						0	0,8	1,6	2,4	3,2	
1	2	3	4	5	6	7	8	9	10	11	12
1	*Stepnaia*; 215; c_6^1	08.09.01-29.10.01	2.8	0.85-0.87	min	4	16	17	20	20	77
					average	7	28	27	28	29	116
					max	9	36	40	36	42	163
2	*Stepnaia*; 213; c_6^1	14.07.01-27.08.01	2.6	0.85-0.87	min	4	18	19	17	20	78
					average	6	24	28	28	33	119
					max	8	36	40	36	40	160
3	*Stepnaia*; 211; c_6^1	06.08.03-26.09.03	3.4	0.85-0.87	min	5	18	19	16	21	79
					average	7	27	29	28	34	127
					max	10	36	38	35	39	158
4	*Stepnaia*; 217-bis; c_6^1	20.06.00-24.08.00	2.4	0.87-0.9	min	16	17	19	22	27	101
					average	18	26	30	31	35	142
					max	10	35	37	40	42	162
5	*Stepnaia*;215-bis; c_6^1	12.03.03-04.05.03	2.9	0.85-0.87	min	14	21	23	24	28	111
					average	17	24	29	30	32	153
					max	21	37	35	35	45	173

6	*Stepnaia*; 213-bis; c_6^1	11.05.03-03.07.03	3.1	0.85-0.87	min	6	14	20	20	22	109
					average	19	22	36	38	41	166
					max	21	30	37	52	56	196
7	*Stepnaia*; 119; c_6	15.07.02-28.08.02	2.7	0.95	min	10	25	32	-	-	67
					average	12	32	35	-	-	79(179)
					max	15	38	40	-	-	93
8	*Stepnaia*; 117; c_6	10.09.02-22.10.02	3.2	0.97	min	12	22	30	-	-	64
					average	15	26	33	-	-	74(278)
					max	18	34	36	-	-	88
9	*Stepnaia*; 117-bis; c_6	02.10.03-17.11.03	3.0	0.95	min	10	22	29	35	42	131
					average	12	25	34	35	42	190
					max	16	29	42	42	50	198
10	*Stepnaia*; 106; c_6	23.10.02-18.12.02	2.5	0.83	min	8	18	20			91
					average	10	25	29			122(257)
					max	12	32	38			158
11	*Stepnaia*; 110; c_6	15.03.04-03.05.04	3.0	0.87	min	9	16	21			93
					average	12	26	30			129(211)
					max	15	36	36			162

Note: ( )* is average convergence value on the line of worked-out area

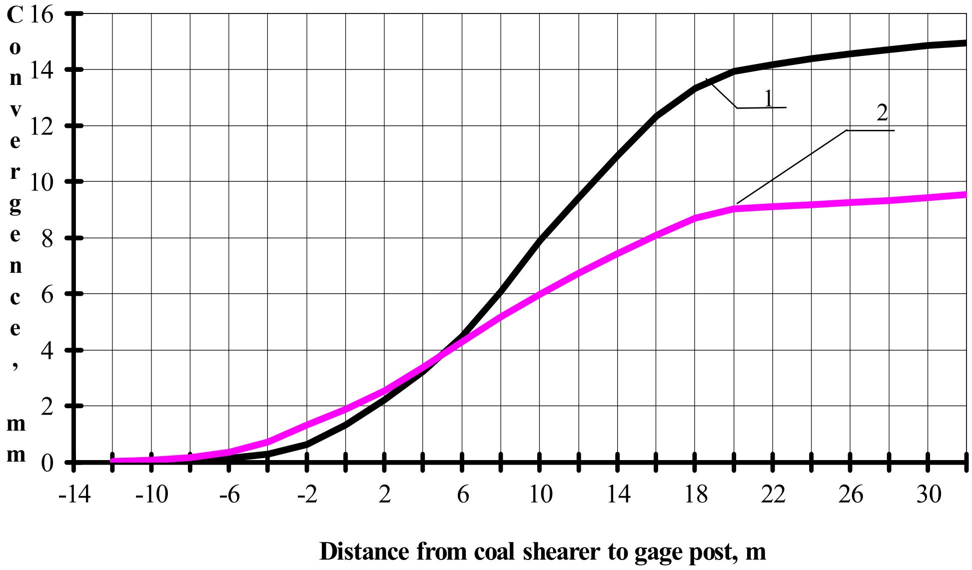


Fig. 4.8 Dependence of convergence value of enclosing rocks on the distance to coal shearer while mining

1. *In the context of a seam within overworking zone*
2. *In the context of a seam out of overworking zone*

The research results have helped recognize that both velocity and total value of convergence of enclosing rocks within end sections of all analyzed longwalls are 15 to 25% higher to compare with those in the central part of the longwall. That can be explained by the availability of high pressure zones within the sections which depends on overlapping of support pressure zones from longwall and development working.

In large, it may be concluded that mining of contiguous seams with thin parting (8 to 11 m in this context) in longwalls mined seams within overworking zones degrades state of rocks. Thus if rocks of seam C_6 immediate roof within zones (where the seam is not overworked) they are characterized as "mean-stability roof". In terms of overworking zones, immediate seam roof is characterized as "unstable roof".

Thereby, underground investigations of power parameters and flexibility of hydraulic props of powered support KMK97M have been carried out to estimate its relevance to available mining and geological conditions.

4.2.3 Power parameters and flexibility of hydraulic props of support

Power parameters of hydraulic props of powered support as well as their flexibility were measured within measuring points in longwalls (Fig.4.7) over 60-100 m of a stope advance; that meets the requirements of both specifications and mathematical statistics. Data recorders МП66А were involved for constant pressure registration. The recorders were placed within head ends of front legs and rear legs of sections.

Flexibility of hydraulic props was measured with the help of a ruler. The procedure involved determination of rod surface right after thrust of a section and before its following advance. Average flexibility of supports was 28…35 mm; sometimes it was 45 mm. No "hard" run down of props of support was observed; however, there were cases when margin of extension of hydraulic props is completed. Most of all, such situations are observed within end sections of longwalls.

While analyzing power parameters of support, pressure in hydraulic system was varying within 18…22 MPa; its average value was 20 MPa. Table 4.7 contains measuring results concerning the whole period of observations; moreover, they have been processed with the help of mathematical statistics approaches.

Data in Table 4.7 show that initial setting load in terms of sections arranged in the neighbourhood of conveyor entries is higher to compare with sections in the central part of longwall and those in the neighbourhood of boundary entries. That can be explained by nearness of pump station as well as fluid leakages and flow of working fluid within hydraulic system and hydraulic control valves.

