## *C h a p t e r   1 :      I n t r o d u c t i o n*

### 1.1   Overview of Mine Tailings and Management Issues

In response to growing community concerns and increasing government regulation, mining companies are placing a greater emphasis on mine waste management to ensure that potentially harmful substances are safely stored in a manner that permits the restoration of post mining land uses. These wastes principally comprise waste rock arising from mining operations and tailings produced from the milling process. The milling process is designed to separate the ore (economic mineralisation) from gangue minerals, the latter often comprising of acid forming sulfides, radionuclides and toxic metals/metalloids such as lead, copper, mercury, cadmium, arsenic and selenium.

Milling generally consists of three steps (Ritcey, 1989):

1.  Preparation, whereby the ore is comminuted by crushing and/or grinding;

2.  Concentration, whereby the economic mineralisation is separated from the host rock;

3.  Dewatering.

In the generation of tailings, the physico-chemical properties (particle size, moisture content, chemical composition) of the host rock are altered by the action of comminution (mechanical crushing and grinding) and concentration which invariably requires lixiviants (such as acid, cyanide or oxidants) and other process chemicals. As a consequence the gangue minerals that comprise the tailings are much more susceptible to erosion and chemical weathering which, if not controlled, can lead to the release of harmful quantities of contaminants (far in excess of those released from the original ore body) into the environment. Therefore the security of tailings disposal, particularly in the long-term and effective rehabilitation of tailings repositories are vital attributes of mine planning.

The principal objectives of decommissioning and rehabilitation of tailings storage facilities (TSFs), particularly above ground TSFs, are well known and according to the International Commission on Large Dams (ICOLD, 1996), include:

•        Stabilization of the impoundment - involving consideration of long-term stability, seismology, erosion protection and drainage systems;

- Hydrology - long-term assessment of catchment runoff, diversion arrangements and   risk of breaching during heavy rainfall events;

- Contamination - leachate control or containment, surface and seepage water quality;

- Aesthetic or visual impact;

- After use or final land-use considerations; and

- Safety - public accessibility and inherent dangers, long-term surveillance and monitoring.

However, despite national and international standards/guidelines on tailings management there are still disastrous failures and long-term or chronic environmental impacts arising from TSFs. The most recent and notable catastrophic dam failures include Marcopper in the Philippines (1996), Merrienpruitt in South Africa (Wagener, 1997), Las Frailes incident in Spain (1998) and Baia Mare in Romania (2000). All of these incidents involved a failure that resulted in the immediate release of large quantities of tailings and/or effluent into local rivers and nearby communities.

Chronic impacts from TSFs include the release of contaminated seepage and runoff into local aquifers and aquatic ecosystems. Past practices at two Australian mines, Rum Jungle in the Northern Territory (Verhoeven, 1988) and Mt Lyell in Tasmania (McQuade et al., 1995), typify the devastating impacts on aquatic ecosystems that arise from the long-term (decades) discharge of untreated tailings effluent into local river systems.

Understandably, both past and present tailings management practices have not engendered community confidence in the mining industry. Benchmark surveys of community perceptions (Syme et al., 1992) and other studies by Mining Journal Research Services (1996), show that the management of tailings and their long-term containment is a significant concern for Regulatory Agencies, Non-Government Organisations and the community in general.

To allay these concerns prudent design, planning and management of TSFs for both operation and closure require a systematic risk-based approach to identify potential hazards, controls and mitigating measures. Fundamental to this assessment and planning process is a detailed understanding of the mineralogical characteristics and geochemical mechanisms that control

the long-term evolution of the tailings and possible transfer of contaminants from the storage facility to the environment. Only through a thorough understanding of these mechanisms will it be possible to improve the certainty of environmental impact assessment and, in so doing, better define the requirements for the safe operation and ultimate long-term closure of TSFs.

## 1.2   Uranium Mining and Milling in Australia

Uranium mining and milling is the first step in a complex industrial fuel cycle for generating electricity from nuclear reactors. At present there are 436 commercial nuclear reactors in 31 countries supplying 16 per cent of the world's electricity (World Nuclear Association, 2004). Based on worldwide energy projections over the next 20 years, a further 67 power reactors are to be commissioned (World Nuclear Association, 2004). The demand for Australian uranium, which accounts for approximately 30% of the world's recoverable resources (UIC, 2004), is also expected to grow over the same period. Currently, Australia produces uranium from three mines:

- Ranger, a conventional open pit operation located in the Northern Territory, with an annual production of approximately 4,000 to 5,000 t of $U_3O_8$ ;
- Olympic Dam, an underground mine located in South Australia, with an annual production of approximately 4,500 t of $U_3O_8$; and
- Beverley, an *in situ* leach mine located in South Australia, with an annual output of 850 t of $U_3O_8$. In contrast to open pit and underground mines, *in situ* leaching does not produce tailings.

During 2003/04, production from Australian mines was 9,533 tonnes ($U_3O_8$) or 25% of World production (UIC, 2004). These exports contributed approximately $AUD288 million to the Australian economy. With growing worldwide demand and the potential for further development of new mines, Australia's annual production is projected to exceed 10,000 tonnes by 2005 (ABARE, 2004). It also follows that an increase in uranium production will lead to a concomitant increase in the production of uranium tailings. As most uranium ores contain between 0.05 to 0.2% uranium (UIL, 2004), it is conceivable that for every one kilogram of extracted uranium from conventional open pit and underground mines, two to eight tonnes of tailings will be produced. On this basis the projected increase in Australian uranium production is expected to result in an additional four million tonnes of tailings per year that will need to be managed and safely stored in the long-term.

