

A COMPUTER SIMULATION OF THE DISTRIBUTION OF GASEOUS CONTAMINANTS IN MINE VENTILATION NETWORKS

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GAZ DAĞILIMININ BELİRLENMESİNDE BİR
BİLGİSAYAR BENZETİŞİMİ

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ABSTRACT

The object of this paper is to describe the development and application of a computer simulation program developed at the Mining Engineering Department, Nottingham University. This program allows the mine ventilation engineer to predict the distribution of the steady state gaseous pollutant concentrations to be expected around a mine ventilation network when alterations to the ventilation system or production cycle of a mine are proposed. This information can assist in the correct interpretation of the signals received from any installed environmental monitoring transducers.

ÖZET

Bu makale yazarlar tarafından Nottingham Üniversitesi, Maden Mühendisliği Bölümünde geliştirilen bir bilgisayar programını açıklamayı amaç edinmiştir. Maden havalandırma sisteminde veya üretim aşamasında herhangi bir değişiklik planlandığı durumda, sözü edilen bu program havalandırma mühendisinin, maden havasındaki zararlı gazların konsantrasyonlarını önceden tahmin etmesine yardımcı olur. Bu da dolaylı olarak yeraltındaki monitörlerden gelen sinyallerin doğru yorumlanmasına katkıda bulunur.

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1. INTRODUCTION

The main objective of a ventilation system is to deliver an adequate quantity of fresh airflow to the working areas of the mine. The required airflow should be at the correct velocity and volume flowrate to ensure that any harmful gaseous pollutants are dispersed rapidly and diluted to below prescribed safe general body concentration levels.

The exploitation of deeper reserves and the use of intensive mechanisation methods to obtain higher productivity levels has placed, an increased burden on the mine ventilation system. The success in the design and safe operation of ventilation systems to tackle this problem has been due, in part, to the development and introduction of reliable on line environmental transducers.

To assist in the determination of the correct position of the monitoring transducers within the mine ventilation network and to aid in interpretation of their signals, the mine ventilation engineer should have a knowledge of the major emission sites and rates, and the resulting steady state gaseous pollutant concentration levels to be expected throughout the mine ventilation system.

2. THE PRESENT PROGRAMS AVAILABLE FOR AIRFLOW DISTRIBUTION AND GAS PREDICTION

The main purpose of ventilation analysis is to assist the ventilation engineer in producing a satisfactory solution to an existing or projected ventilation problem .

There are many industrially proven programs available (Penn State, Hartman et.al., 1982; PENVEN, Anderson et.al., 1978; MVNA, Barnes, 1976 and VNKT, McPherson, 1964). This class of programs may be used in conjunction with underground ventilation surveys and/or projected airway design data to assist in the ventilation planning of any mine. The VNET program has been employed by the authors to obtain the airflow-pressure patterns for any given mine ventilation system.

There are a number of prediction methods for gas emission in mines such as firedamp (MRDE, 1982) and diesel (Bandopadhyay, S. et al., 1983) etc. A summary of other firedamp prediction methods may be found in the review article by Curl (1979).

The gas distribution program which is described in a latter section, employs specific site data together with a suitable prediction method to determine the pollutant emission rates and subsequently calculate the steady state gaseous concentrations of pollutants around the given mine ventilation network.

3. MONITORING OF THE GASEOUS POLLUTANTS

Over the years, extensive research has been conducted into improving the safety of underground mining operations. This has led to a steady expansion in the number of, and installation of, environmental monitoring and measuring instruments within mine ventilation systems. The electrical signals produced by the devices are transmitted to an intelligent outstation or central station where they are evaluated and subsequently recorded.

The first generation of transducers measured methane concentration and air velocity. However, there has been a rapid growth in monitoring technology which has produced a wider range of more reliable monitoring transducers which can measure general body concentrations of oxygen, carbon monoxide and carbon dioxide. The use of these devices is now becoming more widespread ensuring that the quantity and quality of the ventilation air is maintained at working places. They are also used for early recognition of spontaneous combustion and fires.

The program developed by the authors can be used to identify the correct locations of environmental transducers within the mine ventilation network. To do this the mine ventilation engineer needs to know both the sites and rates of emission, as well as the steady state general body concentration levels that are expected for the whole ventilation system, in order to compare these concentrations with the measured values. In cases where a considerable discrepancy occurs between the measured and predicted general body concentration,

The A. D. S. R. B. ...

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...

...

the gas ...

the air ...

emissions ...

static and dynamic

stationary

...

Whereas ...

diesel locomotives ...

only static emission source

The formula used to ... concentration is given below

$$C = \frac{Q_c}{V} \times 10^6$$

Where

C is the general ...

Qc is the volume ...

q is the volumetric ...

5.1. The Algorithm of The Program SPLIT

Before the algorithm is described, the input data required from the user are detailed below. The information stored in a ordered data base is :

- branch number,
- beginning junction number,
- end junction number,
- airway type, i.e. -1 for returns, 0 for emission sites, 1 for intakes
- fresh airflow volumetric flowrate

The contaminant volumetric flow rates for each emission site and the total number of branches are to be entered by the user.

A flag number is given to each branch starting from any emission site to the upcast shaft in a sequence which is increased by one after each labelling. This forms an ordered path of branches. Similarly, a subflag number is assigned to each of branches in the path defined above. This indicates the number of splits in the concerned path.

The algorithm of the program SPLIT is given below :

- 1- Start
- 2- Find next emission site, set the counter $n=1$
- 3- Search for the airways leaving this branch site and label them with the updated flag number $n=n+1$. In addition label each of the branches leaving this junction with a subflag number i.e $k= 1,2,3 \dots$
- 4- Calculate the volumetric flowrate of contaminant for each of these branches with respect to their volumetric airflow rate. If there is only one airway leaving this branch, assign the volumetric contaminant flowrate from the previous branch.
- 5- Choose the airway with the largest subflag number and repeat steps 3 to 5 until the surface reference junction has been reached.

6- Search through the column storing the flag numbers. Discover the largest flag number n in this column and check the subflag number in the corresponding column, if it is 0 or 1 it means that it has already been calculated, therefore search for the next flag number with the value of $n=n-1$. Then check the corresponding value in the column storing the subflag numbers. If the concerned subflag number is not 1 or 0 set the flag and subflag values to 0. And then look for the branch holding the flag number with the value of n and the subflag number with the value of $k=k-1$. After this has been done direct the procedure to step 3.

For the case where all the subflag numbers are either 0 or 1, it means that the surface reference junction number has been reached by following the path from the emission site.

7- Search for any more emission sites. If there is, go to step 2 otherwise perform step 8.

8- Using the prescribed airflow quantities and determined pollutant volume flowrates, calculate the general body contaminant concentration levels in all the airways.

9- Print out the results

The concentrations of these contaminants are printed out in terms of general body concentrations and Threshold Limit Values (TLV). Although the general body concentration of a contaminant is expressed as a percentage, the TLV of a contaminant is described either in p.p.m. (parts per million) or mg/m^3 . The program also utilises appropriate warnings. These are the legislative limit of concentrations for CH_4 , CO, CO_2 , NO_2 and H_2S in the U.K. (M&QA,1954; 1980), and are stored in the program as defaults. If the calculated concentrations approach or exceed the limit the warnings are given at three levels.

