2.9.2 Patterns of Land Use

Much of the Namib Desert falls within conservation areas, for example the Namib-Naukluft and Skeleton Coast Parks. The game populations in these conservation areas is highly variable, with the animals migrating into and out of the area in response to seasonal rainfalls. The game population of the upper Kuiseb River was estimated in 1985 at 300 Kudu, 3200 Gemsbok, 2000 Hartmann's zebra and 1200 Springbok (Huntley, 1985). Those areas of the Central Namib Desert which have not been proclaimed as conservation areas usually have no surface water and little or no underground water available. Consequently, they are generally of very low agricultural potential and cannot support formal farming activities.

In some of the nearby communal areas of Damaraland, land-use is often of very low intensity, with the raising of cattle, goats, sheep, donkeys and some game ranching being the major activities. Further towards the interior, on the Great Escarpment and around the higher reaches of the major rivers, the vegetation is relatively well-developed due to the higher annual rainfalls received (Huntley, 1985). Reliable water supplies of moderate to good quality are obtained from boreholes and this area supports several large ranching concerns, which concentrate mostly on cattle, small- stock and game ranching (Huntley, 1985; Brown & Gubb, 1986). All of the farms located on the lower portion of the escarpment/desert transition are considered to be totally unsuited to any farming practice (Joubert *et al.*, 1976).

Nearer the coast, formal farming is undertaken on several smallholdings in the lower Swakop River. The major activity is concentrated on milk production to supply the needs of Swakopmund and Walvis Bay. Some vegetable produce is also produced here for the local market.

Towards the interior portion of the Central Namib Desert, informal farming was conducted along the courses of most of the rivers due to the presence of fodder (mainly *Acacia albida*) and sub-surface water supplies. Perhaps the greatest effect that this informal farming has had is the competition that domestic stock provide for the food and water resources used by wild animals (Department of Government Affairs, 1987). In 1977, small-holdings along the Swakop River were bought out and the land incorporated into the Namib Desert Park. However, informal farming is still continued along the rivers to the north of the Swakop. In addition, several groups of Topnaar Hottentots continue to raise goats, cattle and donkeys along the lower reaches of the Kuiseb River. During periods of drought, the domestic stock greatly reduces the available forage in the areas surrounding the Topnaar villages (Huntley, 1985).

Prior to the start of mining operations at Rössing, several small- to medium-scale prospecting and mining operations were located in the Central Namib region. These endeavours were focused mostly on the recovery of copper, tin and semi-precious stones (Department of Government Affairs, 1987). The Matchless Copper Mine in the headwaters of the Kuiseb River has an annual output of approximately 20000 tonnes of copper and pyrite concentrate from a reserve of 900000 tonnes with an average grade of 2.37 % copper and 15.01 % silver (Huntley, 1985).

2.10 Communications and Infrastructure

At the time of the investigation of the Rössing Uranium ore deposit by Rio Tinto South Africa in the period 1966 to 1973, communications were already good, with the rail and tarmac road from the port of Walvis Bay and Swakopmund to the interior passing within 12 km of the deposit (Berning, 1986). Several good, but untarred, roads linked the towns of Usakos and Karibib with farming areas to the north and south, while a similar gravel-surfaced road over the Khomas Hochland linked Windhoek

to the port at Walvis Bay. In the arid Central Namib Desert tracks were few and far between, but most areas were readily accessible with a four-wheel drive vehicle. The flat sand-floored tributaries of the Khan and Swakop rivers also served as ideal roadways for these vehicles.

Electricity supplies to Swakopmund and Walvis Bay are provided by the South West African Water and Electricity Corporation (SWAWEK), via overhead high tension powerlines. The electricity grid is now linked to the Eskom network in South Africa from which additional power can be drawn if needed. A modern telecommunications system links Swakopmund with other towns in the country, while remote areas are served by radio-telephone.

3. THE EXTENT OF CURRENT MINING OPERATIONS

3.1 Exploration History

The presence of radioactive minerals in the general Rössing area was first noted by Dr E. Reuning in 1910. He found that heliodore (golden beryl), in a pegmatite near Rössing Mountain, was distinctly radioactive and a subsequent analysis by Hauser and Herzfeld proved that it contained 0.02 to 0.04 % U_3O_8 (Berning, 1986).

In 1921, Dr P.A. Wagner described the minerals of the pegmatite near Rössing Mountain (aquamarine, heliodore and rose quartz), and expressed the opinion that the uranium content of the heliodore indicated that the magmatic solution from which the pegmatite crystallized was also radioactive (Wagner, 1921).

In 1928, Captain Peter Louw and his wife Margery conducted an auto-radiography test on a heavy black mineral that he had found some 20 km west of the present-day operations at Rössing. The mineral proved to be radioactive (Berning, 1986).

In 1932, Professor T.W. Gevers investigated the same area and found small lumps of a heavy black mineral resembling ilmenite strewn over an area of several square kilometres. He concluded that while the radioactivity of some samples could not be doubted, the bulk of the mineral was probably ilmenite (Gevers, 1936).

In 1940, two geologists of the Geological Survey of South Africa, C.M. Schwellnus and G.S.J. Kuschke, were given the task of investigating a reported occurrence of uranium near Rössing Mountain. They found the same black mineral scattered over a very large area extending from Rössing Mountain in the west to Salem in the east and southwards to the northern boundary of the Namib Park Game Reserve. However, only two samples out of a large number submitted for analysis, one from near Arandis and one from Salem, were found to contain uranium (Berning, 1986). The radioactive black mineral appeared to be a titanate of uranium and rare earth elements and was of no economic importance.

In 1953, the Louw Syndicate was formed with the objective of relocating Louw's original discovery. In 1954 the Syndicate, having found the original occurrence and having obtained a prospecting grant, entered into a prospecting and option agreement with the Anglo American Prospecting Company.

During the period 1955 to 1958, Anglo American's investigation located a number of radioactive locales (Smith, 1965), one of which proved to be important in that the uranium-bearing mineral was uraninite from which uranium could be recovered by conventional metallurgical processes. This occurrence, known as the SJ anomaly, was investigated by diamond drilling and by means of some underground development. The Company concluded that, although the area contained several million tonnes of low-grade uranium-bearing rock, the grade was too low to be economically viable and the options were dropped in 1958.

The Louw Syndicate offered this uranium prospect to Rio Tinto South Africa Limited on three different occasions between 1958 and 1966, but it was only in July 1966 that the Company finally concluded an agreement with this Syndicate (Berning, 1986).

3.2 Investigation of the Rössing Deposit

The investigation of the Rössing deposit by Rio Tinto South Africa commenced in September 1966 and was completed in March 1973 (Berning *et al.*, 1976). A variety of exploration techniques were used to determine the spread of uranium mineralization on surface and the grade and distribution to a vertical depth of 300 metres.

Airborne and ground radiometric surveys covering areas of 800 km^2 and 200 km^2, respectively were used to delineate the surface outline of the Rössing deposit. These were complemented with detailed topographical and geological mapping on a scale of 1:2000 over an area of 4 km^2 covering the Rössing deposit. The detailed geological map proved to be indispensable when interpreting the results of the subsequent diamond drilling programme.