As a result of the milling process, shown diagrammatically in Figure 1.1, uranium tailings typically contain finely ground gangue minerals, the silicate matrix of the original ore forming host rock, secondary minerals, metal precipitates formed during neutralization, long lived uranium daughter products and chemical residues from the extraction process (Pidgeon, 1982; Ring et al., 1982; Ritcey, 1989). The chemical residues contribute approximately 86 kg/t ore of inorganic chemicals (sulfuric acid, lime, manganese oxide (pyrolusite), iron, ammonia) and 0.5 kg/t ore of organic reagents (Ranger Mine pers comm., 1996). These residual process reagents are known for their ecotoxicity, particularly ammonia which is a toxicant to freshwater fish species at concentrations (uncorrected for pH) ranging from 0.88 to 4.6 mg/L (ANZECC/ARMCANZ, 2000).
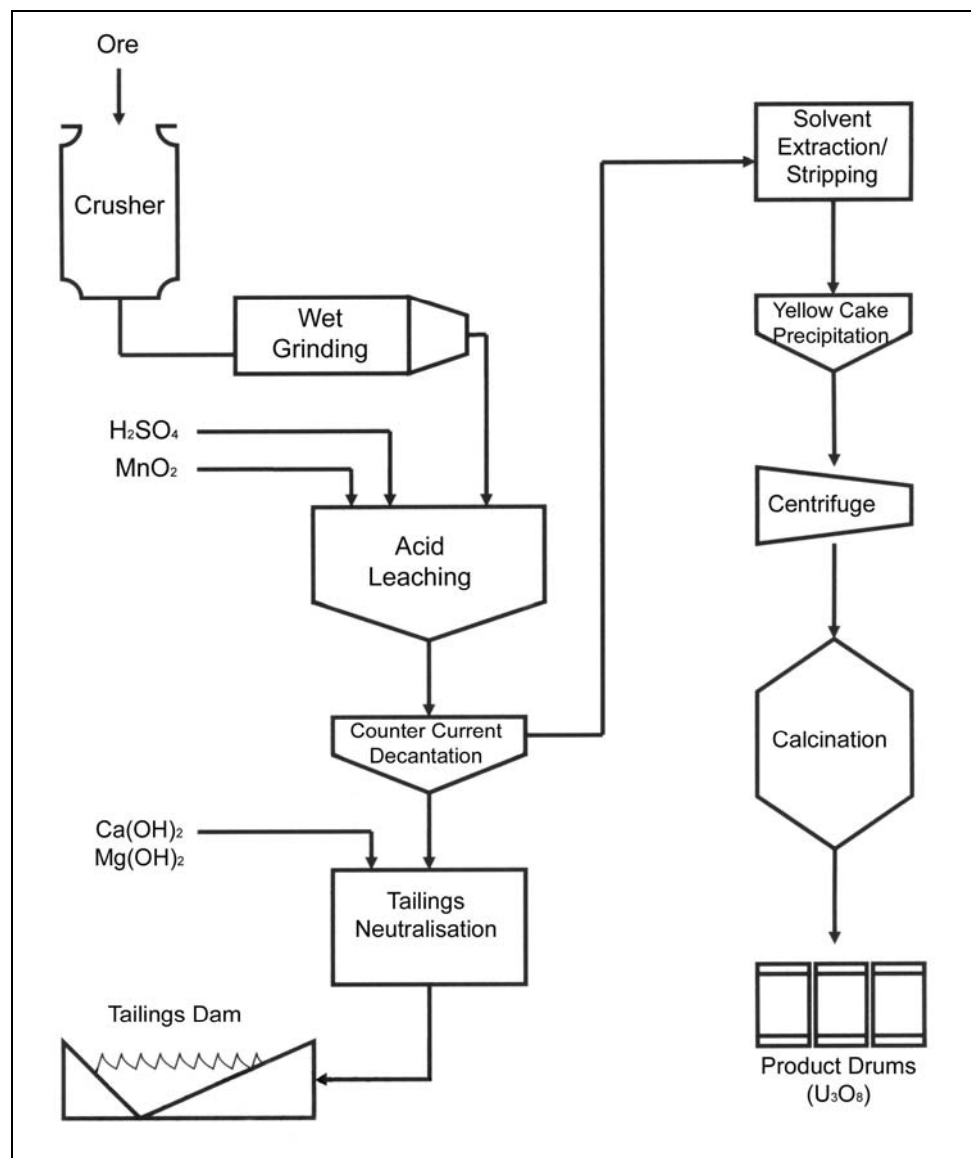
**Figure 1.1:   Acid leach flow diagram for Ranger ores (Ring et. al., 1982)**

Given the final grind size of the ore and relatively strong oxidizing conditions of the leach circuit, the tailings are both chemically and physically altered so as to increase their reactivity and, consequently, the mobility of heavy metals and other contaminants. Although the enrichment of heavy metals in mineralized areas is a uniquely site specific phenomenon, their concentrations typically exceed ecological and human health criteria thus posing a significant environmental risk.

Unlike other hydrometallurgical processes, a unique and distinctive feature of uranium ore processing is the dissolution of uranium decay products during milling. The distribution and abundance of radiogenic elements in uranium mill tailings are less dependent on site specific factors than the non-radioactive contaminants. For example, in a typical uranium mill, about 85% of the total radioactivity contained in the original ore ends up in the tailings (Waggitt, 1994). This is a function of the highly selective extraction process that recovers approximately 95% by weight of the uranium or 15% of the total radioactivity from the ore. The remaining 5% of the uranium and its associated radioactive decay or daughter products constitute a voluminous form of low level radioactive waste. The daughter products of primary concern are thorium-230, radium-226 and lead-210, having long half-lives of 80,000, 1,600 and 21 years, respectively.

