SOME SPECIAL PROBLEMS OF MINE WATER CONTROL

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SUMMARY

Some extreme cases of efficiency reduction of dewatering and filtering wells are presented. Experience on the water control of the Kányás mine is described. Deterioration of submersible pumping wells and its prevention for the Bito incline are discussed. In the Kányás coalfield, the discharge reduction of drainage wells is not caused by filter clogging but by a decrease of the seepage coefficient due to the considerable stress increase in drainage. Against this effect proper drainage control can be used. In the Bito case a properly sized filter structure is recommended.

1. Introduction

One of the general active modes of controlling aquifers and gas reservoirs is to change to potential field by drainage from the surface or mining openings.

This control method has been extensively applied in Hungary owing to its relative simplicity, reliability and economics. However, some cases have occurred, as in Kányás, when the method was ineffective, or, as in Bito, disturbances took place during the drainage process, adversely affecting the efficiency of control.

This study considers some extreme disturbances of water level lowering. It is shown that a deterioration of drainage well efficiency may be caused not only by technological factors but also by a change of state in the aquifer.

2. Water control and development possibilities for the Kányás coalfield

Water control problems of the Kányás coalfield have been addressed for more than 30 years. A concensus was reached as early as in the fifties, that a comprehensive preventive dewatering of the porous aquifer overlying coal seam I is necessary in order the control inflow aggravating exploration /Vigh et al., 1957/. The method would consist of filter wells from roadways in seam I, and predrillings at fault crossings /Schmieder et al., 1960/.

Subsequent drainage experiments have, however, failed. The rate of exploration decreased and the future of shaft Kányás became gloomy. Then, the Hungarian Association for Mining and Metallurgy conducted a competition to solve the problem.

This competition induced fruitful work which can be demonstrated by the relatively great number of competitors.

In this paper some mine water control aspects and conclusions are given from my part of a competition paper prepared jointly with Demeter, Fényes and Verpeléti.

2.1. Water control experience of the Kányás coal reserves

Kányás shaft of the Nógrád Coalmining Company is located in the region of the villages Szupatak, Nagybátony and Mátraverebély /Fig. 1a./, with natural boundaries formed by large faults. Mining exploration and consequently mine water pumping started in 1948. The early, relatively low yield suddenly increased since 1953 /Fig. 2a./, then gradually decreased. The first inflow occurred during the sinking of the ventilation shaft shown in Fig. 1b, by a drillhole from the toe of the shaft.

The inrush of 750 l/min and the conveyed sediment provisionally flooded the shaft but after extensive drawdown the decreasing inflow yield caused no major problem. A similar event occurred in the haulage shaft located 100 m from the ventilation shaft. Here, the initial yield of inrush from overlying porous aquifer was 2 m^3/min together with a considerable amount of methane. This inrush flooded the shaft provisionally, but again, during drawdown the yield decreased rapidly then stopped within relatively short time.

From that time onward, more than 50 inrushes have been observed, with the common property of high initial yield. The typical discharge time function /Fig. 2c./ shows a maximum after some hours and decreases to zero after a certain time.

The highest yield estimated as 30 m^3/min occurred in 1958, with methane inflow, and carried sands of several hundreds of bogies.

Frequency distribution of maximum yields of inrushes accompanied by methane and sand inflows is shown in Fig. 2c. The expected value of q_o is 0,8 - 1,0 m^3/min with a solid volume of 40-50 bogies in the average.

Concerning the spatial distribution of inrushes, they occurred in exploration spaces without exception, and 70 % was connected to faults and 30 % was caused by the ineffective overlying protection layer.

The highest yields occurred in the initial exploration phase during the exploration of new tectonic blocks. A slower rate or stopping of the exploration resulted in smaller frequency and total flow of inrushes. A high proportion of total yield, 85-90 % originates from roadways and only 10-15 % pertains to faces of much greater area. In the Kányás mine there is no water hazard in mining itself as contrast to the exploration. In general, there is no inflow only some dripping in faces. The overlaying sand is generally quite dry, demonstrating that this overlying aquifer is already drained, even if in an uncontrolled way, as an effect of exploration.

In order to improve conditions of roadway driving and fault crossing, special attention was given in the sevaties to drainage preceding fault crossing and following roadway heads. This effort failed. Time function of the yield of drainage wells, water accompanied with sand and often methane, follows that of inrushes /Fig. 2c/

$$q/t/ = q_o \exp - c_o/t-t_o/ \qquad /1/$$

This fact was attributed by local experts mostly to filter clogging until recently.

First Kertész /1976/, then Szilágyi /1977/ recognized that, in addition to filter clogging, the different structure and conductivity of fault zones may be related to the decay of the yield of inrushes and drainage wells. This is shown by the fact that inrushes are connected to fault zones and conductivity of several order of magnitude higher can be calculated from the initial high yield of inrushes and drainage wells. However, water control based on this conception also failed.

2.2 Natural conditions of the Kányás coal reserves

Thick oligocene layers and miocene layers including coal reserves are located on the palaezoic and mesozoic base rock. There are three coal seams in the miocene formation /Fig. 3/, the upper seam I is being mined under the sand aquifer of 15-30 m thick.

Seam I. is directly covered by sandy clay, clay marl of 0-10 m thick, with a unilateral compression strength of ~10 MPa, performing as a protection layer from the overlying aquifer.

The sandy aquifer is covered, after a turf-sandy transition zone, by Helvetian slir of 250~350 m thick with unilateral compression strength of 6~9 MPa. Beneath seam I there is gray sand of 0-7 m thick, then slightly turfy clay of 5-20 m thick, contacting coal seam II.

These formations were fractured by tectonic movements accompanied with vulcanism, and the coal seams were divided into blocks by longitudinal faults of NE-SW direction and transversal faults of NW-SE direction.

Coal reserves are bordered by a fault system of 150~300 m height. Height of faults within this border is mostly less than 40 m. The dip of fault planes is $30 \sim 50^{o}$, that of coal seams is $6 \sim 10^{o}$. According to mining experience the overlying sand of seam I easily collapses. Its unilateral compression strength is 0-1 MPa, much lower than the strength of other clay, marl rocks in the formations. Particle size distribution ranges between 0,1 and 1.0 mm, the effective particle size is about 0,2 mm, uniformity coefficient is 2,1 - 2,3.

Geological and hydrogeological studies over the area and mining operational experience demonstrate that the aquifers drained by mining operations between large faults are closed, and there is no natural recharge either from the surface or other aquifers.

The amount of withdrawals so far corresponds to the static resources of the aquifer /Figs. 2a, 2b/. The temperature of aquifer waters is 25-27^{o} C. They contain small amount of Ca, and high amount of Na, Cl and HCO_3. The highest amount of dissolved gas can be found in aquifer No. 1.

The relative amount of gas is 0,6-0,7 m^3/m^3. Methane, stemming from the aquifer causes that the mine is in the category II of gas hazard.

Piezometric level of the aquifers is related to the depth of the aquifers under the surface, showing that consolidation of the miocene layers is still going on.

