RB 74/222
DM 224/74

1/1/145

November 1974

SOUTH AUSTRALIAN DEPARTMENT OF MINES

OPEN FILE

Amdel Report R/B 74/222

NO.1015

REHABILITATION OF BRUKUNGA MINE: NAIRNE PYRITES LIMITED

by

N. Blesing, J. Lackey and A. Spry

Investigated by: Chemical Metallurgy Section

Officer in Charge: J.E.A. Gooden

F.R. Hartley. Director

THE AUSTRALIAN MINERAL DEVELOPMENT LABORATORIES
Adelaide South Australia

CONTENTS

CONTENTS

SUMMARY

Background

Officers of the South Australian Department of Mines have been concerned about the observed release of acid waters with high iron and other heavy metal concentrations from the Brukunga mine of the Nairne Pyrites Limited into Dawesley Creek. Complaints were received in 1972 of dead fish in Lake Alexandrina (65 km downstream) and there was fear of long-term damage to the limestone aquifer at Milang by dilute acid waters.

Amdel was requested on 7 March 1973 to investigate existing environmental conditions in and around the Brukunga mine, examine methods for treating the acid waters and recommended procedure by which Dawesley Creek could be kept free of polluted waters.

Summary of Work Done

Chemical analyses have been made over a twelve-month period (April 1973 to April 1974) of water from the Dawesley Creek above, within and below the Brukunga mine. The pollutants entering the creek have been identified by chemical analysis on the major acid-water flows. Criteria of acceptable water quality for irrigation and livestock drinking water have been stated and used to determine the acceptability of waters issuing from the mine.

A study has been made of the chemical composition of precipitates and evaporites in the Dawesley Creek, the significance of dyes and alkalinity in the Dawesley Creek water upstream from the mine, the origin of the acid waters, and the total disturbance of the environment has been summarised in the form of impact and cause-effect-correction type matrices.

Treatment method for preventing pollution of Dawesley Creek have been examined, including dilution, diversion, prevention, evaporation, neutralisation and revegetation. The neutralisation method has been studied in the laboratory using waste 'grit' and 'caustic mud' alkali available from the manufacture of hydrated lime and sodium hydroxide.

Estimated costs of various rehabilitation procedures have been determined and recommendations made as to the work required to solve the acid mine drainage problem.

Details of the study have been reported previously in Progress Reports No. 1 to 4 inclusive.

Conclusions and Recommendations

The survey of the mine area and downstream Dawesley Creek over 12 months has shown that significant amounts of acid, sulphate and heavy metals (in particular iron, aluminium, manganese and zinc and trace elements nickel, cobalt, copper and cadmium) are entering the Bremer River system even when using the procedure of collecting northern quarry seepages and the tailings dam seepage and stopping them from entering Dawesley Creek. Under the best control conditions the Dawesley Creek has contained dissolved toxic amounts of one or more of the above heavy metals for at least 16 km downstream. Under adverse conditions when uncontrolled or 'out-of-phase' releases of acid waters have occurred the river system has been polluted with dissolved heavy metals as far downstream as Langhornes Creek (65 km downstream).

An extensive examination of the situation has shown that there is no quick, cheap method of solving the acid drainage problem. The production of acid waters high in dissolved heavy metals has reached an advanced stage in the abandoned mine. From experience elsewhere, the problem can be expected to become worse if it is left unresolved.

Because of the lack of sufficient hydrological data, the exact scheme required to solve the acid mine drainage problem cannot be specified. However, one of three basic schemes, each with a capital expenditure greater than $600 000, must be adopted.

Prevention Scheme:
Diversion of fresh waters including Dawesley Creek, sealing the quarry benches and recycling acid waters from the tailing dam and quarry.

Evaporation Scheme:
Diversion of fresh waters including Dawesley Creek and evaporating acid waters in ponds.

Neutralisation Scheme:
Diversion of fresh waters including Dawesley Creek and neutralisation of acid waters.

The neutralisation scheme is the most satisfying and safest, but also the most expensive scheme, the one most studied for Brukunga and most commonly used in similar situations elsewhere.

The prevention scheme is the preferred scheme because:

(1) The total costs should be equal to or lower than the alternative methods.

(2) The scheme contains considerable preventive measures to keep fresh water away from sulphide-containing materials.

(3) The costs for operation and maintenance are the lowest.

(4) The ponding area required is the smallest of all the schemes. It is however, the least studied of the three schemes and needs further investigation.

The evaporation scheme, based on currently available hydrological data, is economically competitive, is low in operating costs, but is visually unattractive because of the large ponding area required.

Regardless of which scheme is finally adopted it is important to implement, as soon as possible, a revegetation scheme to promote a vegetation cover on the waste dumps, and tailings dam surfaces. This will increase evaporation, reduce entry of surface water, reduce surges in acid mine drainage and reduce the visual impact of the area.

The waste dumps will probably become the major source of polluted water after the tailings dam has been drained but the material is so coarse (rock lumps tens of cms across) that there does not appear to be any simple method of treatment. If soil can be placed and held on the steep sides until vegetation became established, the drainage through the dump may be reduced.

Before the final choice of a scheme is made, it is recommended that further investigations be carried out to verify certain assumptions which have been made. It is recommended that the following investigations be made:

(1) A hydrological and chemical study of the movement of water in the tailings dam to determine the liklihood of the dam remaining stable until it dries out after approximately 20 to 30 years.

(2) An investigation to measure the evaporation rate of water from the upper tailings dam storage to determine the annual amount of acid water which can be 'treated' by this method.

(3) A pilot trial of sealing a portion of the quarry bench at one seepage locality (e.g. QSB) to determine the feasibility of substantially reducing the flow of quarry seepages by this method.

(4) The measurement of levels on the main quarry benches and design of bench drainage.

(5) Surveys during September 1974 to April 1975 to confirm the significant contribution and exact source of heavy metals from the waste dumps into Dawesley Creek and the design of the bypass pipeline.

(6) The requirements for revegetating the tailings dam and the waste dumps be examined by carrying out the appropriate field trials.

To substantially reduce the amount of acid mine drainage currently entering the Dawesley Creek, it is recommended that the drains and channels for Days Creek, QSA and QSBs be cleaned up as soon as possible. Also, a holding pond should be built below QSC and a pump and pipeline installed to transfer seepage to a storage locality either in the QSB area or on the tailings dam (see Fig. 24). The diversion of these fresh-water streams and collection of QSC (and QSD) seepage must be carried out regardless of whichever of the three alternative schemes is finally adopted.

The short-term problem of the annual build-up of acid waters on the tailings dam resulting in illegal discharges of acid drainage during the summer and winter months must be solved as soon as possible. Although the annual discharge of the stored seepage into the Dawesley Creek during the month of the year in which water is not required (e.g., May or June) is the cheapest solution, the preferred method is to use a portable lime neutralisation plant to neutralise the acid overflows on demand.

1. INTRODUCTION

1.1 Initiation of the Project

Following discussions between representatives of the Department of Mines and of Amdel on 7 March 1973, a preliminary proposal (5/0/2145) was submitted and accepted verbally on 30 March 1973. It was requested that the investigation begin immediately and visit was made to the area on 3 April by Messrs K. Harris and J. Selby of the Department of Mines together with Dr A. Spry and Mr N. Blesing of Amdel. The information available was placed at Amdel's disposal and a general outline of the procedure to be followed was agreed upon.

The objectives of the investigation were:

1. To investigate exising environmental conditions in and around the quarry with particular emphasis on the composition of ground water issuing from the quarry and of effluent from the tailings dam.
2. To investigate processes by which the composition of the liquid effluent from the quarry might be improved.
3. To recommend procedures by which the tailings dam contents and effluent could be converted to a liquid of acceptable composition, bearing in mind the importance of economics, an early initiation of the process and the need for avoiding extended operations.

The Department of Mines and Nairne Pyrites Limited have both been involved in the investigation of the problem. It was envisaged that a co-ordinated program of investigation would be established with additional assistance being provided by other State Government Departments such as the Engineering and Water Supply (E & WS) Department and the Department of Agriculature.

1.2 Nature of the Problem

Amdel was advised that a problem existed along the Dawesley Creek (and the Bremer River System) (see Fig.1) due to the release of acid waters with high iron concentration from the mine area of Nairne Pyrites Limited. Heavy metal concentrations were also suspected to be high. Mining ceased in 1972 after approximately 17 years' operations and the lease was due to expire on 31 May 1974.

In July 1972 officers of the Department of Mines surveyed the downstream area of the Dawesley Creek and determined that the pH of the waters was below 3.7

for at least 20 km downstream from the Brukunga mine. Consequently the Company, Nairne Pyrites Limited, was requested by the Department of Mines to stop discharging tailing dam effluent into Dawesley Creek.

A more comprehensive downstream survey was made by Amdel in August 1973 to determine the extent of acid drainage from the quarry area. The results of nine standard inorganic water analyses (Amdel Report AN736/72) showed that:

1. pH values were below an 'acceptable' minimum of 4.8 for at least 13 km downstream from the mine.
2. Iron concentrations were unsatisfactorily high for approximately the same distance; i.e., >10 mg/*l* Fe for 13 km.
3. Aluminium concentrations were above tolerated levels of 20 mg/*l* Al for irrigation waters for the same distance.

Following a complaint of a fish-kill in Lake Alexandrina (65 km downstream) a survey of the waters on September 1972 for copper, zinc and cadmium showed that significant amounts of these metals appeared in Dawesley Creek after passing through the quarry, but by the process of dilution and precipitation the concentration levels became acceptable for fish (i.e., Cu <0.03 mg/*l*, Zn <0.05 mg/*l*, and Cd <1.0 mg/*l*) after flowing 24 km downstream.

Another survey of water compositions was conducted in January 1973 (Amdel Report AN3007/73). Water flowing in the Dawesley Creek 1.6 km downstream from the mine was essentially undiluted acid mine drainage containing a high concentration of iron (845 mg/*l* Fe) and high acidity (pH 2.5). The acid water was accumulating in the upper part of the water system, as shown by the occurrence of stagnant pools 11 km downstream which were slightly alkaline. The Company was requested to stop all major drainage from the northern end of the quarry entering the Dawesley Creek.

Acid water was stated by the Department of Mines to enter the Dawesley Creek from two main sources, namely the tailings dam and seepages in the quarry face (Fig. 2 and Plates 1 and 2). It was also believed that creek water was being similarly polluted by seepage of rain-water through exposed ore and waste dumps of rock rich in pyrite and pyrrhotite.

During the operating period, the Company had temporarily overcome the water pollution problem lower in the river system by providing alternate water supplies for farmers as far downstream as the Nairne Creek-Dawesley Creek junction but will presumably take no such responsibility in the future.

Complaints are believed to have been received from Potts Bleasdale Winery at Langhornes Creek and there was a fear by Department of Mines officers of long-term damage to the limestone aquifer in the underground basin at Milang due to a slow attack by dilute acid water (Bowden and Bleys, 1971; O'Driscoll, 1960; Roberts, 1972).

The water in Dawesley Creek is slightly polluted with organic material and is weakly alkaline before it reaches the mine area as it comes from oxidation ponds at the woollen mills at Lobethal. A small oxidation pond handling sewage from Brukunga contributes only slightly to the general problem.

1.3 Existing Procedure

The procedure at present is to control the seepage from the holding pond at the base of the tailings dam, together with the seepages from the northern part of the quarry, and to pump them to the top of the tailings dam. The tailings dam effluent of approximately 11 m^3/h is thus recirculated and the pH has been steadily decreasing with time.

This water was originally used in the mill circuit in the acid flotation process. Recent procedure is stated to have been to release the highly concentrated water in the pond into the Dawesley Creek in winter when it was running strongly to allow dilution. It is suspected that some acid water from the holding ponds has been allowed into Dawesley Creek during low summer flow. Taken together with the uncontrolled seepage from the quarry, this causes a very low pH in the creek water.

2. THE NAIRNE PYRITES DEPOSIT

2.1 Geology

The geology and mineralogy of the Nairne pyrite deposit has been described by Ridgeway (1949), Edwards (1949), Edwards and Carlos (1949), Skinner (1958) and La Ganza (1959). The relevant facts are that the orebody is a comformable pyritic meta-sedimentary layer enclosed within regionally metamorphised Kanmantoo Group rocks.

This pyritic horizon outcrops widely throughout the region.

The minerals present in the orebody consist mainly of approximately equal proportions of pyrite and pyrrhotite (averaging 10% total) with minor marcasite, galena, sphalerite, chalcopyrite, rutile, arsenopyrite, a group of antimony-silver minerals and nickeliferous and cobaltiferous pyrite with gangue minerals including quartz, muscovite, albite, kaolin, feldspar, biotite, andalusite, kyanite, garnet, carbonates (e.g., siderite) and zeolite.

Weathering produced a resistant limonite-quartz-kaolin gossan containing sporadic pyrite. The more-easily weathered pyrrhotite has not been observed in outcrop (Skinner, 1958, p 549) and is partially altered to marcasite and pyrite below the level of oxidation.

2.2 Physiography

The large quarry has been excavated on the western side of the mature valley of the Dawesley Creek (Fig. 2, Plates 3 and 4). The Dawesley is joined by the Taylors Creek immediately south of the mine area, then by the Nairne Creek (8 km below Brukunga) and it become Mt Barker Creek after a further 5 km (Fig. 1). Mt Barker Creek joins the Bremer River 3 km further downstream and finally reaches Lake Alexandrina after a further 30 km.

The waste dumps extend right up to the Dawesley in some places and in others are only 10 m or so away (Plate 5). The dumps have actually encroached on the creek in the southern end of the workings where a new channel has been cut to contain the water (Plate 6). Seepages and encrustations are present in a number of places along the western bank of the creek where ground water emerges through sandy banks and the old creek bed underlying the waste dumps (Plate 7).

2.3 Mining and Mineral Beneficiation

The open cut is almost 2000 m in length and occupies the western side of the Dawesley Creek valley. The benches reach almost the summit of the hills and very large mullock heaps occur between the quarry and the creek in two main areas near Brukunga village and just south of the disused Nairne Pyrites Limited plant (Fig. 2, Plate 8).

Operational procedures for crushing and mineral beneficiation are described in Appendix A.

When the mine was in operation, ore from the quarry was crushed in jaw and cone crushers, followed by fine grinding in rod and ball mills. The pyrite was separated from waste rock by gravity concentration using jigs and by flotation. The two concentrate containing 40% and 47% sulphur were dewatered and forwarded by road and rail to the works of Sulphuric Acid Pty Ltd. The scale of production was about 300 000 tonne of ore per annum, excluding the removal of overburden. Grinding was such that 50 to 60% of the flotation concentrate and tailings was less than 75 μ (Table A-1). The total concentrate was approximately 25% by weight of crude ore treated.

The tailings dam occupies an area of approximately 27 hectares in a tributory of the Dawesley Creek on the eastern side (Plate 8). It appears to have been built by the upstream method with an initial rock wall and impermeable core; about six steel pipes, 150 mm in diameter, pass through the base of the dam for drainage purposes.

The dam is largely dry with water restricted to a few (approx. 7) hectares in the north-eastern corner where it is stated to be about 2 m in depth (Plate 9).

The dam has extended up the valley almost to the top so that there is a very restricted area of natural drainage into it.

The upper surface of most of the dam is dry and has been covered with rock plus clay. Piezometers through the upper surface of the dam near to the retaining wall in August 1972 showed the water-level to be at about 12 m down. Seepages appear on both abutments of the dam (at the contact between dam-material and the underlying rocks) at this same level. There is apparently drainage through the tailings in the body of the dam, but the flow is concentrated at the contact positions. The seepages are collected in ponds at the base of the tailings dam (Plate 10) and pumped to the upper holding pond along with other seepages from the northern part of the quarry (Plate 11).

3. COMPOSITION OF DAWESLEY CREEK - MT BARKER CREEK - BREMER RIVER

Analyses have been made over a 12-month period (April 1973 to April 1974) of waters from Dawesley Creek at sample sites located above, within and below the mining property, from the quarry and from the tailings dam (results for 14 sampling points are compiled in Appendix B1).

With the recent development of the city of Monarto, the E & WS Department have commenced a water quality survey of waters in the Bremer River catchment system. A progress report summarising the results obtained from August 1973 to December 1973 has been issued and made available to the authors of the Report.

3.1 Water Quality Criteria

Since water in the Bremer catchment is not considered to be suitable as a freshwater resource because of its high salinity (1200 to 5000 mg/*l* Total Dissolved Solids, TDS, in the Bremer River) water quality criteria for agricultural use are the main concern here. Criteria for acceptable quality are currently being compiled for irrigation and livestock watering by the Australian Water Resources Council (Tables 1 and 2). Critical maximum values have been set for most of the heavy metals with especially low concentrations being required for selenium (0.05 mg/*l* Se), molybdenum (0.005 mg/*l* Mo) and cadmium (0.005 mg/*l* Cd). The values should be regarded with some caution because environmental tolerances to the metals depend upon many factors including soil type and climate, plant type for irrigation and animal type, age and conditions for livestock watering.

Acidity and alkalinity as measured by pH have little direct significance for irrigation waters but heavy metals or other elements tend to become soluble and hence potentially toxic at low pH. Based on the fact that acid soils over a period of time yield soluble iron, aluminium and manganese in concentrations large enough to be toxic to plant growth, a lower pH limit has been set at pH 4.8.

Limits on water quality for mining effluents discharged into streams have not yet been specified in Australia but the limits set for mine waters in Pennsylvania, USA, are that the discharging stream be near neutrality and contain not more than 7 mg/*l* Fe (Commonwealth of Pennsylvania Sanitary Water Board 1964).

The Ontaria Ministry of Environment in Canada has established water quality objectives of pH 5.5 to 9.5 suspended solids 15 mg/*l*, nickel 1 mg/*l* Ni, copper 1 mg/*l* Cu and iron 17 mg/*l* Fe for treated acid mine waste waters (Oho and Taylor, 1974).

3.2 Scan Analyses

A scan analysis was run on the major flows and holding ponds in the mine and in Dawesley Creek to identify the main pollutants which do not comply with irrigation and livestock water standards and therefore are potentially likely to cause downstream environmental problems. [A scan analysis is an analysis in which the chemist has attempted to identify and determine the amount of every contaminant in the flows (Hawley and Shikaze 1971)]. Analyses were made on tailing dam storage water, tailing dam seepage, northern quarry seepage, waste dump seepage and Dawesley Creek water as it left the mine for sulphates (SO_4) acidity, (pH and acid values)*; iron (Fe), copper (Cu), zinc (Zn), cobalt (Co), nickel (Ni), manganese (Mn), lead (Pb), vanadium (V), cadmium (Cd), titanium (Ti), lithium (Li), sodium (Na), potassium (K), calcium (Ca), aluminium (Al), magnesium (Mg), arsenic (As), phosphorus (P), antimony (Sb), silver (Ag), uranium (U), chemical oxygen demand (COD), mercury (Hg), tin (Sn), chromium (Cr), molybdenum (Mo) and barium (Ba) (Table 3).

The constituents identified as either existing or potential pollutants have been:

acidity	(pH and acid value)
iron	(Fe)
aluminium	(Al)
manganese	(Mn)
zinc	(Zn)
nickel	(Ni)
copper	(Cu)
cobalt	(Co)
cadmium	(Cd)
sulphate	(SO_4)

Data determined by the E & WS have indicated that the Dawesley Creek is better bacteriologically and lower in nitrogen and phosphorus nutrients than the Mt Barker Creek and Brewer River. This is to be expected since one of the normal methods for the removal of pathogens and phosphorus is by chemical

---

* Acid value is defined in Table 3.

treatment at neutral pH using iron or aluminium salts.

3.3 Effluent from the Lobethal Oxidation Lagoons

Town sewage and wastes from wool processing and the fruit industry at Lobethal are removed from the catchment of the Onkaparinga River by pumping to oxidation lagoons at the Bird-in-Hand Mine near Reefton for biochemical treatment. The treated effluent flows into a tributary of the Dawesley Creek approximately 6 km upstream from the Brukunga mine. Plans have been made by the E & WS to connect treated domestic sewage and industrial wastes from Woodside to this system and to upgrade the operation of the oxidation lagoons by injecting air. The estimated flow into the lagoons is of the order of 500 m^3/day (100 000 gal/d). Flow estimates of treated effluent leaving the lagoons vary according to the time of the year. In the summer months the large surface area of the ponds allows most of the water to evaporate. Estimates show that the treated effluent contributes significantly to the mean flow of the Dawesley Creek for approximately 6 months of the year (Appendix C, Table C1).

As a result of observing pink gelatinous precipitates on rocks where the Dawesley Creek water entered the mine property, a study was made of the upstream treatment lagoons system. Analysis of the treated effluent collected on 21 June 1973 revealed a water high in sodium alkalinity (Appendix B1). A high pH (and alkalinity) would be expected since the biochemical oxidation process removes carbon dioxide from solution, thus raising the pH. E & WS data indicate a variation between 7.8 and 9.0 depending upon the time of year. Lime is added during the treatment process to neutralise acidic fruit wastes and thus maintain conditions for effective biochemical treatment.

The main wastes entering the lagoons from the woollen mills are non-ionic detergents and dyes. The dyes are neutral or slightly acidic. The neutral dyes are of azo-chrome-mordant type containing covalently-bonded chromium. The acid dyes are azo-types which are added as the sodium sulphonate; they change colour with a change in pH, being similar to the normal acid-base indicators (e.g., methyl red).

Examination of the treated effluent and the pink precipitate on the rocks in the creek bed both showed a colour dependence on pH. When alkaline the colour was buff. Upon adding acid the effluent and water extracts of the precipitate turned pink. It was concluded that acid dyes in the treated effluent were not evident until the water reached the acid environment of the Brukunga mine. Assisted by the high concentration of heavy metals mainly iron, the dye would be expected to turn pink and precipitate on the rocks in the creek bed. Analysis of the red precipitate showed a high concentration of iron strongly bound to the dye.

Although there are no known data on the acceptable limits for acid dyes in water for agricultural and pastoral use it is unlikely that the dye in the Dawesley Creek under alkaline conditions could be considered a pollutant. The objection to the acid dye occurs primarily when the water becomes acid, contacts heavy metals and the dye becomes evident as a visual pollutant.

3.4 Water Analysis during Winter Flows

Analyses made during winter flow conditions on 20 September 1973 showed that toxic levels of iron (Fe), aluminium (Al), cadmium (Cd), manganese (Mn), zinc (Zn), nickel (Ni), cobalt (Co) and copper (Cu) were evident in Dawesley Creek water 3 km downstream from the mine (Fig.1; Table 4). The acidity (pH and acid value) and sulphate concentration were outside acceptable limits for agricultural use. The soluble concentration levels of these constituents remained unacceptable until Dawesley Creek water became diluted with Nairne Creek and Mt Barker waters resulting in neutralisation of the acidity and lowering of the sulphate concentration and precipitation of the heavy metals.

Beyond 16 km downstream after the Mt Barker-Dawesley Creek junction the water became suitable for livestock watering and for most irrigation purposes. The salinity of the water remained relatively high (1000 mg/*l* TDS) making it unsuitable for the irrigation of salt- sensitive plants.

3.5 Water Analysis during Summer

Early in summer the water quality of Dawesley Creek 3 km from the mine remained polluted with heavy metals, acidity and sulphates (26 November 1973 analysis shown in Table 5). The concentration of the conventional cations and anions (Ca, Mg, Na, SO_4 and Cl) had increased by 50% at this location relative to winter concentrations whereas most of the heavy metal concentrations had increased by 100 to 200%. Analyses further downstream showed that the water quality had recovered and was free of dissolved pollutants after Dawesley Creek had joined Mt Barker Creek. The salinity of the water at this location had, however, risen to a level of 2000 mg/*l* TDS making it undesirable for irrigation use due to the toxicity of its high chloride content.

Late in the recent 1973/74 summer, the concentration of heavy metals at the location 3 km downstream from the mine had increased substantially to values 400 to 900% higher than those found during winter flows (4 April 1974 analysis shown in Table 6). Chloride levels remained approximately the same as in early summer indicating that the higher heavy metal concentrations were due to additional acid mine drainage from the mine and not to evaporation. The heavy metal concentrations were even higher during the late summer of 1972/73 (Appendix B1, 10 April 73 analysis).

Analyses further downstream showed that the pollutants were not sufficiently diluted or removed from solution as far downstream (20 km) at the junction of Bremer River and Mt Barker Creek and possibly not until the Bremer River entered Lake Alexandrina (65 km downstream).

The toxic heavy metal elements which remained in solution the longest and were the last to be removed by dilution and precipitation were aluminium and cadmium. The concentration values of these pollutants at Langhorne Creek on 6 February 1973 (33 mg/*l* Al, 0.055 mg/*l* Cd) were outside the acceptable levels for irrigation water even for short-term use on fine textured soils.

High concentrations of heavy metals and acidity would be expected in the upper part of Dawesley Creek below the mine during late summer because of the low river flow (Appendix B1: 3 Jan. 73 analyses). However, the extent to which the downstream system has been polluted by heavy metals and acidity during 1973/74 is unusual and is interpreted as being due to the deliberate release of significant quantities of tailings dam and quarry seepage as a result of having insufficient storage capacity on the upper holding pond.

3.6 Precipitates and Evaporites

Precipitates have been evident in Dawesley Creek (Plate 5) as far downstream from the mine as Dawesley village. They have been observed down to the junction of Dawesley Creek and Mt Barker Creek (16 km downstream, Table 7). Reeds have been covered with a white aluminous deposit at the same locality and small amounts of precipitate have been evident at Langhornes Creek (50 km downstream).

Visual pollution is caused by brownish sludge in the water and on rocks; white, cream, yellowish and brownish encrustations on rocks, banks and vegetation; and yellow-brown to black sludge and crusts on the face of the tailings dams (Plate 2). These features are prominent and attract attention to the more-important chemical pollution.

Some of these solids are water-soluble; they are produced by evaporation and disappear by solution. Others are insoluble and persist or even build up with time. The sludge in the river collects in pools at times of low flow but tends to be flushed downstream during flood periods. Ferruginous precipitates are clearly visible at Dawesley village and may be present further downstream.

The main species identified by X-ray diffraction (Table 8) are as follows:

(a) $(Fe, Mg)\ Al_2(SO_4)_4,\ 22H_2O$. Ferropickeringite - a member of the pickeringite (Mg)-halotrichite (Fe) series.

(b) $Na\ Fe_3\ (SO_4)_2\ (OH)_6$ natrojarosite.

(c) $K\ Fe_3\ (SO_4)_2\ (OH)_6$ jarosite.

(d) $Ca\ SO_4.\ 2H_2O$ gypsum

The following have been deduced to be present from chemical evidence (Table 9):

(e) $2\ Mg\ SO_4.\ M\ SO_4.\ 2Al_2(SO_4)\ (OH)_4\ 7H_2O$ where M = Fe, Mn and Zn

(f) $Fe\ (OH)_3$.

(g) Ferric sulphate.

The complex magnesium aluminium sulphate (e) occurs as a soluble white encrustation formed by evaporation during the summer.

Off-white crystalline material rich in ferropickeringite occurs as an encrustation on boulders in the Dawesley Creek just above water-level in the mine area.

The common orange to reddish brown sludge over rocks and the bed of the Dawesley Creek consists mainly of amorphous material (probably ferric hydroxide) with natrojarosite, ferropickeringite (near halotrichite), gypsum, possibly minor colloidal silica and ferric sulphate.

The analyses in Table 8 and 9 are difficult in interpret quantitatively but chemical calculations confirm that the precipitates are mainly hydrated sulphates of iron with aluminium and magnesium. Heavy metals are present in considerable amounts, particularly manganese and zinc.

The precipitates occuring at the junction of Dawesley Creek and Nairne Creek are highly ferruginous (14) and coloured red, whereas precipitates at the junction of Dawesley Creek and Mt Barker Creek are high in aluminium (15) and coloured grey. As the water of the Dawesley Creek moves downstream from the mine the sediments become progressively higher in aluminium but lower in iron (Tables 7 and 9). The concentrations of trace metals (e.g., Mn, Ni, Cu, Zn and Cd) in the sediment are variable and do not appear to be related to a major metal, either Fe or Al. Collins (1973) has shown that trace metals such as Cu are co-precipitated and are actually located within the crystal-structure of the solid hydroxide or basic sulphate.

3.6.1 Origin of Precipitates

The enclosing metasedimentary rocks and the gangue minerals in the sulphide ore consist mainly of the silicate mineral quartz, muscovite, albite, microcline, andalusite, kyanite, spessartite and scapolite. The weathering of these silicates is greatly accelerated by the presence of sulphuric acid derived from the weathering of sulphides; breakdown results in acid soluble and insoluble sulphates, various mobile cations and insoluble hydroxides.

Under normal weathering conditions with moderate pH, the solid residue tends to be a clay such as kaolin, but under stringent conditions, with lower pH, assemblages containing alunite, natroalunite, and jarosite are formed.

These reactions are indicated by the equations listed below (after Garrels, 1967 and Sato, 1964).

Moderate pH

$$\overset{\text{microcline}}{2KAlSi_3O_8} + H_2SO_4 + H_2O \rightarrow \overset{\text{kaolin}}{Al_2\,Si_2O_5\,(OH)_4} + K_2SO_4 + \overset{\text{colloidal silica}}{4SiO_2}$$

$$\overset{\text{albite}}{2NaAlSi_3O_8} + H_2SO_4 + H_2O \rightarrow \overset{\text{kaolin}}{Al_2Si_2O_5(OH)_4} + Na_2SO_4 + 4SiO_2$$

Low

$$\overset{\text{microcline}}{3KAlSi_3O_8} + 6H^+ + 2SO_4'' \rightarrow \overset{\text{alunite}}{KAl_3(SO_4)_2\,(OH)_6} + 2K^+ + 9SiO_2$$

$$\overset{\text{albite}}{3NaAlSi_3O_8} + 6H^+ + 2SO_4'' \rightarrow \overset{\text{natroalunite}}{NaAl_3(SO_4)_2\,(OH)_6} + 2Na^+ + 9SiO_2$$

$$\overset{\text{muscovite}}{2KAl_2\,(Si_3Al)\,O_{10}\,(OH)_2} + 8H^+ + 2SO_4''+4e' \rightarrow \overset{\text{alunite}}{2KAl_3(SO_4)_2\,(OH)_6} + 6SiO_2$$

$$\overset{\text{biotite}}{K_2(Mg,\,Fe)_4Al_2(Si_6Al_2O_{20})\,OH_4} + 2H^+ + 4SO_4'' \rightarrow \overset{\text{alunite}}{2KAl_3(SO_4)_2\,(OH)_6} + 2Fe^{++} + 2Mg^{++} + Al^{+++} + 6SiO_2$$

$$\overset{\text{microcline}}{3KAlSi_3O_8} + 6H_2SO_4 + 6Fe(OH)_3 \rightarrow KAl_3(SO_4)_2\,(OH)_6 + 2KFe_3(SO_4)_2OH_6 + SiO_2 + 6H_2O$$

Both kaolin and alunite have been found in weathered ore.

The composition of the quarry seepages (see Section 5) is compatible with the reactions listed above. The high Al is due to the abundance of alumino-silicates and aluminium silicates but the abundant Na and small K suggests that the attack of albite is much greater than that of muscovite, microcline and biotite. The low pH results in mobility of Al, Fe and Na which are later partly precipitated in the Dawesley Creek as sludges containing natrojarasite, pickeringite and the colloidal forms of silica and ferric hydroxide.

3.7 Lake Alexandrina

The annual flow of the Dawesley Creek has been calculated at 8.2 × 10^5 m^3. Using the analyses of 7 June 1973 as average values for the metal content of Dawesley Creek water leaving the mine, it is calculated that 150 tonnes of Fe, 200 tonnes of Al, 30 tonnes of Zn, 0.7 tonne of Cu and 0.15 tonnes of Cd enter the river each year. These metals are precipitated within the first 20 km, but the resulting precipitates appear to be flushed out of the river during the winter. As the water has been neutralised before the aquifer of the Milang Basin is reached, acid attack of the limestone is unlikely. The metal-bearing precipitate and mud would not enter the aquifer but would probably be deposited in Lake Alexandrina.

If the pollution is assumed to have been taking place over the past 10 years, then about 300 tonnes of Zn, 7 tonnes of Cu and 1½ tonnes of Cd have entered the lake. The metals are probably held in an inert condition on the bottom mud but it is possible that they could be rendered soluble by organic processes and lead to poisoning of fish in the lake.

4. ENVIRONMENTAL DISTURBANCE IN AREA

The total disturbance of the environment may be summarised as the sum of the impacts of a series of actions on a group of environmental characteristics or qualities. An environmental impact matrix (after Leopold *et al.*, 1971) in condensed form is shown in Fig. 3 and a matrix of the cause-effect-correction type is given in Fig. 4.

4.1 Main Relevant Actions

The condensed matrix (Fig. 3) selects the most important activities which have arisen in the area and evaluates their impact on a selection of environmental characteristics following the principles of Leopold *et al.*, (1971) and the NSW Minister for Environmental Control (Anon. 1973). Each impact is represented by a box with a diagonal slash in which are placed figures representing the Magnitude (top left) and Importance (bottom right). The number system of Leopold (1 to 10) has been found too cumbersome in practice and a scale of from 1 to 3 only is used here; 3 represents the greatest, and 1 the least, magnitude or importance, but a maximum value of only 2 is estimated in this instance.

Considerable difficulties are involved in both defining and placing a quantitative value of magnitude and importance, but these problems will not be considered further here. All that need be said is that the number of impacts is small and none are of very great importance.

The main problem here, as elsewhere, is to estimate the total degree of environmental disturbance, compare this with the cost of rehabilitation, and to decide whether the expenditure is warranted.

4.1.1 Modification of Regime

Alteration of ground water hydrology.
Alteration of drainage.
River modification.

4.1.2 Land Transformation and Construction

Excavation of the open cut.
Construction of the industrial site and buildings.
Construction of dams, ponds and pipelines.
Construction of domestic dwellings.

4.1.3 Waste Emplacement and Treatment

Disturbances of overburden and emplacement of coarse waste rock.

Emplacement of tailings in a dam.

Release of mining and milling liquid effluents.

Release of minor domestic waste from Brukunga and upstream from Lobethal.

4.2 Environmental Factors

Existing and previous characteristics and conditions which are being possibly affected by the actions listed above are:

4.2.1 Physical and Chemical Characteristics

River water quality.

Quality of underground waters.

Physical properties of limestone aquifers.

Dump and dam stability.

4.2.2 Biological Conditions

Trees, shrubs, reeds etc., along rivers.

Aquatic fauna.

Animals - domestic and wild.