Thorium is significant as it decays to radium-226 which, in turn, decays to the radioactive gas radon-222. Radon has a relatively short half-life of 3.8 days and being a gas can escape the tailings (relative to water borne contaminants) and be dispersed by wind. Although radon is chemically inert, it decays to radon progeny which pose a significant health risk especially in relation to lung cancer and leukaemia (UMTRA, 1991; IAEA, 1997). Radon progeny or daughter products include lead-210 and short lived isotopes of bismuth and polonium. Being formed from a gas, these isotopes attach to aerosol particles thus exposing humans via inhalation. The longer lived isotope lead-210 is of particular concern as it may give rise to exposure to humans through a variety of pathways, for example, lead can substitute for calcium in bone formation (IAEA, 1997). Given the potential health risks of radon and its progeny, its release into the atmosphere must be carefully controlled to maintain exhalation rates from the tailings repository to acceptable levels.

Waggitt (1994) concluded that the most significant environmental and health hazards that may arise from dispersal of or public exposure to, uranium mill tailings are those depicted in Figure 1.2.
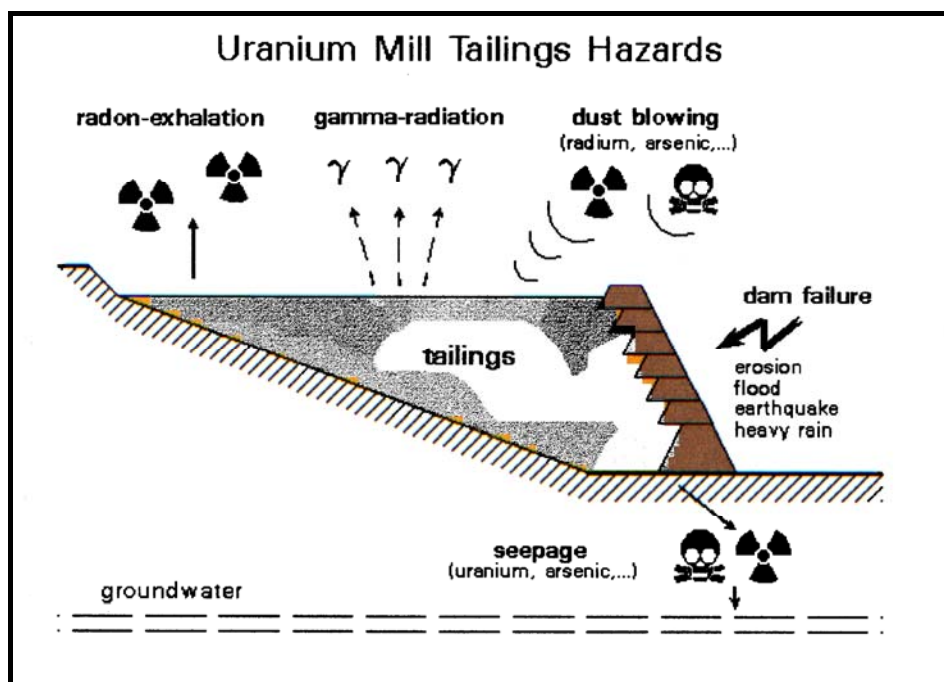


**Figure 1.2:   Uranium mill tailings hazards. (Waggitt, 1994;**
**Uranium Institute, 1991)**

## 1.3   Review of Applicable Case Studies

The mineralogical and geochemical characteristics of uranium mill tailings are primarily dependent on site-specific factors such as local geology/hydrogeology, ore grade (metal enrichment), leach circuit and *in situ* weathering and authigenic processes. Sampling of tailings profiles within operating and/or closed-out storage facilities has provided *in situ* data that are representative of the mineralogical and geochemical characteristics of the tailings pile. These data have, in turn, facilitated the development of site-specific models (Jambor, 1994) that describe the key geochemical processes that control the evolution and long-term stability of the pile over time. Acquisition of such information is required to adequately plan for the safe long-term closure of uranium TSFs.

The Canadian government and mining industry have conducted a lot of research in this area with many of the geochemical and mineralogical studies of uranium TSF's being conducted under auspices of the National Uranium Tailings Program (NUPT). Following NUPT the Mine Environment Neutral Drainage (MEND) program was established to examine aspects of

acid mine drainage (AMD), (Feasby et al., 1997; Taylor and Thornber, 1992). Although AMD is still a significant environmental issue for the mining industry, geochemical processes that lead to AMD are better understood (Andrew, 1980; Jambor, 1994) than processes involving the formation/dissolution of secondary and tertiary minerals within tailings piles and the subsequent release of contaminants in neutral or near neutral mine drainage. Like AMD, near neutral drainage can also contain heavy metals at concentrations that exceed ecological and human health criteria.

Members of the Waterloo Centre for Groundwater Research (Blowes et al., 1988; Blowes and Jambor, 1990; Blowes and Ptacek, 1994) have also extensively studied mineralogical and geochemical data for tailings at various Canadian mines. These studies highlighted the need to adopt an holistic approach to modelling the long-term behaviour of tailings by integrating knowledge of mineralogy, geochemistry and *in situ* pore water chemistry into one interactive solid-water system. Similar studies have been conducted in Australia by the Commonwealth Scientific and Industrial Research Organisation (CSIRO) and the Australian Nuclear Science and Technology Organisation (ANSTO). Unfortunately, much of this latter information is contained in confidential company reports and is not generally available for peer review or to assist in the development of generic guidelines for assessing the geochemical risk and planning for the long-term closure of TSFs.

