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Mine Inundation-Case Histories

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ABSTRACT

Mining in the vicinity of large bodies of water, below a worked out coal seam or under a confined aquifer or abandoned water logged workings is always fraught with danger of inundation. This paper outlines the causes of inundation in underground mining operations together with a brief account of accidental inundation in a range of mining and geological conditions. Statutory provisions to control inundation in various countries are outlined and a risk management approach to solve water danger is suggested.

INTRODUCTION

Seepage of water can cause constant difficulties in underground mining operations and can create a range of stability, safety and operational problems. Thus, handling, pumping, treatment and disposal of mine water is a much larger problem than can be appreciated by a casual observer. The possibility of sudden inundation is a real threat in many underground mines specially in the traditional mining countries having a honeycomb of old, abandoned mine workings surrounding the operating mines. Sudden inrushes of water have resulted in hundreds of fatalities in many countries including Great Britain, India and United States of America. This paper classifies the type of inundation in underground mining operations and gives a brief account of some of the major inundation in a variety of geological and geographical conditions.

Table 1. Selected inundation accidents in England and Scotland (after Moebs and Sames, 1989, and Duckham, 1973).

Year	Mine	Fatalities
1815	Heaton Colliery, Northumberland, England	90
1837	Workington Colliery. W Cumberland. England	27
1883	Diamond Mine, Braidwood. Il.	69
1885	No. 1 Slope. Nanticoke, Pa. A (anthracite)	26
1895	Audley Colliery, N. Staffordshire, England.	77
1889	White Ash Mine. Golden. Co	10
1891	Spring Mountain Mine. Jeansville. Pa. A	9
1892	Lytle Mine. Minersvllle, Pa. A	10
1898	Williams Mine. Middleport, Pa A	6
1901	Donibristle, Scot.	8
1908	Rochburn, Scot.	3
1912	Superba-Lemont Mines, Evans Station, Pa	18
1917	Wilkeson Mine. Wilkeson. Wa.	6
1918	Stanrigg-Arbuckle, Lanarkshire, Scotland	19
1923	Redding Colliery. Falkirk. Stirlingshire, Scot.	40
1925	Montagu Colliery, Northumberland, Eng.	38
1927	Carbonado Mine. Carbonado. Col	7
1950	Knockshinnoch Colliery, Ayrshire. Scot	13
1952	Holmes Slope, Forrestville, Pa. A	5
1954	Newton Chickli Colliery, M.P, India	62
1958	Central Bowrah, Jharia, India	23
1959	River Slope. Port Griffith. Pa. A	12
1960	Dhamua main , M.P., India	16
1970	Karanpura Colliery, Bihar, India	3
1973	Lofthouse Colliery. Northumberland, England.	7
1975	Silvewara Colliery, Nagpur, M.P., India	10
1975	Chasnala Colliery, Jharia, India	375
1977	Porter Tunnel Mine. Tover City. Pa A	9
1978	Moss No. 3. Dante. Va.	4
1979	Mine No. 1. Poteau, Okla.	1
1981	Harlan No. 5. Grays Knob, Ky.	3
1983	Hurrilladih Colliery, Jharia, India	19
1985	Lykens No. o, Lykens. Pa. A	1

CAUSES OF MINE INUNDATION

The classification of mine inundation is necessary in order to understand the underlying cause of sudden inflows so as to provide remedial measures to prevent reoccurrence of such events. A critical review of various past inundation has enabled a classification of mine inundation into three categories, namely; (i) Event Controlled Inundation, (ii) Accidental Inundation and (iii) Spontaneous Inundation (Singh, 1986). The first two types are mining induced inundation whilst the third is a natural phenomenon. The event controlled inundation is associated with caved mine workings below either a confined aquifer or surface bodies of water where the inflow is followed by main and periodic roof falls in the roof strata. The inflow rate of the water is suddenly increased from the background level to a peak rate within a short time and is then reduced exponentially to the background level over a period of time. Spontaneous inrushes, (Sammarco, 1982, and Sammarco, 1986) are a natural phenomenon associated with mining in the vicinity of karst aquifers. Accidental inundation, which is the topic of this paper and is a major cause of concern to the mining industry due to working in the vicinity of large bodies of water, can present a menace to life. A lake or ocean, a large pool of water in an upper seam or water flooding the adjacent old workings, if suddenly released to the lower active workings, could easily flood the current workings with possible fatalities. Table 1 presents selected inundation accidents in the Great Britain, the USA and India which have captured the attention of mining engineers all over the world.

ACCIDENTAL INUNDATION

The danger of a sudden and accidental inrush of water or material which will flow when wet is a traditional mining hazard. An analysis of inrushes in the British collieries during the period 1851 to 1970 by Job (1987) has identified the sources of inrush and the number of fatal incidents arising from each inrush. As can be seen in Table 2, the greatest risk of accidental inundation is from abandoned flooded workings, the total of 162 is well in excess of all the other inrush sources put together.

Table 2 Sources of inrush and number of accidents arising (Job, 1987)

1	Contact with surface water—pond, river, canal or stream	9
2	Contact with surface unconsolidated deposits—glacial or organic	8
3	Strata water entering working	2
4	Shaft sinking	4
5	Clearing old shafts	14
6	Contact with abandoned old workings	162
7	Failure of an underground dam, seal or leakage of a bore hole	9

The danger created by inrushes cannot be defined solely in terms of quantity of water or even rate of flow. More vital may be the position of the working areas relative to the source of the inrush, and the layout of roadways and their gradients which will determine whether they can store water safely or not. The capacity of the drainage system installed underground and the ease with which this can be increased are also important factors.

Job (1987) analyzed and classified the causes of accidental inundation in British collieries. Accidental intersection of waterlogged old workings by the current active developments is a single major cause of accidental inundation. This may happen due to various underlying reasons such as accuracy of old plans, failure in communication or failure in management systems. The sources of inrush through intersection with abandoned old workings can be classified as follows:

- Ineffective barrier between unknown or known waterlogged and active workings.
- Incomplete old plans.
- Absence of plans.
- Incorrect plans.
- Incorrect interpretation of old plans.
- Failure to obtain abandonment plans.
- Absence of protective bore holes.
- Failure to plug borehole.
- Unknown old workings.
- Incorrect seam correlation.

In the past, attempts have been made to prevent or control accidental inundation by adopting prescriptive legislative control as discussed in the next section.

STATUTORY PROVISIONS TO CONTROL MINE INUNDATION

Early mine inundation accidents in Great Britain created a public outcry which motivated the Government to develop mining legislation to prevent inundation.

For example :

1 Accident at Heaton colliery, Northumberland, United Kingdom on 3rd May, 1815 in which 90 persons killed and *subsequent major inrushes* resulted in the introduction of the provision of mine plans to be submitted and the appointment of a qualified surveyor.

2 Mine inundation with peat at Knockshinnock Castle Colliery, Ayrshire on 7th September, 1950 when 13 persons were killed, led to the introduction of the responsibility of the mine surveyor to show on the mine plan information regarding inrush of water or material which can flow after becoming wet, and requiring the manager to take appropriate action.

3 Lofthouse colliery, Yorkshire on 21st March, 1973 - 7 persons killed. This led to the introduction of a requirement to show the position of exploratory boreholes on the mine plan and the provision for all exploratory boreholes to be sealed with cement grout to prevent inter-connection between various seams and different aquifers.

4 Tower/Fernhill mine, South Wales on 30th December, 1974. Responsibility of adjoining mines and quarries to communicate on water problems and update the mine plans.