The 3 levels, at a concentration, C (%), are

- level 1 (*) where $1/3 < C < 2/3$ of the limit
- level 2 (**) where $2/3 < C < \text{limit}$
- level 3 (***) where $C > \text{limit}$

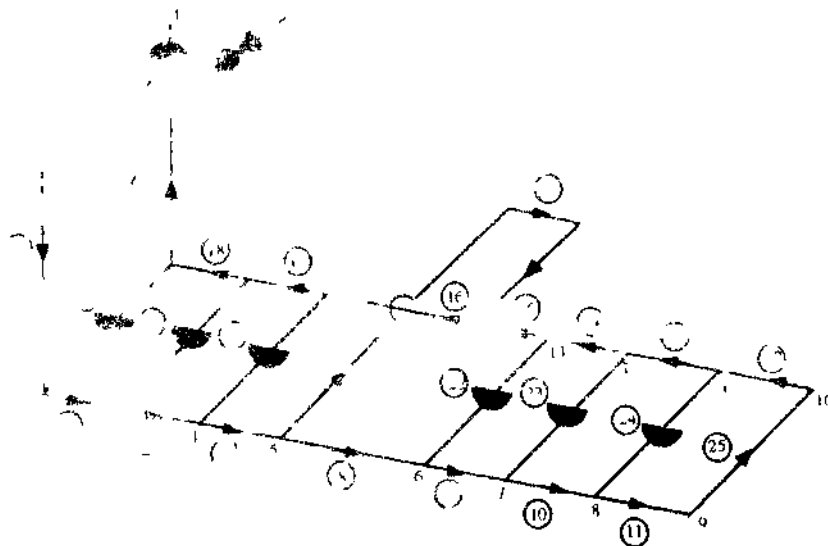


Fig. 1. 100-MHz Vibration Network
(after Jones 1987)

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Appendix I

CIVIL AIRCRAFT DISTRIBUTION PROGRAM
 ***** SPLIT *****
 DEVELOPED AT MINING RESEARCH LABORATORIES
 BY COLONEL G. G. GOUT
 *** UNIVERSITY OF NOTTINGHAM ***
 1988

 INITIAL VALUES OF CONTAMINANTS MEASURED

| BRANCH NUMBER | CO (MG/S) | CH4 (MG/S) | CO2 (MG/S) | NO2 (MG/S) | H2S (MG/S) |
|---------------|-----------|------------|------------|------------|------------|
| 25 | 0 0001900 | 0 0000000 | 0 0650000 | 0 0000100 | 0 0000700 |
| 26 | 0 0002000 | 0 1900000 | 0 3000000 | 0 0000650 | 0 0001300 |

| BRANCH NUMBER | UM FLIGHTS FROM | AIR FLOW QUANTITY | AIRWAY TYPE | CO (MG/S) | CH4 (MG/S) | CO2 (MG/S) | NO2 (MG/S) | H2S (MG/S) |
|---------------|-----------------|-------------------|---------------|-----------|------------|------------|------------|------------|
| 1 | 1 | 53 09 | | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 2 | 14 | 20 | 53 09 RETURN | 0 0005100 | 0 2520000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 3 | 1 | 20 | 61 43 RETURN | 0 0005100 | 0 2520000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 4 | 20 | 1 | 61 43 RETURN | 0 0005100 | 0 2520000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 5 | 2 | 3 | 47 09 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 6 | 4 | 4 | 45 93 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 7 | 4 | 5 | 43 7 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 8 | 5 | 6 | 43 81 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 9 | 5 | 7 | 42 44 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 10 | 7 | 8 | 41 11 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 11 | 6 | 9 | 40 03 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 12 | 10 | 11 | 40 03 RETURN | 0 0001900 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 13 | 11 | 12 | 41 16 RETURN | 0 0001900 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 14 | 12 | 3 | 41 4 RETURN | 0 0001900 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 15 | 13 | 14 | 43 81 RETURN | 0 0001900 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 16 | 14 | 17 | 42 72 RETURN | 0 0001900 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 17 | 17 | 18 | 44 33 RETURN | 0 0005100 | 0 2520000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 18 | 18 | 19 | 44 09 RETURN | 0 0005100 | 0 2520000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 19 | 2 | 19 | 40 03 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 20 | 3 | 8 | 41 6 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 21 | 4 | 17 | 41 1 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 22 | 6 | 13 | 41 4 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 23 | 7 | 15 | 41 25 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 24 | 8 | 11 | 41 12 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 25 | 9 | 10 | 40 03 EM SITE | 0 0001900 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 26 | 5 | 14 | 41 6 EM SITE | 0 0003700 | 0 1900000 | 0 0000000 | 0 0000000 | 0 0000000 |

 CONCENTRATION OF CONTAMINANTS IN PERCENTAL

LIMITS OF CONTAMINANTS CONCENTRATION DEFINED BY THE LEGISLATION
 (in people eqst)

| | |
|-----|-------|
| CO | 0 005 |
| CH4 | 1 25 |
| CO2 | 1 25 |
| NO2 | 0 100 |
| H2S | 0 301 |

W A S H I N G T O N

* over 1/3 of the limit
 ** over 2/3 of the limit
 *** over the limit

| BRANCH NUMBER | CO (%) | CH4 (%) | CO2 (%) | NO2 (%) | H2S (%) |
|---------------|-----------|-----------|-----------|-----------|-----------|
| 1 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 2 | 0 000026 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 3 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 4 | 0 0000742 | 0 472518 | 0 215 373 | 0 0001444 | 0 0000698 |
| 5 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 6 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 7 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 8 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 9 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 10 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 11 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 12 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 13 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 14 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 15 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 16 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 17 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 18 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 19 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 20 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 21 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 22 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 23 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 24 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 25 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |
| 26 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 | 0 0000000 |

**MECHANIZATION AND
ORGANIZATION OF OPENING AND
DEVELOPMENT WORKS IN POLISH
COAL MINING**

POLSKA KOMISJA MIEDNICZEJ I KAZI WE
WĘGLA I KOPALNIE MIECHANIZACJI I
ORGANIZACJI

1. INTRODUCTION

The development of the coal mining industry in Poland has been characterized by a rapid increase in production and a corresponding increase in the mechanization and organization of the work. This has been achieved through the implementation of various projects and the introduction of modern mining equipment and methods.

The coal mining industry in Poland has considerably increased its output and efficiency, raising the output of the coal industry from 27.3 million tons in 1946 to 101.5 million tons in 1976. During this time, the coal output has remained at a steady level of 181 - 210 million tons. At present, Poland ranks the 4th coal producer in the world after the Chinese People's Republic, the United States of America and the Soviet Union.

The development of hard coal production in selected years of the past 4 decades, exclusive of lignite coal, has been presented in Fig. 1.

To ensure the above mentioned development of coal output, during the past 40 years, i.e. the following projects have been implemented:

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Technical Deputy of the Director General,
Polish Coal Board
Jan SZYDLEK, M.Sc.
Managing Director of Mining Development Corp.

- 22 new hard coal mines, there of 10 mines of coking coal, have been built,
- about 350 production, man-riding, supply and ventilation shafts have been sunk and approx. 130 new mining horizons have been developed and put into service,
- a majority of the existing mines have been extended and modernized.

At present, hard coal is mined in 70 active underground mines and next 2 mining units are under construction.

Geological and mining conditions in hard coal mines where mining operations are now underway, can be characterized as follows:

- the average depth for mining operations is 560 m, though in some mines coal is mined at a depth exceeding 1000 m,
- about 55% of the total output comes from methane-bearing seams, among them those of the highest gas hazard degree,
- in 50 mines water hazard is encountered and the water flowing to the mine workings is strongly mineralized and in many cases highly aggressive towards steel and concrete, i.e. elements constituting support structures of mine workings,
- in 14 mines the ambient rocks temperature in development and production workings exceeds 55 °C.

