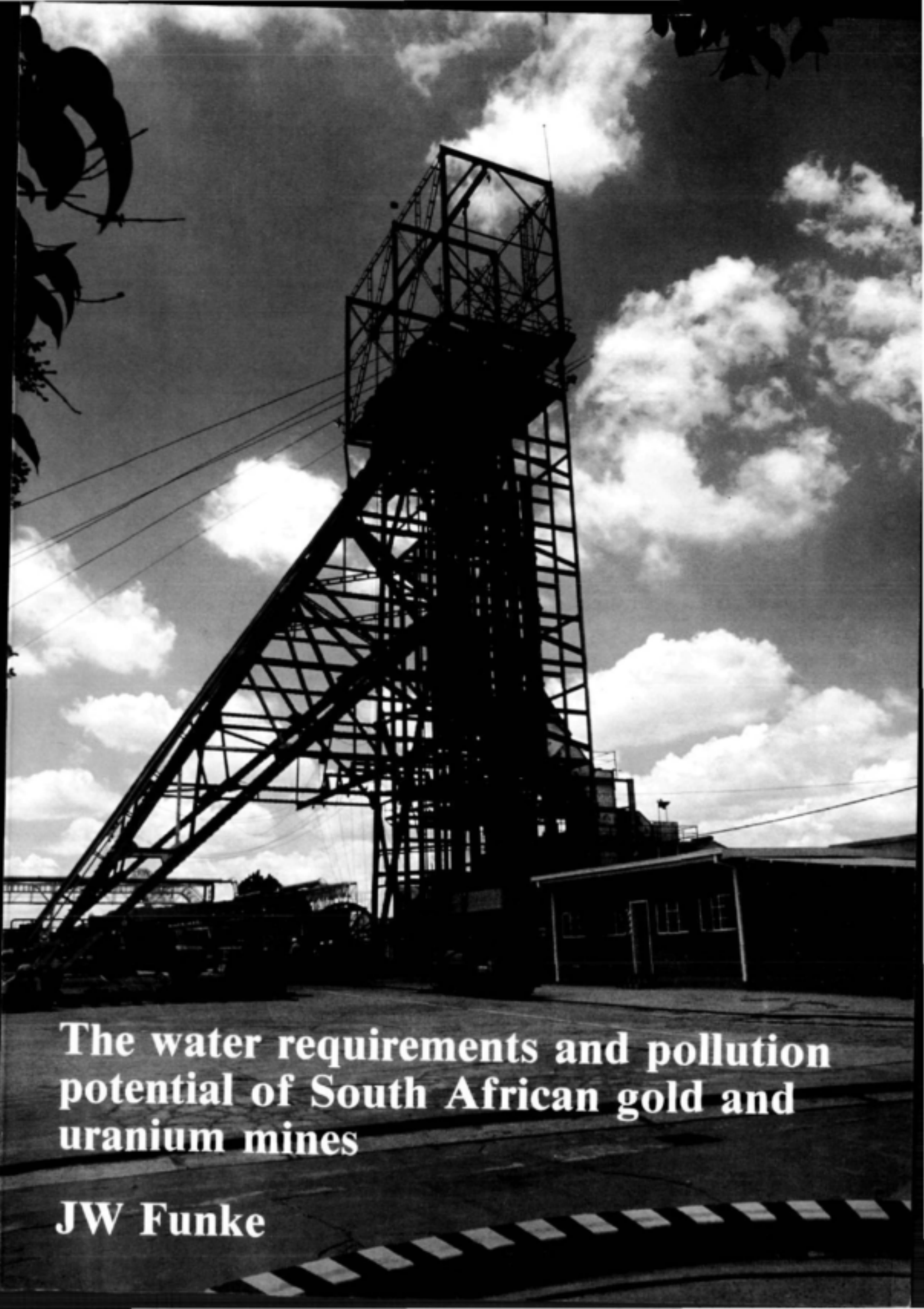

The water requirements and pollution potential of South African gold and uranium mines

JW Funke

THE WATER REQUIREMENTS

AND

POLLUTION POTENTIAL

OF

SOUTH AFRICAN GOLD AND URANIUM MINES

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EXECUTIVE SUMMARY

The 43 larger South African gold and uranium mines (Fig 1) in 1989 disposed of approximately 120×10^6 t of milled and processed ore and about 30×10^6 t of mined waste rock. The 36 small mines in the eastern Transvaal mill and process about $1{,}1 \times 10^6$ t/a of ore. Another about 90×10^6 t/a of low-grade sand dumps and slimes dams is reprocessed for the recovery of gold, uranium and pyrite and the tailings are deposited as slimes deposits or used as backfill material for worked-out stope areas underground.

The major gold-producing areas in South Africa are, from east to west:

- The eastern Transvaal with 36 small mines, many of them not active, stretches from Gravelotte in the north to the mining district of Barberton in the south.

- The Evander gold-field with 4 active mines drains via the Waterval River into the Vaal River upstream of the Vaal Dam.

- The East, Central and West Witwatersrand, which stretches from Heidelberg to the south of Randfontein, with at present 6 active mines and a number of tailings reprocessing operations, draining into the Vaal Barrage.

- The West Rand and the Far West Rand with 13 gold mines, situated in the dolomite area between Randfontein and Welverdiend, south of Carletonville, draining via the Mooi River into the Vaal River.

- The Orkney/Klerksdorp gold-field with 4 gold and uranium producers, draining into the Vaal River.

- The Orange Free State gold-field with 13 mines in operation and one inactive mine, draining into evaporation pans, with no effluent reaching the Vaal.

With the exception of the small gold-mines in the eastern Transvaal and the Evander gold-field, most mines situated further west have the potential of producing uranium as a by-product.

The gold- and uranium mines consume 19% of the electricity sold by Eskom.

WATER INTAKE AND QUALITY

About 50% of the total water intake by the mines consists of a low total dissolved salt (TDS) river water, either abstracted directly by the mines, or supplied by water board authorities such as the Rand Water Board, the Western Transvaal Regional Water Company near Orkney and the OFS Goldfields Water Board at Balkfontein, which treat the water to potable standards . The other water intake is mine water, pumped to the surface and used by the mines. Pan water which is made up of mine pumpage, metallurgical plant and sewage plant effluents and natural runoff is a major source of water supply to the OFS mines. The distribution of water use by mines is shown in Fig 2.

The quality of mine water varies with location, ranging from low TDS water in the Barberton district and at some Far West Rand and Klerksdorp mines to acid, high sulphate and hard water on the East and Central Rand to high chloride water at Evander and high TDS, chlorides and hard waters in the Free State gold-field.

The most objectional properties of mine service water are, in general, corrosivity, a tendency towards scale formation, large volumes of suspended solids and high temperatures.

Obtaining sufficient volumes of water is not a decisive cost factor for most mines. The major cost factors are the replacement of corroded pipes and equipment, the increase in power use to compensate for the reduced heat transfer at refrigeration plants or for pumping through scaled up pipes and the use of treatment chemicals. The cost to the gold-mining industry because of poor quality water is estimated to be in the order of R150 to R300 x 10^6/a.

Water low in suspended solids, TDS, chloride and hardness is required for cooling/refrigeration and for certain applications in the metallurgical plants. A potable supply is required for the about 500 000 persons working and living on mining property.

During 1983 the low TDS water supplied directly from the river by the water boards was used as follows:

- 50% for domestic supply;
- 40% for cooling/refrigeration, chilled mine service water supply; and
- 10% for metallurgical plants and gardens.

Half of the total water intake to the mines is used by the surface metallurgical plants to process about 120 x 10^6 t/a of ore and about 90 x 10^6 t/a of old sand and slimes residues for the recovery of gold, uranium, pyrite and sulphuric acid.

COOLING/REFRIGERATION

The cost for environmental control, that is, keeping the wet bulb temperature at stope face at or below 28 °C and providing a sufficient volume of dust-free air in a modern mine, constitutes as much as a third of the total costs for mining and may rise to about half on some deep mines. The basic elements of a mine cooling system are shown in Fig 12.

The total cooling/refrigeration capacity installed at the large gold-mines increased from 255 MW in 1974 to 1 213 MW at the end of 1989. About 60% of the cooling capacity has been installed at surface plants and the rest underground. As mines get deeper, the cooling/refrigeration capacity is expected to increase at an annual rate of 100 to 200 MW.

Water losses by cooling/refrigeration plants during 1989 were estimated to be:

21 100 M*l*/a for surface installations and

22 900 M*l*/a with ventilation upcast air to give a total of

44 000 M*l*/a which was equivalent to about 0,29 m^3/t of rock mined.

The total water lost by all the gold-mines through evaporative cooling, wind losses and blowdown and carried out of the mine by ventilation upcast air, is less than the volume (55 000 Ml/a) lost by a 3 600 MW power station. Reducing the intake volume of low TDS water as make up for cooling can be achieved by methods such as:

- operating the surface mine service water pre-cooling towers during night and early morning and storing the cooled water in large storage dams in order to reduce the cooling duty of the refrigeration plant;
- recovering potential energy by Pelton wheel turbine, thereby reducing heat gain on downgoing chilled water (2,3 °C/km);
- insulating chilled water pipes, thereby reducing cooling losses at unwanted areas;
- recirculating underground ventilation air, thereby eliminating the heat gained by downcast air of 4 °C/km;
- insulating rock surfaces of intake airways;
- reducing the heat load originating from rock surfaces by backfilling worked-out stopes.

Backfilling worked-out stopes with conditioned slimes appears to be an effective method of decreasing heat load and reducing cooling requirements in a mine, while simultaneously increasing mine safety by reducing damage by rockfalls and rockbursts. Stope face backfills have been found to reduce the heatflow by up to 50%. It has also been shown that backfilling mine-wide can reduce the overall mine heat loads by between 10% and 25%, thereby reducing the capital and operating costs of refrigeration. During 1989 18 mines were pumping about $3,6 \times 10^6$ t of tailings as backfill into worked-out excavations. This is only 3% of the total present annual milling rate of ore. It is expected that within the next ten years the annual rate of backfill will increase to about 15×10^6 t.

ELECTRICAL POWER SAVINGS

Electrical power is costly, particularly during peak demand periods. Electricity savings methods are, therefore, introduced at mines on an increasing scale. Examples are:

- *the Pelton wheel turbine* which recovers 50% to 60% of the down-going potential energy, is widely used to reduce the energy required to pump warm service water back to the surface.
- *the use of ice* rather than chilled water reduces the volume of the resultant hot mine service water to about 20% thereby reducing pumping costs for returning it to the surface. The first 1 000 MW ice plant is already operating at Harmony in the Orange Free State, and a second 6 000 MW ice plant is being constructed at ERPM, its commisioning being scheduled for 1990.
- *the hydro-lift mine cooling system* to convey slurry ice for mine cooling. As the pressure in the rising main, carrying the warm dirty mine water, is effectively balanced by the down-going slurry ice, the pumping power to overcome friction is minimal, less than 10%, compared to conventional pump systems for returning warm mine

water to the surface. The first hydrolift system is being installed at a local gold mine and is expected to be operational by mid-1990. A major advantage of the system is its ability to convey dirty mine water to the surface without the need for underground separation into clear water and mud, thereby eliminating capital and operating costs for underground settlers, clear water sumps, clear water pump chambers, mud water pumps;

- *hydro-power* utilizes gravity as a source of power, thereby conserving electricity. Stoping equipment such as rock-moving water jets, blast hole cleaners, hydraulic props, small Pelton turbines to replace motors driven electrically or by compressed air, e.g. for scraper winches, can all be powered directly with plain water under pressure (Fig 20). For rockdrills, compressed air power is relatively wasteful and inefficient. Two alternative powering technologies utilizing water free of suspended solids, have been developed by the Chamber of Mines Research Organization (COMRO) namely electro-hydraulics and hydro-power.

Electro-hydraulics refers to a system whereby high-pressure water or oil-in-water emulsion is obtained from electrically driven pumps which pressurize the service water to a required working pressure of typically 16 MPa. Electro-hydraulics is the technology to be applied exclusively down to a depth of 900 m, where the hydrostatic pressure of the water column is insufficient, or at greater depths where it would be inconvenient to have high-pressure pipes installed.

Hydro-power technology is applicable at depths greater than 1 500 m, where the water is reticulated to the working areas directly from the shaft water columns, at pressures exceeding 15 MPa. Presently available hydraulic rockdrills, already being used by the mines on a large scale, require 2% additives to be emulsified in water to control wear and corrosion.

Clear-water-powered rockdrills have been developed and successfully tested and are expected to be commercially available by 1992.

The electrical energy consumed for conventional pneumatic rockdrilling is at least 15 kWh/t of rock mined and for electro-hydraulic drilling with emulsion only 1,5 kWh/t, whereas for hydro-powered drilling with water, no electrical input is needed.

Because of the reduced labour costs, the improved drilling rates at least double those obtained with pneumatic rockdrills, and the concomitant electrical power savings, the operating costs of hydro-power systems are lower. The initial capital outlay is about the same as for pneumatic systems. The cost benefits as well as the proven reliability of hydro-power are the reasons why many mines are installing hydro-power systems at an increasing scale, particularly at new extensions.

Non-explosive mining, based on a hydraulically operated (clear-water) impact ripper, is being developed by COMRO and a prototype has already been tested underground. While impact ripping was found to achieve good mining rates on fractured rock that occurs at the stope in deep mines, the development of secondary non-explosive methods to break isolated patches of unfractured rock is still necessary. Methods of suppressing the dust generated by impact ripping also need to be developed because of intermittent high dust loads exceeding the legal limit. In the long term introduction of this hydro-powered rockbreaking system will revolutionize underground mining by changing it from the present one-shift working system to continuous mining around the clock. The expected result will be:

- increased labour productivity,
- a more even load on transport and hoisting facilities,
- concentration of mining with the consequent potential for rationalizing infrastructural ventilation, and reduced cooling requirements.

In addition, waste rock could be sorted at the face and packed in the back area as support.

IMPROVING WATER QUALITY

Drilling and blasting creates dust which must be suppressed by water sprays. The inadequate removal of the abrasive solids in underground settlers is regarded as the major water quality problem at a number of mines. Most settlers cannot cope with the rapid and extreme fluctuations in hydraulic and solids load rates. To improve solids removal and consistently obtain clear water for low-tolerance equipment such as hydro-powered rockdrills (less than 5 mg/*l* suspended, abrasive solids) COMRO in cooperation with mining groups, individual mines, consultants and other organizations is developing and testing new equipment such as high-rate settlers and floating media filters.

A 1 M*l*/d demonstration plant incorporating unit processes for neutralization, settling, filtration, desalination by slurry precipitation and recycle reverse osmosis and disinfection (Aquarius plant) is to be buildt on surface to treat 13 *l*/s of mine waste water pumped from underground. All unit processes have been tested on pilot-plant scale to prove their practical feasibility. This plant is to be constructed in such a way that it can be dissembled and moved to mine sites at different geographic locations with different types of mine service water. The unit processes are shown in Fig 47.

WAYS OF INCREASING THE SUPPLY OF LOW TDS WATER

A large reservoir of low TDS water is available on the mines in the form of treated sewage. The generous use of water at mine hostels of between 0,2 to 0,7 m^3/person.d - an average of about 0,39 m^3/person.d - results in a raw sewage from mine hostels with only a moderate increase in TDS compared to the original water intake.

The upgrading of this large source of low TDS water presently is one of the cheapest ways of augmenting the supply of good quality water on a large scale or of alternatively saving on fresh-water intake. The volume available may be sufficient at many mines to operate sensitive hydro-power equipment such as rockdrills.

Supplementing low TDS potable water supply with low TDS dolomitic water is already practiced by one mine. This appears to be the logical solution for Far West Rand mines which, in spite of having to pump out large volumes of low TDS water to keep the mines operational, still buy potable water which has to be supplied from a source about 90 km away.

Desalination of high TDS mine water by membrane processes such as electrodialysis or reverse osmosis has not been introduced on a large scale, but has been demonstrated on pilot scale to be technically feasible for non-scaling NaCl type OFS mine water. Freeze desalination in combination with mine cooling by slurry-ice has the potential for producing desalinated ice from mine water, regardless of the composition and salinity of the feed water. A full-scale 1 000 t/d plant is being constructed at Vaal Reefs near Orkney and is expected to be commissioned during 1990.

Fan drift condensate from ventilation upcast air is of similar or better quality than water supplied to the Free State mines via a pipeline of about 90 km by the Goldfields Water Board from the Vaal River at Balkfontein near Bothaville. The economic feasibility of recovering fan drift condensate on a large scale by cooling the hot upcast air has not yet been investigated.

Comparative cost figures of providing low TDS water from various sources to the mines are as follows (1990):

	c/m^3	
Rand Water Board	52,75	(includes pumping costs)
OFS Goldfields Water Board	79	" " "
Upgrading treated sewage	12	(operating costs only)
Augmenting with low TDS dolomitic water	12	" " "
Desalination by RO or EDR	50	" " "

UNDERGROUND ENVIRONMENT

All reefs mined for gold contain sulphur in varying amounts ranging from 1% S to pockets of 9% S, mainly as pyrite FeS_2 but also arsenopyrite. Consequently, the wet and oxidative conditions underground are conducive to transforming pyrite sulphur into sulphuric acid.

Due to the leaching of ore, mine service water may contain up to 11 000 mg/*l* TDS. While sulphate is the predominant contaminant of mine service water in most Transvaal mining areas, high concentrations of Na and Cl are the main constituents of underground water in the Evander and OFS gold-fields.

Corrosion and blockage by scale-formation pose a severe problem particularly in the OFS gold-mines. Dissolved oxygen is apparently a major factor of corrosion by mine water with a pH higher than 5. Chloride is regarded as the main culprit for pitting corrosion of stainless steel.

The replacement of corrosion-sensitive mild steel pipes by e.g. epoxy or glass-fibre-coated mild steel or by plastic material corrosion-resistant pipes might be cost-effective in spite of the higher costs for piping, since installation costs constitute the far higher cost factor, accounting for about 75% of the total costs for an underground piping network.

The reduction of underground acid formation by inhibiting acid-forming bacteria on a shaft-wide trial was not successful since the uncontrolled inflow of fissure water made it impossible to control the dosage of inhibitor.

RADIOACTIVITY

Uranium concentration levels in Witwatersrand and Far West reefs are about 100 times lower than in those mined for uranium in Canada, the USA or Australia and range from 50 g/t in the east to 200 g/t U_3O_8 in the west and reach 640 g/t at Afrikander Lease near Klerksdorp and 600 g/t at the Beisa mine at the southern end of the OFS gold-field.

Recently acquired knowledge of radiotoxicity has shown that uranium, thorium and polonium nuclides are of higher toxicity than ^{226} radium. Release of water-borne radioactivity, measured as ^{226}Ra, from old dry gold tailings deposits on the Central Rand apparently does not pose a threat to surface waters accessible to the public, since ^{226}Ra is effectively retained, most probably by adsorption.

While ^{226}Ra is readily adsorbed on to particulate material such as sand, clay or sewage sludge and is thus removed from clarified water, this is not the case with uranium which stays in solution. Uranium can, therefore, be transported with mine service water through the mine but also with mining effluents on the surface. The reuse of dolomitic water from the vicinity of mines should therefore be subjected to approval by prior testing for radioactivity.

High values for radionuclides have been found in mine service water, in water from a dolomite compartment adjacent to a mine and in effluent streams leaving mining properties. Since at one stage, before 1960, 26 mines were feeding 17 uranium recovery plants with ore it can be assumed that radioactive contamination is widespread, irrespective of whether uranium was recovered at metallurgical plants or not.

Lime softening, ion exchange and reverse osmosis have been demonstrated as methods for effectively removing radium from water (93% to 97%). With some modifications, these methods are also capable of removing uranium from water at reasonable costs. Since the radioactive matter will be concentrated with the sludge or with the brine or reject stream, a safe method for the disposal of the radioactive concentrate must be devised.

The experience obtained to date on underground concentration levels of radon daughters has shown that ventilation flow rates necessary for heat control do in some instances keep the radon daughter levels acceptably low, but in other instances not. It cannot be assumed that because a mine is well ventilated no radon daughter problems at will occur. Also, there are very few and probably no adequate routine surveillance programmes in situ to verify this.

The Council for Nuclear Safety, established during 1988, has in consultation with the Government Mining Engineer undertaken to establish the acceptable risk levels for the mining and processing of uranium- and thorium-bearing ores and the burning of uraniferous pyrite in sulphuric acid plants.

Burial of the contaminated structural material (steel, bricks, ceramic Raschig rings, rubber lining, etc) of a now dismantled sulphuric acid plant has been accepted as the method of disposal. There are 10 active sulphuric acid plants at local gold-mines roasting pyrite to produce acid, which some day will be confronted with the same problem.

An investigation by the Council in cooperation with the Department of Water Affairs into problems associated with effluent discharges from mining operations is under way. Results of this investigation to date indicate the need for controls.

Since many communities obtain their drinking water from underground sources which may intercept strata containing radioactive materials e.g. near Warmbaths, in the Transvaal, in the North-Western Cape or in the Karoo (Beaufort West) a radiological survey is indicated for the underground water in these areas. Some of the coal reserves, as yet not exploited, are rich in uranium, e.g. those in the Springbok Flats.

METALLURGICAL PLANTS , WATER REUSE PROBLEMS

On the whole, metallurgical plants use about 50% of the total water intake of the mines.

Typical water intake figures at surface plants are:

- gold recovery only:	0,9 to 1,4 m^3/t of processed ore
- gold and uranium recovery:	2,5 m^3/t of processed ore
- gold, uranium and pyrite recovery:	3,0 m^3/t of processed ore
- gold, uranium, pyrite recovery and production of sulphuric acid:	4,0 to 4,5 m^3/t of processed ore
- H_2SO_4 plant, scrubber/cooling water in circulation:	11,0 to 14,0 m^3/t of acid

Of the eight active uranium recovery plants in 1990, seven use solvent (alamine and isodecanol dissolved in kerosene) for the extraction of uranium. Uranium extraction with solvent - after oxidative acid leaching - results in highly saline contaminated solvent (raffinate), of about 1,1 to 2,0 m^3/t of tailings processed. Constraints in reusing this raffinate in metallurgical processes are:

- the high solvent content of about 250 to 600 mg/*l* which adversely affects flotation of pyrite and gold recovery by the carbon-in-pulp process, and
- the high content of iron and aluminium in solution which, after neutralization with lime, results in a jelly-like hydroxide precipitate which is difficult to clarify.

Possibilities for reusing untreated raffinate are:

- recirculation as make-up for the counter-current leaching of ore with sulphuric acid,
- use as scrubber/cooling water in the Peabody scrubber of an H_2SO_4 plant, provided the chloride concentration is low, and
- use in the gold plant for acid cleaning of clogged filters.

Possibilities for reusing treated raffinate are:

- use after neutralization for re-pulping slimes for disposal as backfill material, and
- use after neutralization/intense aeration as make-up water for monitoring of aged slimes prior to pyrite flotation.

The high-density sludge process (HDS) which is in use at the ERPM gold-mine and at two local coal mines to neutralize acid mine water should be tested for the treatment of raffinate to strip off kerosene and for its capability to remove gelatinous metal hydroxides.

Because of the difficulties encountered with the unrestricted reuse of raffinate and because of the convenient and easy access to low TDS board or river water, most uranium producing mines dispose of their tailings, after neutralization, with double or even three times the volume of water used by gold plants. Consequently, pumping costs to deliver the tailings to the deposit sites are higher.

Management options may exist for closing the water circuit at metallurgical plants recovering gold, uranium and pyrite. A thorough investigation is essential to find methods for reducing the intake of fresh water, and for reducing pumping costs and the pollution potential of the resulting tailings.

TAILINGS DISPOSAL

Sand dumps, of which the last ones were deposited around 1960, are permeable, thereby permitting oxidation of pyrite to depths exceeding 10 m. Of the total sulphur originally present, about 25% has been leached out in the form of acidic ferrous and ferric sulphate over periods longer than 25 years.

Slimes dams which have come into use gradually since 1918, are deposited hydraulically; they consist of material milled to a fineness of $< 75\ \mu m$, with consequent low permeability. Oxidation of pyrite is limited to the outer 2 m layer of the deposit. All tailings deposited hydraulically are neutralized with lime.

Gold plant tailings are deposited at a water to solids ratio of about 1:1. Pollution of ground water by seepage is unlikely, particularly if gold slimes deposits have been built on impermeable soil, since about:

- 30% of the original water is returned to the gold plant via the toe channels;
- 20% interstitial water is retained, and
- the large surface area combined with the high net evaporation loss on the Witwatersrand of 850 mm/a assures high evaporation losses (Fig 31).

Combined gold/uranium plant/pyrite flotation tailings are deposited at a water to solids ratio of 2:1 to 3:1. The surplus water available is now larger in volume than the potential evaporation loss. Since stable tailings deposits cannot be built with such a vast surplus of water, water is decanted and transferred to evaporation dams, which are presently **not** protected against seepage (with one or two exceptions). At some plants raffinate is collected and piped separately to an unsealed evaporation pond.

Since almost all gold/uranium/pyrite tailings are deposited on dolomite and since pyrite is generally removed for conversion to sulphuric acid used for uranium leaching, sulphur is now present as water-soluble $CaSO_4$, $FeSO_4$ and $MnSO_4$. Ground-water pollution by seepage is, therefore, a real problem.

The pollution potential of gold slimes deposits is low.

The pollution potential of combined gold/uranium/pyrite flotation slimes deposits is high as is proved by the manganese pollution of the Vaal River near Orkney and of boreholes between the Buffelsfontein gold/uranium slimes deposit and the Vaal River.

THE ENVIRONMENTAL IMPACT

In the eastern Transvaal gold-field, seepage of arsenic from tailings might become a problem if the drainage water is not treated with lime.

On the Evander gold-field, chlorides and fluorides are the main objectionable constituents in mine drainage. However, most of the saline water from the four mines is retained in the Leeupan for evaporation, with only occasional discharge during the rainy season into the Waterval River. Since the water from this river is used extensively for irrigation, the actual contribution of the Evander/Secunda mining and industrial complex to the mineral pollution load of the Vaal Dam is low.

On the East Rand, dolomite is intercepted by the active Grootvlei mine on both sides of the Blesbokspruit, and by the defunct Sallies mine, west of the Blesbokspruit. Both mines are interlinked. Grootvlei has to pump out 24 300 Ml/a of water from both mines in order to prevent flooding of its underground workings.

The start-up of the new ERGO-Daggafontein operation in July 1987, utilizing effluent from ERGO-Brakpan, the processing of the 8 km^2 tailings dam from ERGO-Brakpan for the recovery of gold and the deposition of the reprocessed tailings at a more suitable adjacent site did not reduce the volume and salt load attributed by water pumped out from Sallies via the Rietspruit into the Klip River.

Of the total flow into the Blesbokspruit between Benoni and Nigel, only 22% is accounted for at Heidelberg. Water losses can be ascribed to:

- evaporation in the reedbeds which extend over 25 km,
- extensive use for irrigation, and
- seepage return into the dolomite.

There is a strong suspicion that the dolomite at Grootvlei is recharged with water, probably from the Blesbokspruit. A thorough hydrological survey is indicated.

Since Marievale mine, south of the dolomite on the Blesbokspruit, is dry and has to import water, the question arises whether it would be useful to pipe the water pumped out from Grootvlei to a point beyond Marievale in order to save costs for dewatering the Grootvlei mine. This should be investigated.

ERPM, the oldest active mine at the age of 94 years, is one of the deepest mines in the world, presently mining at 3 458 m. With the expansion programme under way, the final depth in the South-East shaft area is expected to reach about 4 250 m. In order to reduce the large volume requirement for low TDS water for cooling and to reduce pumping costs for returning hot water to the surface, ERPM is presently constructing a surface ice plant of 6 000 t/d, which will become the largest ice plant in the world. The first module will be commissioned by September 1990. Water sent down for cooling will thus ultimately be reduced to 2 200 Ml/a as compared with 11 000/a if chilled water only and no ice plant were to be used.

The ERPM mine at Boksburg pumps about 9 800 M*l*/a of acidic water through its South-West shaft from the defunct mines of the central Witwatersrand in order to prevent flooding of its own workings. Another 7 200 M*l*/a pumped out from the Hercules shaft. The contribution by three active gold mines, Grootvlei, ERPM and Durban Roodepoort Deep over a period of 16 months during 1988/89 to the flow and load of the Klip River has been as follows:

Flow	9%
TDS load	48%
SO_4 load	60%

The Klip River is the main contributor to the salt load entering the Vaal Barrage.

Controlled irrigation with mining effluent with $CaSO_4$ as its main constituent and with minimum leaching can possibly cause agricultural land to be utilized as a *salt sink* without severe reduction in crop yield, thereby reducing the mineral pollution of the Vaal Barrage water.

Pollution deriving from residue dumps is regarded as of low importance in this particular area. Of the total Klip River catchment area of 1 968 km, only 39 km^2 or 2% is covered by sand dumps (6 km^2) and slimes dams (33 km^2). Of the total Suikerbosrand River catchment area of 3 587 km^2, only 34 km^2 or 0,9% is covered by residue dumps. All of these dumps were built with a water to solids ratio of about 1:1 and many of them are built on clay. When slimes dams are freshly built, leaching of sulphate from the outer crust to a depth of about 2 m takes place. As numerous slimes dams are now being re-worked for the recovery of gold and pyrite ample analytical data are available which indicate that below 2 m no oxidation of sulphur took place. Consequently, the pollution potential of slimes dams is regarded as being low.

Sand dumps, of which the last ones were deposited in the early 1960s, have already lost about 25% of the sulphur present to depths of up to 10 m below the surface.

It has therefore been concluded that the amount of sulphur in residue deposits that still oxidizes at present to cause water pollution is low, particularly when compared with the pollution load deriving from mine pumpage and metallurgical plant operation.

Western Areas had been the mine with the highest pumping rate for dewatering, pumping to the surface close to 60 000 M*l*/a at its North Shaft and about 10 000 M*l*/a at its South Shaft. Until June 1986 the bulk of this water was returned into the dolomite from where it had been abstracted. The mine has now been granted a permit to dewater the Gemsbokfontein compartment. By mid-1989 pumping from the North Shaft had been reduced to about 28 000 M*l*/a. The water pumped out is disposed of at about equal rates via Kleinwesrietspruit and Leeuspruit and reach Lochvaal via the Rietspruit.

Western Areas is the first mine where the radioactivity levels of mine effluents are being measured by both the Atomic Energy Corporation and the mine itself. The results are reported to the Department of Water Affairs. Analytical data indicate that the dolomitic water entering the mine is contaminated with radioactive uranium. Water-soluble uranium has also been reported from the Westrietspruit and the Leeuspruit about 15 km downstream from the mining property.

Most of the 14 active mines on the West Rand and the Far West Rand and the Orkney/Klerksdorp mines intercept the overlaying dolomitic aquifer. In order to increase mine safety and lifespan, dewatering of the Venterspost and Oberholzer compartments commenced in the late 1950s and that of the Bank compartment in 1970.

To compensate for damages incurred due to dewatering, such as the drying-up of boreholes and damage caused by subsidence and sinkholes, two bodies were formed and financed by the State and the Chamber of Mines, viz.:

- The State Technical Coordinating Committee on Sinkholes and Subsidences on the Far West Rand, and
- The Far West Rand Dolomitic Water Association.

The effect of mining and ore processing on the deterioration of the quality of underground and surface waters is evident at many locations on the West Rand, the Far West Rand and the Orkney/Klerksdorp mines.

Uncontaminated dolomitic aquifer water constists of bicarbonate with calcium and magnesium as the major cations, and is characteristically devoid of any sulphate. Borehole waters in the mining area, however, and particularly in the vicinity of uranium tailings, are often severely contaminated by sulphate (up to 1 400 mg/*l*), Mn (up to 300 mg/*l*) and Zn (up to 40 mg/*l*). The contamination of dolomite water with sulphate does not restrict its use for irrigation.

The contamination of borehole and underground water by radioactive constituents can be suspected.

The effect of Far West Rand mining on the quality of the Mooi River before its confluence with the Vaal River is hardly noticeable.

Seepage of manganese from tailings dams via ground water into the Vaal River at Buffelsfontein seriously affected the quality of the water supplied by the Western Transvaal Regional Waterworks to Orkney and Klerksdorp. Remedial measures such as pumping out of polluted ground water and eliminating MnO_2 as oxidant in the acid leaching of uranium have gradually improved the quality of the Vaal River water supply to the waterworks. Ozonation and chlorination at the waterworks further reduced the Mn concentration in the potable water supply to acceptable levels.

Pollution of dolomitic water and its seepage into the Vaal River takes place, because of the surplus of water from tailings and evaporation areas at the Vaal Reefs complex.

None of the Free State gold-mines disposes of its water into a public water body except into pans usually owned by the mines.

Fissure water from the Beatrix mine serves as a water supply to the southern Free State gold-field mines, e.g. St Helena, Unisel, Pres Brand and Pres Steyn.

The closed-down uranium producing Beisa mine is to be re-opened as the Oryx gold mine. However, this mine will have to be dewatered first. Radioactivity of this pumped-out water will probably be higher than that of Beatrix since the ore body of Beisa contains 600 g/t of uranium compared with the 100 g/t or less of Beatrix. Whether or not any fissure water from these mines can be used as water supply without treatment to remove radiotoxic constituents should be established by a radiological survey.

RECOMMENDATIONS

1 Maintain the adequate removal of suspended solids from mine service water by settling/filtration to provide water free of abrasive solids for low tolerance equipment such as hydro-powered rockdrills, valves, flow recorders, high-pressure clear water pumps.

2 Upgrade the locally available large source of low TDS treated sewage from mine hostels to provide low TDS water for use underground and elsewhere.

3 Stop the use of treated low TDS sewage water for watering golf courses and gardens, as this water could be used with greater benefit at the mine.

4 Augment imported low TDS water supply with low TDS dolomite water where such is locally available and not contaminated by radioactivity.

5 Investigate the feasibility of recovering low TDS soft water from the moisture contained in the ventilation upcast air at certain Free State mines which have to pay the highest price for Vaal River water.

6 Because of the complex nature of corrosion, replace corrosion-sensitive mild steelpipes by epoxy-, glassfibre- or polyurethane-lined mild steelpipes or by plastic material pipes, or by stainless steel, or by hot dip galvanized steel, taking due consideration of the possible disadvantages of these materials.

7 Intensify research to remove residual uranium extraction solvents and to obtain a clarified neutralized, partly desalinated solution from raffinate, suitable for reuse. Processes to be investigated for the removal of gelatinous metal hydroxides and residual solvents should include:

  - the high density sludge process with its intense aeration,
  - sulphate removal by anaerobic conversion of SO_4 into H_2S with the possible precipitation of metal hydroxides as metal sulphides (feedstock to sulphuric acid production). This study has been carried out by the DWT of the CSIR.

8 Develop management options to close the water circuit at metallurgical plants, particularly those recovering gold, uranium, pyrite and sulphuric acid, in order to reduce the pollution potential of resultant high water to solids ratio tailings and the consequent expense of sealing such tailings and evaporation dams against seepage.

9 Where a high water to solids ratio for tailings cannot be avoided, seal off such tailings against seepage to prevent pollution of the underground dolomite water.

10 Carry out radioactivity surveys on:

- underground mine water in contact with uraniferous reefs,
- dolomite water in the vicinity of tailings dams, evaporation dams, or surface streams where radioactive contamination may possibly occur,
- pan water derived from Beatrix mine and from the dewatering of the Beisa mine, and
- selected tailings dams for radon and radon daughters.

11 Extend the radioactivity survey to water supplies from underground sources which are suspected of having contact with uraniferous reefs, eg at Springbok Flats, the Northern Cape and the Karoo.

12 Develop treatment processes to remove radioactivity from underground mine water and from effluents leaving mine property or from other affected water bodies.

13 Develop limits (recommended maximum contaminant levels) to cover the radioactivity levels for drinking water and for effluent discharges (General Standards).

14 Investigate the feasibility of reducing the salt load from neutralized mine effluents into the Vaal Barrage by irrigating with a low leaching fraction, thereby utilizing agricultural land as a *salt sink*.

15 There is insufficient information at many mines regarding the volume and composition and flow pattern of mine service water and effluent streams because of the lack of metering devices, regular and analytical procedures and of qualified staff; therefore, -

- carry out a thorough investigation into mine water and effluent circuits and optimum use of chemicals, e.g. of flocculants at underground settlers.

For this investigation and for the management of water systems, employ

- a qualified person as specialist water engineer, or
- an impartial outside consultant firm in order to alleviate the workload of mine management staff.

16 Avoid shifting the responsibility for installing, maintaining and altering water conveying reticulation systems, from one department to another.

1 INTRODUCTION

Gold and uranium deposits are always linked with pyrite (FeS_2) and other sulphide minerals. For example, the pyrite content of the ore body on the Witwatersrand varies from 3,2% S at Durban-Roodepoort Deep to 1,7% S at East Rand Proprietory Mines (ERPM) at Boksburg (Lloyd, 1981). Pockets containing as much as 9% S are occasionally mined. After dilution with barren rock, the milled ore contains about 1% S.

Exposure of pyrite to moisture and air results in the formation of acid mine drainage. Chemical oxidation, accelerated by bacterial action converts sulphide minerals such as pyrite into ferrous and ferric sulphate and into sulphuric acid. This reaction takes place underground as well as on the sand dumps and slimes deposits. Consequently, the discharge of highly saline mine effluents has resulted in a marked increase in the salinity of the water in the Vaal Barrage and further downstream.

Other examples of the impact of gold and uranium mining and processing on the environment are:

- the deterioration of the quality of the underground water in the dolomitic areas on the West and Far West Rand;

- the occurrence of sinkholes caused by water being pumped from the mines to prevent flooding; and

- the deterioration of the water quality in the Vaal River near Klerksdorp due to dissolved iron and manganese.

Although the quality of mine service water is not important for many mining activities, a good quality water with a low dissolved salts content is required for cooling underground, for the increasing use of hydraulic equipment and for some recovery operations. However, the volume of low total dissolved salts (TDS) water which can be supplied to mines, is restricted particularly during extended periods of drought.

The reuse of water by mines varies, due to restrictive factors such as:

salinity, acidity, corrosivity, abrasive fine solids, tendency towards scale formation, presence of organic solvents, bacterial contamination, and temperature.

This report outlines the magnitude and consequences of pollution by mines and related activities. Remedial measures taken by the mines to reduce the pollution load and low TDS water intake are dealt with. New developments which are taking place are described in some detail such as the use of particulate ice for mine cooling, the production of slurry ice from high TDS mine water for simultaneous cooling and desalination, the introduction of energy-saving hydropowered drills and hydrolift mine cooling systems. Those areas requiring further investigation or input of research are defined.

2 GENERAL INFORMATION

Gold and uranium mining and the on-site processing offer employment to about 500 000 people. The major gold producing areas in South Africa are, from east to west:

- The Eastern Transvaal, with 36 small mines, many of them not active, which stretches from Gravelotte in the north to the mining district of Barberton in the south.
- The Evander gold field, with 4 active mines, which drains via the Waterval River into the Vaal River upstream of the Vaal Dam.
- The East, Central and West Witwatersrand, which stretches from Heidelberg to the south of Randfontein, with at present 6 active mines and a number of tailings reprocessing operations, draining into the Vaal Barrage.
- The West Rand and the Far West Rand with 13 gold mines, situated in the dolomite area between Randfontein and Welverdiend, south of Carletonville, and draining via the Mooi River into the Vaal River.
- The Orkney/Klerksdorp gold field with 4 gold and uranium producers, draining into the Vaal River.
- The Orange Free State gold-fields with 13 mines in operation and one inactive mine, discharging their effluent into evaporation pans.

The location of the gold mines except those in the Eastern Transvaal, is shown in Fig 1. Most of the mines west of the small gold mines in the eastern Transvaal and the Evander gold field have the potential of producing uranium as a by-product. The 40 large mines situated between Evander in the Transvaal Highveld and Theunissen in the Orange Free State mill and process about 120 x 10^6 t/a of ore in addition to about 30 x 10^6 t/a of mined-out waste rock. The 36 small mines in the eastern Transvaal mill and process about 1,1 x 10^6 t/a of ore. About 90 x 10^6 t/a of low grade sand dumps and slimes dams are also reprocessed for the recovery of gold, uranium and pyrite and are redumped again as slimes deposits.

In total, the gold and uranium mines consume 19% of the electricity sold by Eskom.

1	Winkelhaak	17	Randfontein Estates (Millsite)	33	Buffelsfontein
2	Bracken	18	Randfontein Estates (Cooke)	34	Loraine
3	Leslie	19	Western Areas	35	Freddies
4	Kinross	20	Venterspost	36	Free State Geduld
5	Wit Nigel	21	Libanon	37	Western Holdings
6	Marievale	22	Kloof	38	St Helena
7	East Dagga	23	East Driefontein	39	Pres Brand
8	Grootvlei	24	Western Deep Level	40	Pres Steyn
9	SA Lands	25	West Driefontein	41	Unisel
10	Vlakfontein	26	Blyvooruitzicht	42	Free State Saaiplaas
11	ERPM	27	Doornfontein	43	Harmony
12	City Deep	28	Elandsrand	44	Beatrix
13	Crown Mines	29	Deelkraal	45	Beisa
14	Cons Main Reefs	30	Stilfontein	46	HJ Joel
15	Durban Deep	31	Hartebeesfontein		
16	West Rand Cons	32	Vaal Reefs		

Figure 1
Producing and developing gold mines
(members of the Chamber of Mines)

3 WATER USE BY MINES

The total water intake by the large mines in 1983 amounted to 337 x 10^6 m^3/a of which 47% was low in total dissolved salts (TDS) and the rest consisted of mine water, pan water and treated sewage (Chamber of Mines Research Organization, 1983). Low TDS surface water is supplied by the:

- Rand Water Board,
- Western Transvaal Regional Water Company,
- Orange Free State Goldfields Water Board, and
- direct withdrawal from the Vaal River.

High TDS water made up of underground water, slimes dams return water, metallurgical plant flows, pan water (which contains water pumped from the mine, metallurgical plant, sewage treatment plant and natural runoff) is a major source of water for the OFS mines. Some mines are dry and need additional water mainly underground for dust suppression and cooling. Other mines, in particular those in the dolomitic area of the West Rand and Far West Rand, have a surplus of fissure water which is pumped from the mines and run to waste.

Mines use water for the following purposes:

Underground:

- dust suppression
- stope cleaning
- mine cooling/refrigeration
- fire fighting
- drinking
- hydraulic equipment

On the surface:

- domestic (drinking, washing, sanitation)
- mine cooling/refrigeration
- ore washing, milling
- gold leaching and processing
- pyrite flotation, sulphuric acid production
- uranium leaching and extraction
- disposal of tailings

Of the total water intake (Fig 2),

49% is used at metallurgical plants;
27% at hostels, offices, houses;
22% underground; and
2% for power.

During 1983, the low TDS water supplied by water boards and directly abstracted from

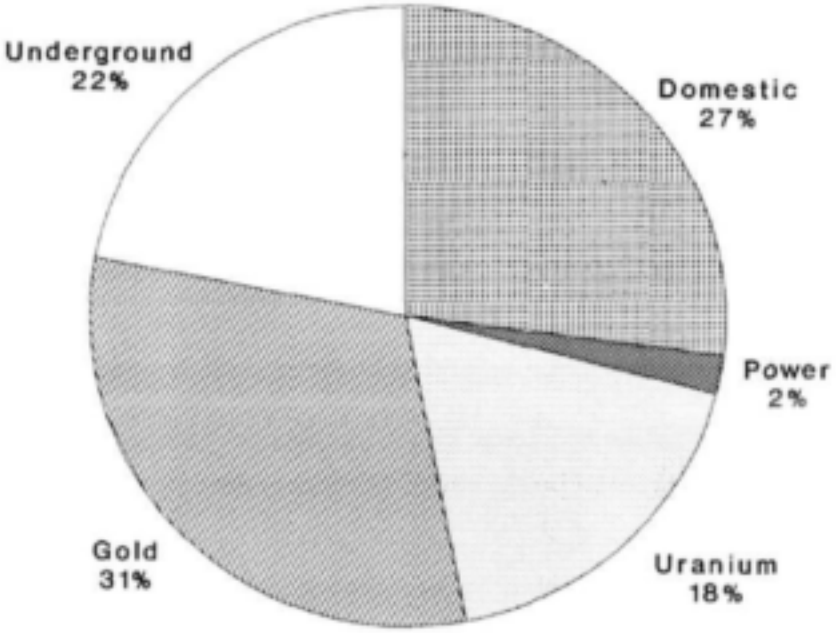


Figure 2
Distribution of water use by mines

rivers was used as follows:

- 50% for domestic supply, hostels, offices, houses;
- 40% for cooling/refrigeration, chilled mine service water supply; and
- 10% for metallurgical plants and gardens.

The supply of 70 542 M*l* of board or river water during 1983 to the then 490 000 strong work force, provided on the average 0,394 m^3/person.d, ranging from about 0,2 m^3/person.d to more than 0,7 m^3/person.d at the various mine hostels.

3.1 Rock handling underground and mine service water quality

Holes on the stope face are mostly drilled by pneumatic drills. The drilling creates large amounts of dust which are injurious to human health. For dust suppression, about 0,3 to 0,7 m^3 of water is supplied per ton of rock mined. Stope cleaning with low-powered, hand-held water jets or, more recently with mounted high-powered water jets, requires variable amounts of water.

The explosives (about 4 kg/m^3 of rock blasted) are charged into the drilled holes. On explosion, most of the chemicals are released as gases such as nitrogen, nitrous fumes, ammonia, carbon monoxide and carbon dioxide. The water-soluble nitrogen compounds have the disadvantage that they increase the chlorine demand for disinfection of mine service water and that they lower the pH of water by forming nitric acid.

Mine service water should be free of abrasive solids for drilling. A low calcium hardness is desirable to prevent scaling in supply pipes and on water jets. With the increasing use of chilled service water at the stope, usually the hottest working area in a mine, it is essential that this water should be disinfected - and therefore be free of pathogens.

To reduce the volume of fresh intake water, mine service water is collected and recirculated, at rates of between 2 to 15 m^3/t rock broken, depending on mine-specific factors like mine layout, water quality, natural inflow into the mine, underground cooling facilities and others (Schmitz, 1984).

Mine water near the stope is often acidic. Pyrite, commonly described in a simplified manner as FeS_2, is the chief source of acidity in mine water. When pyrite is exposed to air and moisture as in the mining environment, it is oxidized to ferrous/ferric sulphate and sulphuric acid. A simplified version of the reactions involved is as follows (Singer and Stumm, 1970):

$$FeS_2 + O_2 + H_2O \dashrightarrow Fe^{2+} + SO_4^{2-} + H_2^{+} \text{ acidity} \quad (1)$$

$$Fe^{2+} + O_2 \xrightarrow{\text{bacteria}} Fe^{3+} \quad (2)$$

$$Fe^{3+} + FeS_2 \dashrightarrow Fe^{2+} + SO_4^{2-} + H_2^{+} \text{ acidity} \quad (3)$$

During stage 1 fine-grained FeS_2 is oxidized, mainly by chemical reaction. At a pH of around 4,5 the abiotic oxidation of Fe^{2+} slows down and bacterial oxidation becomes primarily responsible for producing acidity and ferric hydroxide (stage 2). At a pH of less than 3, the increased solubility of iron results in an abiotic increased rate of oxidation of pyrite by Fe^{3+} (stage 3).

Reactions 1 and 2 require aerobic conditions, while reaction 3, the most acid-producing reaction, is essentially a hydrolysis reaction that can proceed without air. Another very important point is that without the presence of the highly acidophilic bacteria *Thiobacillus ferro-oxidans* reaction 2 could be halted. Inhibition of the bacterial activity would, therefore, greatly reduce acid formation as the rate of acid production will be determined by the slow abiotic oxidation of the Fe_2S by air (1) rather than by the rapid oxidation by Fe^{3+} (3). The acidic water formed in this way underground accelerates the breakdown of clay and carbonate minerals, thus increasing the TDS content and the corrosive properties of underground mine water. Mine service water resulting from ore leaching may contain up to 11 000 mg/*l* TDS (Stephenson, 1982 and Holton and Stephenson, 1983). While SO_4 is the predominant contaminant of mine service water in most mining areas, high sodium and chloride concentrations are the main constituents of the underground water of the Evander and OFS gold-fields.

Fresh and cold low TDS intake water is required to compensate for evaporation losses, increase in salinity due to leaching and concentration, and losses of water brought to the surface with the ore hoisted or pumped out. This volume amounts to 1 to 3 m^3/t of rock mined.

The mine service water picks up substantial amounts of suspended solids during drilling and dust suppression. These suspended solids have to be removed from the water because:

- Suspended solids impair the efficiency of high-pressure recirculating pumps and of high pressure sprays.

- Suspended solids impair the functioning of pressure reducing and sustaining valves and of other water flow regulating devices used in water distribution underground.

- Suspended solids impair the operation of low-tolerance hydro-power machinery, in particular hydraulic drills for which water, free of abrasive solids, is essential.

- Suspended solids promote fouling and loss of efficiency in heat exchanger tubes.

- Workers inhaling the water while it is being sprayed may suffer from health problems if the water contains dust.

- Suspended solids increase the consumption of chlorine used for disinfection of the water.

The dissolved salts concentration in the mine service water increases:

- as a result of evaporation through the ventilation and cooling system;

- as a result of the dissolution of water soluble constituents, particularly from ore dust with its large surface area;

- through inflow of saline fissure water;

- by oxidation of pyrite to sulphuric acid with the increased capacity for leaching and corrosion; and

- by addition of water treatment chemicals, e.g. for neutralization with lime to reduce acidity.

Although mine service water is not intended for human consumption, it should be non-toxic and if drunk in limited quantities, it should show no ill-effects. Highly saline water is laxative, may cause excessive sweating and affect blood pressure. The upper limits of dissolved salts are difficult to establish, and depend on the amount consumed. In the warm underground environment workers could drink as much as 3 *l* of water/shift which is much more than the normally accepted value of 2 *l*/d. Toxic substances include heavy metals, fluoride, chlorinated hydrocarbons, some algae, organic phosphates and some polyphosphates used for water treatment.

Pathogenic micro-organisms in mine service water can be of greater concern than dissolved salts. Unhygienic sanitary habits can promote the spreading of disease. To prevent outbreaks of gastrointestinal disorders, which may cause loss of life and a drop in production, disinfection of mine service water to conform to the bacteriological quality stated in the SABS specification (241-1984) for household water is the recommended approach for every mine. Removal of suspended solids and of ammonia reduces the amount of chlorine required for disinfection.

3.1.1 Removal of suspended solids

The bulk of suspended solids in mine service water is being removed by sedimentation at about 150 underground settler installations.

Underground settlers are generally operated under adverse conditions with rapid and extreme fluctuations in pH values, polyelectrolyte flocculant dosing, and solids and hydraulic rates. The variation of the suspended solids concentration in the inflow to an underground settler during a working day is shown in Fig 3 (Marsden, 1982). Two major peaks in the suspended solids loading occur during a working day and these coincide with peaks in hydraulic loading.