In view of the potential ecological and human hazards associated with uranium mill tailings, international organisations such as the International Atomic Energy Agency (IAEA) and the OECD Nuclear Energy Agency have conducted reviews of contemporary close-out practices of uranium mines with their member states, with a view to developing consistent guidelines for the risk-based assessment and long-term management of uranium mill tailings (IAEA, 1997). As part of this initiative, the IAEA estimated that more than two billion tonnes of residues and materials containing trace concentrations of uranium and other extraction related contaminants must be managed worldwide. A similar review by Waggitt (1994) estimated that there is more than 500 million tonnes of uranium mill tailings located in 18 countries around the world. Australia's operating uranium mines (Ranger and Olympic Dam) have produced, to date, approximately 14 Mt and 61 Mt respectively, with the latter being a co-disposed mixture of Cu and U tailings. Previous mines, as summarised in Table 1.1 (UIC, 2004), produced around 11.7 Mt between 1954 and 1988.

The safe and environmentally acceptable long-term management of these wastes within economic and other country specific constraints represents a formidable but achievable challenge.

Australia, like many other developed countries, has a history of rehabilitation of uranium mill tailings that shows an evolutionary approach in techniques and standards from the minimalistic strategies of the 1950's through the post closure planning era of the 1980's to the holistic life of mine planning of the present time (AECB, 1987; Waggitt, 1994).

**Table 1.1: Tailings Production from Historic Uranium Mines**

| Mine | Period | Average Grade %($U_3O_8$) | Tailings (t) |
|---|---|---|---|
| Radium Hill | 1954-62 | 0.13 | 970,000 |
| Rum Jungle | 1954-71 | 0.35 | 860,000 |
| Mary Kathleen | 1958-63 | 0.15 | 2,896,000 |
| Mary Kathleen | 1975-82 | 0.10 | 6,295,000 |
| Moline | 1959-64 | 0.52 | 127,480 |
| Rockhole | 1959-62 | 1.12 | 13,362 |
| Nabarlek | 1979-88 | 2.0 | 589,142 |
| **Total** | | | **11,750,984** |

Historic uranium mining practices of the 1950's are described by Waggitt (1994). The South Alligator Mill, Moline Mine and Rum Jungle Mine in the Northern Territory were all abandoned in the early 1970's. At all sites, tailings management and containment was non existent and as a consequence significant quantities of effluent and tailings solids were discharged into local river systems. At the Moline site, Waggitt (1994) estimated that approximately 72,000 tonnes or 25% of the tailings were eroded and dispersed into natural waterways.

During the late 1980's and throughout the 1990's, all of these sites were rehabilitated by either returning the residual tailings to the mine voids or capping/stabilising *in situ*. However, there is little or no public data to indicate whether the rehabilitation mitigated those hazards identified by Waggitt (1994), or complied with the Australian Government Code of Practice on the Management of Radioactive Wastes from the Mining and Milling of Radioactive Ores (1982). The Code stipulates the basic requirements for rehabilitation of tailings repositories. Of particular interest to safe closure planning are the following requirements:

- *"….release of radioactivity from the site is minimised and the exposure to members of the public is as low as reasonably achievable and below the limits prescribed..."* in relevant schedules;

- *"….final disposition of radioactive wastes and the need for rehabilitation of the site shall be such that the need for subsequent inspection, monitoring and maintenance is minimised, and preferably rendered unnecessary.…"; and*

- The design life of the rehabilitation measures is specified to be *"….at least 200 years",* and the *"structural life, assuming a minimum renovation, should be at least 1000 years, which could be extended to some thousands by regular attention when required."*

Of all the historic Australian uranium mines, the Mary Kathleen mine in Queensland which operated from 1958 to 1982 appears to be one of the better examples of proactive closure planning. Waggitt (1994) reports that *"the overall objective of the plan was to leave the site in a safe and stable state consistent with the proposed future land use (rangeland grazing) and with no radiological restrictions on public access. The general principle of the works was that they should comply with the code of practice (AGPS, 1987)."* During the planning stage for closure, a collaborative research programme was established between the Australian Atomic Energy Commission (AAEC) and Australian Groundwater Consultants (AGC) Pty Ltd., to study the chemistry and movement of radionuclides within the tailings disposal area. The work programme consisted of three distinct stages:

1.  The concentration of radionuclides in the tailings was measured as a function of depth and the variables pH and sulfate;

2.  The tailings were examined for mineralogy to determine the nature of precipitated radionuclides and associated elements and minerals; and

3.      Tailings samples were reconstituted into stratified columns for leaching experiments to simulate natural conditions in the tailings disposal area during the extended period of rehabilitation to determine the stability of the radionuclides.

The findings of the study (AGC and AAEC, 1982) highlighted the value of understanding the solid state speciation of the long-lived radionuclides, their abundance and distribution throughout the tailings pile in order to develop appropriate engineered controls, such as protective covers, to isolate the tailings from the receiving environment. More recent studies at the site by Lottermoser et al. (2003) report that the dry cover has effectively isolated the tailings solids from the receiving environment, although geochemical reactions within the tailings pile have resulted in acid generation and release of seepage waters laden with dissolved solids, metals and radionuclides. Oxidation of sulfide minerals and the reductive dissolution of Fe and Mn oxyhydroxides at depth were purported to be the key mechanisms for the observed hydrochemistry of seepage waters. Lottermoser et al. (2003) also concluded that both the neutralising potential and adsorption capacity of the various mineral phases within tailings were overestimated at the time of mine closure. This is a significant finding as it highlights both the difficulties and importance of obtaining representative samples of mineralogically complex tailings solids to enable the assessment of diagenetic processes for the development of applicable closure criteria.

Since the 1980's and with increasing community concern (Balkau, 1998), the Australian Government requires all new uranium mines to submit comprehensive tailings management plans, based on both national and internationally accepted practices/guidelines (IAEA, 1992; Atomic Energy Control Board Canada, 1987; AGPS, 1987), for public review and assessment prior to the issuance of a license to operate. Although these guidelines prescribe rigorous design requirements for the expected life and structural stability of the storage facility, there is still little guidance on how to systematically assess the long-term geochemical risks of the stored tailings.