Working resistance of hydraulic props of support within end sections of C_6^1 seam is some higher to compare with that within central part of a longwall. That can be explained by increase in parameters of rock pressure manifestations within the areas. In the context of back props pressure is 10…15% higher to compare with front ones. Normally hydraulic props of support operated in a mode of progressive resistance. (Fig. 4.9 demonstrates it out of overworking zone).

Table 4.7

Measuring results concerning power parameters of support

No.	Longwall	Research period	Longwall advance, m/day	Mined seam thickness, m	Value	Давление в гидростойках крепи, МПа					
						начальное		рабочее		перед разгрузкой	
						I ряд	II ряд	I ряд	II ряд	I ряд	II ряд
1	215, seam $C_6^{'}$	08.09.01-29.10.01	2.8	0.85-0.87	min	10.3	13.4	15.2	18.7	23.2	24.6
					average	16.1	16.0	26.3	28.5	33.1	38.1
					max	19.7	24.7	42.0	40.6	47.7	45.8
2	215-bis seam $C_6^{'}$	20.06.00.-24.08.00	2.4	0.87 –0.9	min	12.4	14.3	16.5	17.7	22.5	24.0
					average	17.2	18.1	25.6	28.3	32.7	39.8
					max	20.5	23.7	40.2	44.0	44.2	48.3
3	117, seam C_6	10.09.02.-22.10.02.	3.2	0.95	min	12.0	8.3	19.2	4.1	23.3	–
					average	15.5	9.6	30.5	6.6	36.4	4.4
					max	20.2	12.4	42.0	10.4	48.2	6.2
4	117-bis, seam C_6	02.10.03.-17.11.03.	3.0	0.95	min	14.7	6.3	20.0	3.3	24.5	–
					average	16.8	9.0	30.5	5.4	38.4	4.2
					max	19.0	11.4	44.3	8.2	48.4	8.3
5	110, seam C_6	15.03.04.-03.05.04.	3.0	0.87	min	12.4	13.7	15.5	17.3	22.4	26.2
					average	17.6	17.5	24.0	27.2	35.7	40.5
					max	20.2	24.3	38.2	40.6	44.4	47.2

Fig. 4.9. Nature of roof rock hanging within worked-out area of C_6^1 seam

In the process of longwall operation within C_6 seam (overworking zones) when excessive fissuring of roof rocks took place, pattern of support functioning was varying: pressure within back props of support was much lower to compare with front ones. That can be explained by roof failure over back prop of the support and real loss of "support-roof" contact. In the process of support advance initial resistance within back hydraulic props was given at 6…12 MPa level to be 2…2.5 times less to compare with front ones. In the context of greater initial resistance back part of the support crushed overlying rocks, and hiked up; shield canopies sloped down which prevented coal shearer from doing pass resulting in loss of contact between stope and front prop of support.

As recording pressure gauges did not perform spatial and temporal recording of rapid processes of mining and advance of props of support, indicating pressure gauges were applied for operative pressure control within hydraulic props of support. Take-off data made it possible to determine that due to regular convergence of enclosing roof and floor within longwalls of C_6^1 seam (if mining did not take place), pressure in hydraulic props demonstrated constant increase at the rate of 0.6…0.08MPa during 0.5 hour. When distance between coal shearer and measuring point was 7…5 m, the rate of pressure change started to be increased. After pass of coal shearer and neighbouring section advance, rate of pressure change became maximum that is 0.2 MPa per minute. Pressure value before unloading of sections was: 23…28 MPa in terms of front props, and 26…35 MPa in terms of back ones. In the context of seam C_6 nature of pressure changes in front hydraulic props corresponded to above; however, values of changes in pressure were 1.2…1.5 times higher. Pressure value before unloading of sections within front props was 28…40 MPa. Principally, back props operated in a mode of initial pressure decrease; before unloading their pressure value was 0…6 MPa. It means that powered support МК-98 is applied under improper conditions (that is for a seam with unstable roof rocks). Therefore we believe in expedient to use powered system of КД90 type to mine C_6 seam within overworking zones instead of КМК97М system being applied. The former system is meant to mine thin coal seams with unstable roof rocks.

4.3. Results of underground investigations in terms of gate roads

4.3.1 General information

Field studies were carried out within boundary entries and conveyor ones drawn in terms of contiguous seams C_6 and C_6^1 (see Table 4.2).

Benchmark stations and control frames were used for measurements in mine workings under analysis. Fig. 4.10 demonstrates their layout in terms of boundary entry 115 of seam C_6. Fig. 4.11 explains layout of equipment within measuring point; Fig. 4.12 shows measuring point in the entry.

As we were interested in data concerning effect of mining operation of seams C_6 and C_6^1 on development workings, measuring points were instrumented subsequent to two-three months after entries had been driven that is after mine pressure was stabilized and initial convergence of enclosing roof and floor attenuated.

Three to five benchmark stations were instrumented in each entry; ten to twenty-five control frames were selected. The control frames were used to measure both width and depth of entry, to record distance from below-ground benchmark up to midline and from the midline up to grider, overlap of components within pipe joints etc. Benchmark stations also registered values of depth benchmark parameters.

4.3.2 Enclosing roof and floor convergence

Convergence of enclosing roof and floor was measured within mine workings to be filled before the event; in the context of mine workings to be reused the value was determined before their retimbering; after that new benchmark stations were equipped to perform measurements up to the moment of the mine working filling.

Fig. 4.13 shows graphs of changes in values of enclosing roof and floor convergence. The graphs are based upon measurement results obtained within benchmark stations of seam C_6^1 boundary entries (## 1 – 4, Table 4.2). Equal-length benchmark pairs (0.5-0.5; 1.0-1.0 and 2.0-2.0m) were measured.