After an initial reconnaissance of the near-surface uranium deposits by percussion drilling, a major programme of diamond drilling was initiated in May 1967 and completed in June 1971. A total of 177 inclined holes, with a total depth of 55000 m, were drilled. The first phase consisted of holes drilled at 120 m intervals, along section lines placed 120 m apart, to a depth of 150 m. In the second phase, the borehole spacing was reduced to 60 m along the same section lines, but their depths were increased to 300 m. The boreholes were inclined at an angle of 45° north (Berning, 1986).

Each drill core was split longitudinally, one half being kept for record purposes while the other half was used for analyses and metallurgical testwork. Half-core lengths used for analysis were crushed to pass a 6mm sieve, quartered and then pulverized to ensure that at least 50 % of the sample passed a 200 mesh sieve for radiometric analysis. All samples were analyzed for U_3O_8 content with a scintillation counter, and every tenth sample plus those showing a high U_3O_8 content were checked by chemical analysis. Tests conducted by the South African Atomic Energy Board (now NUCOR) showed that the radioactive material was in equilibrium.

The amount of sulphuric acid required for uranium extraction, plus that consumed by lime and pyrite in the gangue (or waste host rock), was determined for each sample. Routine beneficiation tests and mineralogical studies of borehole cores were started at an early stage in the exploration programme. This work was undertaken by the South African National Institute for Metallurgy (now MINTEK), and provided the basis for the design of a pilot plant.

A 3.65 m diameter shaft was sunk to a depth of 115 m from May 1970 and an underground bulk sampling programme at this level was completed in December 1971. This consisted of east-west drifts from which cross-cuts along selected surface drill sections were undertaken. Approximately 2750 m of underground development were completed. This was preceded by horizontal diamond drilling with the object of classifying the rocks intersected into grade categories. Each sample was crushed and split, one half being assigned to a particular stockpile and the other half passed through the fine crushing and sampling circuit. The fine crushing plant product was automatically sampled and analyzed for U_3O_8 and a calc index determined, while the remainder was sent to the appropriate stockpile. These calc index determinations provided an estimate of the sulphuric acid consumption by calcite dissolution.

This underground programme enabled comparisons to be made between tonnage and grade figures, derived from surface drilling and from underground drilling and development. The investigation indicated the existence of a very large, low- grade deposit of uranium that could be viable if mined and treated on a large scale.

Pilot plant metallurgical testing was conducted on material obtained from the underground bulk sample stockpiles from October 1970 to March 1972. This was designed to verify that the laboratory results were reproducible on a larger scale, establish the optimum uranium extraction process and assess the economics thereof, and to obtain metallurgical engineering criteria for a full-scale extraction plant.

A preliminary evaluation and analysis of the Rössing uranium deposit, based on production rates of 2500 and 5000 short tons of U_3O_8 per annum, was made in April 1969. This was revised in October 1969 when it became apparent that a major process change was necessary. Towards the end of the pilot plant programme a joint venture of internationally-known engineering construction firms was appointed to manage the engineering and construction of the main plant. The joint venture assisted in the final pilot plant work and confirmed that the ore body was amenable to extraction by low-cost open-pit methods and that a relatively simple extractive metallurgy plant could be constructed to recover U_3O_8 on a commercial scale.

The pilot plant test work indicated that the extraction process would involve:
1. conventional gyratory crushing, cone crushing and open-circuit rod mill grinding to a nominal -6 mesh product;
2. sulphuric acid leaching at approximately 40 °C followed by sand/slime splitting, washing of sand and slime, and liquid/solid separation;
3. concentration and purification by means of resin ion exchange, followed by solvent extraction treatment; and
4. precipitation by ammonia of ammonium diuranate, "yellow cake", followed by calcination in a roaster to yield the final product, a mixture of uranium oxides (Vernon, 1981; Berning, 1986).

3.3 Project Outline and Schedule

Rössing Uranium Limited was formed in 1970 to develop the uranium deposit and a decision to mine the ore body was taken in 1974 and open-pit mining started in 1976 (Vernon, 1987). Initial estimates indicated that the economic life of the mining operation would be approximately 20-25 years. However, subsequent evaluations have indicated that the uranium reserves at Rössing will be sufficient to permit uranium production at current rates to continue well into the second decade of the Twenty-First Century (Vernon, 1981). A continuous search for improved and lower cost mining and treatment methods should also extend the lifetime of the mine. At present, Rössing is the second largest producer of uranium in the world and processes the largest uranium ore tonnage by a very substantial margin (Vernon, 1987).

The Rössing Uranium Mine complex covers an area of approximately 4400 hectares and all mining and metallurgical procedures, from extraction of ore to purification and barrelling of the final product, uranium oxide (U_3O_8), are carried out on site (Vernon, 1981). The mine is operated 24 hours per day and 365 days per year. At full production, some 150000 tonnes of ore and waste rock are excavated on a daily basis from the open pit. Of this, approximately 40000 tonnes of low grade ore (average 0.035 % U_3O_8) are crushed, ground and acid leached per day. The waste material from the uranium extraction process (for practical purposes almost the entire mass of input ore plus waste liquids) is slurried to the tailings pile (Dames & Moore, 1984b).

The mine commenced operations in 1976 and the first few years of mining during the late 1970's were mainly concerned with achieving full production capability and rectifying initial design problems. Today, Rössing Uranium Mine is capable of producing in excess of 5000 metric tonnes of U_3O_8 per annum (Vernon, 1987).

3.3.1 Mining Plans

The Rössing ore body is unique in that it is the largest known deposit of uranium occurring in granite/pegmatite (Sandy, 1987). The ore reserves of the Rössing deposit were developed from data provided by the surface diamond drilling programme undertaken during 1967 to 1971. Using computer techniques, a series of long-term mining plans was developed from the ore reserve data until an optimum 20-year plan was obtained. This called for an open pit mining operation with 15 m high benches and an open pit which at the end of 20 years would be 3 km long, 1 km wide and 300 m deep (Berning, 1986).

A long-term 20-year mining plan was developed from data obtained from boreholes spaced on a grid of 60 m by 120 m. This spacing was too wide to plan actual mining and data obtained from infill drilling on a 20 m by 20 m grid and from blast holes on an 8 m by 8 m grid are used to develop medium- and short-term mining plans of the mining sequences designed to optimize the mining of the ore body. Mining schedules are produced for:

1. A three-year plan - updated every year;
2. A one-year plan - updated quarterly;
3. A three-monthly plan - updated monthly; and
4. A monthly plan - updated weekly.

In addition, an ore schedule is developed in conjunction with the production personnel and updated on a daily basis.

The controlling factor in formulating the mining plans is that sufficient ore at the required grade should be available for mining at any time in order to meet the plan's requirements. Associated with the ore production is the movement of low-grade material and waste rock in order that ore may be continually exposed. All material is mined in such a way as to maintain a logical progression of benches and roadways by which material may be moved to the required destination. Associated with the mining schedules, therefore, are various road designs, stockpile and waste-rock dump designs, and plans for electrical and water reticulation (Berning, 1986).

3.4 Open Pit Mining Operations

The features that make the Rössing operation unique are the scale of the mining operations and the process modifications that have been implemented in order to overcome the difficulties of mining low-grade uranium in a desert environment (Vernon, 1987).