Several settler types are currently employed underground. These include simple *hole-in-the-ground settling dams, horizontal-flow settlers, vertical-flow settlers ranging from shallow rectangular to deep conical tanks, sludge blanket type settlers, and more recently, lamella plate settlers*. These settlers operate with varying degrees of success, particularly when hydraulically overloaded during the daily peak flows.

Chemicals are added to coagulate and flocculate fine suspended matter so that it settles more easily. Flocculants, typically at dosing levels around 1 mg/*l*, should be thoroughly mixed with the service water to be clarified to obtain adequate contact with as much as possible of the suspended matter. During various surveys at the same mine in 1977 and 1985 it had, however, been noted that flocculants were added in excessive dosings of around 1 000 mg/*l*, due to lack of knowledge and control. Oil spilling underground is also a major problem in the operation of underground settlers.

The levels of suspended solids in the settled mine water are generally high. During a four-month surveillance and evaluation programme of water quality, corrosion, fouling and

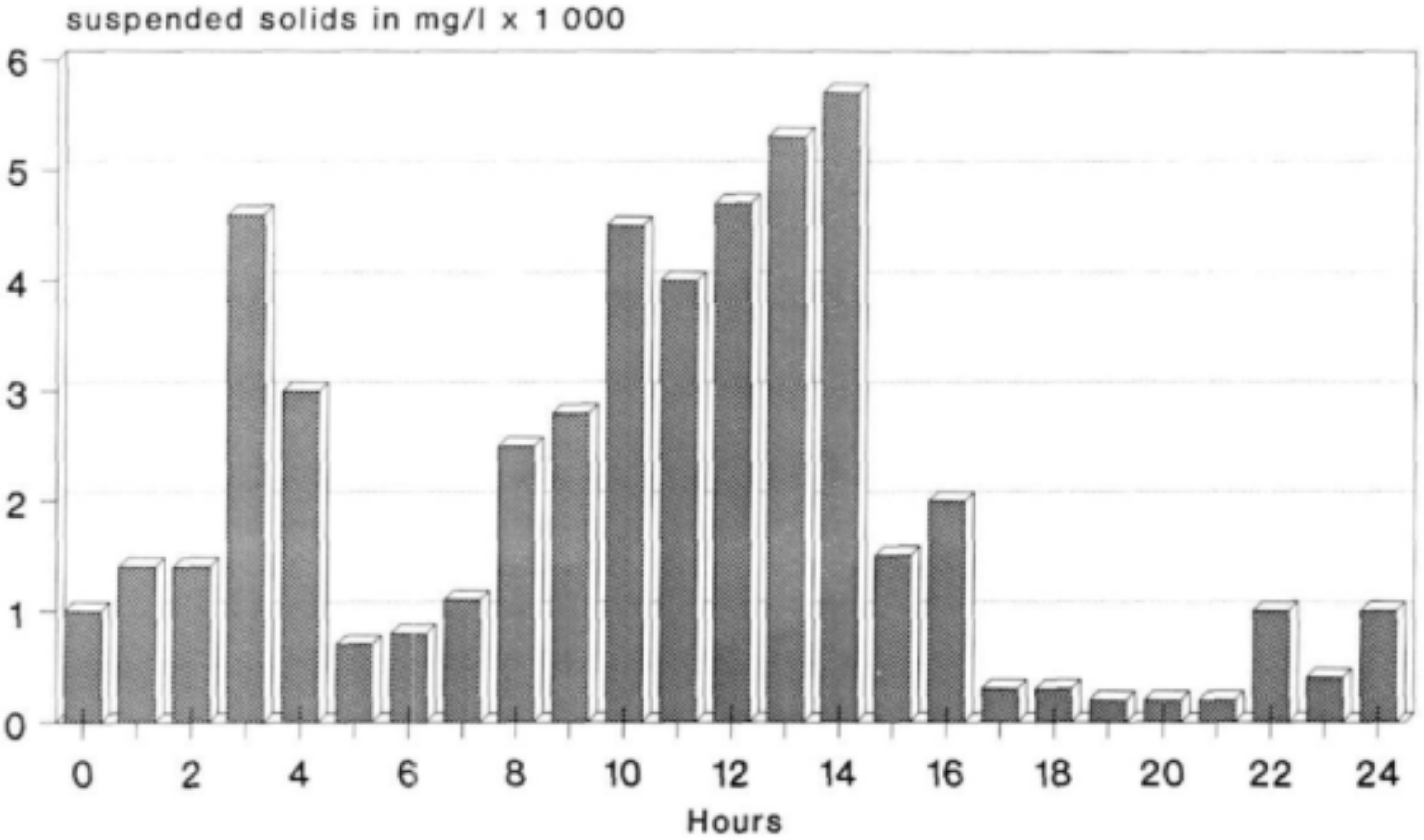


Figure 3
Daily variation in suspended solids content of inflow
to an underground settler

scaling in water reticulation systems at one gold mine, the level of suspended solids in the settled mine service water ranged from 41 to 333 mg/*l* (Howarth and McEwan, 1989).

Due to the significant costs involved in the use of a poor quality settled water, and the operational problems associated with poor solids removal, the Chamber of Mines Research Organization (Comro) in 1987 established a research programme with the main objective of providing a settling technology which could effectively cope with rapid and large fluctuations in solids and hydraulic loading rates. This resulted in the modification of a high-rate tube settler, developed in South Africa, for use underground. This settler comprises a rectangular or circular conical excavation, containing a number of individual settling tubes, submerged vertically below water level, each with a means of introducing the feed water. The feed water, of which the pH is adjusted, and which is dosed with a polyelectrolyte, is fed into the inner cylinder. The suspended solids are removed via the sludge blanket in the outer cylinder and drop-out of the bottom of the settling tube through the annular gap between the inner and outer cylinders to the bottom of the settler. Rigorous pilot-plant and full-scale tests underground at Doornfontein mine have shown that product water quality was unaffected by rapid and extreme surges in loading rate from 0 to 110 $m^3/m^2/h$ provided that pH, flocculant dosing and the rate of desludging were properly controlled. An agitated surge dam ahead of the settler will even out fluctuations of flow and load. An automated control system is currently developed to guarantee adequate product water quality independent of the human factor (Pullen, Ludwig and Hayward, 1990).

Based on the successful demonstration and evaluation of the full-scale (120 *l*/s) test underground, a 440 *l*/s peak flow rate tube settler is to be constructed at Doornfontein some 3 500 m below the surface for a total daily flow rate of 21 000 m^3/d (245 *l*/s average). Commissioning is scheduled for mid-1990. The set-up of the high-rate tube settler is shown in Fig 4.

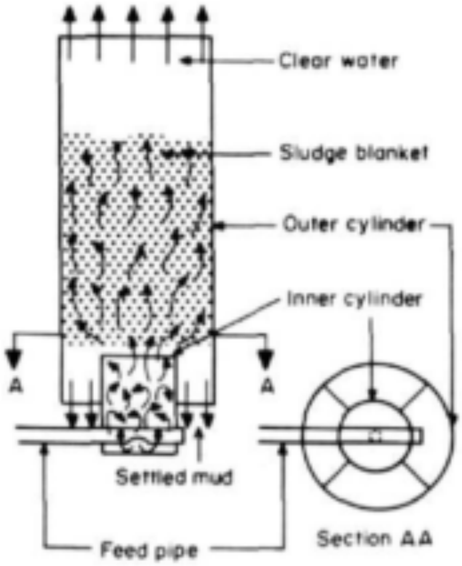
The sludge from the underground settlers, which contains substantial amounts of gold, is pumped to the reduction works for the recovery of gold. Water contained in the sludge, together with the water in the wet rock hoisted to the surface, amounts to about 0,16 m^3/t of rock milled.

In specialized mining applications, however, such as high-pressure pumps or hydraulically powered mining machinery, a further filtration step is required to prevent solid particle erosion of close tolerance equipment. Sand filtration, either downflow or upflow, is considered the best practical method of obtaining large volumes of clear water required for hydraulic machinery. Such high quality water should have a turbidity < 5 NTU and a particle size not exceeding 5 μm (Chamber of Mines, 1988). The selection of suspended solids removal methods based on feed quality, concentration and particle size is shown in Fig 5 (Harries, 1985). A simplified schematic representation of a mine water reticulation scheme is shown in Fig 6 (Holton and Stephenson, 1983).

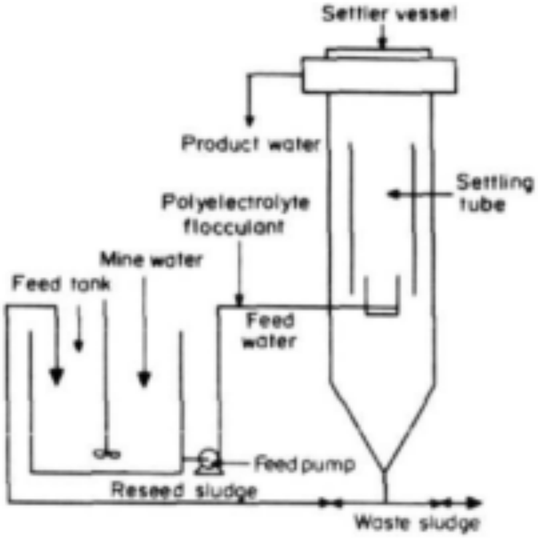
3.2 Mine cooling and refrigeration

All mines, operating at depths between 800 m and nearly 4 000 m below the surface and at virgin rock temperatures of between 30°C to 60°C, require cooling to maintain acceptable working conditions at the stopes. The major causes of heat in deep level operations are:

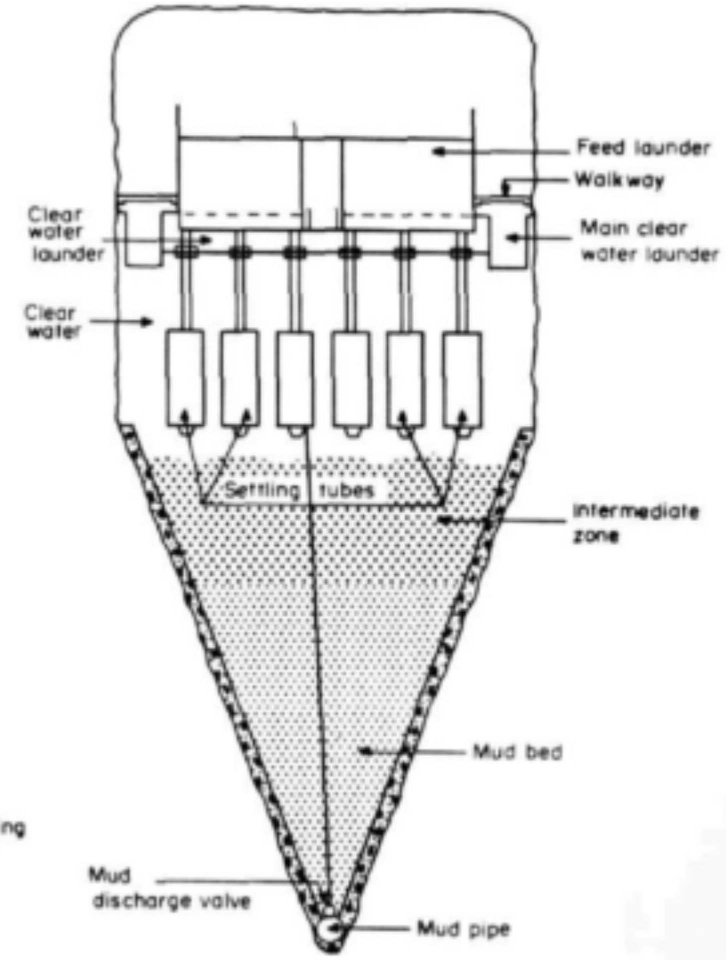
- the high virgin rock temperature;



Flow patterns and details of settling tube



Pilot plant (0,5ℓ/s) process schematic



Cross section of 440ℓ/s high rate tube settler cell

Figure 4
High-rate tube settler

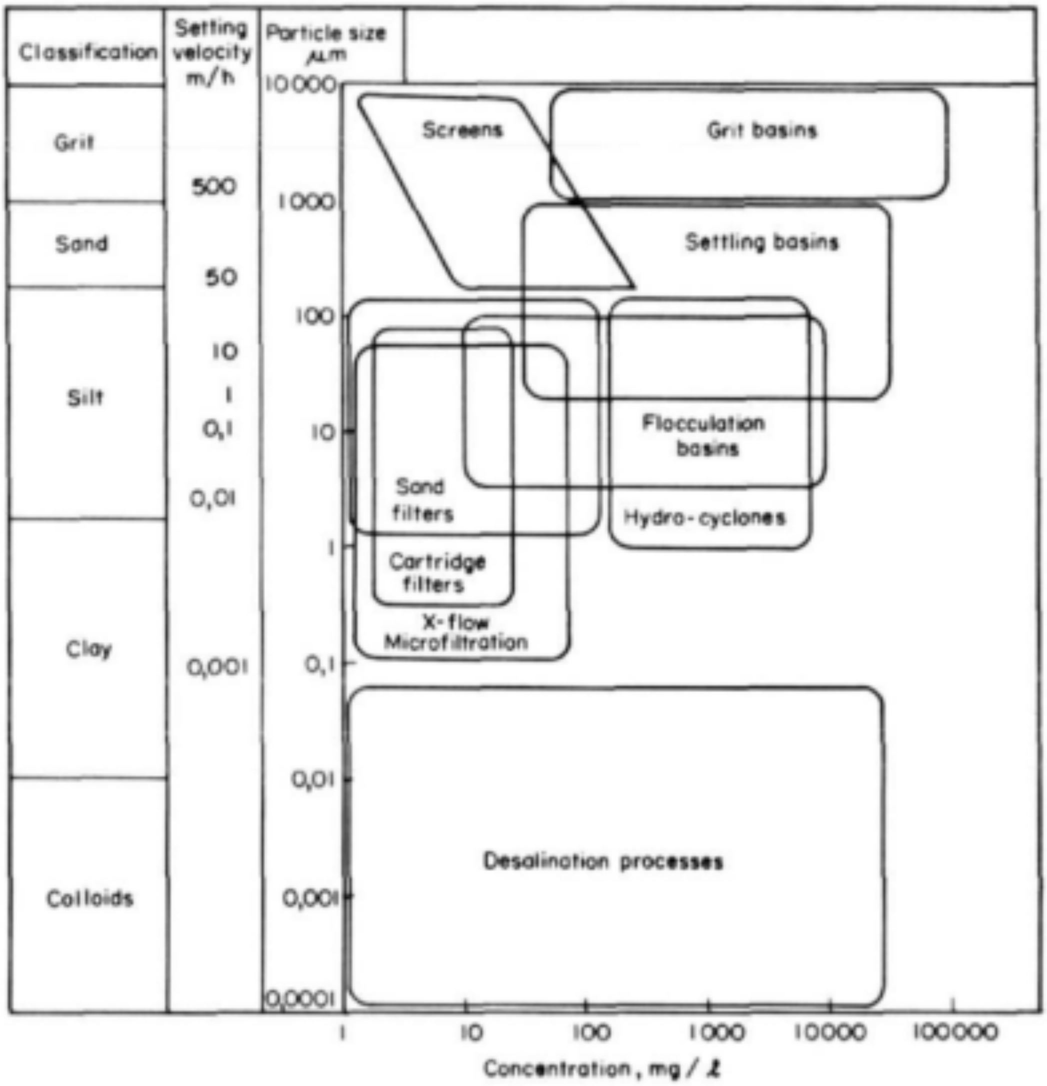


Figure 5
Selection of suspended solids removal methods based on feed quality, concentration and particle size

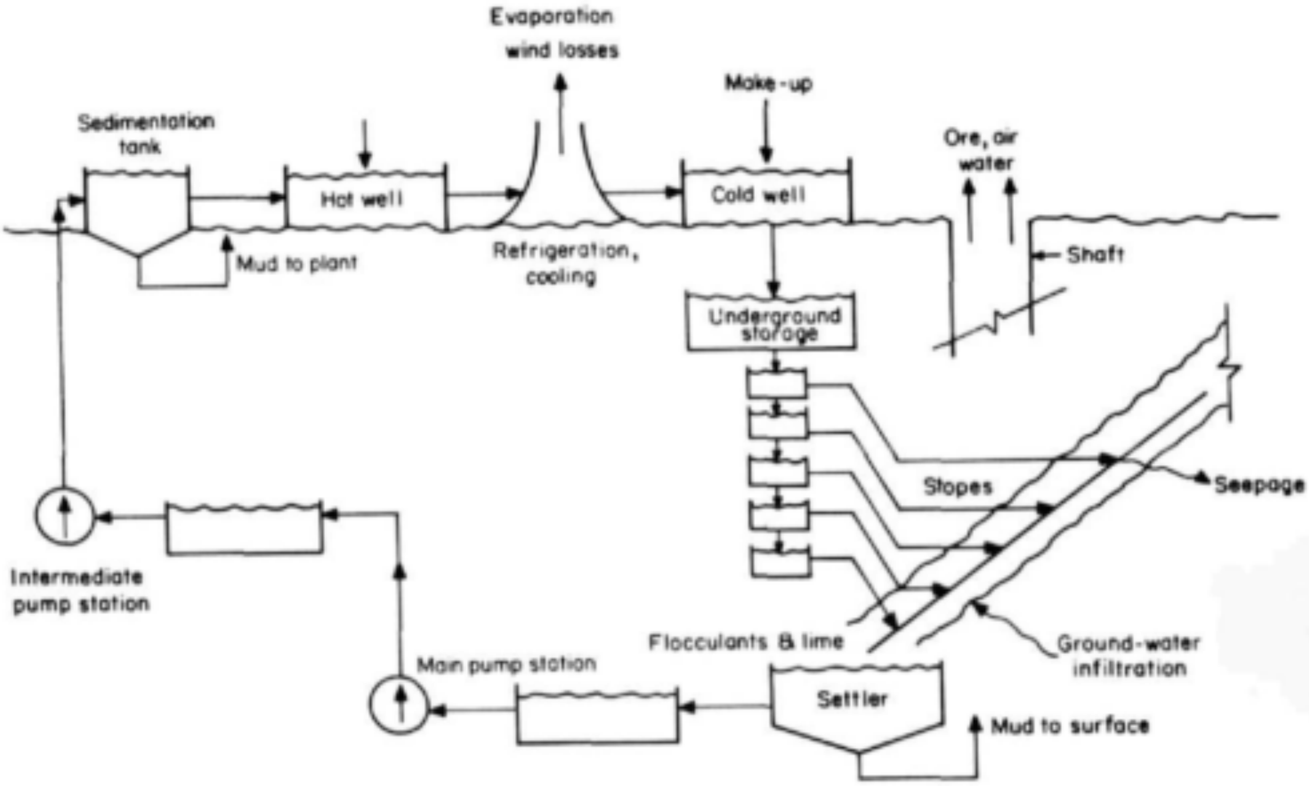


Figure 6
Mine water reticulation system

- the heat gain by the autocompression of the ventilating air which increases the wet-bulb temperature by approximately 4°C/km of depth; and
- the input of heat through mining machinery such as electrically driven equipment or diesel engines.

Virgin rock temperature increases with mining depth. Fig 7 shows the variation of virgin rock temperature with depth, based on 328 rock temperature measurements (Hemp, 1986). The geothermal gradient is noticeably higher in the Orange Free State than in the other mining areas. Many mines have adopted a policy of achieving an average wet-bulb temperature of 28°C to provide environments in which man can work productively and safely. Mine cooling, particularly at the stope face, is therefore a necessity except for shallow depths workings, where ventilation air alone provides sufficient cooling.

Air is supplied to the mines at a rate of about 10 t/t of rock mined. At depths of between 800 m and 1 200 m below the surface, *ventilation air* alone may remove heat sufficiently. There are however, factors which limit the effectiveness of ventilation for mine cooling:

- autocompression which results in an increase of about 4°C in the wet-bulb temperature of downcast air;
- loss of ventilation air to the return airways by leakage because of the high pressure differences between intake and return airways; and
- air losses within stopes where difficulties are experienced in ensuring that air passes along the face rather than through worked-out areas.

Good ventilation is essential not only for mine cooling but also for the removal of dust and nitrous fumes from blasting, and for the removal of radon gas which emanates from certain ore bodies.

At greater depths, chilled water produced at the surface or at underground refrigeration plants is used for distributing cooling. Of the total cooling capacity, about

- 35% is used for service water cooling
- 40% for bulk air cooling; and
- 25% for localized air cooling.

Chilled water is required for each method of cooling. The total installed cooling capacity on all South African gold mines in 1989 was 1 213 MW of which about 75% was in use, with the remainder being required only during limited hot weather periods, or being modified to adjust to the ever changing cooling requirements at different mining sections. Owing to the trend towards deeper mining this total cooling capacity is expected to grow at an rate of about 100 to 200 MW/a (Ramsden, 1990). Water cooling may be drawn basically from three sources:

- low TDS water e.g. Rand Water Board (RWB) water or river water with a temperature of about 18°C and a TDS content of < 1 000 mg/*l*;

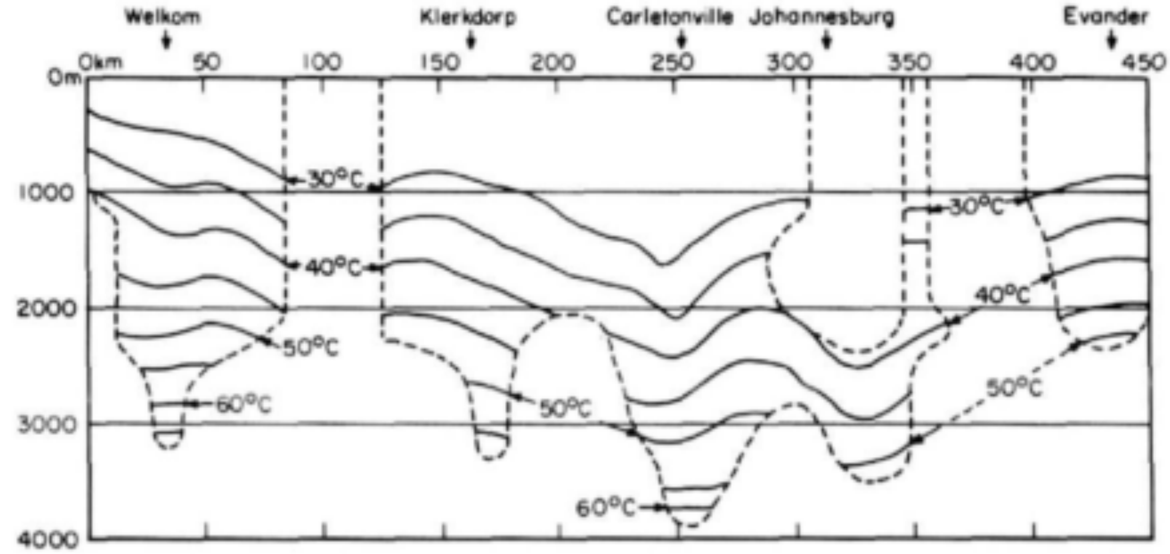


Figure 7
Variation in virgin rock temperature with depth

- underground fissure water with a temperature close to rock temperature, somewhere between 30°C and 50°C and a TDS concentration of between 1 000 mg/*l* (on the Far West Rand) and 3 200 mg/*l* (OFS gold-field/Beatrix); and
- recirculated, settled mine water at about 30°C to 45°C, with a TDS content of up to 5 000 mg/*l*.

Refrigeration plants are located both on the surface (about 60% of the installed capacity), and underground (about 40%).

Underground refrigeration plants have the advantage of producing chilled water close to the working areas. They do, however, have to contend with a number of disadvantages:

- All heat dissipated from the cooling plant is removed via a ventilation shaft, a procedure which can produce potentially lethal temperatures in the shaft as high as 45°C wet bulb. Where the hot air has to pass via a complex system of old workings before reaching the ventilation shaft, large areas of the mine above the cooling plant may become unworkable.
- In underground refrigeration installations the amount of air available for heat rejection is limited because of restricted airway capacity; furthermore, air is at the maximum design temperature for the mine (28°C). Consequently condensing temperatures are higher than for the surface plants, resulting in a considerable increase in power consumption of the refrigeration machines for a given cooling capacity.
- Water and air in an underground heat rejection system are usually contaminated by dust and blasting fumes, hence fouling, scaling and corrosion of pipes and of condenser tubes can become a major problem.

These problems, and mainly the high temperature under which the underground heat rejection systems function, are the reasons for the recent trend towards locating refrigeration installations on the surface and sending chilled water down the shaft.

Surface installations have many advantages. Refrigeration machines usually provide the bulk of the cooling only during summer and less during winter when a larger portion of the cooling comes from the pre-cooling tower. Power requirements and capital costs for a cooling tower are much lower than for a refrigeration machine thus making it the cheapest form of cooling. In contrast, at underground installations all the cooling comes from the refrigeration machine. However, the following disadvantages must be taken into account:

- Chilled water sent underground gains temperature at a rate of 2,3°C/1 000 m.
- The costs of pumping hot water back to the surface for cooling are high.
- Chilled water absorbs heat on its passage down through the normal dam systems, which is not always desirable.
- Problems are encountered in the distribution of chilled water to underground networks.

Ventilation experts believe that below a certain depth, about 2 200 m below the surface, underground refrigeration plants are more suitable than the surface installations.

Refrigeration plants are the central part of most refrigeration systems. At present, the vapour compression type of plant is used almost exclusively in South African mines. The reticulation layout for chilling mine water underground is shown in Fig 8 (Bailey-McEwan, 1979). The chilled water generated by the plant is distributed underground in various ways and returns to the evaporator as hot water for re-chilling. The hot water from the condenser outlet is pumped to an underground cooling tower where its heat is transferred to the upcast ventilation air, and is then returned to the condenser to condense the refrigerant gas.

In mining, the service water and consequently also the refrigeration machines and load are subject to severely fluctuating flow rates. In order to even out these rates and to provide the refrigeration machines with a relatively constant load, large reservoirs are employed at the inlet and outlet of the refrigeration installations. Part of the chilled water from the outlet cold reservoir is conveyed directly via insulated pipes to where it is needed, e.g. at the stope faces as service water. About 35% of the total cooling capacity is distributed by chilled mine service water.

Another portion of the chilled water is used to cool ventilation air. Bulk air coolers at the surface and underground are mostly located near the shafts. Cooling of air is effected by direct water-to-air contact, at the surface in packed spray towers, and underground in water spray chambers. The 2,5 stage spray chamber of Hartbeesfontein mine, as shown in Fig 9, is designed for a cooling capacity of 1,68 MW. Bulk air coolers provide about 40% of the total cooling.

It is sometimes necessary to cool or re-cool air as close as possible to the workings in order to minimize heat pick-up from hot rock and to reduce refrigeration losses through leakage of cold air via unused worked-out sections of the mine. The most widely used method for distributing cooling close to working areas is by means of finned tube heat exchangers known as cooling coils. Mines in South Africa presently distribute about 25% of the refrigeration capacity through cooling coils. Cooling coils operate satisfactorily when clean, but the finned tubes inevitably become fouled or corroded externally and internally during operation and consequently performance deteriorates. To eliminate fouling due to water hardness and dust, a new lightweight entrainment air/water spray cooler has been developed by the Chamber of Mines Research Organization (Comro) which can be moved as the stope advances. This cooler is powered by chilled water alone at normal working pressure so that no electricity is required; it contains no moving parts thus rendering it insensitive to minor damage. The stope air/water entrainment cooler is shown as Fig 10. About 1 000 of the units are in use on South African gold mines, each having a cooling capacity of 10 to 15 kW (Ramsden 1990).

3.2.1 Water quality considerations

The chemical quality of mine service water before neutralization with lime for the mining areas from east to west is given in Table 1.

The mine water on the Central Rand contains mainly sulphates whilst on the Far West Rand, carbonates are more predominant. The Evander and Orange Free State mines have high concentrations of chloride in underground water.

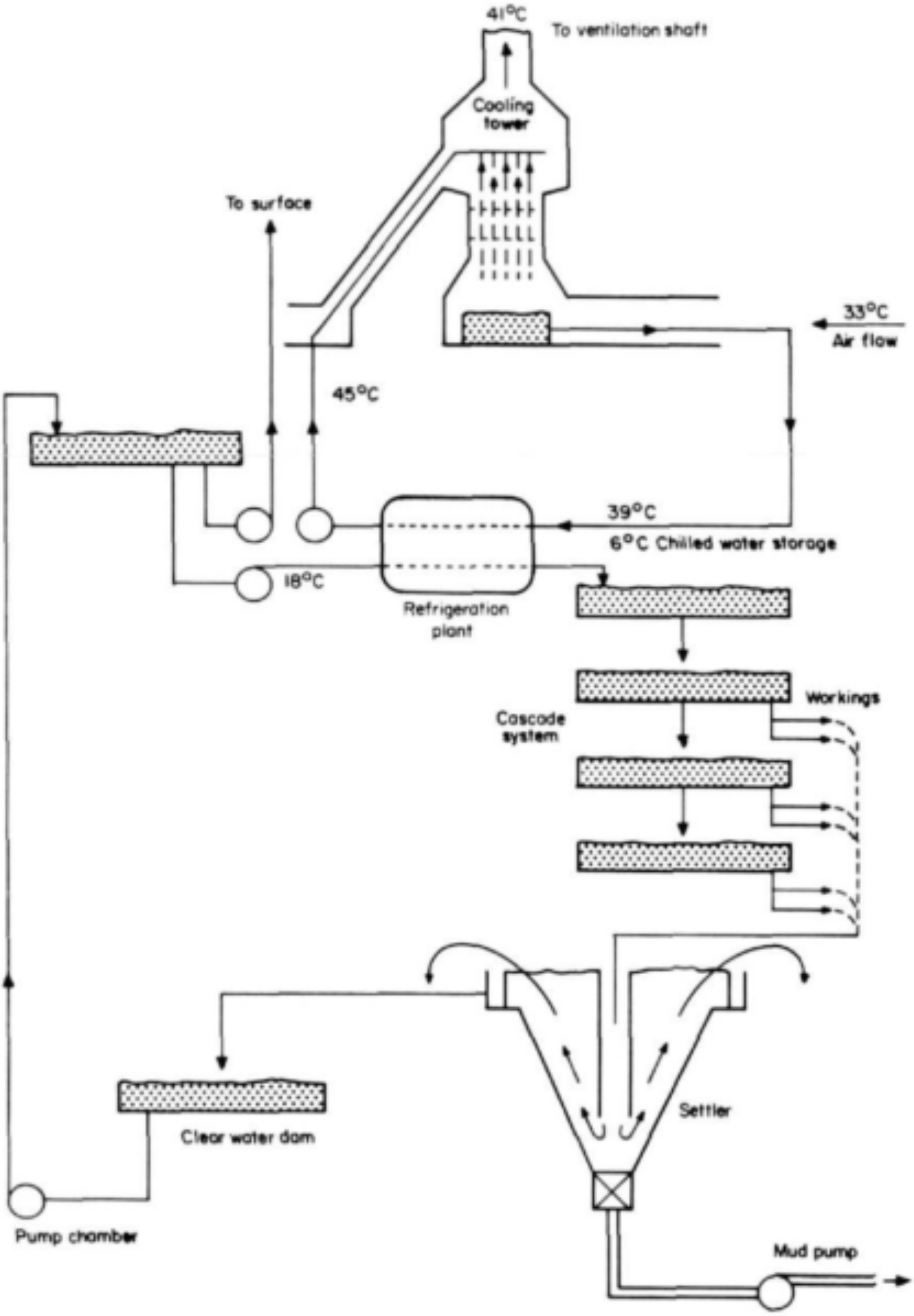
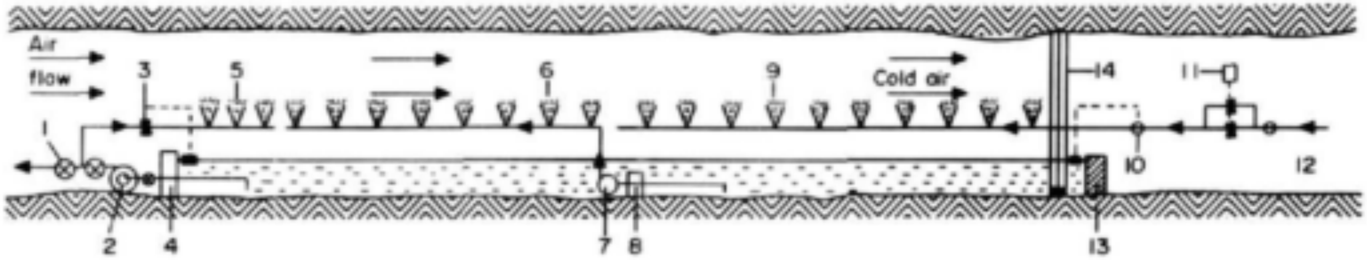
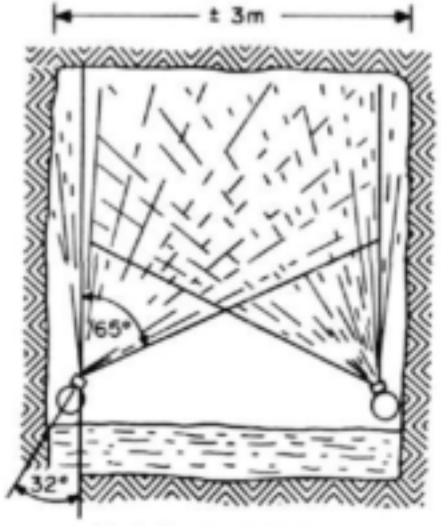


Figure 8
Reticulation layout for chilling mine water underground

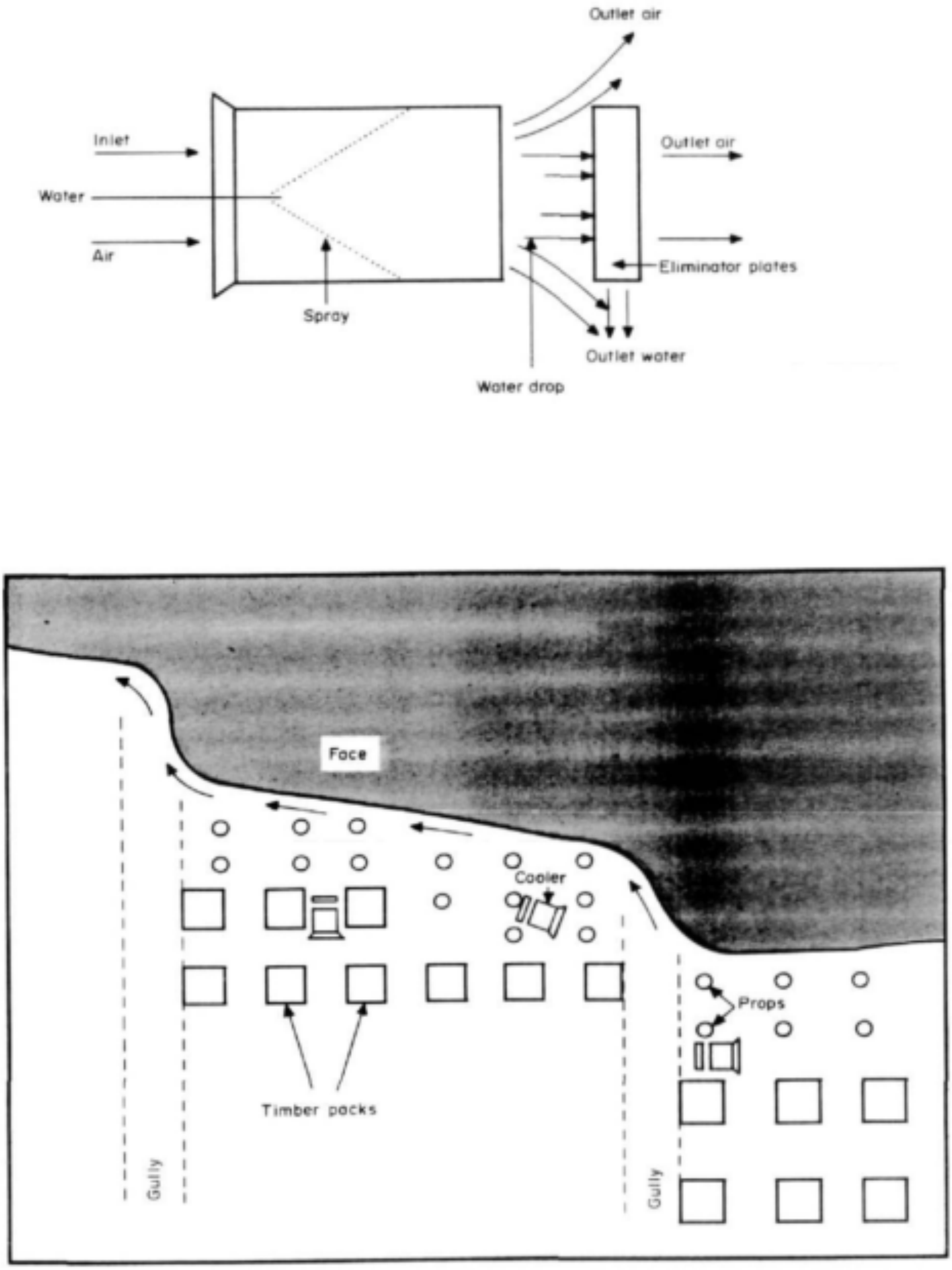


Layout of typical 2- stage spray chamber



Typical spray pattern

Figure 9
Bulk air cooling by spray chamber



Possible positioning of entrainment coolers is between the first row of packs or close to the gully

Figure 10
Stope air/water entrainment cooler (Feio *et al.*,1987)

Table 1: Average chemical quality of mine service water before neutralization in the different mining areas (Chamber of Mines Research Organization, 1982).

Mining area		pH	TDS	SO_4	Cl	Na	Total hardness (as $CaCO_3$)
Barberton		8,3	1 000	380		50	
Evander		6,5	2 200	320	820	600	400
East Rand		7,5	2 500	840	400		520
Central Rand		4,2	3 000	1 410	130	130	1 240
West Rand	Western Areas	7,2	1 300	750	35	100	500
	Other mines	5,8	2 500	1 220	100		1 250
	Far West Rand	6,7	1 300	550	160		540
Orkney/Klerksdorp		6,8	2 200	950	130	40	700
OFS		6,8	4 450	990	1 500	1 500	1 000

Concentrations in mg/l, except pH

High concentrations of dissolved salts, sulphates and particularly chlorides, a low pH and saturation with corrosive gases such as dissolved oxygen and CO_2 promote corrosion of mild steel or even stainless steel used in refrigeration machines and heat exchangers.

Hardness of the water in circulation causes scaling on pipes, mining machinery and particularly on heat transfer surfaces of refrigeration machines, and may even block pipes. A scale deposit of only 0,3 mm of calcium carbonate in a heat exchanger system can cause a 10% increase in the power consumption of a refrigeration plant. The effect of increasing temperature on the rapidly decreasing solubility of $CaCO_3$ and $CaSO_4$ is shown in Fig 11 (White 1985b).

Scale inhibitors such as organic phosphates, interfere with the growth and shape of the crystals of calcium carbonate, calcium sulphate and other scale-forming salts. Polyacrylate type dispersants enhance the effectiveness of organic phosphate scale inhibitors.

For many mines the quality of settled mine service water is adequate for the once-through refrigeration machine chilled water circuit since this water, after chilling, is sent underground and is replaced by new mine service water. Where mine service water is corrosive or scale forming at heat exchanger surfaces in the refrigeration plant or at pumps and pipes underground, treatment must be considered. Any chemical added to this water will be effectively lost. An important quality consideration for chilled water sent underground is that it is disinfected, which implies that pathogenic micro-organisms have been rendered innocuous by chlorination. Chemicals added to inhibit scale formation or corrosion should be compatible with drinking-water requirements (McLaren and Pearson, 1985). As far as dissolved salts are concerned, about 2 000 to 2 500 mg/*l* (equal to an electrical conductivity of approximately 300 mS/m) may be tolerated for drinking purposes (South African Bureau of Standards, 1984).

For the refrigeration condenser circuit, where the same water is recirculated over a cooling tower, the concentration of dissolved salts by evaporation must be kept in check by blowdown. At quite a number of mines, mine service water is now being used on a large scale to replace low TDS water in open evaporative cooling systems. Treatment of the water in circulation to combat scale formation and corrosion is, therefore, essential (Evans, 1985). The effects of running the refrigeration condenser circuit on mine service water from the Carletonville area at 2 and 3 cycles of concentration are shown in Table 2.

As the water concentrates, the operating parameters change from a high Ryznar Index to a low one, creating a water that tends to precipitate rather than dissolve $CaCO_3$. The real possibility now exists, particularly at 3 cycles of concentration and more, that deposition of $CaCO_3$ and $CaSO_4$ will occur. Calcium carbonate is the predominant scale when RWB water is used for condenser cooling, while calcium sulphate scale predominates when mine water is used. Due to the high concentration of sulphate and particularly chloride and the presence of dissolved oxygen, the probability of corrosion taking place simultaneously is real.

While several options are available to reduce the tendency for scale deposition in a condenser circuit, the most practical and easily controllable method is the use of scale inhibitors. Since a suspended solids level of 45 mg/*l* after 3 cycles of concentration would be unacceptable in the cooling water due to the risk of abrasion on metal surfaces, it would be desirable to install side-stream filtration with approximately 5% of the recycle volume passing through a filter.

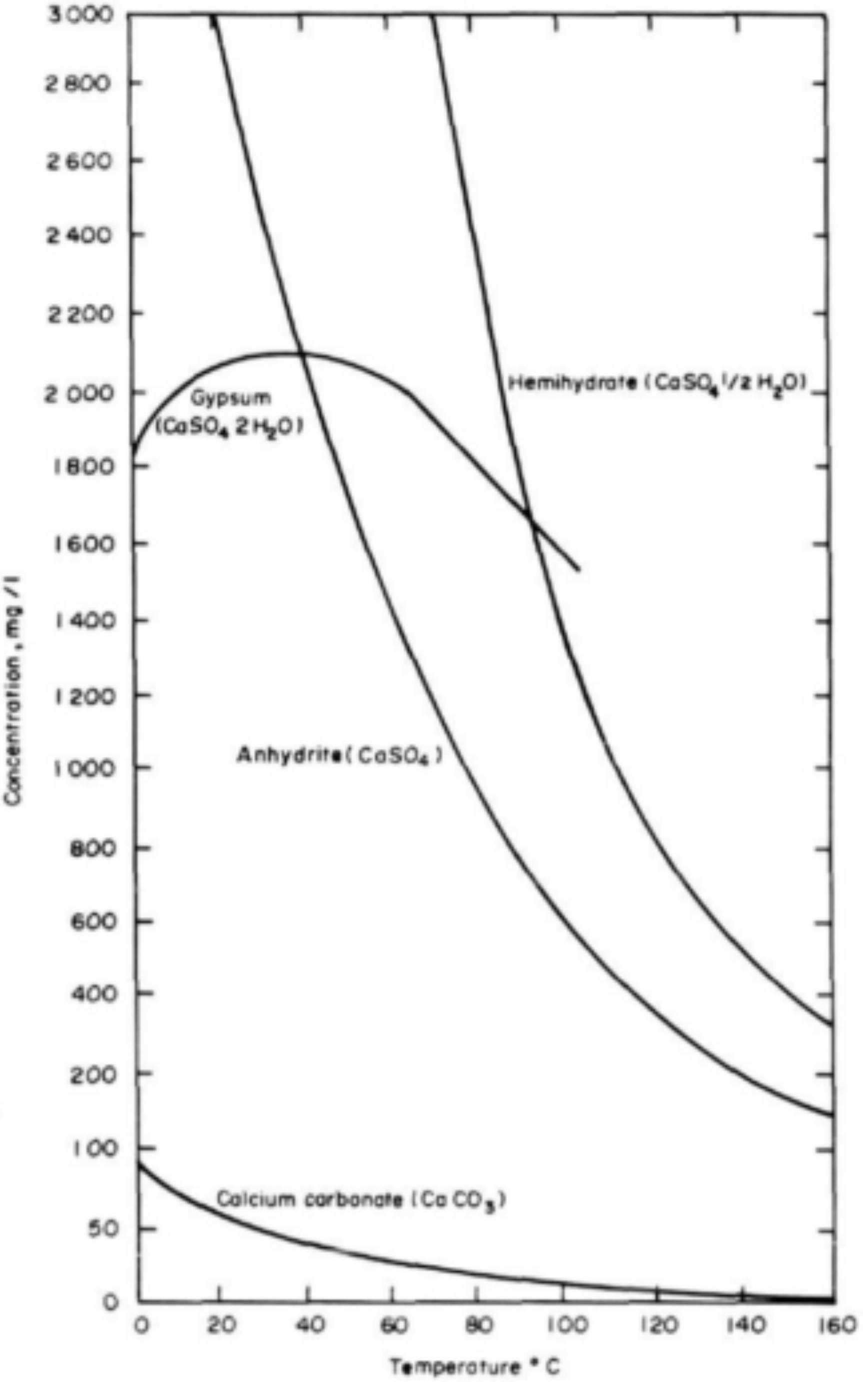


Figure 11
Effect of temperature on solubility of scaling salts

Table 2: Predicted chemical composition of mine water in the Carletonville area after 2 and 3 cycles of concentration when used in the condenser circuit

Determinand		Mine service water	Number of concentrations	
			2	3
Total dissolved salts		2 188	4 376	6 564
Hardness	Total	517	1 034	1 551
	$Ca^{\cdot\cdot}$	431	862	1 293
	$Mg^{\cdot\cdot}$	86	172	258
Alkalinity	Total	80	160	240
Fe	Total	1	2	3
	Soluble	0	0	0
Suspended solids		15	30	45
pH		7,6	8,1	8,3
SO_4		1 349	2 698	4 047
Cl		321	642	963
Ryznar stability index at 25 °C		7,2	5,4	4,5

Concentrations in mg/l, except pH and stability index. Hardness and alkalinity are expressed in mg $CaCO_3$*/l*

3.2.2 Water loss by evaporation and ventilation upcast air

The simplified layout of a mine cooling system is shown in Fig 12. In order to get an idea on how much water will be lost by evaporation during cooling and chilling of hot mine service water *at a surface installation*, the following information will be of value. A spray cooling tower or a spray pond may pre-cool warm mine service water coming from the main pump station (about 30°C) to within about 2°C to 2,5°C of the ambient the surface wet-bulb temperature, i.e. to 19°C on a midsummer afternoon and to 7°C during midwinter nights. The yearly average wet-bulb temperature is possibly about 12°C, therefore the cooling tower should reduce the water temperature to an average of 15°C. As a result of these defined water temperatures, water flow rates are generally directly proportional to the required cooling duty. This water is further chilled in a refrigeration plant to say 4°C and is returned underground. Since the warm condenser outlet water also needs to be cooled, a second cooling tower is incorporated into the system.

Refrigeration plant condenser outlet water temperature is about 30°C to 32°C, with the condenser inlet water temperature after cooling being about 24°C to 27°C. A mine which mines 200 000 t/month at a mean VRT of 41°C at 2 000 m below the surface would require 0,15 MW/1 000 tons mined. The refrigeration requirements are then 200 x 0,15 MW = 30 000 kW. If it is assumed that about 4 m^3 of water pumped from the mine should be used for cooling every ton of rock mined, that is 800 000 m^3/month for a production of 200 000 t/month, equal to 308 *l*/s, then the approximate amount of heat removed by the pre-cooling tower can be calculated as follows:

$$\begin{aligned} Q &= \text{mass flow } (l/s) \times \text{specific heat } (kWs/kcal) \times \Delta t\ (^{\circ}C) \\ &= 308 \times 4{,}184 \times (30 - 15) \\ &= 19\,330 \text{ kW} \end{aligned}$$

Thus, more than half of the required cooling could be provided by the pre-cooling tower. Assuming, that the secondary condenser circuit cooling tower now has to have a cooling duty of 10 700 kW, the actual cooling duty, because of additional heat input from the compressor driver motor load, etc. of about 2 000 kW, will be about 12 700 kW. Water temperatures across condenser cooling towers range between 2°C and 6°C with an annual average of 5°C. The volume of water over the cooling tower required to remove the 12 700 kW can now be calculated as:

$$\text{Mass flow } l/s = 12\,700/(4\,185 \times 5) = 607\ l/s$$

The loss of evaporation is assumed to be 2% of the water flow rate per 10°C cooling obtained in the tower.

Loss of water droplets through wind (windage or drift losses) is in the range of:	1,0% to 5,0% for spray ponds;
	0,3% to 1,0% for natural draught towers; and
	0,1% to 0,3% for mechanical draught towers

of the water flow per 10°C cooling.

The evaporation loss in the above example for 30 000 kW cooling duty for a surface

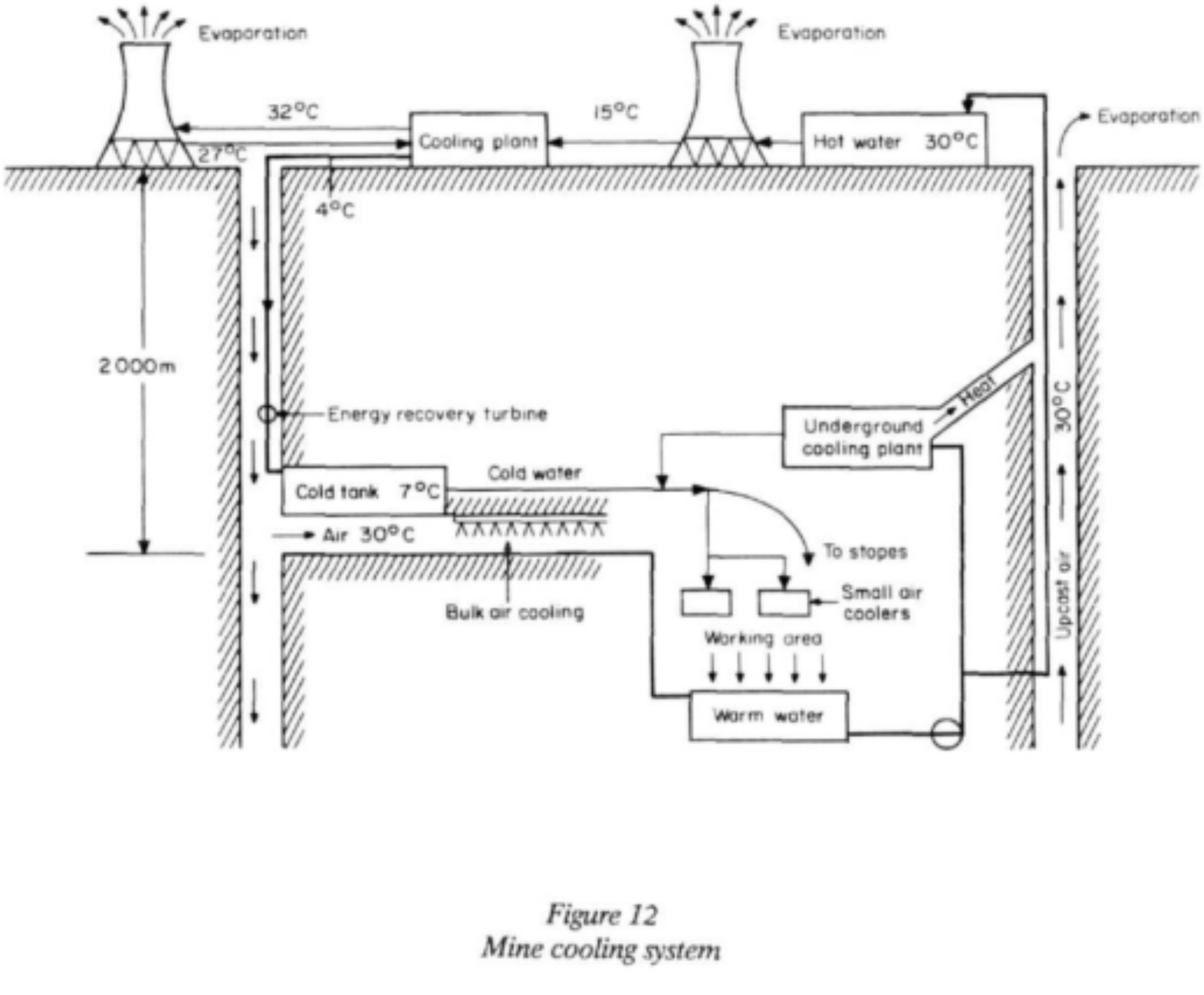


Figure 12
Mine cooling system

refrigeration plant with a pre-cooling tower for 15°C temperature difference and condenser cooling tower for 5°C temperature difference would be:

$$(308\ l/\text{s} \times 3\%) + (607\ l/\text{s} \times 1\%) = 15{,}3\ l/\text{s}$$
$$\text{i.e}\ \ 1\ 323\ \text{m}^3/\text{d or}\ \ 483\ \text{M}l/\text{a}$$

The windage loss would add another 0,5% of the total flow in circulation (915 *l*/s), i.e. 4,6 *l*/s = 397 m^3/d or = 145 M*l*/a.