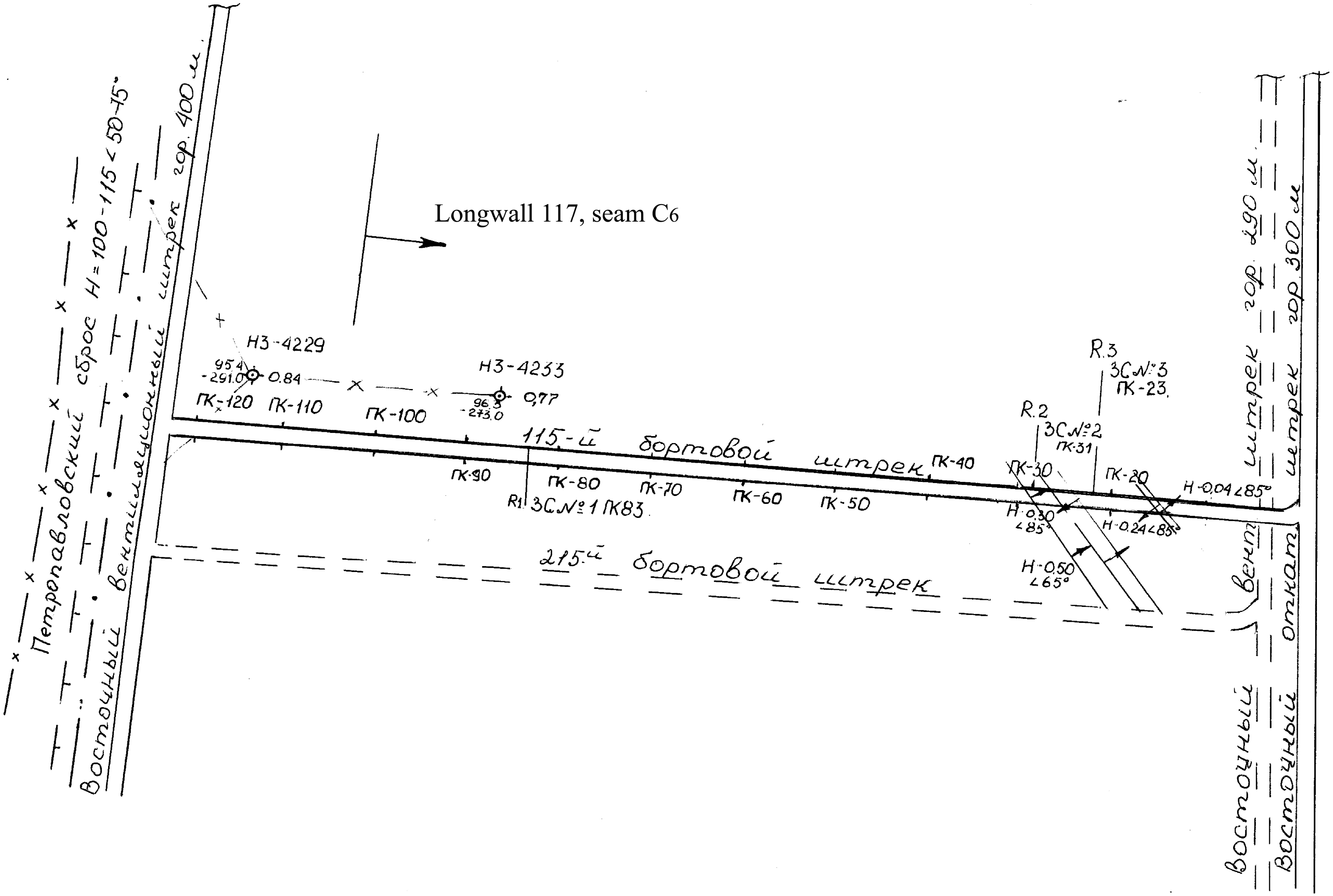


Fig. 4.10. Location of benchmark stations within boundary entry 115:
ПК 110 – 15 are control frames; R1, R2, and R3 are measurement benchmark stations

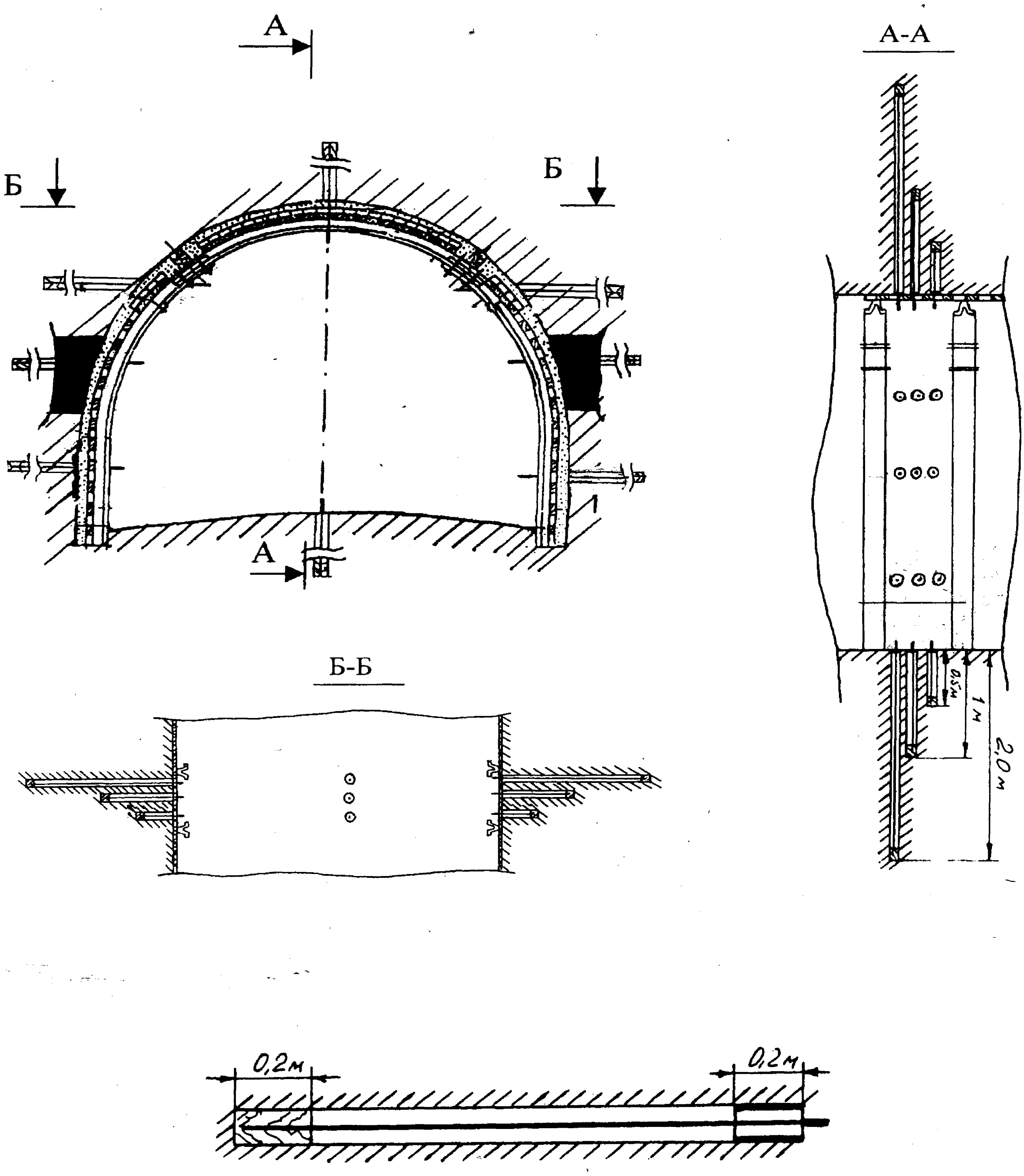


Fig. 4.11. Location of equipment within measurement benchmark station in entry

a)

b)

c)

Fig. 4.12. Measurement benchmark station in entry
a) general view; b) left-side view c) right-side view.