Cost effective design and safe management of uranium mill tailings can only be achieved if the risks to human health and the environment are recognised and understood at the time of mine planning. The application of environmental geochemistry is essential in the early phase of mine planning as it provides a systematic means of understanding the interactions at the tailings solid-water interface. These reactions are fundamental to the evolution of tailings pore

waters and the rate of re-distribution and hence fate of contaminants within the tailings pile. At the macro level, this information is required to assess the long-term integrity of the proposed or existing TSF and its ability to prevent impacts on the beneficial values (ecological, human health) of surface and ground waters.

The paucity of integrated geotechnical/geochemical assessment protocols combined with the community's demand for greater environmental stewardship and conservation, provides a unique research opportunity to study the role of environmental geochemistry in the initial impact assessment of uranium mill tailings and for their subsequent long-term management and disposal.

## 1.4    Hypothesis and Research Program

For reasons outlined in the following Chapter, the Ranger uranium mine was chosen as an ideal site for researching the effects of weathering and diagenesis on the long-term geochemical stability of uranium mill tailings. More specifically this thesis sets out to either prove or disprove the hypothesis that:

*"The solid state speciation and mobility of metals and radionuclides in the tailings pile are governed by the processes of oxidative dissolution of sulfide minerals, weathering of phyllosilicates and organic matter diagenesis. Combined, these processes lead to the formation of authigenic minerals which control the solubility of pore water constituents and the long-term geochemical evolution of the tailings pile."*

Analysis of the statement required the development of an integrated research program to characterise the key equilibrium and kinetic processes that control the long-term geochemical evolution and fate of contaminants such as, evaporites (Mg/Ca sulfate salts), radionuclides (uranium and radium-226) and trace metals (Cu, Pb, Cd, Ni and Co) within the tailings pile.

Specific research activities in support of the thesis statement include:

- A critique of contemporary international research on the application of environmental geochemistry for assessing the risks and long-term closure requirements of tailings storage facility;
- Mineralogical and geochemical characterisation of fresh (run of mill) and aged (permanently stored) uranium mill tailings;

- Establishment of tailings depth profiles within the tailings storage facility for the *in situ* measurement of porewater chemistry;

- An assessment of the solid state speciation or partitioning of radionuclides within the tailings solids;

- Establishment of large scale laboratory kinetic leach columns to simulate the long-term geochemical evolution of the tailings pile and to identify mechanisms responsible for the solubility and release of radionuclides and accessory minerals into the receiving environment;

- Development of a geochemical model that combines both equilibrium and kinetic processes to predict the chemical mass balance and long-term evolution of tailings porewaters;

- Assessment of research findings against the thesis statement or hypothesis; and

- A discussion on gaps in knowledge and opportunities for further research.

Within the context of this research program, this thesis attempts to highlight the important geochemical features of uranium mill tailings that either have positive and/or detrimental implications for their long-term management. Acquisition of such knowledge will assist in the development of appropriate controls and contingency measures to ensure that problematic drainage or seepage losses do not impact on the beneficial uses (human health and ecological) of regional surface and ground waters.

## *C h a p t e r   2 :     T h e   R a n g e r   M i n e   E n v i r o n m e n t*

### 2.1    The Alligator Rivers Region

Of the three operating Australian uranium mines, the Ranger Mine, is probably one of the most publicly scrutinized and regulated mines in the world. Ranger is located 230 km east of Darwin in the undulating lowlands of the Alligator Rivers Region (Figures 2.1(a) and 2.1(b)).
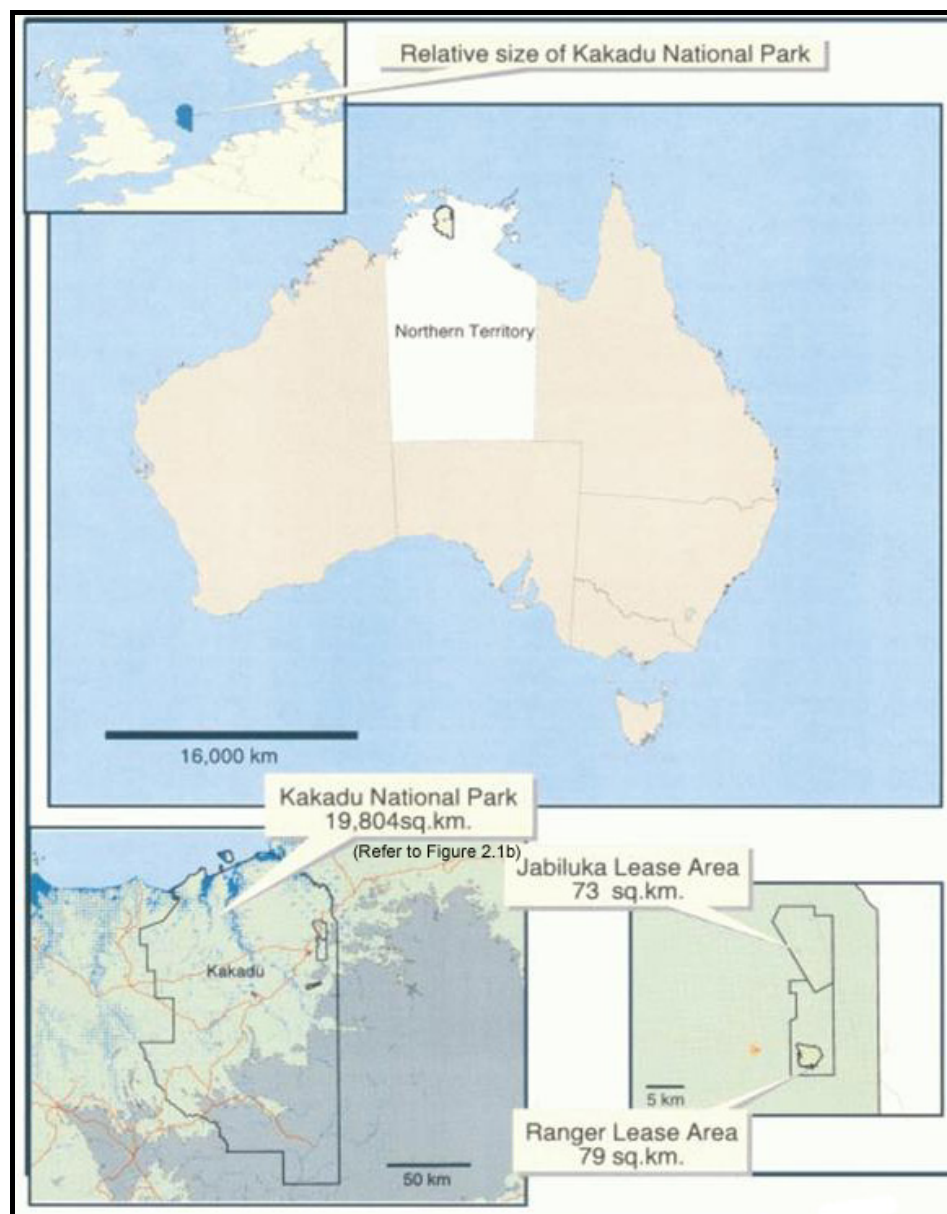