Piezometric pressure on the sandy aquifer No. 1 is 25-30 bar, porosity is 0,3 ~ 0,4.

Good drainage properties are indicated by the fact that the sand above mining operation is generally dry.

Average seepage coefficient of aquifer 1 is $2 \cdot 10^{-5} - 3 \cdot 10^{-5}$ m/s, calculated from pumping test. In a loose condition under atmospheric pressure, this sand has a conductivity of an order of magnitude higher /$2 \times 10^{-5} - 4 \times 10^{-5}$ m/s/. If the sand is compacted again and loaded according to the actual stress conditions, conductivity will be much lower /Szilágyi, 1977/.

The average seepage coefficient of the sand is $1,2 \cdot 10^{-5}$ m/s, calculated from inrush data. This shows that in the vicinity of inrushes, mostly along fault zones, conductivity is much higher, that is, aquifers exhibit double porosity. This assumption of Kertész and Szilágyi explains the initial high yield of inrushes but cannot explain the rapid decay of yields.

If the aquifer behaves really as a double-porosity system, the high yield of inrushes along faults would be sustained for a long time due to the considerable amount of static resources. Even the whole resources could be drained by a fortunate drilling. Experience contradicts this assumption.

The double porosity and the higher conductivity of fault zones are contradicted by the low strength of the sand and the general mining experience that sand is further crushing during displacement in the fault zones. If a large displacement happens, clayey and marl enrichment will decrease the condutivity of the sand.

The width of fault zones in the Kányás coal field ranges between some centimeters and 1-2 m and tectonic blocks bordered by faults behave as closed aquifer units.

The closeness of these tectonic blocks is evident for greater fault height but if this height is smaller than aquifer thickness, the closeness is possible only if fault zones behave as aquitards. Now the questions arise:

- how can the apparent higher conductivity of fault zones be explained?
- what kinds of phenomena govern the yield decay of inrushes and wells?

The answer can be found in the experience itself.

2.3. The change of sand conductivity in time

According to the experience larger inrushes coincide with sand inflow. As a result, the sand will be looser and its conductivity higher around inrushes and wells. This is strenghtened by laboratory experiments.

It is also a fact that pore pressure will change first for a smaller then an increasingly greater rock volume. As singly piezometric pressure $H \varrho_v g$ decreases on the top of the aquifer, the effective main stress will increase in the aquifer after a certain time. An intensive secondary consolidation starts, leading to settling, repeated compaction of the loosened sand and further compaction of intact parts.

Fig. 4. illustrates these processes. The upper part /Fig.4a/ shows the time function of discharge

$$q(t) = q_o e^{-c_o t} \qquad /2/$$

The middle part /Fig. 4b/ shows main stress σ_z in the close vicinity of the inrush and Fig. 4c. shows the seepage coefficient approximated on the basis of laboratory experiments and theoretical considerations as

$$k(\sigma) = k_o e^{-\beta \sigma z} \qquad /3/$$

It is also an experimentally known fact that the creep or compression of clayey-marl rocks is approximately exponential for constant loading. Thus for the slir layer:

$$\sigma_z(t) = \sigma_{zo}\left(1-e^{-\gamma t}\right) \qquad /4/$$

Since for constant drawdown s_o, $qt = A\ k$ and $q_o = A\ q_o$, the following expression can be written using Equations /2/, /3/ and /4/.

$$e^{-c_o t} = e^{-\beta\left[\sigma_{zo}\left(1-e^{-\gamma t}\right)\right]} \qquad /5/$$

Here $e^{-\gamma t}$ is substituted by the first linear term of its Taylor-series

$$c_o \approx \beta\gamma\sigma_{zo} \qquad /6/$$

Since $c_o = 2{,}3\ tg\alpha$ and $\beta = 1/M$

$$tg = 0{,}433\ \frac{\gamma\sigma_{zo}}{M} \qquad /7/$$

That is, the decay time of inrushes is:

$$T = 2{,}3\frac{M}{\gamma\sigma_{zo}} \qquad /8/$$

where β is the coefficient of volume compression of the sand, M is the compression module of the loose sand, γ is the relaxation coefficient of slir and $tg\alpha$ refers to the straight line in Fig. 2d.

Equation /8/ indicates that the decay time depends solely on rock parameters. Since these are quasi constant, the decay time is also constant; this is in accordance with experience.

Since average values of k_o and k are known

$$\beta = \frac{1}{\sigma_{zo}-u}\ \ln\frac{k_o}{k} \qquad /9/$$

and the seepage coefficient k pertaining to total compaction is

$$k = k_o\ e^{-\beta\sigma_{zo}} \qquad /10/$$

The estimated value of $k = 6 \cdot 10^{-7}$ m/s, which is one quarter of the k measured in natural condition.

In conclusion, the whole process of loosening and compaction of the Kányás sand results in a change of 20 times in seepage coefficient.

In addition to this compaction, clogging may also contribute to yield reduction especially for smaller inrushes.

The yield reduction can be controlled - instead of filter change - by preventing initial sand inflow and regulating the rate of drainage.

Such control possibilities are discussed in the next section.

2.4. Development of control possibilities

The main precondition of efficient control is that drainage precede exploration.

At present, the method is subsequent as far as exploration and mining preparation are concerned. Roadways for the above two purposes are driven under water inrushes, sometimes crossing aquifer I. It is thus natural that exploration difficulties caused by water will last despite drainage at roadway heads. As a result, these works of paramount importance for mining will extremely slow down.

This difficulty can be eased only if aquifers 1 and 2 are drained prior to the driving of roadways for seam I from the surface or by wells placed in a water tunnel system at a lower level without water hazard /Fig. 5./.

In principle, one can construct a drainage system from the surface. However, the following problems call for an underground drainage system. Even the average initial yield of a drilled well will not be higher than 100-200 l/min, the total flow will be minimum 1,5-2,0 m³/min with gas content and the average pumping height will be 400 m. Also, the gradual decrease of piezometric head would lead to yield reduction.

In contrast to the present practice, piezometric head reduction and partial pre-dewatering of aquifers 1 and 2 are recommended by main exploration roadways and water cuts: driven in seam II and equipped by special filters.

Technical preconditions of proper filter performance are to construct the filter in the protection of grouted direction pipe through a closed pipe head and to equip with a valve and pressure gauge /Fig. 6./.

Natural condition of placing grouted direction pipe is to have an aquitard of at least 5-10 m thick between the aquifer and drainage roadway. This would require, in turn, to drive the drainage roadways in the underlying rock of aquifer 2, such as the spoil above seam II.

A further important prerequisite of proper pre-dewatering is that the driving of water roadways, the placing of drainage drillings and drainage operation precede mining exploration. Also, a lead time is necessary for drainage between the start of dewatering and that of exploration.

Since piezo-conductivity of the aquifers is small, pressure reduction and dewatering require a relatively long time. This time will be expecially long if the drainage system is not properly planned.

The inclined Kányás aquifers are drained from top to the bottom. As a consequence, retreat mining is recommended in the incline above and advance mining in the incline under the main exploration roadway, following dewatered rock boundary /Fig. 7./.