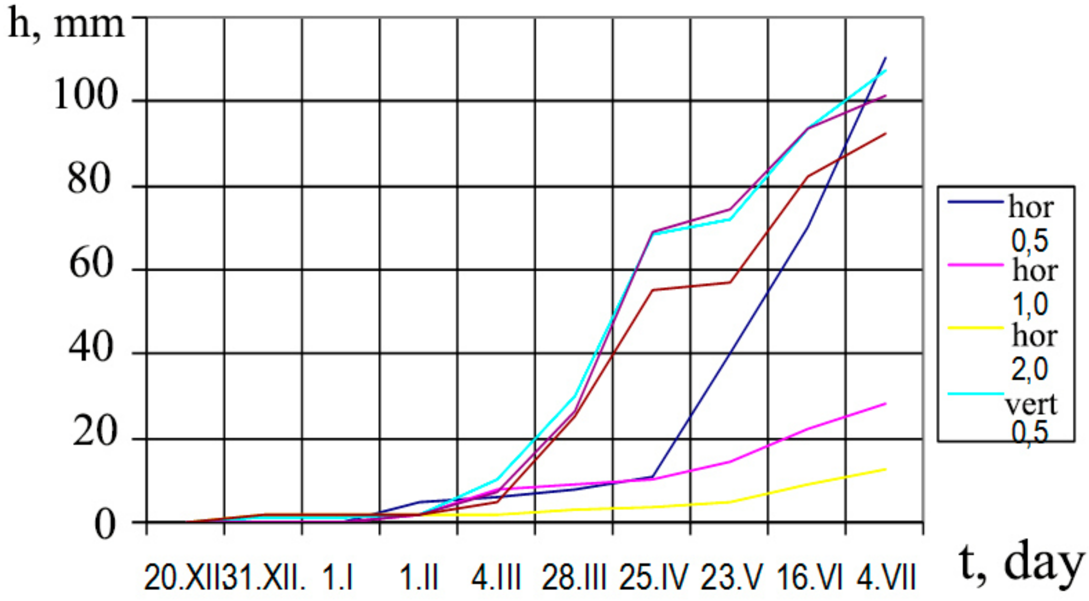


Fig. 4.13 Changes in enclosing roof and floor convergence according to data of benchmark stations within seam C_6 boundary entries

Convergence velocity of both walls of mine workings and roof with floor remained constant in terms of each pair of benchmarks; it was 5 to 8 mm per month. When longwalls of contiguous seam C_6^1 neared benchmark stations for 15 to 30 m in the context of vertical benchmark stations convergence velocity started to be accelerated up to 70 to 90 mm per month; in the context of horizontal benchmark stations it was 30 to 50 mm per month. Moreover, following rule was observed in terms of both vertical and horizontal pairs: the longer is benchmark station, the slower is rock convergence; convergence value also experiences its reduce. It means that destruction of rocks and their foliation took place within walls of the mine working.

Convergence values were measured at benchmark stations within boundary entries of seam C_6 (in a zone of mining effect and out of it) have been processed and shown as graphs in Figures 4.14 – 4.17. The graphs explain that availability of boundary entries of seam C_6 within mining effect of C_6^1 seam at the distance of 15km from upper seam edge results in extra (0.1 to 0.3 m) looses of depth of entry due to convergence intensification while upper seam mining.

Thus, in the context of boundary entry 115 (seam C_6) meeting the requirements of normative technique [31] and located under worked-out area of upper seam at the distance of 15km (Fig. 4.14) from its edge normally the loss of depth was 0.18 to 0.20m at the expense of mining effect in seam 217 of seam C_6^1. In the context of 115-bis boundary entry of the same seam (Fig. 4.15) located at the distance of seam C_6^1 edge the value was 0.08 to 0.10m in the process of longwall 217-bis mining to be 2.0 to 2.5 times less. In the context of boundary entries 111 and 111-bis located at the distance of 40m from upper seam edge (to be in accordance with the developed technique) neither mining of 213 nor 213-bis longwall operates upon its mining (Fig. 4.17).

They "filled" themselves similarly to those entries located under worked-out area out of mining effect; for instance, it concerns boundary entry 119 (Fig. 4.16). Thus, results of underground investigations of ground pressure within development workings depending upon their location prove both provisions of the technique and mathematical tool used in it. Divergence of design parameters and actual ones is not more than 20%.

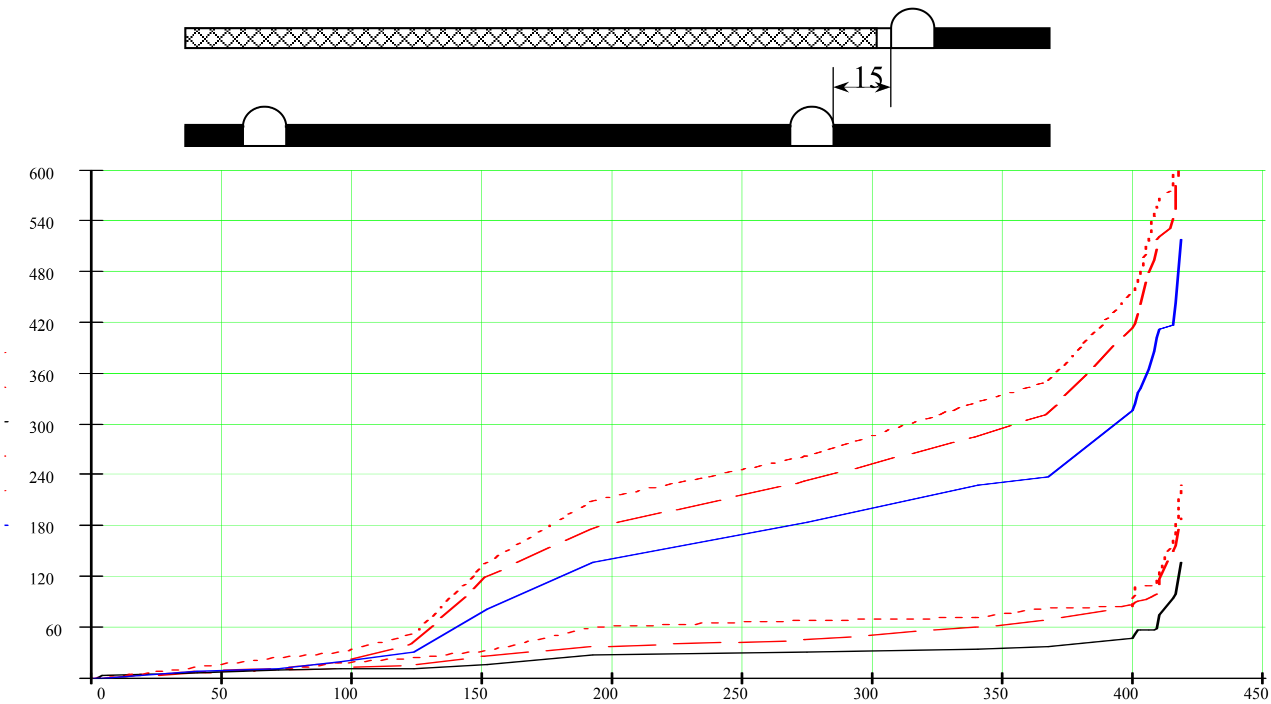


Fig. 4.14. Changes in convergence of floor and roof of mine working located at the distance of 15 m from upper seam edge (in time)