3.4.1 Mine Area Layout

During the early development phase of the Rössing Uranium Mine, the project infrastructure included the following: temporary accommodation and associated facilities for the original construction workers; a double-lane tarred roadway from the main Swakopmund-Usakos road; a full gauge railway line linking the mine's final product plant with the main Windhoek - Usakos - Swakopmund - Walvis Bay railway line; water supply pipelines and storage reservoirs; a link to the SWAWEK 22kV powerline supplying electricity to Swakopmund and Walvis Bay; a biofilter sewage treatment works; an all-weather airstrip; storage facilities for diesoline fuel and explosives; construction of security facilities (including a fence around the mine workings), workshops, laboratories, as well as all personnel, medical and administration buildings. Additional gravel-surfaced (untarred) roads were also built to link the lower portions of Dome, Pinnacle and Panner Gorges with the central area of mine operations.

The layout of the Rössing Uranium Mine with the associated uranium extraction plant, tailings dam, waste rock dumps and infrastructure is shown in Figure 3.1; aerial views are shown in Plate 6.

3.4.2 Mining Method

The open-pit mining sequence at Rössing consists of a conventional drill, blast, load and haul operation (Vernon, 1987), though on a very large scale (Plate 5). The Rössing ore body is extremely complex in that:

- it is a close association between barren metasediments and uranium-bearing alaskite;
- the uranium content of the alaskite is extremely variable;
- in some areas alaskite is present as large masses, while in others it consists of narrow bodies interbanded with barren metasediments; and
- the acid consumption of the rocks varies greatly from low consumption for the alaskite to extremely high consumption for marbles.

Control of uranium grade and acid-consuming rocks is therefore of major importance, though difficult to achieve. The blasting technique used is of fundamental importance in achieving some measure of control (Berning, 1986). Further control is obtained through the use of a centralized scanner operating system which determines the ore grade carried by the haul trucks.

3.4.2.1 *Drilling*

The primary drill fleet consists of 5 Gardner Denver electric rotary blasthole drills (Sandy, 1987). Drill scheduling is, however, made difficult by the large variation in drilling conditions in the various rock types.

Drill patterns vary from 9 m by 10 m in very hard areas to 14 m by 14 m in the softer formations. Each blast hole is drilled in a single pass, 381 mm in diameter and 18 m deep, which includes 3 m of sub-drill. The extremely abrasive nature of the rock results in very high rates of drill wear; typically, drill rods last for between 15 000 and 18 000 metres (Burgess *et al.*, 1987; Sandy, 1987).

3.4.2.2 *Blasting*

On average, 0.3 kg of explosive is use for every tonne of material blasted, though this varies somewhat with the different rock types in the open pit. Bulk explosives are manufactured on site under license.

The explosive used is Heavy Ammonium Nitrate Fuel Oil (Heavy ANFO) which comprises two components: an oxidizer in the form of ammonium nitrate prill and a dense fuel, trade named HEF (High Energy Fuel). The sensitivity of the different explosives is extremely low and they require a heavy boost to initiate detonation. Local legislation requires that the boosters be positioned 1.5

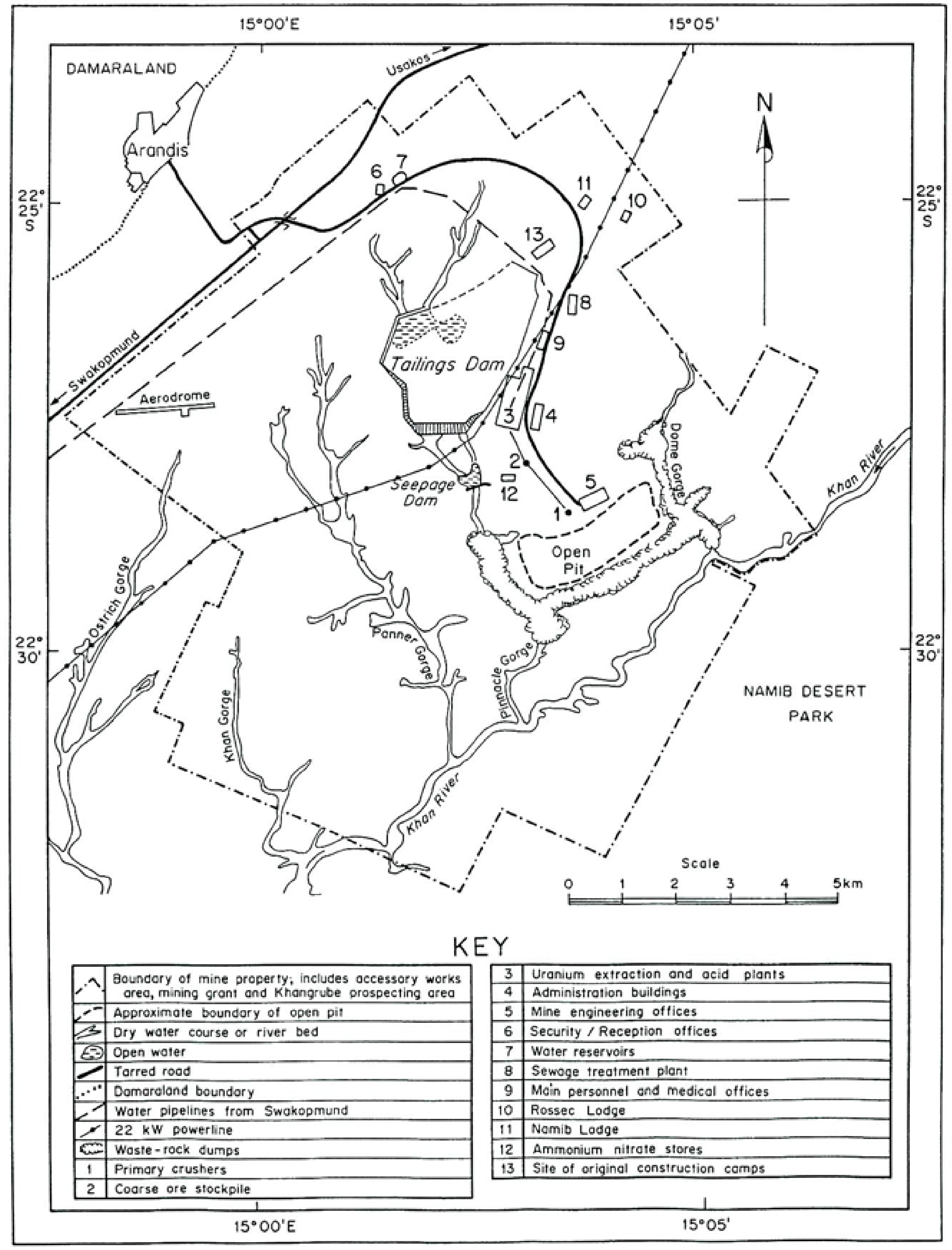

Figure 3.1: Diagrammatic map showing the layout of the Rössing Uranium Mine with its associated uranium extraction plant, tailings dam, waste rock dumps and infrastructure.

Plate 5

(A) Aerial view of the Rössing open pit from the north-west, showing the system of benches and the positions of haul roads and waste rock dumps.

(B) Blast in progress in the Rössing open pit.

(C) Haultruck being loaded with ore in the open pit, whilst water bowser sprays rock pile to reduce dust.