As far as mining is concerned it is required that the velocity in dip direction of drainage boundary line should be greater than the mining rate.

$$v_v \gtrsim \frac{G}{Lm \varrho g /1-\eta/}$$

where G is the annual production, L is mining length in bedding direction, ϱ is coal density, and η is the average mining loss.

It is evident that the smallest dewatering velocity for a given production G is reached if hydrogeologically uniform tectonic blocks are mined in the total bedding length. For a given shaft, the smallest dewatering rate corresponds to simultaneous mining above and under the main exploration roadway.

In addition to gravitational dewatering, compressed air dewatering and vacuum dewatering are also possible, in principle.

Compressed air dewatering may be used for accelerating the dewatering of smaller units such as the Balinka mine. Here, the volume to be dewatered must be closed. In the overlying rock such as aquifer 1, air will escape freely after a relatively fast piezometric head reduction, and air compression will be inefficient.

Considerable efficiency improvement cannot be expected from vacuum dewatering due to the gas content.

At present the above recommended gravitational dewatering is being used. Thus, the task is to improve the operation of the system.

Such a control system is necessary which assures the protection of inclines and exploration, preparation of mining and mining itself, and satisfies the conditions discussed.

Since regional dewatering cannot be used efficiently, a three-step system is recommended in harmony with mining operation.

The first step serves the protection of exploration /Fig.5./ Tasks: pressure reduction and dewatering in the vicinity of exploration roadways, detection of faults to be crossed, preparation of safe crossing of faults.

The second step serves the protection of mining preparation and mining, while the third step fulfils safe dewatering prior to mining.

The first step is constructed from the exploration and water roadways driven in the spoil above seam II /Fig. 5./. Drainage covers three aquifers /1, 2, 3/.

For avoiding soil failure pressure reduction is necessary in sand 3 in the protection of water roadways. Pairs of inclines under and above the main exploration roadways provide for the local dewatering of aquifers 1 and 2 and the conditions of driving roadway pairs in the coal seam.

The second step is developed from the first one, starting from the water roadway beneath the seam and following the bedding line of the lower fault bordering the block.

In the third step the remaining water is pulled out prior to mining. For this purpose PVC filter wells of 3-4 m long, and 5-10 m apart are planned in the roadways of mining preparation. Filtering is envisaged even where the protection layer is thin or missing.

This recommended system is under construction. It is hoped that fruitful mining will be possible in Kányás without water problems.

As long as overall problems were caused in Kányás by the inefficient performance of drainage wells, the same defect resulted in local problems in driving the Bito incline especially at the final phase of the constrcution.

3. Dewatering experience from the Bito inclined shaft

Studies on the sinking of the Bito inclined shaft go back to 10 years ago when the conception of Bito deep-mining was developed. The investigation started by geohydrological and soil-mechanical exploration performed by the Bureau of Surveying and Soil Testing /FTI/. [6, 7].

Next, the Mining Research Institute /BKI/ using previous results and a broader regional geological study of the area, suggested that over the upper, some 200 m thick section of the formation, water level lowering is necessary to protect the inclined shaft [8] .

At the same time, in 1974, the Design Company for Roads and Railways studied several alternatives [9] such as

- cut-off wall,
- grouting,
- water level lowering.

Concerning the third alternative, the company considered uncertain to attain the necessary dewatering effect by wells operated with submersible pumps.

All the same, two alternatives were designed: freezing and water level lowering.

The freezing design was prepared by the Mining Department of the Miskolc University of Heavy Industry and the Shaft Sinking Company [10] and water level lowering was designed by BKI [11] .

The mining company selected the water level lowering alternative and the construction started in 1977. The author participated in this work as the chief designer and construction superviser cooperating with experts of the company; L. Rumpler, M. Tatai and G. Pocsai.

In this section some important construction experience is given with special regard to well deterioration.

3.1. Natural conditions of the inclined shaft

The upper 200 m deep section of the inclined shaft crosses two aquifers /Fig. 8a/.

The upper pleistocene aquifer /1/ of sand, partly loam and gravel with clay lenses is 20-50 m thick. Greater heterogeneity occures at the upper 5-7 m depth. The deeper, cohesionless sand is quite uniform, with particle sizes of 0,1-0,2 mm, coefficient of heterogeneity of 1,7-3,0, average seepage coefficient of $3,5 \cdot 10^{-5}$ m/s, inner friction angle of 25-30°, In a dry state, the sand is stable, as shown in the adjacent Fehérvárcsurgó open-pit mine where steep slopes of 15-20 m high have been formed.

In submersible state the sand immediately collapses, and in saturated state it will be liquidated. This is a quick sand according to FTI analysis [6].

Between aquifers 1 and 2 a clay aquitard of 12~15 m thick is situated with high organic content.

The lower aquifer 2 of 5-7 m thick consists of fine sand with an average seepage coefficient of $1{,}5 \cdot 10^{-5}$ m/s.

The bed rock beneath aquifer 2 is a hard material.

The unconfined water level in aquifer 1 is 2,5~2,7 m under the surface, the piezometric level of aquifer 2 is 4,5~5,5 m under the surface.

It was first endevoured to construct the head of the inclined shaft by pumping from the pit. However, soil failure occurred in a depth of 3-4 m under water level. This showed the necessity of a proper protection.

As a consequence, a two-step water level lowering system was designed with combination of vacuuming [11].

3.2. The system of water level lowering

The first step of the water level lowering system consists of the combination of pumped wells and vacuum wells sunk from the surface /Fig. 8A and b/.

The second step of the system starts from the head of inclined shaft as shown in Fig. 8c.

The provisional vacuum subsystem is required because of topsoil heterogeneity and the adverse experience of initial dewatering.

The system was sized according to the following approximating expression for the joint yield of gravitational and capillary water /Schmieder, 1975/ [12].

$$q = \frac{\pi k (H'^2 - h^2)}{\ln \frac{R}{r_v}} + \frac{2\pi k\, h_1 \left[s_o + \frac{p_o - p_b}{\varrho_v\, g} \right]}{\ln \frac{R}{r_o}} \qquad /12/$$

Under given technical possibilities, the sizing of vacuum well dewatering has no great significance.

In equation /12/ q is the yield of a vacuum well, k is the seepage coefficient, H' is water level height above the closing layer, h is the height of the remaining water level, s_o is drawdown: H'-h,

$$h_k \approx \frac{4 \cdot 10^{-3}}{\sqrt{k}} \qquad /13/$$

is the height of the capillary zone, p_o is atmospheric pressure, p_v is the pressure within the vacuum well, r_v is the radius of the vacuum zone around the well and r_o is the radius of the vacuum well.

The calculation resulted in 40-50 vacuum wells 2 m apart.

The necessary number of water wells for step 1 was calculated as the solution of the linear equations

$$Aq = \Phi \qquad /14/$$

expressing the algebraic sum of velocity potentials. A safety factor of 1,2 was used.

Initial data and results of the calculation are given by the table in Fig. 8b.

Since the average discharge of wells for aquifer 2 was calculated as 20~30 l/min, joint wells for both aquifers were designed /Fig. 8a/.