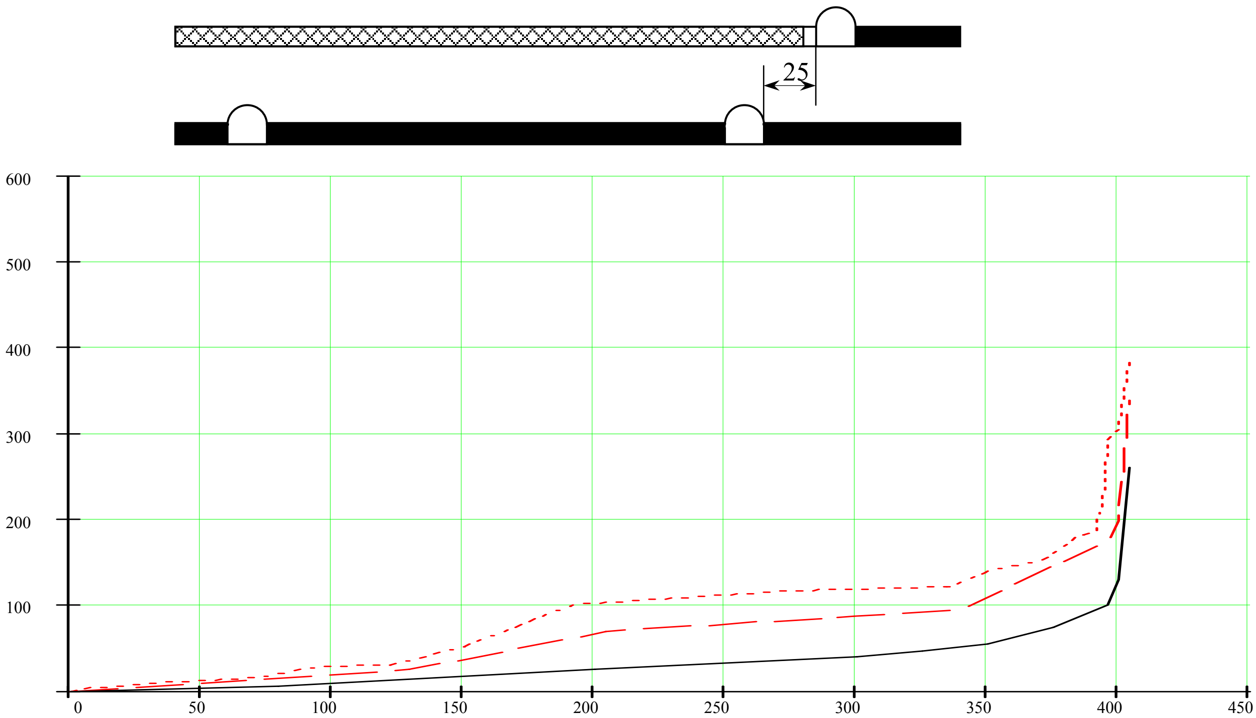


Fig. 4.15. Changes in convergence of floor and roof of mine working located at the distance of 25m from upper seam edge (in time)

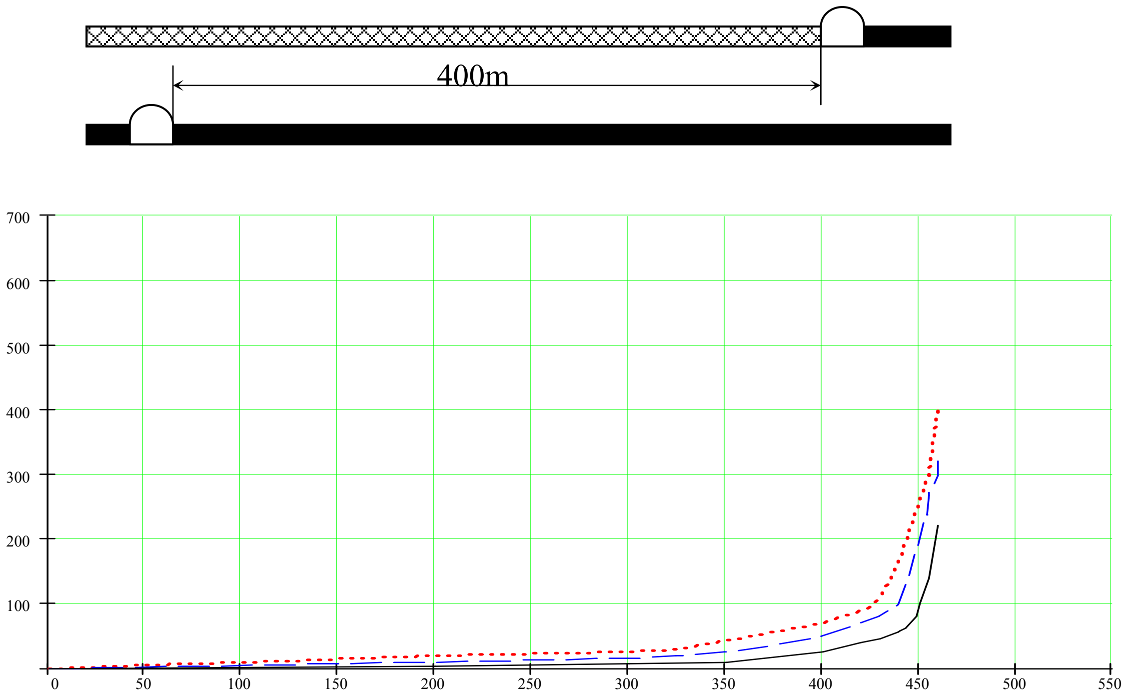


Fig. 4.16. Changes in convergence of floor and roof of mine working located at the distance of 400 m from upper seam edge (in time)