As may be seen from the above quoted facts, mining conditions for coal deposits in Poland must be considered as difficult.

They have also a bearing upon the choice of mechanization and organization systems applied at driftage of opening and development workings, such as vertical shafts and various types of galleries, which will be the subject of discussion in the further part of this paper.

To enable such an extensive development of the production capacity of mines in the past and maintaining it at a steady level at present the Polish Coal Industry since many years has to its disposal its own hinterland facilities, such as:

- research institutes and planning and design offices,
- machine production and repair capacities,
- executive potentials in the line of construction of mines and development of mining levels, extension, modernization and reconstruction of existing mines.

2. DRIFTAGE OF OPENING AND DEVELOPMENT WORKINGS .

2.1. The Quantitative Scope of Works

To maintain the production potential of the Polish Coal Industry at its present level it is imperative to sink annually approx. 5000 m of shafts and small shafts and to drive more than 1500 km of stone, coal-stone and coal galleries meant for making access to and panelling new parts of coal deposits. Irrespective of the above, there is a need for construction every year of many other underground structures, of so-called chamber-type, serving various purpose and functions in the chain of coal extraction technological process.

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work-
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the mining
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over 10 years,
the following have been accomplished:

- about 1500 thou. m of gallery workings, including... other shafts... 150 000 m,
- over 8000 thou. m of gallery workings, including 5400 thou. m of stone workings, as well as more than 35.0 mln.cu.m of chamber workings of any

... of boreholes have been
... and ...
... advance boreholes, etc.

... of ...
... of ...
... of ...

... of projects
... of ...
... of ...

... of gallery workings
... of ...
... of ...

At present the Mining Development Corporation incorporates 15 organisational units, including:

- 10 mine development units dealing with shaft sinking, drilling operations, driving principal gallery and chamber workings,
- 1 mine surveying unit serving the whole coal industry in full-scale mine surveying,
- 3 back-up units responsible for road transport, mining equipment, steel structure manufacturing, repair of mining equipment and auxiliary manufacture of some products for the needs of the industry,
- 1 scientific research unit providing its back-up in mine construction by solving any current and prospective technical, technological and organizational problems.

At present the Corporation employs 29,350 persons including 25,900 employees as directly production personnel and 3,600 employees as the engineering staff.

The organizational chart of the corporation, with the number of employees in the individual units, is presented in Table 2.

2.3» Machinery and Equipment Management for Conducting Mine Development Work

The Corporation, having so remarkable staff potential, also owns the principal machinery and equipment necessary for execution of mine workings and other mining projects with prevailing various geological and mining conditions and with the application of various methods and technological systems.

To ensure the most effective use of the machinery and equipment potential there has been set up, within the Corporation, a centralized management system in this line. The specific feature of that system is that certain organizational units have taken over the whole lot of machinery of a specified type and make it available for the users under a lease arrangement.

The following machinery and equipment have been covered by the centralized management system:

- machinery and equipment for the sinking of shafts from the surface /including special methods/, deepening of shafts and underground small shafts,
- machinery and equipment for driving opening-up and development workings in hard coal mines, both active ones and those under construction.

The organizational units for machinery and equipment management are also responsible for all the problems related to proper use and operation of the equipment. This covers the following:

- purchase of new machinery and equipment,
- conducting major overhaul and medium and routine repairs,
- storage of spares for machinery and equipment,
- running a technical service division to remedy the break-downs or damages directly with the users, without the need of bringing up the equipment from underground to the surface.

So organized system of machinery and equipment management has a number of advantageous features and it has been proved fully successful over many years of its application.

3. Shaft Sinking Mechanization and Organization

3.1. Shaft Sinking Methods

Trying to give a more detailed outline of mechanizational and organizational problems as far as shaft sinking is concerned it seems worthwhile to briefly discuss the geological and mining conditions, under which such kind of workings are sunk, and the methods of sinking.

The hard coal deposits in the territory of Poland originated in the Paleozoic era (the Upper Carboniferous system) and, as a result of subsequent geological processes which took place until the Quarternary, inclusive the deposit rocks, have undergone tectonic disturbances, weathering, erosion, saturation with water- and gasses and they have been covered with younger rock layers whose thickness occasionally exceeds 1000 m.

That rock cover, representing the Quaternary and Tertiary, and in some cases also Cretaceous, Jurassic and Triassic periods, is constituted in its most part of loose layers or those of a small strength, assuming in some places liquid state by the action of water or gas under high pressure.

As a consequence, this results in a necessity of applying special methods for passing through the cover layers.

In Poland about 90 % of shafts constructed had to be sunk by special methods on their length sections ranging from 50 to 760 m.

At present 3 special methods are applied, depending on geological and mining conditions prevailing in the rock body:

- the strata-freezing method,
- the injection method,
- the strata dewatering and degassing method.

3.1.1. The Strata Freezing Method

Although the shaft sinking method based on artificial icing of the water contained in rocks was in use in Poland more than 100 years back, it has found a wide-scale use only after the World War II. This was imposed mostly by the necessity of intensive development of the raw material resources in our country.

With its extending use, the method has undergone continual improvements connected with:

- rationalization of designing methods,
- increasing the capacity of the system and improving the operating data of the freezing units which allowed for increasing the degree of freezing the rocks,
- improving the drilling and casing of the freezing bore-holes,
- increasing the depth of application,
- theoretical learning of the physical and chemical phenomena taking place in the frozen strata and improving the shaft lining applied in the freezing zones.

The present state of the art allows for applying that method to a depth exceeding 750 m in shafts up to 9 m dia., with a temperature of frozen rock reaching up to -20°C .

For instance, this method has been successfully applied at sinking of shafts in the Lublin Coal Area when passing through the layers of quick sand deposited at a depth of approx. 590 m under pressure of 7.0 MPa.

A simplified diagram of a system for the sinking of shaft with freezing method is presented in Fig. 3 while a cross section of a length of shaft sunk with that method - in Fig. 4.

A certain variant of the above said method, based on sectional freezing of the strata, finds application in cases when the layers to be frozen are deposited at a considerable depth under non-waterbearing layers, or else when quicksand layers or strongly water-bearing ones are encountered unexpectedly.

The sectional freezing can be done with the use of bore-holes made either from the shaft foot or from the surface.

A diagram of sectional freezing of strata with bore-holes drilled from the shaft foot is presented in Fig. 5.

In case of applying sectional freezing with bore-holes drilled from the surface in the section where the shaft has been already sunk it is required to apply thermo-insulation in the freezing zone to protect the lining made in the non-frozen section of the shaft.

The so-called compressed air cushion or other low-heat-conduction materials may form the thermo-insulation. Sporadically, there may be used an inner heating circle for separating the frost zone from the shaft lining protection zone.

The experience gained in the Lublin Coal Area proved that an effective protection of the lining over the section undergoing the freezing process right from the surface can be ensured to a depth of 550 m.

A diagram of sectional freezing of strata with bore-holes drilled from the surface according to the variant with the method of thermal protection of the shaft lining (a heating circle or low-heat-conduction materials) is presented in Fig. 6.

5.1.2. The Injection Method

The injection method is applied when the shaft is being sunk in strong but fissured heavy water-bearing strata formation.

The above method is also applied in two variants i.e. by drilling from the surface and by drilling from the shaft foot. It is possible to apply the second modification with utilizing a natural rock shelf over a water-bearing layer or by an artificial concrete plug made in the shaft foot.

Better results are obtained with injection operations performed with bore-holes drilled in the shaft foot, especially in such cases when it is possible to flush water-bearing fissures by letting the water out of them through the bore-holes to the shaft.