The design of wells is illustrated in Fig. 10a. Gravel filter was used. There is an impermeable layer between gravel filters for aquifers 1 and 2 in order to permit communication between the two aquifers within the well.

A filter tube made of metal filter frame and equipped with plastic sieve cloth /Fig. 10c./ supports the gravel filter.

After well construction multi-step cleaning compression was prescribed, then the submersible pump was located, finally the well was set into operation.

The duration of the construction of the whole system was round 3 months.

3.3. Operational experience

The vacuum-well dewatering of the inclined shaft head was realized as planned. As a result, no disturbance occurred during the contruction of the head.

The water-well subsystem was constructed also as planned. Time functions of water level and discharge are shown in Fig. 9A, time function of shaft sinking can be seen in Fig.9b.

Water level lowering was effective to a depth of 15 m under the surface: the sandy aquifer was dewatered and a uniform driving velocity was achieved along an inclined length of 59.4 m. In this point, the remaining peizometric level was reached 5 m above the bottom of the upper aquifer. Now, the second step of water level lowering was set into operation, and the discharge increased /Fig. 9a/ and the water level in front of the head gradually decreased.

From this second step on sinking velocity gradually decreased /Fig. 9b/, and great efforts were needed to sink the inclined shaft down to the clay aquitard.

In this last phase, even the smallest disturbance of operation such as a short failure of a pump caused soil failure.

In order to cope with this situation the highest possible discharge was effected, for a delayed time. As a consequence, the state of wells deteriorated /Fig. 10b/. The lower part of wells was more or less filled with loamy sand mixed with filter gravel originating from sieve cloth failure.

Sinking experience of the Bito inclined shaft can be summarized as follows:

- Water level lowering is a reliable and economic means of water control within its zone of application. At the end of this zone, such as just above an aquitard where there is a water dome due to energy reasons, the method is efficient only if the aquifer has some cohesion. As an example, the eastern inclined shaft in Nagyegyháza can be mentioned where several sandy aquifers were transversed without any disturbance in the protection of water level lowering [13].

- In cohesionless quick-sand, this critical part can be overcome if the gravitational drainage system is absolutely reliable /Fig. 10d/. Since this solution is very expensive, a combination of water level lowering and soil cementation /e.g. the SOLETANCHE method or local freezing/ can be used. In the latter case, the necessary cohesion of the loose soil part saturated by the water dome partly or totally stems from advance soil cementation.

- The application of filter tubes, equipped with plastic sieve cloth - provided statical conditions are met - is efficient only if dynamic loading is small. As the example of Fig. 10a shows, the dynamic water level was fluctuating always under the bottom of the aquifer. As a result, the upper filter worked without any dynamic loading and so without disturbance.

Thus, under such conditions, plastic sieve cloth can be used in order to decrease well resistance to a minimum.

If considerable dynamic loading can be reckoned with, such as in layer 2 in Fig. 10a, it is more advisable to apply metal sieve screening or a filter without sieve.

References

[1] Vigh, F. - Darányi F. - Szebényi F.: Hydrological conditions and principles of water control in the Kányás mine. BKI Research Report, Budapest /1957/

[2] Schmieder, A. - Vigh F. - Darányi F.: Hydrological conditions and guidelines of the control of water and gas in the Kányás mine. Proceedings BKI, No. 2. /1962/

[3] Demeter, F. - Fényes Gy. - Schmieder A. - Verpeléti E.: Solution to engineering and economic problems facing the Kányás mine. Competition study, Second Prize winner Salgótarján, Hungary /1980/

[4] Kertész, R.: Hydrogeological properties of the Kányás coal reserves. VII.Conference on minewater control /1976/

[5] Szilágyi, G.: Possibilities of controlling the water hazard in the Kányás mine. BKI Research Report, Budapest /1977/

[6] Majnes, A.: Hydrogeological investigation of Bito shaft II. Kincsesbánya, FTI Report, Budapest /1974/

[7] Majnes, A.: Hydrogeological study of group wells for Bito shaft II, Kincsesbánya, FTI Report, Budapest /1974/

[8] Schmieder, A. - Havasy I. et al.: Anylysis of bauxite mining in Bito II. BKI Research Report I. Budapest /1974/

[9] Posgay, Gy.: Engineering description of mine opening in Bito II. UVATERV, Budapest /1974/

[10] Mining Department of NME: A freezing technology for the sinking of the inclined shaft Bito II. Manuscript /1974/

[11] Schmieder, A. - Bagdy I. - Havasy I.: Water level lowering for the upper 200 m part of inclined shaft Bito II, BKI Report, Budapest /1977/

[12] Schmieder, A. - Kesserü Zs. et al.: Water hazard and water management in mining /Edited by F. Martos/ Müszaki Könyvkiadó, Budapest /1975/

[13] Mrs Geszler, Ö.: Dewatering plan for the inclined shaft Nagyegyháza East. BKI Report, Budapest /1977/

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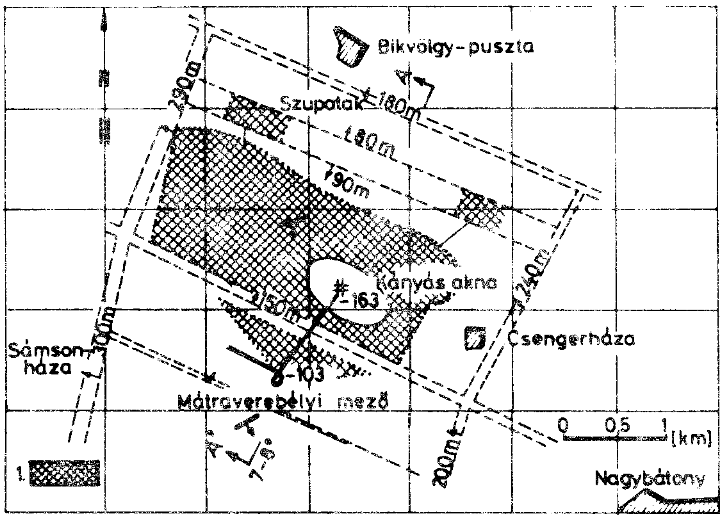
a,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180m
790m
290m
Kányás akna
-163
150m
-103
240m
Csengerháza
Sámson-háza
300m
Mátraverebélyi mező
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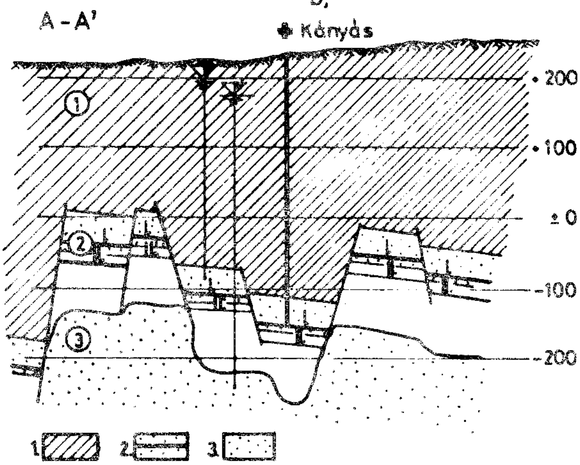
A'
7-8°
1.

