THE FAILURE OF & REMEDIALS TO A RIVER DIVERSION FOR AN OPENCAST MINE IN THE WITBANK COALFIELD OF SOUTH AFRICA

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ABSTRACT

Arthur Taylor Colliery Opencast Mine (ATCOM) is situated in the Witbank coalfield which lies in the eastern highveld of the Republic of South Africa. The catchment typically experiences heavy afternoon thundershowers during the summer season resulting in periodic strong flooding and rivers and streams coming down in spate. A medium size (by South Africa standards) river, the Steenkoolspruit, bisects the mining area of ATCOM and traverses several coal seams at shallow depth, suitable for a mine producing 2,4 million tons per annum of export quality (28Mj/kg C.V.) coal, which prompted the company to construct a diversion to relocate the river some distance to the west of the natural position and permit opencast mining through the river bed and later on again to relocate it to the eastern flank over the mined-out, back-filled ground for final mining of the western section of the property.

The river diversion was constructed to accommodate a flood of peak discharge of 2400 cubic metres per second (recurrence interval of one in 600 years) and consisted of a canal 34 to 36 metres wide, up to twenty metres deep and approximately 5 kilometres long. A portion of the canal passed over previously mined-out ground in which bord and pillar mining had been carried out. This section was excavated to the floor level of the old workings and back-filled with an engineered fill to the invert of the canal.

On the 20th October 1992 the deviation was opened to the full flow of the Steenkoolspruit and on the afternoon of 9th November 1992 a largish flood (estimated at between 50 & 100 cubic metres per second) entered the canal while simultaneously a storm occurred on the mine resulting in the failure of the canal invert and water ingress into the old workings and thence into the newly mined boxcut pit of the opencast mine.

This paper describes the events leading up to the failure, the failure itself and dewatering of the pit; the remedial measures taken and the subsequent performance of the scheme.

INTRODUCTION

As is to be expected, the coal seams of the Karoo System of the coalfields in S.A. are usually shallowest beneath the main surface drainage lines and therefore most economical to exploit by opencast mining. In the past it was the policy of the S.A. authorities not to permit mining within a stipulated distance of a river. The consequences of such a policy were not only to sterilise extensive valuable coal resources but also to form potential blockages to movement of possibly contaminated ground water that could subsequently rise and enter the river. The modern tendency has been to allow deviation of the river under controlled conditions laid down after careful study of the subsequent migration of ground water. Deviating a public stream was subject to compliance with Section 20 of the Water Act (Act 54 of 1956, as amended). (Note: this Act has recently been replaced with a new Act (Act 36 of 1998 National Water Act)).

Arthur Taylor Colliery Opencast Mine (ATCOM) of Tavistock Collieries Limited, at the time a subsidiary company of Johannesburg Consolidated Investments Limited (JCI), is located in the Eastern Transvaal coalfields near Witbank, as shown on Figure 1. As may be seen (Figure 2) the opencast workings are traversed by the Steenkoolspruit, a tributary of the Great Olifants river which is one of the major rivers of southern Africa and passes through the Kruger Park game reserve on its way to Mozambique. The Steenkoolspruit is an important river as it conveys water transferred from escarpment rivers in the adjacent catchment to the Olifants river dams which supply large volumes of water to industry, agriculture, mines and power stations. The catchment area of the Steenkoolspruit upstream of the ATCOM property is 1400 km^2.

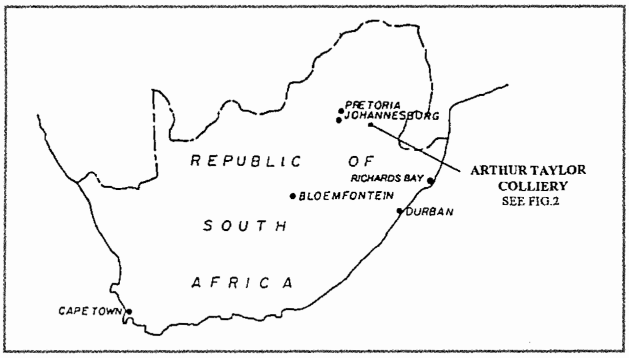

Figure 1. Locality sketch.