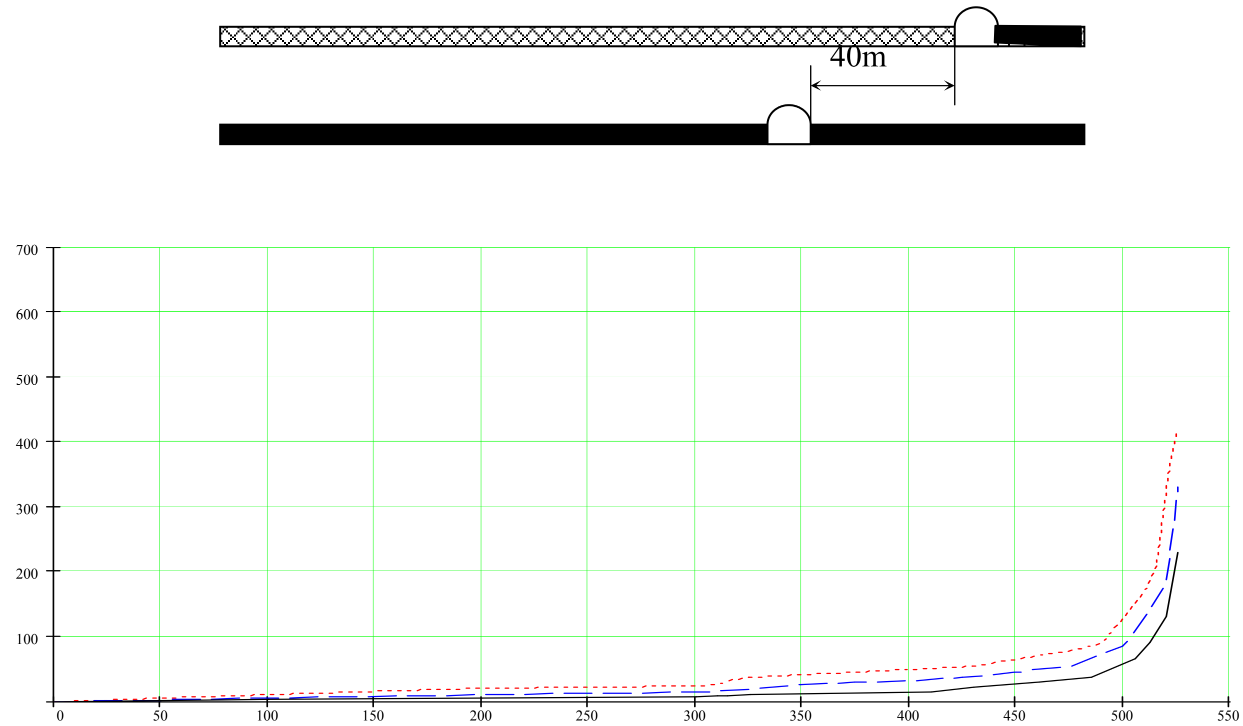


Fig. 4.17. Changes in convergence of floor and roof of mine working located at the distance of 40m from upper seam edge (in time)

Underground investigations have proved that the greatest rock displacements take place in close proximity to mining and within worked-out area right after pass of longwall.

Results of measurements performed within control frames of boundary entries of both seams to be reused have been applied to plot dependences of roof rock and floor rock convergence values on distance to longwall (Fig. 4.18). The graph explains that at the distance of 25-30 m to longwall a value of roof floor convergence is insignificant (almost 200 mm); convergence value is constant being no more than 2 mm per day. In the context of further approximation of longwall, convergence velocity experiences sharp acceleration; in certain cases it was 25 to 35 mm per day. During the period of the investigation start up to the longwall approach to control frame, depth of the entry decreased by 700 to 750 mm.

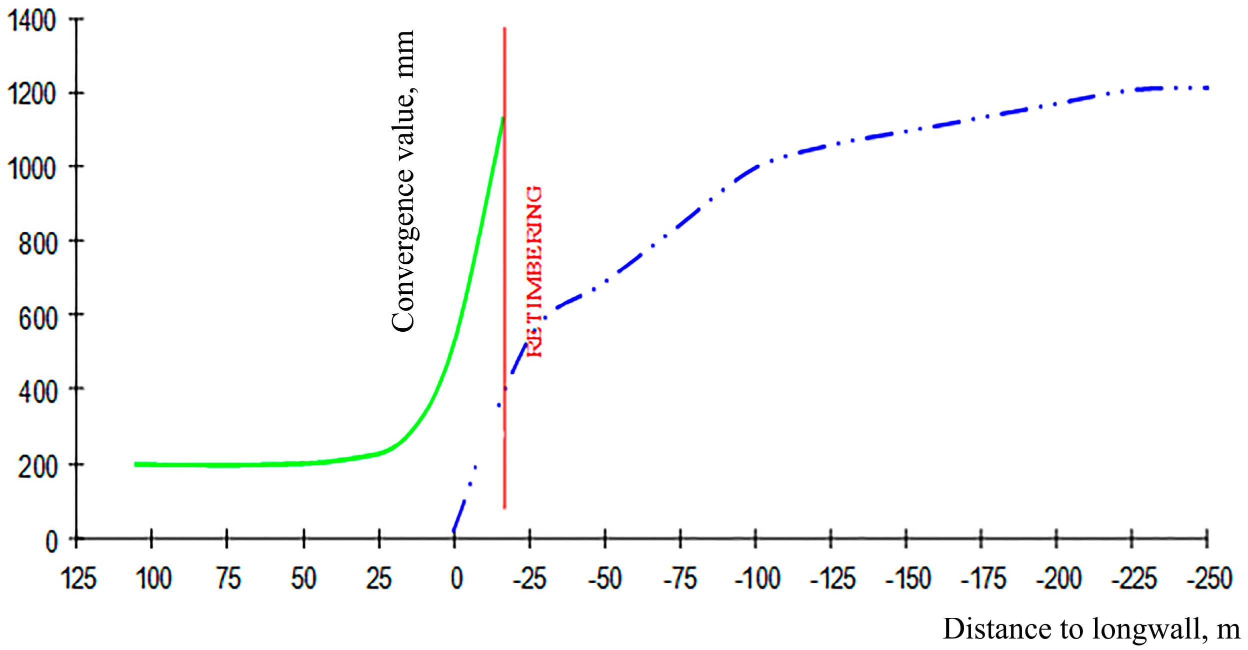


Fig. 4.18. Nature of changes in convergence value of roof and floor rocks within reused boundary entries

In this context, pliability of supports was practically completed, and arch components experienced their deformation (Fig. 4.19). When distance between longwall was 2 to 3 meters and further to retimbering area both velocity and value of convergence experienced their sharp increase. Within the site loss of vertical dimensions of entry was 500 to 700mm. Altogether, from the initial stage of operation to retimbering, vertical dimensions in section of entry experienced their 1200 to 1400mm decrease. Retimbering of entry took place at the distance of 8 to 12m behind longwall by means of roof rock discharge (Fig. 4.20, a).