An example of application of that method can be the sinking of Shaft II of the new " BUDRYK " Colliery (Fig.7) where a sudden water inrush

in the shaft at a depth of approx. 270 m at a rate of 4.7 cu.m/min and under pressure of approx. 2 MPa has been liquidated with that method*

3.1.3. The Method of preliminary Dewatering and Degassing of Strata

That method, also referred to as a depression method and based on dewatering and degassing of the strata around the shaft prior to the sinking of the latter, finds ever wider application apart from its disadvantages.

Among others, all the three shafts of a new Colliery " MORCIMEK " have been sunk with that method from the surface to a depth of 690 m in the cover layers represented by Tertiary formations constituted of thick complexes of silts and mudstones interbedded with exceedingly pressed thin layered sandy powders, sands and slightly firm sandstones having underneath an approximately 25 m layer of slightly firm conglomerates*

Because of a high pressure of water and gas prevailing in the layers of sands and conglomerates (approx. 3-4 Mpa) those formations in their natural condition, after exposing them with a shaft, would undergo fluidization and crushing followed ejection into the shaft excavation. This has been observed in one of the neighbouring shafts sunk across the Czechoslovak border.

The application of preliminary dewatering and degassing of the strata allowed for a considerable reduction of costs of the lining and for shortening the construction time by 2 years.

3.2. The Equipment Used for Shaft Sinking

Exclusively conventional methods of shaft sinking, i.e. with the use of explosives for working the rock solid are in use in Poland. So far no drilling methods have been applied.

For the actual sinking there are used machinery and equipment distinguished for high performance data which allows for sinking shafts of up to 9.5 m dia. and above 1200 m depth.

Among that equipment the following should be listed:

- hoisting headframes of up to 31 m height and effective lifting capacity up to 176 ton,
- winder units, single- or double-ended, provided with a motor of 630 kVA to 2000 kVA power, ensuring the reaching of up to 1500 m depth,
- electric hoists of 10 to 35 ton capacity, with rope drums of 2000 m storage capacity,
- cabin-type shaft loaders provided with buckets of 0.65 cu.m capacity,
- freezing units of 500 thousand kcal/h capacity provided with ammonia compressors,
- air compressors of up to 33 cu.m/min capacity with the working pressure of 8 bar. (Recently a higher efficiency compressors of screw-type are being introduced.)

3.3. Mechanization of the Sinking Process

The extent of mechanization of the principal shaft sinking operations, covering the working of rock and the construction of lining can be characterized as follows:

3.3.1. Bock Body Working and Muck Hoisting

In general, the rock working is performed by blasting. Exceptionally, in case of applying the freezing method, the working of side wall in close proximity to freezing bore-holes is performed with pneumatic pick hammers to prevent possible damage to the freezing bore-holes as a result of blasting.

At present, blast holes are drilled with hand-held rotary percussive drills to a depth upto 4.5 m (or with rotary drills in less compact rocks), driven pneumatically.

The blasting materials used are the rock-type explosives or others, in case of occurrence of methane hazard or when passing through coal seams, and milisecond electric detonators, their design and delay range applied depending on the conditions of performing the blasting work.

Recent research and design work is aimed at implementation of jumbos at the shaft drilling practice to eliminate the laborious manual drilling.

For limitation of the scope of manual working of side walls in shafts sunk with freezing method the Lublin Coal Division has successfully

introduced a special shaft-type shearer of their own design, the unit working mechanically the whole cross-section of the shaft under sinking.

For loading the muck, most frequently very efficient, high-capacity cabin loaders are used with 0.65 nr buckets. The muck is loaded into hoisting buckets having capacity up to 6.0 *TP*.

The discharge of muck from buckets at the shaft top is done automatically.

3.3.2. Construction of Lining

Construction of the lining, most often formed by a monolythic concrete lining, is also fully mechanized. To make the lining there is used a slidable steel shuttering suspended from winches which allows for easy handling and setting it in place.

The concrete mix itself is prepared in shaft site concrete plants or central concrete plants of a high capacity. The latter type of concrete plant finds ever wider and wider application, as being more economical and more suitable having in view technical and organizational considerations.

The lowering of concrete mix in the shaft to the place of its laying is done with the use of special buckets.

In case of applying other types of lining, e.g. tubbing lining or that made from heavy precast concrete segments, there are used special platforms or equipment enabling the construction to be mechanized.

3.4. Work Organization

The organizational systems applied at shafts sinking must take into account both the technical conditions and the limitations resulting from rules and regulations for safe conducting of sinking operations.

For instance, among the technical conditions one must specify the requirement for full utilization of the available work time and obtaining possibly high sinking rates taking into consideration a high cost of equipment installed on construction site. Next, mention must be made about such factor as limiting of the work time because of high or low (With the strata freezing method) temperatures prevailing in the foot of the shaft being sunk.

A long-time experience has led to the application, as a general rule, that the shaft sinking work is conducted on a 4-shift/day basis, the working shifts having 6 hours each.

The number of personnel in the shaft foot (face), depending on the shaft diameter, the technology and mechanization applied, varies from 5 to 9 men.

The off-face works, the attendance of all machinery and equipment and auxiliary work on the surface, are conducted by separate teams working on 3-shifts/day basis, the shifts being of 8 hours each. The magnitude of that manpower is 7 - 10 men depending on the technology and sinking work mechanization applied.

4. MACHANIZATION AND WORK ORGANIZATION IN DRIVING GALLRRY WORKINGS

4.1. Technical Specifications and Driving Method

Applied

Oat of the 1500 km of gallery workings driven annually in the Polish Coal Industry only 267 km (i.e. 17.4 %) are classified as stone workings while the remaining part, as coal and stone or coal workings.

Approximately 88 % of all workings are horizontal workings or those inclined up to $\pm 20^\circ$.

The gyerage cross-section area of the workings, as excavated, is 17.40 m^2 , though a part of the workings has cross-sections larger than 20.0 m^2 . This is due to ventilation and air-conditioning reasons in connection with a high rate of emission of methane from coal seams and with high temperature.

All the gallery workings are secured with steel supports.

In view of the steady increase of depth at which the workings are driven, and the resulting ever harder and harder geological and mining conditions, there has been observed a pronounced trend in applying arch supports with ever increasing unit weight, where the 21 kg/m and 25 kg/m profiles are replaced by supports with 29 kg/m or 36 kg/m unit weight and, recently, even with 44 kg/m.

the gallery workings are driven with the application of conventional methods only, because of the fact that they are located in fairly hard carboniferous formations. Sometimes, certain difficulties are encountered when passing through strongly water-bearing faults and gassy zones, and in such cases preliminary dewatering and degasification of the strata is performed by means of special bore-holes.

As far as the manner of working the rock body is concerned, both conventional and road-header driving methods are applied. The kind of method chosen depends on the cross-section of the gallery driven and the kind of rocks encountered.

In practice, all the stone headings are driven with conventional methods while a part of coal and stone and coal workings are driven with the use of arm-type road headers.