4.2.3 Cultural and Economic Factors

Grazing, agriculture.

Open space qualities.

Health and safety.

4.2.4 Geological Factors

Surface and underground materials.

Composition, surface and underground waters.

4.3 Environmental Impacts

Environmental disturbance has extended from the mine area, along the Dawesley-Nairne-Mt Barker-Bremer river system possibly to the Milang underground water basin. However, it appears that the area downstream from the mining area would rehabilitate itself by natural processes (chiefly dilution) in a short period as acid loss has only been small and over a comparatively short period (17 years).

Corrective measures should therefore be concentrated on the mine area itself.

Significant environmental disturbances and sources of pollution in the mine area may be listed in order of decreasing importance as follows:

4.3.1 Water Pollution

Effluent from tailings dam area.

Seepages from open cut.

Seepages from coarse waste dumps.

Precipitates in, and adjacent to, watercourses.

Pollution due to domestic waste.

4.3.2 Visual Pollution

Slime, encrustations and other stream precipitates.

Large waste dumps.

Large visible quarry faces.

Tailings dam and ponds.

Attention is drawn to the various forms of environmental disturbance but, following instructions from the Department of Mines, emphasis in this project is phased on water pollution with the exception of the industrial and domestic sewerage effluent sources.

The problem is basically that of stabilisation of an abandoned tailings dam and mine area.

The most significant water pollution effect which has arisen as a result of the mining activity is the low pH coupled with very high iron, aluminium, manganese, zinc, nickel, cobalt, copper, cadmium and sulphate values.

The heavy metal pollution problem may not necessarily be solved when the metals pass out of solution as a result of dilution in the river system. The water downstream may be acceptable in its filtered state but contains sulphate sludge and mud onto which the metals have been adsorbed. These solids will pass downstream and it is possible that they could be concentrated at certain sites where they would remain indefinitely as potential sources of metallic ions if the environmental conditions were to change. Irrigation water polluted with suspended solids and cadmium from a nearby mining complex has been implicated in the contamination of rice paddy soils (and subsequently the rice itself) along the Jintru River Japan, the area where Itai-Itai disease has occurred (Yagamata and Shigematsu, 1970).

The heavy metal sludges if spread over the irrigation area lower in Bremer River system during flood time could have a serious effect on plant life in the locality including the vineyards at Langhorne Creek.

5. SOURCES OF WATER POLLUTION IN MINE

The map of the abandoned operation at Brukunga shows the major sources of seepage in the mine. In decreasing order of magnitude these are:

(a) Tailings dam.

(b) Quarry.

(c) Waste dumps.

5.1 Tailings Dam (Plates 8 and 9)

Abundant seepages which occur across the lower face of the dam collect in two ponds at the base of the dam and are pumped back to the pond at the top (Plate 9 and 10).

It has been estimated that there are approximately 230 000 m^3 of water on top of the dam and 420 000 m^3 contained within the tailings mass and that these waters emerge at the rate of 100 000 m^3 per year. The water volume in the tailing material has been estimated from data provided by K. Pile, Senior Lecturer in Civil Engineering, South Australian Institute of Technology.
Volume of dam in December 1970 was 0.94×10^6 m^3 (33.3×10^6 ft^3) Adding 10% for production to 1972, say 1.0×10^6 m^3.
The phreatic surface within the tailing dam shows that 16% of the volume has been drained (Fig. 5).

$$\therefore \text{ Water saturated volume } = 0.84 \times 10^6 \text{ m}^3$$

If there is 50% V/V water in the solids

$$\text{Volume of water } = 0.42 \times 10^6$$
$$\text{or } 420\ 000 \text{ m}^3$$

Because the flow through the tailings mass is laminar, Darcy's law (Cedergran, 1967) is followed:

$$Q = kiAt$$

where Q is the quantity of seepage having a cross-section area, A, normal to the direction of flow under a hydraulic gradient, i, during a length of time, t. The coefficient of permeability, k, is equal to the discharge velocity under a hydraulic gradient of 100%.
The coefficient of permeability for the dam is as follows:

$$k = \frac{Q}{iAt} = \frac{\text{Seepage rate (11.4 kl/h)}}{\text{Gradient of height/length} \left(\frac{6 \text{ m}}{850 \text{ m}}\right) \times \frac{\text{Volume}}{\text{length}} \left(\frac{0.94 \times 10^3 \text{ m}^3}{950 \text{ m}}\right)}$$

$$= \frac{11.4 \text{ kl}}{0.007 \ \ 1000 \text{ m}^2 \ \ 3600 \text{ sec}}$$

$$= 0.003 \text{ cm per sec}$$

This value is equivalent to that found for fine-grained sand.

Based on the assumption that the head of the water in the dam is directly related to the volume, the important implication of Darcy's law is that the discharge velocity is:

$$\frac{Q}{t} = \text{Const} \times \text{head}$$

i.e., directly related to the head

Using the above information, the theoretical discharge rate from the tailings dam has been calculated in Appendix C3. The calculation gives a 90% removal of water from the tailing material in 10 years if there is no recharge and storage on the upper surface and a 70% removal after 10 years, allowing for an annual recharge of 24 000 m^3.

The estimated drainage time is only approximate because of:

(1) Uncertainties in the relationship between volume and head.

(2) The variation of permeability of the tailing material.

(3) The difficulty of removing residual pore water due to capillary action.

(4) The difficulty in estimating the recharge rate from rainfall on the upper surface.

With drainage stored on the upper surface of the dam, the measured seepage rate from the tailings dam has been steady throughout the year. The flow by V-notch measurement and including run-off due to rain has been 9.4 - 16.4 m^3/h during June to September 1973 (Appendix C2). The dry-weather flow has been 11 m^3/h. This represents a flowrate of about 100 000 m^3/year at an acid content of 5000 mg/*l* as $CaCO_3$. Water in the top part of the dam (estimated volume about 230 000 m^3) also contains an acidity of about 5000 mg/*l* as $CaCO_3$.

5.2 Quarry Area

The quarry consists of two to four benches up to 100 m wide with a total height of about 80 m and a length of 2000 m cut into the western side of the Dawesley Creek valley (Fig. 2 and Plate 3 and 4).

Seepages occur at six specific sites (QSA to E), mostly at the lower quarry faces, and streams trickle across the benches and in one area joins the Dawesley

Creek (Plate 1). The rock is weathered but flows take place through fractures. Only two (seepages A and B) are related to small, mostly dry, creeks which have been cut by the excavation.

QSA. This stream has a catchment area of approximately 40 hectares in the hills above the mine. The run-off was estimated (4% of rainfall) to be 8200 m^3/a. Prior to entering the mine property, this stream is fresh. However, it passes through sulphide-containing rocks in the mine area and during winter picks up an acid level of 2000 mg/*l* as $CaCO_3$.

QSBs. This stream has a catchment of about 20 hectares above the quarry area and run-off was estimated (4% of rainfall) to be about 4500 m^3/a. Prior to entering the mine area, this stream consists of fresh water. It flows into the mine and appears to be a fairly significant source of QSB.

QSB (in addition to QSBs). This seepage appears out of the quarry face at the lower levels in the middle of the mine area. Flow-rates of QSB plus QSBs have been measured at certain times of the year and it is estimated the QSB is about 4500 m^3/a. The composition varies throughout the year with an average of about 7000 mg/*l* acidity as $CaCO_3$.

QSC. This seepage appears out of the quarry face just below the bench at the southern end of the mine area. Flow measurements have not been made, but it is considered that QSC is the major source of QSD. Hence the flow-rate of QSC is estimated as 13 500 m^3/a with an average composition of 12 000 mg/*l* acidity as $CaCO_3$.

QSD. This is not really a quarry seepage but is the appearance of QSC (and perhaps other sources) entering the Dawesley Creek from the southern end of the mine property. The estimated flow-rate of this stream (from measured concentration balances) is about 13 500 m^3/a. As with other streams, the composition varies throughout the year with an average of about 14 000 mg/*l* acidity as $CaCO_3$.

QSE. This seepage appears out of the quarry face at the lower levels of the northern end of the mine. There are no reliable figures on its flow-rate and it is estimated to be about 4500 m^3/a. The composition varies throughout the year with an average composition of about 7000 mg/*l* acidity as $CaCO_3$.

Days Creek. This creek has a catchment area of approximately 120 hectares in the hills behind the mine area. Taking the run-off of the catchment area to be 4% of the rainfall, the flow is estimated at 23 000 m^3/a. Prior to entering the mine area, this stream is fresh. However, as it passes to the western side of the northern waste dumps, seepages from the dumps enter this creek giving it an acid level during winter of 700 mg/*l* as $CaCO_3$.

There is a considerable area of benches, suggesting that the quarry seepages QSB, QSC and QSE arise mainly from rain soaking into the benches and seeping out of the quarry face at a lower level. Much of the water issuing as seepage could also augment uphill outside the quarry and enter the area though joints in the rock. Further work is needed to clarify the exact origin of the water passing out as quarry seepage.

The problem is important because the degree of effectiveness of sealing the quarry bench (see Section 7.4) primarily depends on the mechanism in which water contacts the sulphide in the rocks.

5.3 Waste Dumps

Coarse rock-waste produced during excavation of the quarry has been dumped in two large waste dumps 700 to 800 m long and 30 to 40 m high in the northern and the southern parts of the area (Plate 6).

The Dawesley Creek runs along the foot of the dumps; in fact, in the south the dumps have encroached on the creek, which has been relocated about 30 m to the east.

Seepages can be seen emanating from the base or, more often, from the sandy banks of the creek immediately under the dumps.

The waste dump seepages are not a single-point seepage but rather a number of small seepages over the length of the dumps. It is very difficult to estimate total flow-rates under these conditions. However, it is estimated (from chemical balances) that the northern and southern waste dumps each contribute about 1000 m^3/a to Dawesley Creek. As with other seepages, the composition changes throughout the year with an average of 50 000 mg/*l* acidity as $CaCO_3$.

5.4 Seepage Composition

Although they have many features in common, the seepages from the tailings dam, quarry and dumps differ in certain respects (Table 10). The quarry seepages contain iron sulphate in a comparatively unoxidised condition, with high Mn and heavy metals, whereas the tailings dam water is rich in iron but is not high as in the quarry and has the lowest pH (2.5), and the waste dump seepages are high in aluminium and heavy metals but comparatively low in iron.

Detailed sampling and analysis throughout the mine area has allowed the sources of the various polluted waters to be located (Appendix B2 and B3). Values for levels of certain elements in water samples vary for the separate sources. One useful parameter in this regard is the Mn to Ni ratio (Table 10). These two elements have been chosen partly because of their range of values and partly because they are least likely to precipitate at pH between 2 and 4. The Mn/Ni ratio in Dawesley Creek water prior to the entry of the southern quarry seepage is normally between 12 and 15, suggesting that the waste dumps and the northern quarry seepages contribute equally to the pollution problem.

The oxidation state of iron varies considerably between the different seepages. Quarry seepage and tailing dam seepage coming directly from the quarry face or tailing dam wall contain exclusively ferrous iron. After lying in pools or storing in ponds, the ferrous iron gradually oxidises to ferric iron. Seepages from the waste dump, drainage in the upper pond and acid water in the Dawesley Creek all contain exclusively ferric iron. The fact that iron has been oxidised under acid conditions implies the presence of aciduric bacteria which can catalyse the oxidation process at low pH values. Biological activity is seen in the form of algal growth in the creek and in shallow pools through which the seepage slowly flow (Plate 2).

Regular analyses of the seepages has shown (with the exception of the tailing dam seepage) marked variations in the concentration levels for each pollutant. Concentration levels for iron and aluminium, and acid values, over a 12-week period (July to September 1973) are shown in Tables 6, 7 and 8. Some 'flushing out' of acid and heavy metals by winter rains has taken place in the quarry seepages, but only a marginal change has been observed for aluminium and acid values in Dawesley Creek water leaving the mine. At no time during the 18-month study has the composition of the Dawesley Creek water leaving the mine been measured and found acceptable with respect to heavy metals and acid concentrations (Appendix B2). There is some evidence to show the importance of collecting the waste dump seepages and preventing them entering Dawesley Creek. A summer flow survey along the Dawesley Creek in the mine (10 April 1973) revealed that seepages of high acid values were entering the creek from the northern waste dump (Fig. 9). Winter flow surveys (7 June and 10 August 1973) revealed that uncontrolled overflows from storage ponds adjacent to the creek were primarily responsible for the high acidity in the Dawesley Creek.

Further surveys are needed to locate the exact positions of the waste dumps seepages and determine their individual effect on the downstream composition of the Dawesley Creek.

By far the most critical factor controlling the amount of acidity and heavy metals entering Dawesley Creek is the flow of water which contacts sulphide-bearing material in the quarry. The higher the water flow, the greater the amount of acidity and heavy metals produced by the seepage (Fig. 10). This implies that the amount of acid and heavy metal salts available for discharge from the quarry is virtually unlimited and the treatment method most likely to succeed involves preventive measures designed to keep water out of the quarry.

5.5 Lifetime Estimates

The tailing dam of 3×10^6 tonnes contains 1.4% S. Analyses indicate an increase of about 3000 mg/*l* acid value as $CaCO_3$ per pass through the tailing material. This is equivalent to a pick-up of 300 tonnes acid as $CaCO_3$ per year at a flow of 100 000 m^3/a or 96 tonnes sulphur (S) per annum.
Assume the sulphur above the phreatic surface is available for leaching (i.e., 16% of the total weight).

∴ Time to leach out available sulphur is:

$$\frac{0.014 \times 0.16 \times 3 \times 10^6}{96}$$

$$= \quad 70 \text{ years}$$

The main quarry rock exposed to water and air is the 2 m layer below the benches. Containing 10% S the exposed amount of rock has been estimated to be of the order of 1×10^6 tonne.
At an average acid value of 8000 mg/*l* as $CaCO_3$ and annual flow of 35 000 m^3/a (i.e. 90 tonne S per annum) the time to remove all the sulphur at an assumed constant rate is:

$$\frac{0.10 \times 1 \times 10^6}{90}$$

$$= \quad 1100 \text{ years}$$

The lifetime of seepages from the waste dumps is difficult to gauge. However, assuming 1×10^7 tonnes of waste material containing 2% sulphur and 100 tonnes of acid (32 tonnes of S) being removed each year, the time required to leach all the sulphur is:

$$\frac{0.02 \times 1 \times 10^{7}}{32}$$

$$= \quad 6000 \text{ years}$$

Assuming only 10% is accessible to attack, the time required is still considerable; i.e., 600 years.

These calculations, although only approximate, are sufficient to show that the problem has a potential lifetime of the order of centuries rather than decades.

6. ORIGIN OF THE ACID MINE WATERS

Atmospheric oxygen dissolved in meteoric water oxidises sulphides during weathering to produce metal and sulphate ions. Metal ions are ultimately adsorbed on fine detrital mineral particles (clay or mud), or are precipitated as insoluble hydroxides, oxides, sulphates, etc., or as evaporite crusts of soluble salts.

The chemical mechanism has received considerable study (Hoffert, 1947; Harrison, 1970; Hawley and Shikaze, 1971; Petracco, 1973) but is not clear in detail because of the complex chemical behaviour of sulphur which may exist as free S and in sulphate, sulphite, polythionates, thiosulphate ions and as polysulphides. Chemical oxidation may be accelerated by specific organisms such as *Thiobacillus ferro-oxidans* and *Ferrobacillus ferro-oxidans*. Iron bacteria *Gallionella, Crenothrix* and *Leptothrix* have been found at the mine.

The basic oxidation processes may be summarised as follows:

$2\ FeS_2 + 2H_2O + 7O_2 \rightarrow 2\ FeSO_4 + 2H_2SO_4$ .. .. .. (1)
(pyrite)

or

$Fe_7\ S_8 + H_2O + 31(O) \rightarrow 7\ FeSO_4 + H_2SO_4$ .. .. .. (2)
(pyrrhotite)

$4\ FeSO_4 + 2H_2SO_4 + O_2 \rightarrow 2\ Fe_2\ (SO_4)_3 + 2H_2O$ .. .. .. (3)

or

$6\ FeSO_4 + 3(O) + 3H_2O \rightarrow 2\ Fe_2(SO_4)_3 + 2Fe(OH)_3$ .. .. .. (4)

$Fe_2(SO_4)_3 + 6H_2O \rightarrow 2\ Fe(OH)_3 + 3H_2SO_4$ .. .. .. (5)

$Fe_2(SO_4)_3 + FeS_2 \rightarrow 3\ FeSO_4 + 2S^\circ$ .. .. .. (6)

or

$6Fe_2(SO_4)_3 + Fe_7S_8 + H_2SO_4 \rightarrow 19FeSO_4 + 7S + H_2S$ .. .. .. (7)

$S + 3(O) + H_2O \rightarrow H_2SO_4$ .. .. .. (8)

or

$S + 3\ Fe_2(SO_4)_3 + 4H_2O \rightarrow 6FeSO_4 + 4\ H_2SO_4$ .. .. .. (9)

The rates of reaction of pyrite ($Fe\ S_2$), marcasite ($Fe\ S_2$) and pyrrhotite ($Fe_n\ S_{n+1}$) under acid mine drainage conditions are significantly different. Braley (1954) found that pyrrhotite reacts at 18 times, and marcasite at 9 times, the rate of pyrite.

The oxidation rates of sphalerite, chalcopyrite, nickeliferous and cobaltferous pyrite relative to the iron sulphides are slower. Ferric sulphate, however, readily attacks these sulphides according to the equation:

$$Fe_2(SO_4)_3 + MS \rightarrow MSO_4 + 2FeSO_4 + S \qquad \text{.. .. ..} \quad (10)$$

where M = Zn, Cd, Cu, Ni and Co

Thus the source of the Fe and SO_4 in the acid mine waters is pyrite and pyrrhotite, Mn comes from spessartite and sphalerite (35% and 2 to 4% Mn respectively), Zn and Cd from sphalerite (the latter 0.19% in sphalerite), Cu from chalcopyrite, and Ni and Co in pyrite (0.01% Ni and 0.02% Co). It should be noted that the formation of thiosalts is favoured by high pH and hence these are virtually absent at Brukunga.

7. TREATMENT OF ACID MINE WATERS

7.1 Literature Review

Mine drainage waters are significant contributors to water pollution, particularly in the United States. The greater part of such water originates in coal mines and the problem is well documented (Nemerow, 1963; Boyer and Gleason, 1972). Much less is known of the effects of metalliferous mining, but the chief references are Hawley and Shikaze (1971), Schmidt and Conn (1971), Watson (1969), Rivett and Oko (1971), Bailey (1970). Stander, Henzen and Fushe (1970) describe the effect of acid mine drainage from gold mines on receiving waters in South Africa. In Australia research is currently in progress by Peko-Wallsend (Anon., 1972) on the treatment of acid mine drainage. Heavy metal pollution from an abandoned mine in NSW continues to be a problem (Haldane, 1971).

Acid mine drainage (AMD) from metalliferous mines differs considerably in composition from that of coal mines in that the heavy metal concentrations are much higher (1000 to 5000 mg/*l* instead of 2 to 100 mg/*l*) and therefore simple limestone neutralisation may be ineffective.

Acidic effluent may be divided into two categories (Petracco, 1973) according to the physical, chemical and biological characteristics; namely, 'acidic mine drainage' and 'tailings pond effluents'. The water from Brukunga is classed as 'acid mine drainage' (even though this includes the effluent from the abandoned tailings dam) because of the high metal content, low pH, relative absence of organic matter and the presence of bacteria.

Most of the current literature on AMD is related to working mines but the problem of effluent from abandoned mines is becoming increasingly important. Acid mine drainage situations reach a peak after the mine has been abandoned and waters loaded with acid and heavy metals may be produced for decades unless brought under control.

Hawley and Shikaze (1971) present ten basic rules for treating acid mine drainage:

1. Define the problem.
2. Do not ignore the preliminary signs of an acid mine drainage situation.

3. Collect all effluents from a specific property at a common point for treatment before discharge to a water course.
4. Do not mishandle raw seepage flows.
5. Use the forces of nature for acid mine drainage treatment whenever and wherever possible.
6. Make the maximum possible use of required water resources.
7. Do not needlessly contaminate unrequired water resources.
8. Maintain the pH of all effluents at a level which will ensure the optimum precipitation of all metals including iron.
9. Do not ignore the action of wind on a tailing area.
10. Revegetate abandoned tailings areas and the downstream slopes of all tailings dams whenever and wherever possible.

Their work indicates that the Brukunga situation is in an advanced stage because the pH of the water is below 3 (hence heavy metals are mobile and neutralisation is expensive) and the acid water has been mishandled. At present, the seepage from the base of the tailings dam, together with fresh seepages from the quarry, is pumped to the top of the tailings dam. Hence the acid water is passed and repassed through the finely divided sulphides in the tailings mass so that more and more metals are taken into solution (equations 6, 7, 8). It is a heap-leaching situation.

A summary of possible treatment methods was given by Barthauer (1966) and a combination of neutralisation with lime or limestone, together with aeration, is commonly used. Processes which have been proposed include: ion exchange (Kunin and Downing, 1971), reverse osmosis (Mason, 1970; Hill and Wilmoth, 1973), activated-carbon catalysis (Mihok, 1969) foaming (Bikerman *et al.*, 1970), freezing (Anon. 1970a), addition of sulphides (Pugsley *et al.*, 1971), production of microbial polymer absorbents (Dugan, 1970), flash evaporation (Schmidt and Conn, 1971), neutralisation and aeration (Harrison, 1970; Wilmoth and Hill, 1970; Temmel, 1971; Anon. 1971b) and microbial treatment (Anon., 1971a).

The treatment of the acid at its source or the prevention of its formation has been considered in the United States (Besselievre, 1969).

Procedures examined include:

(1) Backfilling abandoned mines with rock.

(2) Diverting drainage water.

(3) Chemical grouting to coat sulphide minerals.

(4) Spraying bacteriophage viruses to kill acid-forming bacteria.

Whether found to be useful or not they have proved to be very expensive.

The use of vegetation to stabilise mine tailings areas has been described by Young (1969). Care has to be taken in the type of vegetation, and soil fertilisation and pH adjustment. Fully developed vegetation will transpire from 50 to 100 m^3 of water per day per hectare (Leroy and Keller, 1972).

7.2 Dilution

Controlled dilution of the acid mine drainage by the upstream Dawesley Creek water to precipitate the heavy metals and neutralise the acidity is one method which has been considered.

The total flow of acid mine drainage has been estimated as follows:

Source	Annual flow m^3	Acid Value as $CaCO_3$ mg/*l*
Tailing dam	100 000	5 000
QSA	8 200	2 000
QSE	4 500	7 000
QSB	9 000	7 000
QSC + QSD	13 500	12 000
Days Creek	23 000	700
Waste dumps	2 000	50 000
TOTAL	160 200	5 500

The average alkalinity concentration of Dawesley Creek water has been estimated from measured data (Appendix B1) to be 130 mg/*l* as $CaCO_3$. Therefore, to achieve at least some residual alkalinity after blending, a 42:1 flow ratio of Dawesley Creek water to acid mine drainage would be needed. This is far greater than observed 5.1:1 ratio based on the estimated average flow data for Dawesley Creek (Appendix C1). As indicated by the chemical analyses already carried out, the Dawesley Creek water upstream from the mine is unable to effectively dilute even the existing uncontrollable seepages entering the creek. The existing procedure for controlling seepages in the mine is shown diagrammatically in Fig. 11.

7.3 Diversion

The needless contamination of fresh water streams by the pollutants must be avoided according to Rule 7 of Hawley and Shikaze (1971).

When flowing during winter, Days Creek, QSA and QSBs enter the quarry as surface water from outside the area and pick ups acidity and heavy metals. By improved channelling or piping, it should be possible to prevent these streams from contacting the sulphide-bearing rock in the mine.

Although further design details are needed, Dawesley Creek must be diverted away from the waste dumps in the mine. The bypass pipeline with weirs proposed by D. Kingston (E & WS 2717/73) overcomes the fresh water-seepage separation problem by using the Dawesley Creek bed in the mine as the collection point for the 'difficult-to-collect' seepages.

The concentrated seepages from QSE, QSB, QSC and QSD, as well as those from the waste dumps, must be dammed and pumped to the upper part of the Q tailings dam for storage. The tails dam seepage is recycled as it is and stored on the top of the dam.

The estimated cost of constructing the total diversion scheme is:

1.	'Cleaning up': QSA Removing any obvious sulphide rocks and generally cleaning up by making path of QSA as straight as possible	$ 2 000
2.	QSBs directed into QSA: Redirect along the quarry face by a small trench to run into QSA	$ 2 000
3.	Damming of Dawesley before it enters the mine area: Dam to hold 250 to 500 m^3 water	$15 000
4.	Diversion pipe with connections from combined Days Creek, QSA, QSBs and Brukunga sewage effluent	$80 000

5. Damming of QSE:

Construction of a small wall approximately 1 m and covering area with acid-proof lining	$ 2 000
50 mm dia. PVC pipe running to QSB storage (installation incl.)	$ 1 700
50 *l*/min pump with power supply, level control (installation incl.)	$ 2 400
	$ 6 100

6. Damming of QSB:

Construction of a small wall approximately 1.5 to 2.0 m and covering floor and walls of area with acid-proof lining (all auxiliary equipment is available and is presently feeding seepages QSA, B and E and Days Creek to holding dams below the tailings dam)	$ 3 000

7. Damming of QSC:

Consolidation of dam foundations, construction of a 1 m wall around dam	$10 000
100 mm dia. PVC pipe running to QSB storage (installation incl.)	$ 4 500
250 *l*/min pump with power supply, level control (installation incl.)	$ 3 600
	$18 100

8. Excavation of hole at QSC (lower):

Excavation of hole and blocking of water path flowing to QSD	$ 5 000
250 *l*/min pumps to elevate water up to QSC (dam) (installation and power supply incl.)	$ 7 200
100 mm PVC pipe passing from QSC (lower) to QSC (dam) (installation incl.)	$ 1 500
	$13 700

9. Construction of small dam on Dawesley:

Small dam of 50 m^3 capacity	\$ 5 000
100 mm PVC pipe passing from creek to QSC (lower) (installation incl.)	\$ 5 700
250 *l*/min pumps to elevate water up to QSC (lower) (installation and power supply incl.)	\$ 7 200
	\$17 900

Total estimated cost	=	157 800
+ 10% contingency	=	17 200
		\$175 000

The total estimated cost of separating the AMD from the creek water is \$175 000. Additional expenditure will have to be made to treat or dispose of the water collected.

There will also be an additional cost of pumping and some labour supervision and maintenance of the equipment to ensure proper running. Pumping operating costs are estimated as \$1500/a and labour at 1 day/fortnight as \$500/a. This gives an operating cost of \$2000/a.

The diversion scheme and collection of seepages offers several distinct advantages including:

1. In-built control of all seepages entering Dawesley Creek. Acid waters enter the creek only during flood periods when the bypass pipeline cannot handle the flow. Design characteristics of the weirs should ensure that any acid overflowing is effectively diluted by the fresh water.
2. The concentrated seepages from the waste dumps containing the more toxic elements Ni, Co, Cu, Zn and Cd are effectively impounded. All the concentrated acid seepages can be collected in the creek at the bottom of the mine and pumped along with all the quarry seepage for storage on the tailing dam.
3. Overflows from the ponds used for holding quarry seepages need not enter the creek uncontrollably as at present.

The diversion scheme is shown diagrammatically in Fig. 12.

7.4 Prevention

Preventive measures of keeping water, air and sulphides apart have been tried overseas, usually without any degree of success. In the Brukunga mine situation it is possible that sealing the quarry benches as proposed by E & WS Officers (2717/73, 15 May 1974) can substantially reduce the flow of quarry seepages and effectively control pollution from this source.

The solution proposed is to: (quote from E & WS Report)

'(a) Clean up, regrade and seal the surfaces of the quarry benches so that run-off from these areas is prevented from mixing with the quarry dust on the surface of the benches. All materials cleaned off the benches shall be transported to the tailings dump. It will probably be advantageous to catch the run-off in a drain at the base of the slope above the bench and then transport it to the downstream end of the quarry. The bench at the base of the cuts at locations QSE and QSB would require grading and sealing as these localities are a ready source of pollutant.

Aerial contours of the bench areas to a scale of 1 to 500 with a contour interval of ½ m would be required before a design of a drainage system could be finalised.

(b) Provide a small storage (approximately 1 m deep) across the old Dawesley Creek at seepage D for the storage of pollutants - the main source of which, having carried out (a) above, would be run-off and seepage from this southern mullock heap. From the end of September to approximately the 1 July the following year (i.e., after the first major run-off) this pondage would be pumped to the evaporation basin. During the winter period the evaporation system could be maintained or a dilution release could be made downstream. This decision would require accurate measurement of the seepages including acidity.

A very preliminary estimate of cost of cleaning, sealing and draining the benches, small creek diversion and the construction of the pollutant pond including the pipeline and pipe is $500 000. The unit rate assumed for bench cleaning, sealing and draining was $2.4 per square metre ($2.00 per square yard).'

With the added requirement of dewatering Dawesley Creek to prevent waste dump seepage contamination and collecting the concentrated seepage (see Section 7.3 on diversion), the cost of $120 000 must be added. The complete scheme shown diagrammatically in Fig. 13 is therefore estimated to cost approximately $620 000.

The scheme provides for the indefinite recycling and evaporation of the seepage from the tailings dam and an additional cost to operate and maintain the pumps. This operating cost has been estimated at $2000/annum.

Further field studies are needed before implementing the scheme, to determine the effectiveness of a bench sealant in reducing the amount of quarry seepage and to measure the natural evaporation rate from the seepage stored on the tailings dam.

7.5 Evaporation

The comparative rates of precipitation and evaporation are an important factor in the disposal of water from mining operations. Many such operations are in dry areas where disposal of liquid is no problem because evaporation exceeds precipitation.

Figures for Brukunga are as follows (Appendix C4):

Total annual precipitation	512 mm (20.13 inches)
Calculated evaporation	1233 mm (48.56 inches)
Net annual evaporation	722 mm (28.42 inches)

Thus an alternative to chemical neutralisation with its high plant and staffing costs plus the problem of disposal of voluminous sludges (see below in Section 7.6) is to collect the concentrated polluted water into evaporation ponds, allow it to dry up, and dispose of the dried residues.

The area necessary to evaporate 140 000 m^3 per year for a net evaporation rate of 722 mm is 19 hectares. However, the required area would be greater in a wet year, and it is calculated that 1 year in 10 require an area of 27 hectares to evaporate 140 000 m^3. It is assumed that 25 hectares would be sufficient to cope with most circumstances.

Unfortunately, the area is rather hilly and there is little area flat enough to construct evaporation ponds. The top of the tailings dam could provide 21 hectares, and flat benches in the quarry another 18 hectares, but not all of this area would be suitable. In addition it would be essential to seal off these evaporation ponds so that water did not re-enter the system through the dam or the benches.

A number of sealing materials have been considered (clay, polymers, concrete, bitumen, soil-cement, etc.). Polymer membrane (e.g., PVC, polyethylene) is the only lining known to give satisfactory and reliable service (Kumar and Jedlicka, 1973); however, to line 25 hectares with plastic sheet is expensive.

The cost of evaporation pond site preparations will be minimal and it is estimated that covering of 18 hectares of ponds will cost (including burying the lining with earth or caustic mud):

$220 000 for polyethylene lining ($1.80/m^2; $1/yd^2).

$330 000 for PVC lining ($1.80/m^2; $1.50/yd^2).

Under neutral conditions, PVC lining has been shown (Anon., 1970) to completely seal and it is the recommended material for lining.

An additional sum of $50 000 should be allowed for cleaning up the evaporation ponds once the problem diminishes. Hence a total expenditure of $380 000 for evaporation is estimated.

It is not known if evaporation has been used as a method of treating AMD on a large scale. For this reason, it will be necessary to carry out some investigations to test the feasibility of evaporation. This would involve:

(1) Testing of evaporation rates at Brukunga.
(2) Testing of seepage rates through membrane lining materials.
(3) Testing the breakdown of fabric membrane lining by AMD and checking the need to place a layer of limestone or any alkaline material to neutralise the AMD in contact with the membrane.

The total evaporation scheme assumes the necessary prerequisite of diverting the fresh water streams as presented diagrammatically in Fig. 14 (first year's flows).

The siting of the evaporation ponds within the quarry should result in a lower discharge rate (say 50%) from the quarry seepages, assuming most of recharge occurred via the benches. Annual discharge would be of the order:

	m^3
QSE	3 000
QSB	3 000
QSC and QSD	8 000
Waste dumps	2 000
Run-off	18 000
	34 000

The evaporation ponds on top of the dam would also minimise the amount of water entering the tailing material.

It has been calculated that the upper storage dam would be dry in 3 years and the tailings dam would be 70% drained in 10 years. After 10 years, the area required for evaporation would be reduced to 12 hectares. The treatment of approximately 10 000 to 65 000 m^3 per annum would then continue indefinitely. The area required for evaporation would then progressively decrease down to a minimum of approximately 9 hectares.

Not including caretaker charges and pumping costs of approximately \$2000/a, the total evaporation schemes has been estimated to cost:

Diversion of fresh waters and collection of seepages	\$175 000
Evaporation ponds	380 000
	\$555 000

The advantages of the evaporation scheme at Brukunga are:

(1) A much lower operating cost than neutralisation, and therefore more suited to the abandoned mine situation.

(2) Less dependence of cost on the estimated drying-out time of the tailings dam.

The disadvantages include:

(1) Any failure of the lining used to separate the acid water and the underlying sulphide-bearing materials would be disastrous, especially for the ponds in the quarry area. The contact of the acid drainage with the sulphide would not only result in a loss in the

amount of water to be evaporated but the flows and the acid (and heavy metal) content of the seepages would increase substantially.

(2) The large area of ponding would be aesthetically unattractive and would have to be fenced off so as to be no danger to children.

(3) A dust problem may arise when the ponds are evaporated to dryness, but it is expected that a non-dusting residue will be left.

Construction and installation procedures would have to be such that the above objectionswere kept to an absolute minimum.

7.6 Neutralisation

The addition of alkali to acid mine drainage is the most widespread treatment method.

Where feasible to minimise chemical and plant costs for neutralisation, fresh water diversion methods and preventive measures must first be incorporated. As suggested by Hawley and Shikaze (1971), the forces of nature for acid mine treatment must also be used. To permit complete oxidation of ferrous iron to ferric iron in the seepages and introduce savings in the amount of alkali chemical used, ponding is the recommended procedure, for which some important conditions need satisfying:

(a) The retention time for ponding must be as great as possible (of the order of months rather than days).

(b) The pond should be as shallow as possible (to enhance oxygen transfer from the atmosphere).