The proposal at Arthur Taylor Colliery Opencast Mine was to deviate the Steenkoolspruit temporarily to the west, as shown on Figure 2 so as to clear a large area for opencast mining (including the seams below the riverbed) and subsequently to relocate the river permanently over mined-out ground along a route to the east of the presence course. Stringent specifications are imposed by the South African authorities for diverting a river, particularly for re-locating a river over mined-out backfilled ground.

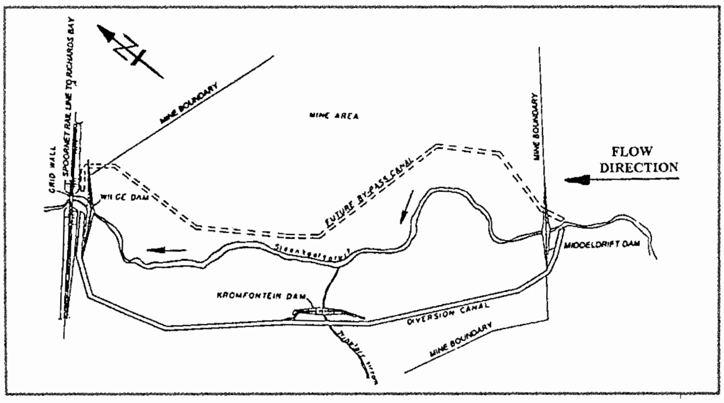

Figure 2. Layout of river diversion.

The major works involved for the Phase 1 scheme were:

- the main upstream diversion dam (Middeldrift)
- the damming of the tributary (Kromfontein)
- the downstream dam (Wilge)
- the large diversion canal.

Stipulations by the Government authority, Department of Water Affairs and Forestry (DWAF) in terms of the Water Act are usually to the effect that a permanent river diversion should preferably be designed to accommodate the Regional Maximum Flood (RMF). Temporary diversions may be designed for flood discharges of other frequency of occurrence but usually the capacity of a deviation is determined by the economics of the situation.

Furthermore, the requirements are that the energy at the downstream confluence of the deviation with the original river course must be the same via the diversion as before the diversion.

The major components of the river deviation, the sizing of which was determined by an economic analysis which in turn was based on the frequency distribution of flood discharges and volumes are described.

DESIGN AND CONSTRUCTION

Hydrology

The area of the catchment drained by the Steenkoolspruit upstream of the mine is approximately 1400 km^2. Flood peak discharges for various frequencies of occurrence were calculated by standard procedures (Midgley, updated 1979; Alexander, 1990; Kovaes, 1988). For the statistical approach of Alexander[2] (1999), the annual maximum flood peaks were also abstracted from the reasonably long and reliable record of official (DWAF) streamgauging station BIMOI (1903 – 1953) commanding a catchment of 3 989 km^2 of the Olifants river downstream of Witbank dam.

As shown on Figure 3 the frequency distribution of flood peaks derived from the streamgauging and that derived from unitgraph procedures[1] plot closely together. Kovacs[3] (1988), however, has suggested values for the 200 – 100 – and 50-year return periods as fractions of the RMF and these are invariably higher than those derived by other methods. Figure 3 shows the three distributions, with the RMF value plotted at the projection of the Kovacs values, viz. approximately 600-year return period. Also shown is a PMF (Probable Maximum Flood) value derived by unitgraph procedures with probable maximum precipitation (PMP). For convenience some values are listed on Table 1, the shorter return period values being averages of the statistical and unitgraph values.

Return Period (Years)	Peak Discharge Cumecs (m^3/s)
30	600
50	750
100	1 050
200	1 450
600 (RMF)	2 400
10 000 (PMF)	4 000

Table 1.