Plate 6

(A) Oblique, low-level view of the mine plant from Point Dave, with the hills along the Khan River visible in the background.

(B) Oblique aerial view of the Rössing tailings impoundment, prior to initiation of the paddock disposal system, showing large expanse of open water.

metres above the cut grade of the bench, almost in the centre of the explosive column (Sandy, 1987). Each blast hole is loaded with approximately 1600 kg of explosive, the type of explosive used in each blast hole is determined by the amount of water in the blast hole, its inflow rate and the rock strength (Sandy, 1987).

The blasts are scheduled in with shovel and crusher services as well as digging requirements. Normally, blasting takes place twice a week on Tuesdays and Thursdays, during the dayshift (Plate 5; Sandy, 1987).

3.4.2.3 *Loading*

Three types of shovels are used for loading ore in the open pit, namely: the P&H 2100 rope shovels, Marion 201 M-HR rope shovels and the Demag 485 hydraulic shovels.

The ageing fleet of P&H 2100 rope shovels were reduced from 8 to 3 and are used mainly for ore blending. Each P&H 2100 is fitted with an 11.4 m^3 bucket and a 13.4 m boom. The loading rate of this type of shovel depends on the rock type available and its degree of fragmentation. A 140 tonne capacity haultruck will take between 5 and 7 minutes to load, resulting in a production rate of between 1000 and 1500 tonnes per operating hour, taking truck efficiency into account (Sandy, 1987).

The P&H 2100 shovels have started to make way for two Marion 201 M-HR and one Demag 485 hydraulic shovels which have bucket volumes of 18 m^3 and 22 m^3, and boom lengths of 15.7 m and 15 m, respectively. The typical loading time for a 140 tonne haultruck is 2 minutes and the truck is fully loaded in 3 to 4 passes. The production rate of these two types of shovels is between 1500 and 3000 tonnes per operating hour, which is a great improvement on the P&H 2100 shovels.

The difficult digging conditions and abrasive nature of the ore at Rössing result in a high rate of wear of all ground engaging components and in a reduced life of consumables such as hoist ropes, tips and buckets (Plate 5; Sandy, 1987).

3.4.2.4 *Hauling*

The present haultruck fleet at Rössing comprises 19 Wabco and 11 Euclid trucks, each with a capacity of 140 metric tonnes. In 1986, an electrified trolley assist system was introduced to the pit when 2.4 km of overhead trolley line was erected from bench 15 to the primary crushers on an 8 % ramp (Sandy, 1987). Since that time a number of changes have taken place.

In 1990, trolley line 5 was moved across to the south-eastern side of the open pit, to allow mining on the west and north ends of the pit. Trolley line 5 starts at bench 17 and runs for 2 km along an 8 % ramp from where it continues up to waste dump 7525 as trolley line 4 for another 1 km along a 10 % ramp. Trolley line 5A, which is 0.3 km long, assists the trucks between trolley lines 5 and 1, along a 10 % ramp. Trolley line 1 runs for 1 km from the primary crushers, along the route used by the trucks exiting from the open pit. At present, the total length of trolley line in the open pit is 4.1 km.

All of the haultrucks have been converted to run on trolley assist. The haultrucks are also fitted with diesoline engines, allowing free manouvering within the confines of the open pit and movement from shovel to exit ramp. On reaching the trolley system at the base of the exit ramp, the truck driver selects trolley mode. The truck automatically switches over the wheel motors from parallel to series,

whilst at the same time the pantograph is raised to the overhead electric line. The line reject rate is less than 1 %.

Each haultruck cab is air conditioned and the air is filtered to minimize the driver's exposure to dust and heat. Each haultruck cab is also insulated against noise.

The pit control office in the open pit controls traffic in the pit by means of a radio communications system with each truck. The truck despatcher is able to despatch the truck to either the crushers, ore dump or waste rock dumps, depending on scanner and other information, while feeding information directly into a Management Information System (MIS).

3.4.2.5 *Construction*

Good haulroads are absolutely essential for the efficient operation of an open pit. For this purpose, a relatively large fleet of auxiliary machines is operated at Rössing. Because of the hard and abrasive nature of the rock one track dozer is assigned to each working shovel and active dump to maintain footwalls in good condition (Sandy, 1987). This also ensures that any haulroads that are subsequently developed in these areas have a firm construction base.

Haulroad and ramp construction consists of a 200 mm layer of fine material which is initially levelled with a track dozer. The rough surface is then broken up with a grid roller and then graded and top dressed with 50 mm of river sand transported from the nearby river gorges. All road bends are elevated on the outside curve, with a 4 % fall across the road, to minimize spillage and to enable haultrucks to negotiate bends without slowing down. As a safety measure, 2 m high rock berms are placed on the open edge of all haulroads and benches (Sandy, 1987).

3.4.2.6 *Wall control*

The position of the Rössing ore body on the limb of a syncline impacts slope stability. The Khan formation has a generally consistent southerly dip of about 70°, whereas the Rössing formation is intensely folded and the axis of the syncline falls within the southern portion of the present open pit (Sandy, 1987).

Despite being heavily jointed, most rock types at Rössing have high strength values and will support steep pit walls. Inter-ramp slope angles of the final pit limits vary from 47° in the softest formations to 60° on the south wall where the dip, joint structure and rock strength are extremely favourable (Sandy, 1987).

In order to achieve these relatively high slope angles, it has been necessary to pre-split many of the bench faces. A typical section through the South and North walls of the Rössing open pit is shown in Figure 3.2. Because the main rock structures dip at 70° to the south, pre-split faces on the North Wall are inclined at 70° parallel with the foliation planes. On the South Wall, the pre-split is drilled at 80° to enable a steep overall slope design with a 20 m wide safety berm every four benches (i.e. 60 vertical metres).