Most countries in the world have their own statutory requirements to control mine water inrush by imposing prescriptive regulations to restrict mining activities near old abandoned mine workings likely to contain water. These statutory requirements can be grouped into two headings viz. the surveyors responsibilities and the management's responsibilities. Anon (1982)

***Surveyors Responsibilities** are as follows:-*

(i) Provision of accurate mine plans.
(ii) Provision of updated mine plans before abandonment.
(iii) Provision of mine plan showing information of water danger, or any other dangerous deposits which may flow when wet.

Management Responsibilities

Provision to maintain advanced central and flank bore holes to locate the presence of water logged workings and drain water when current active working is approaching the stipulated distance (50m) from the following workings (Anon 1982):

(i) Underground water-logged workings
(ii) Surface bodies of water.
(iii) Surface deposits which can flow when wet.

Inadequacies of prescriptive legislation and importance of risk management and management rules

In spite of the above mentioned statutory controls one of the major causes of underground inundation is the presence of unchartered workings or inaccurate abandonment mine plans. Remedial measures may include improved standard of surveying practice (if necessary) and the use of outside consultant surveyors to update mine plans before abandonment.

Secondly, the provision of drilling advanced and flank bore holes ahead of an active development or mine working approaching old mine workings likely to contain an accumulation of water are inadequate for the following reasons.

(i) Short advanced bore holes may leave an inadequate barrier between the blind end of the bore holes and the old workings which may subsequently fail (Figure 1).

(ii) Long advanced bore holes may completely miss a narrow old workings containing an accumulation of water.

(iii) An advanced bore hole through highly stressed ground or an outburst prone ground may completely close due to strata pressure thus giving wrong impression that the old mine workings are free of water.

(iv) There are no statutory provisions for controlling the inflow of water through precautionary bore holes once they have intersected the old workings.

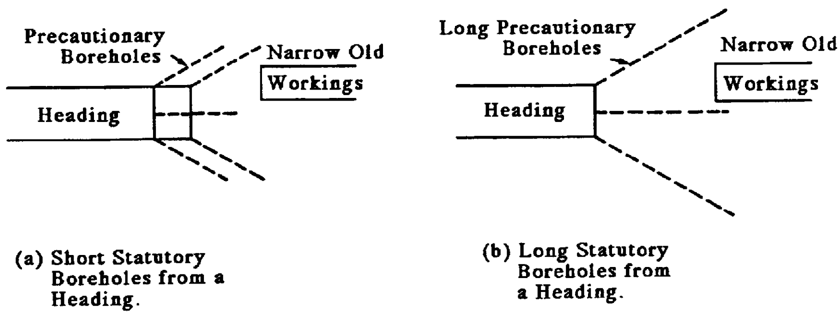


Figure 1 Statutory bore holes when current workings approach water logged workings

It is therefore, important that the management take much wider range of precautions to prevent inrush of water than those stipulated by the statutes.

- o More scientific control
- o Risk analysis approach to the inundation problem
- o Manager rules

Scientific Control Methods

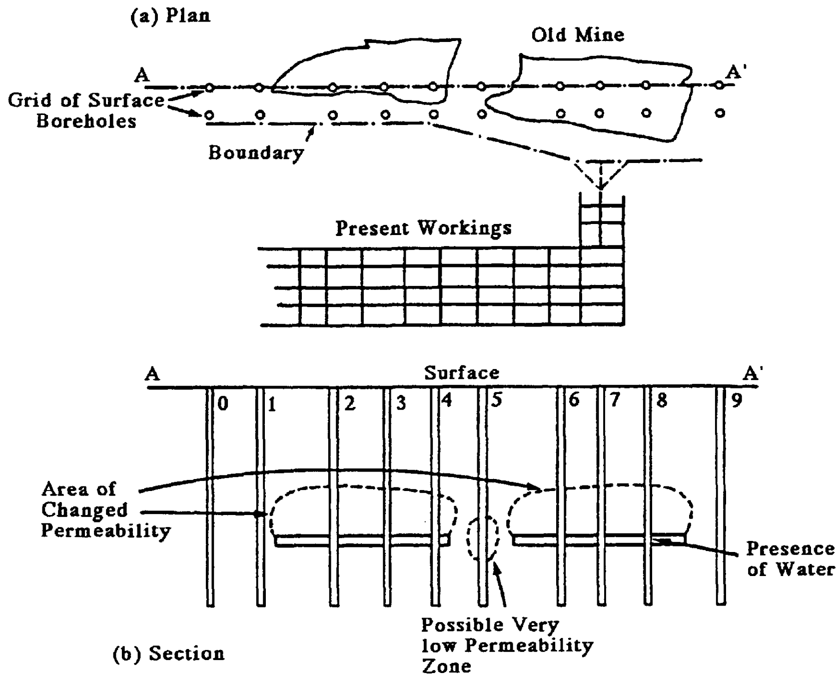


Figure 2 Surface bore hole technique for locating old mine workings

From the foregoing discussions, it can be concluded that there is a need for applying new scientific techniques for solving water danger problems. Special considerations should be given to assert strict management control of individual mines during planning, design and operation of the mine. More scientific control is needed by measuring hydrogeological behavior of the surrounding strata in the individual mines, thus creating a data base. Use of existing methods and developing new techniques for locating permeable and impermeable zones around the mining operations should be supplemented by detailed studies of surface water hydrology of the mining catchment area. It is suggested that some hydrogeological tests should be conducted in the precautionary bore holes or surface bore holes to locate zones of low and high permeability and also to measure in situ pressure profiles. Continuous flow metering tests in open bore holes will permit plotting of the cumulative water velocity and depth profile of a bore hole, thus indicating radial permeability of the rock mass (Spichack, 1988). This may give an indication of zones of high and low permeability, leading to the detection of old mine workings.

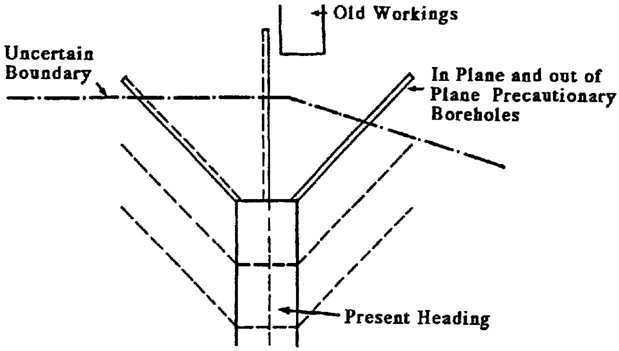


Figure 3 (a) In-seam bore hole testing technique.

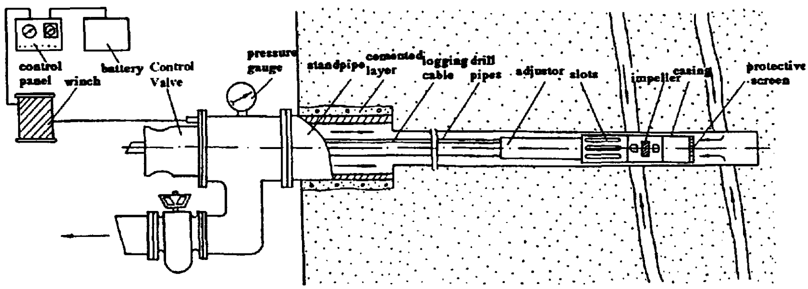


Figure 3 (b) Arrangement and device for testing underground bore hole (Spichack 1988)

CASE HISTORY ANALYSIS OF ACCIDENTAL INUNDATION

(1) Inundation during shaft sinking or during major development

Coal mine workings in concealed coal fields below the base of younger and unconsolidated formations or under unconfined aquifers have not given much water problems in the U.K during the operation stages of mining (Saul 1970). However, shaft sinking through confined aquifers, formations geologically younger than the Coal Measures (like Permian or Trias), have given water problems during shaft sinking, particularly, at shaft lining stage. (Priest and Leggat, 1914 and Hunter, 1957). Sudden inundation of water in a mine not only causes danger to men, property and equipment, but direct costs of dewatering against high hydraulic head, costs of recovery and restoration are very high. Moreover, consequential costs of lost production and social cost of loss of employment opportunities are phenomenal. Water problems during shaft sinking are not necessarily prevalent in coal mining only. A case example of inflow of water and unconsolidated debris in a potash mine in Canada is described below.