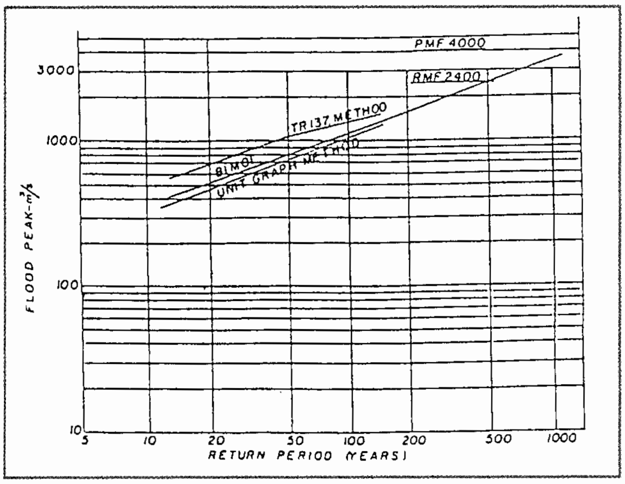

Figure 3. Frecuency distribution of steenkoolspruit flood peaks.

The approximate return periods assigned to RMF and PMF are specifically for the purposes of the hydro-economic analysis. Flood hydrographs of the Steenkoolspruit were generated by unitgraph procedure as shown in Figure 4 for purposes of establishing volumes of flood water likely to have to be diverted during floods of various return periods. A hydro-economic analysis was conducted on the diversion from which the 600 year recurrence interval flood was chosen as the design flood with a peak discharge of 2400 cumecs, meaning that spillways to the inlet and outlet dams would not be required in terms of DWAF requirements. Figures 5(a) and 5 (b) indicate the levels in the canal for this flood.

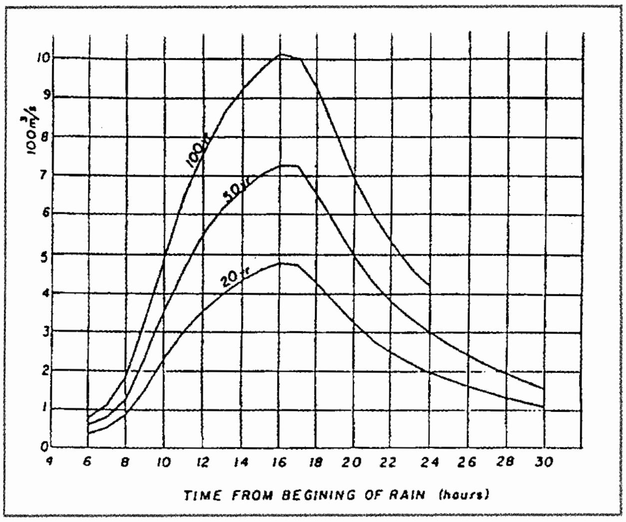

Figure 4. Typical hydrographs for varius return periods.

Hydraulics

Routing of floods through the system required determination of the tailwater rating curve for a section at the canal outlet to provide starting levels for backwatering the system so as to meet the DWAF requirement that there be no variation in the energy line before and after deviation. Backwater calculations for a range of flood discharges from several kilometers downstream of the works back to the diversion canal outlet were performed. Backwater calculations were then extended up the diversion canal to the upstream diversion dam. Canal cross sections are shown on figure. 5(a) and 5(b). The backwater water surface profiles determine the required full supply level of the canal and the heights of Kromfontein and Middeldrift dams. A Manning "n" value of 0.035 was adopted for the canal excavated by blasting without presplit. This mean an absolute roughness of $\pm$ 0,5m – a value which in fact is being realised.

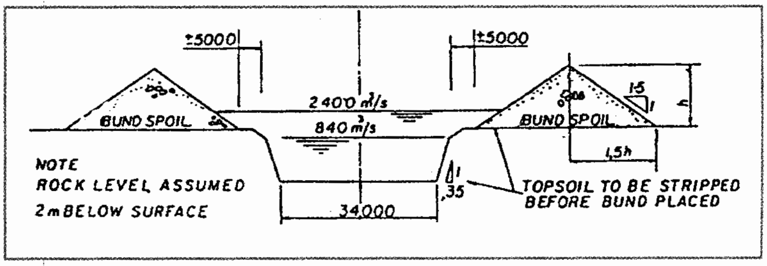

Figure 5(a). Typical shallow cut. Secction thru Canal (E.C. Chainace 0 562 m).