4.2. Driving of Stone and Coal-Stone Headings with Conventional Methods

4.2.1. The equipment Used

The principal equipment for driving such workings with conventional methods include the following:

a/ for drilling of blast holes:

- Hand-held percussive drills pneumatically driven, of 22 kg weight, mounted on air legs,
- Electro-hydraulic-driven Caterpillar mounted drilling jumbo units provided with two heavy-duty percussive drills of 70 kg class,

b/ i'or loading the muck:

- Overhead loaders with 0.25 m scoop, pneumatic-driven, rail-mounted,
- Scraper loaders with electric drive,
- Side-discharge loaders with a scoop of 1.0 - 1.2 nr capacity, caterpillar-mounted, having electric or pneumatic drive, the latter type of drive being meant for operation in conditions rendered by a high degree of methane hazard or high temperatures,
- Stripper-loaders with an electric drive.

c/ For the haulage of muck from the face there are used mine cars of up to 3 nr capacity or electric-driven scraper conveyors.

d/ For delivery of supplies there are used the following technical means: mine cars, platforms, timber bogies, suspension-type mono-rail systems, winches with electric or pneumatic drives, track tractors with pneumatic drives, and, in the face zone, also the scoops of side-discharging loaders.

4.2.2. Mechanization of Heading Drivage

Depending on the prevailing conditions and needs (cross-section, inclination, run of the working) the above discussed equipment is utilized in determined technological systems presented in Table 1, the individual process operations being denoted with the following symbols:

F - drilling of blast holes,
L - loading of muck,
'f - haulage of muck,
M - supplying of materials to the face,
S - making of supports,
A - auxiliary operations.

Examples of specific technological systems based on the equipment discussed above are presented in drawings 8 and 9»

It has to be underlined that the whole equipment for driving gallery workings is represented by products of Polish factories. They are subject to continuous modernization and better adaptation (improvement) for use under various requirements and conditions set forth by specific applications.

One of the essential trends in this line is the desire to eliminate the non-economical pneumatic drives in favour of electro-hydraulic ones.

Table 1

| Sl. No. | Operation | Basic equipment | Kind of working |
|---------|----------------------------|--|--|
| 1 | F L T and M S | 2-4 percussive drill units with air legs 2 overhead loaders Mine cars of small capacity (0.6-1 m ³) Manual work | Horizontal gallery workings with a small cross-section of 8 - 12 m ² |
| 2 | F L T M A S | 3-5 percussive drills with air legs or drilling jumbo Side-discharging loader Mine cars of medium capacity (2-3 m ³) Tubs, platforms, timber carriers Turn-out plate with winch and rail shunting tractor Manual work with a loader | Horizontal gallery workings 15 - 30 m ² |
| 3 | F L T M S | 3-5 percussive drills with air legs or drilling jumbo Side-discharge loader Scraper and belt conveyor Suspended monorail system Manual work with a loader | Gallery workings with $\pm 8^\circ$ inclination and 15 - 20 m ³ cross-section |
| 4 | F L T M S | 3-5 percussive drills with air legs 2 overhead loaders or side-discharge loaders with brakes Scraper and belt conveyor Suspension-type monorail system Manual work | Inclined gallery workings / $\pm 8^\circ$ / |

4.2.3» Heading Drivage Organization and Results Obtained

The work in stone headings is basically performed in 3 shifts, rarely in 4 shifts, applied at a limited effective time of work in the face (at a considerable distance from the man-riding shaft or at higher temperatures) .

The teams consist of 4 - 7 men depending on the method applied. The effective shift index for stone gallery workings exceeds 2.9. The average daily advance rates obtained are 2.3 - 2.5 m/day while the monthly rates are contained within 60 m/month. When driving coal-and-stone headings daily rates of 5.3 - 5*5 m/day are obtained and the monthly rates range to approx. 135 a«

4.3» Drivage of Stone Coal Galleries with Shearer Methods

4.3.1. Equipment Used

As already mentioned, for coal-and-stone headings arm-type road headers are used.

In recent years their range of application has been greatly extended and they are used in approx. 70° of the whole of gallery workings, i'or domestic use there is applied an AM-50 road header manufactured under licence of the Austrian Voest Alpine. Its design is a good one and it fully meets the requirements as far as mechanical working of the rock solid is concerned.

However with the experience gained nowadays some disadvantages of this design may be also observed, such as:

- limited application to the workings with a cross-section within 12 - 17 m,
- impossibility of cutting the coal accompanying rocks of the hardness 40 - 60 MPa

4.3.2. The Drivage Mechanization

In principle, one mechanization system is in use, but of two variations. The system is based on use of two scraper conveyors arranged in series behind the shearer.

Depending on the variant of the system, the intermediate conveyor may be set on the floor of the working and it can be either extended in accordance with the face advance or suspended to a travel carrier and shifted in whole with the advance of the face.

Most of supplies are delivered to the face with the use of suspended monorail systems.

4.3.3» Drainage Work Organization and Results Obtained

when driving mine workings with application of shearer technology 4-shift work system is used with three working shifts and the 4th one devoted to the maintenance of the equipment, including routine repairs and the extension of transportation system.

The personnel in each production shift amounts to 5 persons.

When the shearer system is used, the average daily advance rates reach 8 - 9 m while the monthly rates reach 200 - 230 m.

In favourable conditions and in case of urgent need the rates obtained in Poland reach 30 m/day and 800 - 900 m/month.

5. CONCLUSIONS

As a summary to this paper the following conclusions may be drawn:

1. In the Polish Coal Industry the opening and development works are much diversified and they are carried out to a wide scale.

The mining work is often conducted in adverse geological and mining conditions with a high degree of various natural hazards.

The amount of equipment in operation is considerably high and, though their designs do not represent the top modern level they are utilized to the utmost. However, recently there has been noted a systematic development of mechanization and a trend to mechanize all the operations related to shaft sinking or heading drivage.

2. The mine development works are based on conventional methods. In applying those methods use is made of the vast experience so far gained in sinking of more than 240 km of vertical mine workings and 8000 km of gallery workings.

The use of fall face drilling methods in shaft sinking and fall-section cutting road headers in gallery drivage is not predictable as yet. This is due mainly from the fact that there occur frequent and varying quite serious problems of mining, geological and hydrological nature, and that the costs related to the application of that technology are very high,

3. The engineering staff and workmen of the mining of the mine development set-up represent an exceptionally high level of skill and experience and they can undertake the execution of any most complex and serious tasks related to the drivage, reconstruction, extension, furnishing, even the maintenance of underground workings necessary for mines or cities, or else for other industries and specialistic firms.