Case History 1: Inundation Cominco potash mine, Saskatchewan, Canada, 2 August, 1970

The Cominco Potash mine is situated 40 km southwest of Saskatoon, Saskatchewan where a potash bed is mined at a depth of some 1100 m . The potash bed is underlain by 69 m thick anhydrite bed and 79m of halite, sylvanite and banded clays which can be classified as aquiclude. The total 1060 m thick overlying beds consist of some 400m of interbedded mudstone, dolomite, limestone and anhydrite forming multi-layered aquifer system, overlain by siltstone and some 110m of unconsolidated sandstone and siltstone aquifer. This aquifer is covered by 450m of shale and 65m of glacial till forming an unconfined aquifer, near the surface.

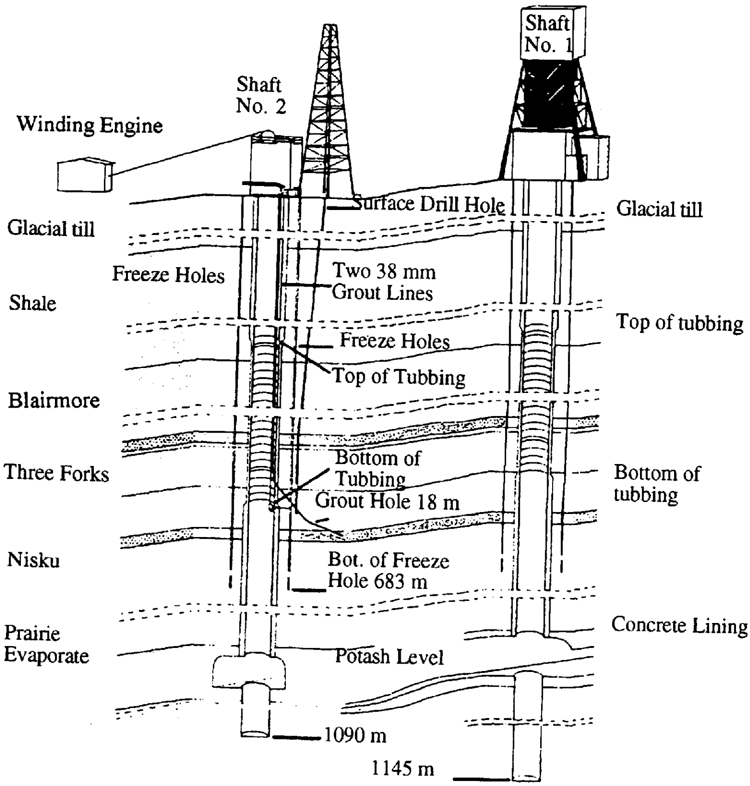


Figure 4 Section through No.1 and No.2 Shafts, Cominco Potash Mine (Prugger, 1979)

The mine comprised two shafts sunk 152 m apart, No 1 shaft being 1090m deep whilst No 2 shaft was 1145m deep, the mining horizon being at 1078m depth . The shafts were sunk by a freezing method using 18, 260 mm diameter freeze holes drilled to the depth of 685 m to negotiate 110m unconsolidated aquifer comprising the Baltimore Group of the Lower Cretacious formation. The shaft which had a finished diameter of 4.93m (excavated diameter 7.07m) was lined with 1.070m thick concrete lining. No 1 shaft was lined with cast iron tubbing from a depth of 498m to 636m and No. 2 shaft 500m to 639m depth.

On 2nd August, 1970, during routine shaft grouting, a major water feeder was intersected at the bottom of the tubbing column. Water and quicksand under very high pressure entered the shaft at a rate of 65 m^3/min (1083 l/s, Prugger, 1979) and flooded the mine including both shafts. Figures 4 shows the cross section through both shaft showing straitigraphical column and the point of intersection.

The breach was sealed with cement grout simultaneously from and within the shaft through a relief hole drilled from the surface. The mine was dewatered, rehabilitated and the equipment was restored and the production resumed two years after the accident. A total of 515,130 m^3 of brine had to be removed before clean-up and rehabilitation of underground mine openings and equipment could begin. To provide access to the mine, 15,000 m^3 of fine quartz sand and mud had to be removed. One low-lying area was not dewatered, but was abandoned.

Case History 2: Inundation in Eastern Karadon group of mines, Zonguldak coal field on the Black Sea coast, Turkey

The Eastern Karadon group of mines is situated in the Zonguldak Coalfield on the Black Sea coast, Turkey. In this mine, a number of steep dipping coal seams namely Kurtsan seam, Kozlu seam, Papas Seam and Karadon seams are being mined from Karadon shaft by the horizon method of mining. The main coal washing plant was situated on the surface near the Catalagzi area to service the group of mines. The Catalagzi shaft had been sunk as the main coal winding shaft for the mining group and it was located immediately adjacent to the central washing plant, connected to the shaft by a bridge conveyor. Due to the necessity of the plant location, minor consideration was given to the hydrogeology of the underlying strata. (Figure 5). The Catalagzi shaft was sunk through the hanging wall side of the Karodon series coal measures, mainly through limestone of cretaceous age. These limestones were highly karstified causing many sinkholes towards the south east of the Catalagzi shaft. At the 360 m level a main haulage level was being driven to connect the new shaft with the Karadon mine workings.

The Catalagzi shaft had an inside diameter of 6.5 m and was concrete lined. As the shaft was sunk through the karstic limestone, extensive grouting work had already caused a delay of 18 months during the sinking due to constant water problems. After sinking was completed the underground loading station loop had to be driven through the vuggy, water bearing limestone and the shaft station had to be connected with the existing mine workings by a main haulage roadway through limestone which dipped at about 30 ^{o}. The development work was carried out observing all precautions. Pilot holes were kept in advance of the face and the rock mass was grouted where necessary. After 300 m of drifting through the karst limestone in the loading station loop, a major water inflow occurred. It can be seen in Figure 5 that the karstic limestone forming the basin has many solution caverns and is intersected with major faults. It may also be noted that in spite of these hydrogeological features there were no test holes drilled in the vicinity and at the centre line of the shaft at the time of shaft sinking. During the drivage of the loading station hoop, water rushed in at a rate of 32.5 m^3 per minute. As a consequence, the drift was completely flooded and had to be abandoned.