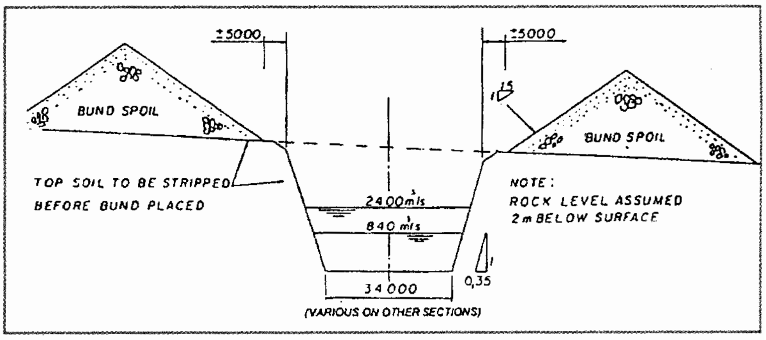

Figure 5(b). Typical deep cut. Secction thru Canal (E.C. Chainace 2156 - 4350).

Dams

As large volumes of mined rock were available on site, the upstream Middeldrift dam and downstream Wilge dam were designed as rockfills with central clay cores (Figure 6). The central Kromfontein dam was built before mined rockfill became available and is a conventional earth dam with horizontal downstream filter and uniform rolled earth fill; rip rap protection is provided to the upstream face. This dam was built ahead of the other two as it was required to trap runoff from the small tributary that led down into the workings and this it did very satisfactorily. In view of its temporary nature only a nominal temporary spillway was provided. As these dams do not have to retain water permanently and merely have to hold back the flood for a few days, the usual requirements for impervious foundations, viz. grout cap and grout curtain, could be eased and a simple cut-off provided. The uniform rolled earthfill of Kromfontein dam and the impervious zone of the rockfill dams came from the decomposed sandstones, shales and mudrock overlying the fresh strata of the pit area. Filter material was obtained either by passing waste rock over a grizzly with bars set at 75 mm or from essential excavation for the canal.

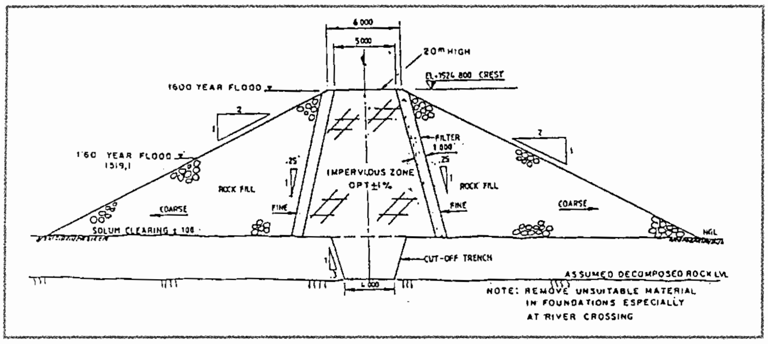

Figure 6. Embankment. Typical cross-section.

Two 1050 diameter R.C. pipes with valves were included in each of the upstream and downstream dam walls as drainage pipes to control future drainage of the mining area once the dams were commissioned. These pipes would permit a maximum discharge of 10 cumecs under flood conditions.

Canal excavation

The river diversion is 5,2 km long and is, on average, 17 m deep and 34 metres wide. To conform to the mining programme, excavation of the inlet and outlet sections of the main canal was carried out by contractors as the mine's 60 m^3 dragline used for opencast mining was not available for this operation. The dragline was however, used over a substantial length of the canal. This minimised the cost of the excavation as was evident in the hydro economic analysis. The total volume of the canal excavation was 2 500 000 cubic meters.