(c) The pond should not be allowed to come into contact with iron sulphides.

The pH of the treated drainage is important in that the value must be kept high enough to ensure complete precipitation of all the heavy metals. Manganese is a potential pollutant in the Brukunga drainage and therefore the pH of treated drainage must be raised to a value of 8.5 (Fig. 15).

A laboratory study was conducted to determine the neutralisation requirements to provide an effluent of acceptable quality.

Five types of alkaline were considered:

(1) Lime (Hydrated lime, $Ca(OH)_2$)

(2) 'Caustic mud'

This finely divided material is precipitated calcium carbonate produced during the manufacture of caustic soda from soda ash:

$$\underset{\text{lime}}{Ca(OH)_2} + \underset{\text{soda ash}}{Na_2CO_3} \rightarrow \underset{\text{caustic soda}}{2NaOH} + \underset{\text{caustic mud}}{CaCO_3\downarrow}$$

A small residual of caustic soda remains with the washed 'caustic mud'.

(3) 'Milk of Lime' Grit

The material is produced during the slaking of lime:

$$\underset{\text{quick lime}}{CaO} + H_2O \rightarrow \underset{\text{hydrated lime}}{Ca(OH)_2}$$

The part of the quick lime which does not hydrate is termed the 'milk of lime' grid. It is essentially coarse unburned limestone containing encapsulated hydrated lime.

(4) Marble

This is a high-grade form of calcium carbonate used for manufacturing high-grade lime.

(5) Agriculatural limestone

This is lower-grade limestone, containing a higher proportion of silica, iron, magnesium etc.

The suppliers of the alkali, the estimated cost of each material at Brukunga and the composition of the alkali are given in Table 11.

Neutralisation studies were carried out on a sample of acid water obtained from the upper tailing dam storage on 7 June 1973 (analysis, Appendix B3).

A 500 m*l* sample of water was stirred vigorously while certain quantities of alkali were added. Stirring was continued for ½ h. A 10 m*l* sample was then taken for analysis. The procedure was repeated until a pH above 6 to 8 was reached. Samples were analysed for Fe, Zn, Mn, Ni and SO_4. The results are presented in Table 12. With the exception of lime, a considerable excess of reagent was required to reach a suitable pH. As expected, the pH of the solution to which caustic mud was added did not rise very high above 6 but a much higher value (> 8) was obtained with the 'milk of lime' (MOL) grit and the lime. Although it was not examined, quicklime would be expected to react as efficiently as hydrated lime in the stirred system.

Using the same stirring apparatus but a smaller (250 m*l*) volume of acid water, a series of rate tests was conducted on the limestone and waste alkalis. The pH of the slurries was measured with time to determine the rate of neutralisation. Three consecutive series of tests were done to determine the effect of the:

(1) Alkali type.

(2) Particle size of marble.

(3) Particle size of MOL grit.

(4) Removal of CO_2 by aeration.

(5) Stepwise addition of reagents.

The results showed that of the raw materials used, only caustic mud could be utilised effectively to produce a pH of 6 (Fig. 16). Marble and MOL grit performed satisfactorily when ground to minus 72 micrometre (Fig. 17). In the case of MOL grit and caustic mud, aeration had a marked effect on the rate at which the pH increased (Fig. 18).

To achieve the removal of Mn down to an acceptable level (< 2 mg Mn/*l*), lime either as fine grit or hydrated lime was needed to raise the pH above 8. After adding 50% excess of caustic mud to the acid water and aerating for 25 minutes a pH higher than 6 was attained. To reach a pH of greater than 8 the addition of lime was needed. The stepwise addition of grit to take full advantage of the lime content at higher pH values was effectively demonstrated (Fig. 19). When added stepwise a final pH of 8.7 was obtained (cf. pH 8.2) and the Mn concentration was 2 mg/*l* (cf. 20 mg/*l*).

After the neutralisation tests, the settling rates of the sludges produced were plotted. The MOL grit crushed to minus 72 micrometre provided the best settling rate with adequate neutralisation. The final volume of sludge obtained was considerably small for the grit and the caustic/lime combination (10%) relative to that produced by lime alone (25%) (Fig. 20).

The laboratory study showed that the addition of MOL grit or caustic mud/lime in two stages was the preferred neutralisation procedure. The cost differential between grit and caustic mud plus lime was marginal but both were considerably cheaper than lime alone for both hydrated lime and quicklime (Table 13).

Because of its ready availability as a waste material, caustic mud is the preferred basic alkali. The flow sheet for the addition of caustic mud followed by lime is shown in Fig. 21.

The following design basis has been used:

Plant capacity 140 000 m^3/a

Labour required 2 men

Plant operational time 40 h/week (i.e., 1 shift)

and 40 week/a

or 1600 hours/annum

Average flow rate of AMD will be $\frac{140\ 000\ m^3}{1600} = 90\ m^3/hr$

Equipment Requirements

1st Neutralisation Tank:

1 h minimum residence time. Size = 100 m^3. Provision has to be made for the addition of air to the tank from an air compressor.

2nd Neutralisation Tank:

½ h minimum residence time. Size 50 m^3.

Lime Tank:

Lime requirements are 110 tonnes/a or 70 kg/h. Arranging this circuit so that make-up is necessary once per day, a 10 m^3 capacity tank will be required (say, 2.3 m diameter by 2.1 m deep).

An effective lime feeding device for the control of pH in the second neutralisation tank is required.

Pumps:

It will be necessary to have three 2 m^3/m pumps (one to pump feed of AMD to plant, one to pump outlet of second neutralisation vessel to 15 m thickener, and one spare) that are acid resistant. A thickener underflow pump is also required to pump slurry to the disposal area (approximate capacity 250*l*/m).

Operation Plant:

It will be necessary to charge caustic mud to the first neutralisation tank (8 to 10 tonnes/day) and the simplest way appears to be a bulk addition of 0.5 tonnes every half hour. The first neutralisation tank will have to be provided with a ramp by which a front-end loader can feed the vessel. National Lime Association describes suitable equipment (Anon., 1971c).

Thickener:

Using the method of Couche and Goldney (1959), the thickener area requirement was determined from settling tests as 0.084 m^2/m^3 liquid treated/24 h.

Thickener area for 100 m^3/h

$$= 0.084 \times 100 \times 24$$

$$= 201.6 \text{ m}^2$$

Diameter of thickener

$$= \sqrt{\frac{201.6 \times 4}{\pi}}$$

$$= 16 \text{ m}$$

Cost Estimate of Neutralisation Plant:

		$
1.	Two off, neutralisation tanks 90 m^3 (installed)	90 000
2.	Air compressor and air lines	4 000
3.	Thickener 16 m diameter $30 000 purchased (installed)	90 000
4.	Three off, 2 m^3/m rubber-lined pumps (installed)	1 200
5.	Two off, 250 *l*/m slurry underflow pumps	5 000
6.	pH controller, activated feeder for lime	5 000
7.	Lime holding tank (installed)	5 000
	Total	211 000
	20% contingency factor	39 000
		250 000

Operating Costs:

Labour 2 men @ $5000 each		10 000
Chemicals Caustic Mud	1600 t/a	
Lime	110 t/a	9 500
Power		2 000
Payroll overhead, plant overhead maintenance of equipment (including front-end loader) etc.		6 000
		27 500
Contingency 10%		2 500
Total		30 000

Acid Values and Minor Constituents*

Date	Acid Value as $CaCO_3$	mg/l				pH
		Ni	Co	Cu	Cd	
		Sampling Point (2)				
11/ 8/72	Nil	-	-	<0.05	-	7.4
4/ 9/72	Nil	-	-	<0.01	<0.01	8.2
10/ 4/73	Nil	<0.05	<0.05	<0.05	<0.01	7.9
7/ 6/73	Nil	<0.2	<0.2	<0.1	<0.02	7.6
13/ 7/73	Nil	<0.1	-	-	-	7.6
20/ 7/73	Nil	<0.1	-	-	-	7.6
3/ 8/73	Nil	<0.1	-	-	-	7.6
10/ 8/73	Nil	<0.1	-	-	-	7.05
17/ 8/73	Nil	<0.1	-	-	-	7.35
23/ 8/73	Nil	<0.1	-	-	-	7.9
31/ 8/73	Nil	<0.1	-	-	-	7.3
7/ 9/73	Nil	<0.1	-	-	-	7.25
14/ 9/73	Nil	<0.1	-	-	-	7.55
20/ 9/73	Nil	<0.1	-	<0.1	<0.005	7.9
21/ 9/73	Nil	<0.1	-	-	-	7.5
28/ 9/73	Nil	<0.1	-	-	-	7.65
4/ 4/74	Nil	0.04	<0.05	<0.02	0.02	6.9
		Sampling Point (3)				
11/ 8/72	-	-	-	0.25	-	3.8
5/ 1/73	-	-	-	2.0	-	2.7
10/ 4/73	-	1.8	3.4	2.2	0.67	2.8
7/ 6/73	1920	1.4	1.0	0.60	0.11	2.8
20/ 9/73	945	0.5	0.30	0.30	0.09	3.4
4/ 4/ 73	4650	3.6	1.45	0.95	0.30	2.7
		Sampling Point (4)				
4/ 9/72	-	-	-	0.44	0.15	2.8
7/ 6/73	2200	1.6	1.0	0.80	0.17	2.9
6/ 7/73	-	-	-	-	-	3.1
20/ 9/73	1070	0.8	0.5	0.3	0.11	3.3
6/ 2/74	2850	1.8	0.91	1.1	0.22	2.7
6/ 3/74	5500	3.9	2.0	-	0.33	2.6
6/ 4/74	1630	1.3	1.45	0.95	0.10	2.9

---

* Sampling points described under Major Constituents.

Acid Values and Minor Constituents*

Date	Acid Value as $CaCO_3$	mg/l				pH
		Ni	Co	Cu	Cd	
Sampling Point (5)						
11/ 8/72	-	-	-	0.35	-	3.5
20/ 9/73	515	0.40	0.30	0.20	0.045	3.45
Sampling Point (6)						
11/ 8/72	Nil	-	-	<0.05	-	7.8
20/ 9/73	Nil	<0.1	<0.1	<0.1	<0.005	8.4
Sampling Point (7)						
11/ 8/72	-	-	-	0.25	-	3.6
20/ 9/73	400	0.3	0.3	0.2	0.035	4.3
Sampling Point (8)						
20/ 9/73	230	0.3	0.1	0.2	0.025	4.0
Sampling Point (9)						
20/ 9/73	-	<0.1	<0.1	<0.1	0.01	8.7
Sampling Point (10)						
20/ 9/73	Nil	<0.1	<0.1	<0.1	0.015	6.4
Sampling Point (11)						
11/ 8/72	-	-	-	0.25	-	4.9
20/ 9/73	Nil	<0.1	<0.1	<0.1	0.005	7.35
4/ 4/74	1120	1.0	0.60	0.26	0.08	4.0
Sampling Point (12)						
11/ 8/72	Nil	-	-	<0.05	-	<7.4
20/ 9/73	Nil	<0.1	<0.1	<0.1	0.005	8.1
4/ 4/74	Nil	<0.02	<0.05	0.02	<0.01	8.6
Sampling Point (13)						
11/ 8/72	Nil	-	-	<0.05	-	6.5
20/ 9/73	Nil	<0.1	<0.1	<0.1	0.01	7.65
6/ 2/74	270	0.28	0.175	0.17	0.045	4.3
4/ 4/74	350	0.46	0.30	0.12	0.05	4.3
Sampling Point (14)						
6/ 2/74	320	0.36	0.24	0.10	0.055	4.2

In implementing the neutralisation scheme the quantity of AMD to be treated has a considerable bearing on the total cost. For the flows for the first years' operation, see Fig. 22.

Estimated annual discharge of acid mine water under the diversion scheme outlined earlier involves:

Source	Annual Flow m^3	Acid Value (as $CaCO_3$ in mg/l)
QSE	4 500	7 000
QSB	4 500	7 000
QSC + D	13 500	12 000
Waste Dumps	2 000	50 000
Surface run-off	18 000	negligible
Total	42 500	8 000

The seepage from the tailings dam is the greatest source of AMD (100 000 m^3/a).

Calculation of the total volume feeding this seepage is very complex and it has only been possible to arrive at an approximate value (see Section 5.1).

Calculations based on the dimensions of the tailings in the dam, the tailings' permeability, the use of Darcy's Law and the contribution of rain, suggest that if recycling of the seepage were to cease, then 70% of the water in the tailings dam will drain away after 10 years.

Thus, in the first year of control:

Seepage rate from quarry and waste dumps	42 500 m^3
Net soakage to dam due to rain	24 000
Volume of water on top of tailings dam	230 000
Volume of water within tailings dam	420 000
Total water content available (dam+seepages)	716 500
Less natural evaporation	35 000
Net water available for treatment	681 500
Amount treated by neutralisation	140 000
Amount remaining in storage	541 500

Thus 541 500 m^3 will remain at the end of the first year, of which 420 000 m^3 will remain within the body of the dam and the volume on top will be reduced from 230 000 to 121 500 m^3.

On the figures suggested, the pond at the top of the tailings dam should dry up after 3 years and drainage from the dam itself should become negligible in less than 10 years. Neutralisation would initially treat 140 000 m^3 per year, and after the first decade would treat annually only 42 500 m^3, but the period of treatment would be indefinite.

The operating cost for the 10-year neutralisation programme would be as follows:

Year	Volume of drainage, m^3	Labour Cost, $	Power Chemical Cost, $	Maint. etc., $
1	140 000	10 000	11 500	8 500
2	140 000	11 000	12 600	9 300
3	140 000	11 600	13 300	9 900
4	140 000	12 200	14 000	10 300
5	100 000	12 700	10 500	10 800
6	85 000	13 400	9 300	11 300
7	75 000	14 900	8 600	12 000
8	70 000	14 800	8 400	12 600
9	63 000	15 500	8 000	13 200
10	58 000	16 300	7 700	13 900
		132 000	104 000	112 000

Total $348 000 say, $350 000

(A 10% per annum escalation of costs has been included)

Acid drainage to the extent of 45 000 to 50 000 m^3/a will continue to be produced in the quarry for many decades and will have to be disposed of as economically as possible.

The most convenient way would be to use an evaporation pond on the top of the tailings dams or on the quarry benches. An area of 9 hectares would be required and the same pond would be used to dispose of the sludge produced by neutralisation. The cost of such a pond would be of the order of $100 000.

Not including the cost of pumping the acid drainage and caretaker charges (at approximately $2000/a) the total neutralisation scheme has been estimated as follows:

Diversion scheme	175 000
Capital Cost of plant	250 000
Evaporation ponds	100 000
Operating cost for 10 years	350 000
	$875 000

It is unfortunate that the crushing and milling equipment on site has been dismantled and sold.

Considering the case prior to the selling of the plant equipment at the mine, and using tanks, compressors and thickener already installed, the capital cost of plant become $50 000 and the total cost $675 000.

7.7 Revegetation

As suggested by Hawley and Shikaze (1971) it is important to revegetate abandoned areas whenever and wherever possible. Regardless of which ever treatment method is finally adopted, a revegetation scheme to develop grasses on available benches, waste dumps and tailing dam surface should be carried out.

This will ensure that surge flows of acid drainage are contained and then are less likely to enter the Dawesley Creek. Experience overseas has shown that the development of suitable top soil for supporting growth on pyrite material is difficult. Most problems of heavy metal toxicity have been overcome by covering the pyrite material with limestone or its equivalent, e.g., caustic mud (up to 200 tonnes per hectare). Vegetation would not only control the flow but assist in preventing contact between water and the sulphide-containing materials. Some water would also be removed by evapotranspiration.

7.8 Comparison of Schemes

A cost comparison summary of the basic treatment methods is given in Table 14 and Fig. 23. The detailed description of the schemes have been provided elsewhere in the Report.

EXISTING SCHEME - See Fig. 11.
DIVERSION SCHEME - See Section 7.3 and Fig. 12.
PREVENTION SCHEME - See Section 7.4 and Fig. 13.
EVAPORATION SCHEME - See Section 7.5 and Fig. 14.
NEUTRALISATION SCHEME - See Section 7.6 and Fig. 22.

Although necessary, none of the schemes include the cost for an accelerated revegetation programme in the mine. Neutralisation is the most satisfying (but expensive) scheme, in which fresh water entering the quarry area are diverted and the acid waters after impounding are neutralised by caustic mud and lime. The main advantages of this scheme are:

1. The acid waters are completely treated before being disposed of.
2. The effectiveness of the scheme can be easily controlled and the water quality reliably monitored for pH.
3. The scheme is flexible and caters for the possible future reopening of the mine.

The main objection to the scheme in the abandoned-mine situation is the high proportion of operating costs relative to the capital expenditure. The Prevention and Evaporation schemes are comparable in cost and effectiveness in keeping acid waters away from Dawesley Creek. Provided experimental investigations confirm adequate stability of the tailings dam for the next 20 to 30 years and that the quarry benches can be effectively sealed by a bitument-type lining prevention is the preferred method because it introduces more preventive measures than Evaporation and avoids the need for a large area of evaporation ponds.

The cheaper Diversion scheme is only a partial solution to the problem because no provision has been made to treat the acid waters once they are diverted from the fresh-water streams.

The Diversion scheme is an important part of the other three schemes, however, and therefore can be implemented as the first stage of solving the Brukunga acid drainage problem. The Dawesley Creek bypass design has yet to be verified but other aspects of the scheme have been well studied and can be implemented without delay. The initial quarry area diversion scheme is shown diagrammatically in Fig. 24.

8. CONCLUSIONS AND RECOMMENDATIONS

The survey of the mine area and downstream Dawesley Creek over 12 months have shown that significant amounts of acid, sulphate and heavy metals (in particular iron, aluminium, manganese and zinc and trace elements nickel, cobalt, copper and cadmium) are entering the Bremer River system even when using the procedure of collecting northern quarry seepages and the tailings dam seepage and stopping them from entering Dawesley Creek. Under the best control conditions the Dawesley Creek has contained dissolved toxic amounts of one or more of the above heavy metals for at least 16 km downstream. Under adverse conditions when uncontrolled or 'out-of-phase' releases of acid waters have occurred the river system has been polluted with dissolved heavy metals as far downstream as Langhornes Creek (65 km downstream).

An extensive examination of the situation has shown that there is no quick, cheap method of solving the acid drainage problem. The production of acid waters high in dissolved heavy metals has reached an advanced stage in the abandoned mine. From experience elsewhere, the problem can be expected to become worse if it is left unresolved.

Because of the lack of sufficient hydrological data, the exact scheme required to solve the acid mine drainage problem cannot be specified. However, one of three basic schemes, each with a capital expenditure greater than $600 000, must be adopted:

Prevention Scheme:

Diversion of fresh waters including Dawesley Creek, sealing the quarry benches and recycling acid waters from the tailing dam and quarry.

Evaporation Scheme:

Diversion of fresh waters including Dawesley Creek and evaporating acid waters in ponds.

Neutralisation Scheme:

Diversion of fresh waters including Dawesley Creek and neutralisation of acid waters.

The neutralisation scheme is the most satisfying and safest, but also the most expensive scheme, the one most studied for Brukunga and most commonly used in similar situations elsewhere.

The prevention scheme is the preferred scheme because:

(1) The total costs should be equal to or lower than the alternative methods.

(2) The scheme contains considerable preventive measures to keep fresh water away from sulphide-containing materials.

(3) The costs for operation and maintenance are the lowest.

(4) The ponding area required is the smallest of all the schemes. It is, however, the least studied of the three schemes and needs further investigation.

The evaporation scheme, based on currently available hydrological data, is economically competitive, is low in operating costs, but is visually unattractive because of the large ponding area required.

Regardless of which scheme is finally adopted it is important to implement, as soon as possible, a revegetation scheme to promote a vegetation cover on the waste dumps, and tailings dam surfaces. This will increase evaporation, reduce entry of surface water, reduce surges in acid mine drainage and reduce the visual impact of the area.

The waste dumps will probably become the major source of polluted water after the tailings dam has been drained but the material is so coarse (rock lumps tens of cms across) that there does not appear to be any simple method of treatment. If soil can be placed and held on the steep sides until vegetation became established, the drainage through the dump may be reduced.

Before the final choice of a scheme is made, it is recommended that further investigations be carried out to verify certain assumptions which have been made. It is recommended that the following investigations be made:

(1) A hydrological and chemical study of the movement of water in the tailings dam to determine the likelihood of the dam remaining stable until it dries out after approximately 20 to 30 years.

(2) An investigation to measure the evaporation rate of water from the upper tailings dam storage to determine the annual amount of acid water which can be 'treated' by this method.

(3) A pilot trial of sealing a portion of the quarry bench at one seepage locality (e.g., QBS) to determine the feasibility of substantially reducing the flow of quarry seepages by this method.

(4) The measurement of levels on the main quarry benches and design of bench drainage.

(5) Surveys during September 1974 to April 1975 to confirm the significant contribution and exact source of heavy metals from the waste dumps into Dawesley Creek and the design of the bypass pipeline.

(6) The requirements for revegetating the tailings dam and the waste dumps be examined by carrying out the appropriate field trials.

To substantially reduce the amount of acid mine drainage currently entering the Dawesley Creek, it is recommended that the drains and channels for Days Creek, QSA and QSBs be cleaned up as soon as possible. Also, a holding pond should be built below QSC and a pump and pipeline installed to transfer seepage to a storage locality either in the QSB area or on the tailings dam (see Fig. 24). The diversion of these fresh-water streams and collection of QSC (and QSD) seepage must be carried out regardless of whichever of the three alternative schemes is finally adopted.

The short-term problem of the annual build-up acid waters on the tailings dam resulting in illegal discharges of acid drainage during the summer and winter months must be solved as soon as possible. Although the annual discharge of the stored seepage into the Dawesley Creek during the month of the year in which water is not required (e.g., May or June) is the cheapest solution, the preferred method is to use a portable lime neutralisation plant to neutralise the acid overflows on demand.

9. ACKNOWLEDGEMENTS

The co-operation provided by officers of the Engineering and Water Supply Department in supplying reports relevant to the investigation is appreciated.

The assistance given by Mr K. Pile of the South Australian Institute of Technology in supplying maps of the tailing dam is also acknowledged.

10. REFERENCES

ANON. (1968). Water quality criteria, *Report of the National Technical Advisory Committee FWPC, U.S. Department of Interior.*

ANON. (1970a). Purification of mine water by freezing. *Applied Science Laboratories Inc. Report F.W. & A. Grant No.* 14010 DRZ.

ANON. (1970b). Pond linings for desalting plant effluents. O.S.W. Report No.602, U.S. Department of Interior.

ANON. (1971a). Microbiological treatment of acid mine drainage waters. *Continental Oil Company. Programme 14010 ENW for Environmental Protection Agency.*

ANON. (1971b). Pilot-scale study of acid mine waters. *Ohio State University, Programme 14010 EXA for Environmental Protection Agency.*

ANON. (1971c). Lime — handling application and storage. Bulletin 213 Publ. National Lime Assoc. Washington DC 2nd Edition.

ANON. (1972). Neutralisation of acid mine waters, Peko-Wallsend. *Bulletin Australian Mining Industry.*

ANON. (1973). Draft manual for assessing environmental impact in N.S.W., *Minister for Environmental Control.*

BAILEY, R.P. (1970). Case for reclamation of mineral processing water. *Can. Min. J.* 87-92, June.

BARTHAUER, G.L. (1966). Mine drainage treatment, fact and fiction. *Coal Age,* 79-82, June.

BICKERMAN, J.J., HARRISON, P.J. and ROSE, S.H. (1970). Treatment of acid mine drainage by foam separation. *Water Pollution Control Research Series* 14010 DEE, December.

BOYER, J.F. and GLEASON, V.E. (1972). Coal and coal mine drainage. *J. Wat. Poll Cont. Fed.,* 44 (6), 1088-1092.

BOWDEN, P.R. and BLEYS, C. (1971). Hydrogeology of the Milang-Langhorne Creek area. *S.A.D.M. Min. Res. Rev. 131,* 84-94.

BRALEY, S.A. (1954). Acid mine drainage; Part 2. *Sources Mechanisations,* February.

BESSELIERRE, E.B. (1969). *The Treatment of Industrial Wastes.* New York: McGraw-Hill, p 100.

CEDERGREN, H.R. (1967). Seepage, Drainage and Flow Netts, John Wiley, p 27.

COLLINS, B.I. (1973). The concentration control of soluble copper in a mine tailings stream. *Geochemica et Cosmochimica Acta,* 37, pp 69-75, Oxford: Pergamon Press.

COUCHE, R.A., and GOLDNEY, L.H. (1959). The design of continuous thickeners for flocculated materials. *Proc. Australas. Inst. Min. Metall.,* 191.

DUGAN, P.R. (1970). Removal of mine water ions by microbial polymers. *Third Symposium on Coal Mine Drainage Research, Mellon Institute.* pp 279-283.

EDWARDS, A.B. (1949). Pyrite deposit at Nairne, South Australia. *Min. Rev. 90, S.A.D.M.* 89-91.

EDWARDS, A.B. and CARLOS, G.C. (1949). The Selenium content of some Australian sulphide deposits. *Proc. Aus. I.M.M.,* 172, 31-67.

GARRELS, R.M. (1967). Genesis of some ground waters from igneous rocks. *Researches in Geochemistry, Vol.2.* New York: Wiley.

HALDANE, A.D. (1971). Zinc pollution in the Molonglo River System. *B.M.R. Rec. 1971/110, Canberra.*

HARRISON, V.F. (1970). Neutralisation and aeration of acid mine waters (A literature survey). *Mines Branch Ottawa, Inf. Cir.* IC.227.

HAWLEY, J.R., and SHIKAZE, K.A. (1971). The problem of acid mine drainage in Ontario, *Can. Min. J.,* 82-93. June.

HILL, R.D., and WILMOTH, R.C. (1971). Limestone treatment of acid mine drainage. *Transactions Soc. of Mining Engineers AIME Vol.* 250, 162-250. June.

HILL, R.D., and WILMOTH, R.C. (1973). Mine drainage pollution control via reverse osmosis. *Min. Eng. Soc. Min. Eng.,* 45-57 March.

HOFFERT, J.R. (1947). Acid mine drainage. *Ind. Eng. Chem.* 39(5), 642-6.

KUMAR, J., and JEDLICKA, J.A. (1973). Selecting and installing synthetic pond-linings, *Chem. Engr,* 80, Feb.5, 67-70.

KUNIN, R., and DOWNING, D.G. (1971). Ion Exchange system boasts more pulling power. *Chemical Engineering,* June, 67-9.

LaGANZA (1959). Pyrite investigations at Nairne, South Australia. *Econ. Geol.* 54, 895-902.

LEOPOLD, L.B., CLARKE, F.E., HANSLOW, B.B., and BALSEY, J.B. (1971). A procedure for evaluating environmental impact. *U.S. Geol. Surv. Inf. Circ.* 645.

LEROY, J.C., and KELLER, H. (1972). How to reclaim mined areas, tailing ponds, and dumps into valuable land. *World Mining,* 34-41.

MASON, D.E. (1970). Treatment of acid mine drainage by reverse osmosis. *Third Sumposium on Coal Mine Drainage Research, Mellon Institute,* 227-40.

MIHOK, E.A. (1969). Mine water research: Catalytic oxidation of ferrous iron in acid mine water by activated carbon. *(Washington) U.S. Dept. of Interior, Bureau of Mines,* RI7337.

NEMEROW, N.L. (1963). *Theories and Practices of Industrial Waste Treatment.* Massachusetts U.S.A: Wesley, 557 pp.

OKO, U.M., and TAYLOR, W.L.W. (1974). Treatment of acid mine wastewater at Falconbridge's Hardy Brown Pillar Project. *CIM Bulletin,* 43-9, April.

O'DRISCOLL, E.P. (1960). The hydrology of the Murray Basin Province, South Australia. *Bull. Geol. Surv. S. Aust.* 35, 148.

PETRACCO, E.F. (1973). Pollution control in the base-metal mining industry: An integrated approach. *Aust. Min.*, June, 50-52.

PUGSLEY, E.G., CHENG, C.Y., UPDEGRAFF, and ROSS, L.W. (1971). Removal of heavy metals from mine drainage in Colorado by precipitation. Water, 1970. *Chemical Engineering Progress Series*, 67(107), AI Chem. 75-89.

RIDGEWAY, J.E. (1949). Gibralter pyrite mine. *Min. Rev. S.A.D.M. 90*, 81-2.

RIVETT, L.S., and OKO, U.M. (1971). Tailings disposal, generation of acidity from pyrrhotite and limestone neutralisation of waste water from Falconbridges' Onaping Mines. *C.I.M. Bull.* 64(712).

ROBERTS, G.T. (1972). Hydrogeology of the Milang-Langhorne Creek area (second report). *S.A.D.M. Min. Res. Rev. 133*, 148-158.

SATO, M. (1960). Oxidation of sulphide ore bodies, parts 1 & 2. *Econ. Geol.* 555, 928-61, 1202-31.

SCHMIDT, J.W., and CONN, K., (1971). Abatement of water pollution in the base-metal mining industry. *Can. Min. J.*, March, 49-52.

SKINNER, B.J. (1958). The geology and metamorphism of the Nairne pyrite formation. *Econ. Geol.*, 53, 546-62.

STANDER, G.J., HENZEN, M.R. and FUNKE, J.W. (1970). The disposal of polluted effluents from mining, metallurgical and metal-finishing industries, their effects on receiving waters and remedial measures. *Journal of the South African Institute of Mining Metallurgy*, 71(5), Dec., 104.

TEMMEL, F.M. (1971). Treatment of acid and metal-bearing wastewaters by the high-density sludge process. *Amer. Iron & Steel Inst. Reg. Tech. Meet.*, 1-15.

WATSON, A.E.P. (1969). Pollution problem associated with base metal mines. *Can. Min. J.*, June, 50-61.

WILMOTH, R.C., and HILL, R.D. (1970). Neutralisation of high ferric iron acid mine drainage. *Dept. Int. Fed. Water Qual. Admin. Prog.* 14010 ETV. August, 1970.

YAGAMATER, N., and SHIGEMATSU, I. (1970). *Bull. Inst. Public Health*, 19(1), 1-27.

YOUNG, C.A. (1969). The use of vegetation to stabilise mine tailings area at Copper Cliff. *Canadian Mining Journal*. June, 43-46.

TABLE 1: TRACE ELEMENT TOLERANCES FOR IRRIGATION WATERS*

Element	For Water Used Continuously on all Soils mg/*l*	For Short-term Use on Fine Textured Soils Only mg/*l*
Aluminium, Al	1.0	20.0
Arsenic, As	1.0	10.0
Beryllium, Be	0.5	1.0
Boron, B	0.75	2.0
Cadmium, Cd	0.005	0.05
Chromium, Cr	5.0	20.0
Cobalt, Co	0.2	10.0
Copper, Cu	0.2	5.0
Lead, Pb	5.0	20.0
Lithium, Li	5.0	5.0
Manganese, Mn	2.0	20.0
Molybdenum, Mo	0.005	0.05
Nickel, Ni	0.5	2.0
Selenium, Se	0.05	0.05
Vanadium, V	10.0	10.0
Zinc, Zn	5.0	10.0

* Reference - Compilation of Australian Water Quality Criteria Australian Water Resources Council, April 1974 extracted from US Dept. of Interior. Anon. (1968).

TABLE 2: DERIVED WORKING LEVELS FOR IONS IN LIVESTOCK DRINKING WATER SUPPLIES*

Element	Derived Working Level** mg/*l*	Remarks
Arsenic, As	1.0	
Cadmium, Cd	0.01	
Calcium, Ca	1000	
Iron, Fe	10	Problems experienced with animals feeding on pastures irrigated with high Fe water.
Lead, Pb	0.5	
Magnesium, Mg	250-500	
Molybdenum, Mo	0.005	May affect animal through accumulation in pasture.
Nitrate, NO_3	90-200	
Selenium, Se	0.05	May affect animals through accumulation in plants.
Sulphate	1000	

* Reference: Compilation of Australian Water Quality Criteria, Australian Water Resources Council April 1974.

** Derived Working Level is defined as the maximum level of pollutant designed to ensure that under a specific circumstance a primary protection level is not exceeded.

TABLE 3: SCAN ANALYSIS 10 APRIL 1973

Water Sample	Northern Quarry Seepage QSE	Tailing Dam Storage	Tailing Dam Pond 1 (Recycle)	Tailing Dam Pond 2 (static)	Waste* Dump Seepage	Dawesley* Creek out of Mine
Analysis mg/l except for Hg						
SO_4	17 200	6 900	6 620	6 900	13 700	2 800
Acid Value (as $CaCO_3$)**	15 000	6 000	5 000	6 500	12 300	1 920
Fe	5 300	1 300	2 100	1 300	68	210
Cu	0.1	0.50	0.15	0.70	8.6	0.6
Zn	140	60	13	40	155	25
Co	0.7	0.8	0.3	0.8	8.0	1.0
Ni	2.1	3.2	0.3	1.4	12.0	1.5
Mn	240	215	155	200	38	19
Pb	<0.5	<0.5	<0.5	<0.5	-	-
V	<0.5	<0.5	<0.5	<0.5	< 0.25	<0.25
Cd	0.06	0.14	<0.01	<0.05	2.0	0.11
Ti	<0.3	<0.3	<0.3	<0.3	< 0.5	<0.5
Li	0.2	<0.1	<0.1	<0.1	0.4	0.15
Na	420	285	280	170	145	350
K	13	3	17	3	3	2
Ca	355	540	380	490	180	100
Al	695	220	55	230	2000	190
Mg	540	360	230	390	400	130
As	0.01	0.01	<0.01	0.02	<0.01	<0.01
P	0.33	0.18	0.69	0.21	0.04	<0.01
Sb	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Ag	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
U	0.18	0.08	0.01	0.07	0.21	0.025
COD (as O_2)***	<50	80	<50	100	130	125
Hg µg/l	<0.1	0.1	0.4	<0.1	<0.1	<0.1
Sn	<0.4	<0.4	<0.4	<0.4	<0.1	<0.1
Cr	<0.1	<0.1	<0.1	<0.1	0.5	<0.1
Mo	0.1	<0.1	<0.1	0.1	0.1	<0.1
Ba	0.1	0.1	<0.1	0.1	<0.1	<0.1

* Sample collected 7 June 1973.

** Acid Value determined by hot titration with NaOH to pH 8.3 and includes acid salts acidity in addition to free mineral acidity.