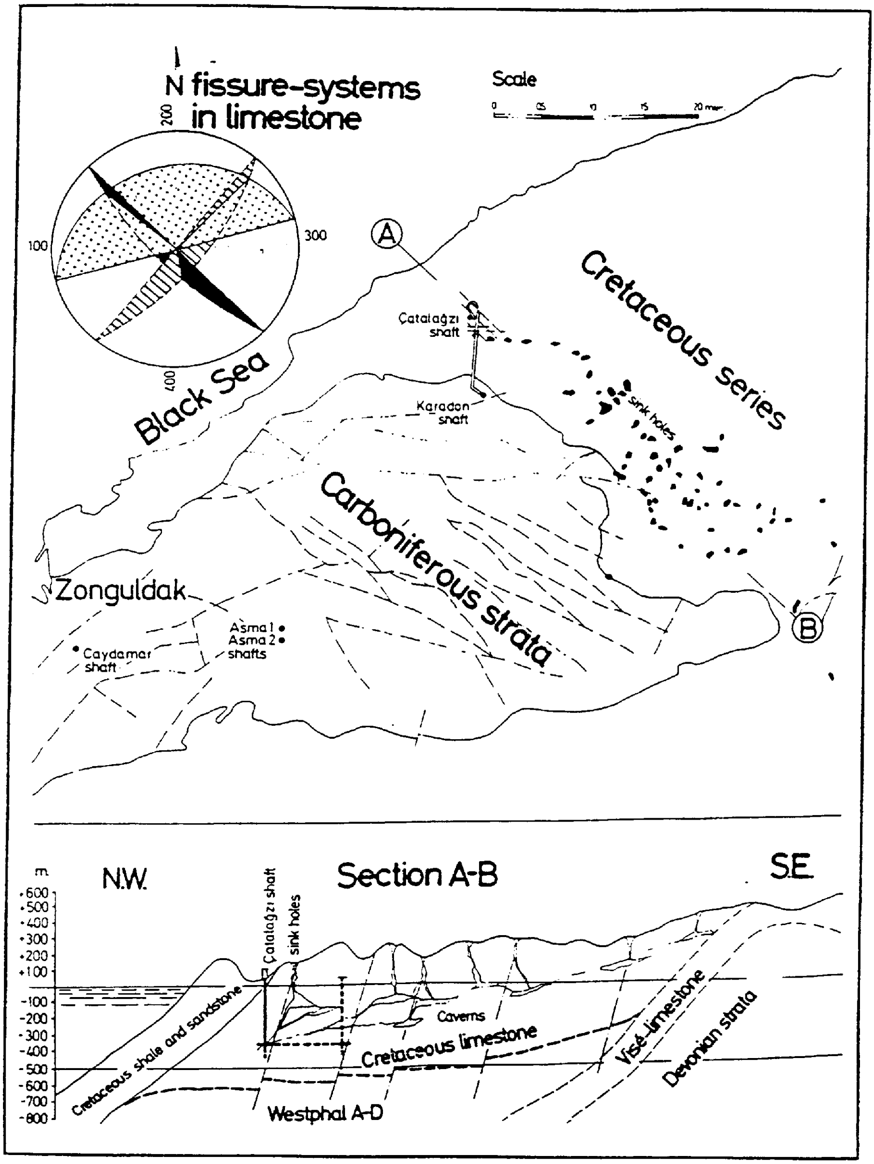


Figure 5 Geological structure of site 2 in the vicinity of Karadon and Catalagzi shafts (Mattes, 1975)

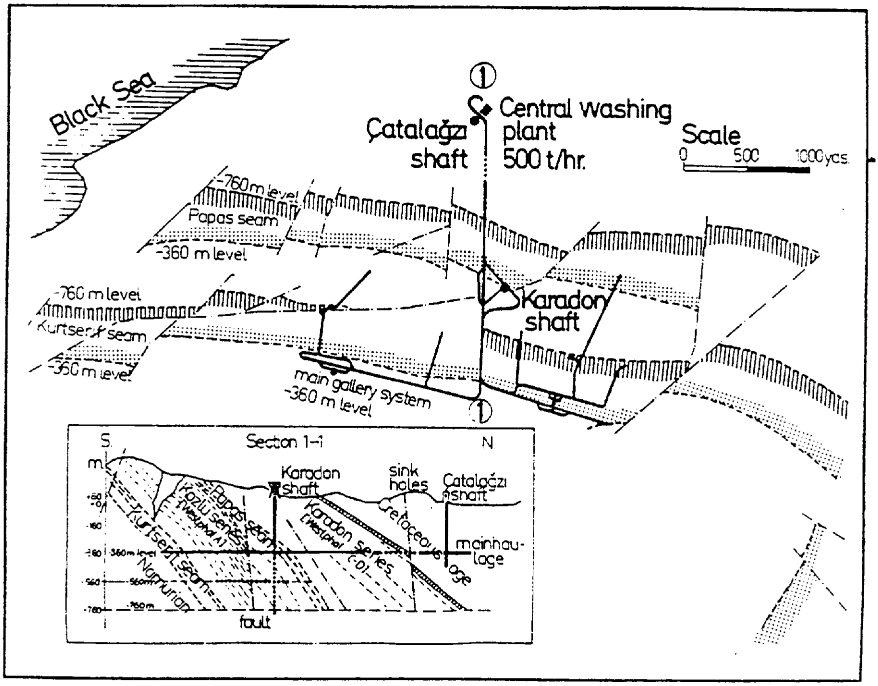


Figure 6 Plan and section of Karadon and Catalagzi section (Mattes , 1975)

The Catalagzi shaft subsequently filled up with water to a level of 16 m below the shaft collar. The shaft was then dewatered by using 11 stage submersible electric pumps with a capacity of 7.2 m^3/ min. It may be noted that the help of a diver was needed to remove shaft debris. After two and a half years and four major floods the shaft was secured by a water dam 120 m north of the shaft. In order to isolate the major feeder intersecting the karst aquifer concrete water dams with watertight welded steel doors were required to seal the water feeder. A second water dam was also built on the south side of the shaft and equipped with a water tight door. Only after the shaft had been protected by these dams, the mining of the main haulage to the Karadon mine started. It rrequired $2\frac{1}{2}$ years using 11-stage submersible pumps and at a cost of $1.6 million to dewater the mine.

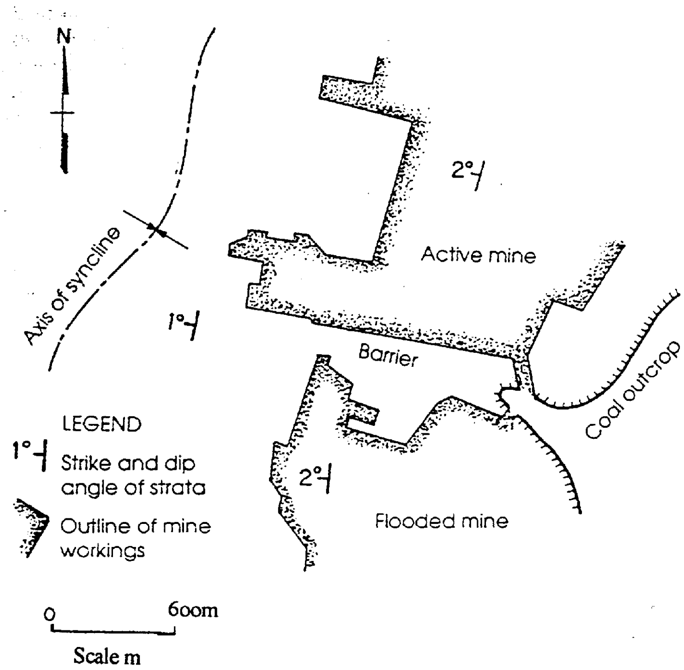
The lessons learnt from this case example are that geological and hydrogeological studies are necessary requirements for major development projects in coal mining. An advanced bore hole should be drilled at the site of major shaft and underground main development headings.