Underground bord and pillar workings of the previously mined areas at depths of, on average 6-8 m below the canal bed over a length of approximately 350 metres complicated operations at the downstream end (Figure 7). It was feared that heavy blasting could cause the roof of the workings to collapse. It was therefore decided to excavate the remaining coal beneath the diversion down to coal floor and backfill to engineered fill specifications with waste rock overlain with a 750 mm thick impervious layer and 250 mm thick coarse rock wearing course over a suitable filter (Figure 8). The revenue of the coal mined was offset against the extra cost of this operation. As an additional measure a berm wall 3 metres wide by 3 metres high of engineered impervious fill was constructed on either side of this section of the canal.

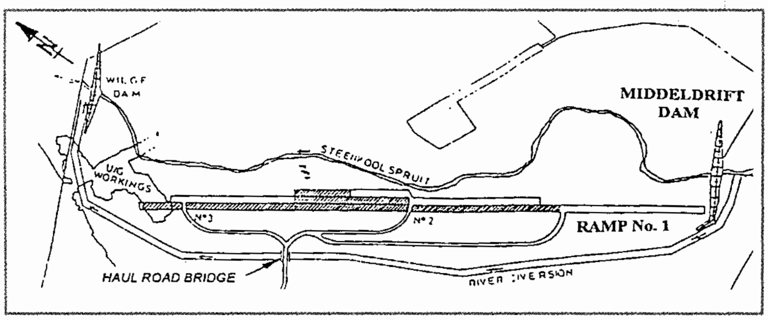

Figure 7. General mine layout. Showing the Steenkoolspruit and river diversion.

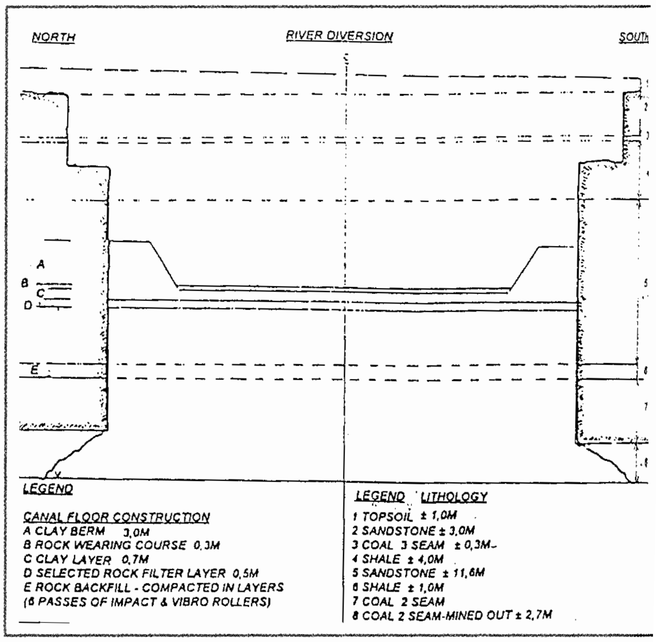

Figure 8. Diagramatic section. Showing the lithology and canal as built (backfilled portion).

Permanent bypass

Once the eastern area of the pit has been mined the present diversion will be replaced by a permanent diversion canal to be formed in the backfill on the east flank of the valley. This canal will be clay-lined to limit seepage into the backfill. DWAF stipulates that the capacity must be such as to accommodate the 1:50 year flood. Therefore spillways will be required for the Middeldrift dam to spill the excess flow into the pit and a similar spillway to handle this spillage at the downstream end at Wilge dam. These structures will be added at a later date when the balance of the pit on the west side is mined. (Subsequent to this agreed procedure considering the performance of the existing canal, DWAF has ruled that the canal should not be relocated).

Grid wall

To redirect the flow at right angles to pass under the Spoornet railway bridge (Figure 7) at the downstream end of the canal, a novel free-standing guide wall was designed using reinforced rockfill. This wall was founded on rock and on the downstream rockfill zone of Wilge dam. It was a simple solution to a potential problem, (Figure 9).