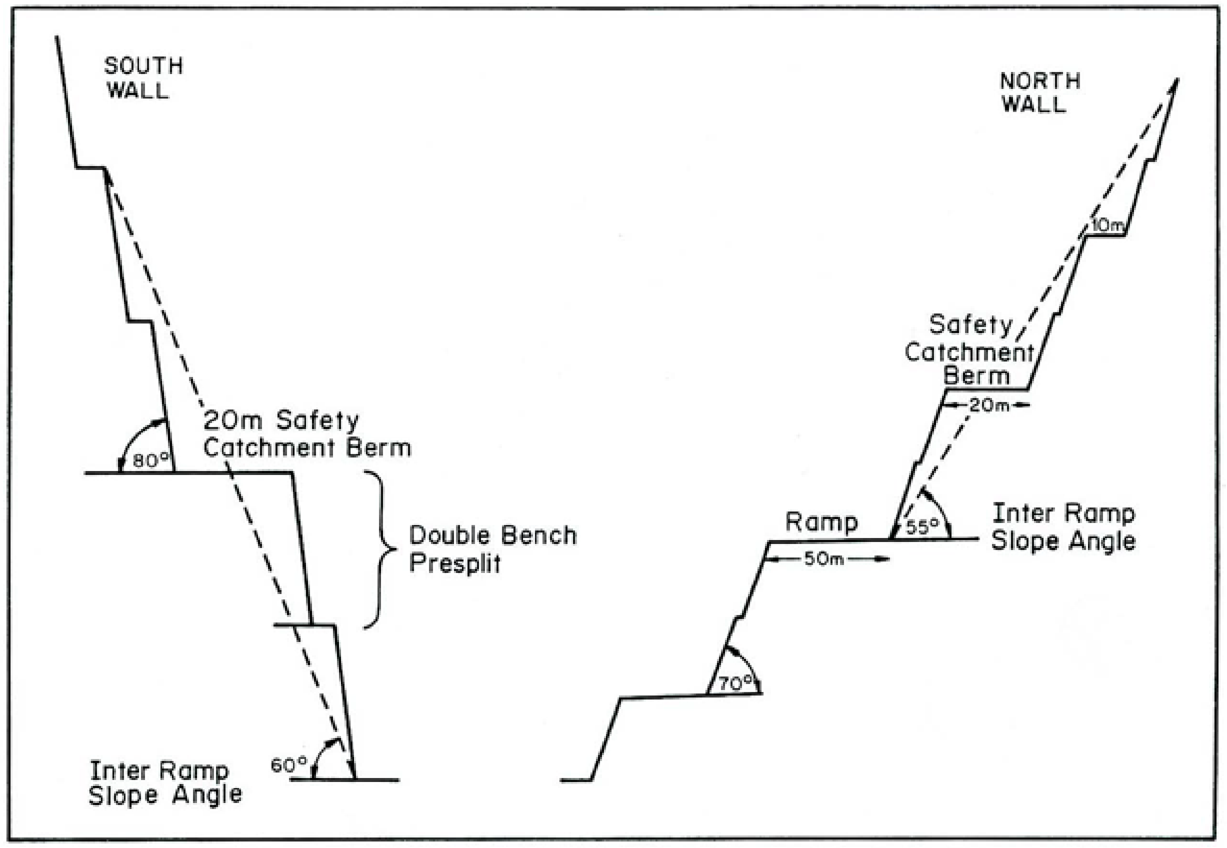

Figure 3.2: Typical sections of the North and South Walls at the Rössing Uranium Mine open pit; (Redrawn from Sandy, 1987).

3.5 Processing Operations

The current ore processing operations at Rössing Uranium Mine are the result of considerable testing and development work conducted during the early stages of mine development (Rössing Uranium, 1987). A variety of metallurgical processes were tested on a large pilot plant with a capacity of 100 tonnes day^{-1} to reduce final scale-up problems. In certain instances, design changes had to be introduced subsequent to the start of mining operations (Vernon, 1981). A diagrammatic sketch of the processing operations at Rössing Uranium Mine is shown in Figure 3.3.

3.5.1 Radiometric Truck Scanning

Due to the erratic nature of the mineralization and the movement of material when it is blasted, there is considerable intermingling of waste rock and ore. To eliminate waste rock from ore blocks and optimize the ore grade fed to the primary crushers, radiometric scanners were installed at the Rössing open pit (Corner *et al.*, 1986).

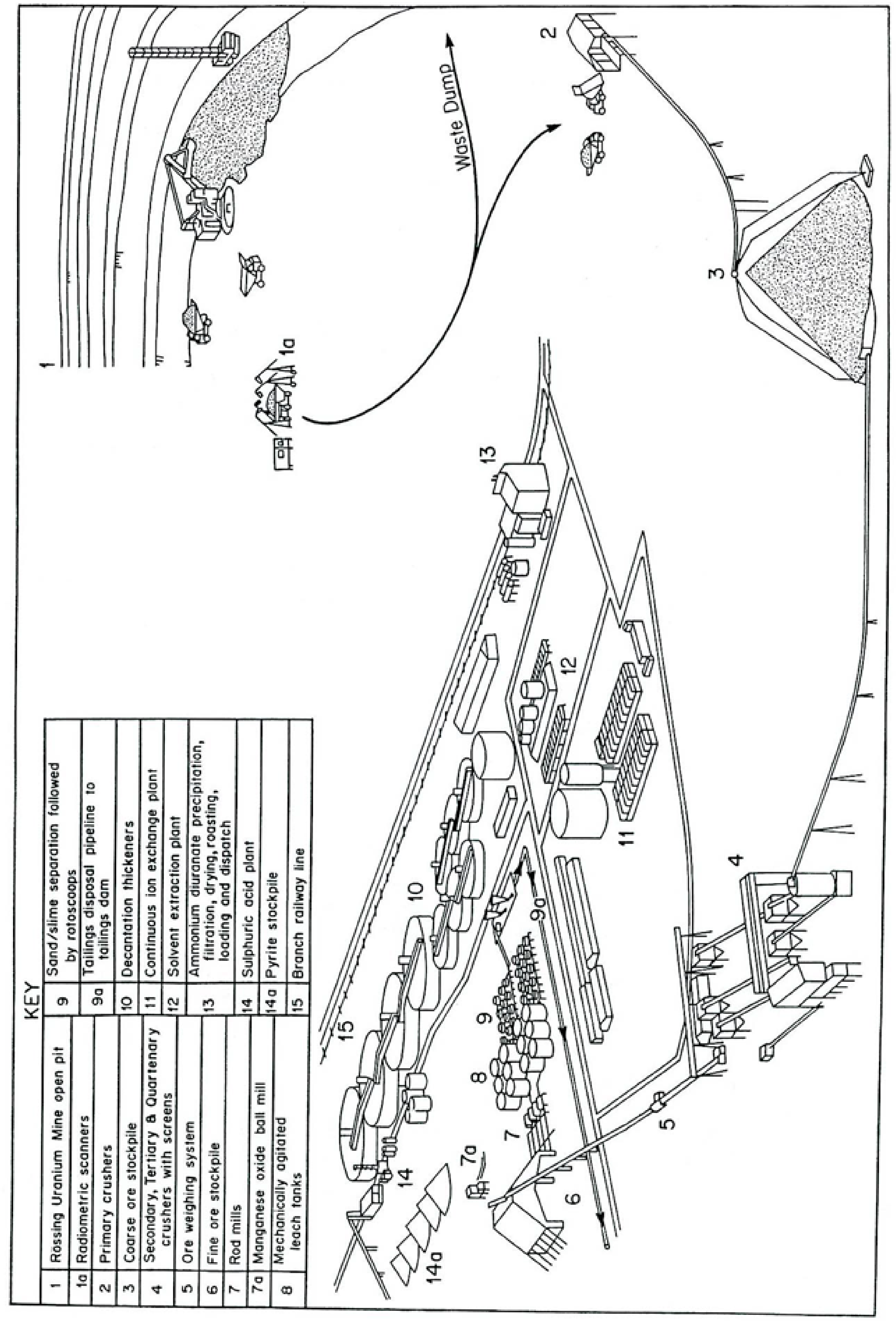

Figure 3.3: Diagrammatic sketch of the ore processing operations at Rössing Uranium Mine.

Each haultruck loaded with ore and marginal composites drives under one of the five radiometric scanners and stops. Each scanner is fitted with four scanner heads, each containing a lead-shielded crystal of sodium iodide so that only radiation from the rock in the haultruck can reach the crystal. A sodium iodide crystal has the property of emitting light impulses when hit by gamma radiation. A photomultiplier converts the light impulses into electrical impulses which are then added up over a period of 30 seconds and displayed on a digital readout. The grade of material in the haultruck is proportional to the amount of gamma radiation given off (Corner *et al.*, 1986). This radiation reading is assessed by a computer which then dispatches the haultruck to either the primary crusher, low-grade ore stockpile or waste rock dump, depending on the readout obtained. From these readings, a daily grade is calculated for the crushed material.