(2) Ineffective Barrier between the active and the waterlogged mine

Case History 3: Inadequate thickness of barrier, Northern West Virginia coal mine

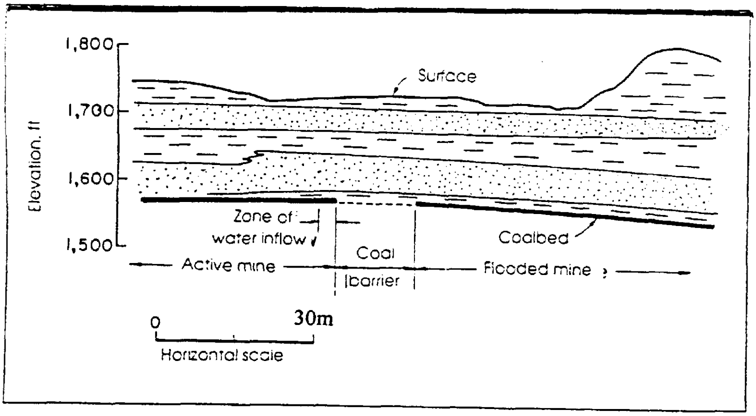
The North Virginia Mine, which was opened in 1982, adjoined a flooded mineand was designed to work in close proximity old water logged workings. As a consequence, a safety barrier designed to be 143 m wide (Figures 7 and 8). In addition to the adequate size of barrier, some test bore holes were drilled before approval was granted for mining in the vicinity of water-logged workings by the local mining inspectors, American MSHA. At the time of granting approval for mining the following precautions were imposed on the mine operators when working in the proximity of old mine workings:

- Monitoring the water level in an adjacent mine.
- Indicating the minimum time available for warning, evacuation or rescue.
- Assessing the impact of water inrush on escape ways and mine ventilation.
- In estimating water inrush, account must be taken of the depth of the impounded water, which affects the pressure at which inrushes will occur.



In this mine a barrier of apparently more than adequate dimensions failed to prevent a substantial influx of water from the adjacent flooded mine. Water first entered through roof-bolt holes and slickensides in the roof with only minor seepage through the coal seam. Analysis confirmed that the seepage was mine water rather than ground water. Dewatering the flooded mine was estimated to take a minimum of 180 days continuous pumping at a rate of (63 l/s) without allowing for any recharge.

Three bore holes drilled in the mine roof and one from surface revealed a sandstone-filled paleochannel (Figure 8a and b) adjacent to the zone of heavy water inflow. Pathways for the water lay along the margin of the channel where deformation of the roof strata had occurred. Although no evidence was obtained which could confirm the



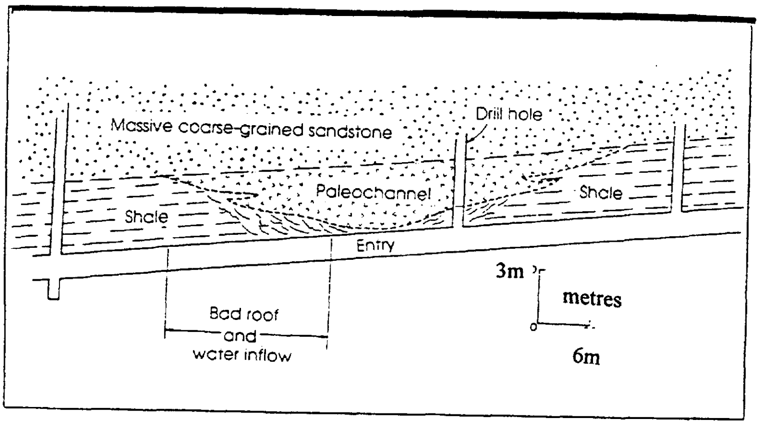


Figure 8 (a) Transverse profile of the barrier zone (b) Paleochanels in the roof strata of the active mine.

continuity of the paleochannel across the entire width of the barrier, no explanation other than an anomalous, permeable geological structure seemed warranted.

This case example demonstrated that the barrier pillars between the water logged workings and the active working should be free of geological structures and anomalies.

(3) Dangerous Mining Practice: Lack of risk assessment

Case History 4: Subsidence zone intersecting the Tailings Dam, Mufulira mine, Zambia - 25th September, 1970 - 89 persons killed

Mufulira mine situated in copper belt, Zambia was one of the most important producer of copper ore in Roan Consolidated Mines Ltd. (Anon, 1971, Nellor, Sandy and Oliver, 1973). The mine produced some 7.6 million tonnes per year of copper ore at 2.49% copper, producing some 176,750 tonnes copper metal. At the time of the incident total ore reserves were 176 million tonnes at 3.28% copper. The ore deposit comprised 3 ore horizons having a dip of 45° to the north east. The lowest and largest ore body 'C' had a strike length of 5.4 km., and average true thickness of 13.5 m. The mine was pumping well over 62,297 m^3 of water daily (721 l/s).

In the eastern part of the mine where the three orebodies are super-imposed, a block caving method was used for many years. In the later years this was replaced by a variation of sub-level caving called the cascade method. The eastern part of the mine was served by No. 5 and No. 7 vertical shafts and the three Peterson sub-inclined shafts from the surface to give access to the deeper part of the ore body. The underground, sub-level caving operation at Mufulira created a collapse structure on the surface. In 1963 it was decided to backfill the subsidence trough or sinkholes with mill tailings.

On September 30, 1970, a sink hole appeared on the surface through which a mud flow, composed essentially of concentrator tailings, poured into the mine, killing some 89 miners and causing serious damage to the mine. Within a 15-minute span in the early-morning hours of 25th September, 1970, an inrush of about 453,070 m^3 of mud from the surface flooded much of the Peterson section of the mine, trapping and killing miners and wreaking extensive damage to the mine itself (Figure 9). The bulk of this mud originated from a large accumulation of saturated tailings from the concentrator, which had been impounded on subsiding ground above the mine workings. A sinkhole with an associated collapse chimney developed beneath this accumulation, creating channels leading to the underground workings; through these channels, tailings flowed into the mine through several major ingress points between the 500m and 580m levels, and thence to lower levels via ore passes, raises, and the Peterson shaft system. As a result, the Peterson section filled with mud and water from the 500m level downwards.

This inflow of water and mud was a case of accidental inundation unprecedented in the history of mining. The risk associated with bad mining procedures and geotechnical instability of backfill could not be foreseen by the mining officials as well as the government inspectorate. Erroneous beliefs, poor communications and lack of engineering judgment can be attributed as the basic cause of the disaster.

While the 5-man Commission, chaired by a mining expert Dr. Nicole Mihailovici, found that no individual can be blamed for the tragedy, its report does maintain that "since no freak occurrences of nature were involved, the final responsibility inevitably must rest with the company. The accident was caused by faulty operational procedure: a (mining) method, a tailings disposal practice, and a drainage scheme, all sound enough in their isolation, became a dangerous combination."