FAILURE OF THE CANAL

The canal was opened to the full flow of the river on 20th October 1992. On the afternoon of 9th November 1992 an estimated peak flow between 50-100 cumecs entered the canal owing to a series of storms in the upstream catchment of the Steenkoolspruit. At the same time a short duration storm with 34 mm rainfall occurred on-mine and large volumes of water

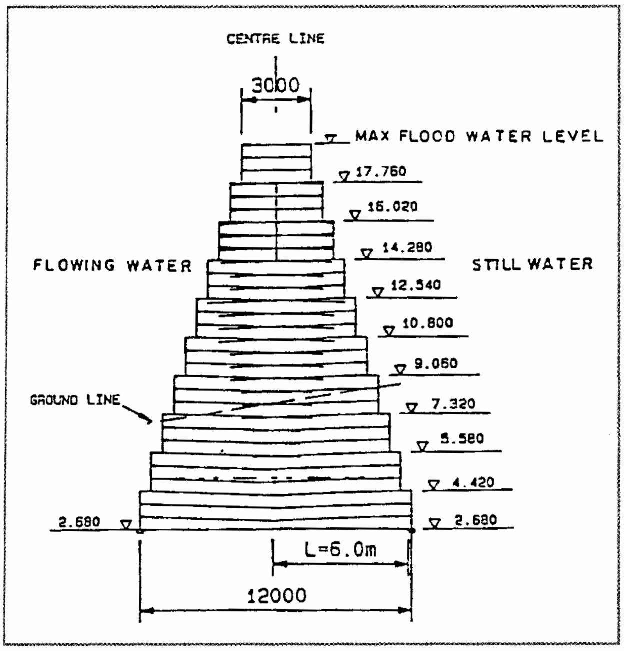

Figure 9. Typical section thru grid wall.

were observed cascading down the side rockfaces of the canal. Leaks developed through the backfill leading to the formation of large openings in the fill. The water entering the openings flooded the underground workings adjacent to the canal which in turn flooded into the opencast pit. This was as a result of the mining of the boxcut at the north end of the pit intersecting the old workings to remove additional coal (pillars and roof coal) as part of the opencast mine plan (Figure 7). A total of approximately eight hundred and fifty thousand cubic metres of water flowed into the pit in a period of about twelve hours.

Fortunately the mining operations were such that a 300 m long x 45 m wide void had been coaled out and only one diesel pump (size 50 l/s) was located there to handle normal seepage. Immediately south of the void, some 700 metres of blasted overburden from Strip 3 had been cast blasted into the Strip 2 void, upon which the dragline stood working. This 700 metres blasted material effectively served as a temporary plug by damming up the water which allowed time for all major mining equipment within the pit confines, to be safely removed.

An attempt was made to fill one of the holes in the diversion floor using two D10 dozers to doze loose fill material, located on either side of the diversion, into the hole. This operation commenced immediately and continued for 12 hours. This proved futile as the rate of flow of the flood water was such that all dozed material was washed into the underground workings.

The decision was then made to stop the flow in the diversion and return the water to its original course. By this time the estimated flow rate had dropped to between 5 and 7 cubic metres per second. The valves in both dams were consequently opened and the entrance into the diversion was blocked off with a rapidly constructed earth coffer dam wall. The flow of water into the diversion was halted by noon on the 10th November. A further series of coffer dam walls were constructed, one at the north end of the diversion to prevent the backed-up water from entering the failure area, and one in the middle of the haul road bridge diversion to provide storage capacity for future pumping to preclude water flowing down the R.D. and into the failure area.

On the 10th November the rate of water seeping through the 700 metres of blasted overburden rapidly increased to the point where the 9 diesel pumps installed to dewater the pit, each capable of 50 litres per second at the required head of 30 metres, were not having any noticeable effect on the inrush of water. Additional pumping capacity was installed including electric units to provide a total of 1 250 litres per second on 11th November. The estimated volume of water to be pumped was 850 000 m^3 of which some 600 000 m^3 had to be removed before coal production could re-commence. This was achieved and production was resumed on 23rd November using Ramp No. 2. Throughout the pit dewatering operation, water samples were taken twice daily to ensure that the Steenkoolspruit was not being polluted. Once the water volume in the pit had decreased, the sulphate levels and the conductivity levels increased and pumping into the Steenkoolspruit ceased. The water was then pumped into two sets of large settling dams prior to discharge into the river.