The total volume of water lost by evaporation and wind for a 30 MW cooling duty is thus 483 + 145 = 628 M*l*/a.

For all South African gold mines the total installed cooling surface capacity for 1989 is as follows:

Total:	1 213 MW
Surface plants:	728 MW (60%, estimated)
Underground plants:	485 MW (40%, estimated)

For surface plants the loss of water by evaporation and windage can now be calculated as approximately (728 x 628)/30 = 15 240 M*l*/a at an assumed 100% cooling duty.

In addition to the *evaporation/windage losses* there is the loss by blowdown. Blowdown is a function of evaporation and cycles of concentration which, on South African gold mines, can vary between values of 2 to 8. Blowdown is calculated from the relation $E/(C - 1)$ where E is the rate of evaporation and C the permissible cycles of concentration.

When taking, as an example, the above cooling duty of 30 000 kW and 3 cycles of concentration (as shown in Table 2 for Carletonville mine service water), the rate of blowdown is:

$$(15{,}3/2)\ l/\text{s} = 7{,}65\ l/\text{s} = 661\ \text{m}^3/\text{d} = 241\ \text{M}l/\text{a}$$

When assuming that 3 cycles of concentration would be an average for the whole industry, then the *blowdown loss* for 728 MW surface cooling capacity would be calculated as:

$$(728 \times 241)/30 = 5\ 848\ \text{M}l/\text{a}$$

Make-up requirements to replace losses due to evaporation, wind and blowdown at South African surface cooling plants can therefore be assumed to be in the region of:

$$15\ 240 + 5\ 848 = 21\ 088\ \text{M}l/\text{a},\ \ \text{rounded off to}\ 21\ 100\ \text{M}l/\text{a}\ (1989)$$

For the loss of water with the *ventilation upcast air* the following information is supplied:

Ventilation air (about 41 000 m^3/s on all South African gold mines, Ramsden, 1990), removes gases, dust, heat and large quantities of water, mainly in the form of vapour and to a lesser extent as carry-over in the form of droplets. The moisture content of the downcast air on dry days varies from about 4 g/kg of air in winter to about 14 g/kg in summer. On rainy days the moisture content increases substantially (Hemp, 1986).

At the bottom of the shaft temperature and moisture conditions are likely to vary depending on whether underground refrigeration plants are used or not. A moisture content averaging 29,1 g/kg for these conditions has been found on mines in the Orkney/Klerksdorp and OFS gold-fields areas.

As the air moves up the upcast shaft, cooling occurs through auto-decompression. The upcast air becomes supersaturated and condensation occurs. In modern deep mines the air velocities in the upcast shafts are normally within the range of 18 to 20 m/s which is well in excess of the critical air velocity range of between 7 m/s and 12 m/s where water droplets remain in suspension. At these high air velocities of about 18 m/s and more, all the moisture at the bottom of the upcast shaft will be taken out of the mine. Psychrometric data from 31 shafts in the Orkney/Klerksdorp and OFS gold-fields are shown in Table 3.

The average moisture content of ventilation air at fan drift is 25,5 g/kg with about 4 g/kg of supersaturation in the form of droplets as free moisture. Moisture pick-up is higher during winter than during summer, since the mass flow of air is higher due to a greater difference in density between the downcast and the upcast streams, compared with summer.

Assuming that the data given in Table 3 are representative for the total of local mines, then at 41 000 kg/s air flow through the mines, the water removed or moisture pick-up by the ventilation upcast air can be calculated to be:

41 000 kg/s x 17,7 g/kg = 725,7 *l*/s = 22 886 M*l*/a rounded off to 22 900 M*l*/a

The total possible water loss through evaporation, wind and blowdown at the surface refrigeration plants and through ventilation/cooling of the mine shaft, assuming that all plants are operating at 100% cooling duty, can thus be estimated as:

21 100 M*l*/a	at the surface plants
22 900 M*l*/a	through ventilation shafts
44 000 M*l*/a	

for an installed cooling capacity of 1 213 MW at the end of 1989. As mines go deeper, the installed cooling capacity will increase at an estimated average rate of 100 to 200 MW/a. Consequently, water losses will increase.

To place mine cooling water losses in the right perspective: One 3 600 MW coal-fired power station requires about 55 000 M*l*/a of low TDS river water to replace losses due to evaporation, wind and blowdown, which is more than is at present required for all of the 43 large gold mines. There are two 3 000 MW and five 3 600 MW stations on the Transvaal Highveld and a number of smaller power stations with a total generating capacity of about 32 000 MW in operation.

Cooling water make-up requirements for individual mines of the Gold Field Group were reported as follows (1985):

Venterspost	620 M*l*/a	own dolomitic water supply
Libanon	1 500 "	50% potable water + 50% own water

Table 3: Psychrometric mine ventilation data of 31 shafts in the Orkney/Klerksdorp and OFS gold-field areas

	Fan drift temp, dry/wet-bulb °C	Air flow, kg/s	Moisture						
			g/kg air			t/h			
			Fan drift	Down-cast	Gain upcast	Pick-up	Bottom upcast	Fan drift	Free
Average	24,4/24,4	500	25,5	7,8	17,7	31,9	52,5	45,9	6,6
Range	15,6/15/6 to 38,0/38,0	133 to 900	13,7 to 55,5	3,8 to 14,1	7,1 to 43,3	2,8 to 121,3	7,0 to 141,3	5,3 to 134,8	1,1 to 19,3

Moisture fan drift - moisture downcast air = moisture pick-up
Moisture bottom upcast - moisture fan drift = free moisture

Kloof	1 650 "	50% potable water + 50% treated sewage
East Driefontein	700 "	from West Driefontein
West Driefontein	1 530 "	own dolomitic water supply
Doornfontein	950 "	own dolomitic water supply
Deelkraal	950 "	potable water

At Vaal Reefs, the make-up requirements for cooling and refrigeration for 1986 were estimated to be 3 100 M*l*/a for all 7 shafts combined. Thus, for 8 gold mines the total estimated annual cooling water make-up is close to 11 000 M*l*/a. Evaporation losses from cooling towers on the surface installations are irrevocably lost. Water carried with the saturated upcast air from underground may probably be partially recovered, provided mine ventilation layout is favourable or could be modified.

3.2.3 Options for energy reduction

(i) Cooling systems operation

Operating the cooling towers for the hot mine service water pumped to the surface at full load during night and early morning and storing the cooled water in large storage dams, preferably covered, will increase the cooling duty of the pre-cooling towers and decrease the cooling duty of the refrigeration plant. The make-up water for the condenser cooling circuit for the refrigeration machines must preferably be low TDS water.

(ii) Energy recovery by Pelton turbine

Chilled water, when sent underground, discharges potential energy, which results in an increase in temperature of 2,3°C/1 000 m depth. By installing an energy recovery turbine at the bottom of the shaft, up to 66% of the potential energy of the chilled water sent down can be recovered with the temperature increase being correspondingly reduced. For example, if no energy recovery turbine is used, chilled water which leaves the surface at 4°C arrives 2 000 m underground at a storage dam at 4°C + 2 x 2,3°C = 8,6°C. With an energy recovery turbine, the rise in temperature would be 2 x 2,3 x (1 - 0,7) = 1,4°C and water would enter the storage dam at 5,4°C. Consequently, more refrigeration capacity could be sent down or less water and energy be needed for a required cooling duty. To date, a large number of Pelton energy recovery turbines have been installed. The power thus saved, approximately 1 MW in a typical installation, partially replaces the pumping energy needed to return the water to the surface.

(iii) Use of ice to reduce volume of warm return water

The costs of producing chilled water and of pumping back hot water are equal for a depth of about 2 000 m below the surface. Sending ice underground reduces the pumping volume for a required cooling capacity because of the latent heat absorbed for melting. The attractiveness of sending down ice increases with depth as, after melting of the ice underground, the amount of warm water to be pumped to the surface would be approximately *20%* of that pumped with a conventional chilled water system for the same refrigeration capacity. Studies have indicated that for deep mines the total cost for an ice system could be equal to or lower than that of a chilled water system incorporating energy recovery turbines. Similar studies have also indicated that the refrigeration capacity of an existing water chilling plant on the surface could possibly be best augmented by adding an ice-making plant in series, rather than by providing additional water chillers. Pneumatic conveyance through plastic pipelines has emerged as the most suitable technique for transporting ice in a dry, or nearly dry state underground.

Extensive pilot plant tests on producing ice and transporting it horizontally and down the shaft, and on suitable means to melt the ice underground, were conducted at ERPM by COMRO in the early 1980s. The first 1 000 t ice (5 MW) refrigeration installation, at that time the largest in the world, has been installed and was commissioned during 1986 at Harmony (Eschenburg, Middleton and Hemp, 1986; Middleton and Muller, 1986 and Eschenburg, Hemp and Roux, 1986). Ice at 1 000 t/d is formed by spraying water onto the outer the surface of ammonia-cooled tubes. Ice-formation is, therefore, in the shape of a hollow cylinder with a wall thickness of 8 to 10 mm which is subsequently broken into small pieces by heating with hot ammonia gas. The plant operates on a 12 min cycle (11 min ice-production and 1 min harvesting). Broken ice from the surface plant is conveyed pneumatically 80 m to the top of the shaft by refrigerated air through rotary valves and 200 mm core-insulated PVC pipe, and then vertically by gravity through the same type of pipe to an ice-melting dam, 1 088 m below the surface. The ice plant and a section of a tube ice-making module are shown in Figs 13 and 14.

The plant had originally been designed to use mine service water for ice-making with the following composition:

TDS = 4 650 mg/*l*
Cl = 1 140 "
SO_4 = 560 "
Ca = 530 "

The use of this water has so far resulted in a yellow-coloured spongy type of ice being produced which is not brittle enough to shatter from the ice-maker tubes when harvested. It appears that this ice is not subcooled sufficiently during the 11-min ice-making period. Because of the poor quality of ice obtained with high TDS mine water, only water from the OFS Goldfields Water Board with a TDS of about 800 to 1 000 mg/*l* is used for the production of ice. The production over the past years has been kept at a level of 860 t/d, resulting in a reduction of the volume of water to be returned to the surface by about 3,4 M*l*/d or 1 250 M*l*/a with a corresponding reduction in pumping costs.

Based on the immediate success of the Harmony ice plant, a second, larger, 6 000 t/d tube ice plant, the largest in the world, has been built at the new ERPM Far East shaft system to provide cooling at depths exceeding 3 000 m. This ice plant, also ammonia-cooled, is of a different design, combining a cluster of small diameter tubes in a large diameter tube. The first 1 000 t/d module is expected to be commissioned in the second half of 1990, with the second module shortly afterwards.

A 350 t/d pilot plant to produce slurry ice for cooling is presently being installed at Western Deep Levels and will be commissioned in about mid-1990. Another 1 000 t/d slurry ice plant for cooling and desalination is being planned for Vaal Reefs.

Evaporation loss: As the ice plant uses more energy to provide cooling than a chilled water cooling installation, the condenser load and evaporation loss is higher for ice-making.

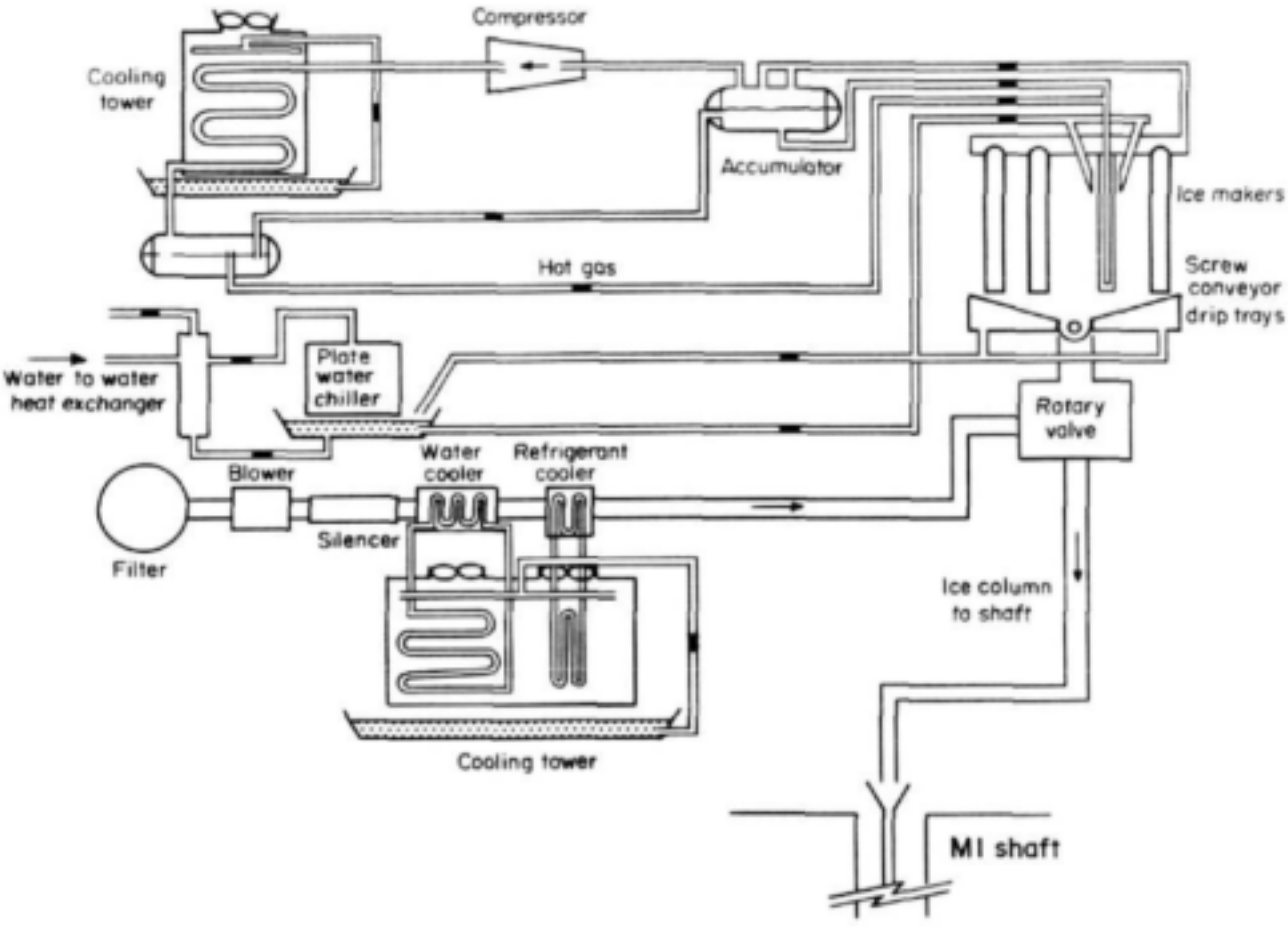


Figure 13
1 000t/d surface ice plant at Harmony

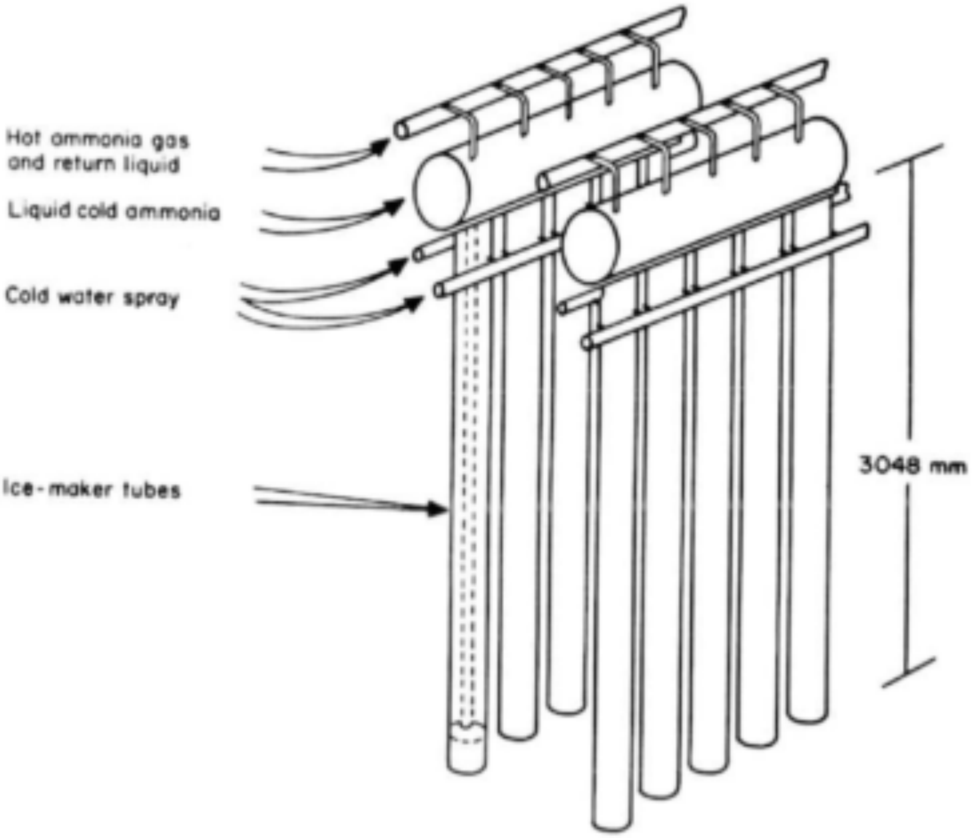


Figure 14
Section of tube ice-making module

(iv) Insulation of chilled water pipes

In South African gold mines approximately 450 km of pipes are insulated each year. Prior to 1987, about 80% of lagging was made of polyurethane foam. After the use of polyurethanes for insulation was condemned due to their high toxic emissions when burning and their linkage to the major mining disasters at Kinross (1986), Western Deep Levels and Vaal Reefs, all traditional methods of insulation were banned by the Government Mining Engineer. Subsequently, all insulation material regarded as combustible, had been stripped from underground pipework. Most of this will have to be replaced by other lagging. In the meantime, non-toxic insulation systems have been developed, based on glass or mineral wool as well as foamed phenol-formaldehyde, with a vapour barrier, e.g. galvanised steel, to protect the very porous insulant material. These lagging materials are more expensive than polyurethane. Numerous measurements of the thermal conductivities of different types of pipe insulations under various conditions have shown the following:

- For all types of insulation tested there is no significant increase in heat transfer with age, provided the vapour and mechanical barriers are not damaged.
- Even when waterlogged, the insulation still reduces the heat transfer to about 10% of that of an uninsulated pipe.
- The rise in the temperature of chilled water flowing in insulated pipes due to friction is similar to heat flow across the insulation from the (warm) ventilation air to the water.

Therefore, in spite of the harsh conditions encountered underground with the inevitable damage and water-logging of the insulation material, it is economically justifiable to insulate chilled water pipes in order to minimize the temperature rise due to heat flow. The rise in temperature of chilled water due to pipe friction must also be accounted for.

(v) Recirculation of ventilation air

As air flows down a vertical shaft, its wet-bulb temperature increases by about 4°C per 1 000 m depth because of adiabatic compression. In most mines, therefore, the air has to be cooled in bulk before entering the workings. Refrigeration required to achieve this is proportional to the air flow rate. The quantity of heat to be removed is significant, particularly in deep mines, as some 10 t of air is used in an average mine for every ton of rock hoisted. Consequently, by lessening the flow of downcast air, recirculating air underground reduces the cooling needed to offset the effect of auto-compression.

At Loraine gold mine, where the first air recirculation system is operating since 1982, 35 m^3/s of return air is cooled and washed in a spray chamber, mixed with 15 m^3/s of fresh downcast air and then forced back into the working section in a remote part of the mine at a depth of 1 800 m where the virgin rock temperature is 45°C. After commissioning, the mixed-air-intake wet-bulb temperature dropped almost immediately from 27,9°C to 24°C. This was followed by a gradual trend towards about 22°C. The return-air wet-bulb temperature followed the gradual trend from 31,5°C to a level of 28°C over the first months. Typical wet-bulb temperatures at various points around the circuit prior to air circulation and at design flow rates for air circulation are shown in Table 4 (Fleetwood *et al.*, 1984; Burton, 1988).

Table 4: Wet bulb temperatures prior to air circulation and at design airflow rates at Loraine

		Prior to recirculation	At design airflow rates
Airflow rate (m/s)	Intake air	35	16
	Recirculated air	–	34
	Mixed air intake	(35)	50
Air wet bulb temperature °C	Mixed air intake	(27,9)	22,3
	Mean air intake	30,0	23,6
	Stope return air	31,3	28,2
	Return air	35,5	28,4

Because of the increase in air flow rate, the main benefit lies in providing an effective means of distributing the required amount of refrigeration. Consequently, rock skin temperature has been reduced over a wider range and productivity has increased at a fraction of the cost to provide the same quantity of air from the surface. Air recirculation has proved to be of advantage for cooling in *scattered mining* with long horizontal distances at great depth.

At present (1989), seven controlled recirculation schemes are operational on five gold mines. These involve a total of 280 m^3/s of fresh air and 185 m^3/s of recirculated air.

Another nine schemes are being considered on eight mines with a total of 410 m^3/s of fresh air and 290 m^3/s of recirculated air. Trials at Loraine showed that direct-contact spray air coolers are ineffective for removing respirable dust. However, atomizing nozzles which make use of compressed air to create small water droplets, (20 μm) are effective. It has been found possible to combine air cooling and dust filtration in one air-conditioning unit, with no extra water pumping costs incurred.

(vi) Reduction of underground heat load

Heat flow from the surrounding rock into the many kilometres of ventilation intake airways on a mine constitutes a significant component of the underground heat load. In many mine sections the heat from the rock in worked-out areas through which the air would pass contributes 50% or more of the heat load. The use of plaster walls, airtight curtains, ventilation doors and backfilling are established practices to reduce the flow of cooled ventilation air through unused airways and mined-out sections. Another method of reducing underground heat is by insulating the rock surfaces of intake airways. The effect has been investigated by COMRO at Western Deep Levels. Two test sections were insulated, one totally and one partially, excluding the footwall which carries the traffic. A third control section was left without insulation. Polystyrene was used and the average insulation thickness was 70 mm. The data obtained showed that a reduction in heat flow of up to 57% was possible when the airway was fully insulated and that a 30% reduction in heat flow was possible when the footwall was left uninsulated. The average heat flow was 35 W/m for the fully insulated section as opposed to 56 W/m for the partially and 81 W/m for the uninsulated sections.

The inflammability and fire hazard of organic foam material manufactured from phenol-formaldehyde or polyurethane have so far precluded its introduction. Investigations are, however, being conducted by COMRO and commercial companies to develop and test insulating materials which cannot burn e.g. materials based on alumina-silicate fibre.

(vii) Backfilling of worked-out stopes (Matthews,1986)

Backfilling with waste rock is an old mining practice which has the significant advantage of keeping the crushed waste underground. This method, however, has the disadvantage that large passages still remain open through which cooled ventilation air may leak and pick up heat. Backfilling with tailings has been practised as far back as 1910, but was abandoned between 1930 to 1940. Since that time mines were changing over to ball mills and the sliming process with the result that the permeability and dewatering of the placed fill underground was severely affected.

Since 1977 backfill systems have been reintroduced in deep mines, using mainly either dewatered or deslimed tailings and to a lesser extent milled waste rock material, cemented tailings or cemented tailings/aggregate mix. If done properly with paddocks and full contact with the hanging wall, air leaks can be minimized. The beneficial results of backfilling are:

- improved safety conditions in deep level mines by reducing the incidence of rockbursts;
- increased lifetime of mine and increased recovery of gold since stabilizing pillars containing up to 30% of the ore reserves of a mine could largely be dispensed with;
- reduction of heat flow into the stope with consequent reduction of refrigeration plant load;
- reduction in timber requirements for stope support with a profound effect on timber yard capacity, loading and unloading equipment, labour, hoisting costs and shaft availability; and
- reduction in fire hazard.

There are three types of backfill materials in use:

- deslimed tailings with a relative density of about 1,65 (65% solids);
- dewatered slimes at a relative density of 1,90 to 1,98 (75% to 78% solids); and
- crushed waste rock, 6 mm to 25 mm, mixed with cement.

Addition of ferrous sulphate at a rate of 0,2 kg/t of tailings to inactivate any residual cyanide is required. Case studies (COMRO, 1988a) for scattered and longwall mining situations to evaluate the mine wide effects of backfilling on refrigeration and ventilation requirements indicated that the mine wide refrigeration requirements could typically be reduced by between 10 and 25%.

At stope face, backfilling with tailings from the metallurgical plant has been found to reduce the heat flow by up to 50%.

Consequently, cooling/refrigeration requirements would be reduced, as well as the concomitant requirements for low TDS water.

The costs for preparing and placing suitable backfill material are outweighed by the savings in refrigeration, timber handling, labour and the increase in recovery of gold since stabilizing pillars can largely be dispensed with. During 1989, 18 mines were using about $3,6 \times 10^6$ t of tailings for backfill, which was equal to about 3% of the ore milled. It is expected that the amount of backfill will increase to about 15×10^6 t/a within the next ten years (Nami, 1990).

(viii) Hydro-lift mine-cooling system (Sheer *et al.*, 1986; Adams *et al.*, 1986)

In a conventional surface refrigeration system sending chilled water down the mine, the power requirements to return warm water from underground by far exceed the power needs for refrigeration - if no energy recovery device has been provided.

When a Pelton Wheel turbine is installed, where up to 66% of the energy in the down-going chilled water can be recovered, the electrical energy required for the refrigeration plant to produce the *coolth* is about equal to the energy required for pumping the warm water back to the surface for a system designed to operate at a depth of 2 000 m.

The three-chamber mine-cooling system developed in Hungary, implemented in West Germany and successfully put into operation at the Heinrich Robert coal mine over three years ago, offers the advantage of substantial power savings for returning warm mine water. Since the pressure in the rising main is effectively balanced by the down-going pressure, only a small amount of energy is required to overcome the frictional resistance. At the Heinrich Robert coal mine operating at 970 m below the surface, only about 100 kW of energy is used to return the warm water to the surface as opposed to approximately 1 650 kW which would be needed in a conventional installation operating without an energy recovery device. The pumping power needed for that distance is therefore reduced to only 6%.

In short, chilled water or slurry ice from the surface displaces the warm water underground from each of the three parallel chambers and in turn forces the warm water, or even dirty warm mine water, up to the surface with the assistance of a pump to overcome the friction. Since the chilled water, or the slurry ice, will not gain heat when going down as is the case with conventional methods of bringing down cold water, - (temperature gain 2,3°C/km depth) - less surface refrigeration capacity is needed, resulting in a saving of water for heat rejection on the surface.

Because the three-chamber system is capable of passing coal slurries through the chamber as effectively as clear water, the following media can also be considered for transportation by the system

- slurry ice to underground
- dirty run-of mine water to the surface
- crushed ore and crushed waste rock to the surface.

The possibility of passing slurry ice through Pelton Wheels to obtain some energy recovery has been examined but has not been found to be feasible. Here, the three-chamber system is superior.

The possibility of passing crushed ore through the valves must first be tested, since the hard quartzitic rock differs from the soft, crushable coal, transported to the surface at the Heinrich Robert coal mine. The tonnage hoisted by the hydraulic method can be considered as supplementary beyond the normal shaft hoisting capacity.

Because there are three distinct operations in the cycle, i.e. filling, pressure change and transportation, the three chambers operate 120 degrees out of sequence, resulting in a continuous operation and flow through the system.

The operation of the valves of the three-chamber system with slurry ice is shown as Fig 15.

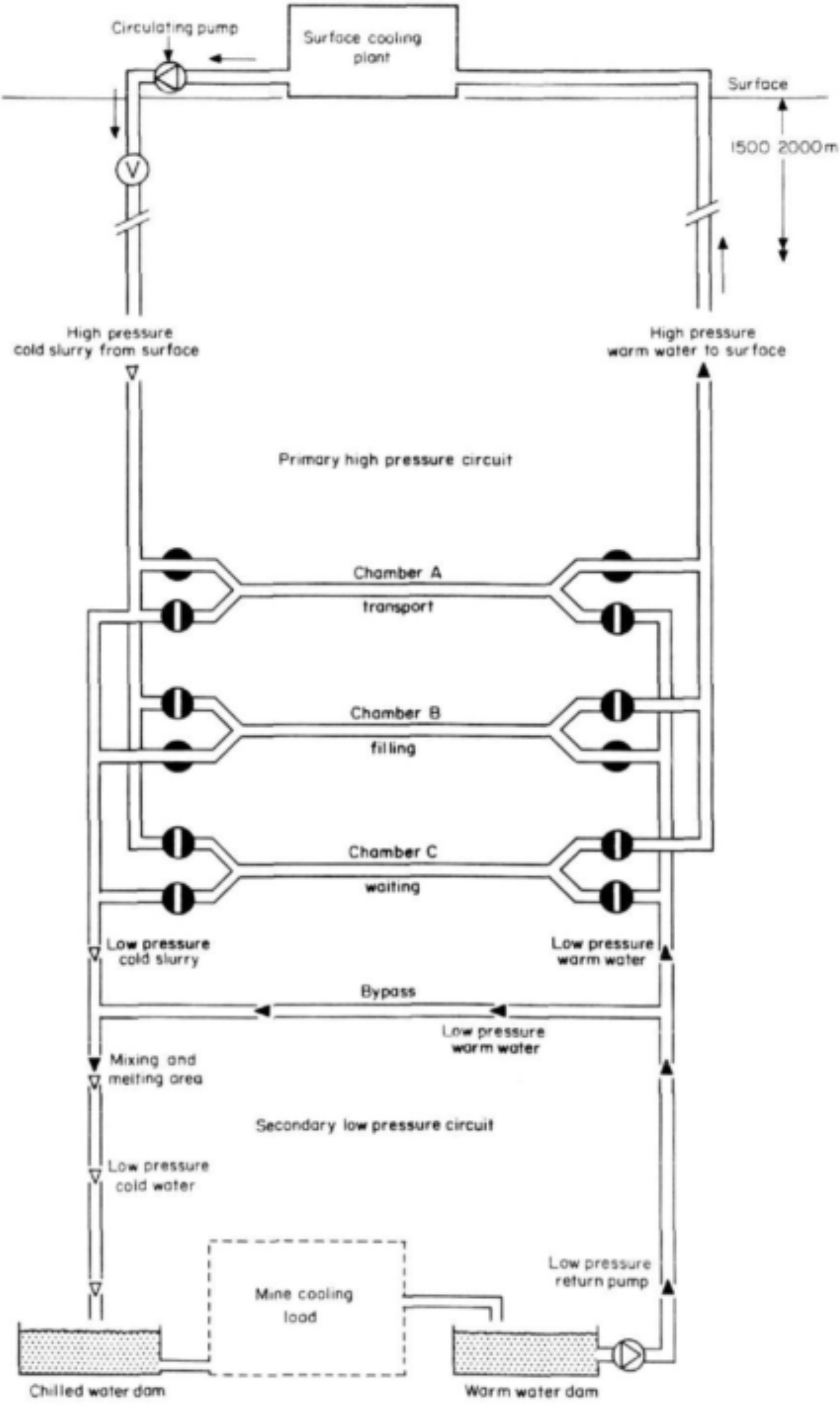


Figure 15
Three-chamber system used as a heat exchanger with slurry ice

The pipe chambers underground are 250 mm in diameter, each 220 m long, giving a chamber volume of 10,8 m^3. In order to site all the valves in a single location, the pipe chambers are of a *hairpin bend* design, which allows valves and hydraulic controls to be centralized.

The secondary circuit pipes which convey the cold water to the working face and the warm return water back to the pipe chambers are initially 350 mm in diameter. Owing to the length of pipe in the secondary circuit totalling more than 30 km, the secondary circuit pump is rated at 14 MPa. To reduce the loss of "coolth" and to lower the friction factor, the pipes will be lined with HDPE (high-density polyethylene). The advantages of the three-chamber system transporting the cooling medium into the mine can be summarized as follows:

- substantial power savings with respect to the pumping of hot return water to the surface for subsequent re-cooling and reuse;
- ability to pass slurry ice, thereby transporting greatly increased cooling capacity to underground;
- ability to pump dirty run-of mine water to the surface, without the necessity for prior underground settling, thereby eliminating capital and maintenance costs for underground settlers, clear water sumps, clear water pump chambers, mud water pump chambers and the like; and
- the probability of large-scale pumping of suitably sized pre-concentrated ore to the surface.

The three-chamber hydro-lift and the hydro-power systems need not be incompatible and could probably be combined, provided that the quality of the high-pressure cold water meets the quality criteria for hydro-power equipment.

Because of its attractiveness in conserving energy by reducing the heat gain of the down-going cooling medium through eliminating the Joules Thompson heating effect, and the savings in pumping costs incurred in returning the hot water to the surface, the three-chamber system is being seriously considered for a number of shaft systems. At Western Deep Levels, a 350 t/d pilot plant for transporting slurry ice down to a depth of 1 200 m by a hydrolift system is expected to be operational around mid-1990.

3.3 **Hydro-power** (Du Plessis *et al.*, 1988; Joughin, 1986)

Traditionally, gold mining operations are largely dependent on compressed air and electricity. Compressed air power for large-scale underground operations such as rockdrilling is relatively wasteful and inefficient. Electrical power, though more efficient, is only suitable for certain types of application, and constitutes a potential safety hazard.

Both compressed air and electricity underground can be replaced on a large scale by hydro-power.

Service water, chilled on the surface, is taken underground via a strong-walled pipe. In a mine, the water column can be 2 000 to 3 000 m deep and this produces a high hydrostatic pressure at the bottom of the pipe. At 2 000 m water is released at a pressure of about 20 MPa. This water can then provide hydraulic power and cooling to the working areas,

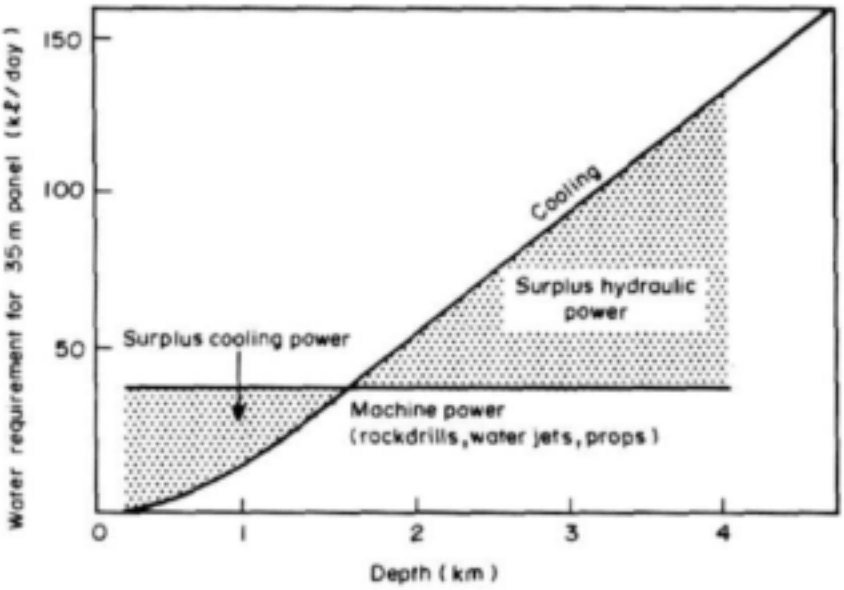


Figure 16
Water requirements for cooling and machine powering

thus satisfying both needs with one energy distribution system. The quantities of water for cooling and hydraulic requirements are equal at a depth of about 1 500 m whereas at depths greater than 1 500 m a surplus of hydraulic power would be available (Fig 16).

The use of hydro-power improves cooling of the working area through more consistent use of chilled water and by eliminating the additional heat input associated with other forms of energy (electrical, diesel). One of the attractions of hydro-power is that it is highly practical for use in mines, consisting simply of pipes and valves which have a good reliability record.

A programme to develop hydraulic rockdrills which are superior in performance to pneumatic drills and less noisy, was started by COMRO in 1975. Hydraulically operated rockdrills have been developed and fully tested underground and are now gradually being introduced by a number of mines.

Two alternative powering technologies have been developed -

(i) *Electro-hydraulics* refers to a system whereby high-pressure water or emulsion is obtained from electrically driven pumps which pressurize the service water to the required working pressure of typically 16 MPa.

Electro-hydraulics is the technology to be applied exclusively down to depths of 900 m, where the hydrostatic pressure of the water column and the volume of chilled water for cooling would be insufficient to operate hydraulic equipment, or even at greater depths where it would be inconvenient to have high-pressure pipes installed.

(ii) *Hydro-power* technology is applicable in deep mines where the water required for stoping is reticulated to the working areas at high pressure directly from the shaft water columns, typically at pressures above 15 MPa. The hydrostatic energy of the water is used to power positive displacement linear and rotary activators as for example in rockdrills. It can furthermore be converted to kinetic energy as water jets which can power turbines or move rock.

The basic alternative stope powering and water reticulation schemes for a mine are shown in Figs 17, 18 and 19 for:

(i) a mine powered by compressed air and electricity,
(ii) an electro-hydraulic powered mine, and
(iii) a hydro-powered mine operating on plain water.

Stoping equipment such as rock-moving water jets, blast hole cleaners, hydraulic props as well as small Pelton turbines to replace conventional motors driven electrically or by compressed air, e.g. for scraper winches, can all be powered directly from hydro-power lines with plain water.

Presently available hydraulic rockdrills require additives to be emulsified in water at a concentration of 2% to control wear and corrosion. The 98/2 emulsion is recirculated locally in the stope and is pressurized to 16 MPa by a water-powered emulsion pump referred to as a hydro-transformer. For these, emulsion additives have been developed with high hardwater stability. Total hardness levels resulting from calcium and magnesium salts as high as 1 500 mg/*l* can be tolerated.

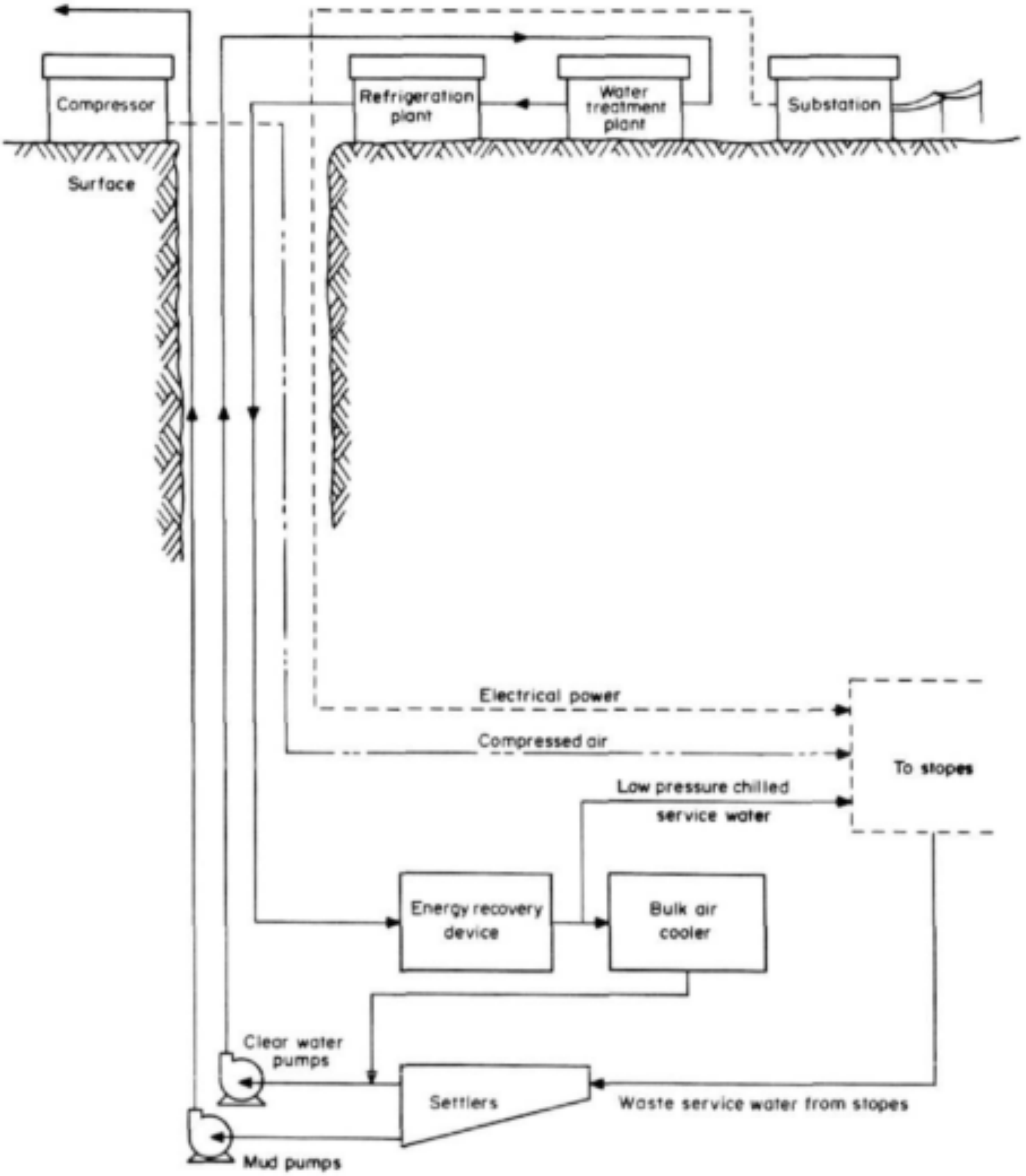


Figure 17
Basic stope powering and water reticulation in a mine powered by compressed air and electricity

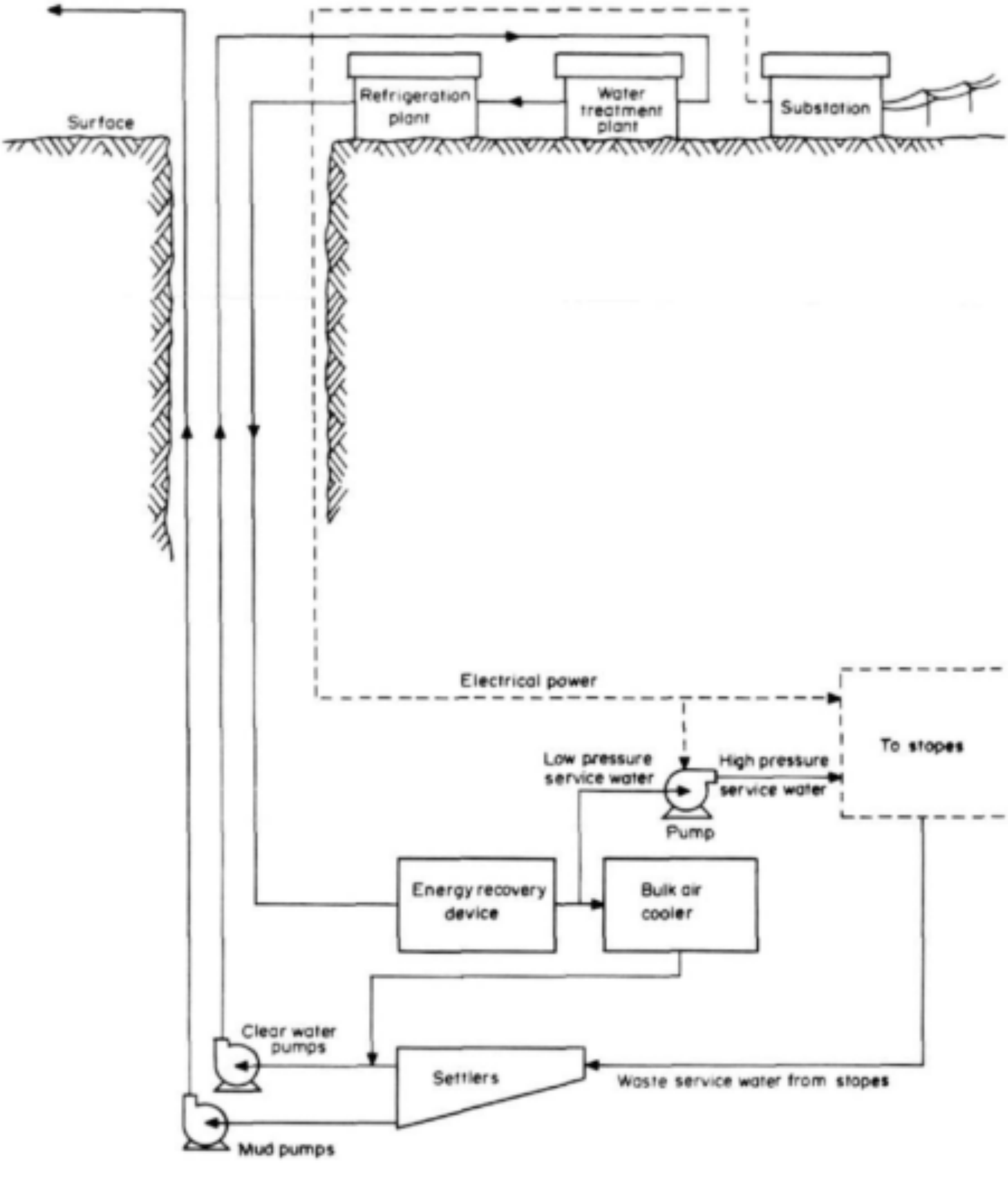


Figure 18
Basic stope powering and water reticulation in an electro-hydraulically powered mine

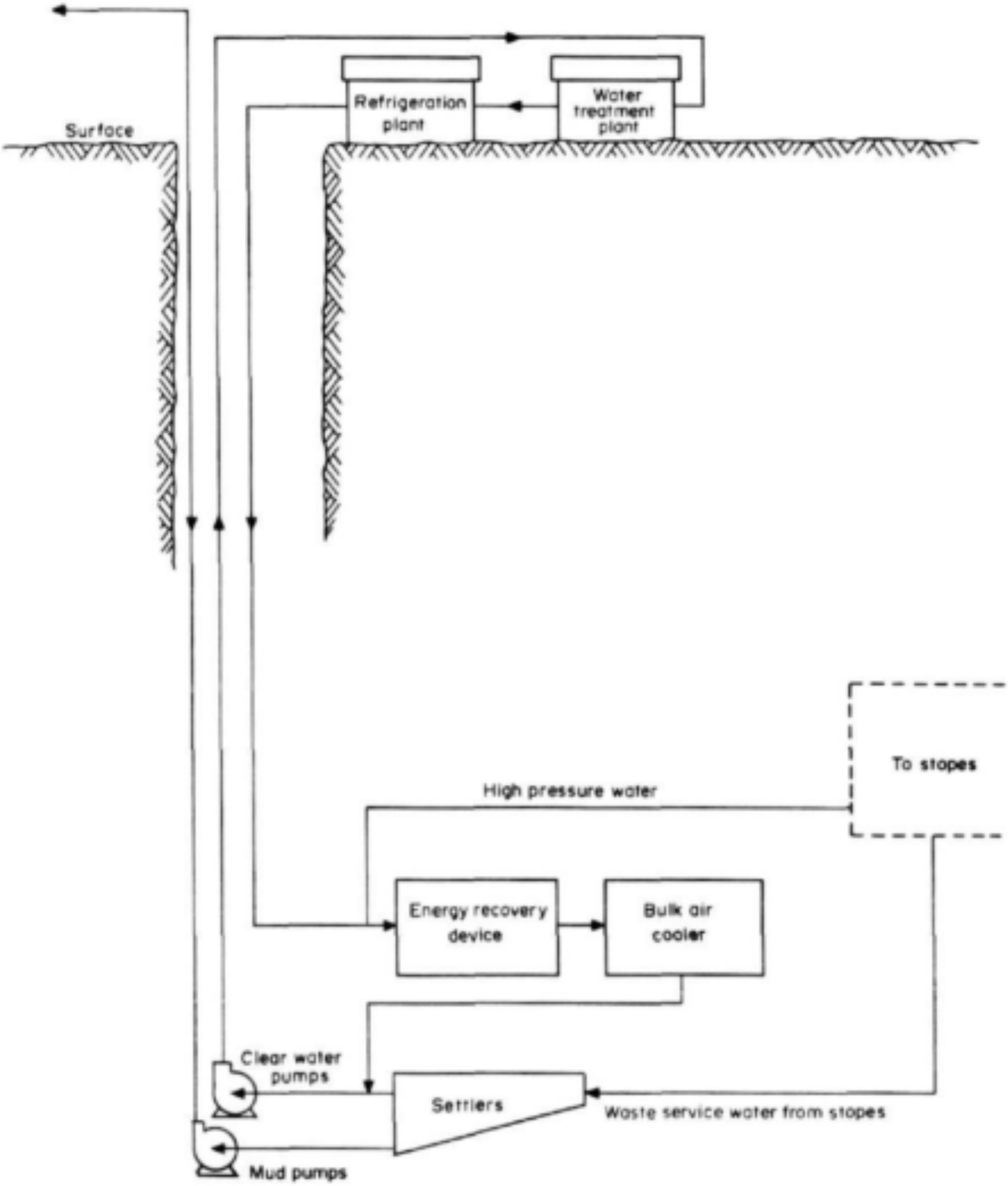


Figure 19
Basic stope powering and water reticulation
in a hydro-powered mine

The *emulsion-powered rockdrill*, already being installed by the mines on a large scale, provides numerous advantages over conventional pneumatic drills. These include:

- drilling rates more than twice those of pneumatic drills,
- reduced labour cost,
- reduced energy input,
- reduced noise levels, and
- elimination of exhaust fogging.