*** COD - Chemical Oxygen Demand by dichromate digestion.

TABLE 4: BREMER RIVER SYSTEM WATER ANALYSIS WINTER FLOWS 20 SEPTEMBER 1973

Analysis mg/*l*	1*	2	3	4	5	6
Ca**	-	56	-	74	72	60
Mg**	-	151	-	57	58	47
Na/K**	-	522	-	338	285	333
HCO_3**	-	Nil	-	134	146	146
SO_4**	-	1170	-	259	86	182
Cl**	-	480	-	550	590	530
Fe	1	40	3	<0.3	<0.3	-
Al	<1	115	35	<1	<1	-
Mn	<0.1	12	6	1.4	0.9	-
Zn	0.1	8.5	5.8	0.5	0.4	-
Ni	<0.1	0.8	0.3	<0.1	<0.1	-
Co	<0.1	0.5	0.3	<0.1	<0.1	-
Cu	<0.1	0.3	0.2	<0.1	<0.1	-
Cd	<0.005	0.11	0.035	0.005	0.01	-
Acid Value (as $CaCO_3$)	-	1070	400	-	-	-
TDS**	-	2380	-	1350	1160	1230
pH	7.9	3.3	4.3	7.35	7.65	-

* Localities shown in Fig. 1:
1: Dawesley Creek upstream from mine.
2: Dawesley Creek 3 km downstream from mine.
3: Dawesley Creek below junction with Nairne Creek.
4: Mt Barker Creek before junction with Bremer Rivers.
5: Bremer River after junction with Mt Barker Creek.
6: Bremer River at Langhorne Creek.

** Data from E & WS Bremer River Quality Survey Progress Report 18/2/74 for water samples collected 27/9/73.

TABLE 5: BREMER RIVER SYSTEM WATER ANALYSIS SUMMER FLOW
26 NOVEMBER 1973*

Analysis mg/*l*	2**	4	5	6***
Ca	112	88	88	-
Mg	190	78	73	-
Na/K	794	561	573	-
HCO_3	Nil	287	268	-
SO_4	1860	230	211	-
Cl	600	910	930	-
Fe	80	0.4	0.44	-
Al	290	1.56	0.10	-
Mn	24.5	0.11	0.05	-
Zn	32.1	0.20	0.21	-
Ni	1.74	0.05	0.05	-
Cu	1.10	0.02	0.03	-
Cd	0.142	0.006	0.006	-
TDS	3560	2010	2166	-
NO_3	<0.1	<0.1	<0.1	-
PO_4	0.02	0.02	<0.01	-
pH	2.7	7.9	8.1	-

* From E & WS Bremer River Quality Survey Progress Report 18/2/74.

** Localities shown in Fig. 1.
2: Dawesley Creek 3 km downstream from mine.
4: Mt Barker Creek before junction with Bremer River.
5: Bremer River after junction with Mt Barker Creek.
6: Bremer River at Langhornes Creek.

***No Flow.

TABLE 6: BREMER RIVER SYSTEM WATER ANALYSIS SUMMER FLOW
4 APRIL 1974

Analysis mg/l	1*	2	3	4	5	6**
Ca	67	-	-	140	115	-
Mg	61	-	-	150	130	-
Na	385	-	-	475	675	-
K	14	-	-	19	24	-
HCO_3	205	-	-	Nil	Nil	-
SO_4	96	-	-	1780	960	-
Cl	710	-	-	750	1100	-
Fe	1.0	180	14.5	0.7	0.3	0.4
Al	0.16	630	105	140	32	33
Mn	0.17	42.0	12.5	21.0	12.0	9.0
Zn	0.03	73.5	12.0	19.1	9.1	8.1
Ni	0.04	3.55	0.74	1.0	0.46	0.36
Co	<0.05	1.45	0.40	0.60	0.30	0.24
Cu	<0.02	0.95	0.19	0.26	0.12	0.10
Cd	0.02	0.30	0.06	0.08	0.05	0.055
Acid Value (as $CaCO_3$)	-	4650	950	1120	350	320
TDS	1440	-	-	3495	3060	-
pH	6.9	2.7	3.0	4.0	4.3	4.2

* Localities shown in Fig.1.
1: Dawesley Creek upstream from mine.
2: Dawesley Creek 3 km downstream from mine.
3: Dawesley Creek below junction with Nairne Creek.
4: Mt Barker Creek before junction with Bremer River.
5: Bremer River after junction with Mt Barker Creek.
6: Bremer River at Langhornes Creek.

** Sampled 6 February 1974.

TABLE 7: PRECIPITATION OF HEAVY METALS IN BREMER RIVER SYSTEM 20 SEPTEMBER 1973

Analysis	1*			2			3		
	Solution mg/l	Sediment as mg/l	Total mg/l	Solution mg/l	Sediment as mg/l	Total mg/l	Solution mg/l	Sediment as mg/l	Total mg/l
Fe	35	9.5	43	3	2.85	5.5	<0.3	0.7	0.6
Al	96	0.6	96.5	35	2.3	37	<1	9.3	7.9
Mn	11	0.07	11	6	<0.01	6	1.9	0.04	1.95
Ni	0.5	<0.01	0.5	0.3	<0.01	0.3	<0.1	<0.01	<0.1
Cu	0.3	0.03	0.33	0.2	0.02	0.22	<0.1	0.04	<0.15
Co	0.3	<0.01	0.3	0.3	<0.01	0.3	<0.1	<0.01	<0.1
Zn	12.5	0.06	12.6	5.8	0.02	5.8	1.2	0.24	1.4
Cd	0.09	0.001	0.09	0.035	0.001	0.035	0.015	0.0015	0.0016

* 1: Dawesley Creek leaving mine.
2: Dawesley Creek after junction with Nairne Creek.
3: Mt Barker Creek after junction with Dawesley Creek.

TABLE 8: X-RAY DIFFRACTION OF PRECIPITATES IN DAWESLEY CREEK

ample No.*	
1 and 2	X-ray diffraction indicates that the sample is mostly amorphous with gypsum, natrojarosite and quartz present.. The analysis suggests that the sample consists mainly of ferric hydroxide with natrojarosite, halotrichite, and a little quartz and gypsum.
3	X-ray diffraction and chemical analysis suggest that this is similar to 1 and 2 but with a little albite.
4	X-ray diffraction identified natrojarosite and gypsum. The chemical analysis suggests that there may be ferric hydroxide, possibly a ferric sulphate, natrojarosite gypsum, ferropickeringite, jarosite and quartz.
5	X-ray diffraction identified a member of the halotrichite-ferropickeringite series. The analysis suggests the presence also of quartz (probably some colloidal silica) and natrojarosite.
6	X-ray diffraction and chemical analysis indicate that the sample consists mainly of fine grained elastic minerals (quartz, albite and mica) with natrojarosite and possibly halotrichite.

Sample locations for Nos 1, 2, 3, 4 and 5 shown in Fig.2. No.6 sample, sludge in creek at Dawesley 7/6/73.

TABLE 9: CHEMICAL ANALYSES OF PRECIPITATES IN DAWESLEY AND MT BARKER CREEK

	Sample No.*														
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
MgO	1.04	0.83	0.80	0.15	2.89	0.58	3.88	4.69	4.35	2.64	0.85	-	-	-	-
FeO	-	-	-	-	-	-	1.44	-	-	3.95	6.86	-	-	-	-
Fe_2O_3	28.74	34.89	26.88	34.03	1.23	8.65	-	-	-	0.68	1.12	51.10	50.98	29.5	6.9
Al_2O_3	6.61	3.27	8.50	0.30	9.45	6.33	19.7	11.87	12.30	10.99	10.54	-	-	15.6	21.4
MnO	0.12	0.12	0.17	0.26	0.37	0.07	1.68	-	-	0.24	0.47	-	-	0.04	0.03
SO_3	9.27	13.86	7.68	26.39	25.23	3.82	28.7	37.29	37.84	35.61	36.42	28.57	30.96	14.0	11.5
H_2O	-	-	-	-	-	-	32.5	46.15	44.66	46.18	43.63	10.56	11.15	-	-
K_2O	0.40	0.32	0.70	0.34	0.32	1.27	-	-	-	-	-	7.13	0.35	-	-
Na_2O	2.08	0.30	1.21	1.21	1.89	1.22	-	-	-	-	-	0.84	6.03	-	-
ZnO	0.15	0.15	0.09	0,02	0.50	0.04	0.36	-	-	-	-	-	-	0.02	0.02
SiO_2	4.00	3.95	11.0	3.6	24.2	62.9	10	-	-	-	-	-	-	-	-
CaO	0.70	0.60	0.60	8.3	0.60	0.60	0.16	-	-	-	-	2.40	0.47	-	-

* 1.)
2.)
3.) Brown sludges, Dawesley Creek, mine area 7 June 73; see Fig.2.
4.)
5. Ferropickeringite crust on rocks, Dawesley Creek, Mine Area 7 June 73; see Fig.2.
6. Sludge, Dawesley Creek at Dawesley 7 June 73.
7. White encrustation, probably 2 $MgSO_4.MSO_4.2Al_2(SO_4)(OH)_4$ 7 H_2O mine area 10/4/73; see Fig.2.
8. Mg $Al_2(SO_4)_4.22H_2O$, theoretical.
9. Pickeringite ($MgAl_2$ $(SO_4)_4.22H_2O$) Dana (1966) p 525.
10. Ferropickeringite ((Fe, Mg) $Al_2(SO_4)_4.22H_2O$) Dana (1966) p 525.
11. Halotrichite ($Fe^{2+}Al_2(SO_4)_4.22H_2O$), Dana (1966) p 525
12. Jarosite (K $Fe_3(SO_4)_2(OH)_6$), Dana (1966) p 561.
13. Natrojarosite, (Na $Fe_3(SO_4)_2$ $(OH)_6$) Dana (1966) p 563.
14. Precipitate in Dawesley Creek after junction with Nairne Creek 20 September 1973.
15. Precipitate in Mt Barker Creek after junction with Dawesley Creek 20 September 1973.

TABLE 10: WATER COMPOSITIONS: BRUKUNGA AREA

	Dawesley Creek		T. Dam	Quarry	W. Dumps
	1*	2	3	4	5
Total annual flow m^3	820 000		100 000	58 200	2000
Fe mg/l	1	210	2150	4025	68
Al mg/l	<2	190	95	1050	2000
Mn mg/l	<0.1	19	120	133	38
Zn mg/l	<0.2	25	16	186	155
Ni mg/l	<0.2	1.5	0.6	7.5	12.0
Cd mg/l	<0.2	0.11	0.05	0.68	2.0
Cu mg/l	<0.1	0.60	0.2	2.1	8.6
Co mg/l	<0.2	1.0	0.5	3.0	8.0
Ca mg/l	38	100	435	330	180
Mg mg/l	28	130	230	437	400
Na mg/l	395	350	245	100	145
K mg/l	19	2	12	2	3
SO_4 mg/l	100	2800	7060	16 125	13 700
Cl mg/l	395	355	325	190	155
TDS mg/l	1250	4180	10 700	22 600	16 800
Acid Value (as $CaCO_3$) mg/l	-	1920	5050	13 650	12 300
Ratio Mn/Ni	-	13	200	17	3
pH	7.6	2.8	2.5	2.9	3.2

* 1: Dawesley Creek entering area 7 June 1973.

2: Dawesley Creek leaving area 7 June 1973.

3: Tailings Dam, lower holding pond, 7 June 1973.

4: Quarry seepages average of QSC and QSE 7 June 1973. Individual flows on Fig. 11.

5: Waste dump (north) seepage, 7 June 1973.

TABLE 11: ALKALI SUPPLIERS, COST AND COMPOSITION

Alkali	Nominal Sizing	Supplier	Cost fob $/tonne	Estimated Cost at Brukunga $/tonne	%			
					$Ca(OH)_2$	$CaCO_3$	H_2O	Total
Lime (hydrated lime)	- 10μ	Hydrated Lime Ltd Ridleyton	27.5	30	86	7	Nil	93
Lime (hydrated lime)	- 10μ	Adelaide Brighton Cement* Birkenhead	30.5	40.55				
Lime (quick lime)		Adelaide Brighton Cement* Birkenhead	29.10	35.80				
Caustic Mud	- 10μ	ICI Aust. through Pearce Transport Pty Ltd Pt. Adelaide	Nil	3.9	1.9	68.6	29.5	100
Milk of Lime (MOL) Grit	-500μ	ICI Aust. through Pearce Transport Pty Ltd Pt. Adelaide	Nil	3.3	26.9	57.6	11.3	96
Milk of Lime (MOL) Grit	-500μ	Hydrated Lime Ltd Ridleyton	2.0	4.7	21.0	76.0	Nil	97
Marble grit	-710μ	ICI Aust. Penrice	1.20	4.7	Nil	97.5	Neg	97.5
Agricultural Limestone								
- crushed	- 50mm	Cawte Industries Murray Bridge	2.45	5.1	-	-	-	-
- grit	-710μ	Cawte Industries Murray Bridge	7.9	10.6	Nil	81.5	Neg	81.5
- fine	- 72μ	Cawte Industries Murray Bridge	11.9	14.6	Nil	76.5	Neg	76.5

* Provided by M. Pascoe, October 1974.

TABLE 12: EFFECT OF ALKALI TYPE ON REMOVAL OF HEAVY METALS AND SULPHATE
Acid Value of AMD 4800 mg/*l* (as $CaCO_3$)
Iron 100% Fe^{3+}

	pH	Analysis mg/*l*				
		Fe	Zn	Mn	Ni	SO_4
	(Theoretical required 6.7 g/*l*)					
'Caustic Mud' added, g/*l*						
0	2.4	1400	39	165	3.0	7000
2	2.55	700	36	160	3.0	-
4	2.7	300	36	160	2.5	-
8	5.0	<0.1	31	155	2.4	2900
12	5.8	<0.1	3.5	145	2.0	-
16	6.5	<0.1	1.0	115	1.2	2900
	(Theoretical requirement 5.1 g/*l*)					
'Milk of Lime Grit' added, g/*l*						
0	2.4	1400	39	165	3.0	7000
4	2.65	650	34	155	2.5	-
8	2.80	220	32	147	2.4	-
16	4.3	<0.1	25	123	2.0	-
24	10.7	<0.1	<0.05	<0.05	<0.2	1500
	(Theoretical requirement 3.8 g/*l*)					
Hydrated Lime added g/*l*						
0	2.4	1400	39	165	3	7000
2	3.5	30	34	160	40	3300
4	6.5	<0.1	6	3	0.2	-
5	10.2	<0.1	2	<0.05	<0.2	1800

TABLE 13: ESTIMATED MEAN ALKALI REQUIREMENTS AND COST
Estimated Mean Flow 18 m^3/h
Estimated Mean Acid Value 5.0 g/*l* as $CaCO_3$

	Requirements tonnes/year		Cost $/a
	Basic Alkali*	pH adjustment**	
Lime (Hydrated Lime)	700	80	23 000
'Caustic Mud'/lime	1600	110	9 500
'Milk of lime' grit	1700	360	8 600
Marble grit/lime	1600	110	12 700
Lime (Adelaide Brighton Cement - hydrated)	700	80	32 000
Lime (Adelaide Brighton Cement - quicklime)	460	40	18 000

* To pH 5.

** To pH 8.5.

TABLE 14: RELATIVE COST OF SCHEMES*

Scheme	Cost $			Effectiveness
	Preventive	Treatment	Total	
Existing	-	-	-	Water quality poor all the year.
Diversion	175 000	-	175 000	Water quality good except during discharge.
Prevention	620 000	-	620 000	Water quality good all the year.
Evaporation	175 000	380 000	555 000	Water quality good all the year.
Neutralisation	175 000	700 000	875 000	Water quality good all the year.

* Capital or investment cost for treatment only. Does not include revegetation costs nor caretaking and operating charges approximately $2000/a.

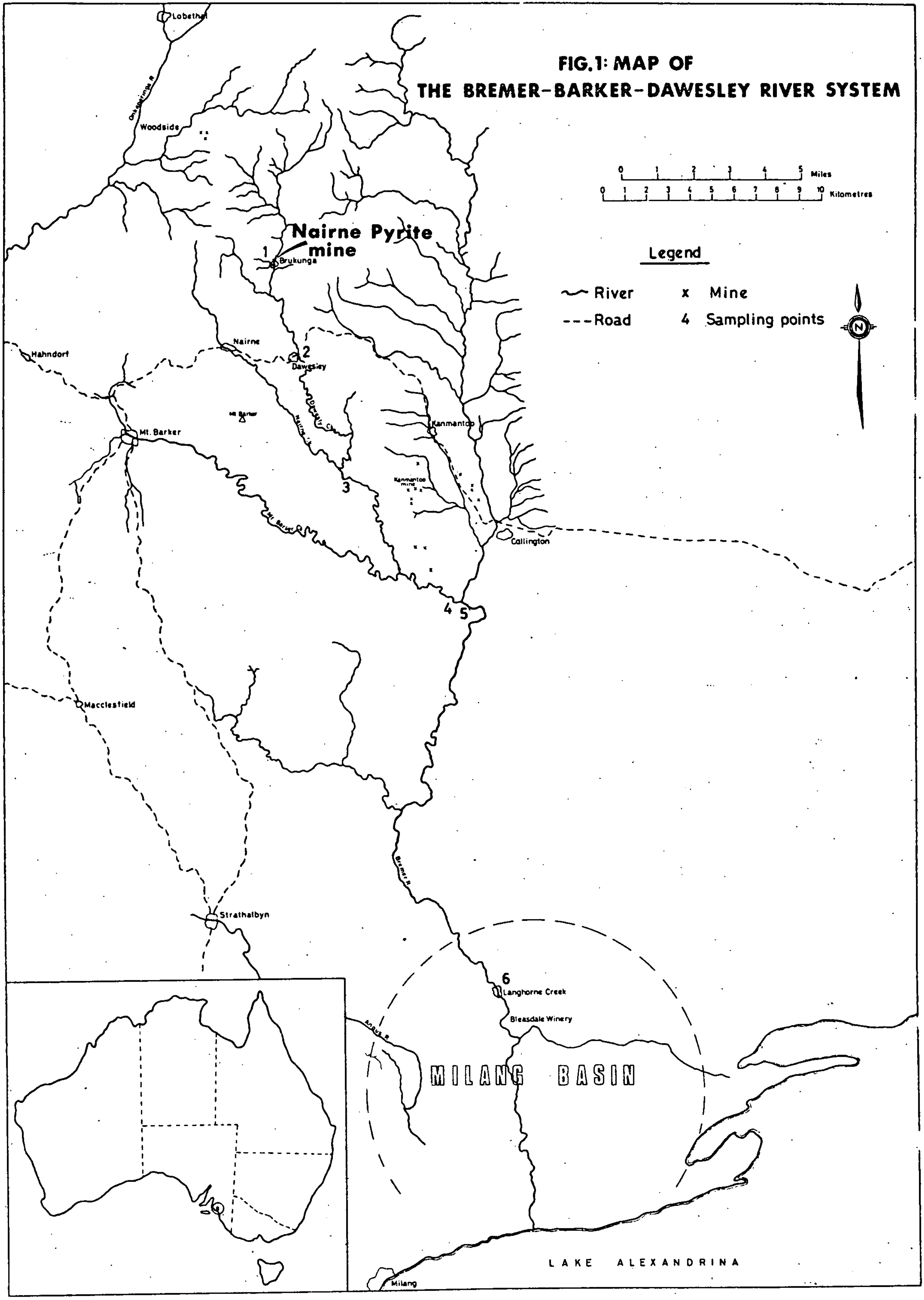
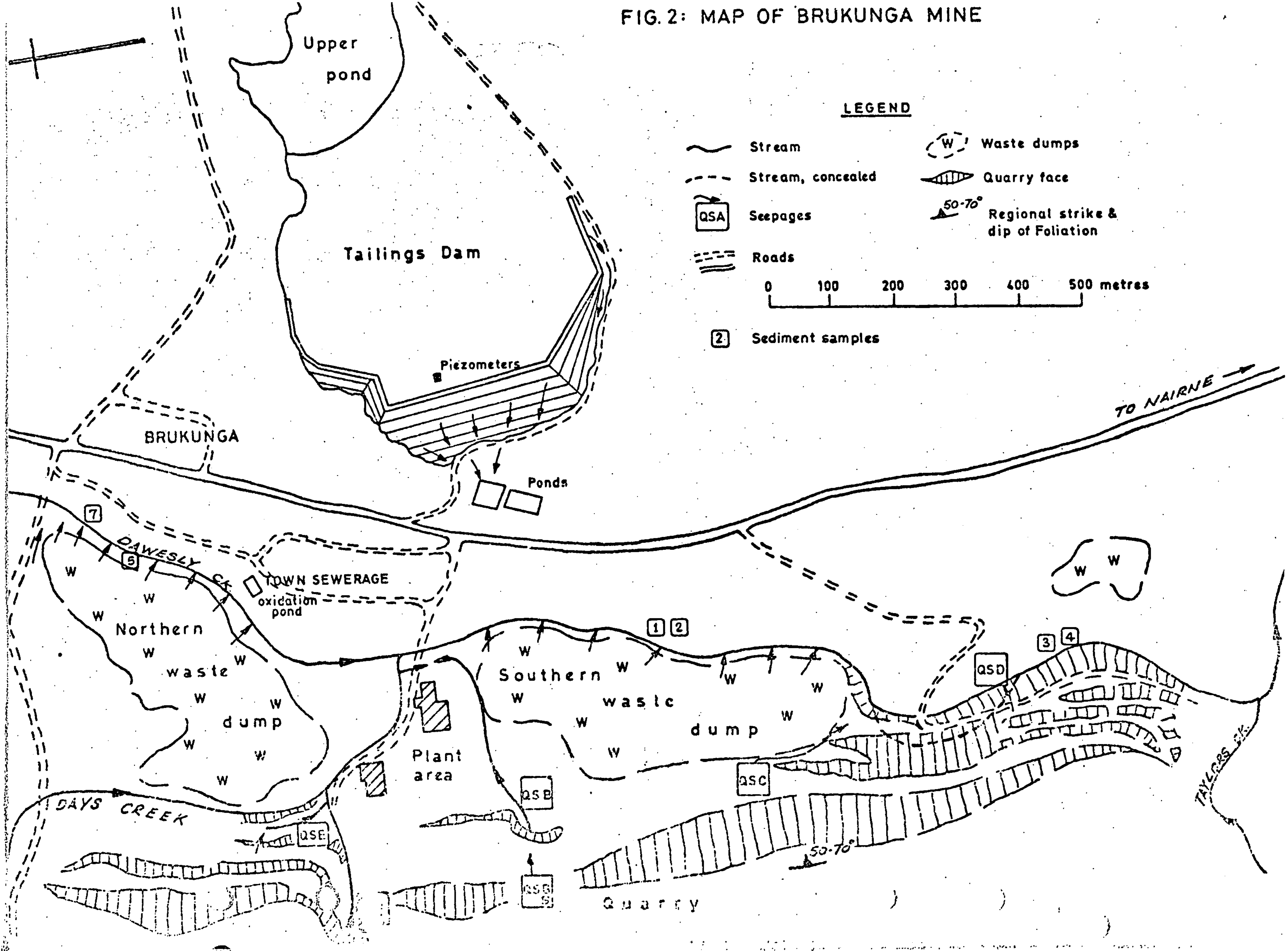


FIG.1: MAP OF THE BREMER-BARKER-DAWESLEY RIVER SYSTEM

FIG. 2: MAP OF BRUKUNGA MINE



M/I

ENVIRONMENTAL CHARACTERISTICS / ACTIVITIES	Alteration of ground, surface hydrology	Open cut excavation	Tailing dam construction	Waste dump emplacement	Dwellings (Brukunga)	Domestic & Lobethal wastes	Industrial plant	Pipelines
River water quality	2/2	1/2	2/2	1/2	1/1	1/1		2/2
Sludge-free rivers	2/2	1/2	2/2	1/2	1/1	1/1		2/2
Soils		1/1	2/1					
Dump & dam stability			1/1					2/2
Stability aquifer, Milang Basin	1/1	1/1	1/1	1/1				1/1
Fish & shellfish	1/1	1/1	1/1	1/1				1/1
Trees, shrubs along river	1/1	1/1	1/1	1/1				1/1
Animals, domestic & wild	1/1	1/1	1/1	1/1				1/1
Aquatic flora								
Grazing	1/1	1/1	1/1	1/1				1/1
Agriculture, irrigation	1/1	1/1	1/1	1/1				1/1
Swimming, recreation	1/1	1/1	1/1	1/1				
Open space qualities	1/1	1/1	1/1	1/1		1/1	1/1	1/1
Health & safety		1/1	1/1	1/1		1/1		

FIG. 3: ENVIRONMENTAL IMPACT MATRIX

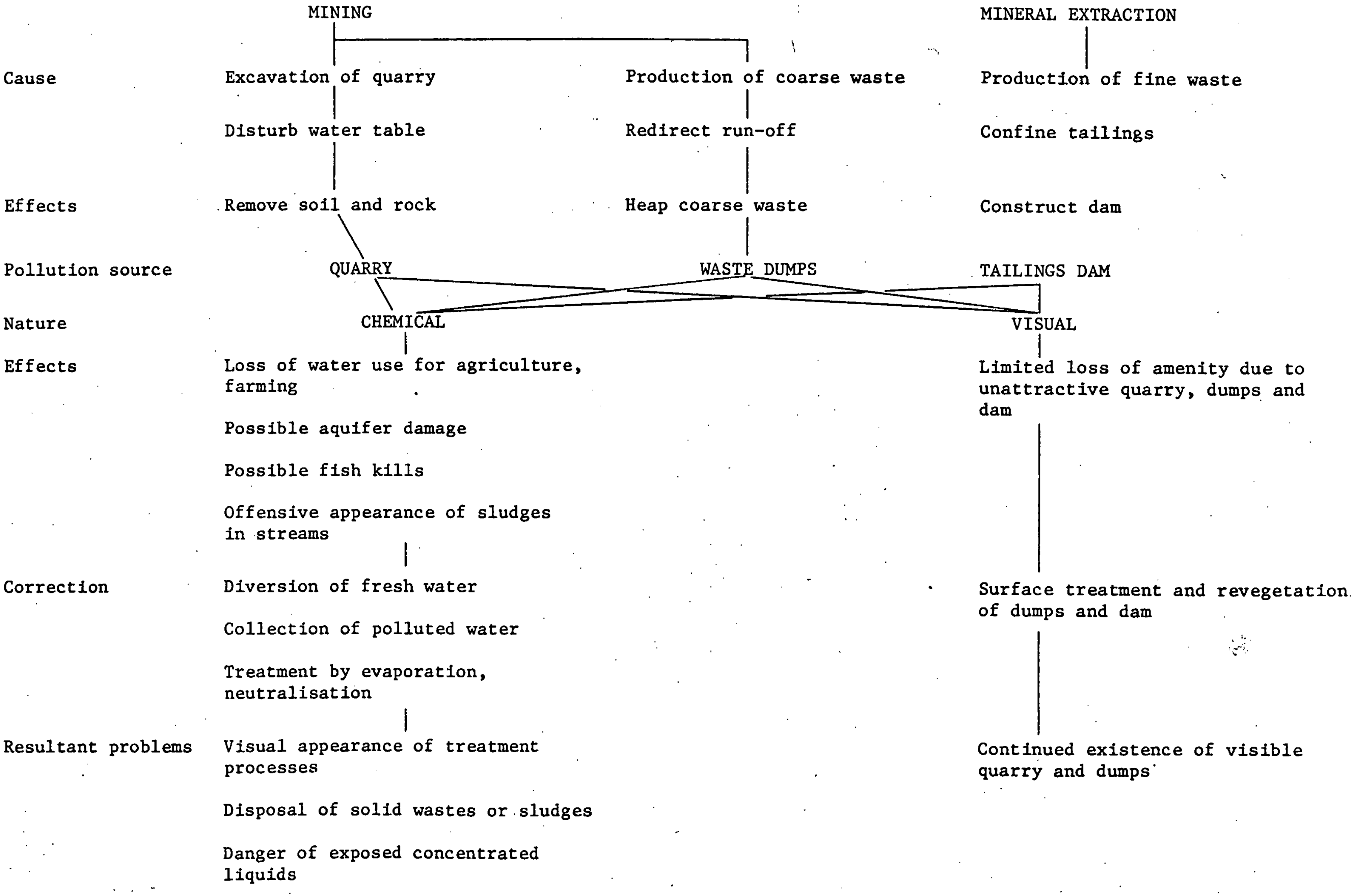


FIG. 4: CAUSE-EFFECT-CORRECTION MATRIX

	MINING		MINERAL EXTRACTION
Cause	Excavation of quarry	Production of coarse waste	Production of fine waste
	Disturb water table	Redirect run-off	Confine tailings
Effects	Remove soil and rock	Heap coarse waste	Construct dam
Pollution source	QUARRY	WASTE DUMPS	TAILINGS DAM
Nature	CHEMICAL		VISUAL
Effects	Loss of water use for agriculture, farming Possible aquifer damage Possible fish kills Offensive appearance of sludges in streams		Limited loss of amenity due to unattractive quarry, dumps and dam
Correction	Diversion of fresh water Collection of polluted water Treatment by evaporation, neutralisation		Surface treatment and revegetation of dumps and dam
Resultant problems	Visual appearance of treatment processes Disposal of solid wastes or sludges Danger of exposed concentrated liquids		Continued existence of visible quarry and dumps