Obviously the remedials were extremely urgently required as the pit was under imminent threat of re-flooding since the wet season had commenced. A risk analysis based on storm events yielding floods greater than 10 cumecs (the capacity of the drain pipes through the dams) showed that there was a potential risk of flooding of the pit of 1:2 (50%). Several alternative remedial actions were considered to safeguard the mine from further diversion failure of which the following were investigated more thoroughly.

- immediate refilling of the openings to the old workings and the laying of a concrete lining over the undermined section to seal off the canal invert. This option was rejected owing to the fact that the concrete lining would have to be designed to span large holes which could occur beneath the lining, while there was also no guarantee that the backfill would not fail again through water infiltrating between the rock-concrete interface (i.e. not possible to ensure 100% water tightness without tedious and time-consuming preparation work on the rockface.);
- replacing the backfill and sealing off the canal invert with a plastic lining beneath the impervious layer. Rejected owing to the large design pressures the plastic would have to withstand coupled with the difficulty of achieving a water-tight seal at the rock interface on the side walls;

- the reinstatement of the original design layering but including a geofabric layer placed beneath the impervious layer to act as a support mechanism to ensure minimal settlement of the impervious layer, while assisting the filter layer. Rejected owing to the difficulties in fixing the geofabric layer into the rockface;
- grouting of the rockfill beneath the canal lining. This option was rejected by grouting experts owing to the tightness of the highly compacted rockfill which would be too resistive to grouting;
- construction of concrete plugs in the bords of the old workings designed to withstand the maximum head of 30 metres under the RMF design flood of 2400 cumecs. This was the method accepted as the most suitable and reliable in the circumstances, and plugs of 10 metres length selected with a Factor of Safety exceeding 5.

REMEDIALS OF THE CANAL

The layout of the old underground workings was such that only five concrete plugs were required in the bords on either side of the river diversion (Figure 10). The plugs to the north of the diversion are uniform in size while those to the south are extremely variable owing to the configuration of the mined out ground. Access for construction of the north plugs was obtained through a ramp formed into the old workings from the invert of the canal through the fill material. The southern plugs were accessed from a similar ramp, an old (1904) vertical shaft (Figure 10) and an incline shaft adjacent to the area. Cleaning out of the plug sites was carried out with a TORO LHD on loan from another mine in the group.

The concrete was initially pumped into the southerly plugs S1 and S4 via the ramp in the canal after installing shuttering of chicken wire mesh, steel fabric and gumpoles with tube-a-manchette grout tubes to the roof and side walls of the bords. This was altered to concreting direct from surface down holes drilled into the workings above the plug site which proved more successful. The tube-a-manchettes were not successful in grouting the sides and roof concrete to coal interfaces and this operation was also subsequently changed to pumped grout from surface. Work on the plugs was completed on 11^{th} December and backfilling of the ramps commenced.

Difficulties were experienced in sealing the interface concrete/coal and during an inspection leakages were noted at the roof of plug S2 which had not been completely grouted and approximately 0,7 m^3/s was spilling over the plug while minor spillage was also occurring over plug No. S4. This was discovered after protracted daily rains forced the mine to redirect the flow down the canal earlier than planned (the flow had begun exceeding the 10 m^3/s the dam drain pipes could discharge) – rainfall recorded on mine during the previous week was as follows (Table 2).

7 December	8 December	9 December	10 December	11 December	12 December	13 December
21 mm	50 mm	26 mm	50 mm	19 mm	38 mm	22 mm

Table 2. Daily Rainfall – (Portion of) December 1992.
Note: the maximum rainfall recorded for a 24 hour period in December is 117mm; which is also the maximum for any day recorded. December is the third wettest month with a mean monthly rainfall of 120mm. Despite consistent rain, excessive floods did not occur in the period.