In the case of *water-powered rockdrills* operated directly by high-pressure plain water without the use of hydro-transformers, successful tests have been conducted with two different prototype versions. A drilling performance similar to that of emulsion-powered rockdrills has been achieved. Wear-resistant materials for the highly stressed internal components had to be developed first. Water-powered rockdrills are expected to be available commercially by 1992.

The electrical energy consumed in generating compressed air for pneumatic stope drilling has been found to be at least 15 kWh/t mined, assuming 1,5 m drilled/t.

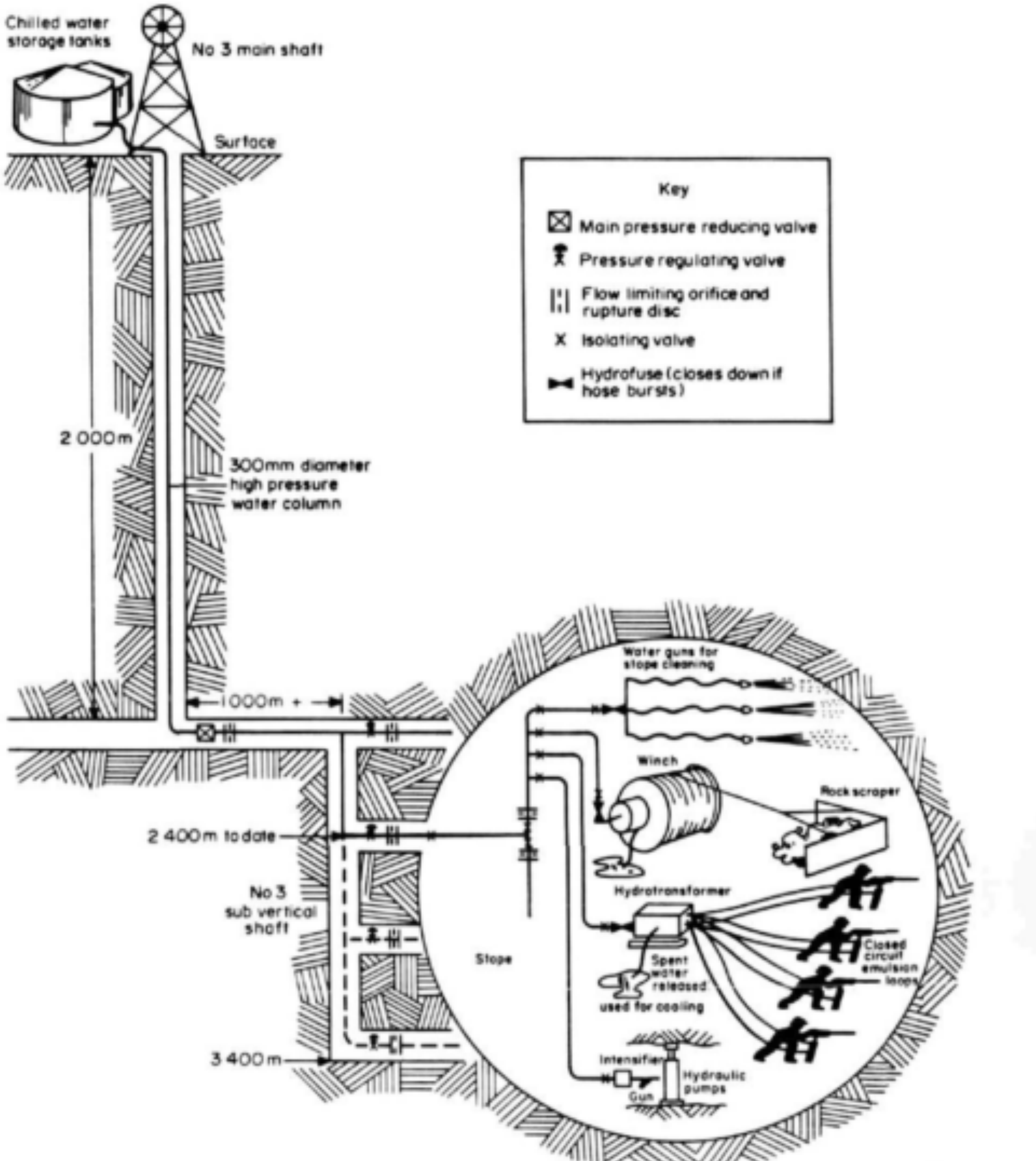
Measurements conducted over many tens of thousands of holes drilled electro-hydraulically under similar conditions indicate an electric energy consumption of only 1,5 kWh/t mined.

Although no direct electrical energy input is required for hydro-powered drilling with plain water, the hydraulic energy consumed by the drills could have been converted to electrical energy in a Pelton recovery turbine. This energy is, indirectly, the energy required for drilling, and is about 0,76 kWh/t mined.

A diagram for a typical hydro-power installation with emulsion-powered rockdrill is shown in Fig 20 (Chamber of Mines, 1986).

COMRO water quality guidelines for mine service water, including hydro-power systems are given in Table 5. Water conditioning equipment to treat mine service water to the required standard consists of surface and underground settling tanks, together with sand filters capable of handling suspended solids up to 500 mg/*l* and strainers located in the underground high-pressure system to protect specific components from debris generated within the pipes. These comprise a large 0,4 mm strainer to protect the automatic control valves near the shaft and small strainers of 0,4 and 0,1 mm to protect each item of hydro-power stoping equipment.

Hydraulic impact mining or non-explosive mining with hydro-power is currently being developed and tested by COMRO. Impact ripping, that is breaking the rock with a hydro-powered hammer, has been found to have the best potential for exploiting the natural rock fracturing which occurs in deep mines, below 2 000 m depth, around stope faces. Filtered mine service water is used as the driving force (Pickering and Haase, 1986).



All water joins the mine's normal pumping system for return to surface

Figure 20
Typical circuit diagram of hydro-power installation
(based on Kloof Gold Mine)

Table 5: Chamber of Mines Research Organization water quality guidelines for mine service water, including hydro-power systems

Parameter		Range and upper limits *
pH for	galvanised steels	7,0 to 8,5
	carbon steels	5,5 to 8,5
Suspended solids	general	30
	for close tolerance equipment e.g hydro-power	5
Chloride	general	limited by TDS
	chromium steels	250
Langelier saturation index		non scaling
$CaSO_4$ precipitation potential		non scaling
Total chlorine (as Cl_2)		0,5 to 1,0
Total dissolved salts (TDS)		10 000

** concentrations in mg/l except pH*

At present, corrosion of steel pipes presents a great problem. With mild steel pipes, the protective lining comes off, starting at pipe welds. Pitting corrosion occurs with stainless steel as well as with 3CR12. A probable solution might be the development of pressure resistant steel-reinforced epoxy pipes, able to withstand a pressure of 20 MPa and more.

The main problem areas identified were delays in the provision of services (spares and power), adverse mining conditions, and failure of the hydraulic components subject to pressure pulses.

In trials conducted at Doornfontein at 2 700 m depth throughout 1988, with mining rates consistently above target being achieved, low system utilization prevented the achievement over a prolonged period of 2 m^2 of rock mined per shift hour. The set-up of the hydraulically driven impact ripper, currently under evaluation, is shown in Fig 21.

Should this method of non-explosive mining be developed to a stage where it could replace conventional drilling or explosive mining, mining could be changed drastically from one-shift mining to continuous staggered shift mining. This would result in a continuous high rate of face advance and would alleviate underground travelling and shaft congestion.

3.4 Water distribution systems and corrosivity of mine waters

For water distribution underground, the piping often exceeds 100 km in length, while pumping heads often exceed 1 km. Until fairly recently, carbon or mild steel was used almost exclusively as piping material. Its strength, ductility, ability to withstand high pressure, impact resistance and ease of welding are all suited to the robust service conditions.

Experience, however, has shown that two characteristics of mild steel pipe are not ideal for mining, namely, limited corrosion resistance and mass.

The rate of corrosion and of mechanical damage promoting corrosion is most severe at the stopes and less near the shaft. Likewise, corrosive conditions are extreme at the shaft bottom, but mild at the shaft collar. In order to understand why corrosion control is frequently unsuccessfull, it is essential to describe the basic principles of the corrosion mechanisms. Corrosion may be defined as an electrochemical process in which differences in electrical potential develop between dissimilar metals, or between different areas of a single metal. These sites constitute *corrosion cells* (Fig 22). The anode is the region of lower potential, and the cathode the region of higher potential. At the anode, metal ions go into solution. Anodic reactions can be expressed as follows:

$$Fe \dashrightarrow Fe^{2+} + 2e^-$$

The electrons migrate through the metal from the anodic area to the cathode where they react in any one of several ways. Only a few examples are given:

(a)	$2H^+ + 2e^- \dashrightarrow H_2$	important in acidic solutions,
(b)	$2H_2O + 2e^- \dashrightarrow H_2 + 2OH$	occurs in natural waters,
(c)	$O_2 + 4H^+ + 4e^- \dashrightarrow 2H_2O$	occurs in aerated acidic solutions, and
(d)	$O_2 + 2H_2O + 4e^- \dashrightarrow 4OH$	important in naturally aerated waters.

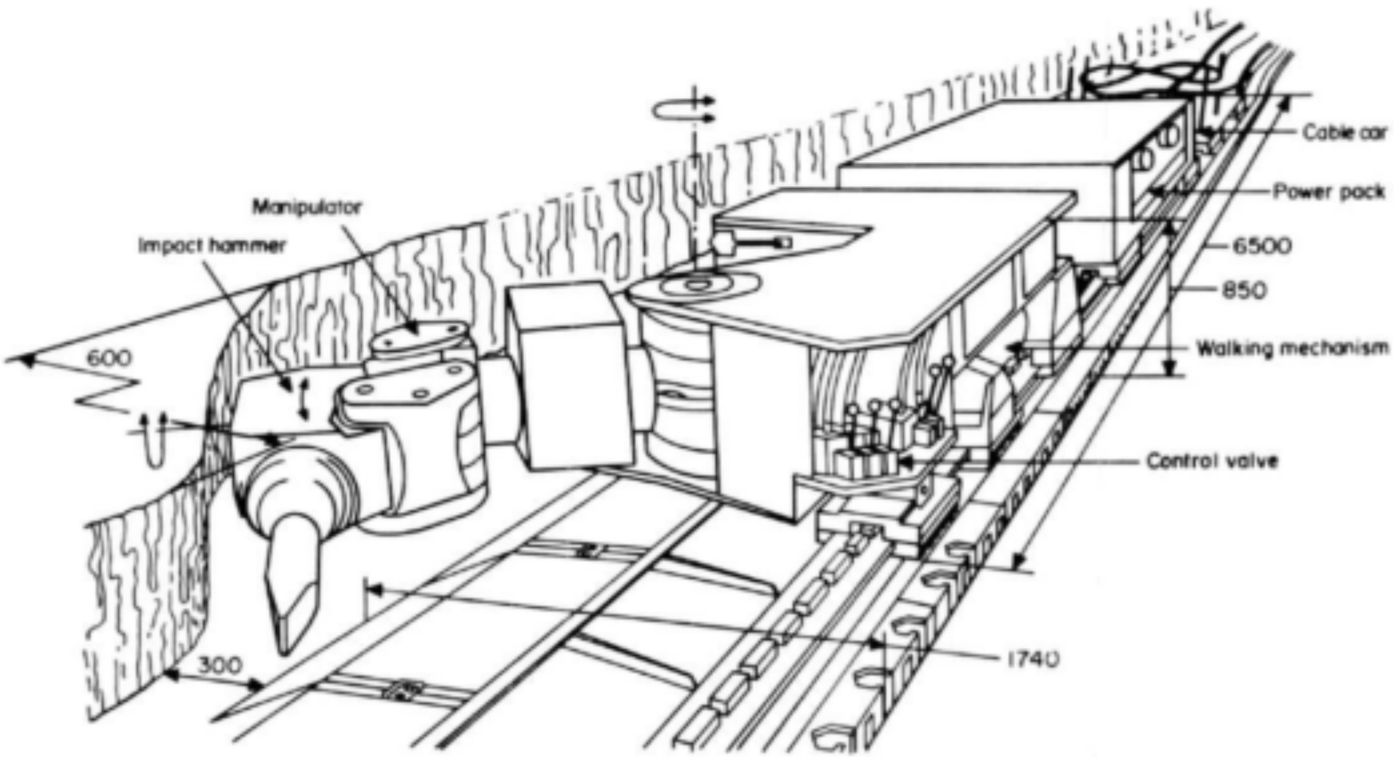
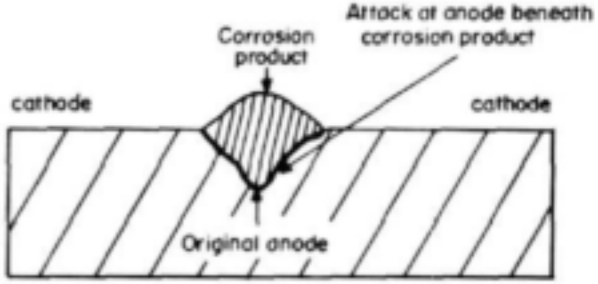


Figure 21
Impact ripping system currently under evaluation

Simple corrosion cell

Cross-section of an impingement pit



Pitting beneath porous deposit

Figure 22
Various forms of corrosion cells

A basic concept of kinetics states that the rate of a reaction can be controlled by the rate of its slowest step. Cathodic reactions are generally much slower than those occurring at the anode. Therefore, control of the overall corrosion rate can be accomplished by controlling the corrosion rate at the cathode. For mild steel the main cathode reactions - hydrogen evolution or oxygen reduction - will proceed if hydrogen gas is generated at the cathode, (reactions (a) and (b)), or if a sufficient supply of oxygen is available, (reactions (c) and (d)). Hydrogen is released at the cathode at low pH values (reaction (a)). In natural waters where the pH values are far too high to provide a sufficiently high concentration of hydrogen (reaction (b)) the pressure of dissolved oxygen usually controls the corrosion rate (reaction (d)). One corrosion control method, therefore, involves the reduction of the amount of oxygen available to the cathode surface of mild steel.

If the amount of oxygen diffusion to the metal surface can be controlled, the rate of corrosion can be reduced correspondingly. This is precisely the mechanism of cathodic corrosion inhibitors which form an impervious film to prevent the diffusion of oxygen to the cathode.

Anodic surfaces, on the other hand, can be protected by the formation of a thin, impervious oxide layer. Stainless steel naturally forms such films. Most metals, however, must be aided by the addition of anodic corrosion inhibitors such as chromate, nitrites and others (White, 1985a). In practice many forms of corrosion exist, for example:

- pitting corrosion
- under-deposit corrosion
- crevice corrosion
- stress corrosion
- water line attack
- impingement attack
- cavitation
- formation of corrosion cells by dissimilar metals
- bacterial attack.

All these forms of corrosion are the result of either chemical or physical destruction of the protective films or the creation of galvanic cells.

Work by Mintek on corrosion problems encountered underground has highlighted low pH values and dissolved oxygen as the major factors controlling corrosion in piping networks used for mine water. Except for a few cases, the pH of mine water was generally within the range of 6 to 9, which is an indication that pH control, usually by lime addition, is satisfactory within most mines. Nevertheless, pH values of between 3 and 4 were not uncommon, even in *off-line* refrigeration plants.

The dissolved oxygen level was generally at saturation levels, 7 mg/*l* or higher, depending on the depth and temperature of the water. A few low concentrations of less than 4 mg/*l* of dissolved oxygen were found, chiefly in areas of fissure water with a high concentration of sulphides (White and Higginson, 1984).

Sulphates and chlorides contributed to the greater portion of dissolved salts, which varied from 600 mg/*l* to 8 000 mg/*l* but were normally within the range of 500 mg/*l* to 1 000 mg/*l*. Chloride concentrations showed two peaks, around 1 000 mg/*l* (average value on the Evander gold-field) with about 2 000 mg/*l* being the norm in the mine waters of the Orange Free State. Nitrate and ammonia were minor constituents, deriving from blasting operations. However, values of up to 80 mg/*l* (as N) have been recorded in mine water.

The initial field survey showed that above a pH value of about 5,5 the corrosion rate was fairly independent of pH for aerated and de-aerated artificial mine water solutions under laboratory conditions. However, de-aeration in these neutral pH ranges gave an average reduction of 75% in the corrosion rate.

Fig 23 is based on determinations of mass loss (mild steel) with artificial mine-machine waters and shows the relation between the corrosion rate and pH for aerated and de-aerated mine water at a constant stirring rate of 100 r/min. De-aeration results in a decrease in the corrosion rate of around 90%. Almost all the benefits of de-aeration are achieved by reducing the dissolved oxygen content to 1 mg/*l*. This could be of importance if consideration is to be given to the chemical reduction of oxygen by the addition of sodium sulfite (Na_2SO_3) as a means of corrosion control.

For mine water with pH values of less than 4, the reduction in the corrosion rate with de-aeration was smaller owing to the contribution made by hydrogen ions to the cathodic reaction. Chloride is regarded as the main culprit of pitting corrosion because of its penetration of the protective oxide film of anodic stainless steel.

Mintek reported that no correlations could be found between corrosion rate and chloride, sulphate, hardness concentrations and conductivity for mild steel. These results cast doubt on the use of corrosion indices (Langelier, Ryznar) that are based on the relative concentrations of aggressive and scale-forming anions in mine waters . An increase in the chloride concentration from 0 to 3 000 mg/*l* - a value never exceeded in the sampled mine waters - had no noticeable effect on the corrosion rate of mild steel (Fig 24). This was true in both synthetic acid and neutral waters. A variation in the sulphate content while the total hardness was kept constant, gave similar results to those for chloride.

Welded pipes often exhibit severe localized attack in the weld areas, commonly known as *tramline* corrosion. Experience in the mining industry indicates that water conductivity rather than the composition of total dissolved salts is the principal corrosive factor. Heat treatment (annealing, normalizing) of the finished welded pipe or the weld area can prevent attack in case of submerged arc welding and electric resistance welding.

Because of the complexity of the corrosion mechanisms and the unpredictability of many corrosion inhibitors (mainly film-forming) which in their effectiveness are limited to narrow pH ranges, protective coatings or corrosion-resistant materials of construction are widely introduced to the mining industry. Hot-dip galvanizing offers good protection to shaft steelwork since it can tolerate a degree of damage because of the sacrificial nature of zinc coating. Sophisticated pipe coating materials have been used satisfactorily to combat corrosion in pipes which are not heat-treated. The combination of heat treatment and protective coatings appears to be a practical solution to the weld heat affected zone corrosion in the mining environment (White, 1985b).

The choice of corrosion-resistant materials for pipes for mine water reticulation is another solution. Plastic pipes, made from high-density polyethylene (HDPE), unplasticized polyvinylchloride (u-PVC) or polypropylene (PP) are resistant to corrosion by mine water. They are, however, limited in their exposure to high pressure. The low resistance of u-PVC, for instance to mechanical damage, the cost of joining, and the requirements of extra support detract from its use. Mild steel pipes, lined with rubber or glass fibre/polyester,

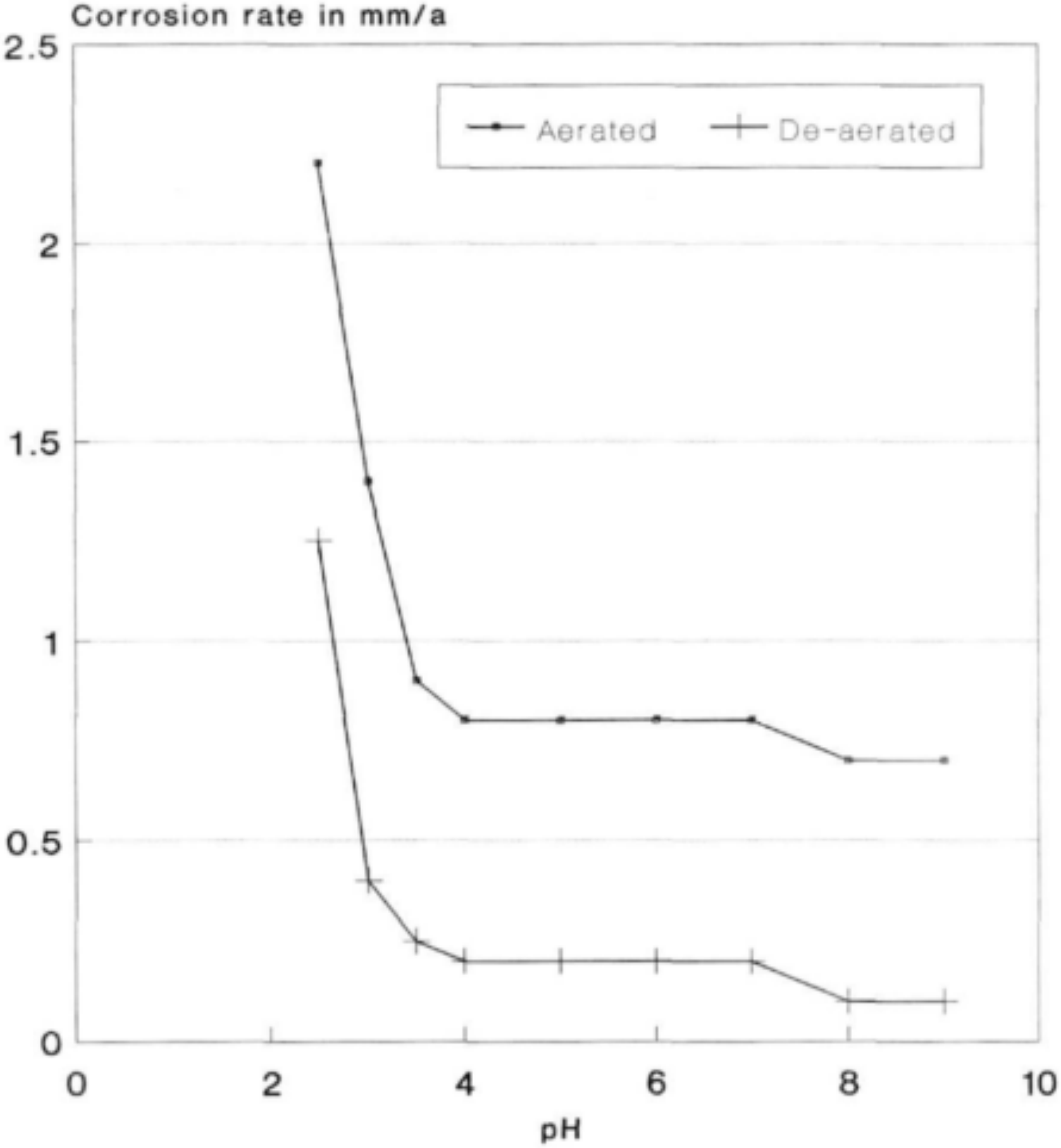


Figure 23
Corrosion rate of aerated and de-aerated mine water when stirred at 100 r/min at different pH values

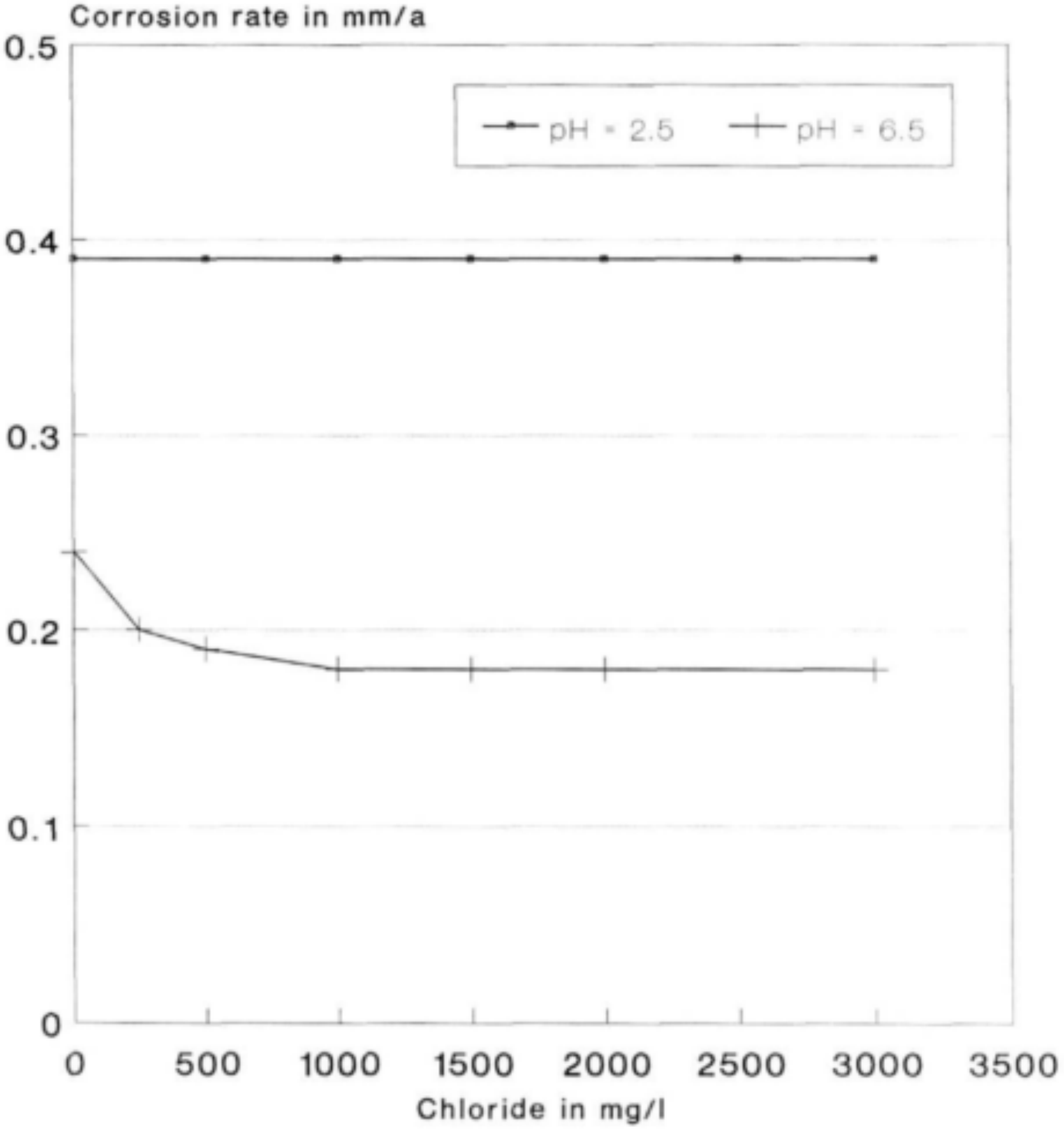


Figure 24
The influence of chloride on corrosion rate of mild steel at two pH levels

are now available locally.

Near or at the working area, the use of corrosion resistant stainless steel pipes is attractive. However, 3CR12 was found to be unsuitable for OFS mines because of the high chloride concentration of mine water of around 2 000 mg/*l*. Tested in mine water with 3 000 mg/*l* TDS and 500 mg/*l* chloride, most stainless steels, including 3CR12 perform well, an exception being welded A1S1 430.

Costs for coupling and general installation costs will be similar for all types of piping material. Life-cycle costing becomes an important exercise. As the movement of equipment underground is expensive, replacement of piping carries a high installation cost penalty. Piping installation costs account for about 75% of the costs of a piping network underground.

COMRO and the Mining Houses are currently testing new corrosion and abrasion resistant alloys. An extensive water quality improvement programme is also under way. The optimum solution is probably a combination of improved material and some water treatment. Corrosion tests on a large scale are carried out by Mintek on surface test stations as well as below ground on a number of mines. The resistance of 3CR12 steel to corrosion is being tested on behalf of the manufacturers, Middelburg Steel, by Mintek at various mines.

3.5 Surface handling of rock

3.5.1 Ore sorting

The first step in the recovery of gold from mined ores is normally sorting to reduce the mass of ore milled by eliminating dilution with waste rock from run-of-mine feed. When sorting by hand is practised, the rock has to be washed clean of adhering grime and mud to enable the ore and waste to be distinguished. Since uraninite is present in most ore, radiometric detectors have been developed to select discardable waste. These detectors have shown promise for certain mines, but gold grades and mass recoveries have generally failed to match those potentially available. Waste rock is either dumped, or sold because it is in demand e.g. as road building or concrete aggregate material.

3.5.2 Crushing and milling

The second step in the recovery process is to expose gold particles locked in the ore. Crushing reduces the size of ore from as large as 450 mm to less than 10 mm. In the milling stages, the 10 mm material is ground very fine, typically to less than 0,1 mm in size.

3.5.3 Classification

The product of the mills is diluted with water and passed to a hydrocyclone, a conical-bottomed vessel with a tangential inlet and outlets at top and bottom. Primary mill discharge, for example, with a water to solids ratio of 2 : 1, is fed via a constant head tank and is separated into a low-density slurry with a higher concentration of fines and into a high-density slurry containing a larger portion of coarse particles. The low-density slurry with a water to solids ratio of typically 4 : 1, is discharged via the top outlet and the high-density slurry with a water to solids ratio of 1 : 2 is collected from the bottom outlet. Water quality at this stage is of no concern.

3.6 Recovery of gold (Adamson, 1973)

A diagrammatical sketch of a typical gold-reduction plant is shown in Fig 25. After crushing and milling, gold is recovered partly by gravity concentration and partly by cyanidation. Slimes dams return water is used to augment the water in the milling circuit.

3.6.1 Gravity concentration

The coarse, dense particles recovered at the bottom of the hydrocyclone are gravity concentrated by passing over a table covered with riffled rubber matting, or through a rotating drum clad with riffled rubber mats, or over a slow-moving endless riffled rubber belt. A concentrate of gold and pyrite is then washed from the riffles and is refined by amalgamation with mercury or by treatment with strong cyanide solution. Gravity concentration is not practised by all mines. However, about 50% of South Africa's gold is produced by this process. The coarse ore, after gravity separation, is re-introduced to milling and combined with the fines.

3.6.2 Thickening

To concentrate the fines from the top outlet of the hydrocyclone the overflow has to be thickened. The mixture of water and fine solids passes to large thickener tanks with virtually flat bottoms. The solids concentrate at the bottom of the thickener is scraped by circular rakes as slime to the centre of the tank. Formation of thick slime is assisted by adding flocculants, typically lime or organic compounds derived from guar gum. The thickened slime is pumped to the cyanidation tanks and the clear water is returned to dilute the feed to the hydrocyclones.

3.6.3 Cyanidation

The purpose of cyanidation is to dissolve gold. The gold-bearing slime is conditioned by the addition of about 0,15 kg NaCN/t. Lime is added, if it has not been done before, at a rate of approximately 1 kg CaO/t of slime to ensure protective alkalinity. Sufficient unclarified pregnant solution is introduced to reduce the specific gravity of the pulp to 1,46 or 50% solids. This represents the optimum density for agitation to ensure maximum dissolution of gold whereas too dilute a pulp results in a high throughput at the filters and a need for additional precipitation capacity later on. Water quality at this stage is not critical.

When the gold-bearing pulp has been suitably conditioned with regard to specific gravity, cyanide content and alkalinity, it is pumped to the top of the air-agitated Brown tank or *pachuca* which is operated in a series of 4 to 6 tanks. Apart from providing an effective means of agitation and ensuring adequate contact between the gold particles and the cyanide solution, the compressed air also supplies the oxygen needed for the dissolution reaction:

$$4Au + 8NaCN + O_2 + 2H_2O \dashrightarrow 4NaAu(CN)_2 + 4NaOH$$

Cyanide is consumed during agitation partly through contact with decomposition products of pyrite, and partly through the action of CO_2 present in the compressed air supply. The total air-agitation period for maximum gold dissolution varies from 15 to 45 h, depending on factors such as fineness of free gold particles, degree of pyrite encasement, and consistency of slime grading. A residual cyanide concentration must be maintained.

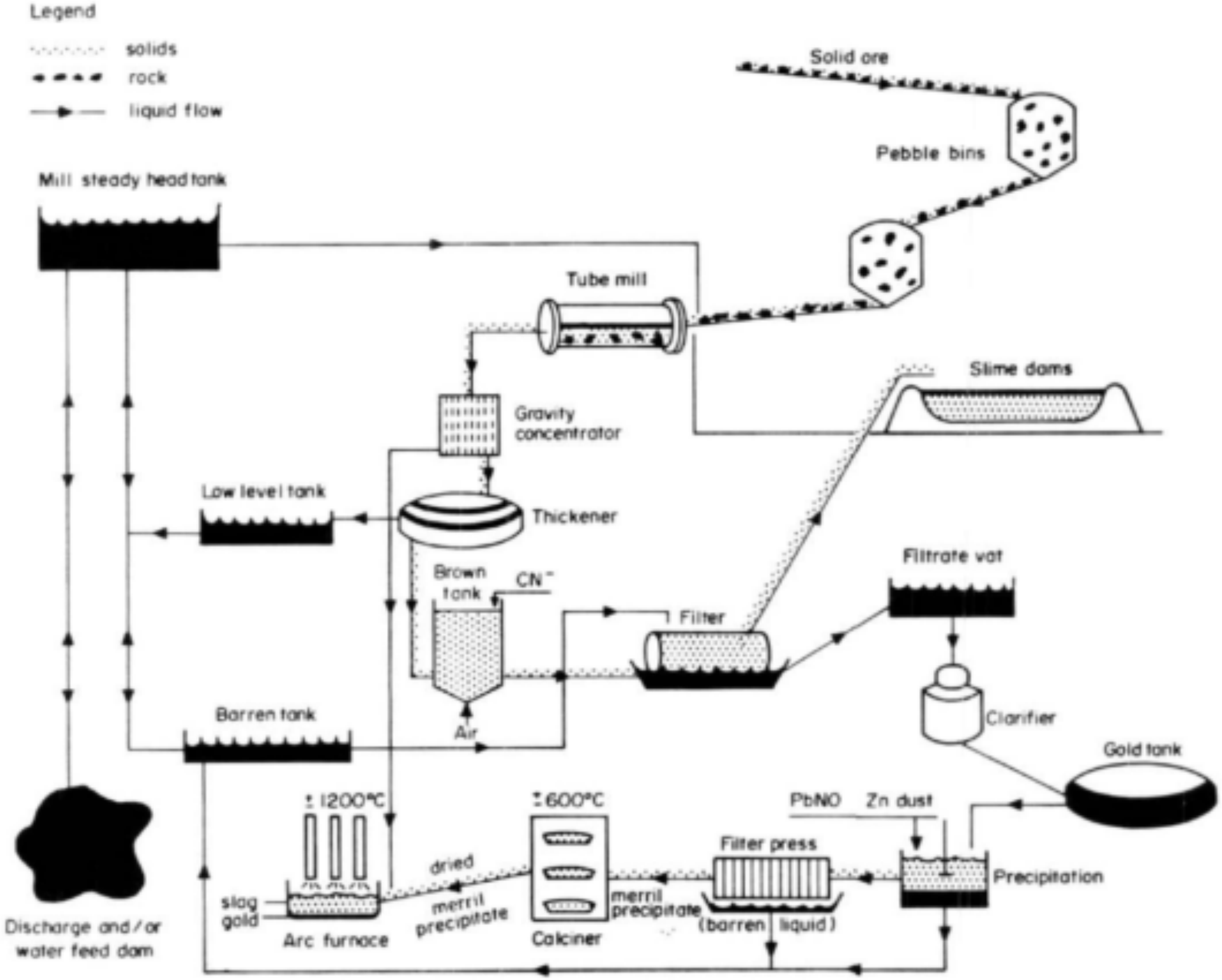


Figure 25
Flow sheet of a typical gold reduction plant

3.6.4 Filtration

Separation of the gold-bearing cyanide solution from slime is currently conducted by filters, e.g. rotary drum or horizontal belt vacuum filters. Due to the presence of calcium hydroxide in the cyanide solution $CaCO_3$ precipitates in the interstices of the filter cloth, thus gradually reducing the filtration rate. Calcium carbonate is formed by the CO_2 present in the agitating air of the pachucas. Fine particles of slime also clog the cloth. Therefore, to maintain filtration capacity, it is necessary to clean the filter cloth by periodic application of 2,5% to 3,5% HC*l*, followed by vigorous brushing of the drum surface. The residue slime, which is blown off from the filter by compressed air, is discharged via a conveyor belt to a repulping tank prior to being pumped to the tailings dam.

3.6.5 Precipitation and smelting

The precipitation of gold from the filtered cyanide solution is by contact with zinc dust, and the addition of a small amount of lead nitrate. Prior to entering the precipitation plant, the gold-bearing filtrate is further clarified by filtration to retain any fine slime particles which would otherwise coat the zinc dust. In order to reduce the amount of zinc dust needed for precipitating the gold, dissolved oxygen in the pregnant solution is removed by vacuum in a deaeration (Crowe) tank. The chemical reaction for precipitating gold from the cyanide solution can be expressed by the equation:

$$\mathrm{Zn + 2NaAu(CN)_2 \dashrightarrow Na_2Zn(CN)_4 + 2Au}$$

The zinc/lead gold precipitate is then removed from the solution by filter presses (Stellar filters). The cake in the Stellar filters is transferred to an acid vat where sulphuric acid is added to dissolve excess zinc and other soluble constituents. A considerable volume of inflammable gas , (a mixture of H_2 and H_2S) is generated. On completion of the acid treatment, the gold slime is dewatered on filter presses. After filtering and airdrying, the gold slime is roasted by calcining at about 600°C. Alternatively, the gold/zinc lead precipitate from the first set of filter presses may be dried by calcining without prior acid treatment. On leaving the calcining furnace the product is transferred to electrode arc furnaces and a borax/silica flux is added. The charge is smelted at approximately 1 200 to 1 300°C for a period of two hours and then poured into a cascade system of moulds. The resulting gold bars are despatched to the Rand Refinery for refining.

3.6.6 The carbon-in-pulp (CIP) process

The carbon-in-pulp process is used to recover gold directly from the cyanide leached pulp by adsorption onto a suitable granular activated carbon without the need for the costly and less than perfect solid-liquid separation as obtained through filtration. The loaded carbon is separated from the slimes by e.g. a DSM screen and is then eluted with hot NaCN under pressure to provide a solution containing the gold, which is recovered by either direct electro-winning or by zinc precipitation and subsequent smelting. Financial savings due to the use of CIP may be considerable. Because of the smaller volume, the capital cost of the post-cyanidation section is said to be about 50% lower than that of a conventional gold plant and the operating costs about 30% less. Also of importance is the simplicity of the operation.

In the CIP process, the screening of large tonnages of finely-ground pulp is an essential pretreatment step for removing coarse particles and particularly woodchips resulting from the use of timber underground, before adding granular activated carbon. Adsorption of gold takes place in mechanically agitated tanks, where the cyanide conditioned slime with a density of about 1,46 or 50% solids is thoroughly mixed with granular activated carbon, typically < 3 mm in size. The gold then slowly precipitates onto the carbon. There are usually 4 or 5 tanks in series and the agitated slime suspension flows by gravity from the uppermost tank to the lowest one. Fresh granular carbon is added to the lowest tank and is air-lifted counter-currently. Contact time between the carbon and the cyanidized pulp is usually between 20 and 60 min/stage. When the carbon loaded with gold, silver, nickel, copper, iron, calcium, and/or silica reaches the uppermost tank, it is recovered by screening, washed free of mud and then transferred to the elution plant.

At the elution plant, the loaded carbon is first treated with HCl to remove the $CaCO_3$, magnesium, some of the nickel and silica. Elution of gold and silver from the carbon is effected by a hot solution (90 to 110°C) of NaOH (1%) + NaCN (1% to 5%) at atmospheric or an elevated pressure in 10 m high, 1 m diameter elution columns with a contact time of 1 to 9 h, followed by a wash with softened water.

The eluant passes through electro-winning cells where the gold and silver are recovered by electrolysis. The gold/silver-loaded cathode is then transferred to the electric arc furnace with a flux added. The resulting gold bars are sent to the Rand Refinery.

Eluted carbon does not have the same adsorption capacity as new carbon. Thermal reactivation by passing the spent carbon through a kiln at 600 to 700°C destroys organic material adsorbed during the preceding stages and restores the adsorption capacity (Moore *et al.*, 1986).

The simplicity and low cost of the carbon-in-pulp route for recovering gold from cyanide solutions has enhanced the interest in exploiting low-grade gold sources, e.g. sand dumps and tailings dams. The application of CIP in combination with cyanide heap leach extraction of gold is shown in Fig 26 (Holloway, 1985).

The following considerations for water quality apply:

In a typical cyanide leach, complexes of gold, copper, nickel, cobalt and iron occur and compete for pore volume on the activated carbon granules. Coating also interferes with adsorption. The following causes for the reduced capacity of activated carbon to adsorb gold from cyanide leach have been identified:

- the presence of large organic molecules, e.g. in treated sewage effluent, or solvents or flotation reagents from preceding processes (Bosman and Burgess, 1985);

- the high magnesium and calcium content of shaft water used in the process, leading to a fine floc being precipitated upon the addition of alkali;

- a high soluble copper content which overwhelms the normal displacement of copper by gold; and

- the presence of ferro-ferricyanide Prussian Blue from the partial decomposition of pyritic materials giving a finely divided precipitate which stains the activated carbon.

It has, however, been found that highly saline water free from entrained organic matter,

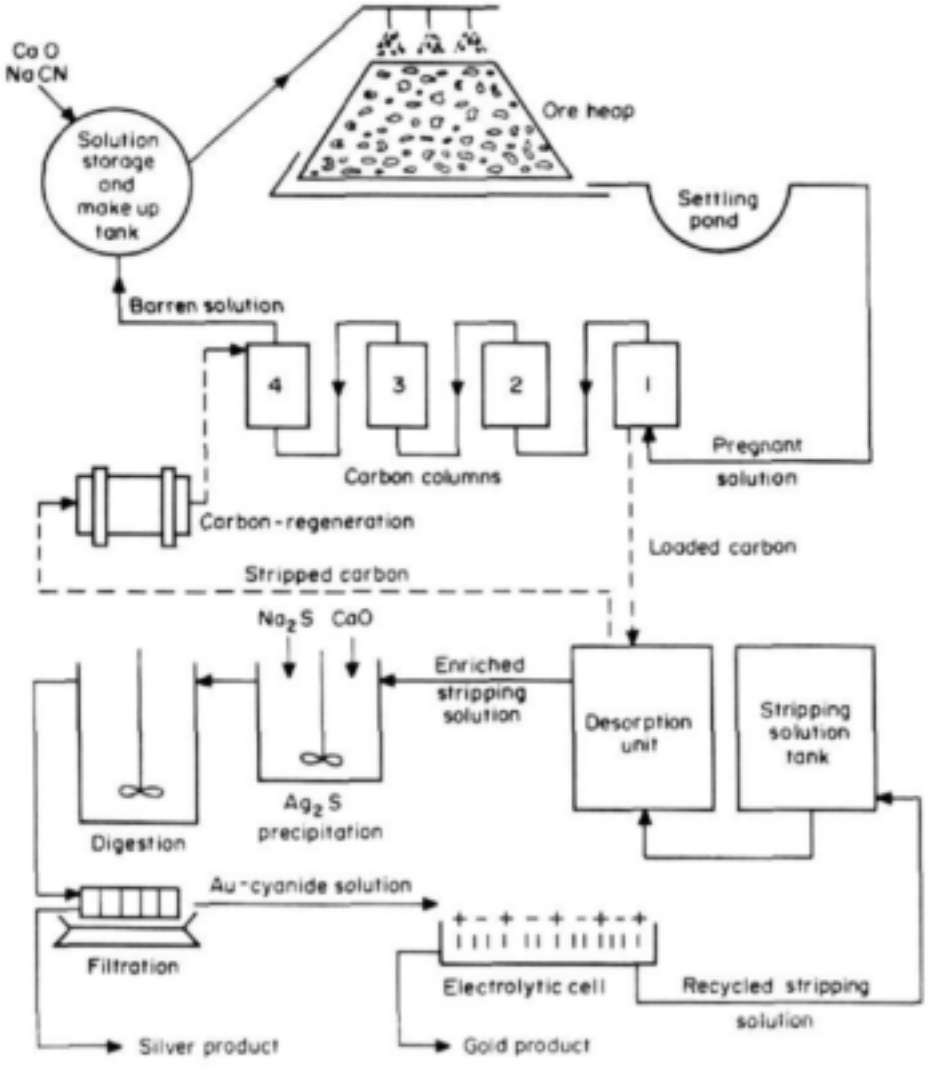


Figure 26
Circuit for heap -leach extraction of residual gold by cyanide and adsorption onto activated carbon (CIP)

e.g. slimes dams return water, is in many cases suitable for CIP or CIL (carbon-in-leach) recovery (Bosman and Burgess, 1985).

Treated sewage would have to be pretreated by anthracite-based activated carbon in a separate column to remove residual organic chemical compounds before being used as a medium for repulping the thickened slime with cyanide and contact with coconut shell-based activated carbon. Because of these pretreatment stages, the reuse of treated sewage is not as attractive as was originally assumed.

For the elution stage, water of low ionic strength, that is softened water, must be used.

3.6.7 Resin-in-pulp (RIP) process (Fleming and Cromberge, 1984)

Strong and weak-base anion resins extract gold cyanide from conditioned pulps under the same operating conditions as those generally adopted for CIP processing (Fleming and Cromberge, 1984). Resins are:

- not as temperature sensitive,
- less affected by organic poisons such as flotation agents and solvents,
- can be eluted at ambient temperatures, and
- do not require thermal reactivation.

Resins suitable for gold extraction are, however, softer, less abrasion resistant, of smaller-size than activated carbon granules, are more expensive.

In spite of some successful trial runs at some local gold reduction plants, the RIP process has not yet been introduced on a large scale in South African gold recovery plants.

3.6.8 Pretreatment of ore by roasting or bioleaching

The bulk of the gold in the Eastern Transvaal gold field is present as auriferous pyrite or as arsenopyrite. Gold associated with pyrite, arsenopyrite, other sulphide containing minerals, antimony and lead minerals and graphite cannot be fully recovered by direct cyanidation.

One route to the economic recovery of gold is, therefore, based on the flotation of sulphide concentrates, roasting to convert pyrite and arsenic sulphides into SO_2 and iron/arsenic oxides, followed by cyanide lead to extract gold. Roasting plants release SO_2 and As_2O_3 into the atmosphere unless these components are removed by expensive after-treatment, to recover SO_2 after oxidation to SO_3 and quenching and to reduce the As_2O_3 elemental arsenic. There are three roasting plants operating in the Eastern Transvaal.

A second route involves solubilisation of metals by bacterial action. The bacteria responsible obtain their energy from the oxidation of inorganic material such as sulphides, sulphur and ferrous iron. Oxygen and carbon dioxide from the air are utilized. Trace amounts of salts of magnesium, calcium, phosphorus and potassium are also necessary. Nitrogen is essential, but as these micro-organisms do not fix nitrogen from the air, their only source is whatever ammonium salts are dissolved in the local waters. There are a large number of chemolithotrophic bacteria. Those most commonly known to be involved in the leaching of ores are listed below with the inorganic matter they oxidize (Bosch, 1987):

Micro-organisms	Oxidized inorganic molecules
Thiobacillus ferro-oxidans	S, S^{2-}, $S_2O_3^{2-}$, Fe^{2-}
Thiobacillus thio-oxidans	S, S^{2-}, $S_2O_3^{2-}$
Leptospirillum ferro-oxidans	Fe^{2+}

Bioleaching is cheaper and pollution-free.

There is presently one bioleaching plant operating in the Eastern Transvaal, treating some 300 t of ore concentrate per month, yielding gold of 100 to 150 g/t or up to 45 kg/month.

3.7 Uranium extraction

The chief uraniferous constituent in the gold-bearing reef is uraninite, of variable composition $xUO_2\ yUO_3$. Whereas uranous oxide UO_2 is only partially soluble in dilute acids, uranyl oxide UO_3 dissolves readily in these dilute acids. Complete dissolution of uraninite requires oxidizing conditions. Oxidizing agents such as ferric iron, sodium chlorate or pyrolusite (MnO_2) are effective for oxidizing uranium from the four-valent (IV) stage to the uranyl ion in the hexavalent (VI) stage. Pyrolusite is the most freely available and therefore the common oxidant. Acid leaching of uranium from the milled ore proceeds according to the following equation:

$$UO_2 + MnO_2 + 2H_2SO_4 \dashrightarrow UO_2\,SO_4 + Mn\,SO_4 + 2H_2O$$

Acid oxidative leaching with steam injection to raise the temperature and concentration of the dissolved uranyl sulphate is done in a series of usually flat-bottomed pachucas. The pregnant solution is separated from the barren slime either by rotary drum filters, horizontal belt filters, disc filters or by counter-current decantation (CCD), followed by sand bed filtration. The flow scheme for a CCD acid leach and solvent extraction (PURLEX) plant at President Brand is shown in Fig 27 (President Brand, 1986). For the acid leaching of uranium oxide, either the *forward* or the *reverse* leaching sequences can be applied. In a forward leach, gold is leached first, and uranium afterwards.

The large quantities of H_2SO_4 and of MnO_2 used for leaching the uranium oxide are then, after neutralization with lime to a pH above 9,5, discharged to the tailings dams. Altogether, about 41,6 x 10^6 t of tailings were processed during 1989 by 9 mines for the extraction of uranium (Atomic Energy Corp of SA Ltd, 1990). Due to the pollution problems encountered with manganese, some uranium extraction plants have replaced MnO_2 as the oxidant with ferric sulphate.