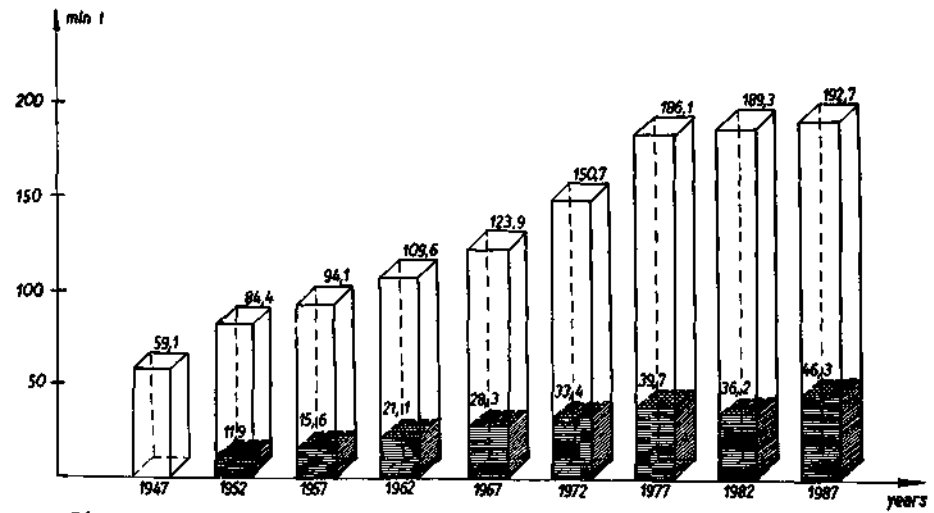
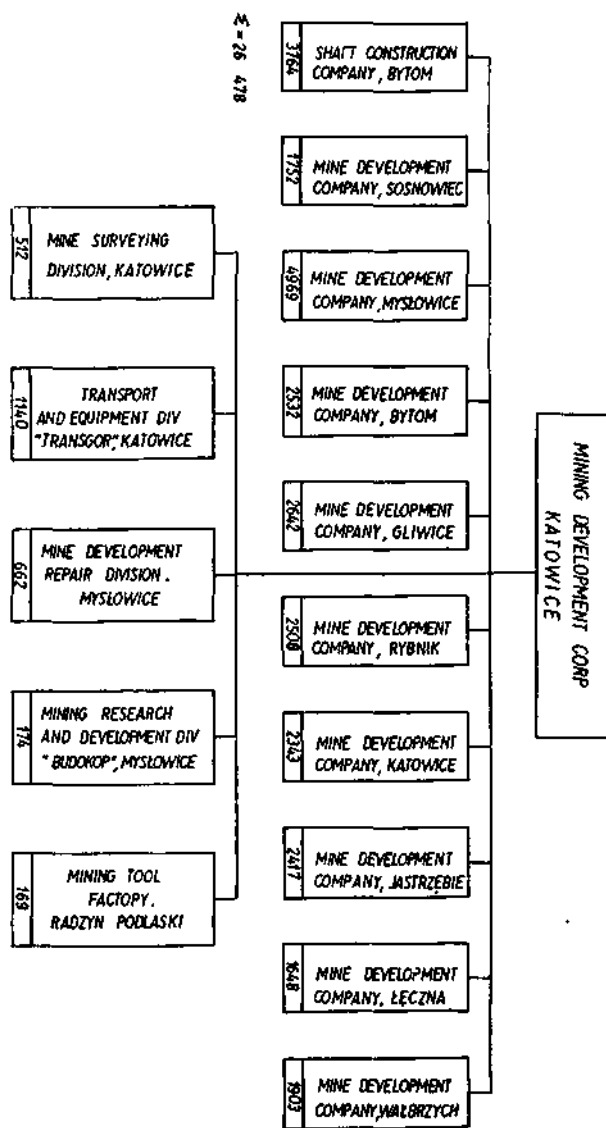


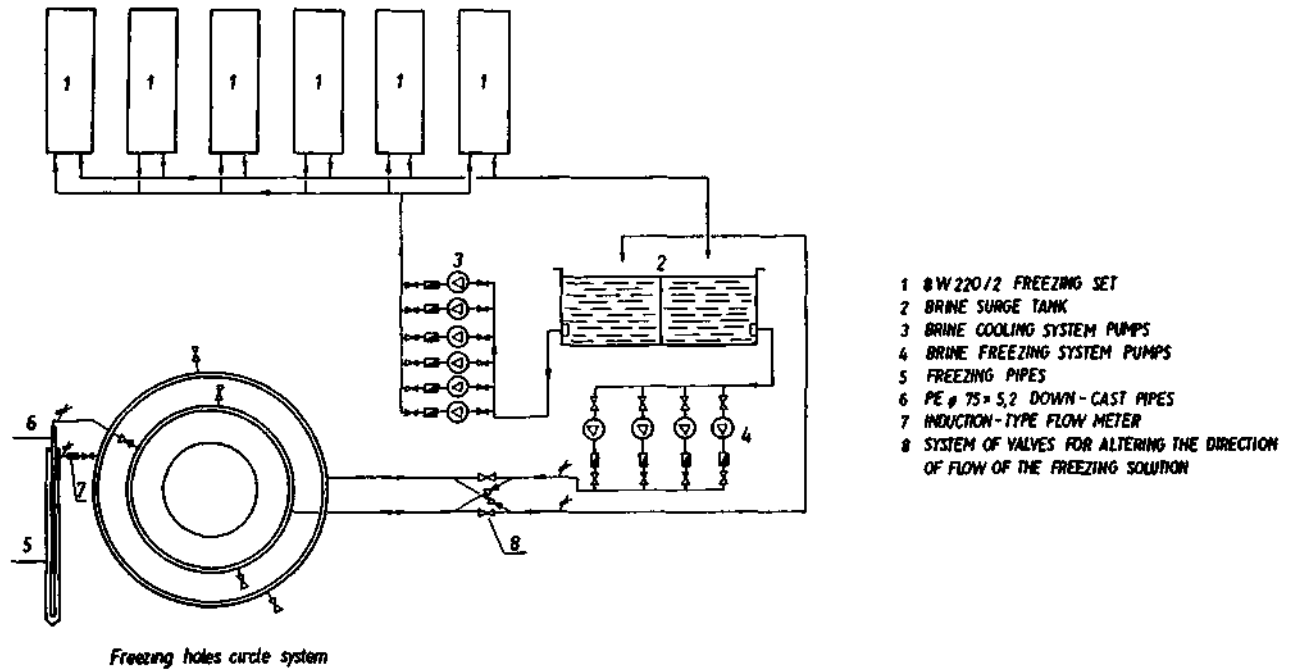
Figure 1. Hard coal output (including coking coal) in 1947-1987

Fig 1

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Figure 2. Organization chart of Mining Development Corporation Katowice

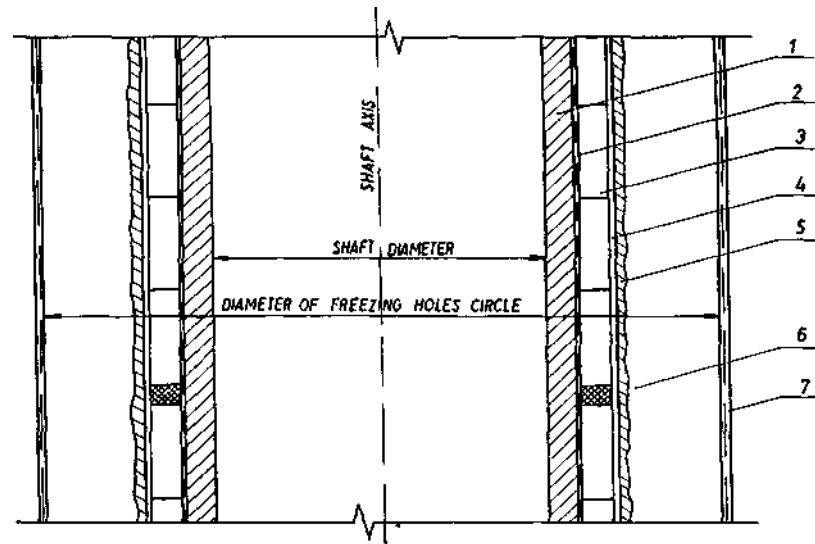




- 1 ØV 220/12 FREEZING SET
- 2 BRINE SURGE TANK
- 3 BRINE COOLING SYSTEM PUMPS
- 4 BRINE FREEZING SYSTEM PUMPS
- 5 FREEZING PIPES
- 6 PE # 75 x 5,2 DOWN-CAST PIPES
- 7 INDUCTION-TYPE FLOW METER
- 8 SYSTEM OF VALVES FOR ALTERING THE DIRECTION OF FLOW OF THE FREEZING SOLUTION

Figure 3. Simplified Diagram of Starta Freezing System

Fig 3



- 1 Permanent concrete lining made with the use of sliding formwork
- 2 Hydro - isolation of plastics foil
- 3 Preliminary lining made from prefabricated elements
- 4 Tension member for suspending preliminary lining
- 5 Concrete filler layer
- 6 Technological link lined with brick or concrete lagging
- 7 Freezing holes