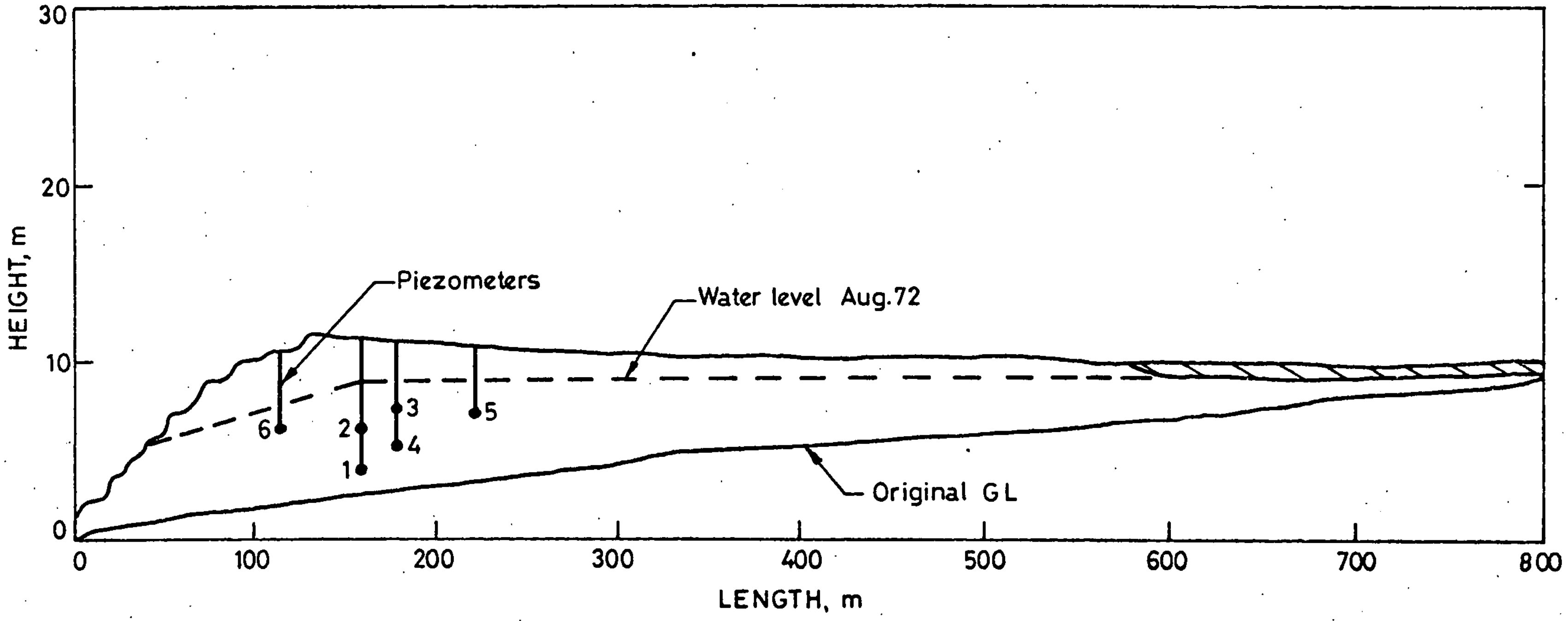


FIG. 5: CROSS-SECTION OF TAILINGS DAM

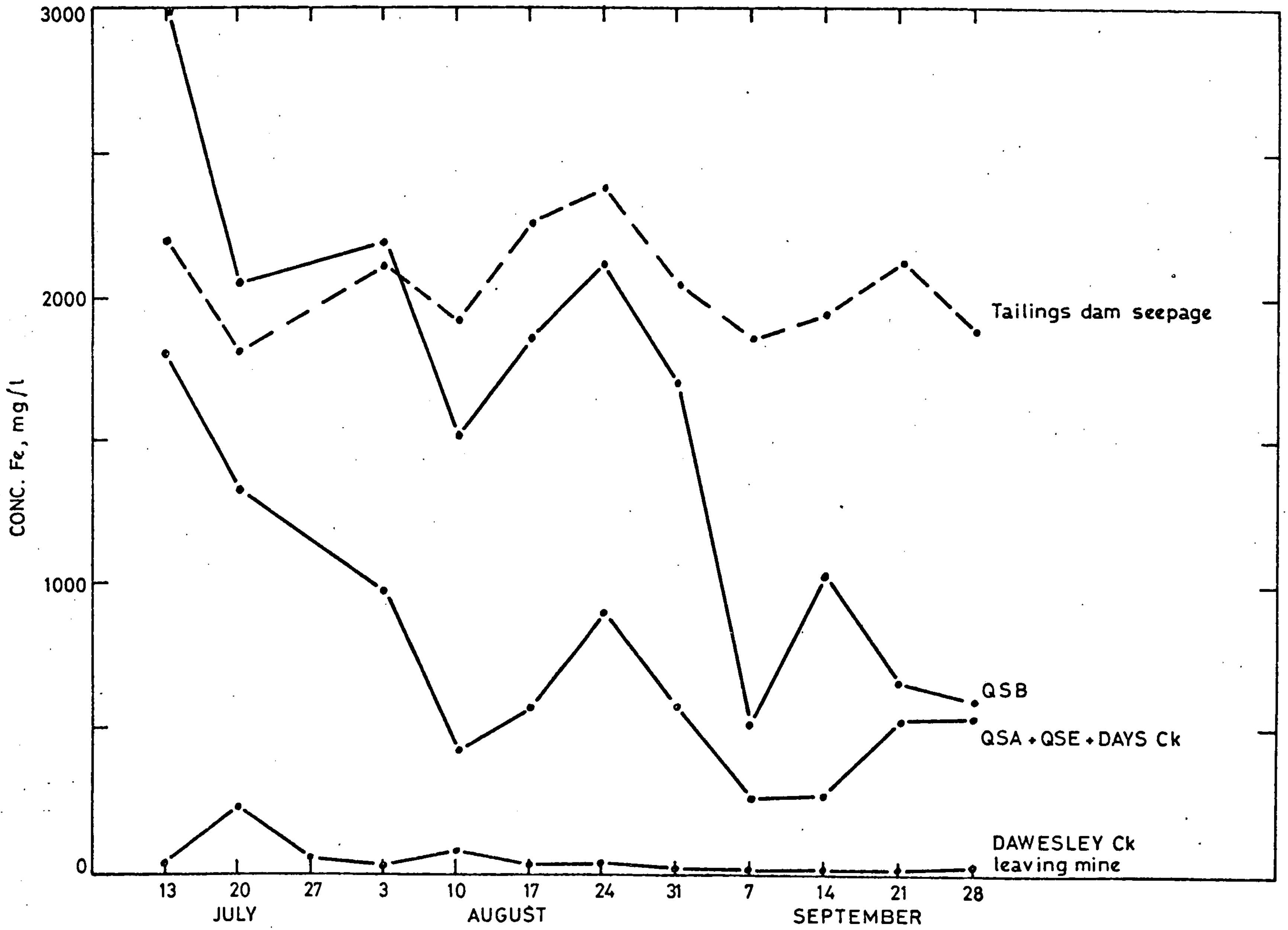


FIG. 6: IRON VALUES OF WATER IN MINE AREA

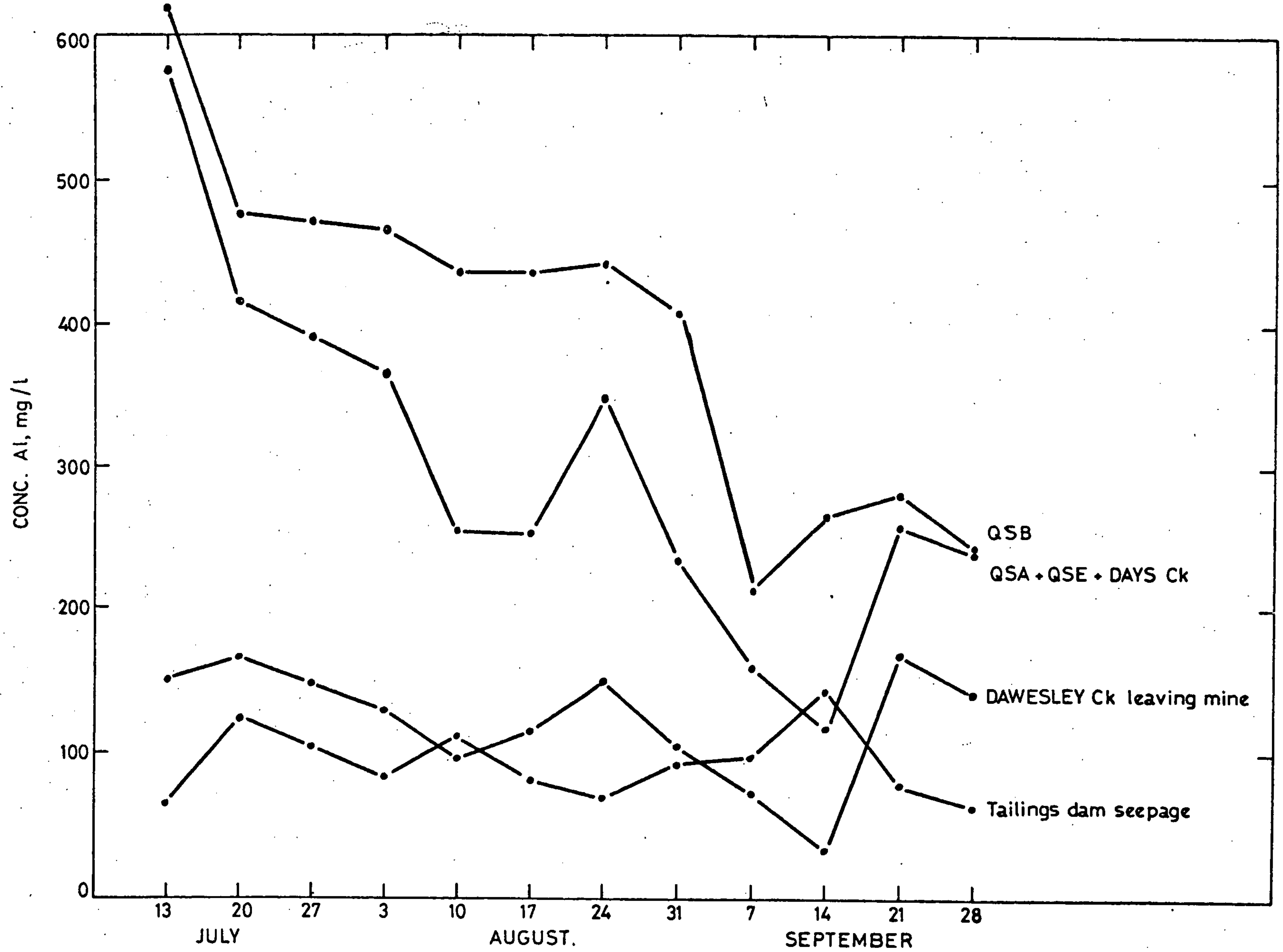


FIG. 7 : ALUMINIUM VALUES OF WATER IN MINE AREA

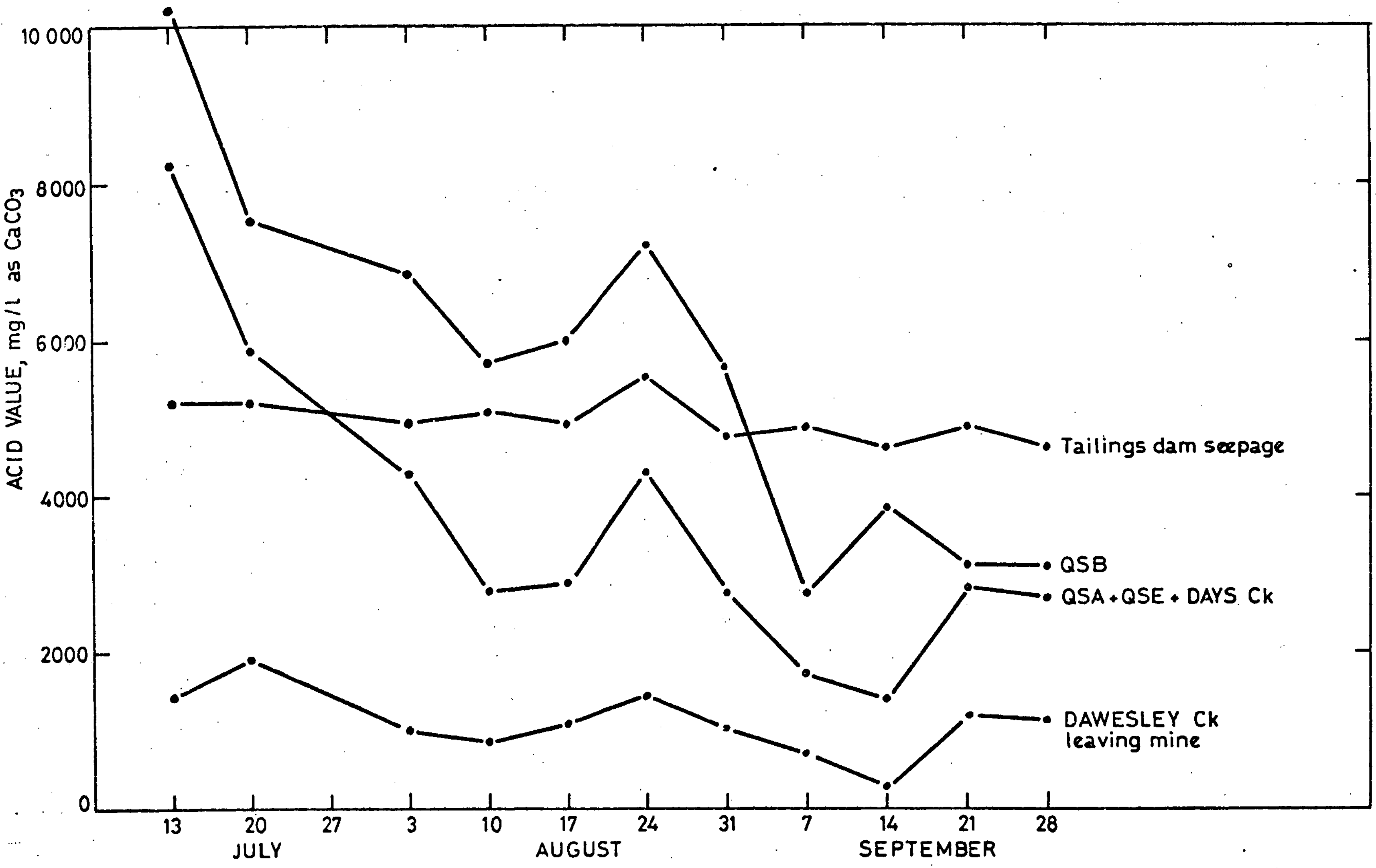


FIG. 8: ACID VALUES OF WATER IN MINE AREA

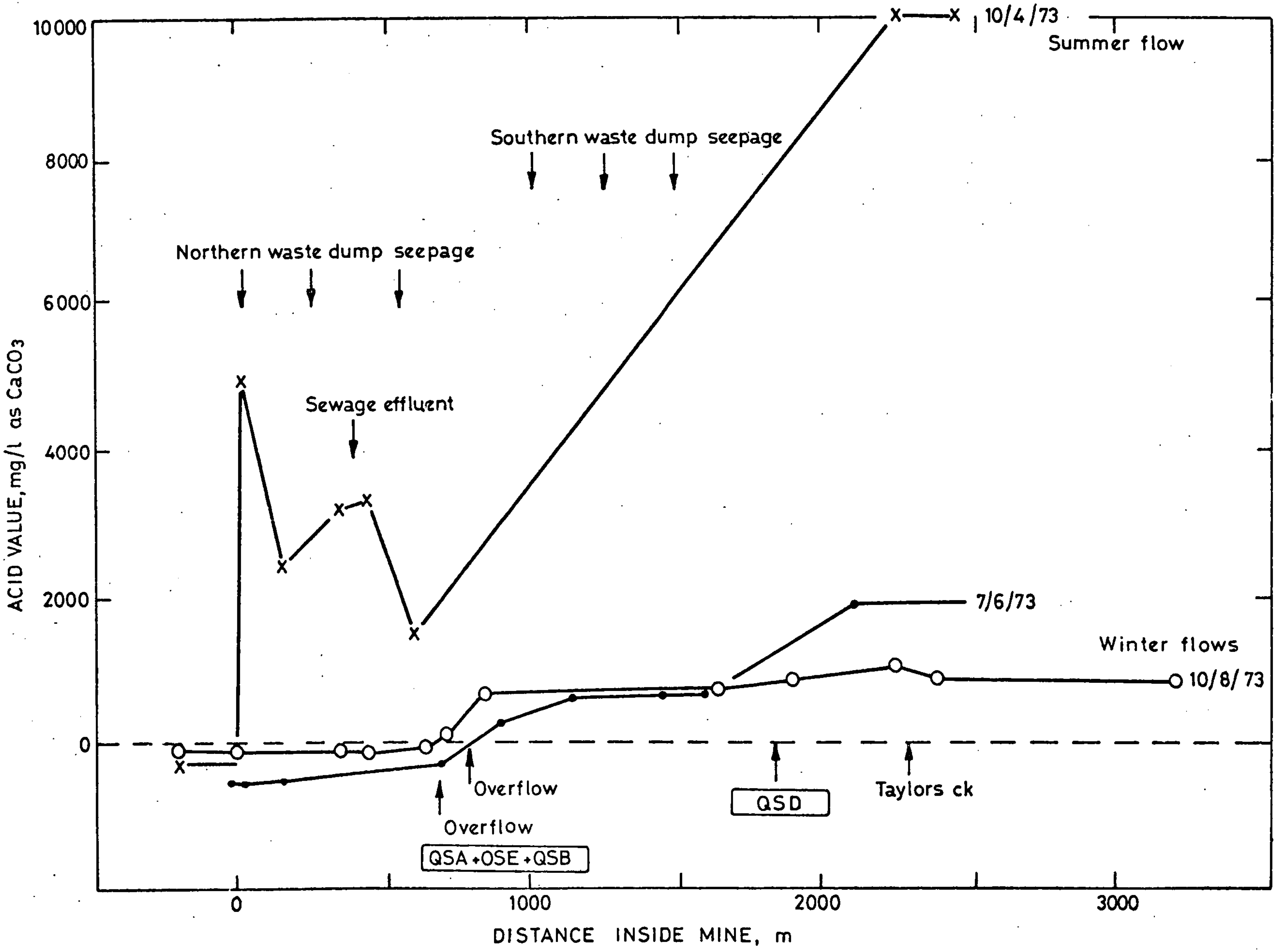


FIG. 9 ACID VALUES ALONG DAWESLEY CREEK IN MINE

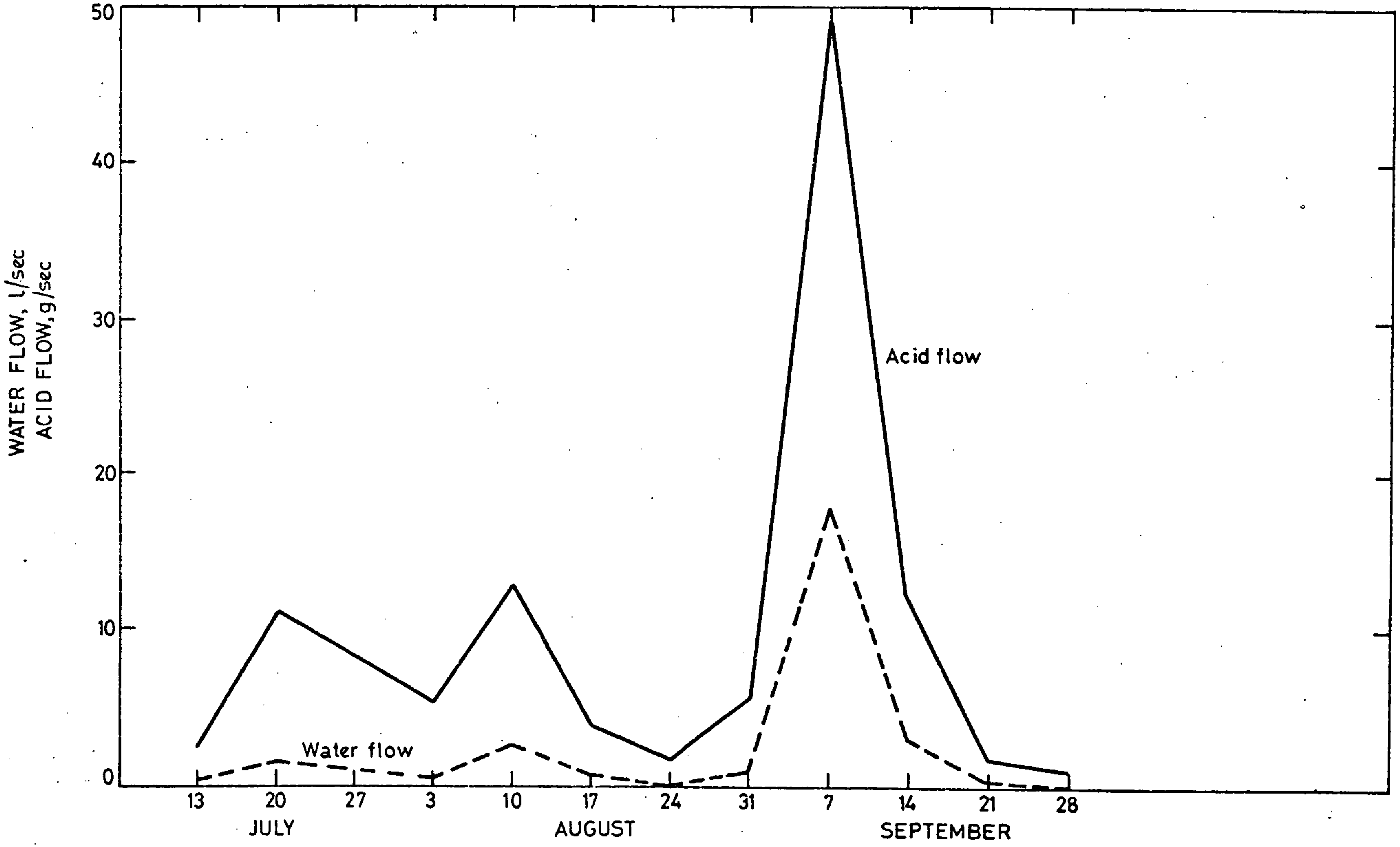


FIG. 10: ACID FLOW RELATED TO STREAM FLOW OF QSB

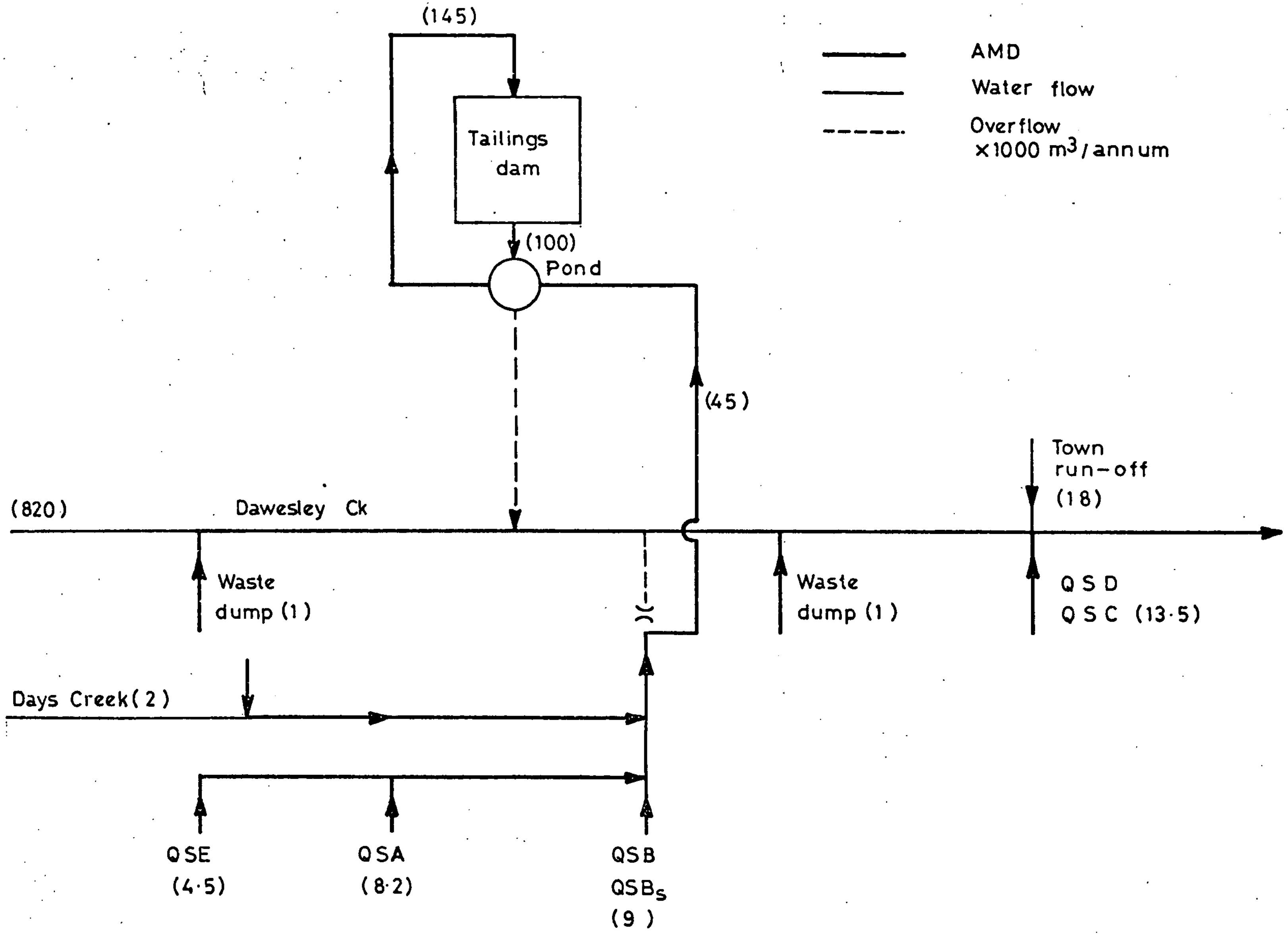


FIG. 11: EXISTING PROCEDURE

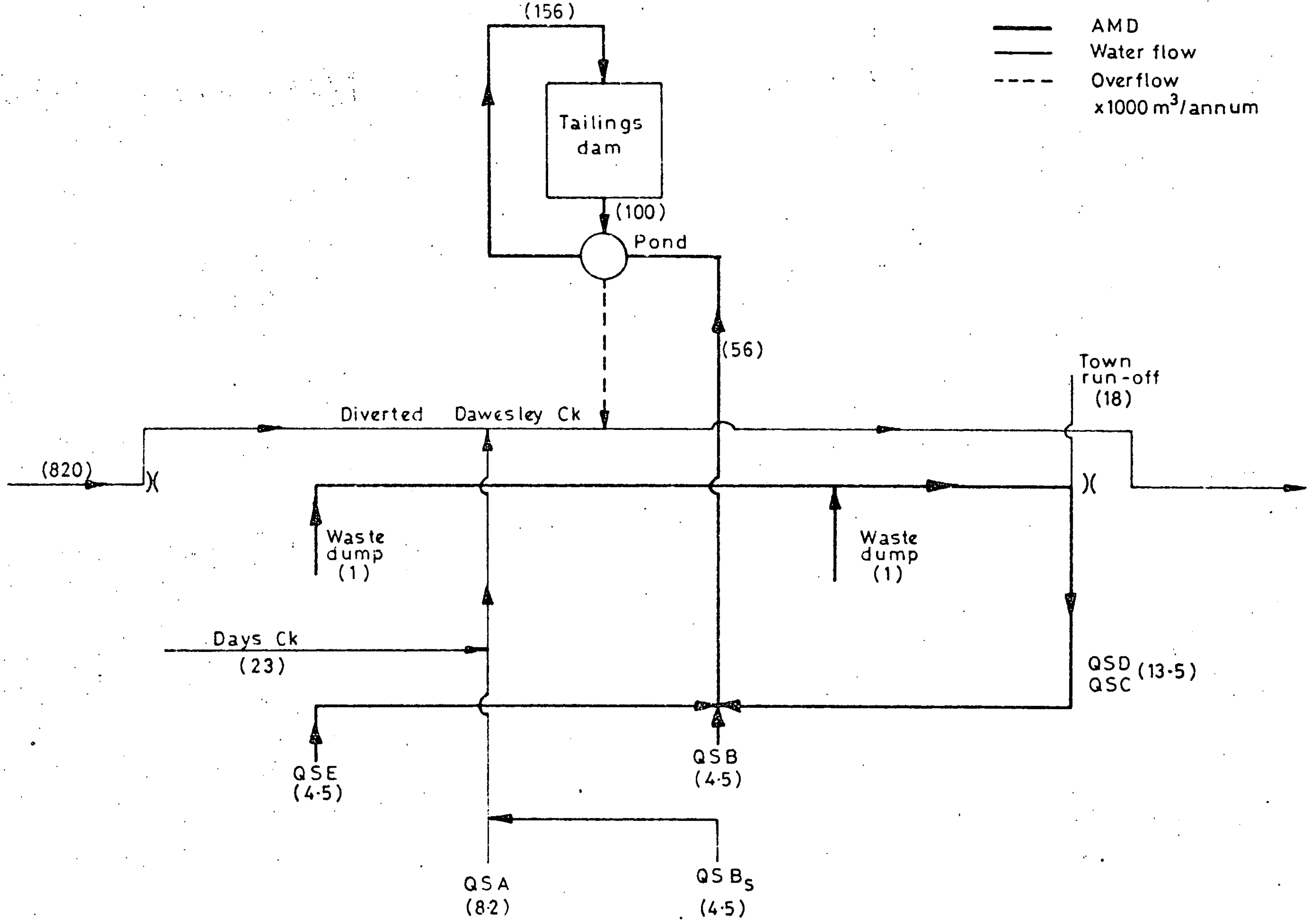


FIG. 12 : DIVERSION SCHEME

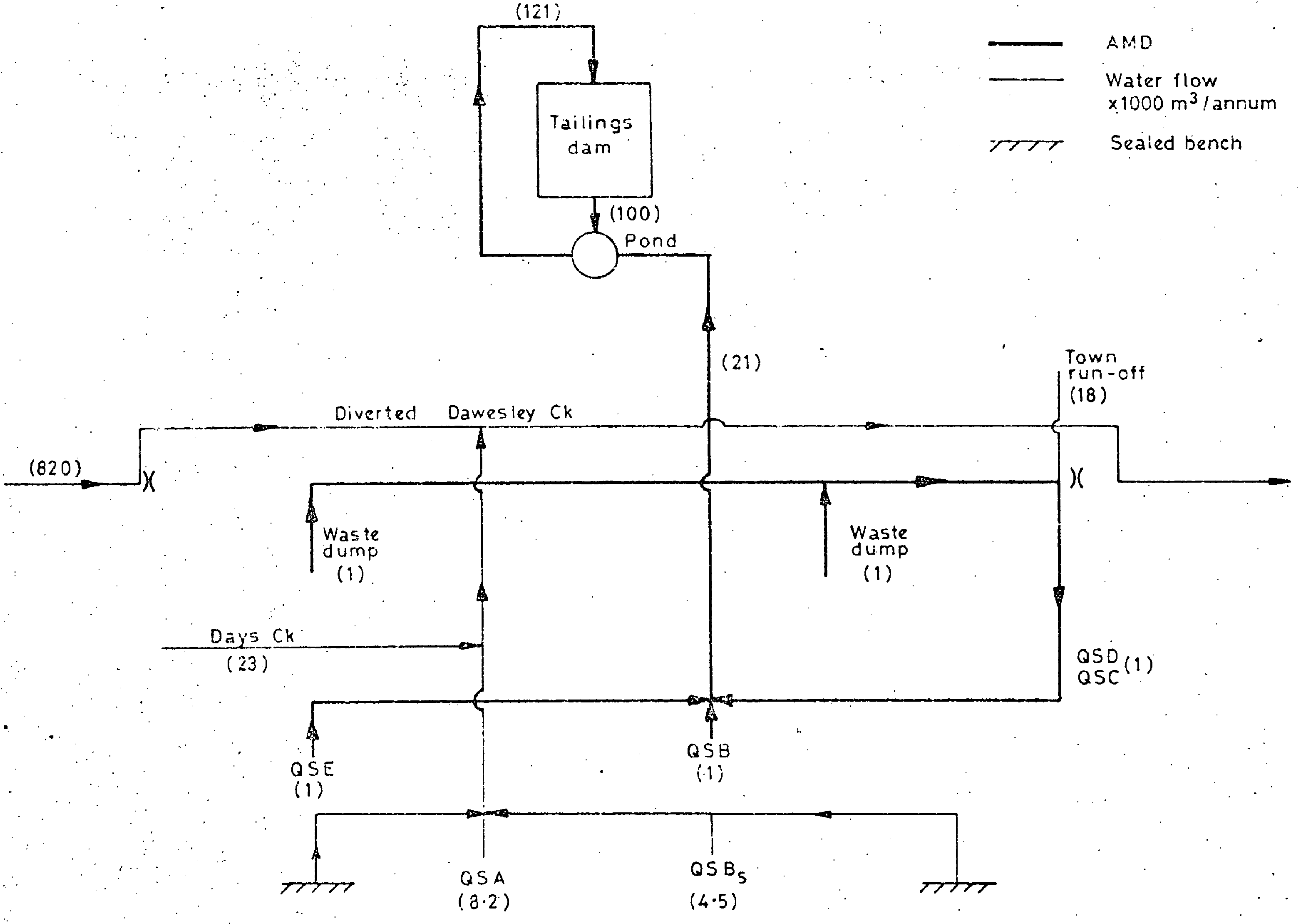


FIG. : PREVENTION SCHEME

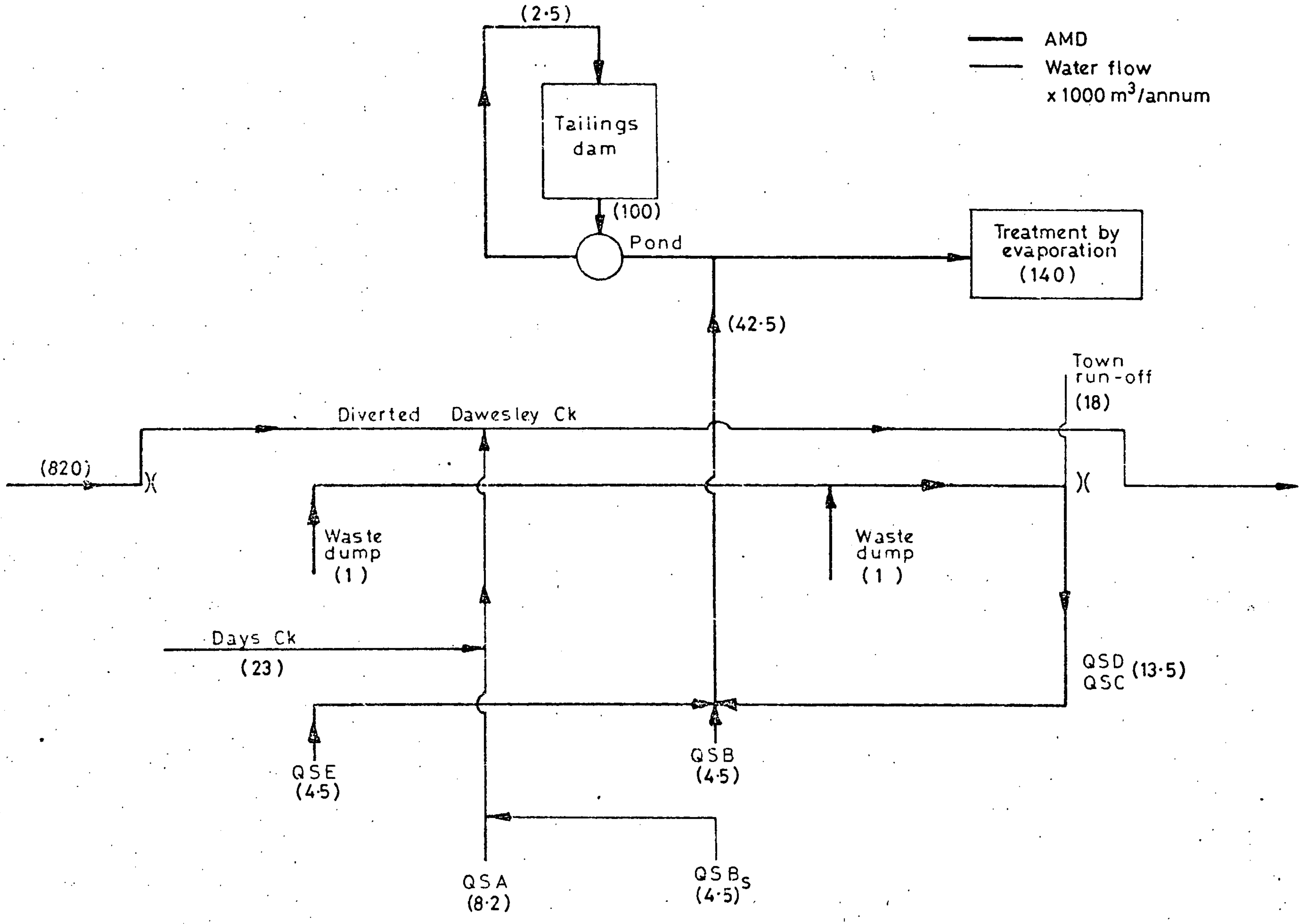


FIG. 14 : EVAPORATION SCHEME

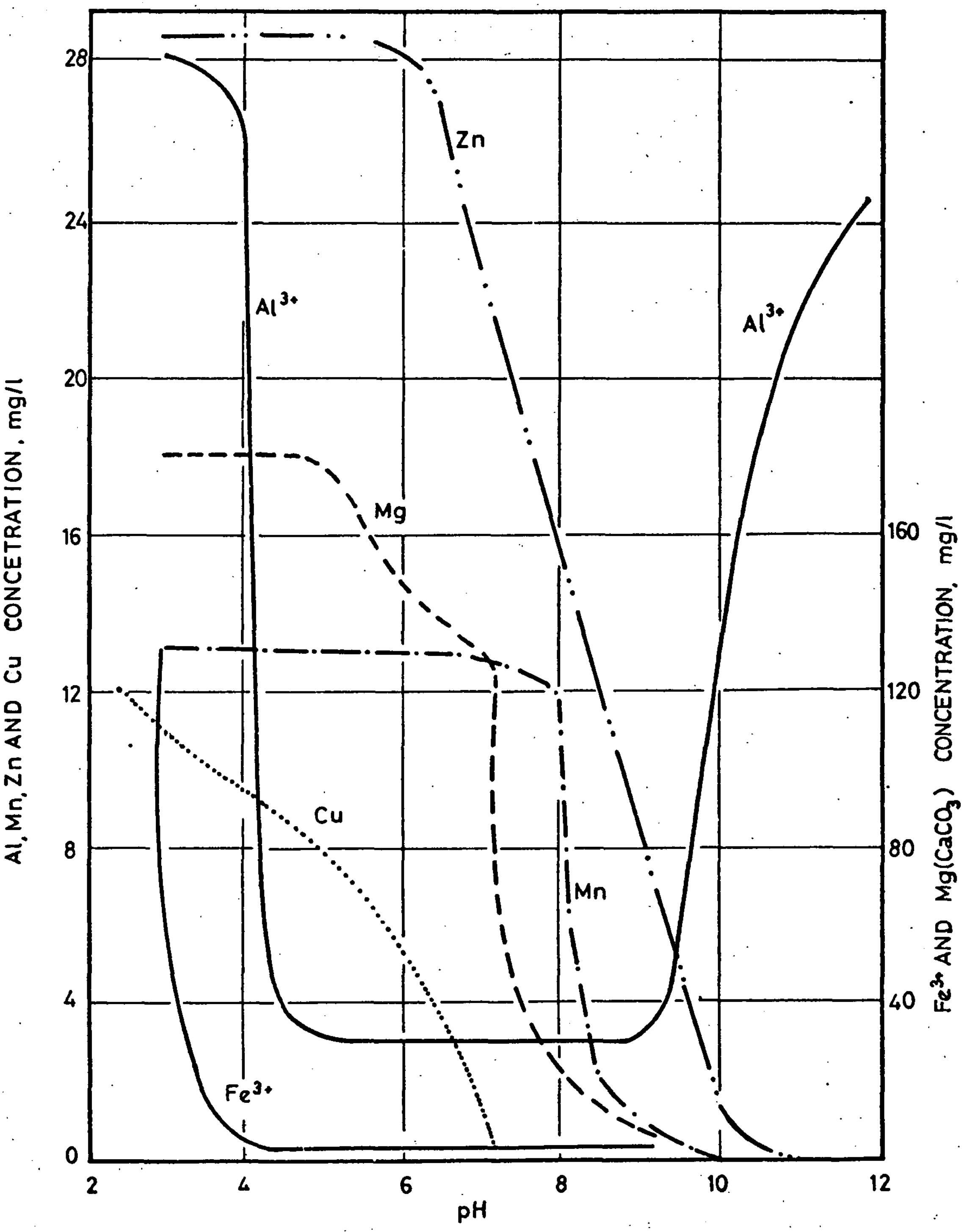


FIG.15: SOLUBILITY OF ACID MINE DRAINAGE METALS AT VARIOUS pH VALUES
(from Hill & Wilmoth, 1971)