3.7.1 Uranium recovery

Recovery of uranium after oxidative leaching from the acid pregnant solution, containing about 5 g/*l* of free H_2SO_4, is achieved by either of the three processes: (plants in operation during 1989 are given in brackets) (Boydell and Viljoen, 1982):

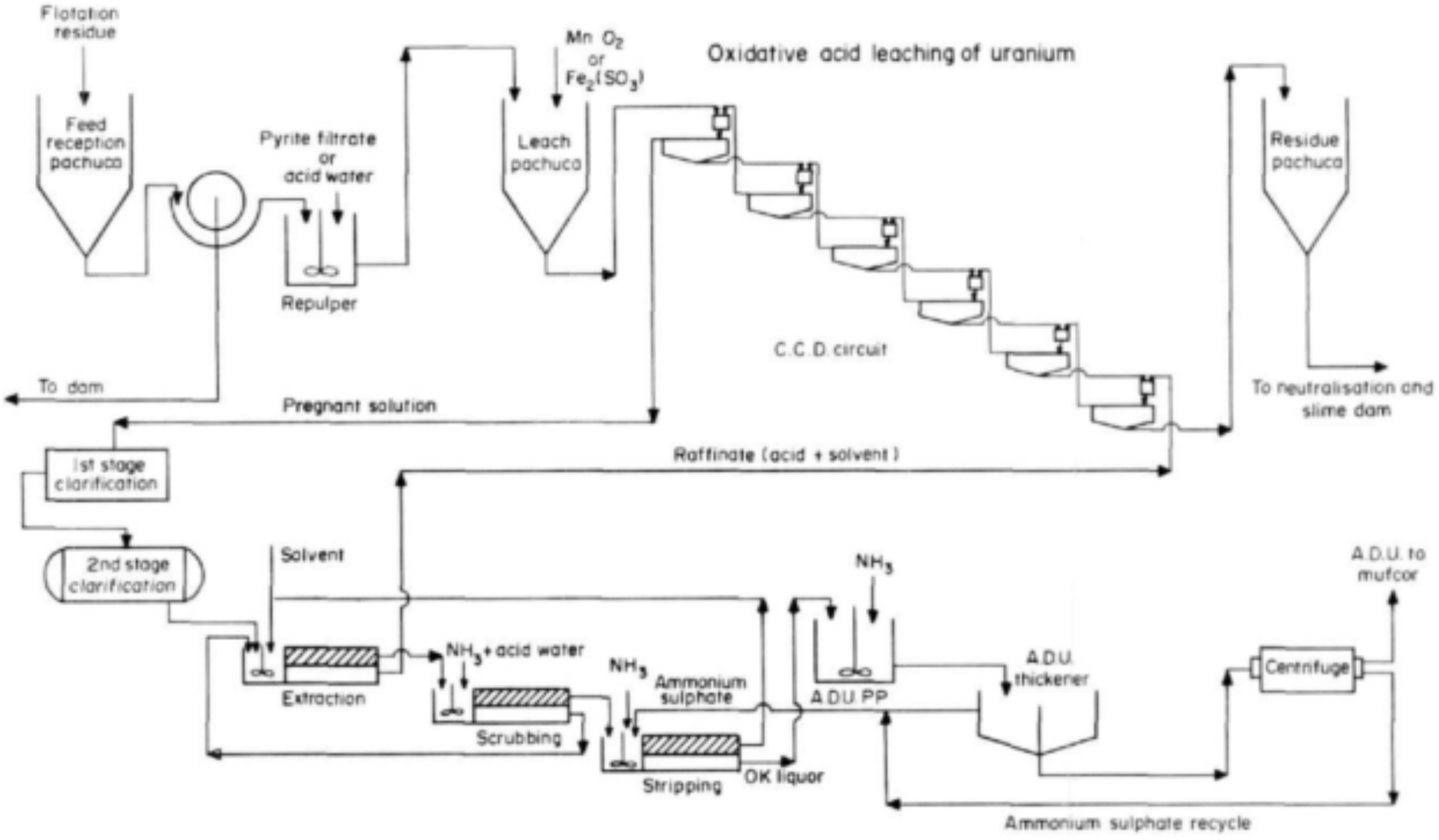


Figure 27
Counter-current decantation leaching and solvent extraction (PURLEX)

- Fixed bed exchange and extraction with nitric acid (1 plant)
- Fluidized-bed continuous ion exchange and small-scale solvent extraction CIX-BUFFLEX (1 plant); and
- Solvent extraction only PURLEX (6 plants).

(i) Fixed-bed ion exchange

- The clarified uranium pregnant solution is contacted with fixed-bed ion exchange resin, from which the adsorbed uranium is extracted by 6-stage elution with nitric acid.
- The eluant is treated with lime to remove $CaSO_4/FeSO_4$ by precipitation at pH 3,8.
- The end-product, ammonium-diuranate, is precipitated by neutralization with ammonia to pH 7.

(ii) Fluidized-bed continuous ion exchange (CIX -BUFFLEX)

Acid oxidative leaching of uranium follows the same principle as already described.

- Coarse solids are removed by belt filtration or by counter-current decantation (CCD).
- The filtered pregnant solution is fed to the fluidized-bed ion exchange resin columns, where the uranium is selectively adsorbed.
- The loaded resin, after being screened and filtered from the barren solution, is eluted with 10% H_2SO_4 at ambient temperature.
- The concentrated eluate is treated at a solvent extraction plant similar to the PURLEX process, but on a smaller scale because of the smaller volume to be treated for the extraction of the uranium oxide.
- The eluted resin is returned to the absorption columns.
- The raffinate from the BUFFLEX extraction section is returned to the CIX section where it is again made up to strength with fresh H_2SO_4 and reused as an eluant.
- Barren solution from the continuous ion exchange section can be reused for filter washing, repulping of eluted slime for pyrite flotation, as a make-up to the counter-current decantation leaching pachucas or as a scrubber/cooling liquid in the H_2SO_4 plant. A bleed-off of about 20% and more is maintained to prevent contamination of the ion exchange resin. Surplus barren solution is disposed of with the tailings.

(iii) Solvent extraction (PURLEX)

The processing stages are as follows:

- The pregnant solution from the oxidative acid leaching stage has to be clarified as described earlier.

- Uranium is extracted from the pregnant solutions at pH 1,3. A 5% solution of Alamine 336 in kerosene with 4% to 5% iso-decanol added is used as the eluant.

  An extraction bank of 4 or more stages as shown in Fig 28 with a long residence time to reduce solvent losses originating from the incomplete separation from the acid phase through formation of an emulsion is used (Robinson, 1972). Solvent losses ranging from 250 mg/*l* to more than 600 mg/*l* are common. However, the presence of solvents in the raffinate prevents its reuse at two major water-using applications:

- Repulping of slimes prior to the flotation of pyrite, because the flotation process is sensitive towards organic chemical impurities other than the specific flotation agent.

- Reuse at the carbon-in-pulp or carbon-in-leach processes, since the activated carbon soon loses its adsorption capacity for gold.

A major cause of solvent loss is a high content of suspended solids in the uranium pregnant solution. Therefore, the almost complete removal of suspended solids prior to solvent extraction is essential. This can be achieved by various methods such as hopper clarification, upflow sand bed filtration, leaf filters, Funda filters or a combination thereof.

- For scrubbing the solvent, an aqueous 0,1 M NH_4OH solution is used with the purpose of precipitating siliceous impurities. In order to achieve as complete as possible removal of iron (the chief co-extracted element) and cobalt, H_2SO_4 is added at the third scrub stage to maintain a pH of 1,5. In the final scrubbing stages, desalinated or softened water is used.

- During stripping, the uranium is transferred from the scrubbed solvent into the water phase by counter-current mixing with NH_4OH and $(NH_4)_2SO_4$, thereby raising the pH gradually to 5,1.

- Uranium is finally recovered from the OK liquor by the addition of NH_3 and subsequent precipitation as ammonium diuranate (ADU) or yellow cake. This is then transported to Nufcor for dewatering by filtration, drying and calcining to produce uranium oxide (U_3O_8).

The solvent, after stripping, is regenerated by washing with Na_2CO_3 to remove thiocyanate which would otherwise rapidly poison the solvent; the regenerated solvent is returned to the fresh solvent tank. The spent Na_2CO_3 aqueous regenerate is disposed of with the tailings. An inherent weakness of the PURLEX solvent extraction process is that it produces large volumes of raffinate, which is acidic and contaminated with solvent from the incomplete phase separation at the extraction stages. Since entrained organic matter adversely affects the flotation of pyrite, raffinate cannot be reused as filter wash or in repulping feed for flotation plants. Raffinate is therefore often disposed of separately to the tailings. The practical implication is that where the PURLEX solvent extraction process is used, the disposal of the large volume of solvent-contaminated raffinate together with the surplus water from pyrite flotation reduces the relative density of the final tailings to a value of 1,2 to 1,3.

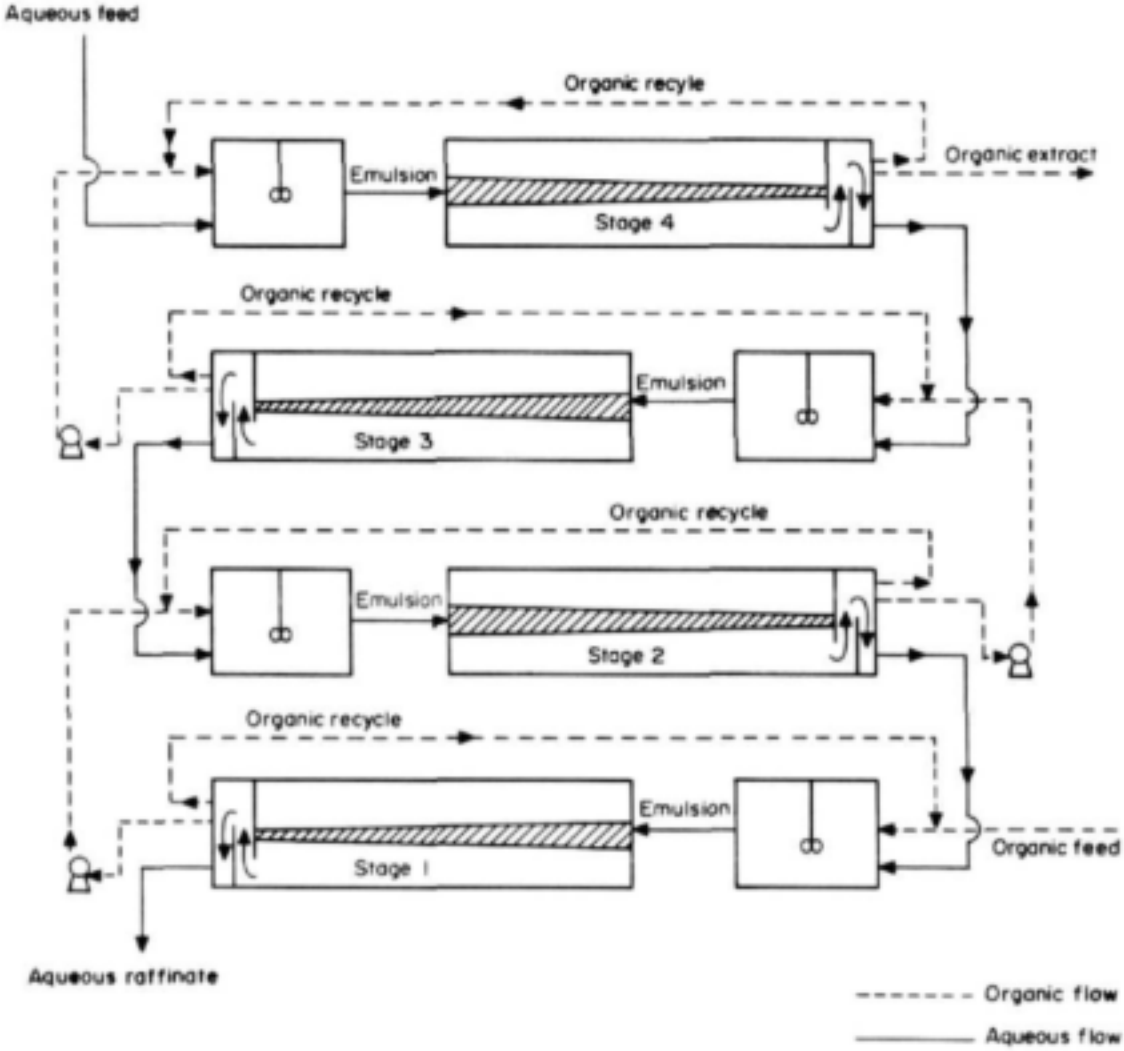


Figure 28
Diagrammatic representation of an extraction bank

Compared with relative density values of about 1,45 from conventional gold extraction, this difference is significant since the volume of water per ton of solids increases from about 1 m^3 to between 2 and 3 m^3. This means that the cost of disposal of tailings increases. Raffinate is a highly saline acid solution and its composition is shown in Table 6.

Present possibilities for the reuse of untreated raffinate are:

- Recirculation of part of the solvent-contaminated raffinate as make-up for the counter-current leaching of ore with H_2SO_4 (Fig 28).
- Use as scrubber/cooling liquor in the Peabody scrubber of the H_2SO_4 plant, provided that the chloride concentration is low (Fig 29).
- Use in the gold plant for acid cleaning of clogged filters.

Neutralization of the acid raffinate is not without difficulties. Due to the presence of high concentrations of Fe, Al and SiO_2, neutralization with lime results in the formation of a jelly-like hydrated precipitate which is difficult to dewater. The neutralized supernatant liquor has a TDS concentration of about 3 300 to 5 500 mg/*l*, which is about the same as that of neutralized return water from gold tailings deposits, but by far less than that of the acid raffinate with a TDS of between 20 000 and 60 000 mg/*l*. Such neutralized return water would be suitable for most processes in the metallurgical section. It is important to find a method:

- dewater the jelly-like hydrated precipitate economically, and,
- remove the solvent simultaneously in order to find a greater reuse potential for the neutralized raffinate.

At ERGO, raffinate is mixed with calcine overflow at a ratio of 1 to 4,7 and neutralized with lime with simultaneous intense aeration. Although the residual solvent is most probably not removed completely, dilution with calcine overflow reduces the solvent level sufficiently to reuse the resultant water for monitoring of aged slimes for the flotation recovery of pyrite.

The magnitude of solvent losses is shown in three examples (costs for 1985):

(i) *Uranium plant A*

ore treated	3,8 x 10^6 t/a (37,5%)
water to tailings	6,4 x 10^6 m^3/a (62,5%)
relative density 1,31 and solids to water ratio of 1:1,67 solvent loss with water to tailings	275 mg/*l*

Replacement:	Alamine	130 m^3/a at R4 500 /m^3
	Isodecanol	130 m^3/a at R2 500 /m^3
	Kerosene	1 520 m^3/a at R600 /m^3
	Total	1 780 m^3/a at R1,82 x 10^6/a

Table 6: Approximate chemical composition of Purlex raffinate

Determinand	Before neutralisation (raffinate)					After neutralisation (barren solution)	
	Plant No						
	1	2	3	4	5	3	5
pH	1,0 to 1,8	1,5	1,5	1,9	1,9	8,9	5,4
Conductivity (mS/m)	2 600		2 930			444	
Total dissolved salts	39 000	30 000	56 500	23 860	22 140	3 850	5 640
Suspended solids		760					
H_2SO_4		5 150	41 400				
SO_4	23 000	17 400		1 690	1 500	2 230	
Fe^{2+}	4 900	1 500	9 200	1 800	2 200		100
Fe^{3+}	1 250	600					
Mn^{2+}	25	2 200	130			0,03	
Al	700			420	3 200		
Cl		240	140			150	
SiO_2	760	620		880	120		nil
Ca	100	620	470	104		570	

Where no figure is entered, the analysis is not available. Raffinate after neutralisation = barren solution; sample No 3 was filtered before analysis to remove precipitated calcium sulphate. Concentrations in mg/l, except pH

Table 6 (continued): Approximate chemical composition of Purlex raffinate

Determinand	Before neutralisation (raffinate)					After neutralisation (barren solution)	
	Plant No						
	1	2	3	4	5	3	5
Mg	400	130	800	760	880	140	490
K	70	70	6	110	110		110
Na	230	310	230	470	420		410
Zn	280	90	157	35	32		3,5
Ni	80	15					
Co	18	5		0,4	0,1		nil
Cr	16						
Cu	130	6	47	7	7		3
Cd	2						
As	70						
Pb	6	5					
NO_3		100					

Where no figure is entered, the analysis is not available. Raffinate after neutralization = barren solution; sample No 3 was filtered before analysis to remove precipitated calcium sulphate. Concentrations in mg/l, except pH.

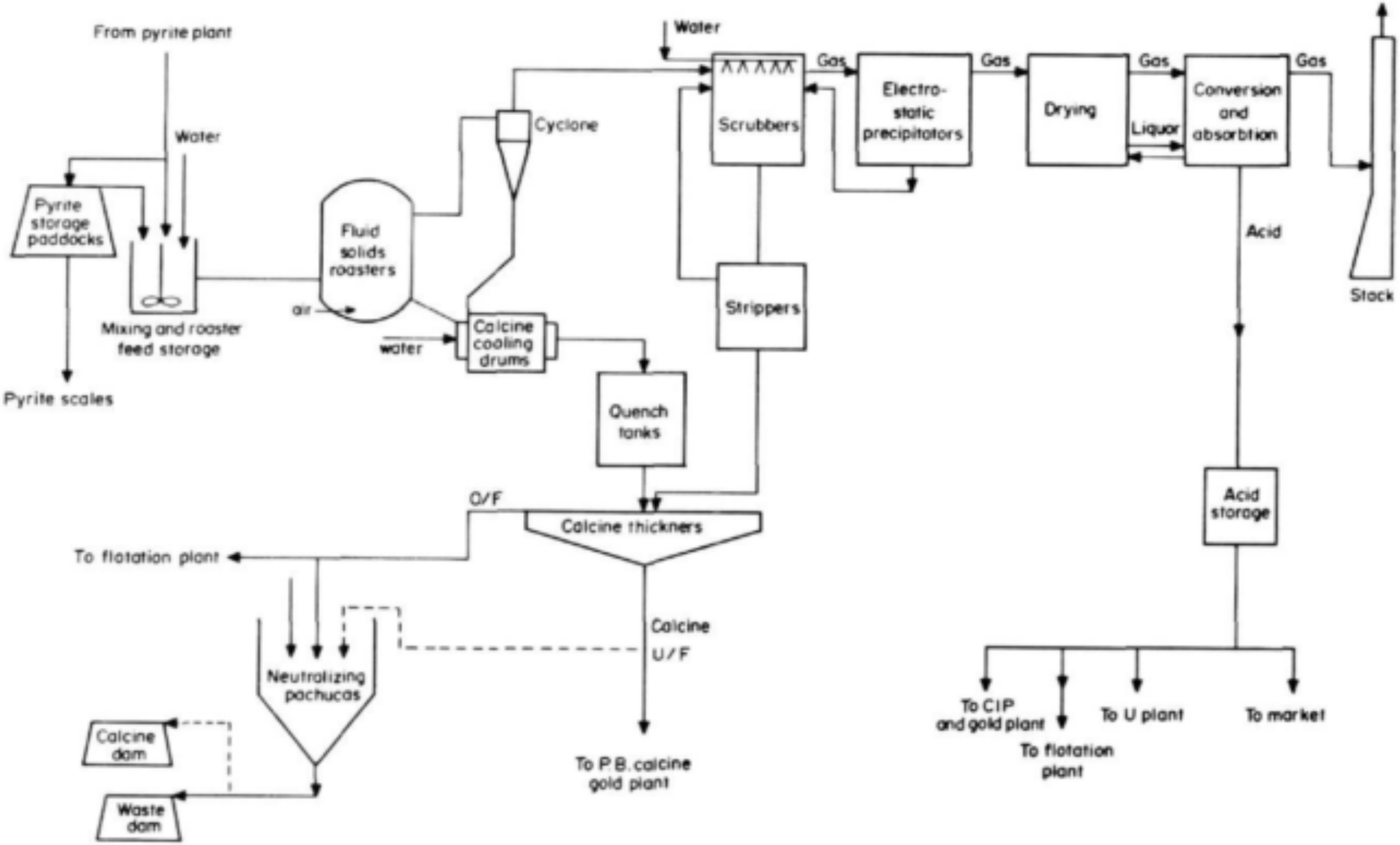


Figure 29
Simplified flowsheet of the acid plant at President Brand

(ii) Uranium plant B

ore treated	$3{,}24 \times 10^6$ t/a
solvent loss with raffinate	1 200 to 1 320 m^3/a
replacement costs for solvent	R$1{,}25 \times 10^6$/a

(iii) Uranium plant C

ore treated	$3{,}024 \times 10^6$ t/a
solvent loss with raffinate	900 m^3/a
replacement costs for solvent	R$0{,}9 \times 10^6$/a

The financial losses via the solvents are substantial. However, the reduced reuse value of raffinate and the higher costs for the disposal of the larger volumes as well as the implications for tighter pollution control, e.g. additional costs for the lining of tailings or evaporation dams to prevent seepage, require strong efforts to be made at the metallurgical plants to find ways and means to reuse spent raffinate more efficiently. Methods to reduce the solvent carry-over into the raffinate (500 mg/*l* at plant A) may include the following:

- Clarification of acid-pregnant solution to remove suspended solids, either by filtration or by clarification in a suitable upflow clarifier with a floc blanket.
- Improving the design and operation of the extraction bank by probably increasing the number of extraction stages in order to increase the time for phase separation, and/or incorporation of a parallel plate separator and a coalescer or a dissolved air flotation unit to improve the recovery rate of solvent.
- The use of biocide on the solvent to inhibit biodegradation thereby possibly improving phase separation.

Coalescers of various designs and/or dissolved air flotation units are installed at most uranium solvent extraction plants. Volumetric overloading may be one reason for solvent still passing through.

3.8 Pyrite flotation

Large quantities of H_2SO_4 are required for the extraction of uranium from gold plant residue. The sulphur for the acid plant is derived from pyrite (FeS_2) and other metal sulphides which are present in the reef. There are basically two processes for the flotation of gold and uranium containing pyrite:

- Flotation recovery of pyrite after cyanidation of gold and uranium extraction. This is the conventional process called *forward leach*.
- Flotation recovery of pyrite from the fresh coarse milled ore before gold extraction and recovery of the uranium from the float by leaching with hot sulphuric acid - called *reverse leach*.

Most flotation agents have a deleterious effect on the cyanidation of gold. Accordingly, flotation of pyrite is usually conducted after cyanidation. However, the effect of the flotation reagents can be destroyed by the hot oxidative acid leaching for the recovery of uranium. Therefore, if "reverse" leach is adopted, it is possible for the pyrite to be floated before the uranium and the gold are recovered. Cyanide is inherently a depressant for pyrite recovery by flotation; even traces of cyanide are quite sufficient to have a significantly adverse affect. Similarly, the lime used to ensure alkaline conditions during cyanidation can have a deleterious effect on pyrite flotation. Consequently, a soft water has to be used for repulping the dewatered slime from the gold plant. The pH value has to be adjusted into the acid range and the feed must be conditioned for a significant period before flotation, both to destroy the effect of the lime and to remove cyanide ions as free HCN. The presence of traces of cyanide is probably one reason why current gold plant residues exhibit a far poorer flotation response than *aged* slimes dam material unless acid is conditioned for about 20 h.

Flotation of pyrite is sensitive to the presence of certain organic chemical solvents. Solvents present in raffinate from the uranium recovery by solvent extraction adversely affect the flotation recovery of pyrite. This excludes the use of raffinate for repulping slimes as feed stock for flotation. The development of a flotation process to enable the reuse of solvent-contaminated raffinate for repulping should be supported.

3.9 Sulphuric acid production

This process involves roasting of pyrite to form SO_2 which is then cleaned, dried, heated and reacted over a catalyst to form SO_3 which is then reacted with water to form sulphuric acid. For roasting, pyrite is slurried with water to a pulp relative density of about 2,0 (80% solids, 20% water) and fed into the roaster. The pyrite is calcined (roasted) with a small amount of excess air to form iron oxide and SO_2. As the iron oxide particle sizes are small, there is almost a 100% carry-over of calcine from the roaster to the cyclones. All the calcine recovered from the cyclones is slurried and pumped to thickeners, and from there to the gold recovery plant.

The gases leaving the cyclones are scrubbed and cooled in Peabody scrubbers which remove most of the remaining dust. The water used to cool the gas from about 700°C to about 40°C should be low in chloride which, at these temperatures, rapidly corrodes stainless steel. Where the chloride concentration of the water is low, e.g. in water of dolomitic origin, waste barren solution from the uranium plant or mine service water can be used, resulting in considerable savings of fresh water. About 14 000 m^3/d of scrubber/cooling water is used as make-up for a production of 1 000 t/d of sulphuric acid. This water is recirculated to the calcine thickener from where part of it is returned as make-up water to the pyrite flotation plant, or it is disposed of with the flotation tailings to the tailings dam.

The simplified flow scheme of a sulphuric acid plant as in use at the President Brand Joint Metallurgical Scheme (JMS) is shown in Fig 29.

3.10 Water use and disposal at a combined gold, uranium, pyrite and sulphuric acid recovery plant (Way and Taylor, 1986)

The water in a metallurgical plant recovering gold, uranium, pyrite and sulphuric acid is recirculated many times. In such a plant, the nature of constituents, the required change in pH for the various recovery processes and the necessity of adding dilution water for washing processes in between, result in a far higher water volume to solids ratio than for straightforward gold recovery. As an example of the many processing steps, the abbreviated process schedule is given below for the metallurgical plant at the Buffelsfontein mine, which recovers uranium, gold, pyrite by *reverse leach*, and sulphuric acid. The sequence of the various processing steps is as follows:

1 Milling of crushed ore with addition of water.

2 Thickening of milled ore and return of thickener overflow to the mill circuit.

3 Adjusting thickener underflow with industrial water, (a mixture of process water and mine water) to a relative density of 1,7 (65% solids) and water to solids ratio of 0,54 :1.

4 Leaching of pulp with H_2SO_4 (40 kg/t), $Fe_2(SO_4)_3$ and steam.

5 Liquid/solids separation by rotary drum filters and horizontal belt filters, necessitating the addition of wash water for cake discharge and cloth cleaning duties and of water for glands and vacuum seals.

6 Extraction of uranium with solvent, followed by scrubbing and stripping using borehole water from the area between the slimes dam and Vaal River and calcine thickener overflow.

7 Neutralization of the solvent-contaminated (400 to 600 *mg/l*) raffinate with lime and separate disposal of this barren solution direct to the tailings.

8 Neutralization of the uranium leach residue with lime to a pH of 12, addition of water to a relative density of 1,46 (50% solids) and water to solids ratio of 1:1 (ex step No 10).

9 Extraction of gold with $Ca(CN)_2$, equivalent to 0,35 kg/t of NaCN.

10 Return of barren liquid from gold extraction, for repulping uranium leach residue, step No 8.

11 Repulping the alkaline gold residue with calcine water to a relative density of 1,35 (41% solids) and water to solids ratio of 1,44 : 1, and acidification to pH 3,5.

12 Flotation of pyrite, and dewatering.

13 Recirculating part of that water to the flotation plant.

14 Roasting of pyrite, production of sulphuric acid. Use of industrial water as scrubber/cooling water.

15 Quenching of hot calcine with slimes dam return water.

16 Recirculation of Venturi and Peabody scrubber/cooling water to the calcine thickener.

17 Thickening of quenched calcine and return of thickener overflow to repulping of alkaline gold residue prior to flotation of pyrite, step No 11.

18 Treatment of calcine thickener solids by agitating with H_2SO_4, over 100 kg/t, for

freeing occluded gold.

19 Return of acid-treated calcine to the oxidative acid leach, step No 3.

20 Neutralization of non-floatable gold residue to pH above 10, resulting in a *thin* pulp of relative density 1,3 (37% solids) and water to solids ratio of 1,7 : 1 because of the admixture of launder water, gland water, cleaner tailings.

21 Disposal of step No 20 tailings to the slimes dam, and decantation of surplus water from steps No 7 and 20 for transfer to an unlined evaporation dam east of the Koekemoerspruit.

22 Treatment of surplus calcine thickener overflow and borehole water to adjust the pH to a value of about 10 for precipitation and removal of iron and manganese in the Enviroclear settlers before discharge into the Koekemoerspruit.

The separately discharged barren solution from the uranium plant (neutralized raffinate) constitutes about 33%, and the thin flotation plant residue about 67% of the total water disposed of to the tailings. At all recovery plants except one, solvent is being used for the recovery of uranium, either by the CIX/BUFFLEX or the PURLEX process. Most of the uranium recovery plants are linked with pyrite flotation and sulphuric acid production, resulting in a disposal of far more water per unit of solids than at gold recovery plants. A list of current uranium extraction, pyrite flotation and sulphuric acid production plants is given as Table 7 and pinpoints mines where tailings can be expected to contain a surplus of water.

3.11 Tailings disposal above the surface

3.11.1 General classification

Residue deposits can be classified into four categories:

(i) Rock dumps consisting of barren overburden material or low-grade material which is not economical to process for the recovery of gold. In order to stabilize rock dumps, unknown volumes of low grade water are sprayed onto the rock dumps during construction. Rock dumps may be used e.g. as source for road building material. The pollution potential due to leaching and wind erosion is considered to be low.

(ii) Sand dumps which were deposited mechanically in a moist state, reaching heights in some instances of over 100 m above ground. Because of the permeability of the loosely packed sand, oxidation of pyrite has occurred to a depth of 10 m and more resulting in the leaching of sulphur in the form of acidic ferrous or ferric sulphate. Estimations based on two sand dumps which were deposited until the late 1930s and the late 1950s indicate that between 22% and 24% of the total sulphur originally present in the dumps has been leached over a period of 40 and 25 years respectively. Leaching and pollution from *inactive* sand dumps in the form of contaminated seepage still continues.

Air pollution by wind-blown dust has been largely eliminated by the planting of grass over the past 20 years.

Table 7: Metallurgical plants recovering gold, uranium, pyrite and sulphuric acid (1989)

Mine	Forward leach	Reverse leach	Uranium plant	Pyrite flotation	Acid plant
ERGO	from slimes	-	Purlex	yes	yes
Western Areas	yes	-	Purlex	yes	yes
Randfontein Estates	yes	-	closed 1988	yes	yes
West Driefontein	-	-	closed 1988	-	-
Stilfontein/Chem Wes	from slimes		closed 1988	yes	yes
Hartbeesfontein	-	yes	fixed bed IX/nitric acid	yes	yes
Buffelsfontein	-	yes	Purlex	yes	yes
Vaal Reefs East	yes	-	Purlex	yes	yes
Vaal Reefs West	yes	-	Purlex	yes	yes
Vaal Reefs South	-	yes	CIX/Bufflex	-	-
Loraine	yes	-	-	yes	-

Table 7 (continued): Metallurgical plants recovering gold, uranium, pyrite and sulphuric acid (1989)

Mine	Forward leach	Reverse leach	Uranium plant	Pyrite flotation	Acid plant
Free State Geduld	from slimes	–		yes	–
President Steyn	from slimes	–		yes	–
President Brand/JMS	from slimes	–	Purlex	yes	yes
St Helena	yes	–	–	yes	–
Harmony	yes	–	closed 1988	yes	–
Virginia	yes	–	fixed bed, closed	yes	yes
Merriespruit	–	–	closed	–	–
Beisa	–	–	Purlex, closed	–	–
Blyvooruitzicht	–	–	CIX/Bufflex, closed	–	–
Western Deep Levels	–	–	Purlex, converted into gold plant	–	–

** Randfontein Estates, Cooke Section changed over from CIX/Bufflex to Purlex*

(iii)	Slimes dams* from the extraction of gold.	Deposited hydraulically as slimes.
(iv)	Slimes dams* from the combined extraction of gold, uranium and pyrite.	" " "

* The word *dam* is used in the mining industry for hydraulically placed deposits.

The following discussion deals with the pollution potential of slimes dams from the extraction of gold only and from the combined recovery of gold, uranium and pyrite.

With the introduction of cyanidation after 1918 to recover gold, mines gradually changed to the use of ball mills and the sliming process. Ore is milled much finer, with about 75% to a size of below 75 μm. Consequently, the mechanical deposition of tailings as sand dumps was gradually phased out, with the last sand dumps deposited probably in the early 1960s. Since then practically all tailings from gold - and later uranium - extraction plants have been deposited hydraulically as slimes. Tailings dams may take a variety of forms but are all characterized by the fact that the fine-grained material is deposited hydraulically, usually into ring dyke impoundments constructed on fairly level ground.

3.11.2 Slimes dams from the recovery of gold

For the deposition of slimes from the recovery of gold, the water content of the tailings is normally equal to the mass of dry solids, with a relative density of about 1,45. Although tailings slurries may be transported as reasonably homogenous materials, this state ceases to exist once the slurry has been deposited in the tailings impoundment. In conventional hydraulic-fill construction, tailings slurry is deposited along the length of the impounding wall into so-called day-paddocks. Because the coarser particles settle faster than the finer particles, a gradation of particle size results, with coarser material settling adjacent to the point of deposition whereas the finer particles remain in suspension. This difference in settling rate is utilized in the construction of tailings dams. The tailings slurry is pumped during the day into one of the day-paddocks along the perimeter, with a freeboard of about 0,7 m to 1 m in height. Construction of freeboards is usually mechanized by tractor and ground-shaping devices. In the late afternoon, the thin slurry from the day paddock is, after settling of the coarser particles, decanted via breeches in the inner wall into the large area of the night-paddock, where the fines settle out. The decanted water is collected in the lower-lying area around the penstocks, from where it is drained as rapidly as possible and returned to the recovery plant.

The plan of a ring dyke tailings dam (Fig 30; McPhail and Wagner, 1987) and the water balance for such a residue deposit are shown in Fig 31. The data as given are for gold slimes residue on the Witwatersrand, namely slurry delivered at a relative density of 1,45 or 50% solids, water content of tailings deposit 20%, rainfall 750 mm and an annual evaporation of 1 600 mm or higher.

For gold and uranium tailings, the maximum rise is the rate of deposition that allows enough cycle time on the dam's day-paddocks to facilitate drying out of the tailings, usually 7 to 14 days. Desiccation of the slime is essential for compaction and gain in strength and to crack the slime and thereby to reduce the ratio of horizontal to vertical permeability. The cycle time should also be such as to limit recharging of the water table (phreatic surface) within the slimes dam. Excessive recharging results in a rise of the phreatic surface

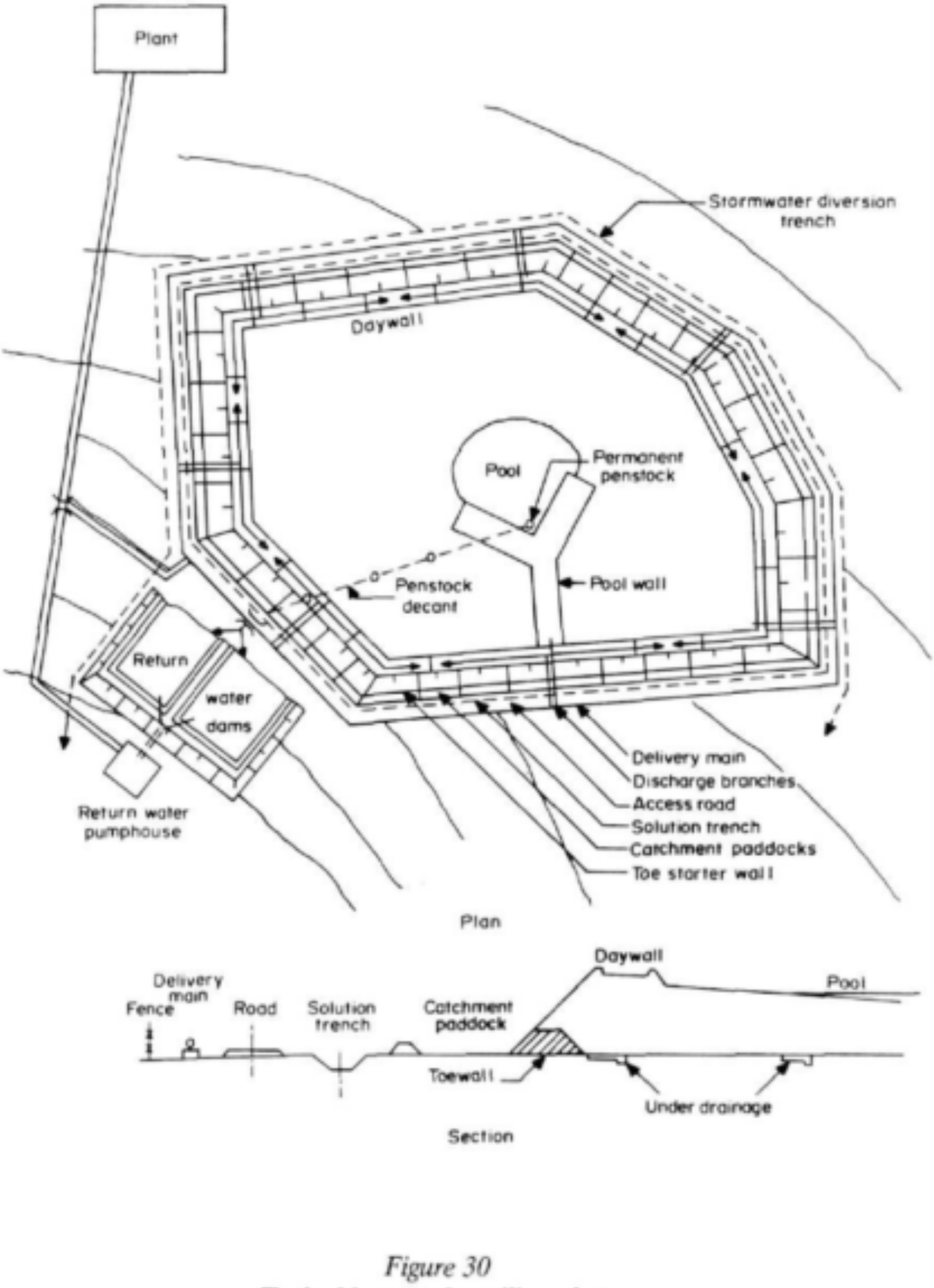


Figure 30
Typical layout of a tailings dam

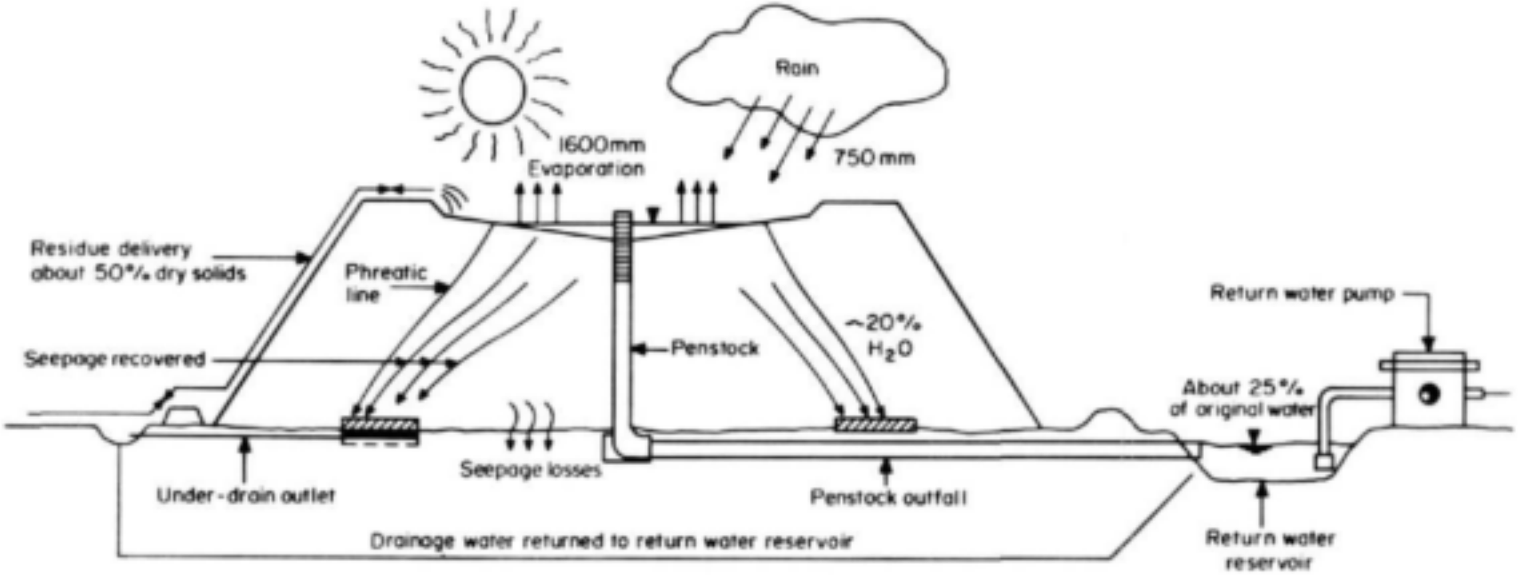


Figure 31
Water balance for tailings deposit

and may cause unstable conditions at the toe of the dam if the drainage is exceeded, (Fig 32).

The network of sub-surface open cracks formed by drying shrinkage may form potential channels for horizontal piping erosion, particularly when depositing a slurry of low relative density, or when water accumulates on the surface of a dam. Experience has shown that the permeability in the horizontal direction can exceed permeability downwards by a factor of about 10 or higher but is usually less. The preferential filling of desiccation cracks with coarser material on the next deposition cycle tends to reduce the effect of layering so that vertical permeability is usually only 1,5 to 3 times lower than horizontal permeability (McPhail and Wagner, 1987) The failure of the (platinum) tailings dam at Bafokeng on 11th November 1979 appears to have been initiated by horizontal piping erosion (Jennings, 1979).

The maximum practical rate of rise, based on effective desiccation, stable surface conditions and access requirements for tailings dams in the Transvaal and OFS gold-fields has been adopted as 2,5 m/a. This is based on experience with gold tailings with a relative density of 1,45 and a cycle time of two weeks. Failures and severe operating difficulties have occurred with uranium tailings at rates of rise well below 2,5 m/a. These have been attributed to high water content, rather than to other factors (Wates, 1983).

Since the risk of foundation instability on dolomite is high, a reduced maximum allowable rate of rise has been proposed. Capital costs for pumps, pipes, valves as well as operating costs increase with lower relative density of the tailings slurry.

Relative pulp densities and water to solids ratios for gold/uranium tailings are shown in Table 8.

The slimes deriving from gold reduction works are alkaline in reaction. Witwatersrand ores and those from the Klerksdorp and OFS gold-fields contain pyrite (FeS_2) in varying amounts, generally between 0,6% and 1,6% . Oxidation of the pyrite, accelerated by bacteria, causes the gradual conversion of the water insoluble pyrite into water soluble ferrous sulphate ($FeSO_4$) and ferric sulphate ($Fe_2(SO_4)_3$), with a concomitant increase of acidity in the surface layer. Acidification occurs on the top and sides of a dam to a depth of about 2 m, with the zone of maximum acidity about 0,3 m below the surface. The pH value near the surface varies, depending on how much pyrite is present and the extent to which oxidation has occurred. pH values as low as 1,5 have been encountered. There is evidence that the acidity moves downwards, particularly during rain, but that this movement is halted or reversed by evaporation from the surface (Marsden, 1986).

Since there is no access of air to the bulk volume of a tailings dam because the pore volumes are filled with water, oxidation of pyrite occurs only during the limited period of exposure to air between the applications of new layers of slimes. The water content of slimes dams which were reprocessed for the recovery of gold, sulphuric acid and uranium was found to be between 16% and 30% with an average value close to 20% (ERGO, 1986). A figure of 20% appears to be acceptable for gold slimes deposits on the Witwatersrand. Gold tailings dams are generally built with a solids to water ratio of about 1:1. Since most of the seepage flows horizontally and along the phreatic line where it is collected at the toe channel as return water it can be assumed that the downward movement of seepage water with the potential of polluting ground water will be low. A case in point is the ERPM tailing complex, consisting of four dams with a combined surface area of 4 km^2. The relevant

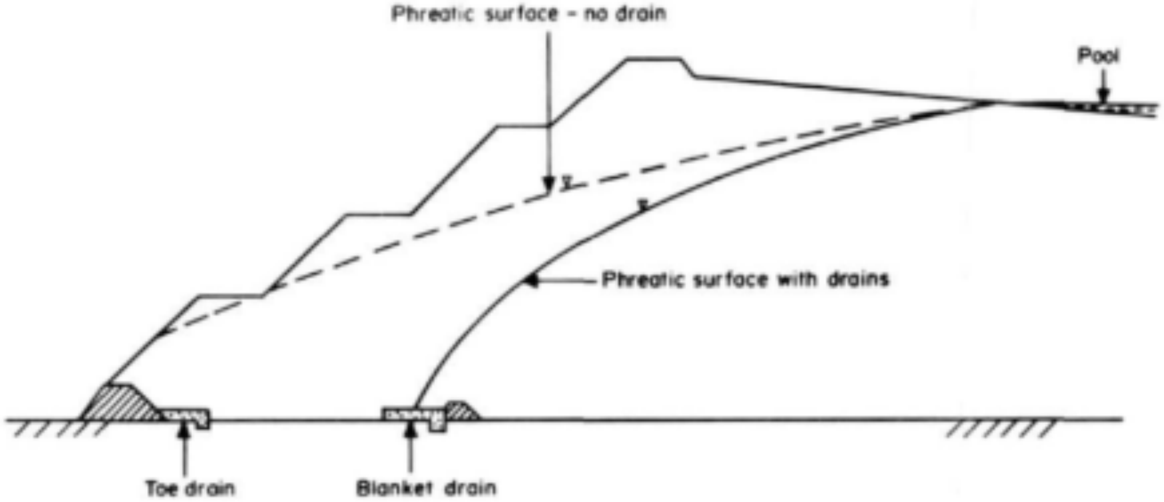


Figure 32
The influence of underwell drains
on the position of the phreatic surface

Table 8: Relative pulp densities and water to solids ratio

Relative density	% Solids	Ratio
1,18	24,0	3,17:1
1,20	26,5	2,77:1
1,22	29,0	2,45:1
1,24	31,0	2,23:1
1,26	33,0	2,03:1
1,28	35,0	1,86:1
1,30	37,0	1,70:1
1,32	38,5	1,60:1
1,34	40,0	1,50:1
1,36	42,0	1,38:1
1,38	44,0	1,27:1
1,40	45,0	1,22:1
1,42	47,0	1,13:1
1,44	48,5	1,06:1
1,46	50,0	1,00:1

Table 8 (continued): Relative pulp densities and water to solids ratio

Relative density	% Solids	Ratio
1,48	51,5	0,94:1
1,50	53,0	0,89:1
1,52	54,0	0,85:1
1,54	55,5	0,80:1
1,56	57,0	0,75:1
1,58	58,0	0,72:1
1,60	59,5	0,68:1
1,65	62,5	0,60:1
1,70	65,0	0,54:1
1,75	68,0	0,47:1
1,80	71,0	0,41:1
1,85	73,0	0,37:1
1,90	75,0	0,33:1
1,95	77,0	0,30:1
2,00	79,0	0,27:1

data for 1986 are given below (ERPM,1986).

On to slimes dams, annual rate of rise less than 1 m/a:

(i)	Alkaline tailings from the gold reduction works	
	437 500 t/month with 56% solids	245 000 t/month
	relative density 1,54; solids:water	56:44
(ii)	alkaline sludge from HDS plant, mainly $CaSO_4$ plus	
	water 80 550 t/month with 18% solids	14 500 t/month

(i) and (ii) total solids = 259 500 t/month	3 114 000 t/year
total water = 258 500 m^3/month	3 102 000 m^3/year
total mass	6 216 000 t/year

Return water : Decantation water ex-penstocks	57 000 m^3/month
seepage water from toe channels, about	10 000 m^3/month
Total, about	67 000 m^3/month
or about	800 000 m^3/year

Return water :
pH: 9,0
TDS: 3 365 mg/*l*
SO_4: 1 850 mg/*l*
Fe: 37 mg/*l*

The annual *possible* evaporation losses from the 4 km^2 slimes dams area can be approximately as follows:

	Average rainfall	750 mm/a
	Average gross evaporation	1 600 mm/a
Therefore :	Net evaporation	850 mm/a

The *potential* loss by evaporation is calculated to be:

$$4\ 000\ 000\ m^2 \times 0{,}85\ m = 3\ 400\ 000\ m^3/a$$

The *actual* evaporation loss is not as high:

water onto slimes dams about	3 100 000 m^3/a
solids onto slimes dams about	<u>3 100 000 t/a</u>
Total	6 200 000 t/a
Water retained within slimes deposit at 20% water content	775 000 m^3/a
water returned to plant	<u>800 000</u> m^3/a
water not available for evaporation	1 575 000 m^3/a
Therefore water available for evaporation and for downward seepage	1 525 000 m^3/a

This is about 45% of the potential loss by evaporation.

Pollution of ground water through seepage from these active dams with the collected water returned to the gold plant must be low. The above ERPM tailings dams are constructed on soil with low permeability (black clay).

Many of the newer gold/uranium tailings dams are equipped with underdrainage and in some cases these drains discharge almost continuously. This would seem to indicate that significant migration of water does occur downward in spite of the lower vertical permeability. Thus the above model of a water balance as for ERPM may not always apply.

Most slimes dams on the Witwatersrand are *inactive* for periods ranging from 20 to about 40 years. Most of them have been protected by the construction of toe dams where seepage accumulates. The sides and tops of many old slimes dams are protected by a cover of grass and trees. Vegetation, however, cannot be established on acid soil and is, therefore, an indication that the acidic oxidation products of pyrite have already been leached out (Marsden, 1986).

Vandalism and the use of old dams as motorcycle racetracks and for horse riding have resulted in extended erosion of some of the dams. Illegal dumping of wastes has, at places, become a serious source of pollution. In general, however, the results of quite a number of surveys of streams draining areas covered by 'inactive' slimes dams have indicated an only negligible contribution to the flow and pollution load of the Klip River system.

Examination of core samples from slimes deposits have shown that the pyrite (sulphide) oxidation is generally confined to surface layers up to 2 m thick. Except for the surface to 0,1 m depth where virtually complete oxidation is obtained in about one month, the rate of oxidation is slow and diminishes rapidly with depth. Completion of oxidation to the 1 m horizon takes 2 to 3 years. When freshly deposited, drain water from these tailings was highly polluted. Today, however, the mass load of dissolved salts from old residues is minor. Borehole samples were taken by ERGO and Rand Mines at a number of "inactive" slimes deposits to assess the values for gold, uranium oxide and sulphur (sulphide and sulphate sulphur). The results given in Table 9 are from ERGO for the Government Gold Mining Area No 3 slimes dam 6/L/3 which was built over a 18 year period between October 1950 and September 1968. Lower than average values for sulphur, indicating a possible loss by leaching, were found only in the surface samples of boreholes which are located near the outer rim of the day-paddock area made from coarse grained material. Variations in sulphur (sulphide) concentration at lower depths throughout the deposit can probably be ascribed to the varying content of pyrite in the slimes during deposition rather than to other factors. This tailings deposit (6/L/3) as well as the other deposits in the Brakpan area were built on black clay with low permeability. Pollution of ground water by seepage is therefore unlikely.

3.11.3 Slimes dams from the combined recovery of gold, uranium and pyrite

Most of these tailings deposits are on dolomitic strata. Thus seepages are likely to leak into the strata and pollute the ground water. On tailings from uranium plants, pyrite has generally been removed to a large extent for the production of sulphuric acid. Sulphur, after leaching out the uranium with sulphuric acid and neutralization with lime, is now present as water soluble $CaSO_4$, iron sulphate and $MnSO_4$. This implies that throughout the slimes dam gypsum and manganese sulphate can be leached by horizontal seepage along the phreatic line and by vertical seepage into the subsoil.