Figur» 4. Cross-Section of a length 'f shaf sunk with strata Freezina method

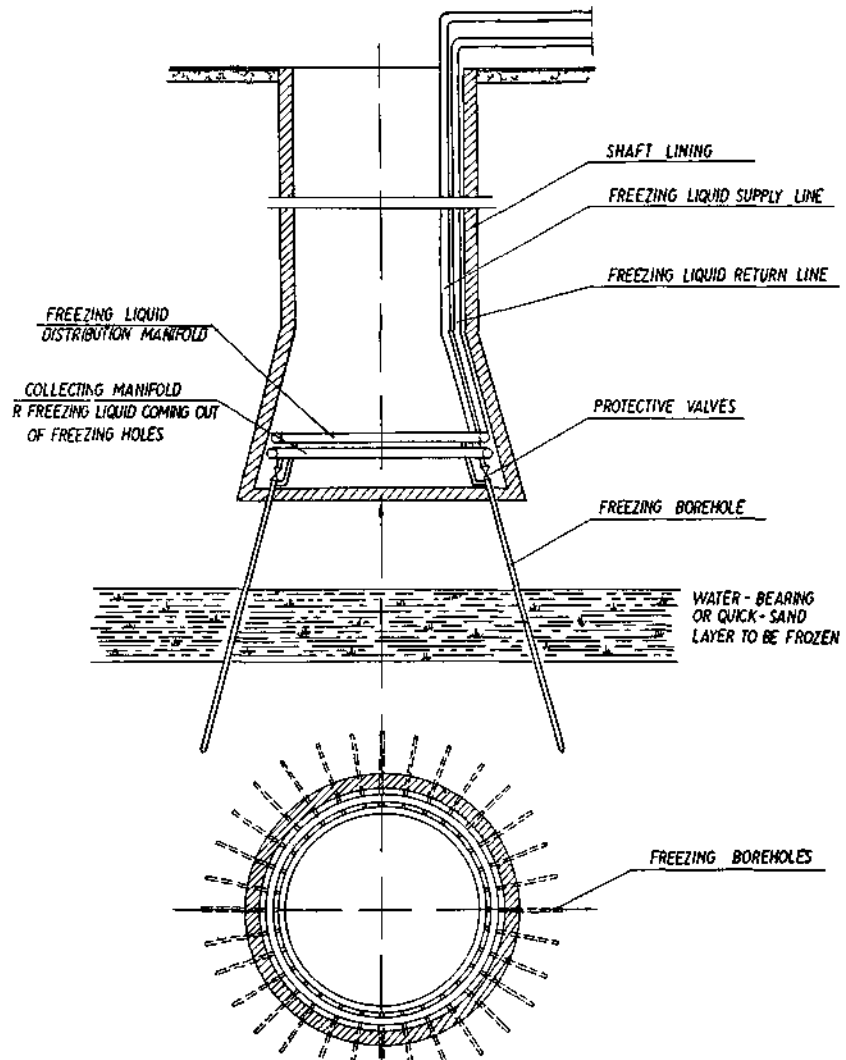


Figure 5. Diagram of shaft freezing of strata with application of boreholes drilled from shaft foot.

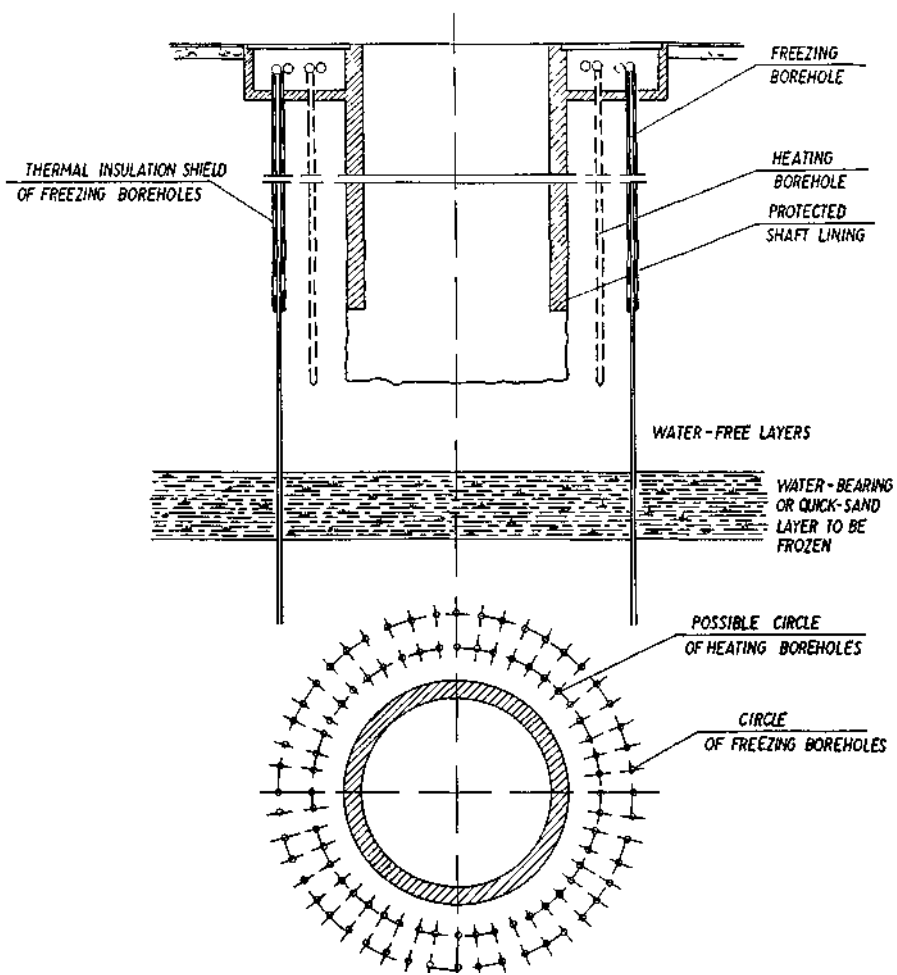


Figure 6. Diagram of sectional Freezing of strata with application of boreholes drilled from the surface