b,


A - A'
Kányás
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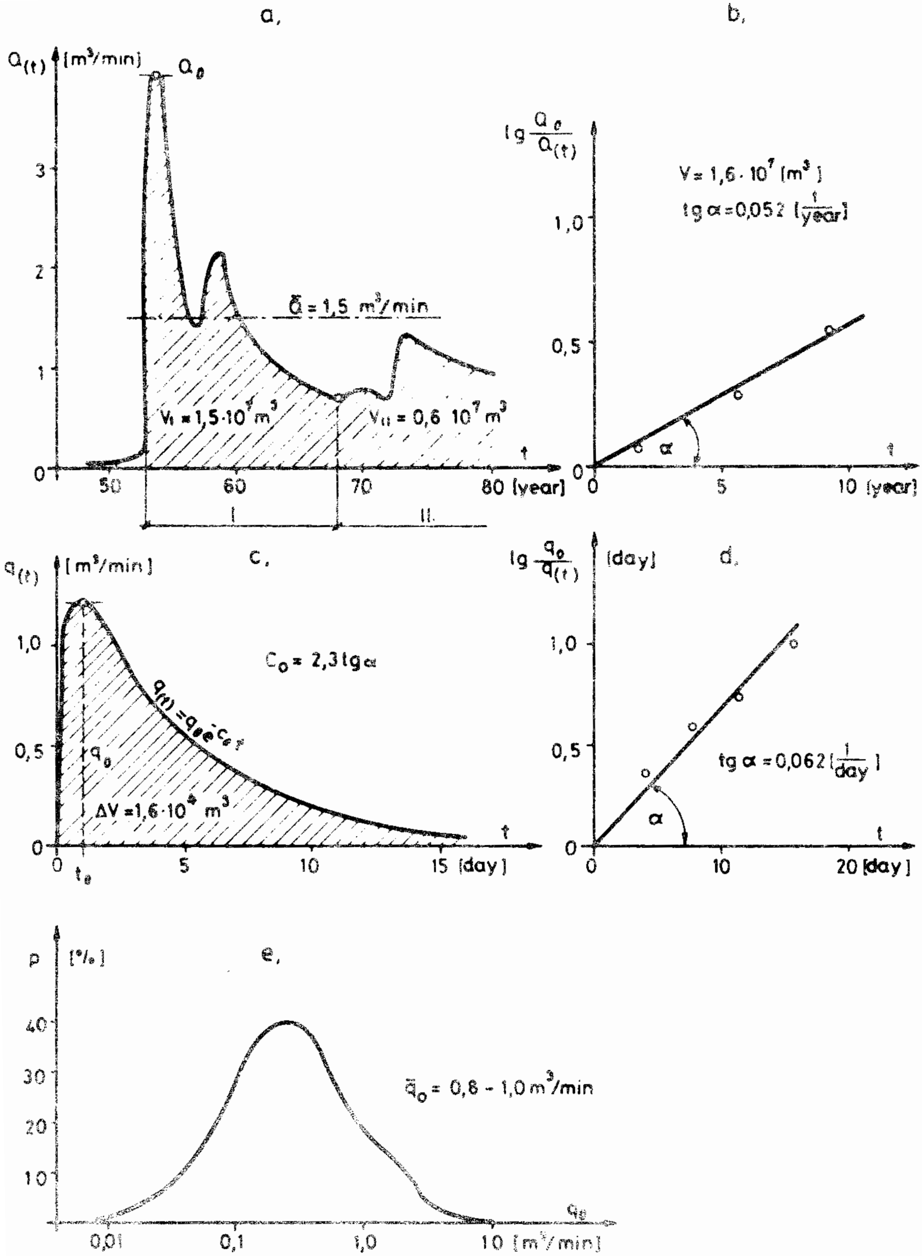
100
± 0
-100
-200
1
2
3

Fig.1.ábra


a,
b,
$Q_{(t)}$ [m³/min]
Q_0
$\bar{Q}$ = 1,5 m³/min
V_I = 1,5·10⁷ m³
V_{II} = 0,6 10⁷ m³
80 [year]
I
II.
lg $\frac{Q_0}{Q_{(t)}}$
V = 1,6 · 10⁷ [m³]
tg α = 0,052 [1/year]
10 [year]
c,
d,
$q_{(t)}$ [m³/min]
C_0 = 2,3 tg α
$q_{(t)} = q_0 e^{-C_0 t}$
q_0
ΔV = 1,6·10⁴ m³
15 [day]
t_0
lg $\frac{q_0}{q_{(t)}}$ [day]
tg α = 0,062 [1/day]
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e,
p [%]
$\bar{q}_0$ = 0,8 - 1,0 m³/min
10 [m³/min]

Fig. 2. ábra

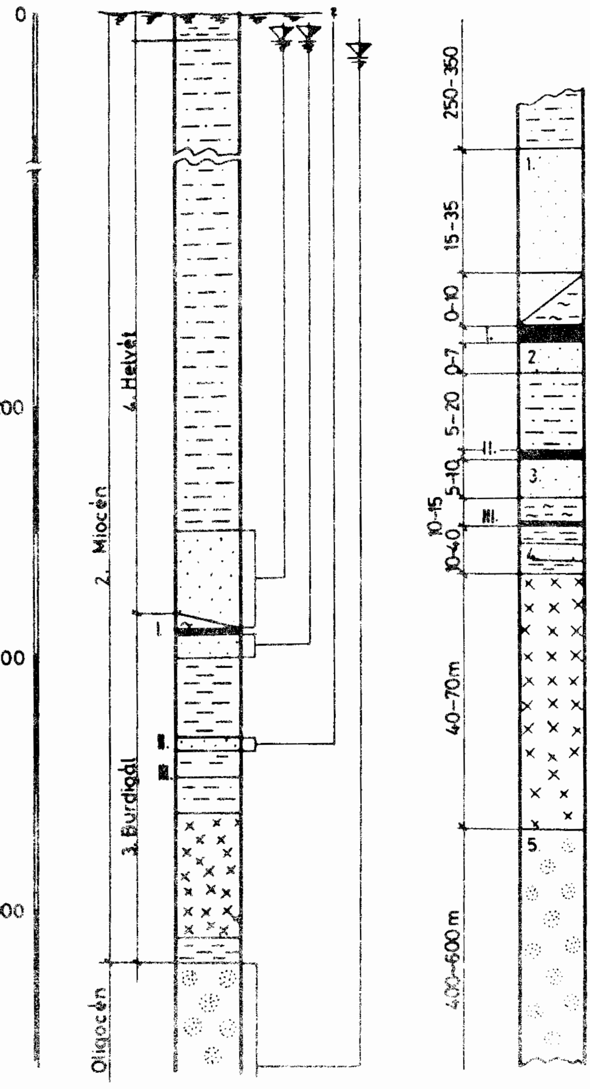

0
200
300
400
1. Oligocén
2. Miocén
3. Burdigal
4. Helvét
250-350
15-35
0-10
0-7
5-20
5-10
10-15
10-40
40-70m
400-600m