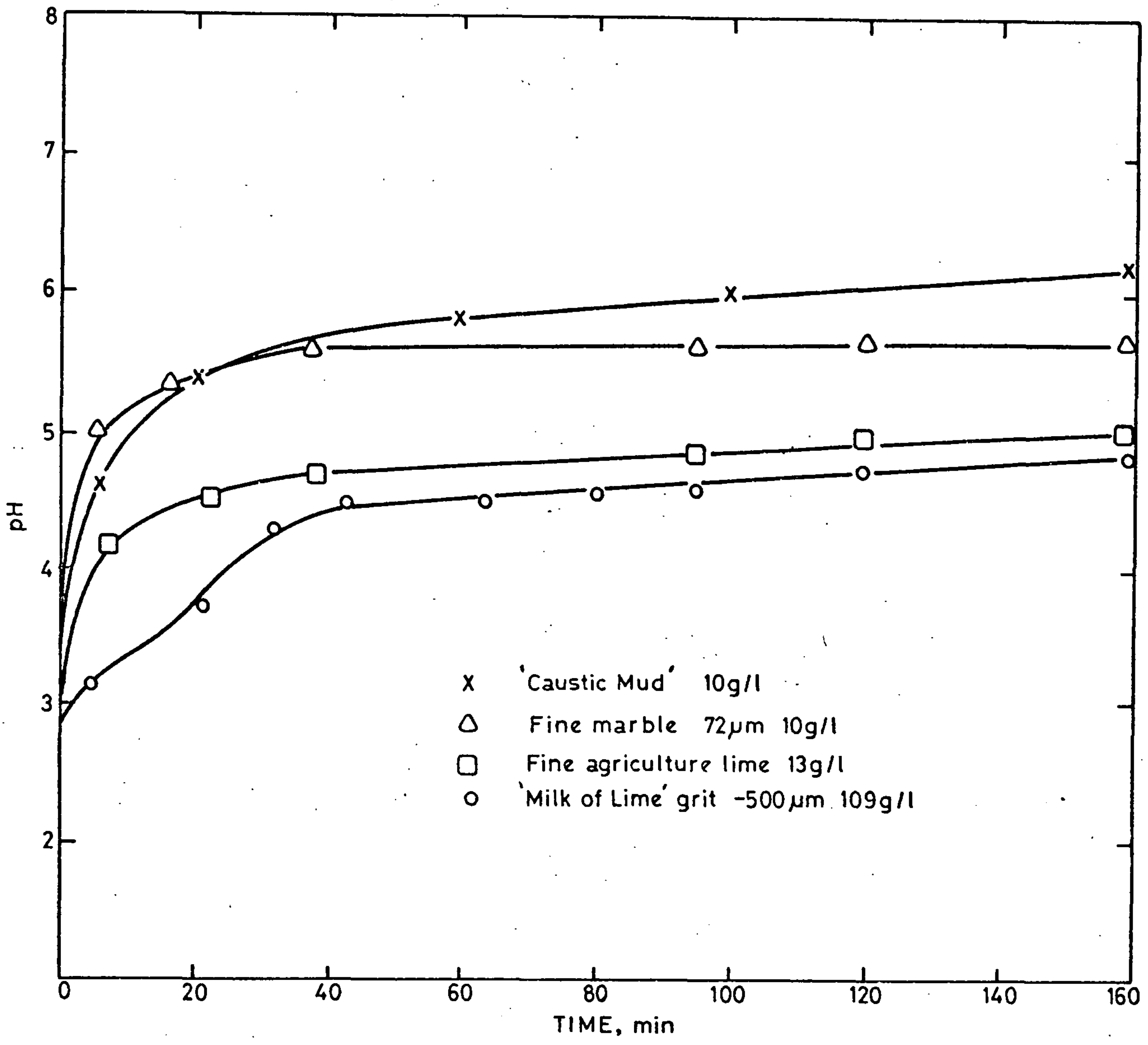


FIG.16: EFFECT OF ALKALI TYPE ON NEUTRALISATION

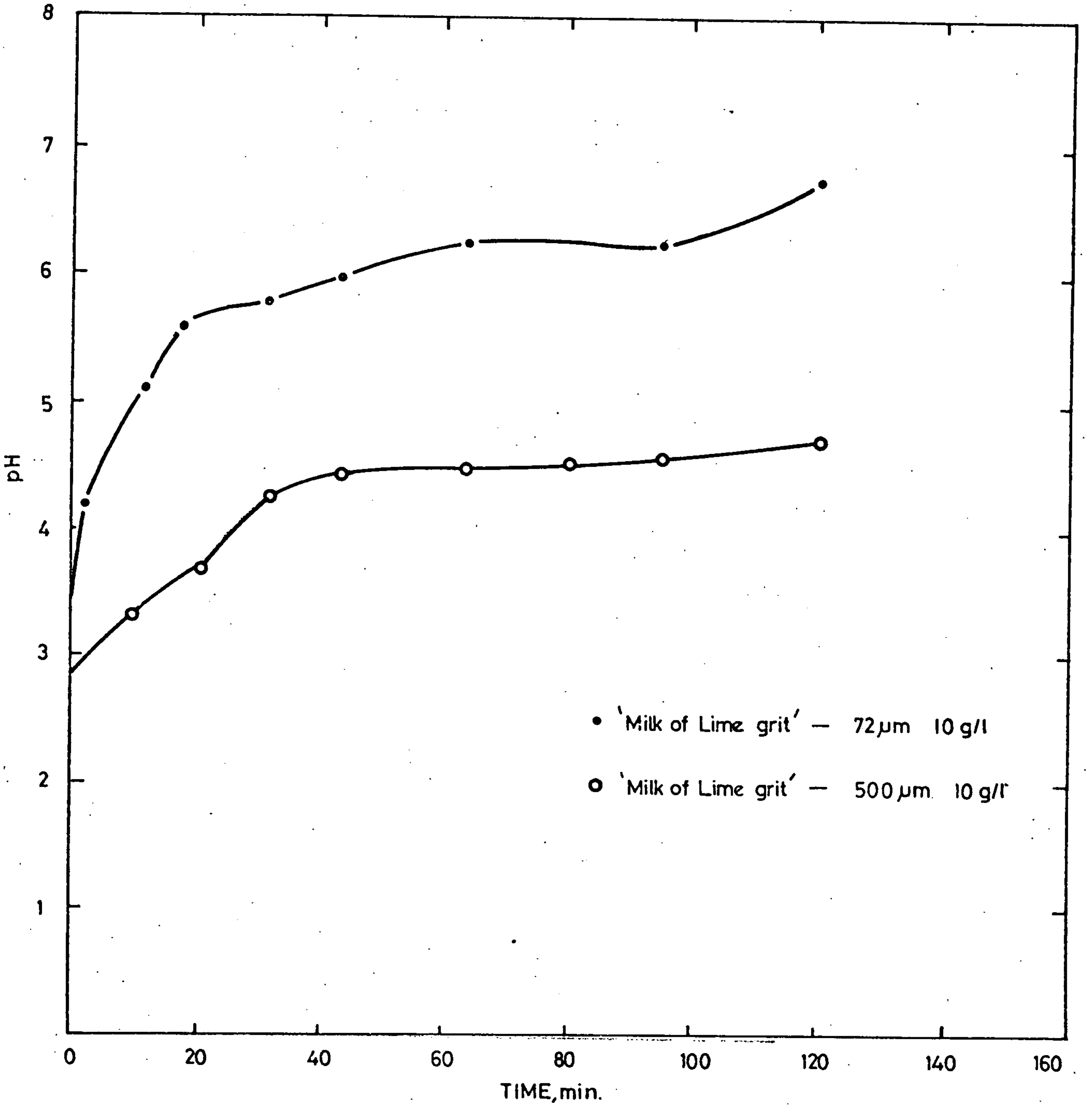


FIG. 17: EFFECT OF GRIT PARTICLE SIZE ON NEUTRALISATION

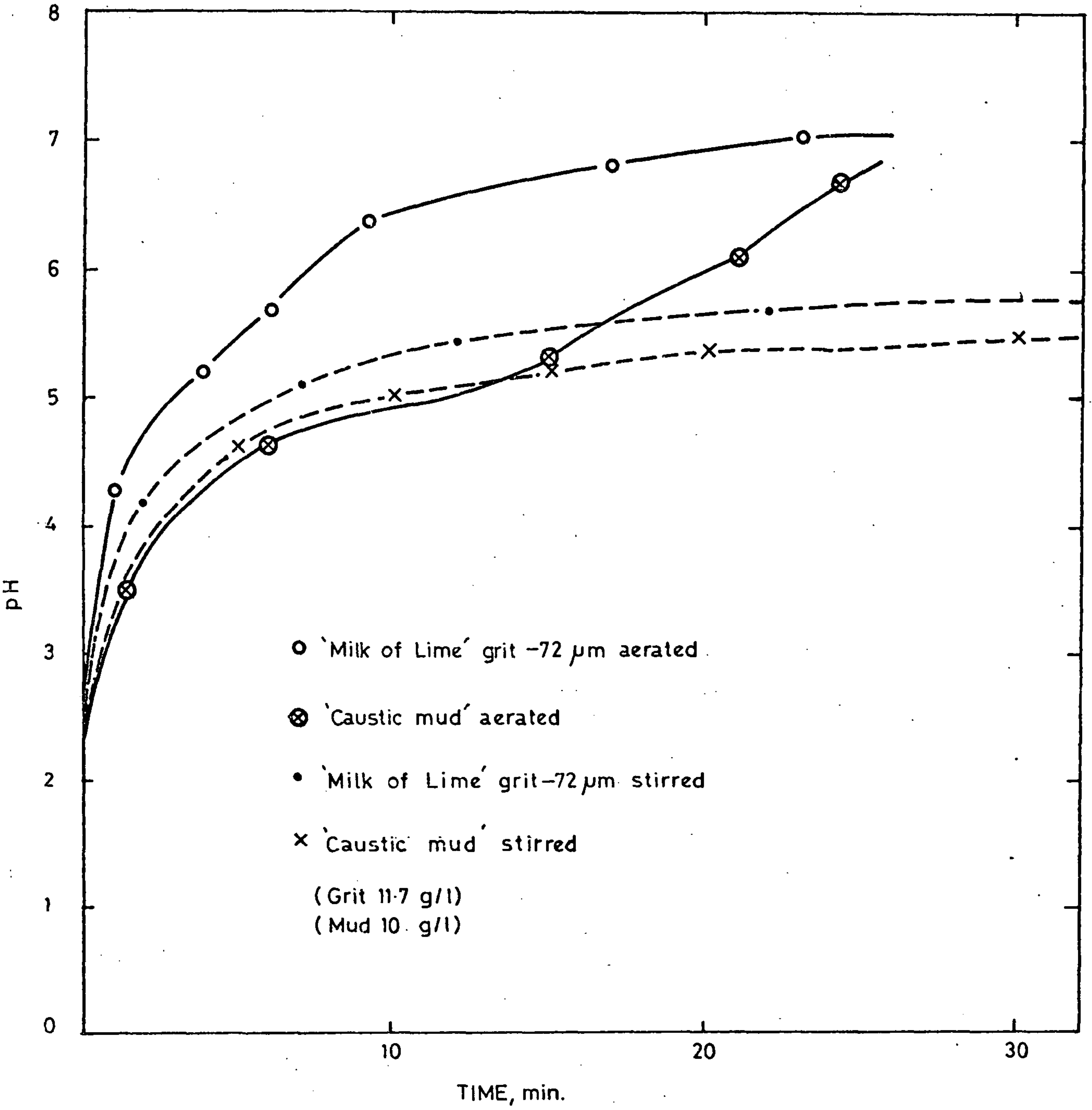


FIG. 18 EFFECT OF AERATION ON NEUTRALISATION

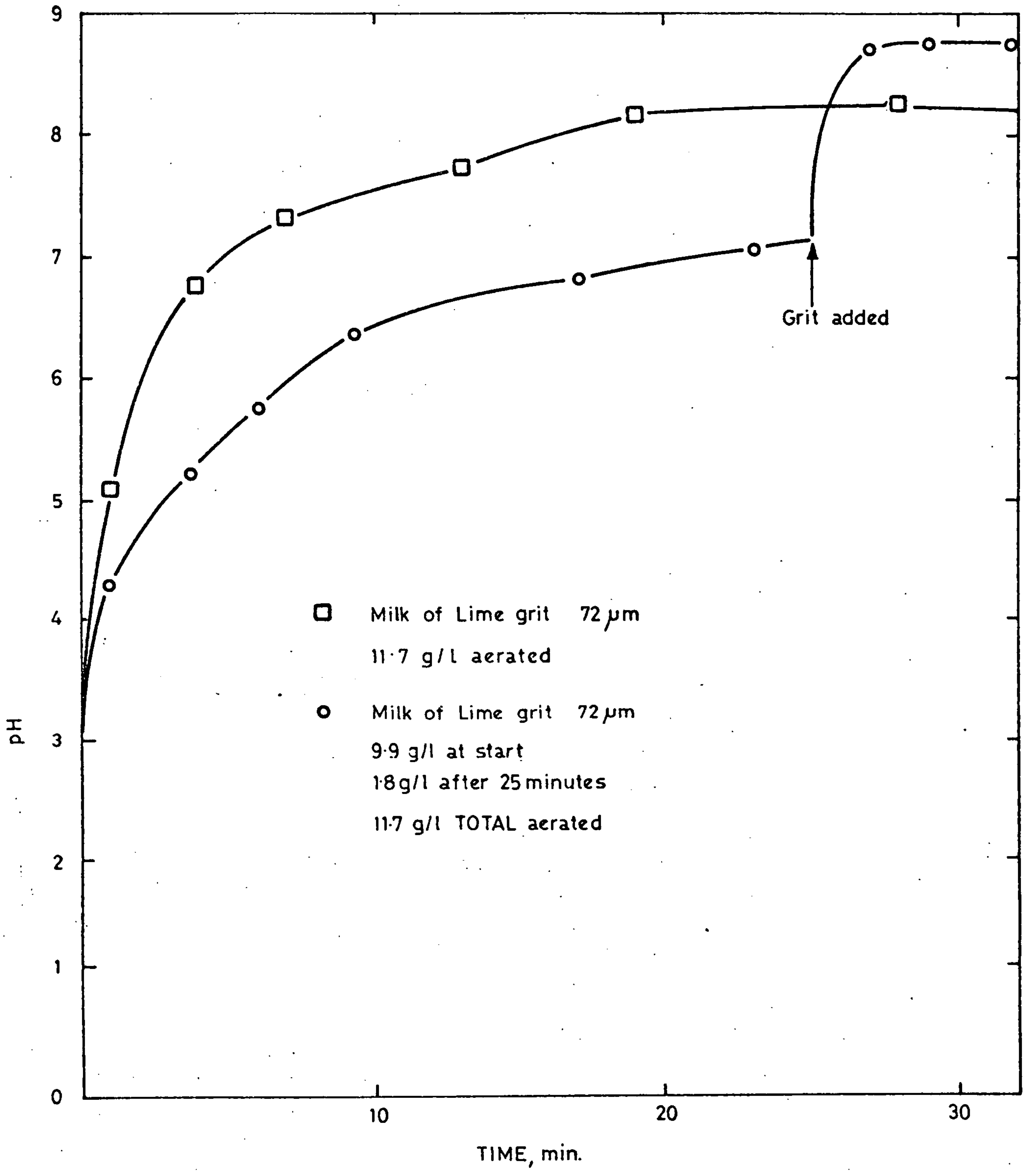


FIG. 19 : EFFECT OF STEPWISE ADDITION OF GRIT ON NEUTRALISATION

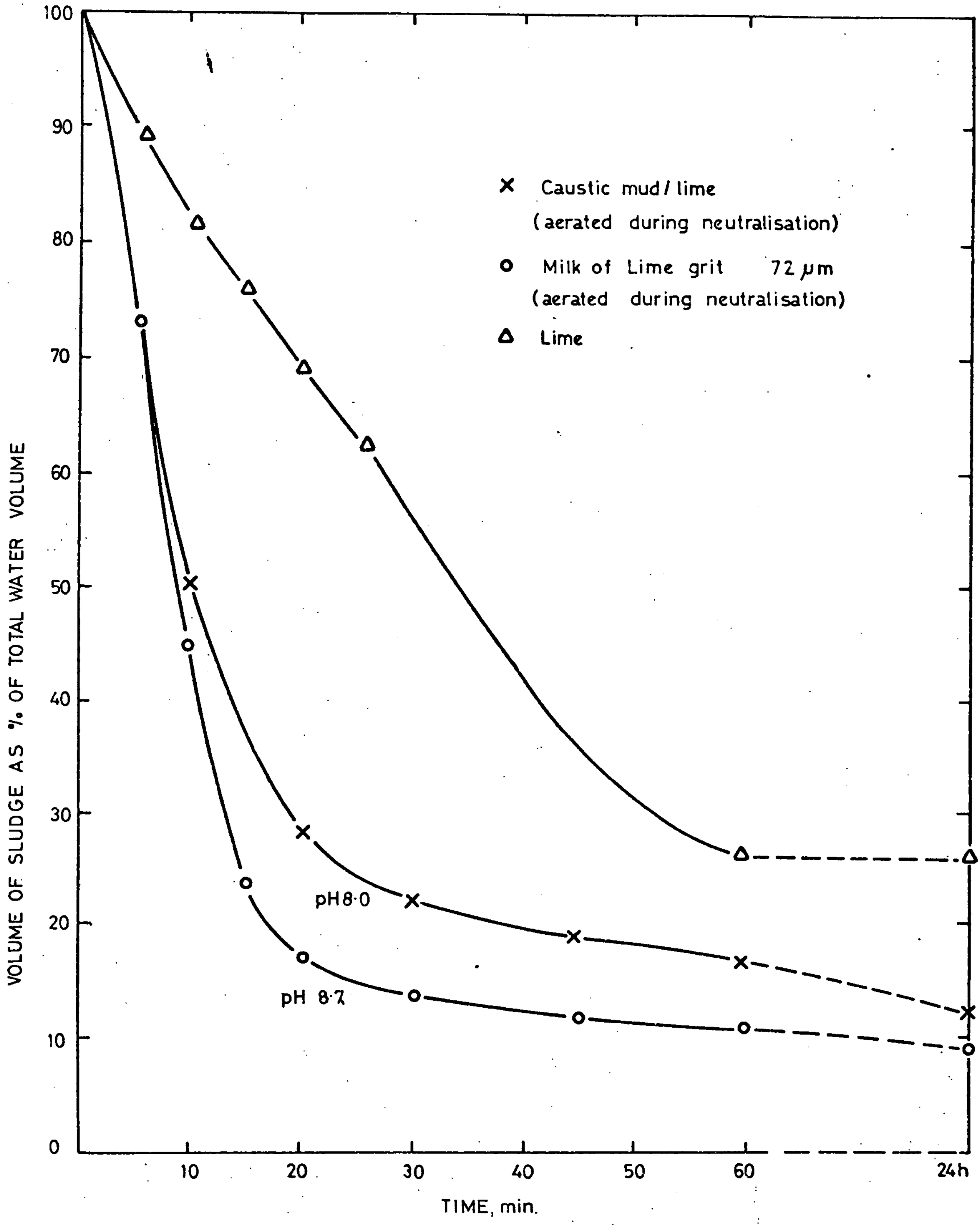


FIG. 20: SLUDGE SETTLING FOR FULLY NEUTRALISED ACID MINE DRAINAGE

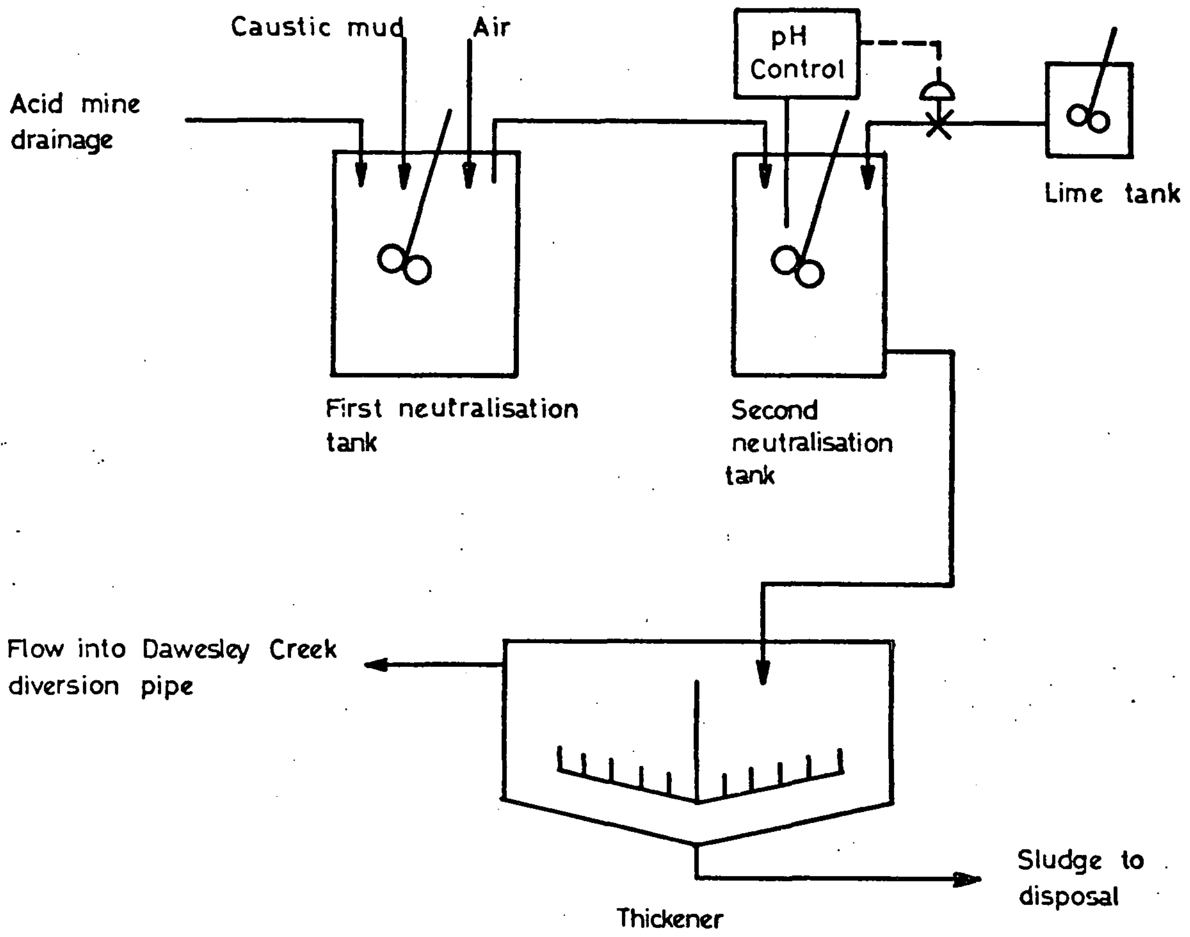
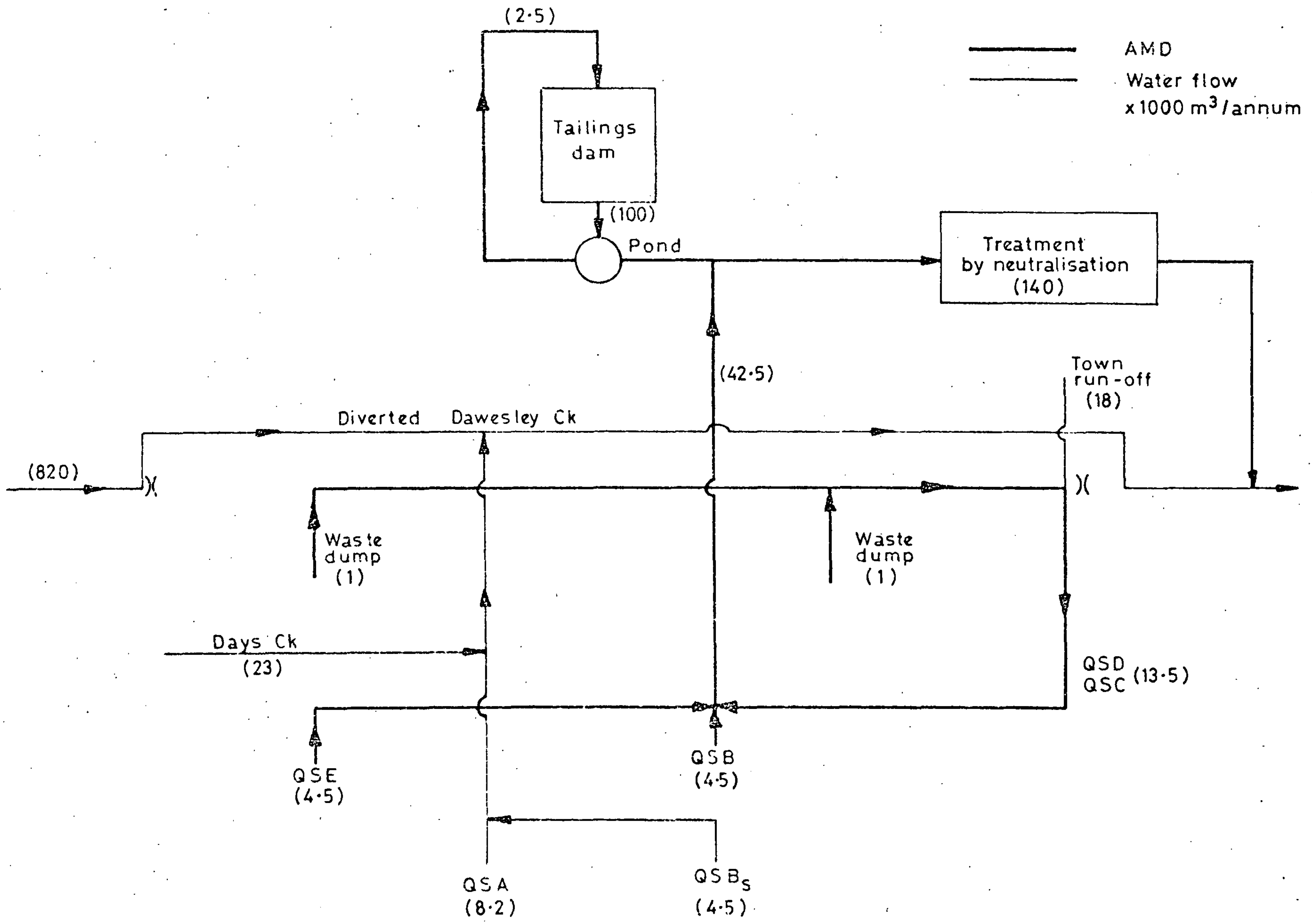


FIG. 21: CHEMICAL TREATMENT FLOW SHEET



Ǝ. 22: NEUTRALISATION SCHEME

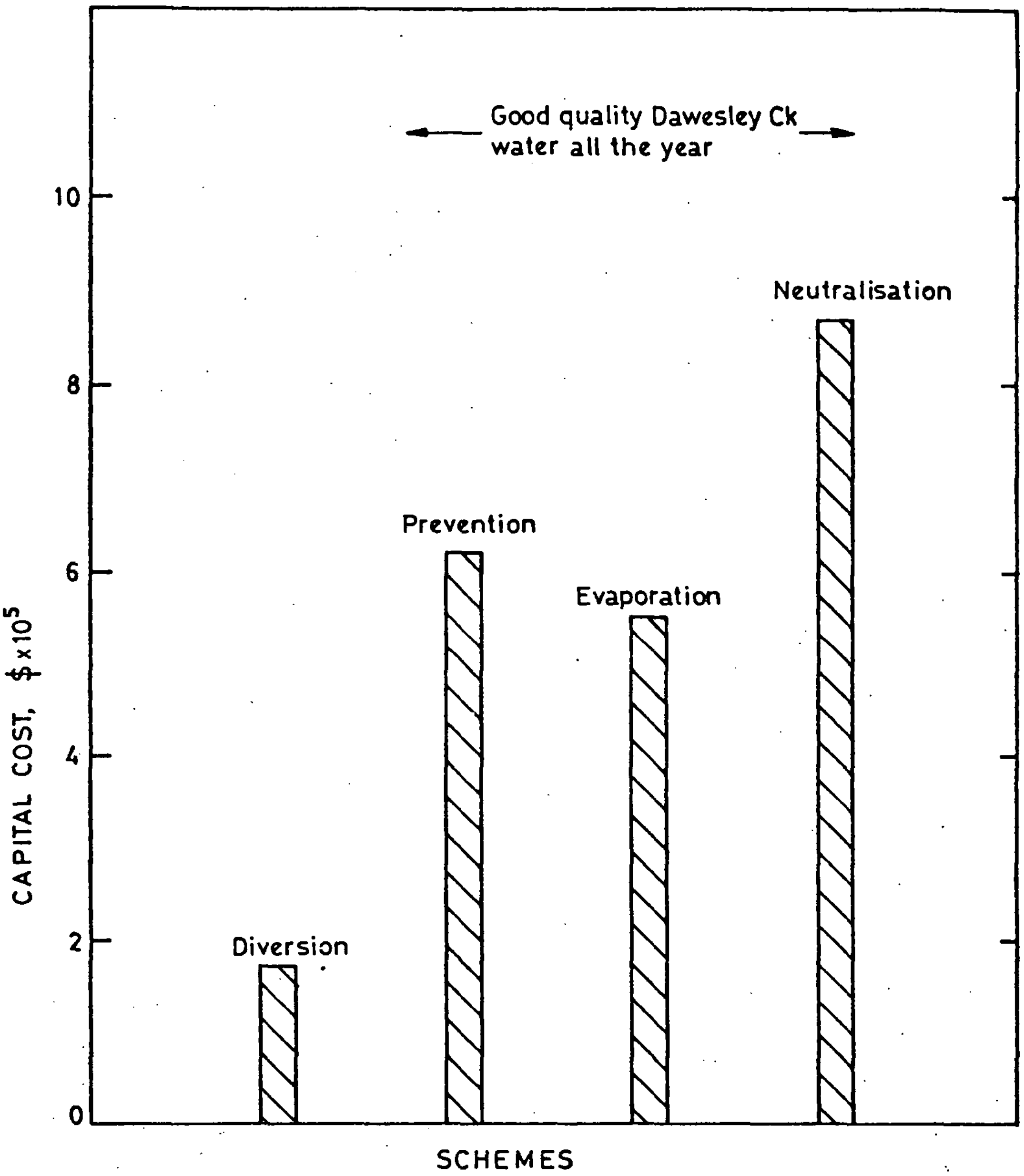


FIG.23: RELATIVE COST OF BASIC SCHEMES

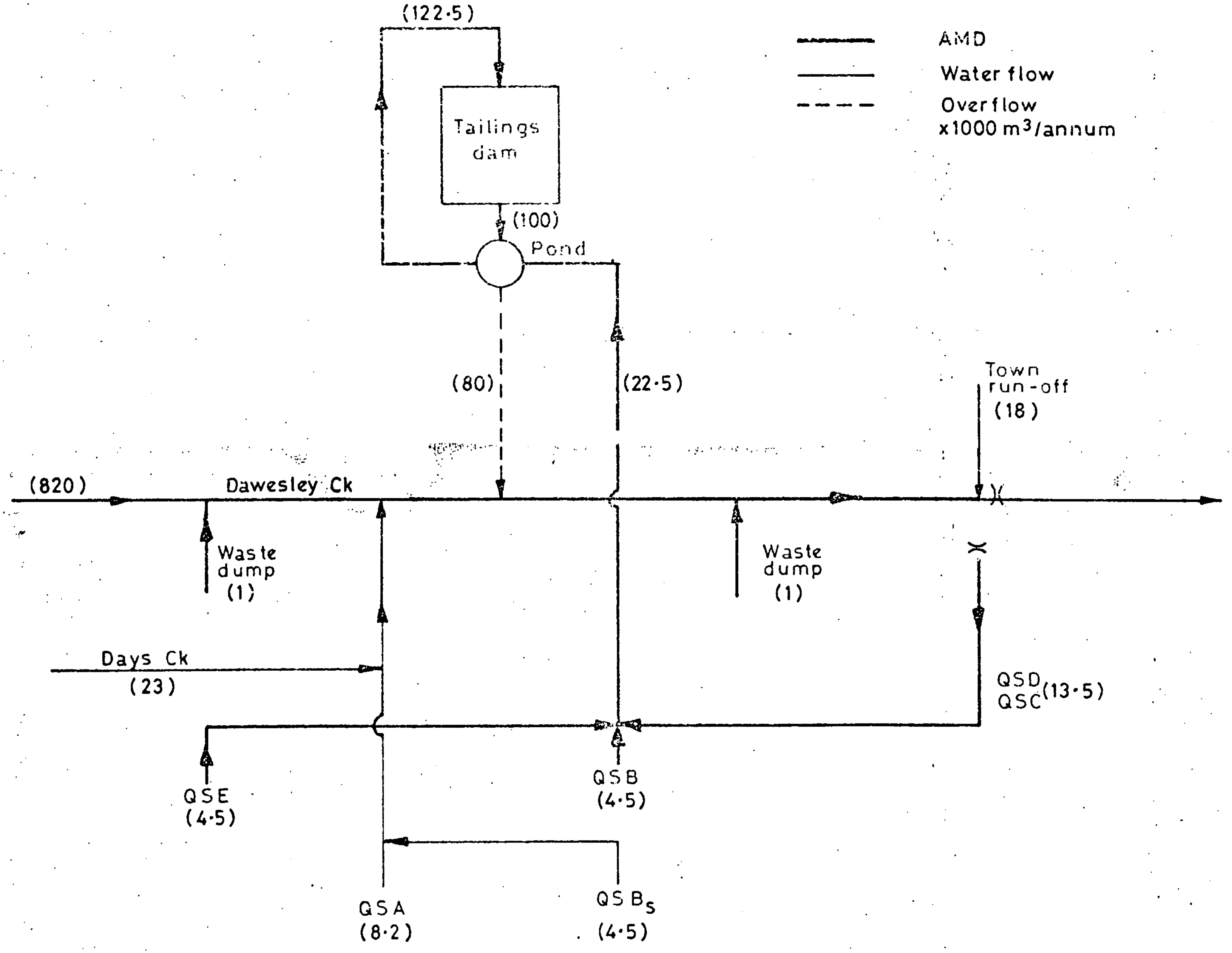


FIG. 24: INTERIM DIVERSION SCHEME

APPENDIX A

NAIRNE PYRITES PTY LIMITED
CRUSHING AND MILLING PROCEDURES 1971*

1. ORE TYPE - GRADE - TONNAGE TREATED

(a) Ore Composition - Grade:

The ore deposit consists mainly of pyrite and pyrrhotite in approximately equal proportions with minor amounts of copper, lead, zinc, antimony and arsenic sulphides. The host rock is composed of greywacke, mica-schist and quartzite. The pyrite and pyrrhotite constitutes approximately one quarter by weight of the rock while the grade of ore sent to the Crushing Section varies from 8% Sulphur to 13% Sulphur and averages 10.9% Sulphur. The pyrite and pyrrhotite lie in parallel bands of varying thicknesses (from 1 to 5 mm) and this causes the rock to break into slabs.