3.5.2 Ore Processing

The facilities and processes employed for uranium extraction at Rössing have several environmental implications. Vernon (1981, 1987) has described the various features in detail and they are summarized here so as to provide an overview of the operation. A simplified flow-sheet of the process is shown in Figure 3.4.

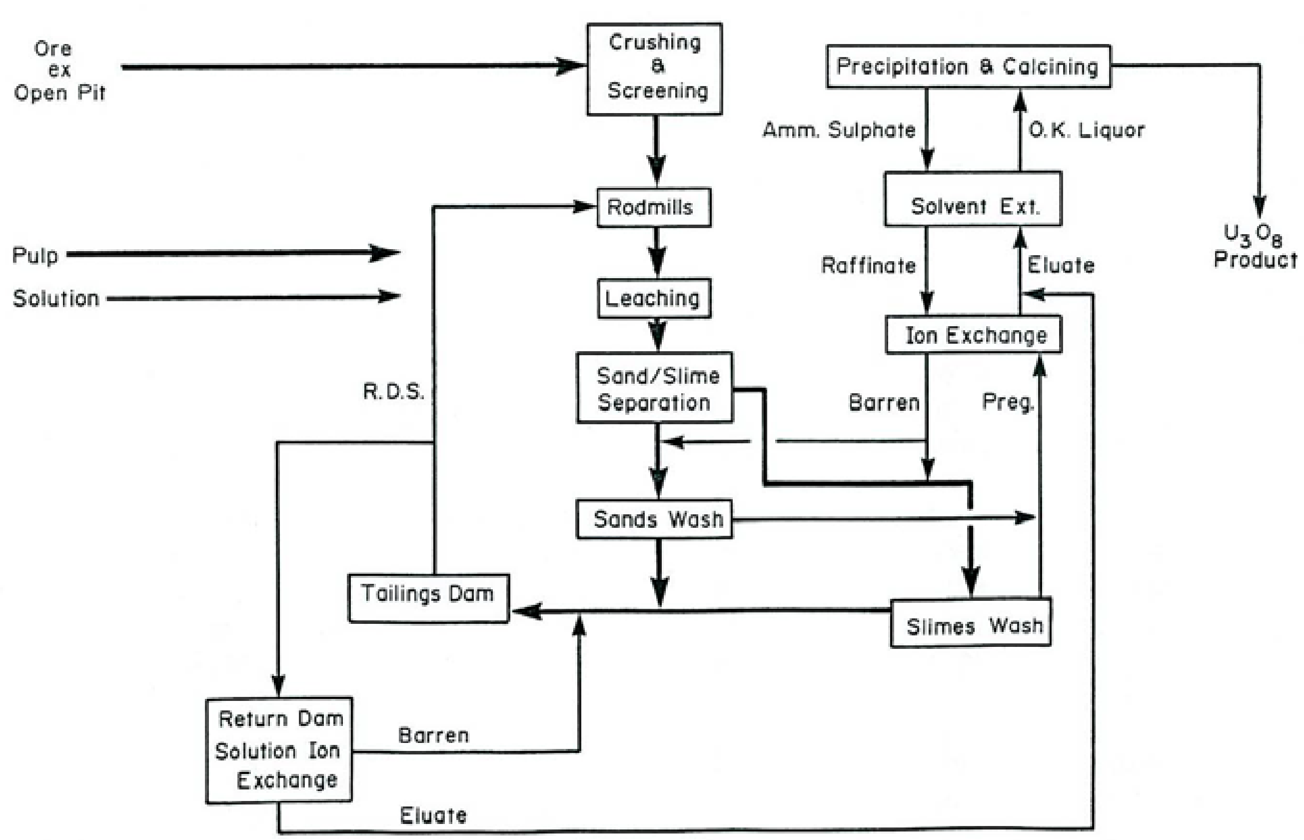

Figure 3.4: Simplified flow-sheet of the uranium extraction process employed at the Rössing Uranium Mine; (redrawn from Vernon, 1987).

3.5.2.1 *Crushing*

Run-of-mine ore is tipped from the haultrucks directly into one of two gyratory crushers, where it is reduced to fragments less than 170 mm in diameter. This crushed rock is fed directly onto a conveyor which elevates the ore to the coarse ore stockpile (Figure 3.3). This coarse ore stockpile has a total capacity of some 400 000 tonnes; approximately 70 000 tonnes of this is live storage (Vernon, 1987).

Ore is withdrawn from the base of the stockpile by vibrating feeders onto a second conveyor which delivers the ore to a 1 000 tonne capacity surge bin. A powerful magnet mounted over the head of this second conveyor removes scrap steel.

Ore is withdrawn from the surge bin and fed into two secondary crushers, which produce ore fragments of less than 65 mm in diameter. This product is then delivered to four vibrating screens equipped with 19 mm openings. The screen oversize is fed to a total of four tertiary crushers. Material passing through the screens (screen undersize) gravitates to conveyors which deliver to the fine ore storage.

The tertiary crusher product is fed to six vibrating screens with 19 mm screen decking. The screen undersize joins the tertiary screen undersize, as crusher plant product, and is delivered to the fine ore stockpile. The screen oversize is fed to two quaternary crushers, with apertures set to approximately 8 mm. The product from these crushers joins the tertiary crusher product and is recycled to the screening plant before joining the feed to the fine ore stockpile (Vernon, 1987).

Crushed and screened fine ore is elevated to the 80 000 tonne capacity fine ore stockpile by a 357 m long conveyor. A weightometer situated on this conveyor weighs the ore fed into the treatment plant.

3.5.2.2 *Leaching*

Four variable speed conveyors feed the crushed ore into four rodmills operating in open circuit. Fresh water and recycled tailings solution is added to give a discharge pulp comprising 78 % solids. Feed water is heated by waste steam from the acid plant in heat exchangers.

Each rodmill operates at about 500 tonnes hr^{-1}, producing a leach feed where about 90 % of the material is below +12 mesh in size. This unusually coarse grind is acceptable because the uranium mineralization is situated at the grain boundaries of the rock-forming minerals (Section 2.3.4), which are themselves quite coarse (Vernon, 1987). Leaching is carried out in two modules, each of six tanks. Residence time in the leach tanks varies from 8 to 12 hours dependent upon the number of tanks on line.

In this leaching process, the response of ore from the north and south sides of the pit is significantly different. That from the south side requires more severe leach conditions and a longer residence time compared with material from the north side of the pit. Ore from the south side of the pit also yields somewhat lower recoveries with greater consumption of acid and other reagents. The leach conditions are therefore flexed to minimize costs consistent with achieving U_3O_8 production targets. Up to the end of December 1990, the year-to-date plant recovery of U_3O_8 was 86.2 % (Vernon, 1991: personal communication).