Although some of the tailings that entered in the mine as mud flow may have dated back to as early as 1933, more than 75% had been emplaced since 1963, when management decided to use this material to fill in the surface depression caused by underground

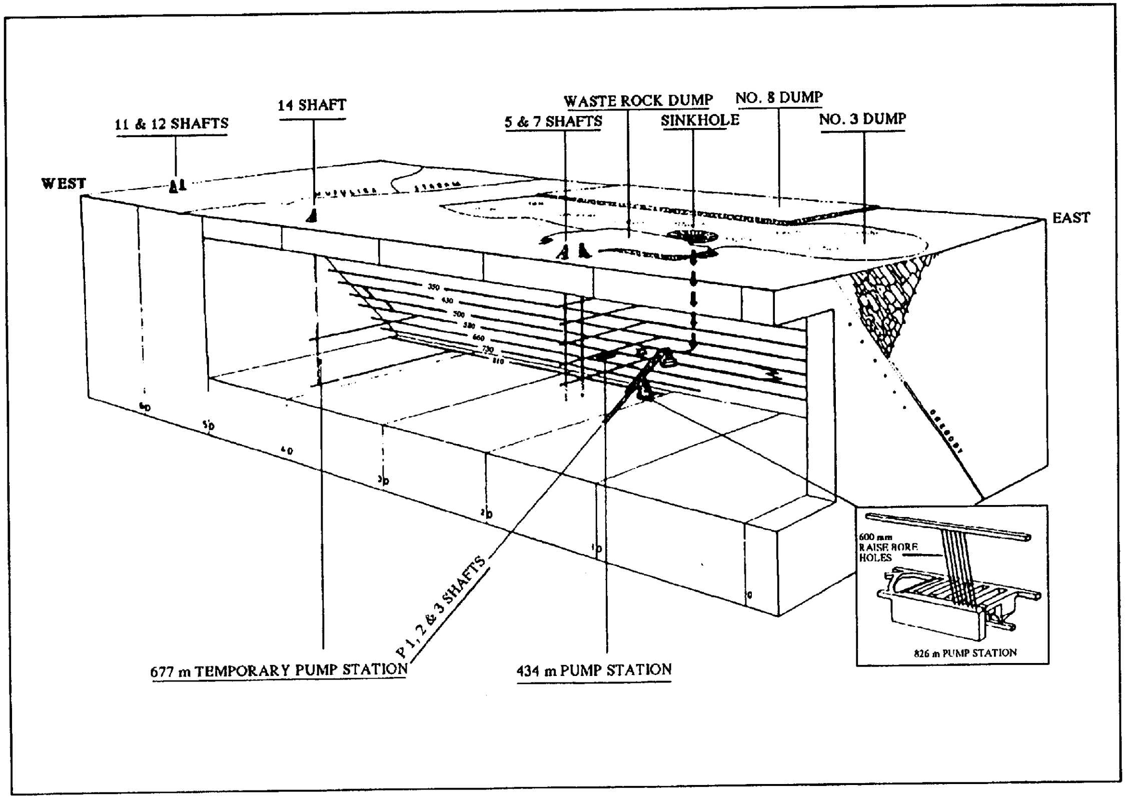


Figure 9. Mufulira inrush arrea showing position of shafts, main levels, pump stations, and the sinkhole on the surface with the probable entry point of the mud and tailing into the mine workings. The inset shows the arrangement for introducing submersible pumps into the sumps (after Neller et al., 1973)

caving of ore, thereby preventing the accumulation of water that might seep into the mine. This practice of disposing tailings on caving hangingwall ground was termed "basically wrong" by the Commission. Although it was recognized in 1967 that there was a potential for the "dangerous combination" which actually caused the disaster, the possibility of such a disaster occurring was discounted, the Commission reported. This happened "at least partly because the re-appraisal (after the Aberfan, Wales, tragedy in 1967) was undertaken by those persons who had made the original decision to infill the surface subsidence area with tailings in 1963. In addition, " the report continued, "everyone concerned was almost certainly unconsciously influenced by the fact that any doubts about the practice raised problems which, by lapse of time, had become almost insurmountable".

"A truly objective appraisal of the situation at Mufulira prior to September, 1970," concluded the Commission in its report, "was almost impossible since both the company staff and the Mines Department had to a large extent been *conditioned* to the situation by long association.". It was further suggested by the commission that chimney caving in the central part of the caving area, and particularly the mud extrusions which occurred underground between April and September of 1970, should have been interpreted as warning signs by management of the impending calamity, but they were not. The Commission said that management was "under the misapprehension" that any tailings penetrating the hangingwall would be a controllable inflow into the mine workings, giving ample warning. This belief was held despite the fact that such an inflow would take place under a hydrostatic head of several hundred metres.

The Commission went on to criticize what it called "ineffective communications" at the mine. "Only a small number of senior officials were aware of the events which had occurred both on surface and underground prior to the disaster. There is a lack of records concerning the critical decisions which were made at various times, namely the decision to adopt a caving method of mining in 1946, the decision to infill the surface depression with tailings in 1963, and the discussions regarding the significance of the underground mud extrusions which took place prior to the disaster. There is no evidence that regular formal discussions between the management and the senior technical personnel were held and vital information was apparently passed to decision-making levels in a very informal manner.

The Mines Department also came in for criticism from the Commission. Although no evidence exists that the Mines Department was ever informed of the decision to infill the surface subsidence with tailings or that the matter was ever discussed with the Mines Department, the Commission pointed its finger at departmental officials by noting that "the (infilling) was there to be seen and, as there is no record of adverse comment, it must be assumed that the practice was accepted. The rehabilitation of the mine has been discussed by Neller et al. (1973).

This disaster proves that before starting a new scheme it is prudent to carry out a risk assessment for men, equipment and process before adopting it.

(4) Spontaneous Inundation: due to mine workings encountering a major feeder of water intersecting natural karst aquifers.

<u>Case History 5</u>: Development workings holing to water bearing fissure in Kombat mine, Namibia - 1988

Goldfields Namibia Ltd. operates three base metal mines in Namibia producing in total 2 Mt/y. There was a major interruption to production at the Kombat mine in late 1988 when a well known water-bearing fissure was holed during blasting. This released an estimated 5 Ml/hr of water into the underground workings. The accident happened as a result of pre-cementation work under way some 535 m below surface. In little more than 4 days, the mine was completely flooded. The water level rose to just 50 m below the shaft collar.

Immediately two drill rigs capable of boring 165 mm diameter holes were used in an attempt to reach the development end and the breached fissure. However, the holes were deflected by adverse ground conditions and it was impractical to continue. Two large oil drilling rigs were at that time available in South Africa which could be used for obtaining access to the site of water inflow if it was feasible to drill holes of sufficient diameter bore holes. Moreover accuracy of drilling was important so as to intersect the 535 m level with the intention of filling about 50 m of the drive with cement up to the fissure to form a plug. It was also feared that the accuracy of drilling would be affected by rock formations and known magnetic anomalies underground which would affect any magnetic instruments. The proposed solution entailed drilling a 312 mm diameter primary bore hole vertically into the 5 m wide drift. This would be used to supply the cement necessary to fill the complete cross-section of an area of the drift some 45 m back from the face forming a primary barrier. This would be followed by more bore holes deflected from the primary hole to intersect the drift at equal distance from the face and the primary barrier. Thus, two more sections of the drift could be isolated with cement forming a 45 m thick plug. To achieve these intersections, the target area for each hole was within a radius of 2.5 m.

The primary bore hole was drilled with Deutag's T-12 rig which is a diesel mechanical Ideco H-525-D Dual Rambler. The mast has a nominal gross capacity of 117 tonnes and a hook load rating at 113 tonnes at six drilling cables. The draw works feature a Micromatic type CB automatic driller powered by two Caterpillar D343PC-TA diesel engines. Major logistics difficulties required in mobilizing the men, drilling equipment and necessary ancillaries a total distance of almost 1500 km. A special South African transport services train of 24 rail cars and 12 road trucks were used to haul a total 580 tonnes of equipment. This enabled drilling to start only ten weeks after the water was released.

The primary bore hole hit the drift 21 days later, a mere 200 mm away from the centre of the target. Directional control was achieved by a steering tool and down-hole motor to correct deviations caused by difficult strata conditions. The steering tool sent data from the bore hole bottom via an umbilical line to the surface. Data included direction of hole, direction of face and drill bit, inclination of bore hole, magnetic intensity and other parameters. This information was used to steer the down-hole motor which, by means of rotation at the bore hole bottom and a bent sub, allows the hole to be directed to the desired direction.