Fig 3. ábra

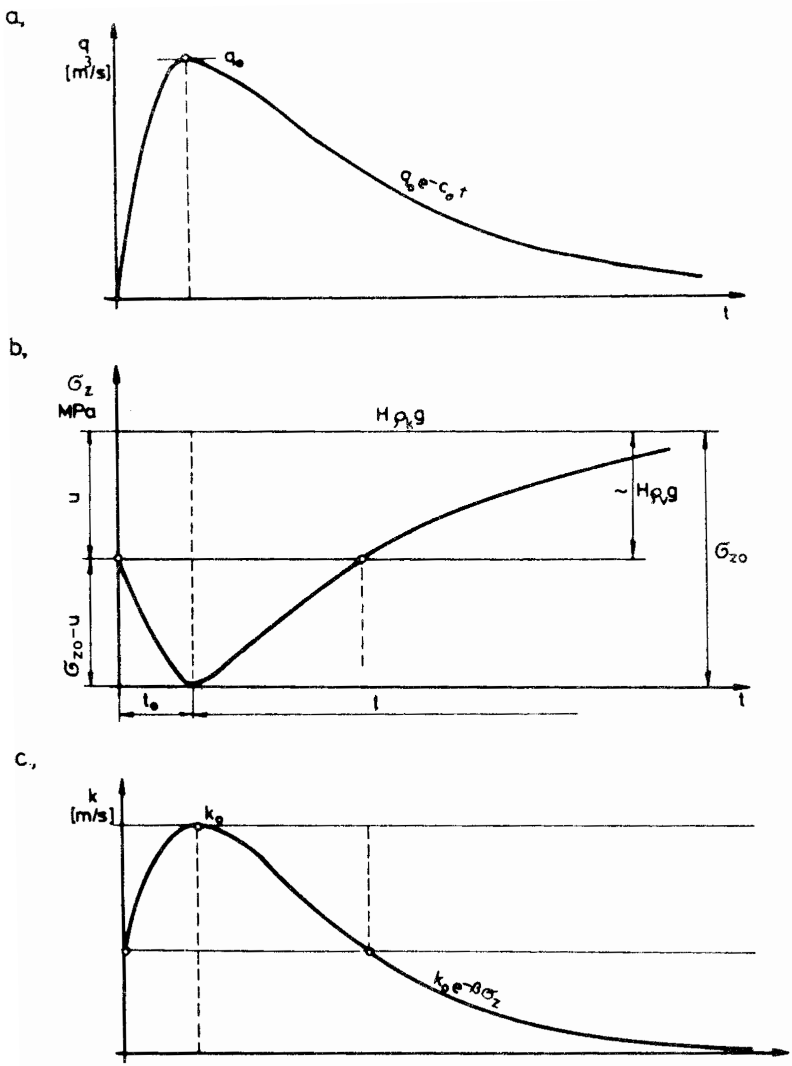
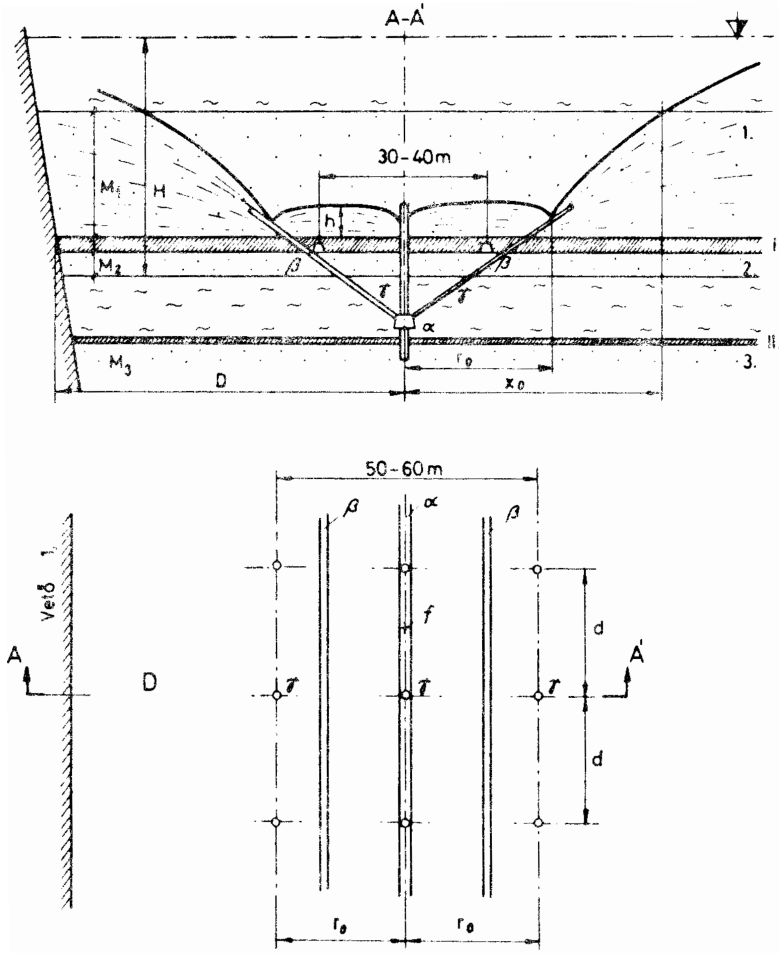

a,
q
[m³/s]
q_o
$q_o e^{-c_o t}$
t
b,
σ_z
MPa
$H \rho_k g$
u
$\sim H \rho_v g$
σ_{zo}
$\sigma_{zo} - u$
t_o
t
t
c.,
k
[m/s]
k_o
$k_o e^{-\beta \sigma_z}$

Fig.4. ábra



1. α vizvágat
2. β sikló v. ereszke pár
3. τ Kassai-Halász féle szűrő (τ)
4. +f vizszintészlelő főteszűrő (f)

Fig. 5 ábra

1. a, b, és c,
Kassai-Halász féle fém szűrő, tolózárral és feszmérővel felszerelve. Beépítése zárt.