**Fig 2.1(a):    Relative sizes and location of Kakadu National Park,
Jabiluka and Ranger Mine leases**
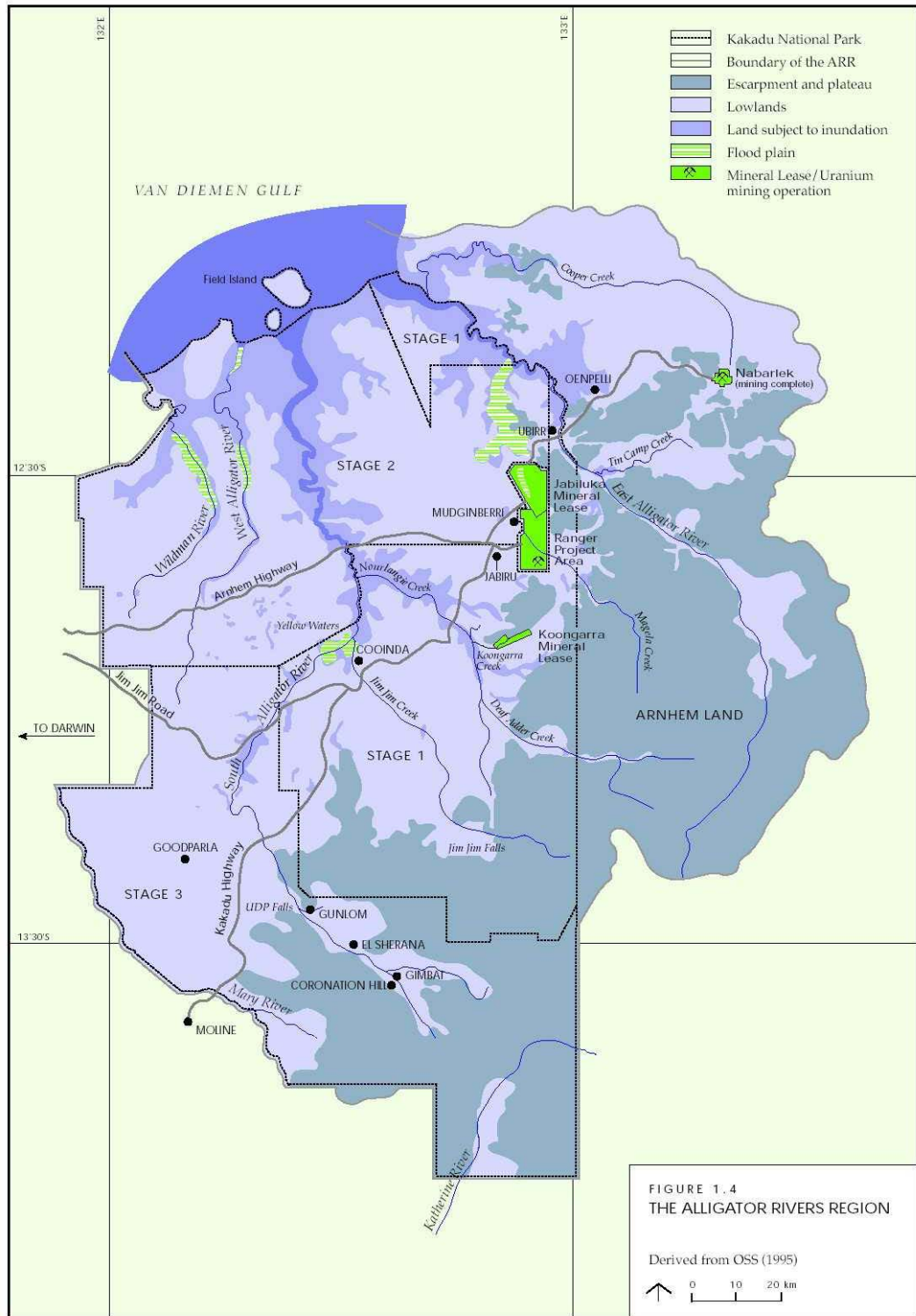
**Figure 2.1(b):   Map of Kakadu National Park and Uranium mining areas**

Apart from being a significant mineral province, the Alligator Rivers Region also possesses an extensive diversity of species and habitats as well as sites of cultural significance (Figures 2.2 to 2.4).