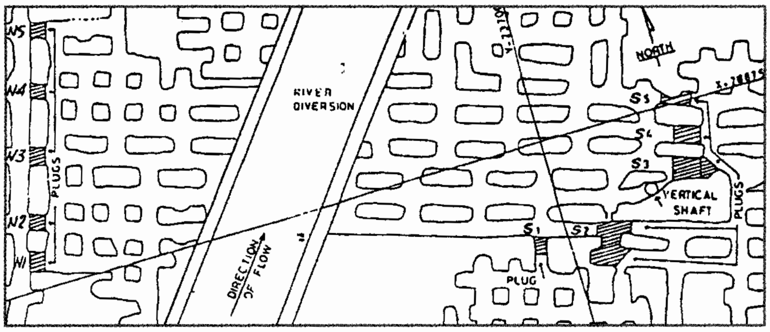

Figure 10. Showing the river diversion. Underground workings and plugs.

The inspection of the plugs was carried out on the 14^{th} December once the river flow had again subsided to 10 m^3/s to allow the flow to be diverted down the water course. Re-grouting of the roofs of plugs S2 and S4 was then to be carried out from surface (6 boreholes per plug). Repair work to these leaking plugs could not commence until the water level between the two sets of plugs dropped to the extent that the overspill abated. This necessitated the removal of some 90 000 m^3 of water through a drain pipe installed in one of the low lying plugs. The time taken for the water level to drop to effect the repairs extended through to the 21^{st} December, by which time the river was again flowing strongly. With this in mind and the fact that the grout needed 24 hours to cure it became necessary for some alternative plan to be used to repair the leak. Wooden wedges, chocks and gumpoles together with heavy duty blasting socks, normally employed in 250 mm diameter pre-split blast holes, were forced into the gap between the roof and the concrete and these were filled with grout pumped from surface. On 22^{nd} December inspection of the leak showed that it had reduced to 150 litres/sec. indicating that the emergency measures were partially successful; the mine could handle this quantity without loss in production. The inspection on the 24^{th} showed the same approximate leakage rate.

On Christmas day (25th December 1992) at 10h20 mine management received a report that the pit was flooding. After re-directing the flow down the river course the plugs could only be inspected on 29th December, when it was found that a large portion of the floor beneath plug No. S3 had washed out. It was subsequently found to be an excavation in the floor, backfilled by the "old timers" which had not been detected. Emergency remedial measures were carried out by installing large grout bags (three No. 5m x 4,5m utilising ventilation brattice cloth) into the void below the plug and grouting from surface. This was completed together with plug S2 remaining work on New Years Day (1st January 1993), substantially reducing the flow into the pit.

Further grouting of smaller voids was carried out and the mine recommenced coaling on 4/1/93. It was also decided to reinforce all the south plugs by rockbolting the floor on the downstream side of the plugs and casting an extension to plug No. S3 to form a buttress, achieving additional stabilisation of this plug. These remedials were monitored for several weeks and proved stable. Water pressure testing for leak detection (to a pressure of 4 bar) was then carried out in holes drilled into the strata round each plug to check the interface coal/concrete and grout and the immediate surrounding coal / rock for fissures, jointing and other potential flow paths. The test work was completed in early February 1993 and a monitoring system including regular plug inspection, leakage monitoring and water level dipping adjacent to the plugs instituted. To date there have been no further problems with these plugs; in fact the decision has now been made to leave the diversion in its current position permanently.

CONCLUSIONS

Notwithstanding the problems experienced on the project including the long hours worked under extremely onerous conditions, the constant threat from the elements, the additional costs incurred and the loss of production, the experience and knowledge gained has been invaluable and is currently being applied on a similar project on an adjacent mine. In particular the danger of old workings being incompletely surveyed, old mining operations not carried out as shown on plans, the potential for failure round the concrete plug and in the surrounding strata are emphasised for close attention in a situation such as this. This is particularly the case where access to inspect the workings prior to construction is not possible.

ACKNOWLEDGEMENTS

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