Nature of roof and floor convergence in entry varied after retimbering process. At the distance of 50m from retimbering stope vertical dimensions of mine working experienced its 600 to 750 mm decrease. Further convergence velocity decreased slightly; only after 100-150 m it stabilized (Fig. 4.18). During the period total convergence value became 1000 to 1200 mm.

During the investigations (5 to 7 months) width of entries experienced 120m to 190 mm decrease to be 168 mm on an average. After mine working has been retimbered both nature and values of walls of entry convergence remained almost similar to those obtained before longwall approach. It should be noted that rock heaving in entry after retimbering increased; in some cases it was 40 to 50% of total convergence value. Thus, retimbering of mine working in terms of such a distance from longwall stope cannot give positive results under the circumstances. Moreover, reuse of the entry involves retimbering.

Fig. 4.20 (b, c) shows boundary entry 115 after its retimbering at the distances of 100 and 250 m. At the distance of 250 m from retimbering stope vertical dimensions were 1.4 to 1.5m only. Therefore, the entry experienced one more retimbering; in this context mining machine performed dinting of floor rocks (Fig. 4.21).

The results of underground investigations and calculations performed according to the technique make it possible to state that to maintenance-free operation of entry depends on retimbering distance (no less than 150-200 m) behind the longwall within a zone of mine pressure stabilization.

a)

b)

c)

Fig. 4.19. Typical damage of arch support:
a) from rock mass, b) and c) from worked-out area

a)

b)

c)

Fig. 4.20. Boundary entry 115 after retimbering:
a) retimbering stope; b) 100m behind the longwall; and c) 250 m behind the longwall

Fig. 4.21. Retimbering of boundary entry 115 with floor rock extraction (1.0 to 1.2 m) with the help of mining machine

The recommended parameters have been implemented in the process of 115-bis boundary entry retimbering (117-bis longwall of C_6 seam). According to retimbering pattern of the entry its lagging behind longwall was adopted as 100 to 120m in comparison with boundary entry 115 which retimbering was performed with 8 to 12 m lagging. Fig. 4.22 demonstrates results of enclosing roof and floor convergence in the context of boundary entry 115-bis.

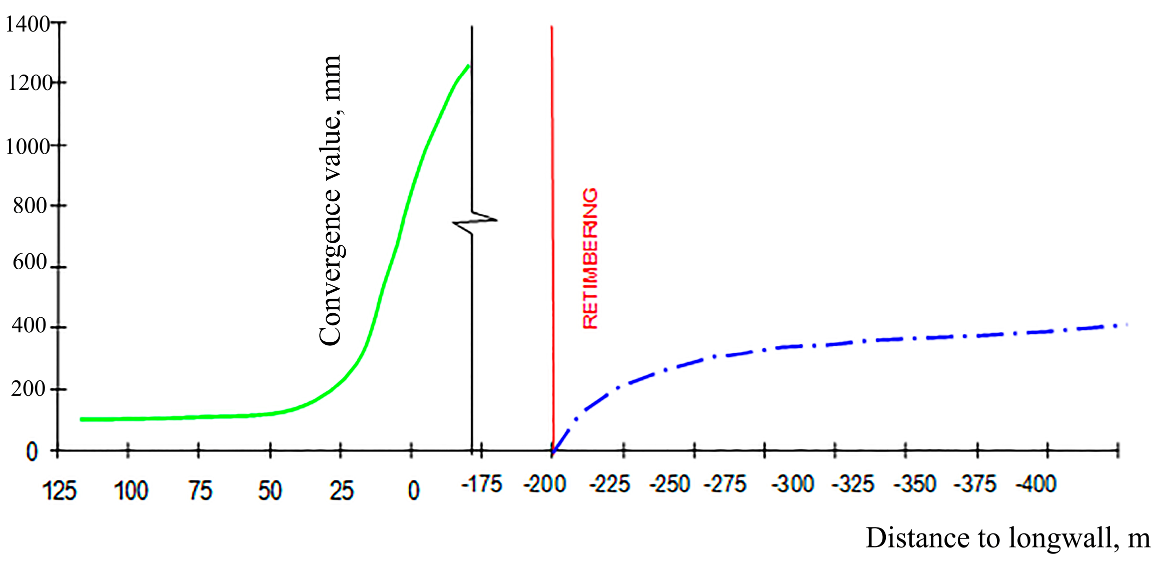


Fig. 4.22. Nature and values of roof and floor rock convergence within boundary entry 115-bis in the context of its retimbering at the distance of 120m behind the longwall

Data in Fig. 4.22 explain that increase in distance up to 120m from longwall stope to retimbering area of boundary entry 115-bis results in significant decrease of total value of roof and floor convergence; that made it possible to reuse the entry without repetitive retimbering. Thus, both retimbering area and time is important parameter to preserve stability of gate roads and their reuse together with a site of mine working. That involves extra investigations for adequate substantiation of rational area to retimber mine workings. Parameters recommended by the paper have been obtained relying upon field studies for certain mine workings.

4.4 Reproducibility analysis concerning experimental results and theoretical ones

4.4.1. Comparison of results of calculations and underground measurements of enclosing roof and floor convergence in stopes

To determine reproducibility of analytical studies and underground investigations according to the algorithm, enclosing roof and floor convergence has been calculated in stopes of upper ($\Delta(u_в)_{max}$) seam and lower ($\Delta(u_н)_{max}$) one under the conditions of overworking when interbed thickness is 8 to 11 m.

2.12 formulas were used for the calculations. Cases when longwall operated in terms of upper seam C_6^1 (the seam is neither overworked nor underworked) and lower seam C_6 was mined within overworking zone were considered. In this context, mining depth values varied within 300 to 350 m with 10 m interval. Other initial data were as follows: L=100 m, $m_в$=0.9 m, $m_н$=0.95 m, h =12 m, $E_п = 0.29\cdot10^{-4}$ MPa, $E_у = 0.36\cdot10^{-4}$ MPa (for C_6^1 seam) and $E_п = 0.28\cdot10^{-4}$ MPa, $E_у = 0.35\cdot10^{-4}$ MPa (for C_6 seam). Table 4.8 demonstrates results of calculations and underground investigations of convergence within stopes.

Table 4.8

Results of analytical studies and field investigations of enclosing roof and floor convergence within stopes

Mining depth H, m	Convergence, Δu_{max}		Divergence of calculation results and field investigations, %
	Theoretical value, mm	True value, mm	
C_6^1 seam			
300	125	116…146	17.2
310	139	125…159	14.4
320	153	132…162	13.7
330	166	133…178	19.8
340	178	147…193	17.4
350	189	166…196	12.2
C_6 seam			
300	182	156…193	14.3
310	194	163…204	16.0
320	203	176…207	13.3
330	211	185…217	12.3
340	219	187…223	14.6
350	226	186…227	17.7

As data from Table 4.8 show, divergence between calculation results and true values of rock convergence within stopes of seams under mining is not more than 20%.