The ore from some sections of the quarry contains secondary iron sulphide minerals formed by the replacement of pyrrhotite. These minerals are not all amenable to flotation.

(b) Tonnage Treated

Tonnage throughput in the crushing section averages 243 tons per hour while the feed rate to the Concentrating plant is from 60 to 85 tons per hour dependent on ore hardness.

2. DESCRIPTION OF FLOW SHEET (Fig. A1)

The ore from the quarry is carted in trucks to the Crushing Section and tipped into a Pan Feeder. Operators regulate the feed rate to Jaw Crusher set at 5½ in. closed.

The jaw crusher discharge is delivered via conveyor to two Secondary Crushers set at 1¼ in. on the closed side. The discharge from these crushers drops onto a conveyor which discharges onto two double-deck screens. Both screens have woven wire 1¼ in. square aperture on the scalping deck and rubber-covered punched plate with ⅝ in. diam. aperture on bottom deck.

---

* Provided by Nairne Pyrites Pty Limited.

Screen undersize reports to the mill bin, live capacity 1600 tons, via conveyor.

Screen oversize is delivered to a 700-ton capacity surge bin on a conveyor, while delivery from this bin is by a Vibrating Feeder onto a conveyor and thence to three Cone Crushers with fine bowls set at $^3/_8$ in. on the closed side.

Tertiary Crusher discharge is delivered, along with the secondary crusher discharge, to the double-deck screens.

Feed to the grinding section is drawn off from the mill bin by four Vibrating feeders which deliver onto a conveyor. The ore is weighed prior to entering a Rod Mill.

The rod mill discharge gravitates either to one or both of two parallel rake classifiers, or 6-way distributor via pumps when the jigs are in operation. The rod mill can be operated in open circuit, but is generally close circuited with part of the sands from No.1 Classifier. The balance of No.1 Classifier sands is delivered by a scoop conveyor to the ball mill, while No.2 Classifier sands gravitate to the ball mill.

Ball Mill and Rod Mill discharges are pumped to a self-propelling 6-way distributor. From this distributor, feed can gravitate either to the two classifiers or to a rougher jig. The rougher jig concentrate gravitates to a cleaner jig, while the rougher jig tailing gravitates to the classifiers. Cleaner jig tailing, containing approx. 15% solids, is used in conjunction with rougher flotation middlings and cleaner flotation tailings to dilute rod mill feed and classifier sands.

Cleaner jig concentrate is pumped to two 550-ton capacity drainage vats, from which it is taken to the rail siding by motor truck.

For simplicity, the rod mill closed circuit sands pump and the scoop conveyor have been omitted from the flow sheet.

Classifier overflow is pumped by pump to a self-propelling 3-way distributor which delivers the flotation feed to three Conditioners.

Reagents are added in the grinding circuit in the following manner (for year to 30/6/70):

Activator: Copper Sulphate added to the pump delivering to the classifiers, at the rate of 0.086 lb/ton of crude ore.

Collector: Secondary Sodium Butyl Xanthate added to the classifier overflow pump at the rate of 0.093 lb/ton of crude ore, in a 10% solution.

Frother: 'Oil 66' added to the classifier overflow pump at the rate of 0.065 lb/ton of crude ore.

Grinding Media:

Rods - 3 in. rods charged as required. Usage 0.696 lb/ton of crude ore.

Balls - 1½ in. and 2 in. balls in the ratio of 2 to 1 by weight as required. Usage 0.874 lb/ton of crude ore.

Conditioning time for the flotation feed is 10 min, followed by 8 to 10 min, of flotation time in the rougher flotation units.

Tailings from the three rougher units are pumped in three stages, to the Tailings Dam. Rougher concentrates produced by the first 5 or 6 cells of each unit are up-graded in an 8-cell Rougher Flotation Machine, while the rougher unit middlings of 2 or 3 cells combined with cleaner tailings are pumped by pump to the grinding circuit to assist in dilution of mill feed and classifier sands.

Cleaner Unit concentrate, the final product, of 6 to 8 cells, is delivered to a cyclone, while cleaner unit middlings, if any, are returned to the head of the cleaner unit for refloating.

The concentrate cyclone discharge gravitates to a pump which delivers to a disc American type filter using top feed. Cyclone overflow is delivered to a 50 ft thickener for settling. 50 ft thickener underflow joins cyclone discharge, and overflow gravitates to mill circuit water tank.

The dried concentrate from the filter drops onto a conveyor and is delivered to a swivel piler on thrower, which stacks the concentrate, under shelter, ready for loading. The filter bath level is kept constant by maintaining an overflow to the filter feed pump. The total volume of feed to the filter is maintained by regulating the underflow from the 50 ft thickener. Production is approximately 12 tons of concentrate per hour.

Loading of the concentrate is by an endloader and motor trucks. The trucks cart it to a rail siding at Nairne 3½ miles distant.

The filtrate water is pumped to the 50 ft thickener through an filtrate pump. The air-water separation in the vacuum line is by means of a receiving tank, several traps, and a cyclone.

Water is pumped back from the tailings dam by pumps operating on automatic level controls in the circuit water tank. Return water from the tailings dam has a pH of 3.5. All water in the concentrating plant is pumped by pumps through an 8 in. rising main to a constant head tank, and an overflow back to the circuit water tank is maintained at all times.

All water take-off points are from the 8 in. rising main so that a constant pressure can be maintained throughout the plant.

Sampling to final jig and flotation concentrates and rougher tailings is done by automatic sampling machines while several other intermediate products are sampled hourly by hand.

Assays and sizings of various products are carried out daily, for each shift of operation.

Part of the rougher tailings is by-passed over a Wilfrey table and tailings losses can be observed instantaneously by noting the band of pyrite showing on the table. Supervisors vary reagent rates according to the 'show' on the table.

3. MILL PRODUCTS SIZINGS AND COMPOSITION

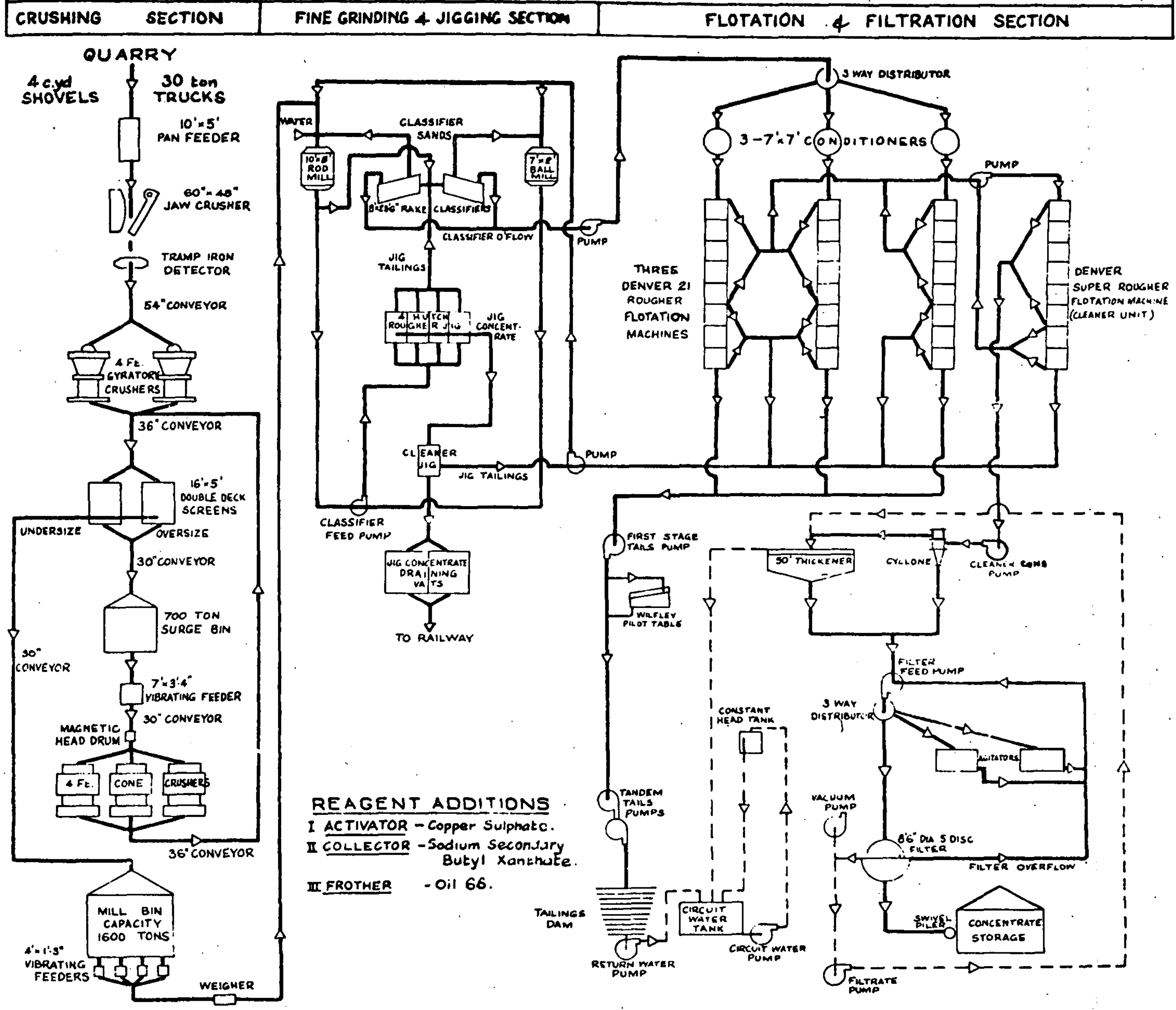
(See Table A1).

TABLE A-1: GRADES, SIZINGS AND PULP DENSITIES OF MILL PRODUCTS

Mesh	Rod Mill Feed	Rod Mill Discharge	Ball Mill Discharge	Class. Overflow	Mill Tailing	Jig* Concentrate	Flotation* Concentrate
+ 15.9 mm (⅝ in.)	Nil						
+ 12.7 mm (½ in.)	6.2						
+ 6.35 mm (¼ in.)	41.6						
+ 1.7 mm (10BSS)	26.3	5.0					
+850 μ (18BSS)	25.9 (-1.7mm)	17.0					
+420 μ (36BSS)		17.8	26.0	0.2	0.4	17.2	0.3
+300 μ (52BSS)		-	19.0	2.5	5.0	23.1	2.6
+210 μ (72BSS)		18.2	14.2	8.0	9.1	26.7	5.0
+150 μ (100BSS)		-	12.5	8.5	9.0	21.1	9.6
+105 μ (150BSS)		13.2	7.0	10.2	10.1	9.3	18.3
+ 75 μ (200BSS)		3.6	3.2	11.1	6.7	1.3	14.3
- 75 μ (200BSS)		25.2	18.1	59.5	59.7	1.3	49.9
Total	100.0	100.0	100.0	100.0	100.0	100.0	100.0
% Solids	Dry	80	80	39-40	35	97.0	91.6
% Sulphur	10.9	14.16	16-18	8- 9	1.4	46.6	39.8

* % Solids shown for jig concentrate and flotation concentrate are after gravity drainage and filtration respectively (i.e., as despatched for Sale). Total concentrate is approximately 25% by weight of crude ore.

NAIRNE PYRITES PTY LTD-CRUSHING AND MILLING FLOWSHEET



REAGENT ADDITIONS

I ACTIVATOR - Copper Sulphate.
II COLLECTOR - Sodium Secondary Butyl Xanthate.
III FROTHER - Oil 66.

1. BREMER RIVER SYSTEM WATER ANALYSIS*

Date	Major Constituents, mg/l										
	Ca	Mg	Na/K	Fe	Al	Mn	Zn	HCO_3	SO_4	Cl	TDS
Sampling Point (1) — Bird in Hand Lagoon Effluent											
21/ 6/73	3	15	380	-	-	-	-	630	96	260	1 063
Sampling Point (2) — Dawesley Creek, 20 m Upstream from Brukunga Mine											
11/ 8/72	40	39	273	-	-	-	-	165	80	440	954
13/10/72	-	-	-	-	-	-	0.16	-	-	-	-
10/ 4/73	140	170	1540	-	-	-	<0.5	460	-	-	-
7/ 6/73	38	28	415	1.10	<2	<0.1	<0.2	560	100	395	1 250
13/ 7/73	-	-	-	0.4	<1	<0.1	-	570	-	-	-
20/ 7/73	-	-	-	2.0	<2	<0.1	-	120	-	-	-
3/ 8/73	-	-	-	1.5	<1	<0.1	-	740	-	-	-
10/ 8/73	-	-	-	1.0	<1	<0.1	-	134	-	-	-
17/ 8/73	-	-	-	1.0	<1	<0.1	-	190	-	-	-
23/ 8/73	-	-	-	0.5	<1	<0.1	-	270	-	-	-
31/ 8/73	-	-	-	1.0	<1	<0.1	-	185	-	-	-
7/ 9/73	-	-	-	1.5	<1	<0.1	-	110	-	-	-
14/ 9/73	-	-	-	1.0	<1	<0.1	-	85	-	-	-
20/ 9/73	-	-	-	1.0	<1	0.1	0.10	180	-	-	-
21/ 9/73	-	-	-	1.0	<1	<0.1	-	210	-	-	-
28/ 9/73	-	-	-	0.5	<1	<0.1	-	230	-	-	-
4/ 4/74	67	61	400	1.0	0.16	0.17	0.03	200	96	710	1 420
Sampling Point (3) — Dawesley Creek, 0.5 km Downstream from Brukunga Mine											
11/ 8/72	50	55	230	67	47	-	-	Nil	880	350	1 700
5/ 1/73	545	720	665	845	-	-	-	Nil	7 900	530	11 000
10/ 4/73	220	410	312	700	1330	115	150	Nil	10 700	610	14 600
7/ 6/73	100	130	352	210	190	19	25	Nil	2 800	355	4 200
20/ 9/73	-	-	-	35	96	11	13	Nil	-	-	-
4/ 4/74	-	-	-	180	630	42	74	Nil	-	-	-
Sampling Point (4) — Dawesley Creek, 3 km from Brukunga Mine											
4/ 9/72	-	-	-	-	-	-	22		-	-	-
7/ 6/73	110	140	280	185	235	24	33	Nil	2 910	335	4 350
6/ 7/73	-	-	-	82	185	-	-	Nil	-	-	-
20/ 9/73	-	-	-	40	115	12	15	Nil	-	-	-
6/ 2/74	-	-	-	209	310	27	36	Nil	-	-	-
6/ 3/74	-	-	-	280	670	62	78	Nil	-	-	-
6/ 4/74	-	-	-	34	210	20	25	Nil	-	-	-

---

* Sampling points located on Map, Fig.B-1.

(Contd.)

Date	Major Constituents, mg/l										
	Ca	Mg	Na/K	Fe	Al	Mn	Zn	HCO_3	SO_4	Cl	TDS
	Sampling Point (5) – Dawesley Creek, 10 km from Brukunga Mine before Nairne Creek Junction										
11/ 8/72	61	64	220	59	57	-	-	Nil	1 080	335	1 870
20/ 9/73	-	-	-	5	53	9	8	Nil	-	-	-
	Sampling Point (6) – Nairne Creek just before Junction with Dawesley Creek										
11/ 8/72	42	34	253	1.5	-	-	-	155	90	390	880
20/ 9/73	-	-	-	0.8	<1	<0.1	0.1	200	-	-	-
	Sampling Point (7) – Dawesley Creek, 10.5 km from Brukunga Mine just after Nairne Creek										
11/ 8/72	57	57	220	24	44	-	-	Nil	845	355	1 600
20/ 9/73	-	-	-	3	35	6	6	Nil	-	-	-
	Sampling Point (8) – Dawesley Creek, 16 km from Brukunga Mine before Mount Barker Creek										
20/ 9/73	-	-	-	0.9	23	5	4	Nil	-	-	-
	Sampling Point (9) – Mount Barker Creek before Dawesley Creek Junction										
20/ 9/73	-	-	-	0.5	<1	<0.1	<0.1	200	-	-	-
	Sampling Point (10) – Mount Barker Creek, 16.5 km from Brunkunga Mine below Dawesley Creek Junction										
20/ 9/73	-	-	-	<0.3	<1	1.9	1.2	55	-	-	-
	Sampling Point (11) – Mount Barker Creek, 20 km from Brukunga Mine before Bremer River										
11/ 8/72	50	39	204	0.8	-	-	-	5	225	340	860
20/ 9/73	-	-	-	<0.3	1.4	<1	0.5	70	-	-	-
4/ 4/74	140	150	495	0.7	140	21	19	Nil	1780	750	3 500
	Sampling Point (12) – Bremer River before Mount Barker Creek										
11/ 8/72	37	47	297	1.0	-	-	-	135	90	510	1 050
20/ 9/73	-	-	-	0.8	<1	<0.1	0.1	200	-	-	-
4/ 4/74	83	105	980	0.6	0.4	0.04	<0.01	420	225	1560	4 860
	Sampling Point (13) – Bremer River, 21 km from Brukunga Mine after Mount Barker Creek										
11/ 8/72	45	40	230	1.3	-	-	-	5	225	340	885
20/ 9/73	-	-	-	0.5	<1	0.9	0.4	115	-	-	-
6/ 2/74	-	-	-	0.3	27	6	9	Nil	-	-	-

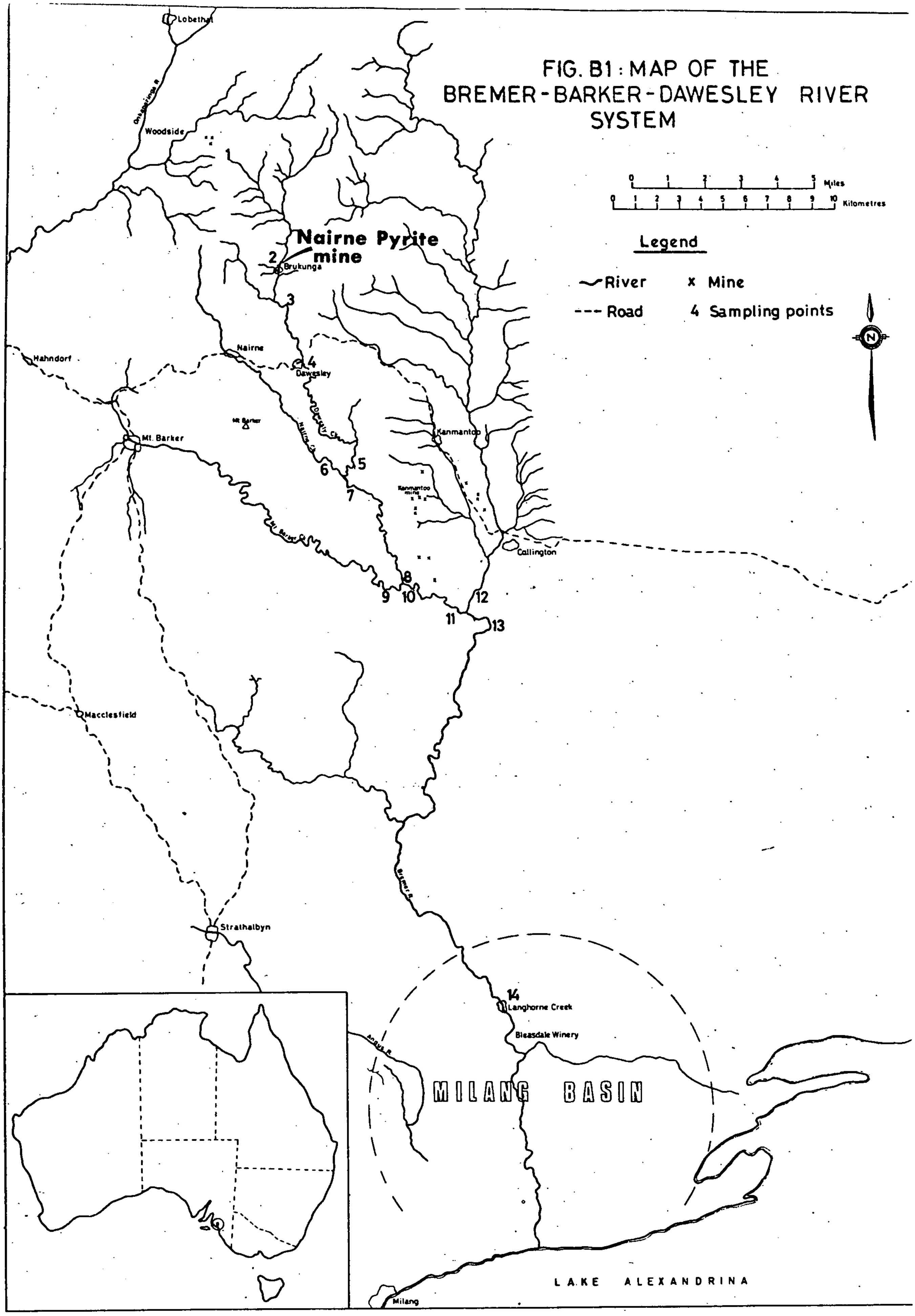


FIG. B1: MAP OF THE BREMER-BARKER-DAWESLEY RIVER SYSTEM

FIG. B2: MAP OF BRUKUNGA MINE
- DAWSLEY CREEK WATER ANALYSIS

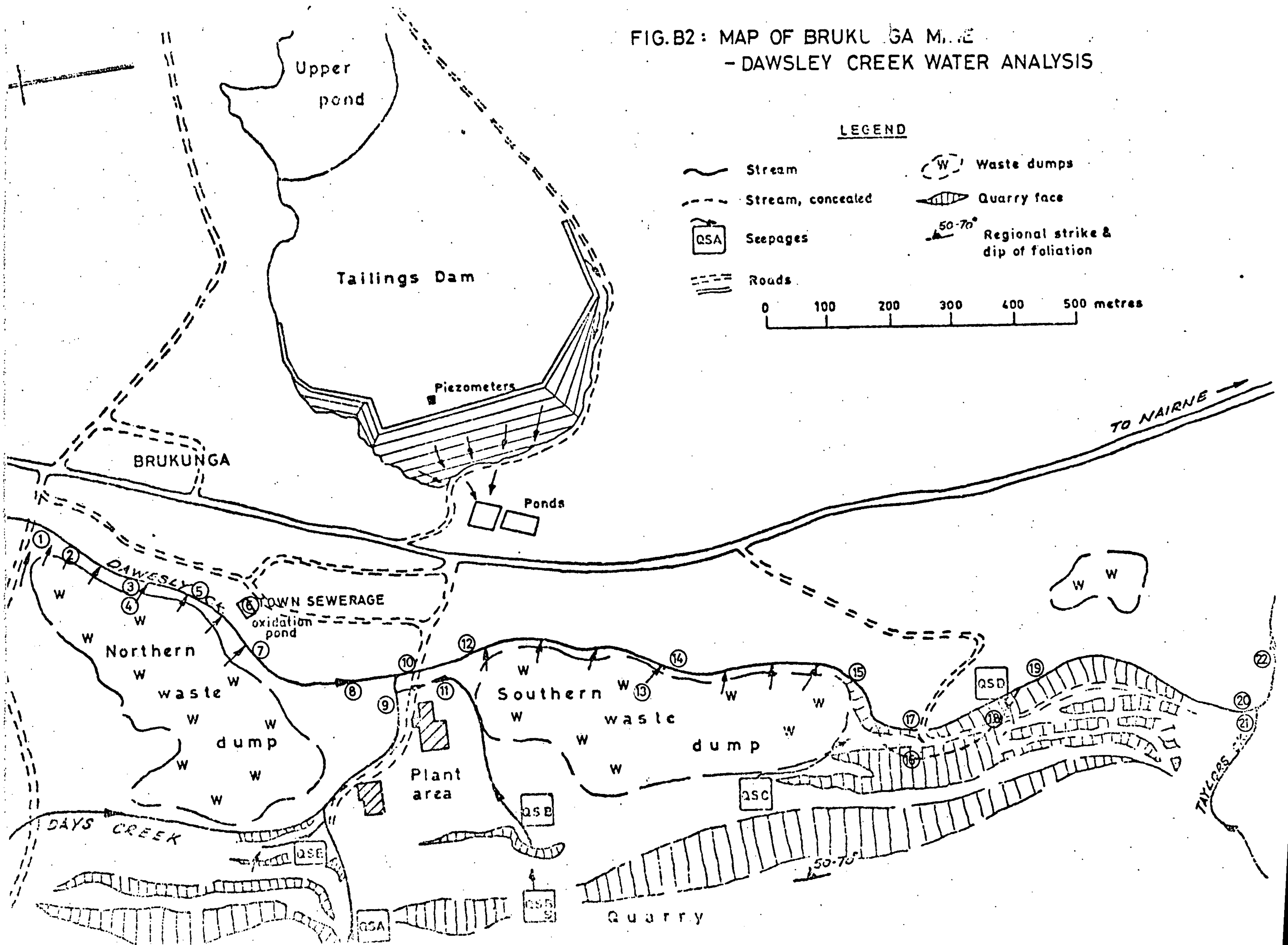
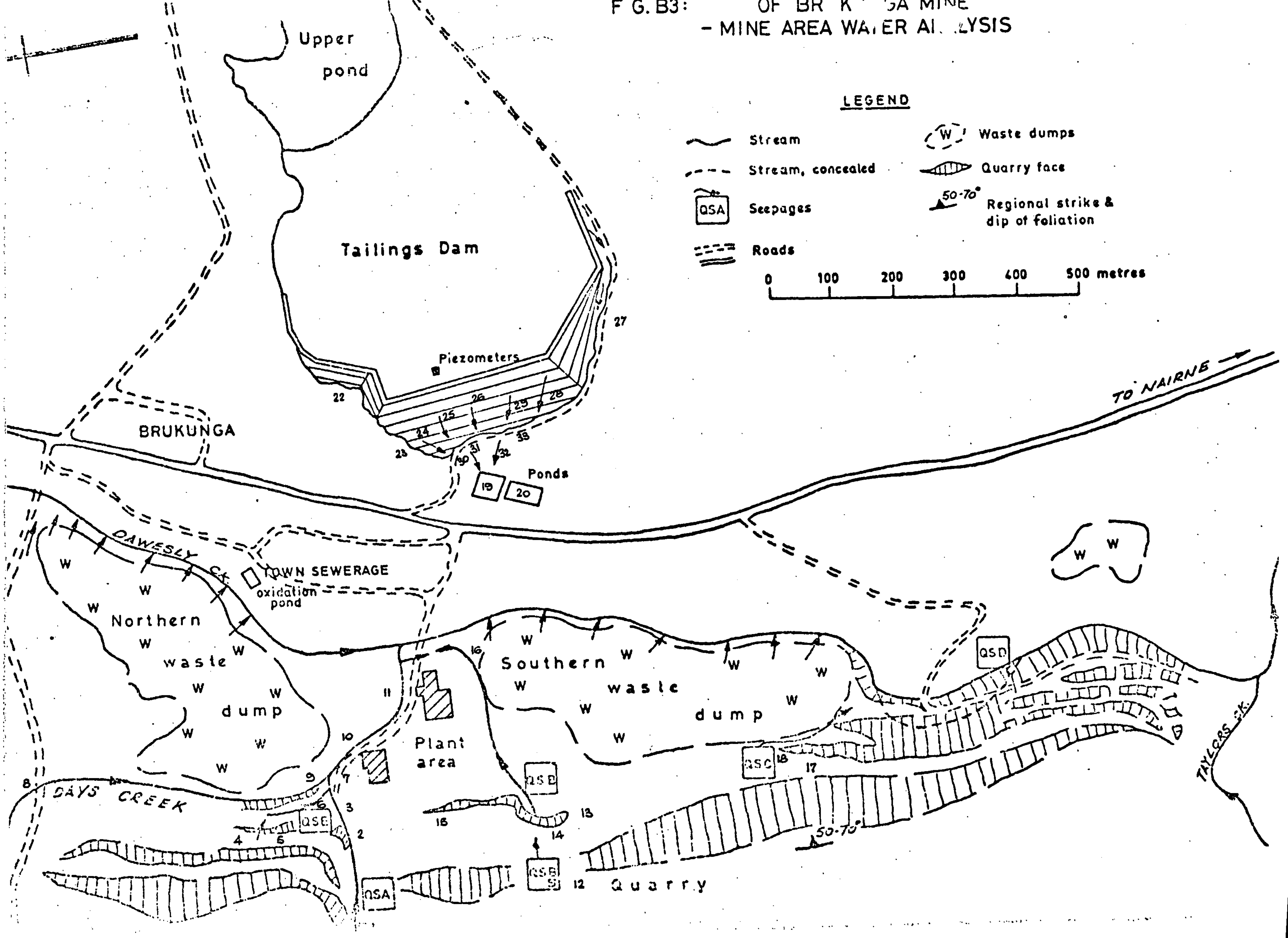


FIG. B3: OF BR K GA MINE
- MINE AREA WATER ANALYSIS



2. BRUKUNGA MINE — DAWESLEY CREEK AREA WATER ANALYSIS*

Date	Major Constituents, mg/l										
	Ca	Mg	Na/K	Fe	Al	Mn	Zn	HCO₃	SO₄	Cl	TDS
				Sampling Point (1) — 1 m Inside Mine Property in Dawesley Creek							
10/ 4/73	230	360	520	1280	135	56	18	Nil	-	-	
7/ 6/73	37	28	445	5.8	2	< 0.1	< 0.2	540	140	395	1 39
				Sampling Point (2) — 50 m Inside Mine Property in Dawesley Creek							
10/ 4/73	330	520	420	970	275	76	34	Nil	-	-	
				Sampling Point (3) — 200 m Inside Mine Property in Dawesley Creek							
10/ 4/73	270	380	570	280	255	44	25	Nil	-	-	
7/ 6/73	43	35	405	5.3	2	0.1	0.2	460	240	390	1 35
				Sampling Point (4) — Waste Dump Seepage, 200 m Inside Property							
10/ 4/73	180	270	142	50	2270	48	180	Nil	-		
10/ 4/73**	370	620	200	800	6900	205	300	Nil	36 000	220	45 70
7/ 6/73	180	400	148	68	2000	38	155	Nil	13 700	155	16 80
				Sampling Point (5) — 300 m Inside Property in Dawesley Creek							
10/ 4/73	230	360	640	100	505	41	35	Nil	-	-	
				Sampling Point (7) — 450 m Inside Mine Property in Dawesley Creek							
10/ 4/73	100	140	340	30	245	16	18	Nil	-	-	
				Sampling Point (9) — Pond at 700 m							
10/ 4/73	270	330	370	2150	465	105	135	Nil	-	-	
				Sampling Point (10) — 750 m Inside Mine Property in Dawesley Creek (at middle calvert)							
7/ 6/73	48	50	405	8.5	3	2	0.7	240	460	385	1 48
4/ 4/74	-	-	-	13.7	88	7	8	Nil	-	-	
				Sampling Point (11) — Pond at 800 m							
10/ 4/73	260	330	335	2200	465	110	135	Nil	8 400	630	12 90
				Sampling Point (12) — 900 m Inside Mine Property in Dawesley Creek							

Date	Major Constituents, mg/l										
	Ca	Mg	Na/K	Fe	Al	Mn	Zn	HCO_3	SO_4	Cl	
Sampling Point (14) — 1200 m Inside Mine Property in Dawesley Creek											
7/ 6/73	72	90	402	38	65	12	15	Nil	1 420	385	2
Sampling Point (15) — 1500 m Inside Mine Property in Dawesley Creek											
7/ 6/73	73	93	420	40	70	12	15	Nil	1 450	385	2
Sampling Point (16) — Pool in Old Creek Bed, 1600 m											
7/ 6/73	350	450	75	2950	1250	80	122	Nil	15 600	145	21
Sampling Point (17) — 1600 m Inside Mine Property in Dawesley Creek (at calvert)											
7/ 6/73	75	93	405	41	70	12	15	Nil	1 480	385	2
31/ 8/73	-	-	-	11	54	6	-	-	-	-	
7/ 9/73	-	-	-	7.5	55	5	-	-	-	-	
14/ 9/73	-	-	-	14	17	3	-	-	-	-	
21/ 9/73	-	-	-	28	135	14	-	-	-	-	
28/ 9/73	-	-	-	26	120	14	-	-	-	-	
4/ 4/74	-	-	-	325	950	79	-	-	-	-	
Sampling Point (18) — Old Creek Seepage, QSD, 1800 m											
10/ 4/73	290	410	92	1650	1340	40	135	Nil	13 200	270	17
4/ 4/74	-	-	-	1350	1450	87	130	Nil	-	-	
Sampling Point (19) — 1900 m Inside Mine Property in Dawesley Creek											
13/ 7/73	-	-	-	20	145	17	-	-	-	-	
20/ 7/73	-	-	-	240	165	16	-	-	-	-	
3/ 8/73	-	-	-	57	130	14	-	-	-	-	
10/ 8/73	-	-	-	23	100	8	-	-	-	-	
17/ 8/73	-	-	-	75	115	11	-	-	-	-	
23/ 8/73	-	-	-	38	150	15	-	-	-	-	
31/ 8/73	-	-	-	54	105	9	-	-	-	-	
7/ 9/73	-	-	-	33	78	7	-	-	-	-	
14/ 9/73	-	-	-	18	25	3	-	-	-	-	
21/ 9/73	-	-	-	18	170	14	-	-	-	-	
28/ 9/73	-	-	-	44	140	15	-	-	-	-	
4/ 4/74	-	-	-	370	866	75	115	-	-	-	

Date	Ca	Mg	Na/K	Fe	Al	Mn	Zn	HCO_3	SO_4	Cl	TDS
Sampling Point (21) — Taylors Creek before Junction with Dawesley Creek (2150 m)											
10/ 4/73	60	65	470	<0.5	<1	<0.1	<0.5	–	–	–	–
Sampling Point (22) — 2200 m Inside Mine Property in Dawesley Creek after Taylors Creek											
10/ 4/73	220	410	312	700	1325	115	150	Nil	–	–	–

Date	Acid Value as CaCO$_3$	Acid Values and Minor Constituents, mg/l*					pH
		Ni	Co	Cu	Cd	SiO$_2$	
Sampling Point (1)							
10/ 4/73	-	1.2	0.6	0.3	0.2	-	2.8
7/ 6/73	-	<0.2	<0.2	<0.1	<0.02	11.5	7.7
10/ 8/73	-100	-	-	-	-	-	6.7
Sampling Point (2)							
10/ 4/73	-	3.3	2.8	0.20	0.11	-	2.7
Sampling Point (3)							
10/ 4/73	-	2.1	1.6	0.40	0.13	-	2.7
7/ 6/73	-	<0.2	<0.2	<0.1	<0.02	10.0	7.4
Sampling Point (4)							
10/ 4/73	-	9.0	7.5	10.0	2.4	-	3.0
10/ 4/73**	40 000	20.0	2.5	17.0	6.0	-	2.7
7/ 6/73	12 300	12.0	8.0	8.6	2.0	150	3.2
10/ 8/73	14 000	-	-	-	-	-	3.0
Sampling Point (5)							
10/ 4/73	-	3.1	2.3	0.60	0.25	-	2.9
10/ 8/73	-70	-	-	-	-	-	6.0
Sampling Point (6) — Oxidation Pond Effluent (400 m)							
10/ 8/73	-105	-	-	-	-	-	6.0
Sampling Point (7)							
10/ 4/73	-	1.9	1.4	0.20	0.14	-	3.7
10/ 8/73	-65	-	-	-	-	-	5.8
Sampling Point (8) — 650 m Inside Mine Property in Dawesley Creek							
10/ 8/73	-25	-	-	-	-	-	5.75
Sampling Point (9)							
10/ 4/73	-	4.0	1.60	0.20	0.50	-	2.6
10/ 8/73	3 100	-	-	-	-	-	2.75
Sampling Point (10)							
7/ 6/73	-	0.3	<0.2	<0.1	0.02	4	7.0
10/ 8/73	120	-	-	-	-	-	4.7
4/ 4/74	680	0.69	0.50	0.20	0.08	-	3.6

---

* Sampling points described for Major Constituents.