Table 9: Sulphur content (%) of core samples at different depths of an inactive slimes dam (6/L/3), near Brakpan

Sample No	Depth in m						Weighted average	Remarks
	0 to 4,3	4,3 to 8,6	8,6 to 12,9	12,9 to 17,2	17,2 to 21,5	21,5 to 22,1		
1	0,49	0,74	0,51	0,82	0,90	0,36	0,66	Black & brown clay
2	0,57	0,74	0,77	0,82	0,69	0,32	0,72	Black clay
3	0,44	0,64	0,64	0,69	0,77	0,26	0,63	Black clay
4	0,32	0,82	0,53	0,77	0,89	0,73*	0,68	Hard object
5	0,71	0,90	0,67	0,79	0,87	0,36	0,78	Black & brown clay
6	0,73	0,89	0,93	1,07	0,89	0,62	0,90	Black clay
7	0,72	0,72	0,76	0,72	0,73	0,56	0,73	Black clay
8	0,81	0,67	0,63	0,81	0,68	0,51	0,73	Black clay

Table 9 (continued): Sulphur content (%) of core samples at different depths of an inactive slimes dam (6/L/3), near Brakpan

Sample No	Depth in m						Weighted average	Remarks
	0 to 4,3	4,3 to 8,6	8,6 to 12,9	12,9 to 17,2	17,2 to 21,5	21,5 to 22,1		
9	0,67	0,71	0,96	0,48	0,76	0,63	0,71	Black clay
10	0,93	0,98	0,77	0,78	1,16	1,02*	0,93	Black clay
11	0,77	0,86	0,92	0,48	0,63	0,70	0,74	Black clay
12	0,82	0,64	1,07	0,78	0,77	0,73	0,82	Black clay
13	0,73	0,81	0,54	0,93	0,67	0,55*	0,72	Black clay & wood
14	0,99	0,90	0,55	0,59	0,88	0,67	0,78	Black clay
15	0,72	0,70	0,83	1,84	0,84	0,44	1,00	Black clay
16	0,55	0,61	0,85	0,86	0,71	0,90	0,72	Black clay

** 21,5 to 25,8 m   tonnage = 147 000 t   average moisture content = 15,8%   volume = 9,3 mil* m^3
present S content = 0,77%

There is surplus water available at most metallurgical plants where, apart from gold, uranium, pyrite and sulphuric acid are being recovered, mainly because of the limited reuse potential of solvent-contaminated raffinate and the surplus volume of flotation plant water. Examples of the relative densities and water to solids ratios for combined gold/uranium tailings in approximate values are:

Mine	Relative density	Water to solids ratio
Buffelsfontein	1,23	2,3:1
Vaal Reefs East	1,08	4,5:1
Vaal Reefs West	1,18	3,2:1

The solids to water ratio in the wet slime is now about 1:2 to 1:4,5, as compared with a gold tailings ratio of 1:1. With such a low solids and high water content, stability problems can arise. The pollution potential of most uranium tailings dams is also higher than that of gold tailings dams. It is, therefore, general practice to decant water as much as possible from the tailings deposit for evaporation in a separate evaporation dam.

Experience at the uranium deposits of the Buffelsfontein mine, built on dolomite, has shown that manganese, sulphate and chloride do seep through and pollute the ground water. It has also been found that neutralization with lime to a pH of 7,0 to 7,5 of the tailings does not prevent migration of dissolved manganese into the dolomitic. The extent of seepage from the tailings dam at Buffelsfontein into the dolomite can be judged by the dissolved Mn concentration in the pumped out borehole water as shown in Table 10 (Department of Water Affairs).

The result was continuous pollution of the Vaal River by manganese, affecting the operation and water quality of the nearby Western Transvaal Regional Water Works. Current practice is to construct adequate underdrains below the embankment in order to intercept and collect the downward flow of a tailings dam. For tailings dams, in particular the water-rich uranium/gold tailings, construction of an impervious lining below the entire tailings deposit area - about 250 ha for 200 000 t/month of solids - could be required in order to protect the ground-water quality below foundations with a high permeability e.g. on dolomite. An alternative solution would be to site new tailings deposits away from areas with a high potential for ground-water pollution. An environmental impact study for choosing a site for a new deposit is then indicated.

Decant dams for holding surplus decant water from water-rich uranium/gold tailings or from metallurgical plants should always be protected against seepage losses by an impervious lining. Lining systems as described in *The design, operation and closure of residue deposits* (Chamber of Mines of South Africa, 1979) include materials such as clay, bentonite, bitumen, synthetic rubber or plastic membranes.

Operating experience at various mines has shown that there are distinct possibilities for reducing the volume of surplus water disposed of for the construction of slimes dams for tailings from uranium plants employing the solvent extraction process. Proposals are:

(i) Improve the clarification of the uranium-pregnant solution since solvent losses with the raffinate are linked with the content of the suspended solids (Buffelsfontein).

Table 10: Volume and Mn concentration of borehole water pumped from dolomite between Buffelsfontein No 4 slimes dam and the Vaal River

Period		Volume	Mn
Months	Year	10^6 m^3	mg/l
Jul to Sep	1982	412,6	106 to 338
Oct to Dec		382,4	98 to 298
Jan to Mar	1983	355,6	75 to 237
Apr to Jun		600,9	80 to 198
Jul to Sep		661,8	64 to 152
Oct to Dec		591,7	59 to 122
Jan to Mar	1984	437,3	71 to 83
Apr to Jun		223,8	37 to 104
Jul to Sep		nil	–
Oct to Dec		420,9	45 to 51
Jan to Mar	1985	459,2	46 to 60
Apr to Jun		423,0	37 to 63

(ii) Treat the raffinate, during neutralization with lime, by intense aeration to:

- oxidize the $Fe(OH)_2$ to $Fe(OH)_3$ which is easier to dewater;
- drive off organic solvent which restricts the reuse of raffinate for bulk applications such as repulping of ore prior to pyrite flotation; and
- use the neutralized raffinate for applications where quality considerations are not critical e.g. moistening of rock dumps.

(iii) Erect a thickener and pump out only thickened slurry to build the dam, and reuse the surplus water elsewhere, e.g. as scrubber/cooling water in the H_2SO_4 plant, provided the chloride concentration is low.

(iv) Make use of the alkaline return water from uranium and uranium/gold tailings with e.g. 3 000 to 5 700 mg/*l* TDS, 2 000 to 4 000 mg/*l* SO_4 and a total hardness of around 2 500 mg/*l*.

(v) Establish a water management committee to determine the volumes and quality of internal effluents before disposal onto tailings deposits with a view to possible reuse of these flows.

3.11.4 Protection against wind-blown dust

A serious pollution problem in the past has been that of wind-blown dust emanating from tailings dumps, mainly from sand dumps. This dust problem has been successfully reduced following the establishment of the Chamber of Mines Vegetation Unit. The primary obstacle to growing grass on slime or sand proved not to be the absence of humus but the presence of acid originating from the oxidation of pyrite. On old *inactive* tailings where the surface zone had already been oxidized, the acid leached away and the pH risen again to a value above 4, grassing has proved to be successful. Rock-cladding the slopes with run-of-mine rock is often a promising alternate solution to prevent wind-blown dust originating from the sides of slimes deposits.

One factor essential for protecting the surfaces of slimes dams against erosion by rain and wind is the formation of a hard outer crust. By analysing samples of slimes and crust material from a large number of gold tailings deposits, it has been found that the most important factor in crust formation is the amount of pyrite present. Upon weathering, the pyrite decomposition products (ferrous/ferric sulphate) react with the lime present in the slime to form a protective gypsum-like product. The conclusion drawn from the behaviour of surface crusts in relation to their chemical composition is that a minimum of 0,7% pyrite in the slimes material is essential for the formation of hard crusts (Donaldson, 1960).

The removal of pyrite during the reprocessing of old tailings for the production of sulphuric acid has, however, little effect on the formation of a stable crust since all the iron and sulphuric acid will be deposited with the uranium tailings, after neutralization with lime. Thus, after drying, a ferrous/ferric/calcium sulphate layer will be formed with cement-like properties. Even after prolonged weathering which over a period of rainy seasons has leached out most of the sulphur from the surface layer as water soluble sulphate, a hard crust remains, probably consisting of iron oxide/calcium sulphate/oxide and reaction products of aluminium and silicon oxides.

Another way to ensure minimum pollution by silt erosion is to reduce the slope by constructing an erosion catchment berm half-way up and around the tailings deposit. Any drainage from the upper section can be intercepted and collected by penstocks for transfer into the bottom drain.

4 RADIOACTIVITY

The Witwatersrand ore contains varying amounts of uranium and thorium and the radioactive isotopes from their decay chains. Indications in the late 1950s were that ^{226} Radium had a radiotoxicity level of about ten times that of the next most toxic nuclide in the uranium decay chain. Hence the concentration of ^{222}Ra came to be used as a common measure of radioactivity in mine water and effluent. However, later reviews on the radiotoxicity hazard at occupational exposure levels - which are about 200 times the public permissible exposure level - have shown that uranium, thorium and polonium nuclides are of higher toxicity. ^{222}Ra is therefore now considered to be the second most toxic nuclide.

Due to the only recently acquired knowledge of radiotoxicity hazards, limited measurements of nuclide radiotoxicity levels other than ^{226}Ra have been carried out. In general, analytical methods to determine radioactivity hazards are complicated, long-winded and costly. The following paragraphs refer to results based on ^{226}Ra determinations and, therefore, probably do not reflect the actual level of radioactivity hazard.

Underground mine waters which are in contact with uranium-bearing reefs can be contaminated to a high degree. Values as high as 8,2 Bq/*l* and 12 Bq/*l* for ^{226}Ra have been found in underground mine water at one West Rand mine. Keeping in mind that at one stage 26 mines were feeding 17 uranium recovery plants it can be assumed that the contamination of mine water is widespread.

Concentration levels of uranium in the Witwatersrand and Far West reefs range from below 50 g/t to about 200 g/t of U_3O_8. At Afrikander Lease near Klerksdorp, uranium grades reach 640 g/t of ore. At Beisa at the southern end of the OFS gold-fields, uranium grades reach 600g/t of ore. This contrasts with the situation in areas mined primarily for uranium such as Canada, USA and Australia, where ore grades of between 2% and 4% uranium are extracted. There is therefore a difference of about two orders of magnitude in the radium concentrations in Canadian, USA and Australian tailings compared with those in South Africa. The concentration of radium in tailings is small, about 50 μg of radium/t of tailings for a 0,02% ore. Analyses of tailings dams to assess the prevailing ^{226}Ra levels gave values of 0,6 to 3,1 Bq/g of ^{226}Ra for the Klerksdorp and OFS mining areas where the richer uranium ores associated with gold are to be found, and of 0,2 to 0,6 Bq/g on the East Rand (Chamber of Mines of South Africa, 1979). Consequently, treatment of mine and tailings effluents by $BaCl_2$ to precipitate dissolved radium has not been practised in South Africa, since the radium concentrations in waste streams have been considered too low to justify such treatment.

The principal form in which radium is present in the tailings is $RaSO_4$, but a strong likelihood exists that the radium ion is also adsorbed on to the surface of particulate material. Laboratory leaching tests using columns of 1 250 mm depth, packed with different types of tailings and on to which predetermined doses of ^{226}Ra had been adsorbed have shown that after 550 days of continuous percolation of water through the columns, radium had moved only 25 mm. Little of the original radium in the bulk of the tailings was leached. The results for all test columns are consistent amongst themselves and point to an extreme low mobility of radium in the tailings impoundments studied (De Jesus, 1985). *In situ* tests on tailings dams also indicate that radium moved 1 300 to 2 900 times more slowly than water, which could explain the rather low radium content (0,06 Bq/*l*) of seepage water measured at the base of some impoundments.

All uranium extraction locally is done by means of acid leaching. Oxidation of pyrite in the tailings results in the production of H_2SO_4 which may solubilize uranium and its daughter products such as radium. Again, laboratory investigations did not show an increased leaching rate of radium with increased acidity.

Contamination of surface waters by ^{226}Ra has not yet been a problem in the vicinity of sand dumps and of tailings dams from gold processing only. The fact should, however, be taken into account that these deposits from gold only recovery are too dry to produce significant amounts of seepage.

Contamination of ground water by seepage from gold tailings does not appear to be a problem, since the migration of radium in the tailings is so slow. In the case of the Buffelsfontein ground-water pollution, it has been found that the radium content of seepage into the dolomite was below 0,04 Bq/*l* (De Jesus, 1980).

Dams, lakes and streams, some of them in close proximity to old gold residue deposits to which the public has access, show radium concentrations close to those of natural waters (0,02 to 0,06 Bq/*l*). Radium concentrations in water bodies with free access to the public and situated in the vicinity of mine dumps are shown in Table 11.

Contamination of surface waters can occur by direct discharge of large volumes of radioactive process water or underground mine water into surface streams which by-pass a tailings dam with its adsorption capacity for ^{226}Ra.

Increases in ^{226}Ra levels in surface streams have been reported. In a few cases where ^{226}Ra concentrations in effluents of up to 1,4 Bq/*l* were measured, these fell to levels of about 0,03 Bq/*l* within a distance of 7 to 8 km from the point of release, most probably due to adsorption of radioactive material by the bottom sediment of the stream (Table 12).

Radioactive contamination of underground water may take place due to the common practice of uranium processing plants decanting surplus water from the tailings and pumping it to unlined evaporation dams. Most uranium processing tailings deposits and evaporation dams are on dolomite. Annual evaporation losses of 2 m/a to 4 m/a have been reported for some evaporation dams while the actual net evaporation rate should be around 1 m/a. Seepage losses into the dolomite strata may, therefore, take place. Whether or not seepage constitutes a threat to ground-water quality should be established. Lining of such evaporation dams could be a remedy.

While ^{226}Ra is readily adsorbed on to particulate material such as sand, clay or sewage sludge and is thus removed from clear water, this is not the case with uranium, which stays in solution. Uranium can, therefore, be transported in mine service water through the mine, but also in mining effluents over wide distances.

At one mine, pumped-out mine water had to be returned into the adjacent dolomite compartment in order to prevent ground subsidence and the formation of sinkholes. In order to increase the lifespan of the mine and to reduce the high cost of pumping, this mine obtained permission to dewater the compartment and to discharge the effluent via a public stream. The average uranium level of the dolomite water still entering the mine from this

Table 11: ^{226}Ra content of water bodies on the East Rand with free access to the public

Sampling site	Bq/l
Victoria lake	0,0074
Elsburg stream	0,0629
Elsburg dam	0,0185
Boksburg lake	0,0004
Geduld dam	0,0222
Cowles dam	0,0296
Nigel dam	0,1110

Table 12: ^{226}Ra content of some surface streams

Site			Bq/l
East Rand	ERGO-Brakpan	Waste stream, leaving property	1,400
	Rietspruit	After confluence with ERGO effluent	0,222
		At N103 crossing about 8 km downstream	0,026
West Rand	Western Areas	Leaving mine property from North Shaft	0,190
	Western Rietspruit	About 15 km downstream of North shaft	0,037
	Western Areas	Leaving property from South shaft	0,222
	Leeuspruit	About 8 km downstream of South Shaft	< 0,037

compartment after three years of dewatering is 67 $\mu g/l$, which is above the presently accepted standard for the public of 44 $\mu g/l$ uranium. Taking this situation into account, the mine is conducting a programme to measure radioactivity underground to ascertain whether any danger areas exit. Emphasis is placed on educating the workers not to drink mine service water.

Results of recent surveys have indicated levels of radium and uranium in many effluents leaving mining property which are in excess of the following accepted standards for the public, namely:

^{226}Ra: 0,19 Bq/*l*

Uranium: 44 $\mu g/l$*

The measured radium and uranium levels in effluents for some mines are given in Table 13 (Council for Nuclear Safety, 1988/89):

* *Uranium is measured colorimetrically and is expressed in μg/l. This method is more convenient and preferred because establishing the radiation emission from uranium is laborious and long-winded. Unfortunately only grab samples had been taken and the volume of the waste flows was not recorded. The data recorded in Table 13 therefore, indicate only the presence but not the extent of a radioactivity.*

The increase of concentration of ^{210}Po in beef liver from cattle raised on irrigated land downstream of a mine to a level double that of control samples has been confirmed (NUCOR, 1984).

In the OFS Goldfield, the water supply to a number of mines is augmented by surplus fissure water from the Beatrix mine over a distance of up to 40 km. Beatrix is situated next to the closed-down formerly maximum-producing Beisa mine which has now been re-opened as the Oryx gold mine, mining different ore bodies. Whether or not fissure water pumped from Beatrix to the St Helena, Unisel, Pres Brand and Pres Steyn mines can be used as water supply without treatment to remove radioactive constituents should be established.

In the United States there are around 60 000 community water supplies, of which some 500 drinking-water supplies (almost all ground-water supplies) exceed the 0,19 Bq/*l* concentration limit for ^{226}Ra, and up to 650 supplies exceed the uranium concentration of 0,74 Bq/*l*. Radon concentrations in drinking water in the USA range from a few Bq/*l* to more than 100 000 Bq/*l* in the North-east, with the average value being between 18,5 and 185 Bq/*l*. In areas where the Rn concentration is high, private wells probably deliver more Rn into the household than do community water supplies (Cross *et al.*, 1985). Yet, the EPA is still in the process of developing revised regulations for radioactivity in drinking water under the Safe Drinking Water Act, due to the difficulty in obtaining information from all related areas and disciplines.

Lime softening, ion exchange and reverse osmosis have been demonstrated as removing from 93% to 97% Ra from drinking water (De Jesus, 1985). For the removal of uranium, studies on anion exchange, lime softening and reverse osmosis are still incomplete. One local mine group has, however, developed a method to remove uranium from mine water at reasonable cost, and is presently conducting tests on pilot-plant scale. The question now arises as to how radioactive waste concentrated in the brine originating from ion exchange,

Table 13: Radium and uranium concentrations in the effluents from some mines (Metcalf, 1990)

Site		^{226}Ra	Uranium
		Bq/l	ug/l
East Rand	A1	0,67	618
	A2	0,66	151
	B	0,12	47
	C1	0,18	336
	C2	0,60	539
West Rand	D	0,21	262
	E	0,22	16
Far West Rand	F1	0,03	15
	F2	0,05	118
	G	0,31	305
	H	0,48	204
Orkney/Klerksdorp	I	0,03	402
	J	1,23	45

or in the lime softening sludge, or as reject stream from reverse osmosis or any other process should be disposed of.

Two methods to remove dissolved radon gas from drinking water appear possible:

- aeration

- adsorption onto granular activated carbon.

It is known that uranium is present not only in the quartz-pebble conglomerates of the Witwatersrand group, but also in other widespread mineral deposits as well as in certain coal deposits, in particular in the Springbok Flats. A locality map indicating the occurrence of uranium in South Africa is shown in Fig 33. A comprehensive radiological survey for the entire mining industry is being undertaken by the recently established Council for Nuclear Safety. At this stage, progress has been limited due to limited availability of personnel and analytical capability.

There are, at present, no limits set for radioactivity levels within the General Standards for the disposal of effluents into public streams nor in the SABS specification for drinking water.

High values for radioactivity were reported from one, now dismantled sulphuric acid plant at AECI, Modderfontein, apparently originating from the roasting of uraniferous pyrite. Burial of the contaminated structural material (steel, bricks, ceramic Raschig rings, rubber lining) has been accepted as the solution. There are at least 10 sulphuric acid plants, which are confronted with the problem of radioactive contamination from the roasting of uraniferous pyrite (Table 7).

The storage of 18,5 x 10^{14} Bq in the 65 km^2 of slimes dams and the 15 km^2 of sand dumps on the Witwatersrand might probably constitute a health hazard. Tailings containing uranium will constantly emit gaseous radon ^{222}Ra, which decomposes into radioactive solid short-lived decay products ^{218}Po, ^{214}Pb, ^{214}Bi, ^{214}Po, ^{210}Po (Table 14). It is claimed that inhalation of these radon decay products, attached to airborne particulates, is by far the largest contributor to total radiation exposure (90% and over). Because radon gas has a short half-life of 3,8 days, much of it decays before it reaches the surface. Therefore, a reduction in radon gas release can be achieved by retarding the rate of diffusion with the result that much of the radon decays before reaching the atmosphere. Measures stipulated in the USA for remedial action at inactive uranium mill tailings include:

- the covering of existing surface with a thick layer of earth to limit radon releases to below 0,74 $Bq/m^2.s$; or

- deep disposal (e.g. as backfills).

Field tests have demonstrated that both thick earthern covers and thinner asphaltic covers can readily reduce radon emissions by 90% to 99%. This aspect has been investigated by the Atomic Energy Corporation.

Radon gas is emanated from any ore body containing uranium and may be found even in non-uranium-producing mines in significant quantities to constitute a health hazard to workers. In addition radon can be transported in underground waters and released far from the point of emanation in sufficient quantities to constitute a health hazard in mines.

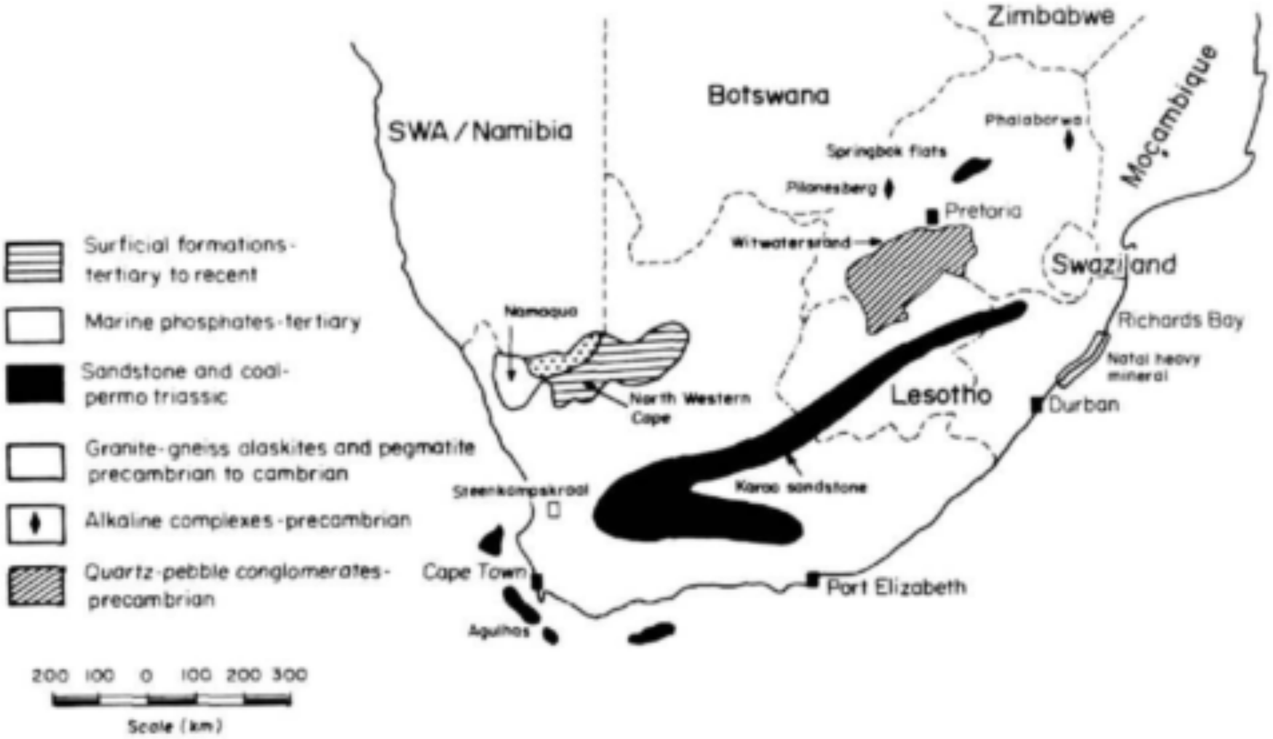


Figure 33
The occurrence of uranium in South Africa

Table 14: ^{238}Uranium decay series

Nuclide	Historical name (element)	Half life	Major radiation energies, (MeV) and intensities		
			Alpha	Beta	Gamma
^{238}U	Uranium I (Uranium)	4,51 x 10^9 years	4,15 (25%) 4,20 (75%)		
^{234}Th	Uranium X (Thorium)	24,1 days		0,103 (21%) 0,193 (79%)	0,063 (3,5%) 0,093 (4,0%)
^{234}Pa	Uranium X (Proactinium)	1,17 min		2,29 (98%)	0,765 (0,3%) 1,001 (0,6%)
^{234}U	Uranium II (Uranium)	2,47 x 10^5 years	4,72 (28%) 4,77 (72%)		0,053 (0,2%)
^{230}Th	Ionium (Thorium)	8,0 x 10^4 years	4,62 (24%) 4,68 (76%)		0,068 (0,6%) 0,142 (0,072%)
^{226}Ra	Radium	1 062 years	4,60 (6%) 4,78 (95%)		0,168 (4%)
^{222}Rn	Emanation (Radon)	3,82 days	5,49 (100%)		0,51 (0,07%)

Table 14(continued): ^{238}Uranium decay series

Nuclide	Historical name (element)	Half life	Major radiation energies, (MeV) and intensities		
			Alpha	Beta	Gamma
^{218}Po	Radium A (Polonium)	3,05 min	6,00 (100%)		
^{214}Pb	Radium B (Lead)	26,8 min		0,65 (50%) 0,71 (40%)	0,295 (19%) 0,352 (36%)
^{214}Bi	Radium C (Bismuth)	19,7 min		1,0 (23%) 1,51 (40%)	0,609 (47%) 1,12 (17%)
^{214}Po	Radium C' (Polonium)	0,164 sec	7,69 (100%)		0,799 (0,014%)
^{210}Pb	Radium D (Lead)	21 years		0,016 (85%) 0,061 (15%)	0,047 (4%)
^{210}Bi	Radium E	5,01 days		1,161 (100%)	
^{210}Po	Radium F (Polonium)	138,4 days	5,305 (100%)		0,803 (0,0011%)
^{206}Pb	Radium G (Lead)	Stable			

Indications are that in terms of mine environmental pollutants radon and its daughter products are the main contributors to the excessive incidence of lung cancer amongst miners. Human data are now available from several groups of underground metal-ore miners at USA, Canadian and Czechoslovakian uranium mines, Swedish and British iron mines, Swedish lead and zinc mines, and Newfoundland fluorspar mines. Although other potential carcinogens such as diesel smoke, traces of As, Ni or Fe ore are found in these mines, the lung cancer response appears to be predictable, based on Rn-progeny exposure (Cross, Harley and Hoffmann, 1985). Lung cancer hazard from inhalation exposure to Rn and Rn-progeny is apparently not from the Rn *per se* but rather from the alpha-dose delivered through lung deposition of the short-lived progeny of radon. Two alpha-emitters, ^{218}Po (RaA) and ^{214}Po (RaC^1) ultimately deliver the carcinogenic dose to basal cells of the bronchial epithelium, the critical tissue for induction of lung carcinoma.

Historically, inhalation exposure is defined in terms of the air concentration of Rn progeny in working level (WL) units. A working level is defined as a concentration of short-lived Rn progeny totalling $1{,}3 \times 10^5$ MeV of potential alpha-energy per litre of air. A working level month (WLM) is an exposure equivalent to 1 WL for 170 h.

The question arises as to whether exposure to radon gas and alpha radiation underground constitutes a threat to the health of workers at local mines. Is radon spread through the whole mine by means of ventilation? Is radon gas expelled sufficiently with ventilation up-cast air? Is air recirculation as practiced at Loraine potentially hazardous? Or is it safe as long as underground workers are not exposed beyond a certain period to a radiation dosis, expressed as WLM or whatever unit might be chosen? An investigation into the level of radon and its short-lived daughter products at uranium bearing horizons of local mines is underway.

It is the opinion of local mine management that good ventilation removes radon gas out of the mine to ensure safe working conditions underground. Additional methods like sealing off sufficiently worked-out areas and removing broken rock in working areas might be required. However, this does not mean that such hazards are in fact maintained at acceptable levels. The experience obtained to date on measuring underground levels of radon daughters has shown that ventilation flow rates necessary for heat control in some instances maintain the radon daughter levels acceptably low but in other instances not. It cannot be assumed that because a mine is apparently well ventilated no radon daughter problems will prevail. In addition to this there are very few and probably no adequate routine measuring programmes *in situ* to verify this.

Radiological conditions during surface operations are shown in Table 15 (Metcalf, 1990).

In view of the growing concern locally and internationally with regards to radiation hazards, a comprehensive and systematic assessment of this situation in the mining industry - underground and on the surface - is now regarded as necessary. To this effect, the Nuclear Energy Amendment Act (Act No 56 of 1988) established the Council for Nuclear Safety with regulatory powers over nuclear installations and activities involving nuclear-hazardous materials. These include

- mining and processing of uranium - and thorium - bearing areas;
- other operations involving such materials including, for example, monazite processing plants, as well as burning of pyrites in sulphuric acid plants.

In view of certain shortcomings in the existing Mines and Works Act it was decided, in consultation with the Government Mining Engineer and as an interim measure, for the Council to proceed with licensing in terms of the Nuclear Energy Act, in order to ensure adequate radiological safety (Council for Nuclear Safety, Annual Report 1988/89).

Table 15: Radiological conditions prevailing in various surface plants

Type	Dose rate	Long lived airborne alpha activity
	mrem/h	
Uranium plant	0,05 to 0,5	1 DAC*
Sulphuric acid plant	0,05 to 2,0	
Calcining facility	0,1 to 4,0	0,01 to 10 DAC
Phosphate (Baddeleyite)	0,05 to 2,0	Up to 8 DAC
Monazite (Mineral sands)	0,05 to 2,0	Up to 20 DAC
Monazite (Mine)	0,5 to 10	Up to 200 DAC
Uranium (High thorium content in ore)	0,5 to 300	Up to 10 DAC

DAC = Derived air-borne concentration (ore dust and uranium concentrate aerosol) equal to the inhalation during a whole year or about 2 000 exposure h/a representing the annual dose limit

5 DEVELOPMENTS TO REDUCE THE INTAKE OF LOW TDS WATER INTO MINES

5.1 Upgrading of treated sewage for reuse as mine service water

At Kloof the volume of potable water intake for the hostels, housing 13 500 occupants averages about 4 000 m^3/d or 0,296 m^3/person.d. The intake of potable water by the nine hostels at Vaal Reefs gold mine varies widely but is typical of mine hostels (Table 16). At Buffelsfontein the daily water intake for hostels averages 0,34 m^3/person.d . About 75% of the volume comes from the ablution blocks and about 25% from the kitchen and beerhall. Because of the generous intake of about 0,2 to 0,71 m^3/person.d of low TDS potable water, the raw sewage derived from mine hostels has a low TDS content compared with mine service water (Table 17). Consequently, scale formation and corrosion would be easier to control if mine hostel sewage would be upgraded for use underground. Investigations by the Division of Water Technology (DWT) of the CSIR and Gold Fields at Kloof indicated the possibility of producing a high-quality industrial water from treated sewage plant maturation pond water at a cost lower than that of Rand Water Board water. Ferric chloride dosage to maturation pond effluent followed by dissolved air flotation (DAF) removes phosphates, algae and other suspended solids and improves the quality of the water with respect to COD and turbidity. The low ammonia content in the maturation pond No 8 effluent of 0 to 1 mg/*l* indicates that complete disinfection with moderate chlorine dosage is achievable. The full-scale plant to produce high-quality water for use underground (of which a flow diagram is shown in Fig 34) was commissioned early in 1986.

This water treatment plant at Kloof will probably be further improved by incorporating ozone treatment for oxidizing nitrite, thereby reducing the chlorine demand for complete disinfection (McLaren, 1990).

5.2 Supplementation of RWB water with dolomitic water (Pullen and McLaren, 1989)

The drought experienced during the early 1980s and its resultant restrictions forced the mining industry to look for alternative sources of water. Driefontein Consolidated being situated on the water rich dolomites of the West Witwatersrand has isolated at its West Driefontein North Shaft the upper sources of underground dolomitic water some 960 m below surface, from where it is pumped to the surface at a rate of about 80 M*l*/d. This volume is split into a service water fraction and a potable water fraction. The potable water fraction of 10 M*l*/d is treated in a dual-media, downflow, pressure filter plant after which it is chlorinated and blended with RWB water at a ratio of 1:1 and reticulated into the existing potable water circuit.

The service water fraction is pumped to both the East and West Driefontein refrigeration plants. Excess water overflows into the existing canal system to feed the No 7 West Shaft service water intake and the Wonderfonteinspruit.

The plant was commissioned early in 1986; it is automated and requires little operator attention.

The North shaft water is made up of two sources, namely 55 M*l*/d of relatively uncontaminated dolomitic fissure water and 25 M*l*/d of spent service water from the No 4 shaft

Table 16: Monthly potable water intake and specific use by hostels at Vaal Reefs for January 1986

Hostel	No	Occupants	Water intake	Specific use
			m^3	m^3/person.day
East	1	4 723	41 983	0,296
	2	3 140	17 036	0,181
	5	4 363	66 963	0,512
West	3	3 506	21 840	0,208
	4	3 230	22 928	0,237
	6	2 843	14 177	0,166
	7	2 322	14 789	0,212
South	8	9 040	106 641	0,393
	9	5 744	118 149	0,686
Total		38 911	424 506	0,352

These figures do not include water used for sports fields

Table 17: Chemical quality of Pretoria, Kloof and Vaal Reefs settled sewage and mine service water

Determinand	Settled sewage			Underground mine service water	
	Pretoria	Kloof	Vaal Reefs	Kloof	Vaal Reefs
pH	7,2 to 7,5	7,1	6,8 to 7,1	7,3	5,0 to 7,0
TDS	550 to 650	800	590 to 860	1 200	1 400 to 4 200
SO_4	155	100 to 140	180 to 280	390	400 to 1 200
Cl	60 to 130	50	80 to 90	460	110 to 780
Total P	10 to 12	5 to 8	6 to 8	1	ND
NH_3-N	35 to 38	9 to 22	19 to 26	2	ND
NO_2+NO_3-N	10 to 12	1 to 13	2 to 13	24	ND
COD	340 to 410	600	600	15 to 50	ND

Concentrations in mg/l, except pH *ND = not determined*

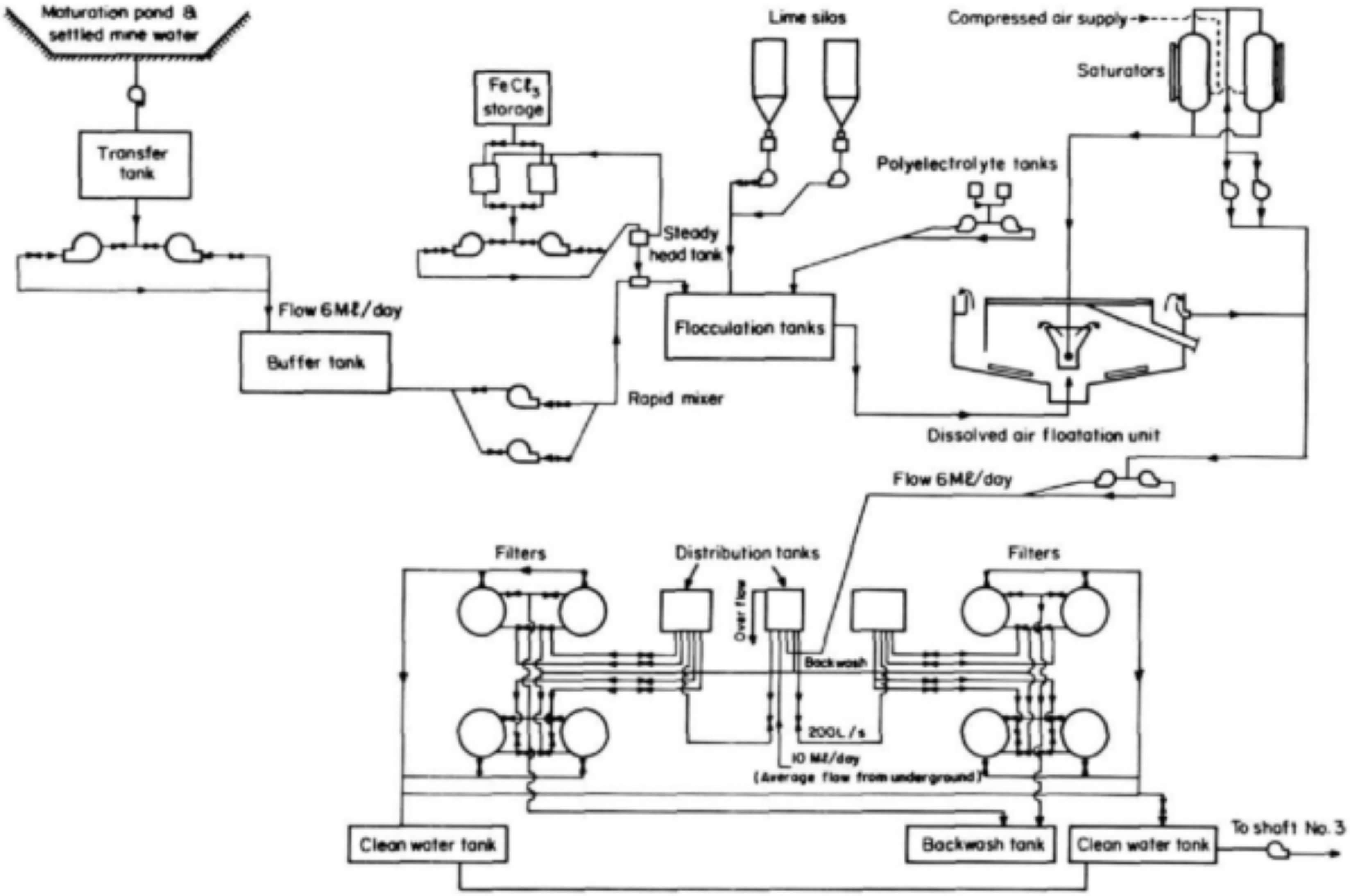


Figure 34
Flow diagram of dissolved air flotation (DAF) and filtration plant for the recovery of high-quality water from treated sewage/settled mine service water

working area. The 5 M*l* buffer tank receiving water from both sources was modified during 1987 to allow dolomitic water to pass peferentially to the potable water treatment plant.

The chemical composition of the water is shown in Table 18.

5.3 Desalination of mine service water

The desalination of water can be done by a number of different processes or a combination of such processes, which are discussed hereafter.

5.3.1 Membrane processes

The major problem for the desalination of mine water by membrane processes arises from the presence of sparingly soluble salts in the water, notably calcium sulphate. All desalination processes are concentrating processes which produce a fresh water and a concentrated brine stream from which salts will precipitate, once the solubility limit has been exceeded. Table 19 shows the average calcium sulphate saturation levels in various mining areas and also in a simplified way, disregarding interference by other salts, the corresponding possible water recovery before calcium sulphate scaling is likely to occur.

Water recoveries achievable before $CaSO_4$ saturation are low in some areas and can be as little as 5% to 20% at some mines. For conventional membrane processes to work effectively on these waters, an extensive pretreatment system would be necessary. Suspended solids removal, reduction in iron and manganese and chemical softening using lime and soda ash would be required to precipitate calcium and magnesium and to reduce silica levels. Pretreatment requirements would thus tend to make desalination of mine service water by conventional reverse osmosis uneconomical and complex.

Electrodialysis (ED) is a membrane process based on the principle that certain synthetic membranes will allow only negatively charged ions (anions such as Cl^-) to pass, while other types of membrane will pass only positively charged ions (cations such as Na^+). Electricity is used as the driving force. Pairs of membranes are placed alternatively in a cell, between two electrically charged plates (Fig 35). Positively charged ions in the diluting compartment are attracted to the negative electrode and pass through the cation-permeable membrane, thus becoming entrapped in the concentrating compartment. Similarly, negative ions attracted to the positive electrode are removed from the diluting compartment as they pass through the anion-permeable membrane to be trapped in the concentrating compartment. The process results in the desalination of water in the diluting compartments. Conventional ED has a unidirectional mode with cations moving to the negative fixed cathode and anions moving to the positive fixed anode.

In *electrodialysis reversal (EDR)* the polarity of the electrodes is reversed 3 to 4 times/h and in this way various fouling and scaling problems encountered with the conventional ED process have been overcome. A small pilot EDR plant which has been operated with Water Research Commission support at ERPM experienced frequent operational problems, primarily the blocking of the membrane stack by scale deposits.

Indications are, however, that EDR may be suitable for treating mine water with a low concentration of calcium sulphate. With equipment supplied by the Water Research Commission, COMRO has been operating a 140 m^3/d EDR pilot plant (Ionics) and a 30 m^3/d tubular RO plant (Bintech) for the desalination of mine water at the Beatrix mine. The

Table 18: Chemical composition of different sources of water at Driefontein Consolidated

Determinand	North shaft water (Sept 1983)	Dolomitic water (1988)	RWB water	1:1 Mixed potable water (1988)
pH	8,1	8,1	8,3	7,7
TDS	531	ND	386	ND
Conductivity*	ND	48,5	ND	< 50
Hardness**	247	252	143	180
Alkalinity**	142	139	ND	180
SO_4	128	76	68	60
Cl	27	22	34	21
NO_3-N	15	2,6	0,5	1,2
NH_4-N	0,1	0,3	< 0,1	< 0,1

** mS/m*

*** Total in mg $CaCO_3$/l*

Concentrations in mg/l, except pH and conductivity

Table 19: Calcium sulphate saturation level of mine water in various mining areas

Mining area	Average saturation level	Maximum water recovery before saturation limit
	%	%
Orange Free State	20	80
Klerksdorp	33	67
Far West Rand	55	45
East Rand	50	50
Evander	14	86

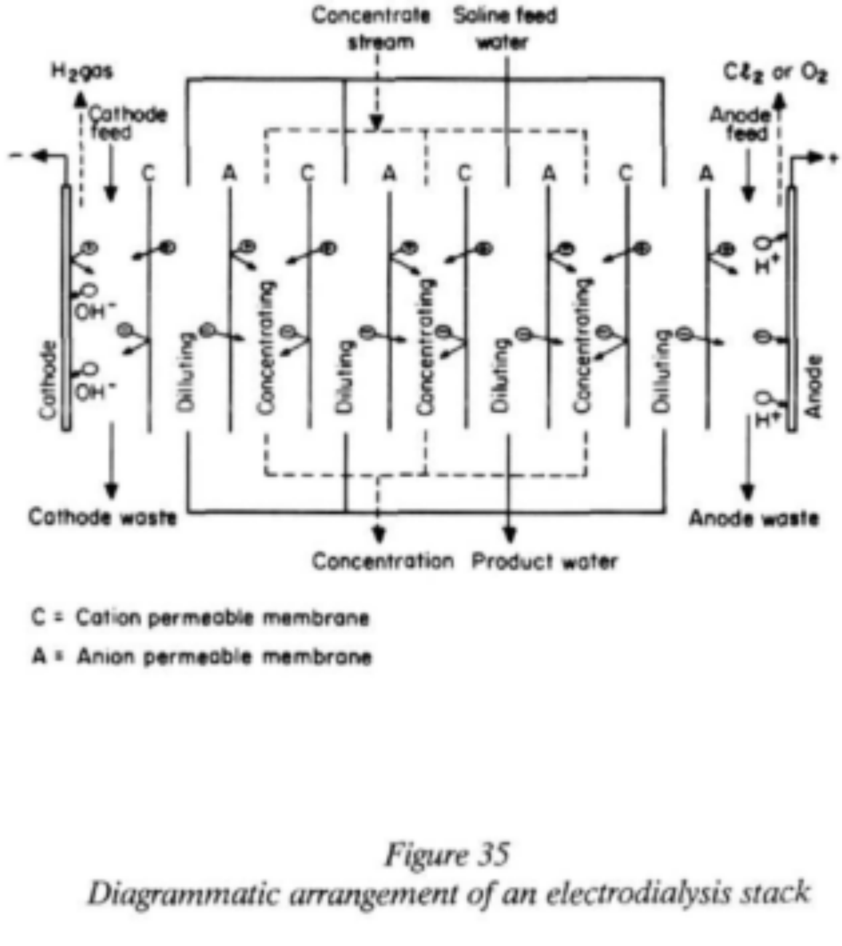


Figure 35
Diagrammatic arrangement of an electrodialysis stack

chemical composition of the feed, product water and brine is given in Table 20.

Pretreatment for the EDR plant consists of potassium permanganate oxidation of Fe^{2+}, flocculation, multimedia filtration and filtration through a cartridge filter (10 μm). Water with a temperature of as much as 40°C is tolerated. The water recovery is 80% with a 80% salt rejection. The flow scheme of this EDR pilot plant is shown in Fig 36. Pretreatment for the RO plant consists of pH adjustment, potassium permanganate oxidation of Fe^{2+} and multimedia filtration. Since the cellulose acetate RO membrane is temperature sensitive, the maximum temperature of the feed water is limited to 30°C. Water recovery is 75% with 80% to 85% salt rejection. Tests with both plants were completed by August 1987 (Juby, 1989).

The seeded reverse osmosis (SRO) process was developed to overcome the scaling and fouling problems encountered in some applications of conventional reverse osmosis. The SRO process is shown in Fig 37. Briefly, a slurry of *seed* crystals, principally calcium sulphate, is incorporated into the feed water of a tubular reverse osmosis system. These seed crystals then serve as preferential sites for the growth of additional crystals of calcium sulphate, silica and other salts and prevent the formation of scale on the surface of the membrane. Tubular reverse osmosis modules are required for SRO as slurries of 3% to 10% solids are circulated continuously in the system. Such concentrations of suspended solids would rapidly result in complete blockage of either spiral or hollow-fibre RO modules.

With support by the Water Research Commission COMRO has operated a small 5 m^3/d pilot SRO plant at ERPM, with 92% reliability and a water recovery of between 92% and 96%. The tests showed that SRO was capable of producing a good quality water with 350 mg/*l* to 980 mg/*l* TDS from mine water averaging 2 850 mg/*l* at satisfactory membrane fluxes and using minimal pretreatment. There has been no evidence of membrane scaling by calcium sulphate or fouling by contaminants present in the mine water. However, tests with a larger 50 m^3/d SRO pilot plant at the ERPM gold mine had shown that the SRO process was inefficient energy-wise and required a high rate of recirculation, resulting in heating, necessity for cooling and a high degree of wear on the slurry pumps. Consequently, a new process has been developed by COMRO which should eliminate the abovementioned disadvantages, known as the slurry precipitation and recycle reverse osmosis (Sparro) process. A 0,85 *l*/s (73 m^3/d) pilot plant has been erected at ERPM with support by the Water Research Commission and is currently being tested and evaluated.

5.3.2 Distillation

The most common desalination process for sea water is distillation. However, about 80% of the salt content of sea water is NaCl and, furthermore, a relatively low rate of recovery of desalinated water is acceptable because the supply of saline feed water is unlimited. Studies by COMRO have shown that generally distillation processes are unsuitable, mainly because of high capital and operating costs and also because of the severe scaling expected to occur at operating temperatures. No further studies in this direction are planned in the near future.

5.3.3 Desalination in combination with ice-making for mine cooling

Freezing is claimed to have several advantages over other methods of desalination, including the following (Shone, 1987):

Table 20: Average chemical composition of product water and brine of the EDR and RO pilot plants at Beatrix mine over 3 500 h of operation

Method of desalination		TDS	Cl	Na	Ca	SO_4	Fe *	Mn	Ba	Turbidity	pH
										NTU	
Electrodialysis reversal (EDR)	Feed	3 184	1748	1392	100	74	0,6	0,6	0,6	1,0	6,4
	Product	637	376	193	25	5	0,5	0,2	0,3	0,4	6,1
	Brine	9 135	4 990	3 110	ND	344	0,9	1,4	1,0	0,4	5,4
Reverse osmosis (RO)	Feed	2 952	1 725	1 280	ND	70	1,0	0,6	0,6	1,0	5,7
	Product	470	281	152	ND	3	0,4	0,2	0,3	0,4	5,9
	Brine	9 950	5 400	3 000	ND	260	3,0	1,2	2,0	2,0	5,4

Concentrations in mg/l, except pH and turbidity ** Total* *ND = Not determined*

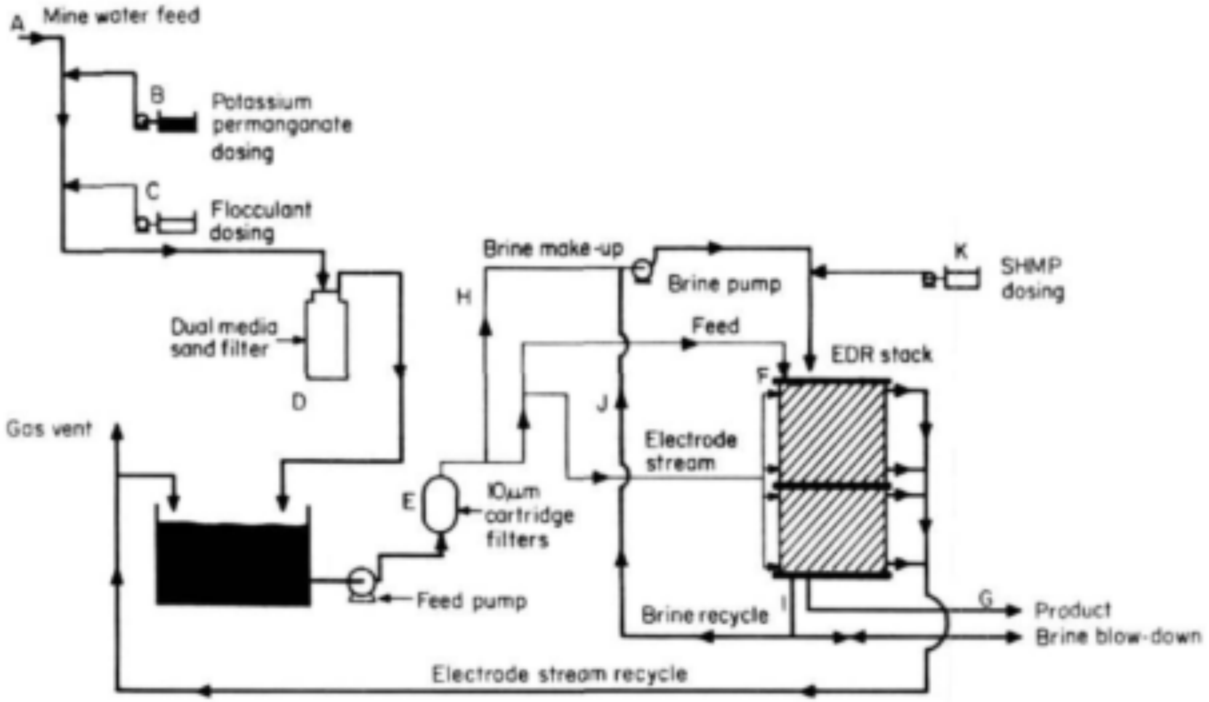


Figure 36
Flow scheme of EDR pilot plant

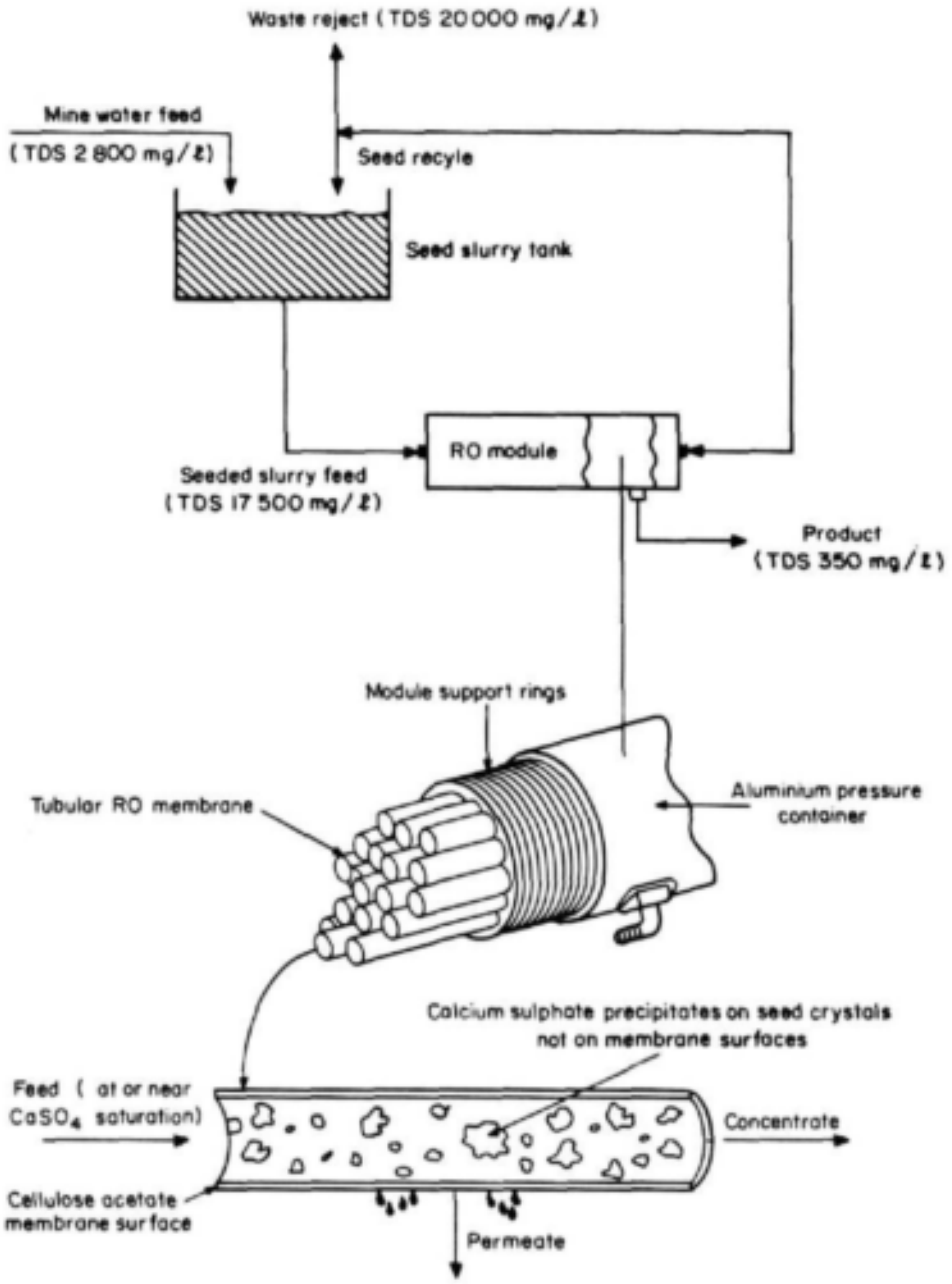


Figure 37
Arrangement of a seeded slurry reverse osmosis system

(i) The quantity of heat to produce 1 kg of clean water by freezing is about one-seventh of that needed to recover the equivalent amount of water from the condensed vapour of a distillation process.