4.4.2. Comparison of calculation results and underground measurements of enclosing roof and floor convergence in the context of development workings

To estimate reproducibility of experimental observations and theoretical values of roof and floor rock convergence within development workings in terms of their different locations as for upper contiguous seam selvedge, proper true convergence values obtained at measurement stations have been compared with its theoretical values. In this context theoretical values of convergence were determined taking into consideration specific characteristics of enclosing rocks, mining depth, interbed thickness, time parameters etc. Table 4.9 involves results of calculations and measurements.

Table 4.9

Results of analytical studies and field investigations of enclosing roof and floor convergence within development workings

Measuring point	Convergence, Δu_{max}		Divergence of calculation results and field investigations, %
	Theoretical value, mm	True value, mm	
Mine working: boundary entry 115, l = 15m			
1	308.4	368	19.5
2	356.8	422	18.2
3	348.5	404	15.9
Mine working: boundary entry 115-bis, l = 25m			
1	172.2	197	14.4
2	196.5	235	19.6
3	211.4	242	14.5
4	225.7	264	17.0
Mine working: boundary entry 111.l = 40m			
1	109.3	124	13.4
2	112.5	127	12.9
3	120.2	142	18.1
4	126.6	147	16.1

As Table 4.9 demonstrates divergence between true convergence values and theoretical ones within development workings is not more than 20%.

Conclusions

Following conclusions may be drawn according to the results of underground investigations in stopes and development workings of contiguous seams:

1. In the context of contiguous seams with thin interbed (up to 10-15 m) state of immediate roof rocks worsens sharply in longwalls operating in overworking zones: fracturing intensity experiences its 3 to 5 times increase; inrush intensity experiences its 2 to 2.5 increase; enclosing roof and floor convergence increases by 20 to 25%. At the distance of nor more than 2-2.5 m from a stope, roof failure takes place. As a result, it loses its continuity.

2. Equipment designed to mine seams with unstable roofs should operate within overworking zones that is КД90 powered systems should replace КМК97М ones.

3. In the context of contiguous seams, development workings should be located within discharge zone out of mining operations performed in adjacent seam. For the considered mining and geological conditions, distance from edge of upper seam down to mine working should not be less than 35 to 40m.

4. The greatest displacements of enclosing roof and floor within a development working takes place in close proximity to mining operations (up to 15m before longwall and up to 150 behind it). Thereby retimbering of extraction entries meant for reuse is recommended to be performed at the distance of 150m behind the longwall within a zone of rock pressure stabilization.

5. On the whole, results of underground investigations confirm theoretical calculations. Difference in calculation results and actual values of parameters under study is not more than 20% to be satisfactory for required calculation accuracy.

OVERALL CONCLUSION

The monograph is weighty research as it has solved topical scientific problem concerning substantiation of boundary parameters of mining concentration basing upon determining rules of rock pressure manifestation in terms of mine workings and development ones while mining contiguous coal seams in the context of mines of Western Donbass.

Basic scientific and practical recommendations are as follows:

1. Boundary-element method was applied to estimate effect of simultaneous mining of to contiguous coal seams on stress-strain state of rock mass in terms of advance overworking.

Calculation results according to the developed algorithm has been generalized and represented in the form of ratios characterizing rules of rock pressure manifestation in the process of contiguous seams mining in the context of mining and geological conditions being considered.

The accepted mathematical tools as well as the developed correlation ratios to determine maximum displacements of rocks of floor and roof of the considered seams taking into account mining depth, thickness of seams and interbed, physical and mechanical properties of coal and enclosing rocks, and distance between longwalls are the basis to specify safety mining boundaries within contiguous seams.

The developed correlation ratios making it possible to estimate maximum equivalent stresses within rock mass as well as enclosing roof and floor displacement in terms of lines of mine working contribute to identification of safe distance between development workings and mining operations of adjacent seams.

2. Rational areas of mining concentration providing both safe and expedient extraction and development operations in terms of contiguous seams have been determined using analysis of influential mining and geological factors as well as simulation results.

In the context of mines in Western Donbass following distances between stopes of contiguous seams are safe in terms of rock pressure: no less than 70 m if $h = 5$ m, and 60 m if h=15 m.

Determination of minimally allowable boundaries (safe from the viewpoint of rock pressure) and expedient location of mine workings in terms of concentration of mining operations within contiguous seams should involve total consideration of equivalent stresses and maximally allowable displacements of rocks along a line of the mine working.

Distances along overworking seam up to the selvedge of upper seam have been determined for the conditions under analysis. The distances are as follows: 60m if interbed thickness is h = 5m, and 25m if h = 15m. When the distances shorten, stresses within roofs of the mine workings under analysis experience sharp increase; right under edge of the seam they are more than five times higher to compare with ultimate compression strength of rocks. That results in loss of cross-sections of workings, necessity for their retimbering and maintenance.

3. Underground investigations concerning rock pressure manifestations within stopes and development workings while mining contiguous coal seams have been carried out. It has been determined that concentration of mining operations within contiguous seams is both possible and expedient within certain limits. Boundaries of such a concentration should be substantiated from the viewpoint of rock pressure safety and expenditures connected with construction, maintenance, and retimbering of mine workings.

Control of reproducibility of theoretical calculations and actual values of parameters under study has shown that divergence is not more than 20%; thus, results of the work are available for substantiation of boundaries of concentration of mining operations within contiguous seams.

Values of multiply correlation ratios of the determined dependences are 0.83…0.98 and values of Fisher's ratio test are much higher than one-percent divergence limits. The fact confirms rather close correlation between random values being analyzed and variable parameters as well as statistic value of the specified dependences.

4. The developed correlation dependences as well as results of underground investigations were used for "The procedure to determine rational spacing of development workings while mining contiguous seams in the context of Western Donbass". The procedure is applied by mines of "DTEK Pavlogradugol" PJSC and engineering company "Dneprogiproshakht".

Recommendations concerning production control of mining including contiguous coal seams have been formulated for mining and geological conditions of mines in Western Donbass.

The recommendations have also been taken into consideration while planning mining operations and their implementing in *Stepnaia* mine of "DTEK Pavlogradugol" PJSC.

Real economic effect resulting from decrease in expenditures connected with maintenance and reuse of boundary entry 115 of longwall 117-"bis" when it is initiated at the distance of 25m from seam c_6^1 edge was UAH808860.

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ГРАНИЧНІ ПАРАМЕТРИ КОНЦЕНТРАЦІЇ ГІРНИЧИХ РОБІТ ПРИ ВІДПРАЦЮВАННІ ЗБЛИЖЕНИХ ВУГІЛЬНИХ ПЛАСТІВ ЗАХІДНОГО ДОНБАСУ

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