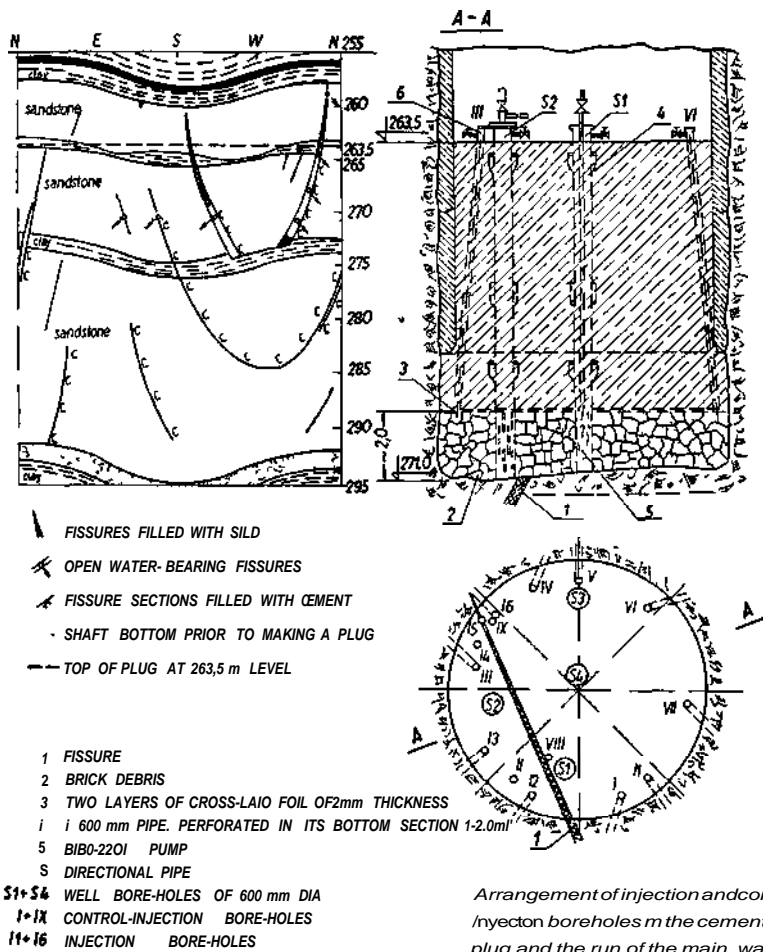


Figure 7a. Developed view of cylindrical surface of the 225-295 m section of shaft II of Bulryk colliery

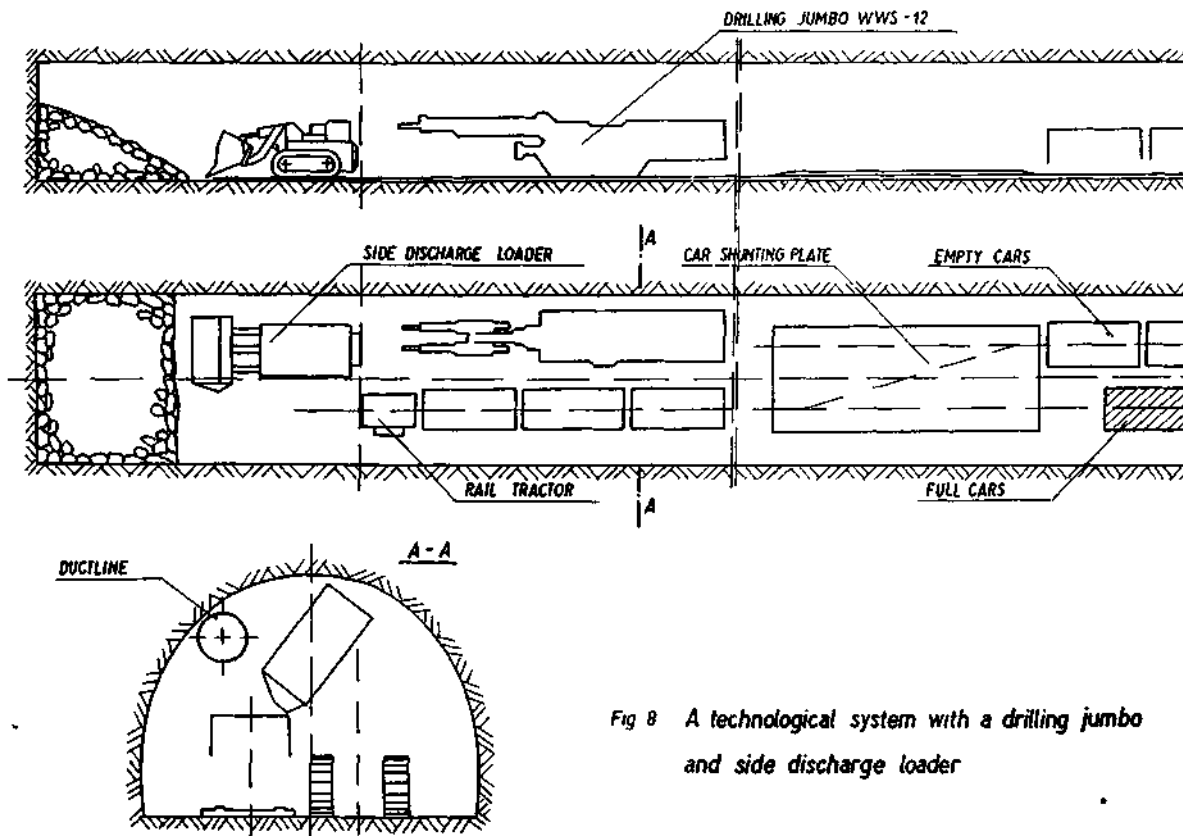


Fig 8 A technological system with a drilling jumbo and side discharge loader

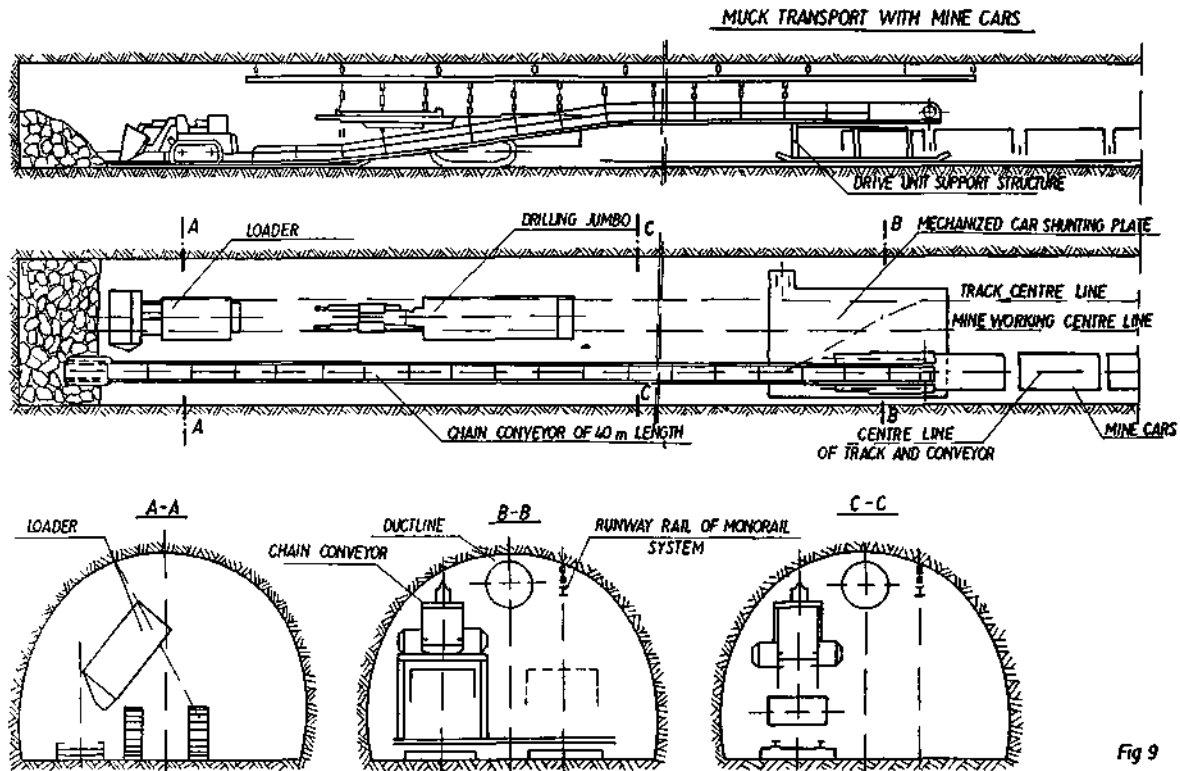


Figure 9. A technological system with a drilling jumbo and side discharge loader. Intermediate conveyor for muck transportation.