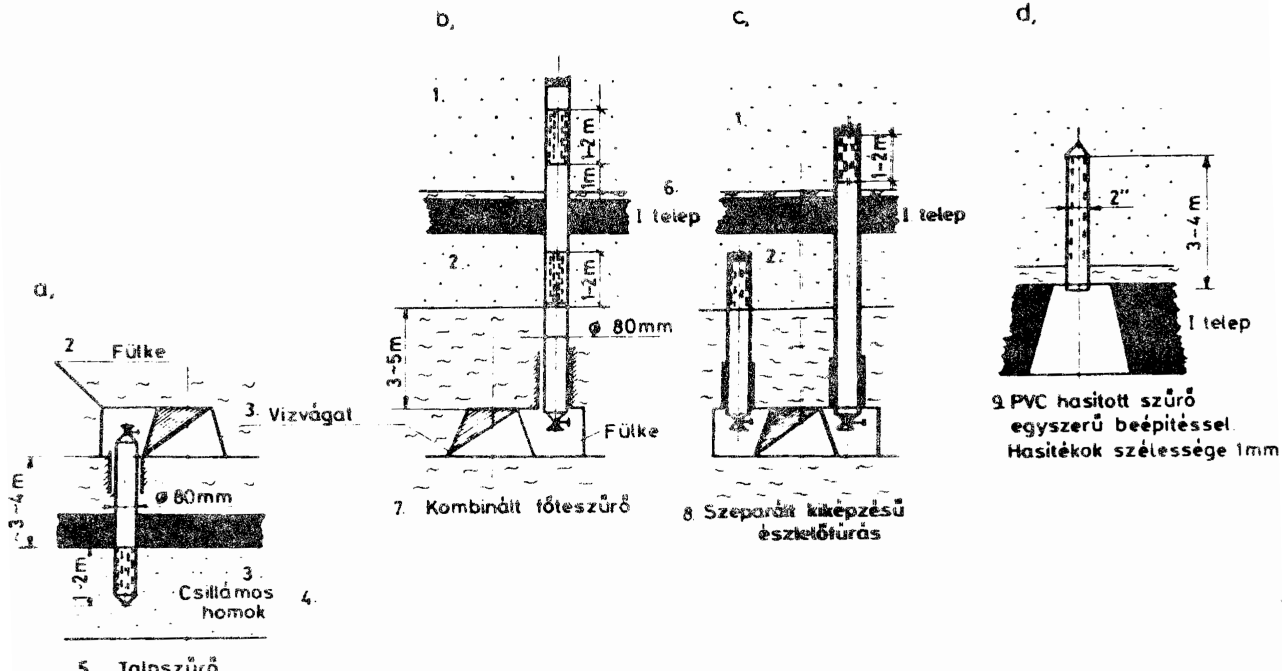


Fig. 6 ábra

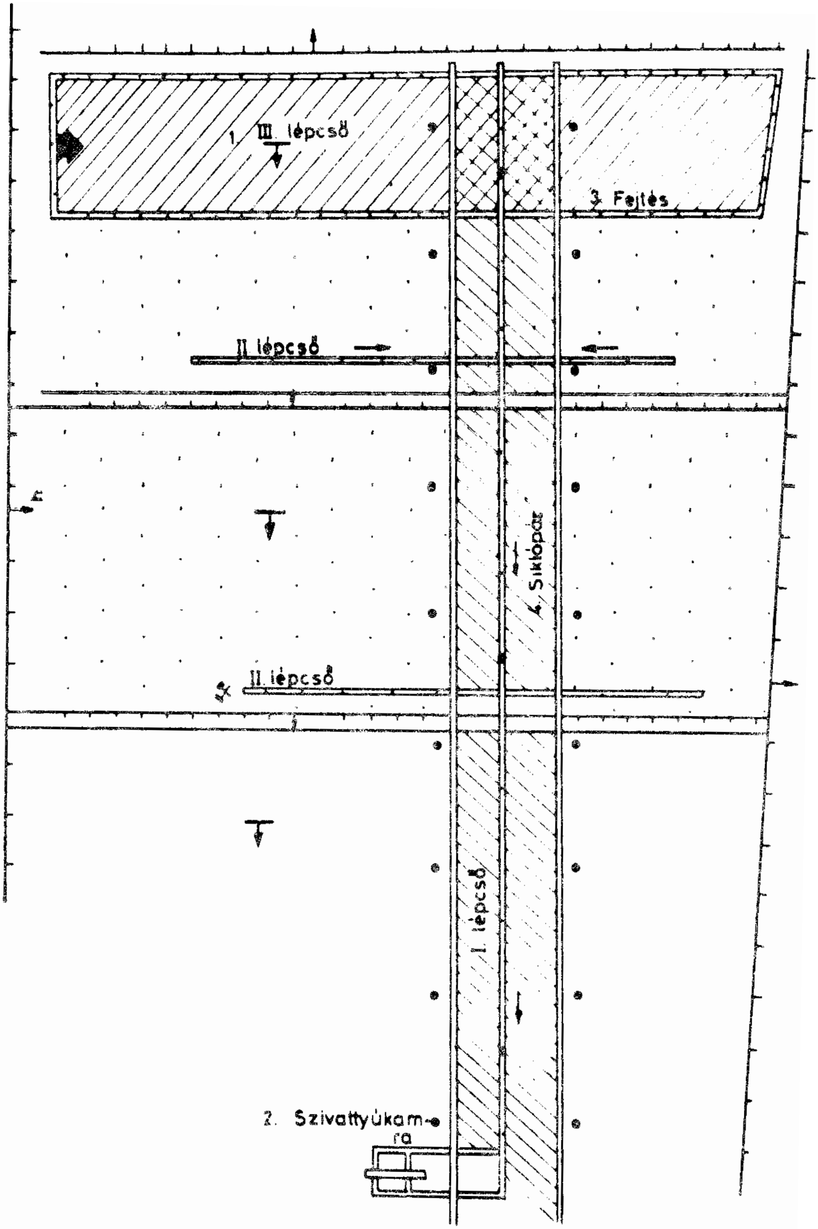
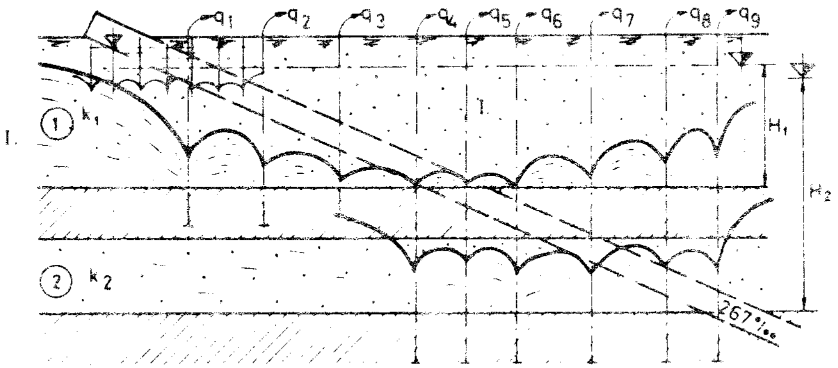
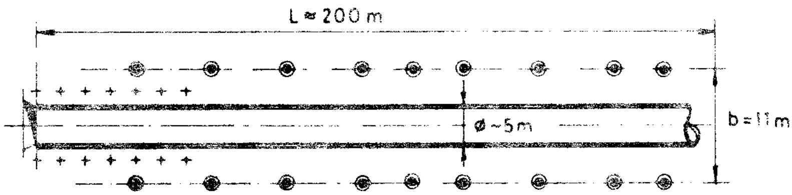