Leached sand and slime are separated and washed in rotoscoops and in a five-stage counter-current decantation thickener circuit, respectively, using barren solution from the ion exchange plant (Vernon, 1981, 1987). The waste solid material is re-slurried with recycled solution and transported by pipeline to the nearby tailings dam. Control and recycling of the tailings dam solution has important economic and environmental implications and is described in detail in Section 3.6.2.3. The recycling of return dam solution contributes a further 3 % to the overall recovery of U_3O_8 (Vernon, 1987).

3.5.2.3 *Ion exchange*

Almost 80 000 m^3 day^{-1} of solution containing very low uranium levels, arises from the sand and slime washing operation. In order to treat such large volumes of solution, the Porter Continuous Ion Exchange (CIX) system was developed specifically for the Rössing operation.

The uranium-containing solution from the washing circuit (so-called "pregnant solution") is treated in four parallel lines of six rectangular contactors, each containing about 30 m^3 of resin. Solution is pumped into the bottom of the first contactor fluidizing the bed and overflows to a pump sump for transfer to the next stage. The principle of operation is shown in Figure 3.5.

The loaded resin is transferred into one of three fixed columns per line for elution. The eluent is acidified raffinate from the subsequent solvent extraction plant and flows consecutively through all three columns. Eluted resin is returned to the last loading contactor (Vernon, 1987). The concentrated eluate typically contains about 3.5 g litre^{-1} U_3O_8 together with a substantial range of impurities, particularly iron, nitrates and chlorides (Vernon, 1987). This is then fed to a conventional solvent extraction plant.

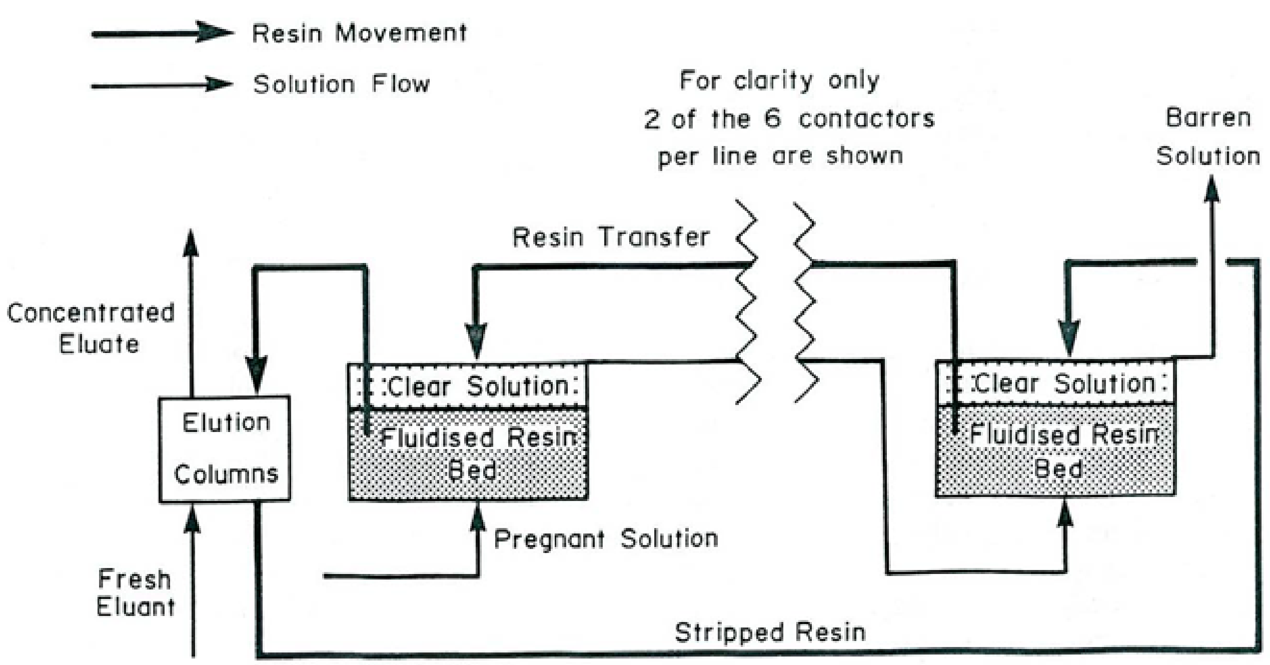

Figure 3.5: Simplified diagram of the Porter Continuous Ion Exchange (CIX) system as employed at Rössing Uranium Mine; (redrawn from Vernon, 1987).

3.5.2.4 *Solvent extraction*

The solvent extraction plant consists of mixer-settlers arranged in two lines of seven stages of extraction, two stages of water scrub, four stages of ammonia stripping and one regeneration stage. The product from the solvent extraction plant typically contains 9 g litre^{-1} and few impurities are fed forward to ammonia precipitation in the Final Product Plant (Vernon, 1987).

3.5.2.5 *Final product plant*

In the Final Product Plant, the uranium is precipitated as ammonium diuranate, or 'yellow cake', thickened, washed in two stages on rotary drum filters and calcined (roasted) to uranium oxide. This is then loaded into steel drums, (350-430 kg nett weight), containerized and dispatched by rail. At over 97 % U_3O_8, the Rössing product is recognized as one of the purest uranium oxide concentrates in the world (Rössing Uranium, 1987; Vernon, 1987).

3.5.2.6 *Acid production*

By world standards, acid consumption at Rössing is low in terms of kilogrammes of acid per tonne of ore treated. However, the total acid requirement is about 250 000 tonnes per annum (Vernon, 1987). An acid plant was therefore built as part of the original Rössing installation to convert pyrite obtained from the Otjihase Mine near Windhoek, to sulphuric acid (Vernon, 1987). The Rössing acid plant is capable of producing over 700 tonnes day^{-1} of sulphuric acid, which satisfies most of the mine's acid requirements (Vernon, 1987). Any shortfall in acid requirement has been made up by the importation of sulphuric acid by sea to the port of Walvis Bay and rail to the mine. Imported acid has always been substantially more expensive than the acid made at Rössing (currently by a factor of approximately 2).

A comparison of the quantity of sulphuric acid produced on site and imported shows that the total import requirement has fallen steadily from 57 % in 1980 to 6 % in 1986. This reduced acid requirement is due partly to reduced ore throughput, acid plant improvements and to increasingly good ore grade control in the open pit. The reduced imports of acid have had a very significant effect on stabilizing the production costs in the metallurgical plant (Vernon, 1987).

3.5.2.7 *Centralized process control*

During the early days of operation, the uranium extraction process was monitored from various control panels, each located near to the specific facility being monitored; trouble spots could be identified at a glance by the operators. However, these panels were scattered throughout the plant and required a large staff to man and maintain them. Difficulties in communication, supervision and accurate recording of process performance were often apparent. This state of affairs was considered to be ideal for centralization of process control using the latest computer technology. Since its inception, the degree of automatic control exercised by the central computer has been extended and the whole metallurgical operation is now controlled from one air-conditioned room. The Rössing Centralized Process Control (CPC) is now considered to be one of the most advanced operations of its kind on a mineral processing plant in Africa (Plate 7).

The system is linked to all monitoring points on the plant and information is updated several times each minute. All incoming data is checked and compared before being stored in the database; alarm, backup and self-diagnostic systems are also incorporated. The system allows rapid evaluation of incoming information, diagnosis of problems and the implementation of appropriate remedial measures.