(ii) Unlike many desalination processes, freeze desalination is virtually insensitive to the salinity and composition of the feed water. A product water of low salinity (400 mg/*l* TDS or less) can be obtained, and a high proportion of the feed water can be recovered as clean water.

Despite these advantages, no full-scale commercial freeze desalinator has yet been built. The reason is that, because of high capital costs, no significant savings in total costs have been anticipated in the production of desalinated water by freezing over other more established methods. In the mining industry, however, the potential for increased cooling by utilizing the latent heat of melting ice combined with the potential for desalination makes freeze desalination attractive.

The existing batch production plant at Harmony (1 000 t/d) produces ice sheets with almost all the impurities in the feed water contained in the ice produced. In contrast to the later production of particulate dry ice from ice making installations as at Harmony and at ERPM which require low TDS feed water, slurry ice can be produced continuously as a mixture of ice and saline water, directly from high TDS mine water. Slurry ice has two major advantages (Shone and Sheer, 1988; Gebler, *et al.*, 1988; Ramsden and Carvalho, 1988 and Ramsden, Bluhm *et al.*, 1988):

- There is no build-up of ice on a freezing surface as in conventional ice makers. This results in a more efficient ice-making process due to less thermal resistance.
- Micro-crystalline ice is pure and after washing and melting, desalinated water is obtained.

A slurry ice-maker could thus be adapted as a dual-purpose ice-maker and freeze desalinator.

There are three basic ways of making slurry ice :

- indirectly where water flows through tubes with refrigerant evaporating on the outside;
- by vacuum, where ice is formed at the triple point of water; water vapour is drawn off resulting in evaporative cooling and ice formation; and
- directly where brine and an immiscible refrigerant are sprayed into a vessel; the refrigerant evaporates and removes heat from the brine causing ice crystals to form.

Perhaps the most important single technical factor relating to the production of ice slurry is that of the composition and salinity of the water. Ice made from relatively clean water will tend to grow quickly as long dendrites which will interlock to form sheets or conglomerate lumps and, if adjacent to a cold surface, will form a layer on the surface. When, however, ice is made in a concentrated and stirred salt solution the crystals tend to remain discrete and pure. The slurry is homogenous and up to a point pumpable. With suitable modifications the process can yield desalinated ice.

The concentrated solution results in reduced performance because the freezing temperature is lowered by an amount known as freezing point depression. Fig 38 shows the freezing point depression as a function of salinity for water containing mainly sodium chloride as in sea water or in OFS mine water and a typical Transvaal mine water (mainly magnesium sulphate).

A pilot plant based on the *indirect freezing process* as developed by the Chicago Bridge and Iron Company has been tested during 1987 at ERPM using mine water (Fig 39). The freezer tubes were specially treated to prevent ice from sticking to the inner surface. It was found that the ice-production rate was low but that the ice product was of good quality despite very poor quality mine water. After separation of the small ice crystals from the concentrated brine containing up to 17% salt and after washing to remove surface salts a clean ice-water slurry exceeding 95% of the feed water was obtained at a purity better than 500 mg/*l* TDS.

The principal merit of the *vacuum freezing process* is that it is a direct contact heat transfer process using water as its own refrigerant. The basic process is well known and operates at the triple point, at which water can exist in three phases: solid ice, water and water vapour (Fig 40). For pure water the triple point exists at an absolute pressure of about 600 Pa at 0 °C. For a salt solution the pressure and temperature are lower. The water to be frozen is contained in a vessel and water vapour is withdrawn, producing a vacuum and resulting in the evaporative cooling of the bulk of the water. For vacuum ice production, the pressure in the ice-maker must be at or below the triple point pressure for the given quality of water. The ice will normally form at the surface of the water because this is where the principal heat and mass transfer occurs; the water must, therefore, be suitably agitated to produce a homogenous ice water mixture.

The major component of the vacuum freezer is a rather special vapour extraction pump which has to handle a large volume flowrate because of the high specific volume of water vapour at low pressure. For example, to produce 1 kg/s of ice approximately 40 m^3/s of vapour must be withdrawn. The experimental vacuum freezer on which COMRO tests are being conducted is shown in Fig 41. The extracted slurry ice here shown to be recycled will eventually be discharged to a growth vessel and a steady rate of ice production will be maintained.

Ice crystals are best made in a concentrated salt solution. In the saline slurry the crystals are pure but surrounded by brine which adheres quite tenaciously to the surfaces. Therefore after separation from the brine the crystals must be washed with clean chilled water to remove surface salts. The key to a cost-effective freeze desalination process is to produce crystals which are sufficiently large and uniformly sized - ideally spherical - to facilitate washing with a minimum volume of wash-water.

Washing is done in most cases in a wash column (Fig 42). The ice-brine mixture is introduced at the base and the ice floats to the top, forming a permeable bed. Most of the brine drains immediately and that which adheres to the surface of the crystals is removed by the clean wash water which is distributed over the top of the ice float.

In order to obtain large and washable ice crystals, a *growth* vessel is used between the stages of ice formation and washing as shown in Fig 43. The growth vessel is an insulated vessel in which the ice-brine mixture is stirred at freezing temperature. In a solution containing different sizes of crystals, the larger crystals tend to grow at the expense of the

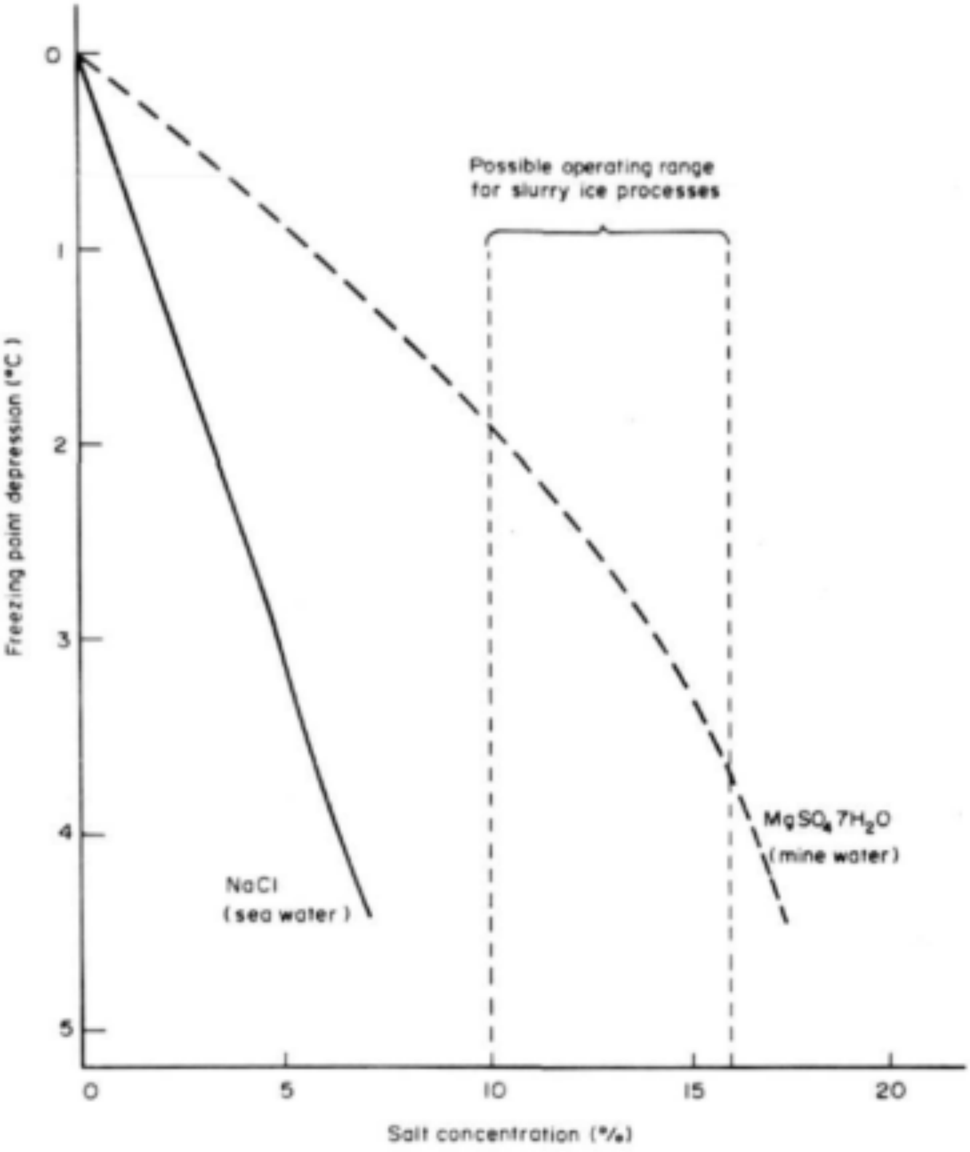


Figure 38
Freezing point depression for water containing sodium chloride (NaCl) and magnesium sulphate ($MgSO_4$-$7H_2O$)

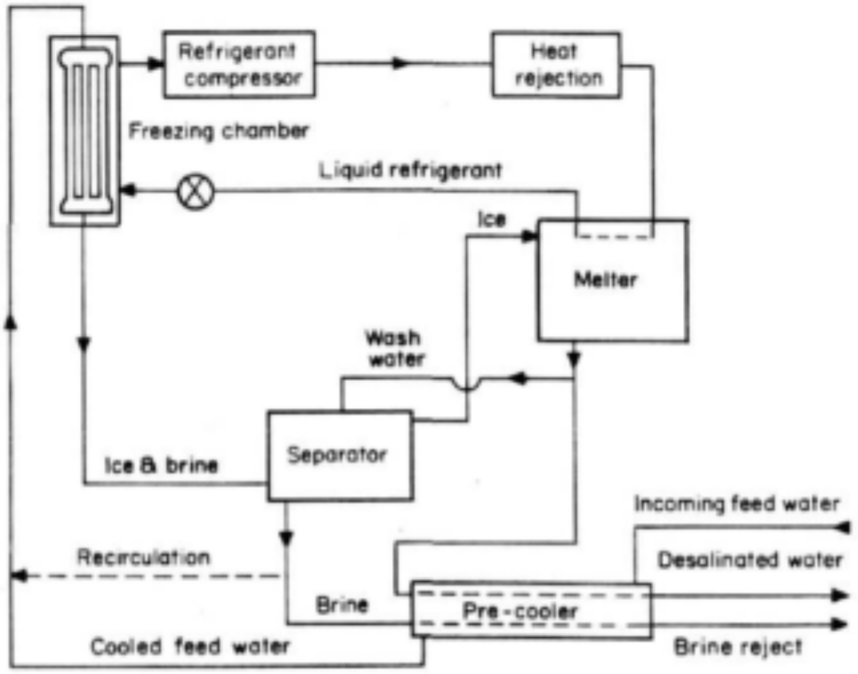


Figure 39
Schematic arrangement of a freeze desalination process
(ERPM pilot plant, Chicago Bridge and Iron Company)
(COMRO, 1987b)

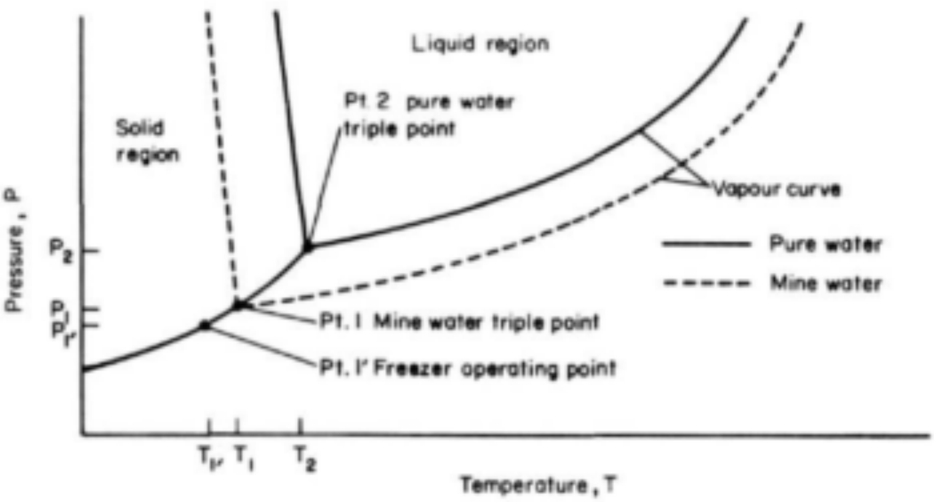


Figure 40
Phase diagram for pure and mine water

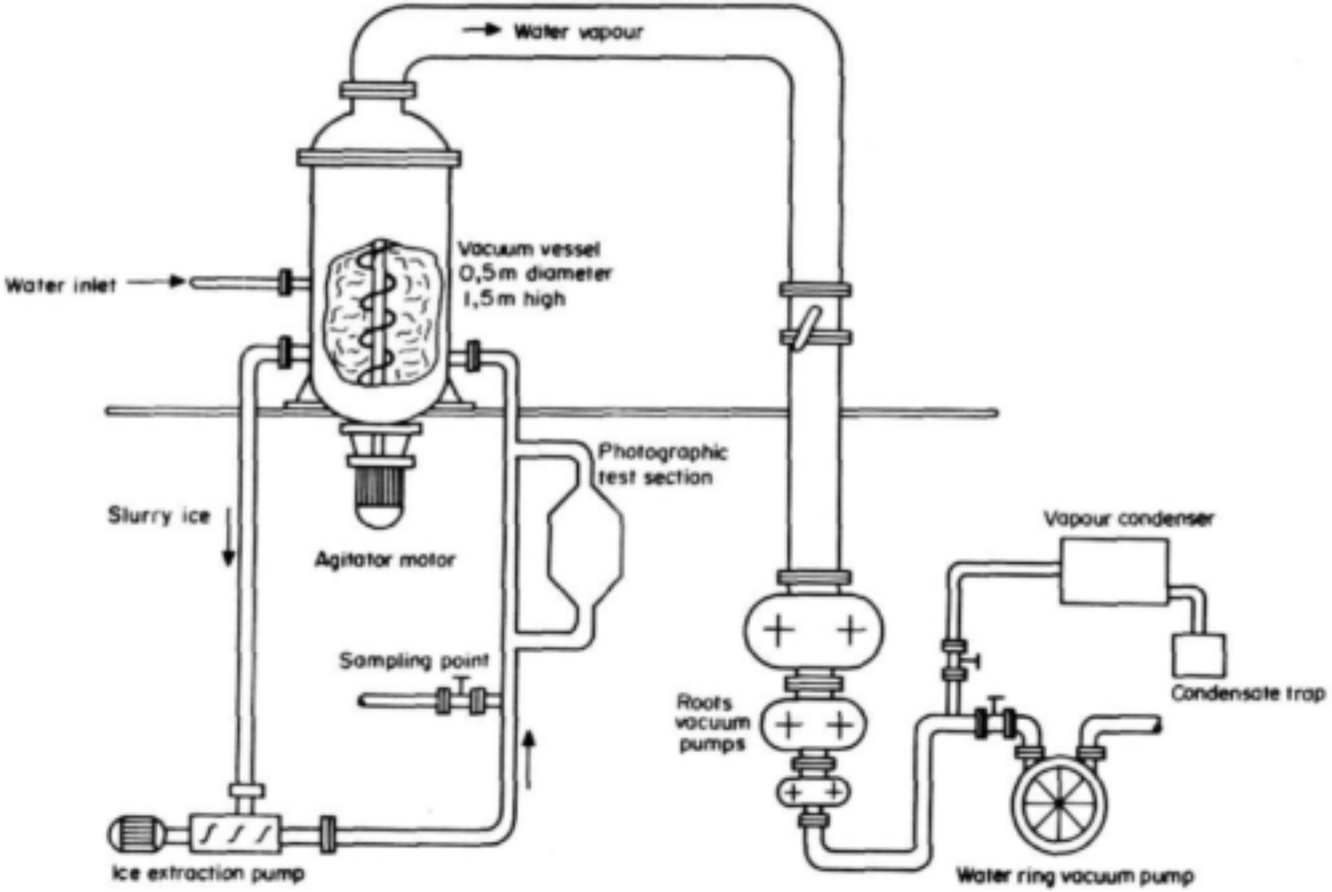


Figure 41
Experimental vacuum freezer

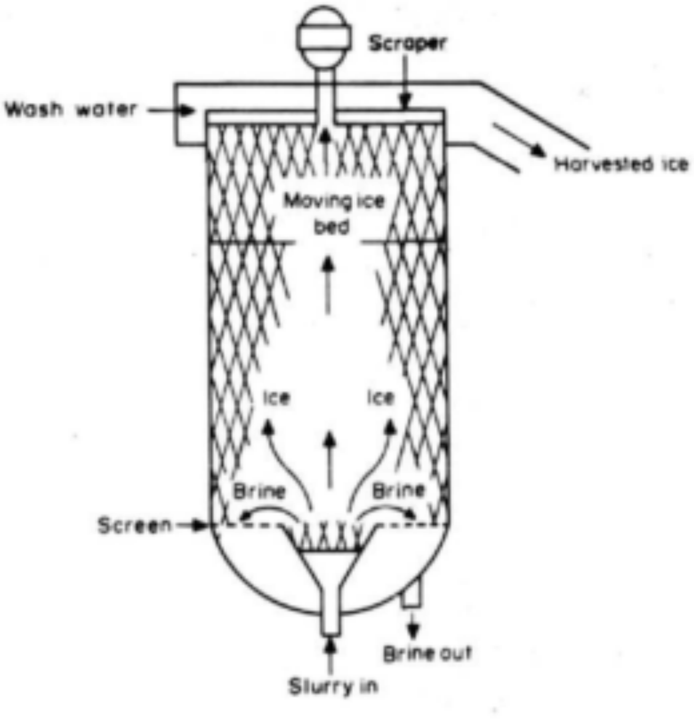


Figure 42
Wash column

smaller ones. The residence time required for growth differs according to the salinity of the solution and size distribution of the crystals and it may take several hours to grow large enough crystals as shown in Fig 44.

Any desalination process is a concentrating process in which water is extracted from brine, the remaining brine becoming more concentrated. For a high extraction of low TDS water the brine concentration may become very high. Considering for example, a mixed water feed with a concentration of 2 500 mg/*l* TDS from which it is desired to extract 97% of the pure water, the remaining brine will have a concentration of about 83 000 mg/*l* TDS, if all the salts remain in solution. As is often the case with mine waters containing sparingly soluble salts such as calcium sulphate, some salts precipitate and the final TDS concentration of the brine will be lower.

A design to produce *vacuum ice* has been developed in Denmark by Sabroe & Company in cooperation with Israeli Desalination Engineering Limited. The great advantage of this vacuum-ice process is the lower energy consumption of only 29,6 kWh/t of ice as compared with 43,1 kWh/t for plate and tube ice.

A pilot plant of 350 t/d capacity (as dry ice) to produce slurry ice for cooling has been installed at Western Deep Levels. Since the surface water supply contains only about 1 000 mg/*l* TDS, desalination is not the aim of the testing. NaCl is added to increase the concentration of the *brine* supply to the vacuum ice plant to only about 6 000 mg/*l*. The result is a slurry ice of 15% to 17% concentration with a low content of impurities. After dewatering and, if required, washing the ice concentration will be increased to between 75% and 90%. Brine is to be returned to the ice maker. The make-up requirements for fresh brine and disposal of spent brine will thus be kept at a minimum.

Slurry ice pumping and transportation in pipes are limited by the ice mass fraction. Depending to some extent on the size and shape of the ice crystals, possibly at about 60%, the slurry will cease to flow normally; the flow will tend towards extrusion and pumping energy will be excessive.

Testing to transport a slurry with a high ice mass fraction of up to 90% is under way and may possibly include a three-chamber hydrolift system as described in Section 3.2.3. (viii), extending to a depth of 1 200 m.

A 1 000 t/d slurry ice pilot plant of Canadian design, Sunwell Eng. Limited, is presently being constructed for the Vaal Reefs No 9 shaft. It is intended that this slurry ice plant will produce desalinated ice for bulk air cooling 2 300 m below surface. The brine will be made up of mine service water, enriched with NaCl to a solution strength of about 50 000 mg/*l* TDS. The brine will be subcooled to -3 °C in a pack of plate heat exchangers cooled with Freon or ammonia. The ice produced as a slurry with 5% to 10% ice mass is scraped from the plates. The slurry ice/brine solution is then transferred to a growth vessel sufficiently large to allow for a one-day retention period. Microcrystalline ice slowly rises to the surface, thereby growing in size to about 1 mm.

A continuously operating wiper collects the ice which is to be sent down the shaft either as a 25% to 40% slurry ice, or even as almost dry ice into a melting dam where it will be mixed with mine service water. The ratio of ice to adherent brine water will determine the degree of desalination after melting. Insoluble salts such as calcium and magnesium sulphates and carbonates precipitate and collect at the bottom of the freezer/growth vessel

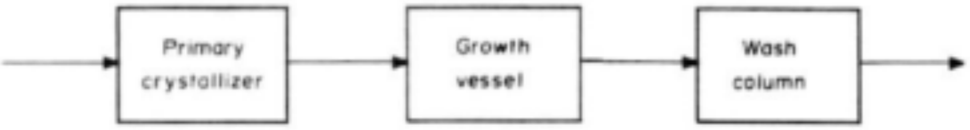


Figure 43
Unit operations for a freeze desalination plant

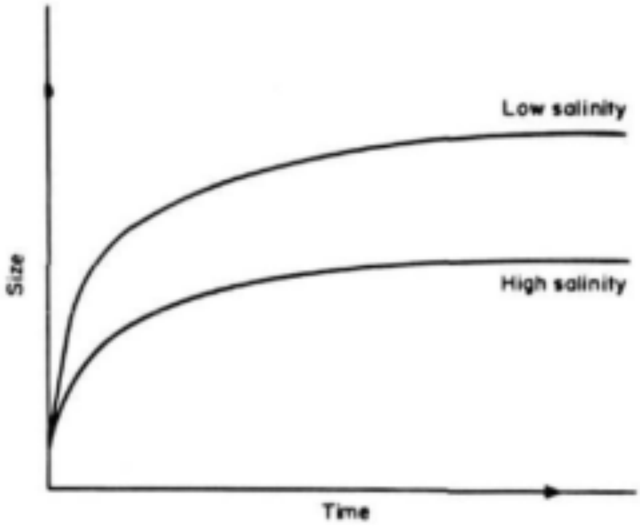


Figure 44
Crystal growth rate

below the heat exchangers and are removed for dumping. To compensate for this *desalination* of the brine, mine service water and probably NaCl will be added as feed to the freezer growth vessel. The schematic set-up for this slurry ice plant is shown in Fig 45. Operating results are expected during early 1991.

5.4 Air-cycle refrigeration

The basic principle as indicated in Fig 46 is to compress a portion of the downcast ventilation air on the surface, pipe it underground and there expand it through a turbine (driving an electric generator) to sub zero temperatures. This very cold air (- 50 °C), would be mixed with recirculating ventilation air to provide satisfactory ambient conditions in the mine.

Feasibility studies carried out by COMRO to evaluate the viability of this concept at various depths, included -

(i) The method of waste heat recovery on the surface by

- absorption refrigeration to produce chilled water;
- a steam turbine cycle for power generation; and
- production of domestic warm water.

(ii) A compressed air-drying system, since ice formation might create problems in the expansion turbine.

Air-cycle refrigeration could make good use of any surplus air compression capacity that might result from future changes on mines from pneumatic rock drilling to hydraulic rock drilling. Costs of pumping water to the surface would be greatly reduced. Low TDS water would not necessarily be required for heat rejection on surface. As far as cooling make-up water requirements are concerned, the volume would depend on evaporation/windage blowdown losses and would be the same for a given cooling duty, irrespective of whether cooling is distributed underground by water, ice or air. However, technical problems have been experienced with compressed air drying and with air turbine freezing. Costwise, air-cycle refrigeration appears to be feasible only for mines at depths of below 4 000 m which can use the waste heat on the surface. At present mines are not yet working at sufficient depths to pursue air-cycle refrigeration further.

5.5 Recovery of moisture from ventilation upcast air

A rough calculation, provided by Livanos (1986), is used to illustrate the possible recovery rate of water from the ventilation upcast air for a typical shaft at conditions shown below:

		Moisture g/kg		
Fan drift temp °C dry/wet bulb	Air flow kg/s	Fan drift	Bottom upcast	Free in fan drift
23/23	600	21,4	24,8	3,4

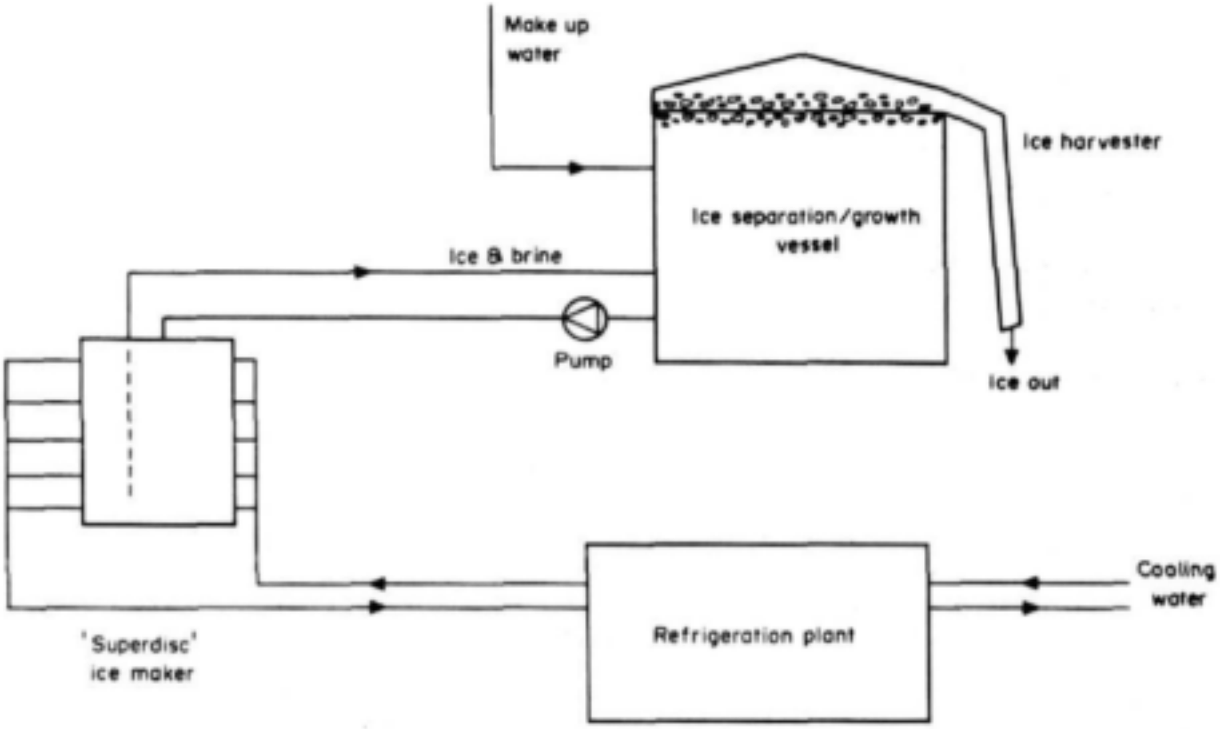


Figure 45
Sunwell superdisc slurry ice production for desalination

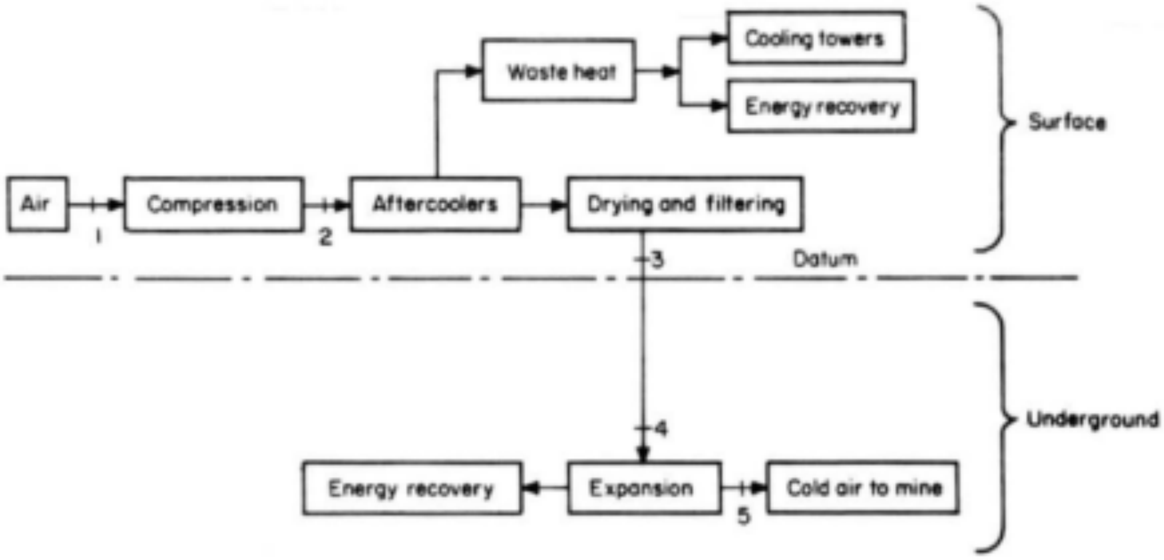


Figure 46
Basic principle of the air-cycle refrigeration system

The distribution of moisture as vapour and as condensed droplets at fan drift will change with cooling at saturated air leaving conditions as shown below:

Temperature in °C					
23	20	15	10	5	
21,4	17,7	12,8	9,2	6,5	g/kg moisture as vapour
3,4	3,4	3,4	3,4	3,4	g/kg available condensed moisture at 23 °C
Nil	3,7	8,6	12,2	14,9	g/kg additional moisture condensed
3,4	7,1	12,0	15,6	18,3	g/kg total moisture available for recovery at 100% efficiency

Thus, at 100% efficiency and 600 kg/s upcast air mass flow, at 23 °C the free moisture recovery only is 600 kg/s x 0,0034 kg/kg = 2,04 kg/s = 64,3 M*l*/a. At 10 °C dew point the free plus condensed moisture available amounts to 600 kg/s x 0,0156 kg/kg = 295 M*l*/a.

To recover either free moisture only, or free moisture and condensed vapour, there is an energy penalty in as much as some form of inertial separator must be used which has a pressure drop that must be overcome by the fan. The inertial separator must meet the following conditions:

(i) It should not foul up over time.

(ii) It must be installed in the physical space available.

(iii) Any installation time at an existing shaft in production longer than <u>one day</u> would be inadmissible in terms of loss of gold production:

- Assume 4 000 t/d of rock x 7 g/t Au = 28 kg/d Au x R25 000/kg = R700 000/d loss.

If condensed vapour as well as free moisture is to be recovered it needs to be borne in mind that:

(i) cooling of free moisture to the vapour dew point uses energy that must be included in the calculation; and

(ii) whatever device does the cooling it would ultimately use water which has to be deducted from water recovery.

One point in favour of such a recovery scheme would be that refrigeration capacity on surface plants is invariably oversized since it is designed for cooling duties at hot midsummer conditions. Spare refrigeration capacity is, therefore, available for extended periods.

With regards to mineral dissolved solids, condensed water recovered at fan drift should be of almost distilled water quality, contaminated for a few hours a day by nitrous fumes and by dust from blasting. Unfortunately, this is not always the case since fissure water or droplets of mine service water will contaminate the upward flow of moisture. Analytical data for fan drift condensate from OFS ventilation upcast shafts and other waters are given in Table 21.

Mines in the OFS gold-fields depend much on saline fissure water, pan water or water ex Beatrix mine, and therefore recovered water with a low concentration of dissolved solids is in demand as:

(i) cooling water in the refrigeration plant open condenser circuit, thereby saving costs for chemical treatment to reduce corrosion and scale formation,

(ii) cooling/scrubber water for sulphuric acid plants, since high chloride water attacks stainless steel; and

(iii) scrub liquor in the PURLEX solvent extraction of uranium after its leaching with sulphuric acid.

Recovery of water from ventilation upcast air which has a low TDS should be seen as an alternative to desalination of fissure water. This form of moisture recovery will require a high planning input and probably high capital outlay, but probably will operate at lower energy costs than other forms of desalination. One likely candidate for such a recovery scheme is probably President Steyn, Shaft No 4 at Welkom. The relevant psychrometric data are given below.

Shaft No 4	Fan drift t °C	Air density kg/m^3	Mass air flow kg/s	Moisture pick-up
Summer	38,0/38,0	0,8763	657,2	75,84 M*l*/m
Winter	36,0/36,0	0,8860	778,9	87,36 "

A substantial volume of low TDS water could probably be recovered, thereby saving costs to pump a corresponding volume ex Vaal River from Balkfontein over a distance of some 90 km.

5.6 Improving mine water quality for reuse, full-scale testing

Mine water is often abrasive, corrosive, scale forming and fouling. This has major practical and economic consequences in terms of reduced equipment life and increased power requirements. The costs to the gold mining industry are estimated to be as high as R300 x 10^6/a (Wiechers, 1990). COMRO in collaboration with mine groups, individual mines, other research organizations, water treatment specialist consultants and equipment and instrument suppliers has developed, refined and tested various surface and underground unit processes for treating mine water for reuse underground. However, the practical implementation of new or improved water treatment technology within the industry has not taken place to a major extent. The main reasons for this is:

(i) The lack of sufficient data to demonstrate the cost benefits associated with improved water treatment.

(ii) The difficulty in accurately predicting and quantifying the scale-up factor from research pilot plants to operational full-scale plants. Consequently, COMRO with the financial support of the Water Research Commission is erecting a 13 *l*/s

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Table 21: Chemical composition of fan drift condensate and other sources of water supply of OFS mines

Source of water		pH	Conductivity	TDS	Hardness		Alkalinity	SO_4	Cl	Na	NO_3-N	NH_3-N
			mS/m		Ca	Total	Total					
OFS Goldfields Water Board		7,8	87,5	680	200	310	90	255	72			
Loraine	Fissure	6,6	517	3 390	560	700		255	1 560	865		
	No 4 Fan	6,5	32	250	100	100		39	58	26	2,3	0,9
	No 5 Fan	7,1	56	435	80	100		50	97	97	4,4	0,5
from Beatrix		7,6	527	3 450	274	302	29	40	1 710	1 150	44	
Western Holdings	No 1 Fan	6,3	96	750	60	160	120	200	150	146	14	
	No 2 Fan	6,7	65	502	100	180	140	200	64	66	12	
	No 3 Fan	7,6	110	860	200	320	150	110	150	97	11	
	No 5 Fan	6,8	50	390	80	100	50	120	73	60	14,4	
	Return dam	8,3	710	6 400	2 145	2 230	112	1 990	1 940	1 400		

Concentrations in mg/l except pH and conductivity. Hardness and alkalinity are given in $CaCO_3$. The results were kindly supplied by the Division of Water Technology (CSIR) demonstration plant incorporating unit processes for neutralization, settling, filtration, desalination by the slurry precipitation and recycle reverse osmosis (Sparrow) process and disinfection (Aquarius plant, Fig 47). The plant will be studied on the surface and will treat mine waste water pumped from underground and will have a guaranteed supply capacity of 12 *l*/s (1 M*l*/d). All unit processes have been tested on pilot-plant scale to prove their practical feasibility. The plant is to be constructed in such a way that it can be dissembled and moved to different mine locations resembling mine water quality for that area. Three test sites for the Aquarius plant are envisaged as Westonaria/Carletonville, East Rand and Welkom.

The project will be carried out as follows:

Plant construction and operation at site 1	January 1990 to March 1991
Plant construction and operation at site 2	April 1991 to March 1992
Plant construction and operation at site 3	April 1992 to March 1993.

The product water will be monitored for scale formation, fouling and corrosion.

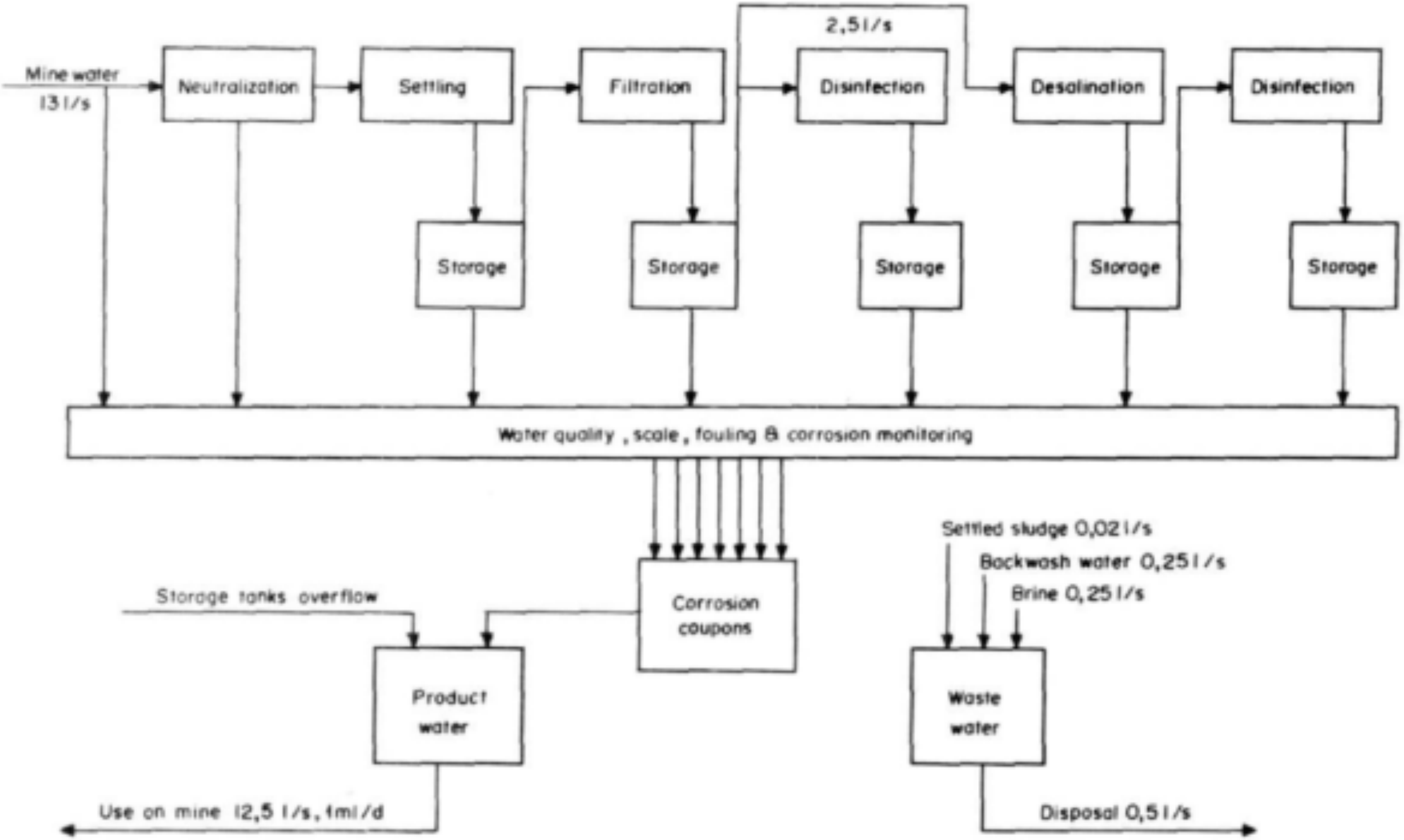

Figure 47
Aquarius plant

6 RESEARCH PROJECTS TO REDUCE POLLUTION

6.1 Reduction of underground acid formation

In gold mining major problems occur as a result of the presence of inorganic salts in mine water. These arise from:

- salts from fissure water and leaching of broken rock;
- the oxidation of pyrite, giving rise to acidity, sulphate and iron in solution; and
- the addition of lime to neutralize acidity giving rise to high calcium concentrations in solution, with the potential to precipitate gypsum.

Consequently, the primary problems which result are:

- corrosion (acid ferric sulphate is very corrosive); and
- fouling of pipes and heat transfer surfaces due to gypsum precipitation and scale formation.

The application of about 25 to 50 mg/*l* of a suitable anionic surfactant, sodium lauryl sulphate (SLS), to the service water, offers a unique way of controlling the formation of acid mine drainage and the associated need for neutralization with lime. Under experimental laboratory conditions the addition of 50 mg/*l* of sodium lauryl sulphate (SLS) had a significant effect in minimizing the generation of acidity from gold bearing reefs containing pyrite or pyrrhotite.

In view of the positive results obtained COMRO undertook underground trials to test the effect of SLS application in actual mining operations. A trial was started in three shaft systems at West Rand Consolidated near Roodepoort. The underground mine water was dosed continuously with SLS to maintain a 50 mg/*l* concentration level to compensate for bleed off over a period of about three months. Unfortunately, the results were not up to expectations since ingress of fissure water made the control of SLS dosage impossible. Further trial runs indicated that it would be impossible to add the anionic surfactant to the mine water before the acidification reactions had already started. This project has, consequently, been abandoned.

6.2 Inhibition of bacterial oxidation of pyrite

A research project to study the inhibition of bacterial oxidation of pyrite was started in 1983. Prof Loos of Stellenbosch University concluded during 1984 that due to pyrite oxidation and leaching, sulphur had largely disappeared from the outside region of the sand dump investigated. In a desk study on the likely contribution of slimes dams and sand dumps to the salt pollution of the Klip River entering the Vaal Barrage during the season of 1982/83, it was concluded that this contribution was very small, probably in the order of 2% or less (Funke, 1984). Consequently, it was decided to discontinue relevant research on gold mine dumps.

6.3 **Sulphate removal from effluents** (Schutte and Maree, 1989)

Mining effluents on the Witwatersrand are generally saturated with respect to calcium sulphate. On laboratory scale it was shown that sulphate, calcium and heavy metals present in mine effluents can be removed by a three-stage anaerobic/aerobic/anaerobic biological process, with virtually only sodium, potassium, magnesium, chloride and high concentrations of bicarbonate remaining in the water.

By using molasses as carbon source or generator gas consisting of H_2, CO and CO_2 in either an upflow anaerobic packed bed or sludge blanket reactor, sulphate is reduced to sulphide, which precipitates heavy metals. Approximately 90% of the inflow sulphate was removed at a hydraulic retention time of 10 h when 3 g molasses/*l* of effluent was added, from a concentration level of about 2 500 mg/*l* to about 200 mg/*l* SO_4. Surplus H_2S which has not reacted with the heavy metals, is removed by gaseous stripping. This H_2S gas can be converted biologically to elemental sulphur in additional process stages, or chemically by reaction with SO_2. The complete removal of H_2S is essential since traces of H_2S attack copper in electrical substations by forming a non-conductive CuS, resulting in power failure and burnt-out motors.

In the subsequent aerobic stage, the residual organic matter from molasses addition is, to a large extent, degraded and removed as CO_2. Unreacted heavy metals are oxidized and removed as insoluble hydroxides or oxides. Calcium is precipitated as insoluble $CaCO_3$ and can be returned to the acid mine effluent feed-stock for neutralization, thereby saving the costs for neutralization chemicals prior to the first anaerobic stage.

During secondary anaerobic treatment, residual organic matter consisting mainly of phenol-like components, is further degraded. By adding this stage, heavy metals are removed completely, including complex nickel cyanide. The effluent from the three-stage process is of good quality and appears to be suitable for all metallurgical processes. It has, however, so far only been tested on mine effluents and not on the raffinate from the extraction of uranium. The interference of gel-like iron hydroxide formation as in the process of neutralization with lime or raffinate would be eliminated by the removal of Fe as insoluble FeS, which in turn could be added as feed-stock to the roasting stage of the H_2SO_4 plants. This anaerobic/aerobic/anaerobic process offers a route to remove residual organic solvent, to neutralize and to desalinate raffinate partially for reuse in the metallurgical section, with the possibility of drastically reducing the large volume of intake water for uranium extraction.

6.4 **Disposal of mine water by irrigation** (Du Plessis, 1982)

Lime-treated mine waters are virtually neutral, but remain high in TDS, a factor which limits their usefulness. The fact that in many mine effluents the main constituents are calcium and sulphates, whereas sodium and chlorides are present only in low concentrations, leads to interesting possibilities of using them for irrigation. The chemical composition of the neutralized mine effluents presently being discharged into the Vaal Barrage, is given in Table 22.

Minimum leaching is necessary if a significant reduction in salt load is to be achieved through irrigation. The marked reduction in soil solution conductivity for the sulphate water relative to chloride water is the result of precipitation of gypsum as the soil solution becomes increasingly concentrated due to water uptake by plants. Relatively high yields

Table 22: The chemical composition of neutralized mine effluents discharged into tributaries flowing into the Vaal Barrage

Effluent		pH	TDS	SO_4	Cl	Na	K
From	Discharged to						
Grootvlei	Blesbokspruit	7,0 to 8,1	1 550 to 2 360	520 to 970	-	-	-
ERPM HDS plant	Elsburgspruit	6,4 to 8,1	3 800 to 4 500	1 750 to 3 000	100 to 140	140 to 220	13 to 20
DRD *	Klip River	4,7 to 10,0	2 800 to 3 900	1 900 to 2 500	35 to 75	60 to 130	6

Concentrations in mg/l except pH * *Durban-Roodepoort Deep* can be achieved with sulphate water for widely grown crops such as maize and lucerne . Much more salt (gypsum) is stored in the soil under irrigation with sulphate water compared to chloride water. It is thus not only feasible but also desirable to use lime-treated sulphate mine water for irrigation and to produce a crop without suffering soil physical problems, in order to minimize salt pollution of agricultural runoff water.

Soil structure and depth are important factors when it comes to the combined use of agricultural land to:

- produce crops without severe reduction in crop yield; and to
- act as a *salt sink* in order to prevent large quantities of salt, mainly gypsum, from reaching a water body.

The relationship of area irrigated and the amount of salt leached from soil under different leaching practices when the same volume of water and quantity of salt are applied (20 000 m^3/a; 38,6 t/a TDS) is shown in Table 23 for an idealized sulphate lime-treated mine water of the following composition:

pH	7,4	
conductivity	229	mS/m
TDS	1932	mg/*l*
SO_4	1200	"
Cl	177	"
Na	115	"
Ca	360	"
Mg	89	"

Decreasing amounts of salts are stored in the soil with increasing leaching fractions. Drainage water from the irrigation area will, however, be highly saline. To utilize agricultural land as a salt sink in a sensitive area, e.g. the Klip River catchment area draining into the Vaal Barrage, would require a high degree of irrigation efficiency and management skill. The potential benefits of reducing the salt load into the Vaal Barrage do warrant a full commitment to such a project which would best be started on pilot scale. Information on suitable land areas, soil properties and depths is obtainable from the Soil and Irrigation Institute in Pretoria.

6.5 Guidelines for water quality requirements for the industry (Wiechers *et al.*, 1989)

The quality of water currently in use in South African gold mines for condenser cooling, mine service water, potable water and water pumped to the surface in many instances does not meet the quality requirements appropriate to its use. In order to minimize working costs as well as to protect workers' health and meet existing and impending water quality legislation, improved water quality management is indicated.