1. III. lépcső
3. Fejtés
II. lépcső
4. Siklópár
II. lépcső
I. lépcső
2. Szivattyúkam-ra

Fig. 7 ábra

a,



b,



—	H	M	n_0	k	a_0	Q	N
	m	m	—	m/s	m^2/s	m^3/min	db
1	20	20	0,15	$3{,}5 \cdot 10^{-5}$	$5 \cdot 10^{-3}$	1,3 ~ 1,5	13 - 14
2	42	6,5	0,10	$1{,}5 \cdot 10^{-5}$	$2{,}2 \cdot 10^{-3}$	0,3 ~ 0,5	16 - 18

c,

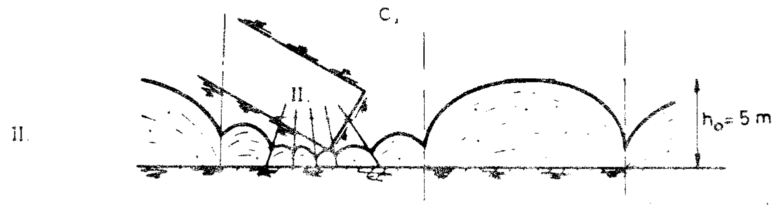
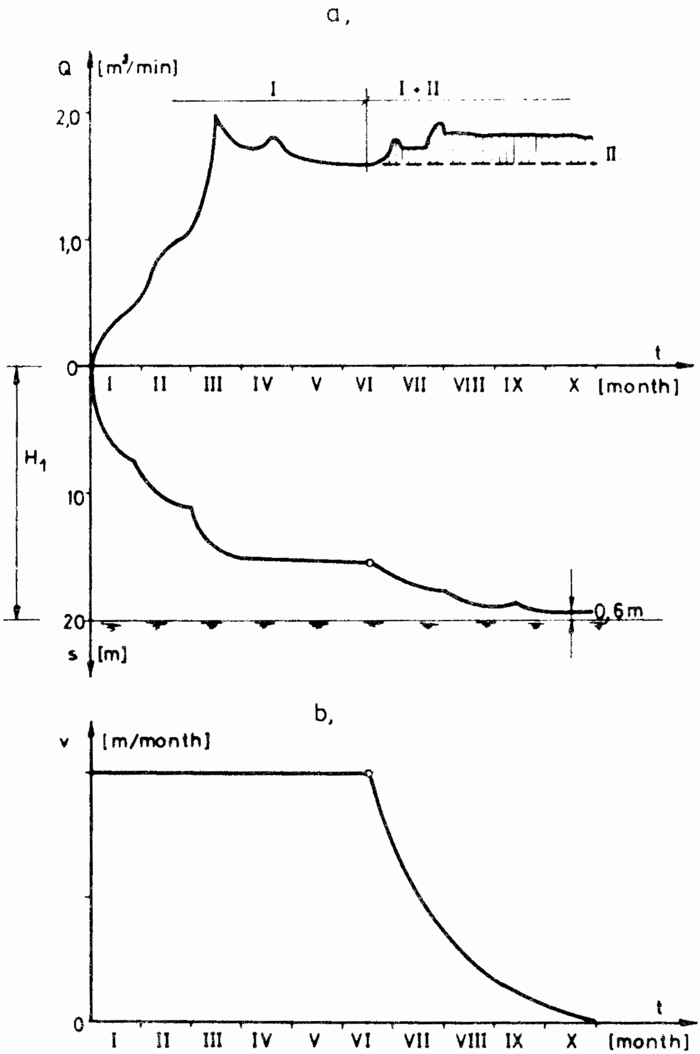


Fig 8 ábra



a,
Q [m³/min]
I
I + II
II
2,0
1,0
0
t
I II III IV V VI VII VIII IX X [month]
H1
10
0,6m
20
s [m]
b,
v [m/month]
0
t
I II III IV V VI VII VIII IX X [month]

Fig. 9. ábra

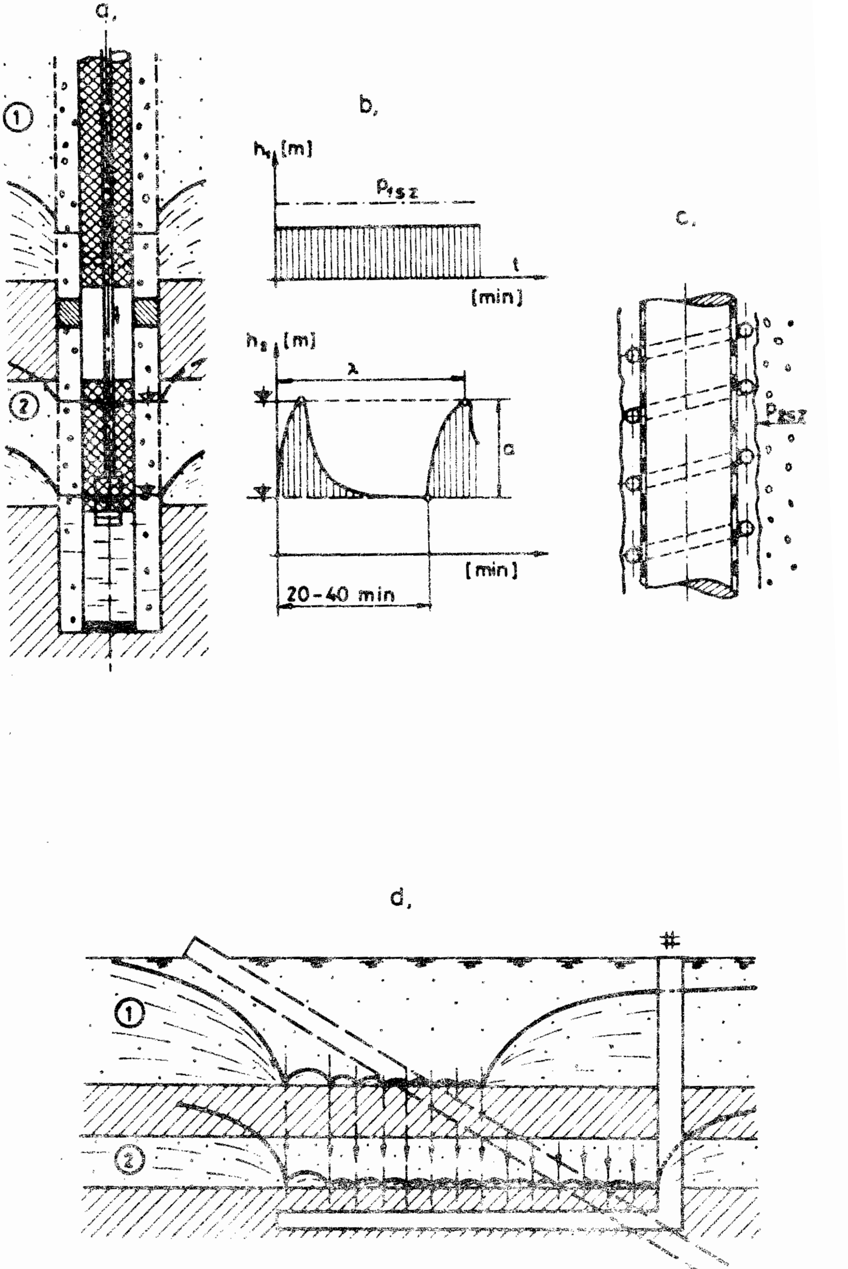


Fig. 10. ábra