** Sampling point 400 m inside property.

Date	Acid Value as $CaCO_3$	Acid Values and Minor Constituents, mg/l*					pH
		Ni	Co	Cu	Cd	SiO_2	
		Sampling Point (11)					
10/ 4/73	-	4.0	1.8	0.30	0.49	-	2.6
10/ 8/73	3 650	-	-	-	-	-	2.85
		Sampling Point (12)					
7/ 6/73	350	0.40	0.50	0.15	0.05	20.5	3.45
10/ 8/73	630	-	-	-	-	-	4.20
	Sampling Point (13) — Waste Dump Seepage (1000-1400 m)						
10/ 8/73	66 000	-	-	-	-	-	2.25
		Sampling Point (14)					
7/ 6/73	680	1.0	1.0	0.30	0.07	19.0	3.0
		Sampling Point (15)					
7/ 6/73	685	0.9	1.0	0.20	0.07	19.5	3.0
		Sampling Point (16)					
7/ 6/73	13 800	6.2	3.0	3.6	0.47	160	2.8
		Sampling Point (17)					
7/ 6/73	735	0.7	1.0	0.20	0.07	21.0	3.0
10/ 8/73	660	-	-	-	-	-	4.25
31/ 8/73	518	0.4	-	-	-	-	4.48
7/ 9/73	570	0.4	-	-	-	-	4.30
14/ 9/73	246	0.3	-	-	-	-	4.60
21/ 9/73	1 180	1.0	-	-	-	-	3.55
28/ 9/73	1 040	0.8	-	-	-	-	3.52
4/ 4/73	7 650	5.95	2.1	1.27	0.41	-	2.9
		Sampling Point (18)					
10/ 4/73	-	5.8	2.9	3.2	0.48	-	2.6
10/ 8/73	16 300	-	-	-	-	-	2.4
4/ 4/74	12 100	6.0	2.70	2.45	0.41	-	2.3
		Sampling Point (19)					
13/ 7/73	1 400	1.3	-	-	-	-	3.25
20/ 7/73	1 950	1.2	-	-	-	-	3.2
3/ 8/73	1 000	0.9	-	-	-	-	2.95
10/ 8/73	874	0.5	-	-	-	-	3.2
17/ 8/73	1 054	0.5	-	-	-	-	3.69

Sampling Point (19) (Contd.)

Date	Acid Value as $CaCO_3$	Acid Values and Minor Constituents, mg/l					pH
		Ni	Co	Cu	Cd	SiO_2	
23/ 8/73	1 506	1.0	-	-	-	-	3.30
31/ 8/73	918	0.5	-	-	-	-	3.48
7/ 9/73	760	0.5	-	-	-	-	3.65
14/ 9/73	260	0.3	-	-	-	-	4.4
21/ 9/73	1 250	1.0	-	-	-	-	3.32
28/ 9/73	1 140	1.0	-	-	-	-	3.1
4/ 4/74	7 800	5.5	2.0	1.33	0.38	-	2.7
		Sampling Point (20)					
10/ 4/73	-	6.8	3.5	2.3	0.68	-	3.7
7/ 6/73	1 920	1.4	1.0	0.60	0.11	37	2.8
		Sampling Point (21)					
10/ 4/73	-	<0.05	<0.05	<0.03	<0.01	-	8.0
		Sampling Point (22)					
10/ 4/73	-	6.8	3.4	2.2	0.67	-	2.8

3. BRUKUNGA MINE — QUARRY AND TAILING DAM SEEPAGE CHARACTERISTICS*

Date	Major Constituents, mg/l										
	Ca	Mg	Na/K	Fe	Al	Mn	Zn	HCO_3	SO_4	Cl	TDS
Sampling Point (4) — QSE											
5/ 1/73	680	670	810	2230	-	-	-	Nil	7 780	565	-
3/ 4/73	-	-	-	3200	-	-	-	Nil	10 000	-	-
10/ 4/73	355	540	433	5300	695	240	140	Nil	17 200	1 100	26 000
Sampling Point (6) — QSE on Quarry Bench											
7/ 6/73	310	425	133	5200	850	185	250	Nil	16 650	235	24 200
Sampling Point (11) — QSA + QSE + Days Creek											
10/ 4/73	290	320	365	2200	460	110	135	Nil	-	-	-
13/ 7/73	-	-	-	1800	575	95	-	-	-	-	-
20/ 7/73	-	-	-	1320	415	70	-	-	-	-	-
3/ 8/73	-	-	-	980	365	45	-	-	-	-	-
10/ 8/73	-	-	-	430	255	26	-	-	-	-	-
17/ 8/73	-	-	-	560	255	28	-	-	-	-	-
23/ 8/73	-	-	-	900	350	50	-	-	-	-	-
31/ 8/73	-	-	-	580	235	30	-	-	-	-	-
7/ 9/73	-	-	-	275	160	18	-	-	-	-	-
14/ 9/73	-	-	-	280	118	10	-	-	-	-	-
21/ 9/73	-	-	-	550	260	32	-	-	-	-	-
28/ 9/73	-	-	-	550	245	31	-	-	-	-	-
Sampling Point (16) — QSB											
10/ 4/73	380	430	387	3650	410	95	90	Nil	10 400	680	16 500
13/ 7/73	-	-	-	3000	620	80	-	-	-	-	-
20/ 7/73	-	-	-	2050	475	52	-	-	-	-	-
3/ 8/73	-	-	-	2200	465	60	-	-	-	-	-
10/ 8/73	-	-	-	1500	435	42	-	-	-	-	-
17/ 8/73	-	-	-	1890	435	56	-	-	-	-	-
23/ 8/73	-	-	-	2150	440	65	-	-	-	-	-
31/ 8/73	-	-	-	1700	410	45	-	-	-	-	-
7/ 9/73	-	-	-	503	215	30	-	-	-	-	-
14/ 9/73	-	-	-	1070	265	40	-	-	-	-	-
21/ 9/73	-	-	-	650	280	35	-	-	-	-	-
28/ 9/73	-	-	-	600	245	34	-	-	-	-	-
Sampling Point (18) — QSC											
			[illegible]	5100	-	-	100	Nil	14 700	245	21 700

Sampling Point (19) (Contd.)

Date	Major Constituents, mg/1										
	Ca	Mg	Na/K	Fe	Al	Mn	Zn	HCO_3	SO_4	Cl	TDS
20/ 7/73	–	–	–	1820	125	135	–	–	–	–	–
3/ 8/73	–	–	–	2130	81	145	–	–	–	–	–
10/ 8/73	–	–	–	1940	110	141	–	–	–	–	–
17/ 8/73	–	–	–	2300	80	147	–	–	–	–	–
23/ 8/73	–	–	–	2400	68	160	–	–	–	–	–
31/ 8/73	–	–	–	2070	94	150	–	–	–	–	–
7/ 9/73	–	–	–	1880	98	142	–	–	–	–	–
14/ 9/73	–	–	–	1950	145	175	–	–	–	–	–
21/ 9/73	–	–	–	2150	78	170	–	–	–	–	–
28/ 9/73	–	–	–	1900	65	160	–	–	–	–	–
Sampling Point (20) — Tailing Dam Seepage Storage (static)											
10/ 4/73	490	390	173	1300	230	200	40	Nil	6900	535	10 300
Sampling Point (21) — Tailing Dam Upper Pond											
10/ 4/73	540	360	288	1300	220	215	60	Nil	6900	485	10 400
7/ 6/73	505	295	236	1400	185	165	39	Nil	7000	385	10 200
Sampling Point (22) — Seepage Pool on Northern Corner of Dam Wall											
10/ 4/73	400	290	335	3600	81	185	30	–	–	–	–
Sampling Point (23) — Sample 22, Flowing in Channel											
10/ 4/73	400	300	310	3300	84	190	30	–	–	–	–
Sampling Point (24) — Seepage Flowing, Meeting Sample 23											
10/ 4/73	440	280	300	4100	12	200	10	–	–	–	–
Sampling Point (25) — Seepage Flowing 10 m South of Sample 24											
10/ 4/73	370	260	295	3600	33	170	30	–	–	–	–
Sampling Point (26) — Seepage Pool of Sample 25											
10/ 4/73	340	240	260	3100	15	160	5	–	–	–	–

Date	Major Constituents, mg/l										
	Ca	Mg	Na/K	Fe	Al	Mn	Zn	HCO_3	SO_4	Cl	TDS
Sampling Point (29)— Seepage Flowing from Centre of Wall											
10/ 4/73	410	220	335	3000	29	155	8	-	-	-	-
Sampling Point (30) — Seepage Combination of Northern Flows											
10/ 4/73	350	260	255	3000	32	175	10	-	-	-	-
Sampling Point (31) — Seepage Flowing Centre (lower level)											
10/ 4/73	240	280	241	2300	45	160	14	-	-	-	-
Sampling Point (32) — Seepage, 0.5 m from Sampling Point (31) (lower level)											
10/ 4/73	380	220	480	2500	8	130	6	-	-	-	-
Sampling Point (33) — Seepage Flowing Along Pipe (Lower level)											
10/ 4/74	400	210	253	2500	37	150	7	-	-	-	-

ACID VALUES AND MINOR CONSTITUENTS

Date	Acid Value as $CaCO_3$	mg/l					pH
		Ni	Co	Cu	Cd	SiO_2	
Sampling Point (1) — QSA at Entrance to Quarry							
10/ 8/73	-45	-	-	-	-	-	6.0
Sampling Point (2) — QSA 200 m Inside Quarry							
10/ 8/73	1 950	-	-	-	-	-	3.05
Sampling Point (3) — QSA 250 m Inside Quarry							
10/ 8/73	3 760	-	-	-	-	-	2.95
Sampling Point (4) — QSE Northwest Quarry Face							
5/ 1/73	-	-	-	0.26	-	-	2.7
3/ 4/73	-	-	-	-	-	-	2.5
10/ 4/73	15 000	2.1	0.7	0.10	0.06	-	2.60
10/ 8/74	10 500	-	-	-	-	-	2.65
Sampling Point (5) — QSE Southwest Quarry Face							
10/ 8/73	9 050	-	-	-	-	-	2.70
Sampling Point (6) — QSE on Bench							
7/ 6/73	13 500	9.0	3.0	0.60	0.90	-	2.95
10/ 8/73	9 300	-	-	-	-	-	2.9
Sampling Point (7) — QSE + SQA							
10/ 8/73	6 000	-	-	-	-	-	2.95
Sampling Point (8) — Days Creek at Entrance to Quarry							
10/ 8/73	-35	-	-	-	-	-	5.40
Sampling Point (9) — Days Creek at 450 m Inside Quarry							
10/ 8/73	760	-	-	-	-	-	3.45
Sampling Point (10) — QSE+SQA+Days Creek 500 m In							
10/ 8/73	3 000	-	-	-	-	-	3.0

Date	Acid Value as $CaCO_3$	mg/l					pH
		Ni	Co	Cu	Cd	SiO_2	
Sampling Point (11) — QSE + QSA + Days Creek, 750 m In							
10/ 4/73	-	4.0	1.6	0.40	0.48	-	2.6
13/ 7/73	8 200	4.0	-	-	-	-	2.95
20/ 7/73	5 900	3.0	-	-	-	-	3.20
3/ 8/73	4 300	2.6	-	-	-	-	2.85
10/ 8/73	2 770	1.5	-	-	-	-	2.9
17/ 8/73	2 820	1.5	-	-	-	-	3.05
23/ 8/73	4 300	2.6	-	-	-	-	3.0
31/ 8/73	2 725	1.7	-	-	-	-	3.20
7/ 9/73	1 740	1.2	-	-	-	-	3.25
14/ 9/73	1 405	1.2	-	-	-	-	3.30
21/ 9/73	2 810	2.0	-	-	-	-	3.10
28/ 9/73	2 750	1.7	-	-	-	-	3.00
Sampling Point (12) — QSB at Entrance to Quarry							
10/ 8/73	210	-	-	-	-	-	3.5
Sampling Point (13) — QSB South Quarry Face							
10/ 8/73	7 350	-	-	-	-	-	2.85
Sampling Point (14) — QSB West Quarry Face							
10/ 8/73	4 250	-	-	-	-	-	2.90
Sampling Point (15) — QSB North Quarry Face							
10/ 8/73	8 550	-	-	-	-	-	3.10
Sampling Point (16) — QSB 350 m from Face							
10/ 4/73	-	2.5	1.1	0.10	0.40	-	2.6
13/ 7/73	10 200	3.4	-	-	-	-	3.0
20/ 7/73	7 500	2.8	-	-	-	-	3.2
3/ 8/73	6 800	2.8	-	-	-	-	2.80
10/ 8/73	5 670	2.3	-	-	-	-	2.80
17/ 8/73	6 000	2.4	-	-	-	-	2.95
23/ 8/73	7 250	2.8	-	-	-	-	3.0
31/ 8/73	5 670	1.0	-	-	-	-	3.05
7/ 9/73	2 700	1.5	-	-	-	-	3.1
14/ 9/73	3 880	2.8	-	-	-	-	3.02
21/ 9/73	3 125	2.2	-	-	-	-	3.05
28/ 9/73	3 125	1.7	-	-	-	-	2.90
Sampling Point (17) — QSC Run-off from Bench							
10/ 8/73	5 200	-	-	-	-	-	2.65

Date	Acid Value as $CaCO_3$	mg/l					pH
		Ni	Co	Cu	Cd	SiO_4	
Sampling Point (18) — QSC Seepage from Face							
10/ 4/73	-	3.5	1.8	0.4	0.5	-	2.50
10/ 8/73	10 800	-	-	-	-	-	2.95
Sampling Point (19) — Tailing Dam Seepage Storage (recycle)							
11/ 8/72	-	-	-	0.35	-	-	3.0
3/ 4/73	-	-	-	-	-	-	2.4
10/ 4/73	5 000	0.3	0.3	0.15	<0.01	-	2.5
7/ 6/73	5 050	0.6	0.5	0.20	0.04	85	2.75
13/ 7/73	5 200	0.4	-	-	-	-	2.9
20/ 7/73	5 200	0.8	-	-	-	-	3.0
3/ 8/73	4 950	0.5	-	-	-	-	2.52
10/ 8/73	5 000	0.6	-	-	-	-	2.60
17/ 8/73	4 950	0.5	-	-	-	-	2.65
23/ 8/73	5 600	0.6	-	-	-	-	2.80
31/ 8/73	4 870	0.5	-	-	-	-	2.75
7/ 9/73	4 900	0.7	-	-	-	-	2.70
14/ 9/73	4 700	0.7	-	-	-	-	2.70
21/ 9/73	4 900	0.4	-	-	-	-	2.65
28/ 9/73	4 700	0.4	-	-	-	-	2.55
Sampling Point (20) — Tailing Dam Seepage Storage (static)							
10/ 4/73	6 500	1.4	0.8	0.7	<0.01	-	2.2
Sampling Point (21) — Tailing Dam Upper Pond							
10/ 4/73	-	3.2	0.8	0.5	0.14	-	2.40
7/ 6/73	4 800	3.0	1.0	0.4	0.12	140	2.80
10/ 8/73	5 000	-	-	-	-	-	2.60
Sampling Point (22) — Seepage Pool on Northern Corner of Dam Wall							
10/ 4/73	-	0.5	0.3	0.07	<0.01	-	2.7
Sampling Point (23) — Sample 22 in Channel							
10/ 4/73	-	0.6	0.4	0.07	<0.01	-	2.7
Sampling Point (24) — Seepage Meeting Sample Point 23							
10/ 4/73	-	0.2	0.3	0.06	<0.01	-	2.9
Sampling Point (25) — Seepage Flowing 10 m South of 24							
10/ 4/73	-	0.2	0.3	0.04	<0.01	-	2.9
Sampling Point (26) — Seepage Pool of Sample Point 25							
10/ 4/73	-	0.2	0.3	0.04	<0.01	-	2.8
Sampling Point (27) — Seepage Flowing in Channel on Southn. Cnr. of Wall							
10/ 4/73	-	0.4	0.3	0.07	<0.01	-	2.7

Date	Acid Value as $CaCO_3$	mg/l Ni	Co	Cu	Cd	SiO_4	pH
Sampling Point (28) — Seeping, Flowing into Sampling Point 27							
10/ 4/73	-	0.2	0.2	0.07	<0.01	-	2.6
Sampling Point (29) — Seepage Flowing from Centre of Wall							
10/ 4/73	-	0.2	0.3	0.06	<0.01	-	2.8
Sampling Point (30) — Seepage Combination of Northern Flows							
10/ 4/73	-	0.3	0.3	0.07	<0.01	-	2.6
Sampling Point (31) — Seepage Flowing Centre (lower level)							
10/ 4/73	-	0.3	0.4	0.15	<0.01	-	2.5
Sampling Point (32) — Seepage 0.5 m from 31 (lower level)							
10/ 4/73	-	0.1	0.4	0.07	<0.01	-	3.0
Sampling Point (33) — Seepage Flowing Centre along Pipe (lower level)							
10/ 4/73	-	0.1	0.3	0.07	<0.01	-	2.6

APPENDIX C

HYDROLOGICAL AND METEORLOGICAL DATA IN MINE AREA

C1: ESTIMATE OF MEAN MONTHLY FLOWS

(data from E & WS 2717/73 report by D. Kingston

Month	from Lobethal Lagoons (cusecs)	Dawesley Creek including effluents (cusecs)	Dawesley Creek including effluents (m^3)	AMD (cusecs)
January	0.01	0.03	2 280	0.06
February	0.01	0.02	1 375	0.05
March	0.15	0.17	12 940	0.06
April	0.23	0.25	18 410	0.15
May	0.29	0.43	32 700	0.26
June	0.29	0.58	42 710	0.30
July	0.38	2.07	157 500	0.32
August	0.42	3.64	277 000	0.29
September	0.25	2.34	172 300	0.25
October	0.20	0.98	74 570	0.21
November	0.16	0.33	24 300	0.11
December	0	0.04	3 045	0.07
Total	2.34	10.88	819 130	2.13

C2: MEASURED FLOWS JUNE TO SEPTEMBER 1973
(Graphically shown in Fig. C1 and C2)

READINGS TAKEN BY W. HUTTON

Gauging Point Flows l/sec

Date	GP 1*		GP 2	GP 3	GP 4		GP 5
	a*	b	b	c	a	b	b
20/6/73	(10)				(5)		
25/6/73			(6)	(6.9)			
2/7/73			1.0	0.32			3.25
3/7/73			1.0	0.38			2.6
4/7/73		10	1.0	0.38			2.6
5/7/73		12.5	1.0	0.35		15.3	2.6
6/7/73		12.5	1.0	0.35	27	15.3	2.6
7/7/73							
8/7/73							
9/7/73		13.8	1.0	0.33	30	25.5	3.6
10/7/73		10	1.0	0.32	24	18.3	3.75
11/7/73		8	1.25	0.28	22	17.2	3.25
12/7/73		6	1.0	0.25	18	13.9	3.5
13/7/73		6	1.0	0.25	18	13.9	3.25
14/7/73d*							
15/7/73							
16/7/73		>170		>8	>1000		
17/7/73				2.3			
18/7/73				1.5			
19/7/73					70		
20/7/73		54		>5			3.25
21/7/73							
22/7/73							
23/7/73	50			0.95			
24/7/73d				0.91			4.55?
25/7/73	>1000			>6	180		4.55?
26/7/73	100	99		1.9	100		4.55?
27/7/73	50	51		0.91	50		3.25
28/7/73							
29/7/73							
30/7/73	35	30		0.76	130		3.25
31/7/73	28	26		1.1			4.55?
1/8/73	36	32		0.82			3.25
2/8/73	36	32		0.72			3.25
3/8/73d	30						
4/8/73							
5/8/73							
6/8/73	130			1.2			
7/8/73d	170			2.3			
8/8/73d	>1000			>10			
9/8/73	> 140						

Date	GP 1		GP 2	GP 3	GP 4		GP 5
	a	b	b	c	a	b	b
10/8/73	200			2.8			3.25
11/8/73							
12/8/73							
13/8/73	45			0.95			3.25
14/8/73	50			1.0			3.25
15/8/73	50			0.9			3.25
16/8/73	50			0.65			3.25
17/8/73	50	50		0.65			
18/8/73							
19/8/73							
20/8/73				0.23			
21/8/73				0.36			
22/8/73				0.33			3.0
23/8/73	28	30		0.32			3.25
24/8/73				0.32			3.25
25/8/73d							
26/8/73				>10			
27/8/73							
28/8/73							
29/8/73	(130)						3.0
30/8/73	(130)			(1.9)			3.25
31/8/73	(100)	45		(1.5)			3.25

* Legend
GP 1 Dawesley Creek upstream from mine.
GP 2 Days Creek including QSE + QSA.
GP 3 QSB.
GP 4 Dawesley Creek leaving mine.
GP 5 Tailings Dump seepage.
GP 6 Dawesley Creek before QSD.

a staff gauge measurement.
b V-notch measurement.
d

14/7/73	36 mm of rain
24/7/73	17 mm of rain
3/8/73-5/8/73	10 mm of rain
7/8/73	9 mm of rain
8/8/73	16 mm of rain
9/8/73	7 mm of rain
13/8/73	1 mm of rain
14/8/73	2 mm of rain
15/8/73	2 mm of rain
25/8/73-26/8/73	24 mm of rain

Bracketed values estimate only.

c flow ex-pipe.

C4: MEAN MONTHLY RAINFALL AND EVAPORATION RATES FOR BRUKUNGA

Month	Rainfall		Tank Evaporation		Practical Evaporation		Nett Evaporation	
	Points	mm	Points	mm	Points	mm	Points	mm
January	56	14	820	208	697	177	641	163
February	45	11	760	193	646	164	601	153
March	59	15	600	152	510	130	451	115
April	142	36	410	104	349	89	207	53
May	250	64	210	53	179	46	- 71	- 18
June	283	72	170	43	145	37	-138	- 35
July	300	76	160	41	136	35	-164	- 42
August	275	70	210	53	179	46	- 96	- 24
September	236	60	380	97	323	82	87	22
October	197	50	500	127	425	108	228	58
November	105	27	610	155	519	132	414	105
December	66	17	880	224	748	190	682	173
Total	2013	512	5710	1450	4856	1233	2842	722

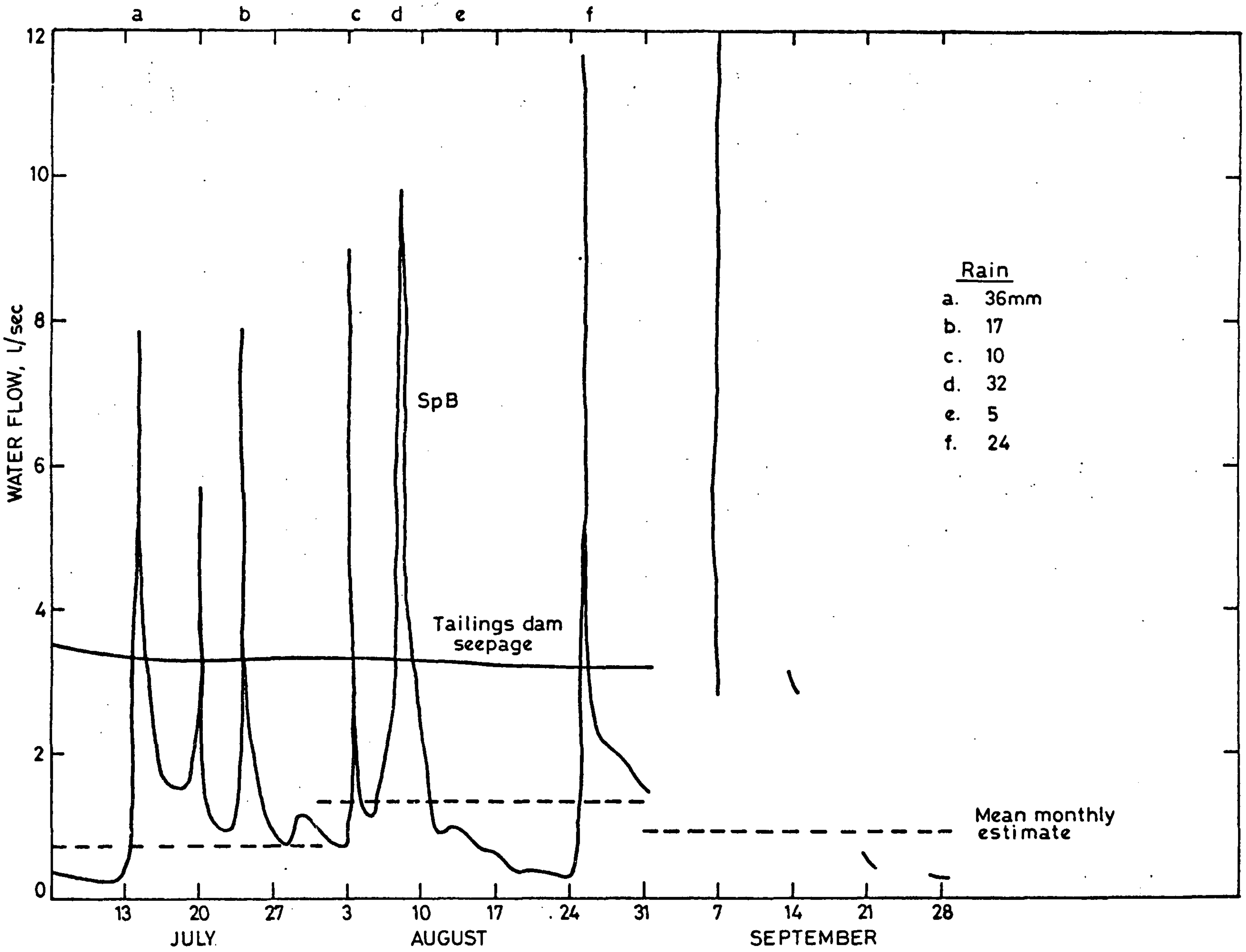


FIG. C1 : FLOW DATA FOR QUARRY SEEPAGE AND TAILINGS DAM

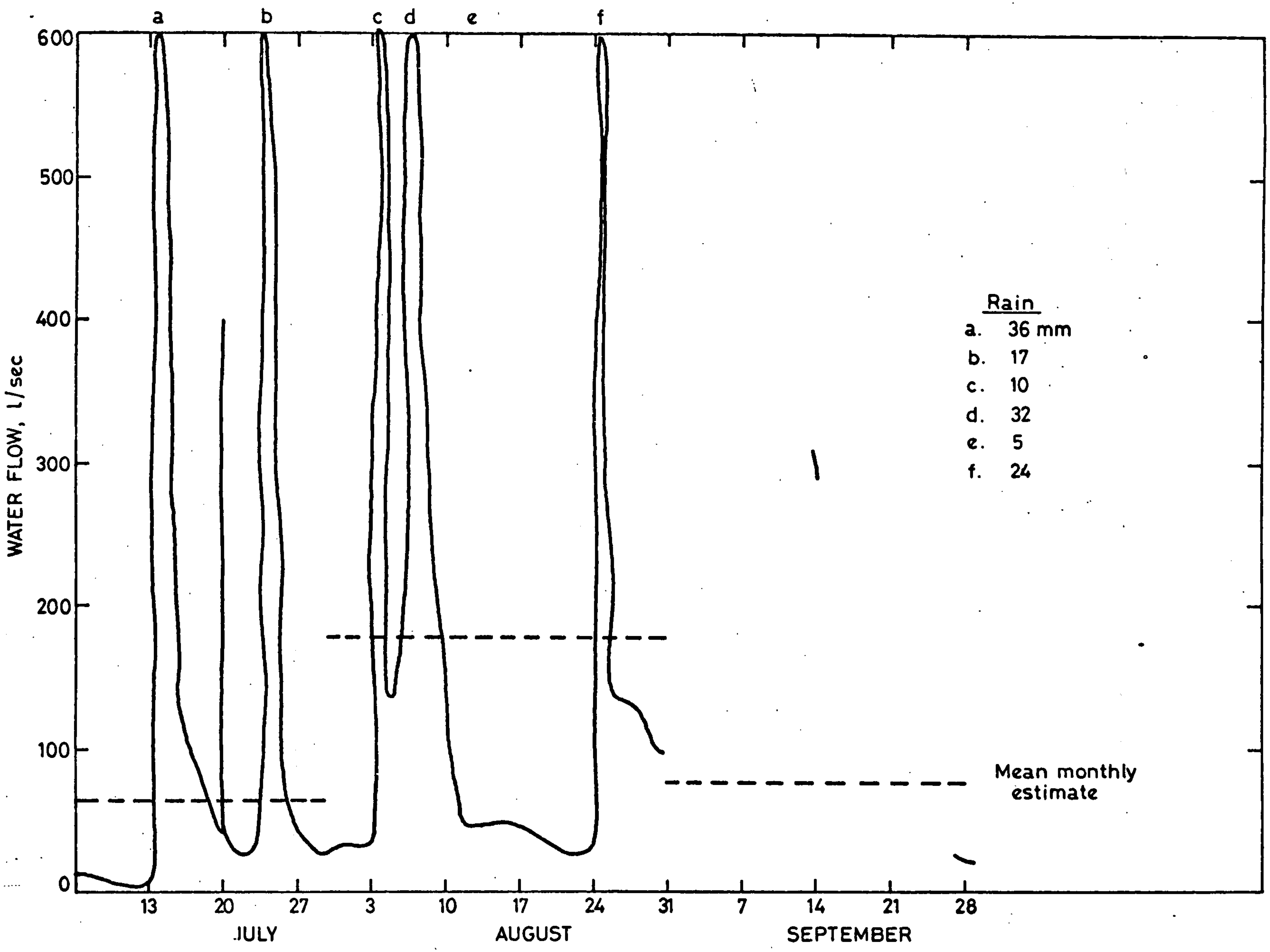


FIG. C2: FLOW DATA FOR DAWESLEY CREEK

PLATE 1: Seepage (QSC) running along quarry bench in foreground; eventually reaching Dawesley Creek in background

PLATE 2: Tailing dam seepages collected in channels; yellow encrustations and dark coloured algal growth on rocks.

PLATE 3: Open-cut quarry facing south from central position; benches to right and coarse waste dumps to left.

PLATE 4: Open-cut quarry facing south-west; bench containing pools of QSB in bottom half of picture.

PLATE 5: Dawesley Creek in central mine area in summer. Facing south yellow-brown precipitates in creek bed.

PLATE 6: Dawesley Creek in southern mine area. Channel cut to allow passage of creek water. Waste dump on left covering old creek bed.

PLATE 7: Seepage (QSD) entering Dawesley Creek in southern mine area from former creek bed below waste material at bottom right.

PLATE 8: View from centre of quarry facing east; quarry benches in foreground, disused plant in middleground; terraced face of tailings dam in background.

PLATE 9: Upper holding pond for AMD on tailings dam showing red-blue coloration of the oxidised drainage and quarry face in background.

PLATE 10: Holding ponds for AMD at base tailings dam. Facing south, the Dawesley runs along base of coarse waste dump on right.

PLATE 11: Holding pond for AMD from northern quarry. QSB enters through pipe at centre. The Dawesley, out of picture on the left, runs adjacent to the pond.