Recognizing the importance of water quality to the gold mining industry, as well as its impact on the public water environment, COMRO through its Environmental Engineering Division, has launched a five-year programme on water treatment and quality management. At the end of 1993 a comprehensive text book as well as a user-friendly computerizer export system based on the result of this programme will be released to the mining industry. A computer programme AQUA-Q for use by mine water technologists for

Table 23: Relation between area irrigated and amount of salt leached from soil when the same volume (20 x 10^3 m^3/a) and salt load (38,6 t/a TDS) were used

Leaching Fraction applied	Irrigation applied	Area	TDS in mg/l		TDS in t/a		
	mm/a	ha	Applied	Leached	Applied	Leached	Stored
0,05	1 053	1,9	1 932	15 824	38,6	15,9	22,7
0,10	1 111	1,8		8 938		17,9	20,7
0,20	1 250	1,6		5 460		21,8	16,8
0,30	1 429	1,4		4 310		25,9	12,7
0,40	1 667	1,2		3 742		30,0	8,6
0,50	2 000	1,0		3 403		34,0	4,6

optimizing water quality management on existing mines is already available (COMRO, 1989a,b). This facility should be used by the mines.

In the Eastern Transvaal gold-field, seepage of arsenic from tailings might become a problem if the drainage water is not treated with lime.

On the Evander gold field, chlorides and fluorides are the main objectionable constituents of mine drainage. However, most of the saline water from the four mines is retained in the Leeupan for evaporation, with only occasional discharge during the rainy season into the Waterval River. Since the water from this river is used extensively for irrigation, the actual contribution of the Evander/Secunda mining and industrial complex to the salt load of the Vaal Dam is of a low order.

On the East Rand, dolomite is intercepted by the active Grootvlei mine on both sides of the Blesbokspruit and by the Sallies mine, west of the Blesbokspruit. Both mines are interlinked. Grootvlei has to pump out a total of 24 300 M*l*/a of water from both mines in order to prevent flooding of its underground workings.

The start-up of the new ERGO-Daggafontein operation in July 1987, utilizing effluent from ERGO-Brakpan, the reprocessing of the 8 km^2 tailings dam from ERGO-Brakpan for the recovery of gold, and the deposition of the reprocessed tailings at a more suitable adjacent site did not reduce the volume of water and salt load pumped out by Sallies via the Rietspruit into the Klip River. Of the total flow into the Blesbokspruit between Benoni and Nigel, only 22% is accounted for at Heidelberg. Water losses can be ascribed to:

- evaporation in the reedbeds which extend over 25 km;
- extensive use for irrigation; and
- seepage return to the dolomite.

There is a strong suspicion that the dolomite at Grootvlei is recharged with water probably from the Blesbokspruit. A thorough hydrological survey is indicated.

Since Marievale, south of the dolomite on the Blesbokspruit, is dry and has to import water, the question arises whether it would be useful to pipe the water pumped out from Grootvlei down to a point beyond Marievale in order to save costs for dewatering the Grootvlei. This should be investigated.

ERPM, the oldest active mine at the age of 94 years, is one of the deepest mines in the world, mining presently at 3 458 m below surface. With the expansion programme under way, the final depth in the South-East shaft area is expected to reach 4 250 m. In order to reduce the large volume requirement for low TDS water for cooling and to reduce pumping costs for returning hot water to the surface, ERPM is presently constructing a $2{,}19 \times 10^6$ t/a ice plant at the surface, which will become the largest ice plant in the world with the first module to be commissioned in September 1990. The volume of water sent down for cooling will thus ultimately be reduced to 2 200 M*l*/a as compared with 11 000 M*l*/a if chilled water only and no ice plant were to be used.

ERPM at Boksburg pumps about 9 800 M*l*/a of acidic water through its South West shaft from the defunct mines of the Central Witwatersrand in order to prevent flooding of its own workings. Another 7 200 Ml/a is pumped out from the Hercules shaft.

The contribution by three active gold mines, Grootvlei, ERPM and Durban Roodepoort Deep over a period of 16 months during 1988/89 to the flow and load of the Klip River has been as follows:

Flow	9%
TDS	48%
SO_4	60%.

The Klip River is the main contributor to the salt load entering the Vaal Barrage (Rand Water Board, 1988;1989).

Controlled irrigation with mining effluent, with $CaSO_4$ as its main constituent, can possibly utilize agricultural land as a *salt sink* without severe reduction in crop yield, thereby reducing the mineral pollution of the Vaal Barrage water.

Pollution deriving from residue dumps is regarded as of low importance in this particular area. Of the total Klip River catchment area of 1 968 km, only 39 km^2 or 2% is covered by sand dumps (6 km^2) and slimes dams (33 km^2). Of the total Suikerbosrand River catchment area of 3 587 km^2, only 34 km^2 or 0,9% is covered by residue dumps. All of these dumps were built with a water to solids ratio of about 1:1 and many of them are built on clay. When these dams were freshly built, leaching of sulphate from the outer crust of slimes dams took place, down to a depth of about 2 m below surface. As numerous slimes dams are now being re-worked for the recovery of gold and pyrite ample analytical data are available which indicate that below 2 m no oxidation of sulphur took place. Consequently, the pollution potential of slimes dams is regarded as being low.

Sand dumps, of which the last ones were deposited in the early 1960s, have already lost about 25% of the sulphur present, down to a depth of up to 10 m below surface.

It is concluded that at present the amount of sulphur in residue deposits that still oxidizes to cause water pollution is very low, particularly when compared with the pollution load deriving from mine pumpage and metallurgical plant operation.

Western Areas had been the mine with the highest pumping rate for dewatering, pumping out close to 60 000 Ml/a at its North Shaft and about 10 000 Ml/a at its South Shaft. Until June 1986 the bulk of this water was returned to the dolomite from where it had been abstracted. The mine has now been granted a permit to dewater the Gemsbokfontein compartment. By mid-1989 pumping from the North Shaft was reduced to about 28 000 Ml/a. The pumped-out dolomite water is disposed of at about equal rates via the Kleinwesrietspruit and Leeuspruit and reaches the Lochvaal via the Rietspruit.

Western Areas is the first mine where the radioactivity levels of mine effluents are measured by the Atomic Energy Corporation and the mine. These results are reported to the Department of Water Affairs. Analytical data indicate that the dolomite water entering the mine is contaminated with radioactive uranium. Water-soluble uranium has also been found in the Wesrietspruit and the Leeuspruit about 15 km downstream of mining property.

Most of the 14 active mines on the West Rand and the Far West Rand and the Orkney/Klerksdorp mines intercept the overlaying dolomite aquifer. Since 12 of these mines were at one stage operating uranium recovery plants, it can be assumed that radioactive contamination is present in the dolomitic mine service water and mine pumpage reaching the Wonderfonteinspruit as well as in adjacent boreholes. This could be established by a radiological survey.

To increase mine safety and lifespan, dewatering of the Venterspost and Oberholzer compartments commenced in the late 1950s and of the Bank compartment in 1970.

To compensate for damages incurred due to dewatering, such as drying-up of boreholes, subsidence and sinkholes, two bodies were formed and financed by the State and the Chamber of Mines viz.:

- The State Technical Coordinating Committee on Sinkholes and Subsidences on the Far West Rand and
- The Far West Rand Dolomitic Water Association.

The effects of mining and ore processing on the deterioration of the quality of underground and surface waters are evident at many locations in the dolomite area, particularly on the West Rand and the Orkney/Klerksdorp gold field.

Uncontaminated dolomitic aquifer water comprises a bicarbonate solution with calcium and magnesium as major cations, and is characteristically devoid of any sulphate. Borehole waters in the mining area, however, and particularly in the vicinity of uranium tailings, are often severely contaminated by sulphate (up to 1 400 mg/*l*), Mn (up to 300 mg/*l*), and Zn (up to 40 mg/*l*). The contamination of dolomite water with sulphate does not restrict its use for irrigation.

The effect of mine effluent discharged from the 13 West Rand and Far West Rand mines via the Wonderfonteinspruit on the mineral quality of the Mooi River before its confluence with the Vaal River is hardly noticeable.

Seepage of manganese from tailings dams via ground water into the Vaal River at Buffelsfontein seriously affected the quality of the water supplied by the Western Transvaal Regional Water Works to Orkney and Klerksdorp. Remedial measures taken such as pumping out polluted ground water and eliminating MnO_2 as oxidant in the acid leaching of uranium gradually improved the quality of the Vaal River water supply to the waterworks. Ozonation and chlorination at the waterworks further reduced the Mn concentration in the potable water supply to acceptable levels.

Pollution of dolomitic water and its seepage into the Vaal River takes place because of the surplus water from tailings and evaporation areas at the Vaal Reefs complex.

None of the Free State gold-mines disposes of their water into a public water body except into pans usually owned by the mines.

Fissure water from the Beatrix mine serves as a water supply to the southern Free State gold-fields mines, e.g. St Helena, Unisel, Pres Brand and Pres Steyn.

The closed-down uranium-producing Beisa mine has now been re-opened as the Oryx gold mine. However, this mine will have to be dewatered first. Radioactivity of this pumped out water will probably be higher than that of Beatrix mine water since the ore body of Beisa contains 600 g/t uranium compared with that of Beatrix which is less than 100 g/t. Whether or not any fissure water from these mines can be used as water supply without treatment to remove radioactive constituents should be established.

8 DISCUSSION

Of the total water intake to the mines, about 50% is low TDS river water, either extracted directly by the mines, or supplied by water board authorities which treat the water to potable standards. The other 50% of the water intake is mine water, pumped to the surface and either used by the mine itself or transferred to other mines. The quality of this mine water varies with location, ranging from good quality water from dolomite compartments at some Far West Rand and Klerksdorp mines, to acid, high sulphate and hard water on the Central Rand to high chloride and hard water at the Orange Free State goldfields. The most objectionable properties, in general, are corrosivity, tendency to scale formation, temperature and suspended solids.

Obtaining sufficient volumes of water is not a decisive cost factor for most mines. It is the quality of the water which affects the operating costs. Major cost factors are the replacement of eroded and corroded pipes and equipment, the increase in power use to compensate for the reduced heat transfer at refrigeration plants or for pumping through scaled up pipes, and the use of treatment chemicals.

The inadequate removal of suspended solids in underground settlers is regarded as the major water quality problem at some of the Evander, Witwatersrand and Carletonville mines. Most settlers cannot cope with the rapid and extreme fluctuations in pH, polyelectrolyte flocculant dosage and hydraulic and solids load rates.

The water required for cooling/refrigeration and for certain applications at the metallurgical plants must be soft and low in suspended solids, TDS and chloride. About 50% of the low TDS potable and river water supplied to the gold mines is used at the hostels, offices and houses for the around 500 000 persons working and living on mining properties. With the generous intake at mine hostels of between 0,2 to 0,7 m^3/person.d - on average about 0,39 m^3/person.d - the sewage from mine hostels is low in TDS when compared with that of mine service water from most mines. A further 40% of the low TDS potable or river water is used for cooling/refrigeration and about 10% at the metallurgical plants.

Costs to provide low TDS potable water from water boards to the mines are high because of pumping over long distances. The cost figures for 1990 are as follows:

Rand Water Board including delivery to the mines (Far West Rand):	52,75 c/m^3
OFS Goldfields Water Board water delivery to the mines (Welkom):	79 c/m^3

To augment the supply of low TDS water to the mines, a few alternative routes may be chosen to upgrade or to recover locally available low TDS water. Since pumping costs incurred in delivering potable board water over long distances fall away, the schemes outlined below have an inherent cost advantage.

Scheme 1

Upgrading of the large source of low TDS sewage water presently appears to be one of the cheapest ways of augmenting on a large scale the supply of good quality water or of alternatively saving potable or river water.

Much of the treated sewage on mines is currently used for applications such as the watering of lawns and golf courses. This appears to be a waste of low TDS water, which could be used with benefit for e.g. cooling/ refrigeration and for hydro-powering of rockdrills. Only a minor proportion of the 0,2 to 0,7 m^3/person.d is required for the preparation of food and drink. Therefore, another proposal for optimizing the use of low TDS water could be:

Scheme 2

(i) Supplying low TDS water only to the kitchen and brewery of a hostel and using mine service water for the showers and washing.

(ii) Supplying a corresponding volume of low TDS water to where it is needed e.g. for cooling/refrigeration, hydro-power or for limited applications at the metallurgical plants.

(iii) Using treated hostel sewage effluent for watering of lawns and golf courses.

Such a scheme has the disadvantage that two different supply lines will have to be provided for a hostel. Furthermore, the mine service water used for washing and showering has to be disinfected. The advantage is that the need for upgrading sewage effluent to potable water standards will be eliminated.

Scheme 3

Supplementing low TDS water supply with low TDS dolomitic water as already practiced with success at Driefontein Consolidated. This solution appears to be logical for the Far West Rand mines which, in spite of having to pump out large volumes of low TDS water from the mines, still have to buy Rand Water Board water which has to be supplied from a source about 90 km away.

Scheme 4

Desalinating mine service water by reverse osmosis, electrodialysis or freeze desalination. Membrane processes have been tested on pilot-plant scale and have been found workable for certain types of mine water. Costs are, however, a factor of constraint and other ways of supplying low TDS water may be more suitable.

Freeze desalination has no commercial advantage over other established methods for producing desalinated water (membrane processes). In conjunction with mine cooling by slurry ice, however, freeze desalination appears to be economically more viable. A 1 000 t/d freezing plant for producing desalinated slurry ice is to be constructed at the Vaal Reefs No 9 shaft, and results should be available during 1991.

Scheme 5

Recovering moisture from ventilation upcast air. Fan drift condensate from ventilation upcast air is generally of the same or better quality than water supplied to the OFS mines by the Goldfields Water Board from the Vaal River via a 90 km pipeline. The feasibility of recovering low TDS soft water from the moisture contained in ventilation upcast air warrants investigation as an alternative to other desalination techniques such as reverse osmosis and electrodialysis.

A survey into available low TDS water supply and low TDS water demand is indicated for each mine, as well as a feasibility study into the practicability and economics of each of these schemes.

The cost of environmental control (i.e. to keep the wet-bulb temperature at stope face at or below 28 °C) and to provide a sufficient volume of dust-free air, in a modern mine may constitute as much as one-third of the total costs for mining and may rise to about half on some deep mines.

Presently, about 40% of all the low TDS water used on mines is used for cooling/refrigeration. Since the total cooling/refrigeration capacity installed has risen from 255 MW in 1974 to 1 213 MW in 1989 and will increase at an average of 100 to 200 MW/a, the demand for low TDS water for cooling can be expected to increase proportionally unless remedial measures are taken to reduce the demand for cooling. Numerous methods have been and are being developed by the mines, such as reduction of the heat load originating mainly from the rock by backfilling of worked-out stopes, thermal insulation, the use of Pelton energy recovery turbines, or by controlled air circulation. Sending down ice provides the same cooling duty as a fivefold volume of chilled water, with concomitant saving in the cost of returning the hot water to the surface. The gold and uranium mines consume 19% of the total electricity sold by Eskom (1989). Electrical power generation at power stations results in huge losses of water, mainly by evaporation, e.g. 55 000 Ml/a for a 3 600 MW station, or 1,75 m^3/ MWh. Methods to save electricity will, therefore, reduce evaporation losses of good quality water at these stations and hence result in water savings for the region as a whole.

Electrical power is costly, particularly during peak demand periods. Electricity saving methods are, therefore, introduced at mines on a growing scale.

The Pelton wheel turbine recovers up to 67% of the down-going potential energy and is already widely used to reduce the energy required to pump back warm service water to the surface.

The introduction of hydro-power to the stopes, as developed over the past year by COMRO in cooperation with the industry, offers large savings in electrical energy by using the same chilled water to cool the underground environment and by using hydrostatic pressure to operate a wide range of energy-consuming stoping machinery and equipment. At insufficient mining depth or where the mining layout forbids the installation of high-pressure water supply, electrically driven pumps increase the water pressure to the desired level. The electrical energy consumed for conventional pneumatic stope drilling is at least 15 kWh/t mined, for electro-hydraulic drilling with emulsion only 1,5 kWh/t, whereas for hydro-powered drilling with plain water no electrical energy input is needed.

Presently available hydraulic rockdrills, already being introduced by the mines on a large scale, require 2% additives to be emulsified in water to control wear and corrosion. For these emulsions, additives have been developed with high hardwater stability.

Because of their many practical benefits, rockdrills powered by clear water have been developed and successfully tested. Wear-resistant materials for the highly stressed internal components had to be developed. Water-powered rockdrills are expected to be available commercially by 1992.

For both emulsion-powered and clear-water-powered rockdrills, water quality is no longer critical, with the exception of low tolerance for suspended solids (5 mg/*l* max.)

Because of the reduced labour costs - drilling rates are obtained of at least double those obtained with pneumatic rockdrills,- and reduced electricity input, the operating costs of hydro-power systems are lower, whereas the initial capital outlay is about the same as for pneumatic systems. The cost benefits and proven reliability of hydro-power are the reasons for many mines electing to install hydro-power systems at an increasing scale, particularly at new extensions.

Non-explosive mining, based on a hydraulically operated (clear water) impact ripper, is being developed by COMRO and is undergoing trial runs. With this low energy rockbreaking method, less dust will be produced, and loss of gold with the fines is expected to be reduced. In the long term, introduction of this hydro-powered rockbreaking system is expected to revolutionize underground mining by changing from the present one-shift working system to a staggered shifts-around-the-clock working system resulting in a more even load on transport and hoisting facilities. Consequently, the demand for low TDS and suspended solids-free water underground might increase.

The removal of suspended solids from mine service water with underground settlers is subject to extreme fluctuations in inflow volume, pollution with oil spillage, absence of flow measurements and lack of control of flocculant dosing. It is difficult to obtain a water with less than 20 mg/*l* suspended solids in the treated water. Even the high rate tube settler, recently modified and tested underground by COMRO, effectively clarifies inflows at widely fluctuating loads only when the pH and flocculant dosing rates are properly contolled.

The introduction of the three-chamber hydro-lift system for sending slurry ice underground and for returning dirty run-of mine water to the surface without the necessity for prior underground settling and clarification would enable mines to carry out the entire process of removing suspended solids above ground. This would largely eliminate capital and maintenance costs for underground settlers, clear water sumps, clear water pump chambers, mud water pump chambers and the like. Obviously, at surface installations clarification of dirty mine water could be achieved with far greater efficiency than underground. The major benefit of the hydrolift system, is the expected large reduction in the power requirement for returning warm water to the surface - less than 10% as compared with pumping.

The corrosivity of mine water is influenced by the presence of dissolved oxygen. Therefore, an investigation into the practicability and benefits of de-aerating mine service water for reducing the level of dissolved oxygen is probably worthwhile.

About 40% of the total water intake to the mines is used at the metallurgical plants to process about 120×10^6 t/a of ore and about 90×10^6 t/a of sand and slimes residues for the recovery of gold, uranium, pyrite and sulphuric acid. Where gold only is recovered and tailings are deposited with a solids to water ratio of about 1:1, the potential to pollute underground or surface water is low. The pollution potential, however, of tailings from the combined recovery of gold, uranium by solvent extraction, and of pyrite by flotation is high, because of the resulting water to solids ratio exceeding 3:1 and due to the conversion of insoluble pyrite into sulphuric acid used for uranium leaching. Sulphur in the tailings therefore, is present as water-soluble $FeSO_4$, $MnSO_4$ and $CaSO_4$. Pollution of surface streams and ground water is now a real problem. This is aggravated by the fact that surplus water is decanted from the tailings to be evaporated in unlined evaporation ponds not protected against seepage losses. Since nearly all uranium producers are situated on dolomite the large underground water reserves may become contaminated by constituents such as sulphates, soluble metals or radioactive material.

An uncontaminated dolomite aquifer comprises a bicarbonate solution of calcium and magnesium, and is characteristically devoid of sulphate. However, high values of sulphate exceeding 1 000 mg/*l* are found in boreholes in the vicinity of such tailings deposits. At the Turffontein eye, west of the Far West Rand mines, the sulphate concentration is around 150 mg/*l*, compared to 0 to 2 mg/*l* at the Maloney's and Schoonspruit eyes in dolomite not affected by mining. Pollution of underground water by $MnS0_4$ and of Vaal River water by seepage through the dolomite as at Buffelsfontein, renders such polluted water unfit for drinking, unless costly treatment (ozonation) is applied.

To protect the quality of the underground water its seepage into underground strata must be prevented by measures such as:
- lining of the foundation of tailings dams and of evaporation ponds in sensitive areas by an impervious layer (clay, butyl rubber, etc.); and
- utilization of tailings return water at the metallurgical plant or elsewhere.

A geological survey will be essential to determine if and where tailings deposits have to be sealed against seepage. The alternative is to close the water circuit at the metallurgical plants for the combined recovery of gold, of uranium by solvent extraction and of pyrite by flotation. Here, the real trouble-maker is the raffinate from the solvent extraction, which contains residual solvent, which in turn adversely affects the flotation recovery of pyrite. Due to the presence of high concentrations of Fe, Al and SiO_2, neutralization of the acid raffinate with lime results in the formation of a jelly-like hydrated precipitate which is difficult to dewater.

Neutralization of the raffinate with simultaneous removal of at least part of the residual solvent could probably be achieved by the Bethlehem Steel high-density sludge process (Fig 48), employing intense aeration during neutralization with lime. The conventional lime neutralization process is shown in Fig 49.

The anaerobic conversion of sulphates into H_2S, using molasses or generator gas as carbon source, could probably be employed to precipitate the heavy metals present in the raffinate and so eliminate the problem encountered with dewatering the jelly-like precipitate. The possible benefits from closing the water circuit and reducing the water to solids ratio at metallurgical plants recovering gold, uranium and pyrite would be mainly in the form of reduced pumping cost, reduced costs for disposal of the wet slime and protection of underground water.

Management options to reduce the volume of waste water to be disposed of by gold/uranium metallurgical plants should be thoroughly investigated.

With the increasing trend to dispose of slimes as backfill underground, neutralized raffinate which does not meet the quality demand for metallurgical processes could probably be utilized.

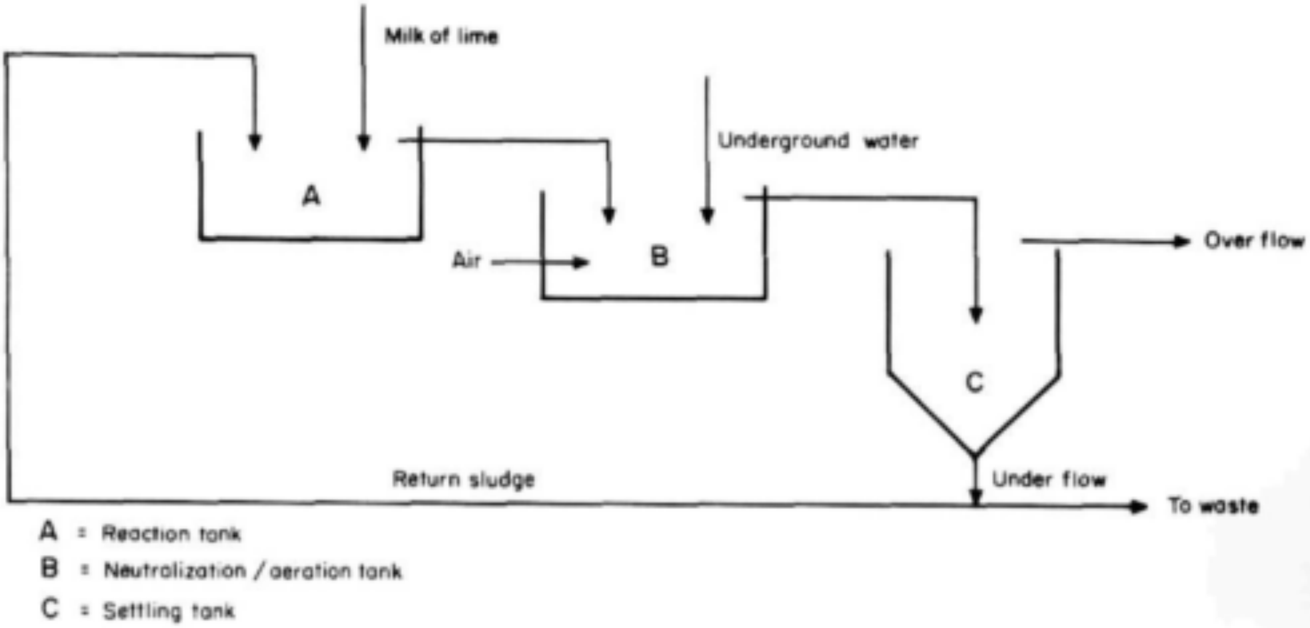

Figure 48
High-density sludge neutralization/aeration process

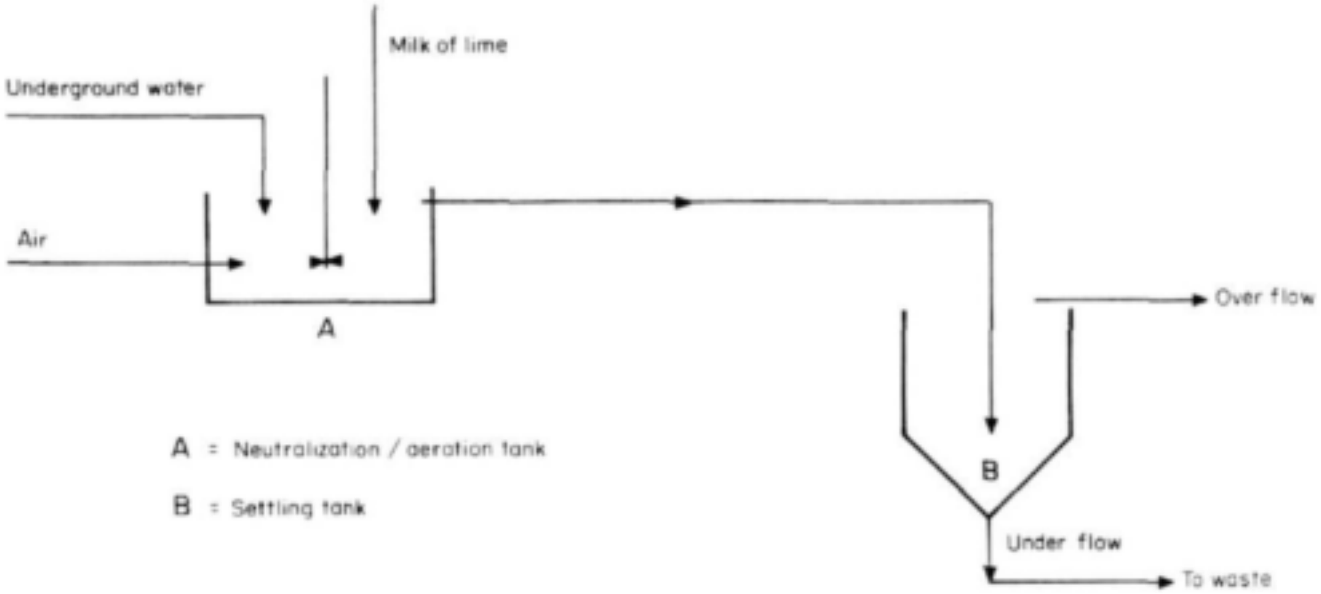

Figure 49
Conventional lime neutralization

During 1985 South Africa was the second largest producer of uranium, Namibia excluded (Atomic Energy Corporation of SA, 1986). Yet, no system had been established to limit and control exposure to radiation arising from the radioactive contamination of underground and surface waters by either mining and processing or by its occurring naturally. High values for radionuclides have been found in mine service water, in water from a dolomite compartment adjacent to a mine and in effluents leaving mine properties. While ^{226}Ra is readily adsorbed onto particulate matter such as sand, clay or sewage sludge, uranium stays in solution and can be transported throughout the mine.

The extend of the exposure of miners to radioactive materials in gold and uranium mines should be measured and countermeasures taken, if necessary. The extend of the exposure of the public living nearby tailings dumps to the effects of radon gas and its radioactive daughter products also warrants investigation.

The Council for Nuclear Safety established in 1988, has, in consultation with the Government Mining Engineer, undertaken to establish the acceptable risk level *inter alia* for the mining and processing of uranium - and thorium - bearing ores and for the burning of pyrites in sulphuric acid plants.

The policy adopted for the control of exposure to radiation must be based on limiting the risk level in terms of effective dose equivalent, a concept which takes into account all the various modes and time scale of exposure.

Preparation of the necessary standards has been carried out to enable the affected facilities to be licensed. An investigation of the problems associated with effluent discharge from mining operations in cooperation with the Department of Water Affairs is under way. Results of the investigation to date indicate a definite need for control.

Since many communities obtain their drinking water from underground sources of which many may intercept uraniferous strata e.g. near Warmbaths (Tvl), in the north-western Cape or in the Karoo (Beaufort West) such a radiological survey should also include underground waters in these areas. Some of the coal reserves, as yet not exploited, contain high concentrations of uranium e.g. those in the Springbok Flats.

As far as the treatment of affected water to remove radium, uranium or other radionuclides is concerned, several processes are mentioned, namely adsorption onto slimes, coagulation/flocculation, lime softening, anion exchange, reverse osmosis and electrodialysis. The effectiveness and economy of these treatment routes should be investigated, including the disposal of the resultant sludge or brine.

The three active mines on the Witwatersrand contribute about one-third of the total salt load entering the Vaal Barrage via the Klip River. Both the ERPM and Grootvlei mines are forced to dewater adjacent abandoned mines in order to continue mining and can do little to reduce the pumping rates.

Since the main constituents of the salt load originating from these mines are calcium and sulphate, irrigation with minimum leaching should be considered as a possible method of using selected agricultural land as a *salt sink*. High yields can be achieved for crops such as lucerne and maize without suffering physical soil problems. Such a practice would require a high degree of management input and skill. The potential benefits of reducing the salt load to the Vaal Barrage justify a full commitment to such a project which would best be carried out on pilot scale.

Since the mines are forced to rationalize and cut costs for labour, energy and replacement of equipment, it is essential to recognise the role of sound water management and its effect on working costs.

The impact on the environment by the various process routes followed to recover gold, uranium, pyrite and sulphuric acid must be understood. Unfortunately there is often a lack of information at mine management level regarding the impact of unsound water management. This is aggravated by the fact that responsibilities for water and effluent treatment and converging facilities are frequently shifted from one mining department to another.

Already now COMRO, through its Environmental Engineering Division is providing advisory services to the mines. A number of consultants have acquired a great deal of expertise in the field of mine water management and tailings disposal. To employ impartial outside consultants will be helpful in alleviating the workload of mine management staff.

9 SUMMARY

1 The 40 larger active South African gold and uranium mines annually dispose of approximately 120×10^6 t of milled and processed ore and in addition about 30×10^6 t of waste rock (1989). The 36 small mines in the eastern Transvaal mill and process about $1{,}1 \times 10^6$ t/a of ore. Another about 90×10^6 t/a of low-grade sand dumps and slimes dams are reprocessed for the recovery of gold, uranium and pyrite and the tailings are deposited as slimes. In total, the gold and uranium mines consume 19% of the electricity sold by Eskom.

2 Of the total water intake about,

- 50% is used at metallurgical plants;
- 27% at hostels, offices, and houses;
- 21% underground; and
- 2% for power.

3 During 1983, the low TDS potable water and river water were used as follows:

- 50% for domestic supply, hostels, offices, houses;
- 40% for cooling/refrigeration, chilled mine service water supply; and
- 10% for metallurgical plants and gardens.

4 Low quality (high TDS) water is mainly used for:

- dust suppression
- stope cleaning
- fire fighting
- ore washing and milling
- gold leaching and processing
- pyrite flotation
- sulphuric acid production
- uranium leaching
- disposal of tailings.

Mine service water is disinfected in order to destroy pathogens and to render it *safe to drink*. The presence of ammonia from blasting and of suspended solids increases the chlorine demand.

5 During 1983 the water emanating from the gold and uranium mines was disposed of or lost as follows:

- to surface streams or returned to dolomite compartments:	132 700 M*l*/a = 39%
- lost by evaporative cooling or with ventilation air:	34 000 M*l*/a = 10%
- to tailings and evaporation dams:	170 000 M*l*/a = 51%
	336 700 M*l*/a

6 All reefs mined for gold contain sulphur in varying amounts ranging from 1% to pockets of 9% S, mainly as pyrite FeS_2 but also as arsenopyrite. Consequently, the wet and oxidative conditions underground are conducive to transformation of pyrite sulphur into sulphuric acid.

7 Mine service water, due to the leaching of ore may contain as much as 11 000 mg/*l* TDS. While SO_4 is the predominant contaminant of mine service water in most mining areas, high concentrations of Na and of Cl are the main constituents in the under ground water of the Evander and OFS gold-fields.

8 Corrosion and blockage by scale formation pose a severe problem particularly in the OFS gold-mines. Dissolved oxygen is apparently a major factor of corrosion by mine water with a pH above 5. Chloride is regarded as the main cause of the pitting corrosion of stainless steel.

9 All mines, operating at depths between 800 m and nearly 4 000 m and at virgin rock temperatures of 30°C to 60°C, require cooling to maintain a maximum stope wet-bulb temperature of 28°C. At depths of about 1 200 m and sometimes even deeper, ventilation air alone can remove all the heat. At greater depths, chilled water produced at the surface or at underground refrigeration plants is used for distributing cooling.

10 Water underground is recirculated at a rate of 2 m^3/t to 15 m^3/t of rock broken and air is supplied at a rate of about 10 t/t of rock mined.

11 The total cooling/refrigeration capacity installed at the large gold mines increased from 255 MW in 1974 to 1 213 MW at the end of 1989. As the mines get deeper, the cooling/refrigeration capacity is expected to increase at an annual average rate of about 100 to 200 MW.

12 Water losses by cooling/refrigeration plants currently are:

21 100 M*l*/a	for surface installations
22 900 M*l*/a	with ventilation upcast air
44 000 M*l*/a	

which gives a specific water loss of 0,29 m^3/t of rock mined.

The total water lost by all the gold mines through evaporative cooling, wind losses and blow-down and carried out of the mine by ventilation upcast air is less than that lost by one single 3 600 MW power station on the Transvaal highveld (55 000 Ml/a).

13 The following methods may be used to reduce the intake volume of low TDS water as make-up for cooling and to conserve energy:

- operate the surface mine service water pre-cooling towers during the night and early morning and store the cooled water in large storage dams in order to reduce the cooling duty of the refrigeration plant;
- recover the potential energy by Pelton wheel turbine, thereby reducing the heat gain of down-going chilled water (2,3°C/km);
- insulate all chilled water pipes to reduce cooling losses at unwanted areas;
- recirculate underground ventilation air, thus eliminating the heat gain by downcast air of 4°C/1 000 m;
- insulate rock surfaces of intake airways; and
- reduce the heat load originating from rock surfaces by backfilling of worked-out stopes.

14 The method of backfilling worked-out stopes with conditioned slimes appears to be an effective method of decreasing heat load and reducing cooling requirements in a mine, while simultaneously increasing mine safety. At stope face the heat flow could be reduced by up to 50% and mine-wide refrigeration requirements could typically be reduced by between 10% and 25%. During 1989, 18 mines were using $3,6 \times 10^6$ t of tailings for backfill, equal to only about 3% of the ore milled. It is expected that the use of backfill will increase to about 15×10^6 t/a within the next ten years.

15 New methods of cooling mines with less energy input which are already in use or showing promise of being economically viable are:

- Sending ice down instead of chilled water reduces the volume of the resultant hot mine service water to one-fifth, thereby reducing pumping costs for returning it to the surface.
- The closed-circuit three-chamber mine cooling system is being considered for installation at two local gold mines and one platinum mine. It offers the following advantages:

  Through eliminating the Joules Thompson heating effect of the down-going cooling medium, energy is conserved for cooling purposes.

  Since the pressure in the rising main carrying the dirty mine water, is effectively balanced by the down-going chilled clean water or slurry ice, pumping power to overcome friction is minimal, as compared with conventional pump systems which return hot mine service water to the surface.

Since dirty hot mine water can be lifted to the surface, the entire process of removing suspended solids can take place above ground, thereby eliminating the ineffective settling/solids removal underground.

16 The use of hydro-power to replace compressed air for rockdrilling results in significant savings of electrical energy during the peak demand period; because of the doubling of the drilling rate, the cost of labour is reduced as well. For the currently exclusively electro-hydraulically powered emulsion rockdrills the electrical energy consumption averages 1,5 kWh/t mined compared to at least 15 kWh/t for compressed air rockdrills. The use of plain water for powering rockdrills at depths exceeding 1 500 m has been successfully tested, and such rockdrills will be commercially available by 1992.

Clear water, free of abrasive solids, is already used for stope cleaning jets, for stope air/water entrainment coolers and for Pelton wheel motors driving winches for rock scrapers.

Non-explosive mining, based on a hydraulically (clear water) operated impact ripper, will revolutionize underground mining by changing it from the present one-shift working system to a staggered shifts-around-the-clock working system with better utilization of man-power, machinery and hoisting facilities. Consequently, the demand for chilled water which is non-scaling and free of suspended solids for underground mining will increase.

17 Corrosivity of underground mine water appears to be a function of conductivity, dissolved ions, mainly chlorides, and dissolved oxygen. The replacement of corrosion-sensitive mild steel pipes by e.g. epoxy or glass fibre-coated mild steel or by plastic corrosion-resistant pipes might be cost-effective in spite of higher capital outlay. Installation costs constitute the far higher cost factor, accounting for about 75% of the total expenditure for a piping network underground.

The high chloride content of OFS mine water precludes its reuse as scrubber/cooling water at H_2SO_4 plants because of its aggressiveness towards stainless steel and so necessitates the use of water with a low TDS and chloride content, e.g. treated sewage.

18 To augment the supply of water which is low in TDS, non-scaling and free of abrasive solids without increased abstraction of board or river water, the following possibilities exist -

- upgrading of treated sewage effluent from hostels;
- supplementing with low TDS dolomitic water;
- desalination of high TDS mine water by membrane processes or freeze desalination; and

- recovery of moisture from ventilation upcast air.

19 The generous use of low TDS water at mine hostels of between 0,2 to 0,7 m^3/person.d (average about 0,39 m^3/person.d), results in a sewage effluent which is low in TDS compared with mine service water. The upgrading of low TDS treated sewage from mine hostels by dissolved air flotation and chlorination for use underground, as already practiced at Kloof mine, improves the quality of mine service water thereby reducing corrosion, scale formation and costs for water treatment chemicals.

20 Desalination of high TDS mine water by reverse osmosis or electrodialysis has not yet been introduced on a large scale, but has been demonstrated on pilot scale to be technically feasible for non-scaling NaCl-type OFS mine water.

21 Comparative cost figures to provide low TDS water from various sources during for 1990 are as follows:

	c/m^3	
Potable water (including pumping to mines)		
Rand Water Board:	52,75	
OFS Goldfields Water Board:	79	
Upgrading of treated sewage:	12	operating costs only
Augmenting with low TDS dolomite water:	12	"
Desalination by RO or EDR:	50	"

22 The introduction of the CSIR (DWT)/Kloof process to replace or augment potable or river water by upgrading hostel sewage water should be seriously considered, particularly at mines with service water which is scale-forming and of high TDS.

23 Desalination of mine service water with 5 600 mg/*l* TDS and 1 360 mg/*l* Cl by commercial short-cycle freezing (11 min - Harmony-type tube ice-maker) is not feasible. The resulting dry ice is of insufficient brittleness during harvesting and contains about 70% of its original chloride.

Slurry ice-making at a slow freezing rate holds the potential of producing desalinated ice from mine water, regardless of the salinity and composition of the feed water. The potential for increased cooling capacity by utilizing the latent heat of melting ice combined with the potential for desalination can make freeze desalination attractive for mining. The first two full-scale slurry ice plants for bulk air cooling, employing different processes for freeze desalination, are presently being installed and tested at two mines in the Western Transvaal.

24 The fan drift condensate already recovered from ventilation upcast air is of the same or better quality than water supplied to the Free State mines via a pipeline of about 90 km by the OFS Goldfields Water Board from the Vaal River at Balkfontein near Bothaville. The recovery of good quality low TDS soft water from the moisture contained in ventilation upcast air might, for certain shafts, be an alternative to other desalination techniques.

25 Water consumption figures at surface plants are:

Gold plant alone to tailings:	0,9 to 1,4 m^3/t of processed ore
Gold and uranium plant residue:	2,0 m^3/t of processed ore
Combined gold, uranium and pyrite flotation plants to tailings:	4,0 to 4,5 m^3/t of processed ore
H_2SO_4 plant, scrubber/cooling water in circulation and lost: (partly disposed of with pyrite flotation tailings)	11,0 to 14,0 m^3/t of acid

26 Uranium extraction with solvent, after oxidative acid leaching, results in a highly mineralized, solvent contaminated *raffinate* of about 1,1 to 2 m^3/t of tailings processed. Reuse of this raffinate in metallurgical processes is hampered by:

- the high solvent content of about 250 to 600 mg/*l* which adversely affects flotation of pyrite and the recovery of gold by the carbon-in-pulp process; and
- the high content of iron and aluminium in solution which after neutralization with lime results in a jelly-like hydroxide precipitate which is difficult to clarify.

Since the annual costs of replacing solvent losses are between R1 and 2 x 10^6 at most uranium extraction plants, research into the reduction of these losses is advisable. Possibilities of reusing *untreated* raffinate are:

- recirculation as make-up for the counter-current leaching of ore with sulphuric acid;
- use as scrubber/cooling water in the Peabody scrubber of an H_2SO_4 plant, provided the chloride concentration is low; and
- use in the gold plant for acid cleaning of clogged filters (Chapter 3.6.4).

Possibilities of reusing *treated* raffinate are:

- use after neutralization for re-pulping slimes for disposal as backfill material; and
- use after neutralization/intense aeration as make-up water for monitoring of aged slimes prior to pyrite flotation.

Because of the difficulties encountered with the unrestricted reuse of raffinate and because of the convenient and easy access to low TDS potable or river water, most uranium-producing mines dispose of their tailings, after neutralization, with double or even 3 times the volume of water as compared to gold plants. Consequently, pumping costs to deliver the tailings to the deposit sites are higher.

27 Management options may exist to close the water circuit at metallurgical plants recovering gold, uranium and pyrite. A thorough investigation is essential to find methods for reducing the intake of fresh water, pumping costs and the pollution potential of the resulting tailings.

28 The following treatment processes already in use or under development might be suitable for upgrading acid raffinate for further reuse at the metallurgical plant:

- The high-density sludge process which is in use at ERPM and at two coal mines to neutralize acid mine water, may be suitable for the treatment of raffinate to strip off kerosene and to remove gelatinous metal hydroxides.

- Removal of sulphate from mining effluents and metallurgical plant streams by anaerobic conversion of SO_4 to H_2S followed by oxidation into elemental sulphur using molasses or generator gas as a carbon source may have the advantage of:

  removing heavy metals, e.g. complex metal cyanides or gel-like metal hydroxides from neutralized raffinate as metal sulphides, and to break down residual organic solvent;

  adding the metal sulphides as feed stock to the roasting stage of H_2SO_4 plants; and

  removing residual organic solvent at elevated temperatures in the oxidation stage to obtain a demineralized raffinate with a reduced level of solvent for reuse.

The possibilities of this process should be fully exploited.

29 Sand dumps, of which the last were deposited around 1960, have a high permeability rate, permitting oxidation of pyrite to depths exceeding 10 m. Of the total sulphur originally present, about 25% has been leached over periods longer than 25 years in the form of acidic ferrous and ferric sulphate.

30 Slimes dams which gradually came in use since 1918 are deposited hydraulically and consist of material milled to a fineness of about 75% below 75 μm, with consequent low permeability. Oxidation of pyrite is limited to the outer 2 m layer of the deposit. All tailings deposited hydraulically are neutralized with lime.

31 Gold plant tailings are deposited at a water to solids ratio of 1:1. Due to the large surface area and the potential evaporation loss on the Witwatersrand of 850 mm/a the net evaporation exceeds the volume of water available for evaporation. The potential for pollution of ground water by seepage is, therefore, limited particularly if gold slimes deposits have been built on impermeable soil.

32 Combined gold/uranium/pyrite flotation tailings are deposited at a water to solids ratio of 2:1 and even exceeding 3:1. The potential evaporation loss is now less than the surplus water available. Since stable tailings deposits cannot be built with that vast surplus of water, water is decanted and transferred to evaporation dams, which presently are not protected against seepage. At some plants raffinate is collected

and piped separately to an unsealed evaporation pond.

Since almost all gold/uranium/pyrite tailings are on dolomite and since pyrite has generally been removed for conversion into sulphuric acid used for uranium leaching, sulphur is now present as water soluble $CaSO_4$, $FeSO_4$, or $MnSO_4$. Ground water pollution by seepage, therefore, is a real problem.

33 The pollution potential of gold slimes deposits is low.

34 The pollution potential of combined gold/uranium/pyrite flotation slimes deposits is high as is shown by the manganese pollution of the Vaal River near Orkney and of boreholes between the Buffelsfontein gold/uranium slimes deposit and the Vaal River.

35 Uranium concentration levels in the Witwatersrand and Far West reefs are about 100 times lower than in the reefs mined for uranium in Canada, the USA or Australia and range from 50 g/t in the east to 200 g/t in the west and reach 640 g/t at Afrikander Lease near Klerksdorp, and 600 g/t at the Beisa mine at the southern end of the OFS goldfield.

36 The release of water-borne radioactivity, measured as ^{226}Ra, from old dry tailings deposits apparently does not pose a threat to surface waters accessible to the public, since ^{226}Ra is effectively retained, most probably by adsorption. In spite of the fact that water within the tailings impoundment can move a few hundred times faster than radium, it is still water that acts as the carrier. Seepage of ^{226}Ra into the dolomite water from the wet slimes deposits and particularly from unlined evaporative dams for the raffinate at the uranium-producing mines can, therefore, be suspected. A radiological survey to prove or disprove this suspicion is indicated.

37 Radioactive pollution of dolomite ground water can be suspected at some mines since levels as high as 12 Bq/*l* for ^{226}Ra have been found in the underground mine water at one mine. Since at one stage, around 1960, 26 mines were feeding 17 uranium recovery plants with ore, it can be assumed that radiotoxic contamination of mine water is widespread, irrespective of whether uranium is presently recovered.

38 While ^{226}Ra is readily adsorbed onto particulate material such as sand clay or sewage sludge, and is thus removed from clear water, this is not the case with uranium which stays in solution. Uranium can, therefore, be transferred with service water through the mine, but also with mine effluents over wide distances.

39 The reuse of dolomitic water from the vicinity of mines should be subjected to approval following prior testing for radionuclides. High nuclide levels in the concentrated sludge or brine will require special methods of containment or disposal.

40 Standard water treatment processes such as coagulation/flocculation, sand filtration, lime softening, ion-exchange softening, and reverse osmosis effectively remove radium, and some even uranium. The high nuclide levels will require special methods of containment or disposal.

41 High values for radioactivity were found at one, now dismantled, sulphuric acid plant, apparently originating from the roasting of uraniferous pyrite. There are ten

active sulphuric acid plants at gold mines which are confronted with the same problem. Burial of the contaminated structural material (steel, bricks, ceramic Raschig rings, rubber lining, etc) has been accepted as the method of disposal.

42 The radioactive contamination level of metallurgical plant structures, be they in operation or moth-balled, should be determined before or during dismantling, to decide on a method of safe reuse or disposal.

43 Should emanation of radon gas from dry tailings deposits pose a threat, covering the deposits by compacted earth or by asphalt will most probably be an effective remedy. The extent to which radon gas and its daughter products may be present at uranium-bearing reefs and whether or not they constitute a health hazard to underground mine workers warrant investigation.

44 The reduction of underground acid formation by inhibition of acid-forming bacteria has not been successful since the uncontrolled inflow of fissure water has made it impossible to control the dosage of inhibitor.

45 Neutralized mine effluents from two mines contribute about one-third of the total salt load entering the Vaal Barrage via die Klip River, with ERPM adding the largest portion from any source. With controlled irrigation of mine effluent with $CaSO_4$ as its main constituent, and with minimum leaching, agricultural land can possibly be used as a *salt sink* without severe reduction in crop yield.

10 RECOMMENDATIONS

1 Maintain the adequate removal of suspended solids from mine service water by settling/filtration to provide water free of abrasive solids for low tolerance equipment such as hydro-powered rockdrills, valves, flow recorders and high-pressure clear water pumps.

2 Upgrade the locally available large source of low TDS treated sewage from mine hostels to provide low TDS water for use underground and elsewhere.

3 Stop the use of treated low TDS sewage water for watering golf courses and gardens, as this water could be used with greater benefit at the mine.

4 Augment imported low TDS water supply with low TDS dolomite water where such water is locally available and not contaminated by radioactivity.

5 Investigate the feasibility of recovering low TDS soft water from the moisture contained in the ventilation upcast air at certain Free State mines which have to pay the highest price for Vaal River water.

6 Because of the complex nature of corrosion, replace corrosion-sensitive mild steel pipes by epoxy-, glass-fibre- or polyurethane-lined mild steel pipes or by plastic material pipes, stainless steel, or hot dip galvanized steel, taking due cognisance of the possible disadvantages of these materials.

7 Intensify research to remove residual uranium extraction solvents and to obtain a clarified, neutralized, partly desalinated solution from raffinate, suitable for reuse. Processes to be investigated for the removal of gelatinous metal hydroxides and residual solvents should include the:

- high density sludge (HDS) process with its intense aeration, and
- sulphate removal by anaerobic conversion of SO_4 into H_2S with the possible precipitation of metal hydroxides as metal sulphides (feedstock to sulphuric acid production).

8 Develop management options to close the water circuit at metallurgical plants, particularly those recovering gold, uranium, pyrite and sulphuric acid, in order to reduce the pollution potential tailings with a resultant high water to solids ratio tailings and the consequent expense of sealing such tailings and evaporation dams against seepage.

9 Where a high water to solids ratio for tailings cannot be avoided, seal off such tailings against seepage to prevent pollution of the underground dolomitic water.

10 Carry out surveys on the radioactivity of:

- underground mine water in contact with uraniferous reefs;

- dolomite water in the vicinity of tailings dams, evaporation dams, or surface streams where radioactive contamination may possibly occur;
- pan water derived from Beatrix mine and from the dewatering of the Beisa mine; and
- selected tailings dams for radon and radon daughters.

11 Extend the radioactivity survey to water supplies from underground sources which are suspected of having contact with uraniferous reefs, e.g. at the Springbok Flats, the Northern Cape and the Karoo.

12 Develop treatment processes to remove radioactivity from underground mine water and from effluents leaving mine property or from other affected water bodies.

13 Develop limits (recommended maximum contaminant levels) to cover the radioactivity hazard for drinking water and for effluent discharges (General Standards).

14 Investigate the feasibility of reducing the salt load from neutralized mine effluents into the Vaal Barrage by irrigating with a low leaching fraction, utilizing agricultural land as a *salt sink*.

15 There is insufficient information at many mines regarding the volume, composition and flow pattern of mine service water and effluent streams because of the lack of metering devices, regular analyses and qualified staff - therefore -

- carry out a thorough investigation into mine water and effluent circuits and the optimum use of chemicals, e.g. flocculants, at underground settlers.

For this investigation and for the management of water systems -

- employ a qualified person as specialist water engineer, or
- employ an impartial outside consultant in order to alleviate the work load of mine management staff.

16 Avoid the shifting of responsibilities for installation, maintenance and alterations of water conveying reticulation systems from one department to another.

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