**Figure 2.2:   Cultural and natural heritage values of Kakadu National Park**



**Figure 2.3:   Kakadu Wetland-Magela Floodplain**



**Figure 2.4: Mt Brockman - Site of cultural significance**

These special values have been recognised by the listing of Kakadu National Park as a World Heritage site (Environment Australia, 2000). As reported by Environment Australia these values include:

Cultural Criteria:
   i.      represent a unique artistic achievement, a masterpiece of a creative genius; and
   ii.     be directly or tangibly associated with events/ideas/beliefs of outstanding universal significance.

Natural Criteria:

   i.      outstanding examples representing significant ongoing geological processes, biological evolution and man's interaction with his natural environment;
   ii.     unique, rare or superlative natural phenomena, formations or features or areas of exceptional natural beauty; and
   iii.    the most important and significant habitats where threatened species of plants and animals of outstanding universal value from the point of view of science and conservation still survive.

Although the Ranger Mining Leases are a separate enclave within the Park, the social license to conduct mining and milling operations in a region which is World Heritage listed and subject to seasonal extremes in rainfall is dependent on the application of best practice environmental management. In this context, the long-term security of disposed tailings and the effective closure of the tailings repositories are by far the most important components of the Ranger Mine decommissioning and rehabilitation plan.

 Ranger's statutory authorisation to operate also sets specific goals in relation to rehabilitation of the Project Area so that it can be eventually incorporated into Kakadu National Park without detracting from Park values. Specific requirements include the need to provide for a rehabilitated landscape that incorporates structures/repositories with a design life of 200 years and a structural life of 1000 years. The closure design must also take into account geomorphological elements and geochemical processes to prolong long-term structural integrity. The inclusion of these latter elements in the design of the mine closure plan warrants a pre-cautionary approach that affords the highest level of protection to the cultural and natural values of Kakadu National Park.

## 2.2  Regional Overview

Ranger is located in the Alligator Rivers Region of the Northern Territory, adjacent to the Arnhem Land Escarpment and surrounded by Kakadu National Park. The region covers an area of 20,000 km$^2$ and is bounded by the catchments of the East, South and West Alligator Rivers, Wildman River, and Cooper, Magela and Nourlangie Creeks as shown in Figure 2.1(b). The climate of the region is monsoonal with two distinct seasons; the wet season extends from November to April while the dry season, from May to September, is a period of drought. October and April tend to have occasional showers only. During the wet season rain arises from tropical monsoonal troughs and localised convection systems such as thunderstorms. Meteorological records at Ranger for the period 1971 to 1989, showed that the mean annual pan evaporation rate was 2530 mm/a (Coefficient Variance 6.6%). This exceeded the mean annual rainfall of 1480 mm/a (Coefficient Variance 20%) (Ranger Mine, pers. comm., 2000).

The regional geology is characterised by deeply weathered metamorphic rocks of the basement Nanambu Complex, which comprises of granite, gneiss and schist ranging in age from 2470 to 1800 Ma (Page et al., 1980). This sequence is overlain by the Lower Proterozoic Cahill Formation, consisting of quartz, schist, mica schist, para-amphibolite, calc-silicate and carbonate (Kendall, 1990). All significant uranium mineralisation including the Ranger orebodies occurs in this formation. Unconformably overlying the Cahill Formation is the Kombolgie Formation, which has an upper and lower sandstone unit separated by the Nungbalgarri Volcanic Member (Kendall, 1990). The Kombolgie Formation outcrops as the Arnhem Land Escarpment. Geomorphologically, the dissected lowland plains of the region, and in the vicinity of Ranger, are known as the Koolpinyah surface. This surface is Late Tertiary in age and is rarely more than 45 m above sea level (Page et al, 1980). It is dissected by braided creeks and low level alluvial plains which merge with coastal plains to the north as the coastline is approached. During the wet season, the creeks become expansive sheets of water that extend well beyond their banks. Conversely in the dry season, the creeks are reduced to a series of billabongs and swampy depressions.

## 2.3  The Ranger Mine Project

Energy Resources of Australia Limited (ERA) owns and operates the Ranger Mine. The Ranger Mine and associated leases occupy a combined project area of approximately 79 km$^2$ enclosing some nine radiometric anomalies. Although the project area is excluded from

Kakadu National Park, it does lie on Aboriginal land and hence is leased from the local Traditional Owners. To the north and adjoining the Ranger Project Area (RPA), is the 73 km$^2$ Jabiluka Project Area which is also leased by ERA. Figure 2.5 shows the areal extent of both the Jabiluka and Ranger mine sites.
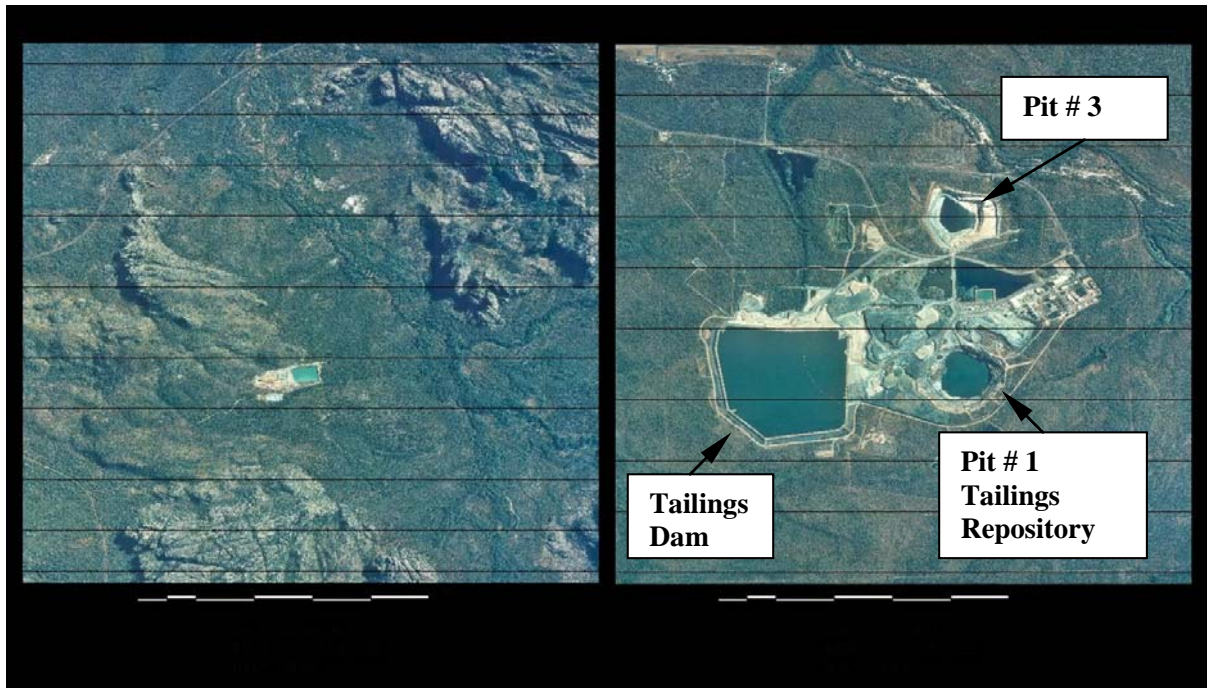