3.6 Waste Disposal

The complex nature and low uranium content of the main uranium-bearing ore-body at the Rössing Uranium Mine necessitate the mining and processing of large tonnages of rock each year, throughout the life of the mine (see Section 3.3). In a mining operation of this size, the efficient and safe disposal of waste rock and tailings have presented, and will continue to present, problems of scale.

3.6.1 Waste Rock

At the Rössing Uranium Mine, waste rock comprises approximately 70% (some 110 000 tonnes) of the 150 000 tonnes of rock mined each day. This consists of barren country rock and of sub-economic uranium ore that has been rejected by the radiometric scanners (Section 3.5.1). The waste rock varies in consistency from large boulders to finer sands and gravel-sized particles. All of this material is transported by haultruck and dumped at one of several designated dumping sites. The location and extent of these sites is shown in Figure 3.1.

Since the start of open-pit mining activities in 1976 (Vernon, 1987), approximately 475 million tonnes of waste rock has been moved and dumped in the 13-year period up to June 1989. This is equivalent to approximately 715 haultruck loads, each of 140 metric tonnes, per day. Initially, waste rock was tipped into those sections of Dome and Pinnacle Gorges that were located nearest to the developing open pit. These waste rock dumps have been gradually extended to the point where the lower section of each gorge is almost completely filled with waste rock. Indeed, the waste rock in the lower portion of Dome Gorge is now within 100 metres of its junction with the Khan River, up to the border of the Mining Grant (Figure 3.1).

In future, waste rock will be continue to be dumped into the accessible portions of Pinnacle and Dome Gorges until these sites are full. After this, a substantial amount of waste rock will be used as back-filling in the mined out north-western portions of the open pit at the end of the life of the mine (R.L. Murphy, personal communication).

3.6.2 Tailings

At Rössing Uranium Mine, all of the waste solids (tailings) from the uranium extraction process are pumped to a large tailings dam (Figure 3.1). This tailings dam is located to the west of a north-east trending ridge which effectively separates the tailings dam from the rest of the mine workings (Section 2.1). Tailings disposal began in mid-1976 and some 190 million tonnes have been deposited in a 13-year period up to mid-1989. Because of the low uranium content of the ore, the tailings consist of virtually the entire mass of input ore (40 000 metric tonnes per day) plus waste process liquids and the calcine residue from pyrite used in acid making (Dames & Moore, 1984b; Vernon, 1987).

3.6.2.1 *Tailings dam construction*

The original tailings dam was designed to encircle an evaporation pond and provide a deposition area of some 1000 hectares (10 km^2). However, pilot plant studies showed that the residual uranium in the tailings dam water could be recovered and a large proportion of the water could be recycled in the plant. This recycling of water from the tailings pond allowed the tailings dam to be redesigned with considerable reductions in capital, as well as operating and projected close-out costs (Kesler, 1987).

Initial construction of the tailings dam commenced with the dumping of a starter wall of waste rock across the upper portion of Pinnacle Gorge, followed by "upstream" deposition of tailings (Figure 3.6). The coarser sand fraction settled out rapidly whilst the finer slimes and water flowed into the main body of the dam. The sands dried out fairly rapidly and were then bulldozed to form a raise from which further tailings could be deposited. Two raises are required to form a 6 m bench and, after surface preparation, a 15 m wide access roadway is formed before the start of the next bench (Figure 3.6a and b). The overall embankment slope is 27° whilst the beach angle starts at 6°, tapering off to less than 1° when the slimes reach the water surface, with an average slope angle of 2° (Kesler, 1987).

As shown on the site plan (Figure 3.1), the tailings impoundment is anchored at its eastern end against the ridge of hills. Initially, the embankment extended approximately 1700 m to the west of the ridge, with a similar but shorter wall extending some 900 m to the north (Kesler, 1987).

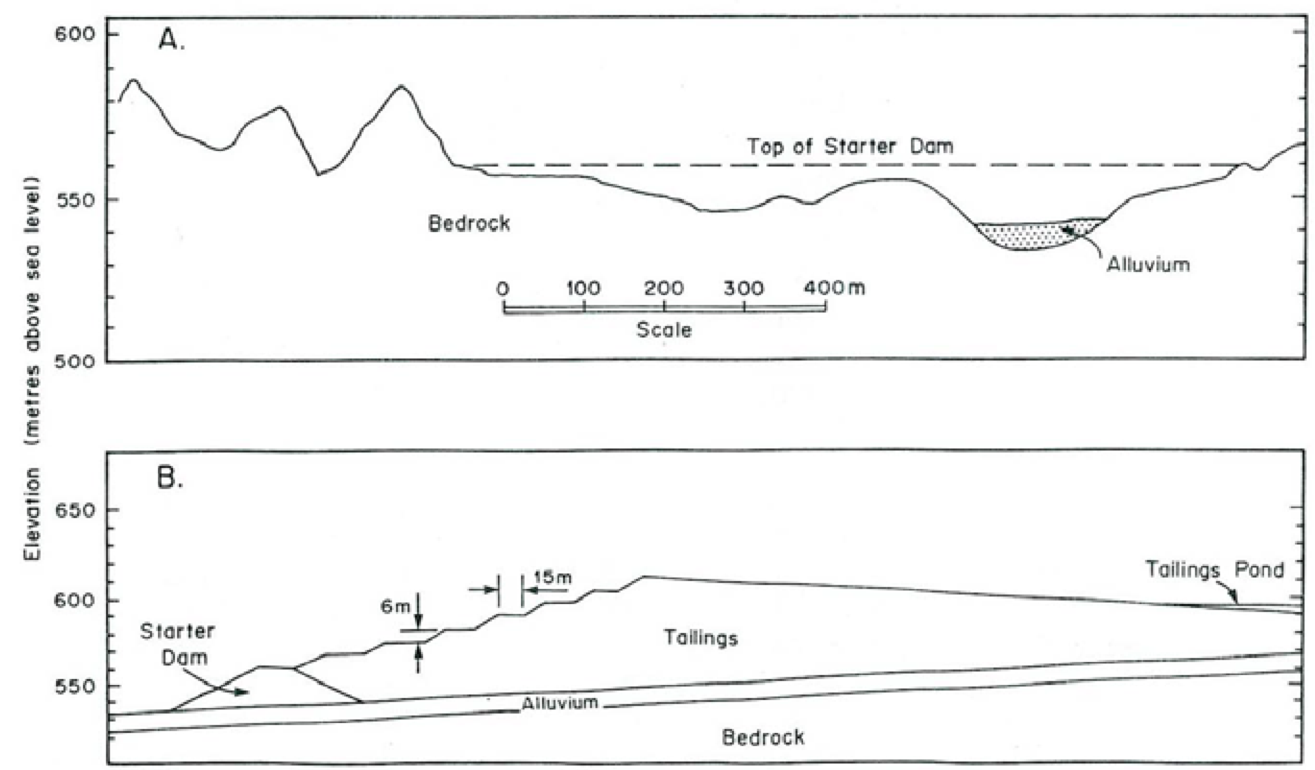

Figure 3.6: (A) - Horizontal profile across Pinnacle Gorge, showing the elevation of the starter dam for the tailings impoundment; and (B) typical longitudinal cross-section of the tailings impoundment showing major construction features; (redrawn from Robinson & Eivemark, 1987).