Once the primary bore hole had been completed, some 140 m^3 of cement and aggregate were delivered down a 965 mm diameter casing pipe to form the primary barrier. This was achieved in five days. The primary bore hole was cemented back to 300 m below surface and the first side track was drilled. This bore hole intersected some 22 m closer to the face along the centre line of the drive. A further 650 m^3 of cement and aggregate were then placed, filling the drift between the primary barrier and the face. This bore hole was cemented back to where it broke away from the primary bore hole and the second side bore hole was drilled. This intersected along the centre line some 40 m from the original bore hole. This second bore hole gave evidence of concrete and it was assumed that the necessary sealing had been achieved in that area. A third deviated bore hole intersected the drift about 2 m back from the face, where it was only necessary to inject 18 m^3 of concrete into a small void which was discovered, thereby demonstrating the effectiveness of the two plugs. Dismantling of the drill rig began on 29th March and shortly thereafter dewatering of the mine commenced. By the end of September water level had dropped to 353 m below surface or 295 m below the highest water level. As soon as , No.3 shaft was dewatered and re-commissioning of the shaft and pump station was initiated .The complete reclamation from setting up the drill to the final concrete tests was completed in the record time of 62 days.

The grouting operation needed special considerations using Forsoc's special underwater additive and retarders to prevent the cement from being washed out of the concrete mix. A strong super-plasticiser additive which retarded the concrete for up to 6 hours was also designed to produce a high slump, very flowable but cohesive creamlike concrete which could be pumped down 530 m vertically as required. The mix allowed for the effects of free fall in the pipe and the placing of concrete under a very high water pressure of 4.8 MPa.

Case History 6: Stope intersecting water bearing fissure in West Driefontein mine, South Africa on 26th October, 1968

The inrush of water into West Driefontein mine, the biggest gold producer in the world, which began on 26th October 1968 and continued unabated for 23 days before the flood (18 Ml /day) was plugged, was unprecedented in the history of mining. Also unprecedented were the tremendous efforts and organization which made it possible to bring the situation under control, thereby saving the mine from being lost for a matter of years, Cousens and Garrett (1969).

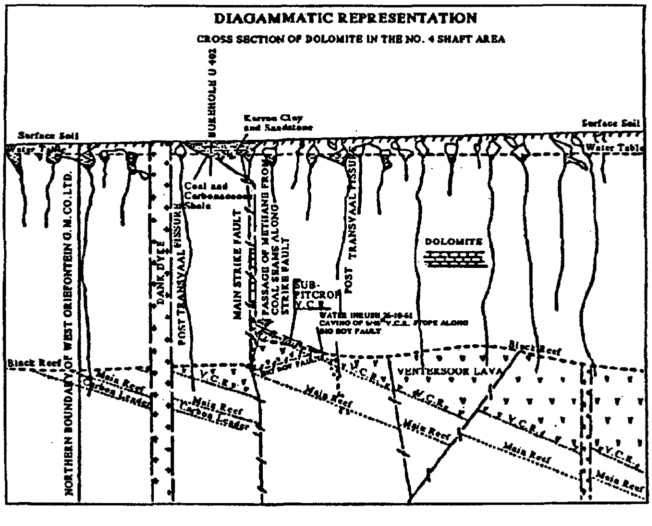


Figure 10 Diagrammatic representation of solution cavities and fissure formation (Coussens and Garret, 1969)

On the morning of 26th October, 1968, an examination of the hanging wall in a stope between 4 and 6 level, No. 4 shaft (Figure 10) at a depth of 863m below surface, indicated some movement and a small amount of water inflow. At 9.30 a.m. a fissure opened in the stope and large quantities of water rushed into the mine.

The first reaction was that it was a minor emergency which could be dealt with in a routine manner - as so many others had been - by plugging the fissure or simply by diverting the water. To those underground, however, it soon became clear that this was no ordinary inflow and that there was serious danger to a great many people working in many parts of the mine - not least in the No. 4 shaft pump stations. A decision was therefore taken to withdraw personnel immediately; in this withdrawal some highly courageous actions were performed.

At the time there were approximately 1350 men in the mine, about 1200 of whom were in the No. 4 shaft area. Some 400 of these were hoisted up No. 4 shaft, the remainder scrambling along 10 and 12 levels, the only connections to No. 3 shaft and other operating shafts. Even at that stage it was not realized that tremendous quantities of water were pouring into the mine; nor was it appreciated that what had been thought to be over-generous precautions against any such emergency would prove inadequate and would have to be supplemented by drastic measures which were only just able to avert complete flooding of the mine.

The plan of campaign adopted after the inrush of water and the reorganization and expansion of pumping arrangements was discussed by the authors. When it became clear that the only means of saving the mine was to construct plugs to isolate the inrush, while maintaining all emergency measures, the work was put in hand at once.

In considering the design of the plugs, some major factors had to be taken into account, the most vital of which was the need for speed - which was complicated by difficulty of access to the plug site because of the huge volumes of water flowing in the drives. How this difficulty and other problems were overcome was also detailed in the paper. This incident highlights the notion that strict management system and technical expertise are necessary to detect and control ground water inflow potential when mining underneath a major karst aquifer. In addition to drilling advanced boreholes to locate a large reservoir of water, a large sump and installed emergency pumping capacity and management plan are necessary to meet such emergencies.

Case History 7: Jefferson City mine, Mascot-Jefferson City Zinc district of East Tennessee 7th November, 1961

The Jefferson Mine is situated in Mascot-Jafferson city in East Tennessee, produces Sphalerite ore imbedded in a dolomite and limestone bed 2.4m to over 30 m thickness which dips at angle of 7° to 10°. Active mining is carried out in the 1st, 2nd, 4th and 6th levels at a depth 286m, 301m, 331m and 361m respectively below the shaft collar. Mine water is pumped in 3 stages, from the 6th level 75 m to the 1st level, from there 57 m to the 96m level, and then 96 m to the surface. In early November 1961, the installed pumping capacity , consisting of 2 sets of 112-kW, 3550 rpm pumps, was 202 l/s. The normal mine inflow was 13 to 19 l/s, thus providing a safe margin of extra pumping capacity. (Miller and Jolley, 1964)

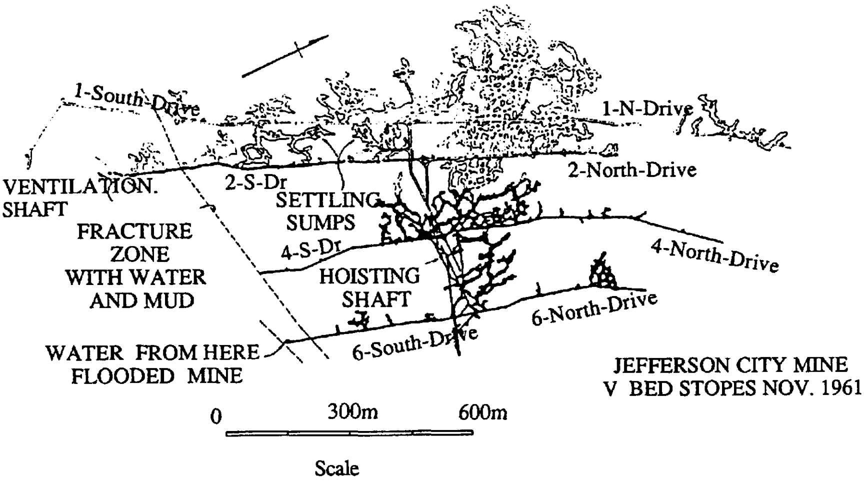
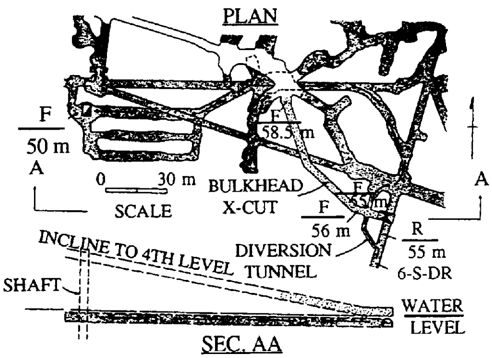


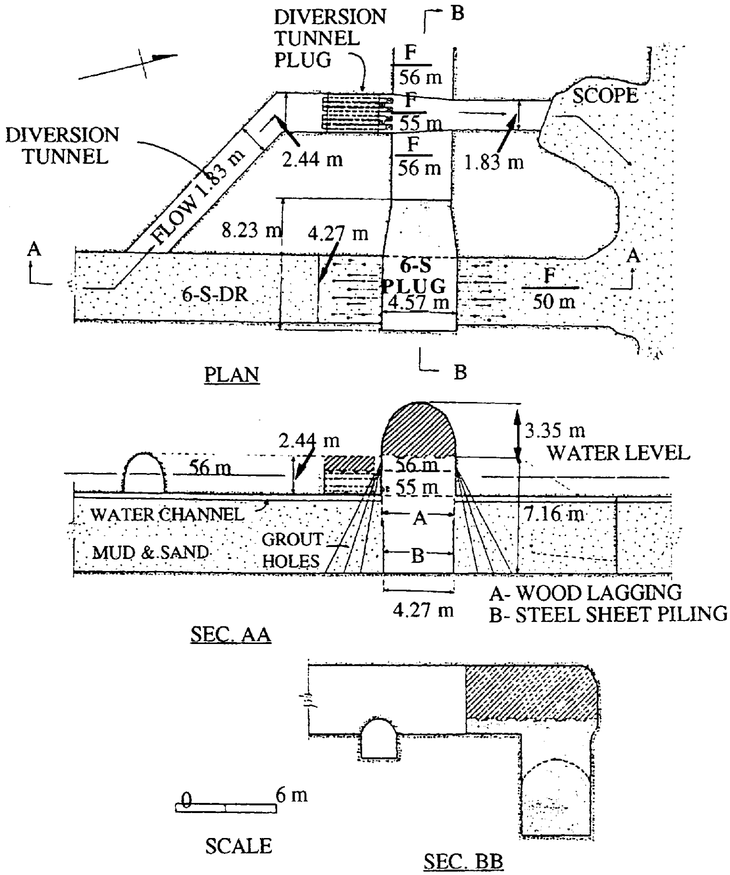
Figure 11 Cross section through Jefferson City Mine (Miller and Jolley, 1964)

On the south western part of the mine a highly fractured water bearing zone comprising steeply dipping fractures and faults with minor displacement s was known to be present. Examination of the exposed fracture zone reveals a series of lenticular voids up to 152 mm in width with the rest of the zone water tight. The pattern of the voids in the plane of the fault resembled a lacy structure. Water, heavily charged with mud and sand, had been encountered within this zone by drill holes during drivage of 1-S, 2-S, and 4-S drifts. Each time the water was controlled by placing packers in the holes. Drilling of precautionary pilot holes in advance of the drift face was, therefore, unreliable. It was believed that at greater depth this zone would be comparatively dry because the fractures and voids would have tightened with depth.

In driving 6-south drift, in moderately oxidized minor fault zone, a newly exposed face intersected the known water bearing zone. There were two or three small lens shaped voids up to 102 mm wide along the fault plane which yielded some 4 to 7.5 l/min of water. The water potential of this fracture zone was under-estimated and proved to be far in excess of any previously known in the district. On 7th November, 1961, a night crew started drilling a round in 6-S drift 472 m southwest of the shaft crosscut and 372 m below the surface, when the drift hit watery ground. Their jumbo had 3 drills which were simultaneously drilling the first bore holes, all 32 mm in diameter. Four 4.3m pilot holes drilled in advance of the previous round had not encountered water. The left bore hole and the central hole had reached a depth of 2.4 to 2.7 m. At a depth of about 0.9 m, the right bore hole encountered mud and water. Initial inflow was estimated at 19 - 26 l/s at an approximate pressure of 3.1 - 3.5 MPa. The first attempt to insert a packer was unsuccessful due to a lack of clearance between the packer and the hole, and as a consequence a leak developed in the packer. Subsequently, another packer was inserted and the water shut off for about 4 hours before a leak developed. The abrasive sand and mud badly eroded the packer, culminating in complete failure 8 hours later. All further efforts to secure a packer in the hole were unsuccessful due to



A: Development (stipled area) for 6-S plug required driving a 81 m cross-cut an 25 m diversion tunnel that connected with 6-S drift. Shaded areas represent working full mud



B: Details of the diversion tunnel and 6-S concrete plug

Figure 12 Details of access drivage to make a friction plug in 6 level south.

the erosion of the bore hole and the high pressure of the mud and water. Efforts to contain the water in three 102-mm pipes by constructing a concrete dam against the face were also unsuccessful. As a result, the 6th level was completely abandoned on the 8th day.

Water continued to enter the mine until the highest water mark occurred on 29th December, 1961, when the water reached a point 3 m above the 2nd level or 63 m above the 6th level (see Figure 11). At this moment, the estimated inflow rate had increased from 57 to 353 l / s. In order to contain the inflow, attempts were made to grout the drivage face from the surface without any success. As an alternative to grouting, the pumping capacity of the mine increased by installing three air lift pumps in the shaft. As the water level decreased the presence of mud increased with depth. It was estimated that 135,000 tonnes of mud and sand entered mine. The total quantity of water stored in the mine was 196,820 m^3 (196 Ml) With great difficulty the water was lowered to 5.8 m above the 6th level by 9th May 1962. By this time it was obvious that it would be practically impossible to lower the water to the 6th level and clean the mud in 6-S drift to a point where a concrete plug could be constructed.

At this point of time, it was decided to build a friction type plug at 6-level South to seal the inflow. To achieve this end, a plan was devised for plugging off the 6-S drift without the need to lower the water level below a 55m datum viz. 6 m above the 6th level station. This plan involved building a large concrete plug in the 6 level south drift from the top of the drift and a small concrete plug with valved pipes in an adjacent diversion tunnel. Both plugs were the friction type with no hitches or reinforcing steels except for some steel pins in the diversion plug. A crosscut 82 m long was driven from a stope which had been cleaned of mud at elevation of 57m, to a point 1.5 m above the back of 6 level south drift. At the elevation of 55m, a 1.8m by 2.4m diversion tunnel 25 m long was then driven both north and south to within one round of connecting with the 6 level south drift. The elevation of the bottom of the diversion tunnel was 54m , the same as the back of 6-level south drift The detail of the diversion tunnel and the 6-S concrete plug is shown in Figure 12. This operation virtually stopped the inflow and the mine was ultimately dewatered and restored after interruption of production for a period of 10 months.

CONCLUSIONS

This paper has demonstrated that accidental inundation is one of the major hazards in mining in the vicinity of old water logged workings or workings near hydrogeological anomalies. Until recently, prescriptive coal mines regulations have been used to prevents these hazards. However, prescriptive regulations alone are not adequate to control all accidental inundation in a range of circumstances. In addition to the prescriptive coal mines regulations, more scientific investigations, risk assessment techniques and management emergency plans are needed to control accidental and spontaneous inundations.

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