**Figure 2.5:   Aerial view of Jabiluka (left) and Ranger (right) Mine sites**

The Ranger operations consists of two orebodies, Ranger #1 and #3 that are mined by open pit methods. Total reserves were estimated at 166,300 tonnes of $U_3O_8$  (Savory, 1994). Processing of ore from Ranger #1 commenced in October 1981 and was continuously mined until 1994 when the economic ore was depleted. Ore is currently sourced from the Ranger #3 pit which has a projected mine life of 13 years that will end in 2008.

Fordham (1993) after Eupene et al. (1975) succinctly summarized the mine geology as being dominated by intense chloritisation with variable lithology. Rock types within the orebody include mica-quartz-feldspar schists subjected to varying degrees of chloritisation, magnesium or dolomitic carbonates, carbonaceous schists, gneisses, dolerites and pegmatites. The primary ore mineral in Ranger #1 is uraninite ($UO_2$ (s)). Secondary uranium minerals such as torbenite ($Cu(UO_2)_2(PO_4)_2 \cdot 8\text{-}12H_2O$) and saleeite ($Mg(UO_2)_2(PO_4)_2 \cdot 8H2O$) are enriched in a 20 m thick lateritic ore  zone (Savory, 1994).

Almost 50% of the gangue is magnesium chlorite, with an additional 40% comprising quartz and sericite. Petrographic analyses (Savory, 1994) show that ore minerals invariably occur within a matrix of fine compact chlorite. The primary ore also contains galena, localized pyrite, smaller quantities of finer chalcopyrite, dolomite, apatite, rutile and hematite. Table 2.1 summarizes the mineralogy of the primary ore, weathered ore and laterite for Ranger #1.

**Table 2.1:   Mineralogy of Ranger #1 (Savory, 1994)**

| Mineral | Primary Ore | Weathered Ore | Laterite |
|---|---|---|---|
|  | (% wt/wt) | (% wt/wt) | (% wt/wt) |
| Mg Chlorite 1 | 25.5 | 20.4 | 2.3 |
| Mg Chlorite 2 | 20.9 | 3.8 | 0.2 |
| Quartz | 37.3 | 42.8 | 53.1 |
| Sericite | 10.4 | 31.3 | 40 |
| Kaolinite | Absent | absent | 3.3 |
| Dolomite | 0.5 | < 0.04 | Absent |
| Apatite | 0.2 | 0.1 | Absent |
| Pyrite | 4.0 | 0.4 | < 0.02 |
| Rutile | 0.5 | 0.5 | 0.2 |
| Hematite | 0.1 | trace | Trace |
| Goethite | Absent | 0.3 | 0.6 |
| Chalcopyrite | 0.1 | < 0.01 | Trace |
| Galena | 0.1 | Trace | Absent |
| Plumbogummite | Absent | <0.01 | Trace |
| U minerals | 0.7 | 0.3 | 0.2 |

Natural water levels in the vicinity of Ranger #1 were relatively high but have progressively declined with pit excavation and dewatering. Typical dry season inflows, which are predominately through the upper weathered zone of Pit #1 (above sea level), are around 600 $m^3$/day. The fresh rock below the weathered horizon is relatively impermeable with the exception of some contact zones and fault planes (Savory, 1994).

## 2.4   Tailings Management

Tailings management incorporates a 107 ha tailings dam and the Ranger #1 (or Pit #1) void (Figure 2.5). The tailings dam has a capacity of 15 $Mm^3$ and currently stores approximately 13 $Mm^3$ of Ranger #1 tailings at an overall density of 1.09 t/$m^3$ (Ranger Mine pers.comm, 2000). Tailings are no longer pumped to the dam, as it is used to evaporate excess process water. Since December 1996, tailings from the milling of Ranger #3 ore and the remaining stockpiled Ranger #1 ore have been deposited into the Pit #1 void. At the end of mine life, the closure plan requires that all tailings be transferred to the Pit #1 and Pit #3 voids for final storage. Thus, the site of the existing tailings dam will be rehabilitated and restored such that the land can be incorporated back into Kakadu National Park. Careful design of the pit repositories and isolation of the tailings from surface and ground waters will be essential for the long-term protection of Kakadu's World Heritage values.

The tailings dam is a 1 $km^2$ ring dyke or turkey's nest impoundment located 500 m due west of the Ranger #1 pit. It lies across the head of a shallow valley which once formed the headwaters of Coonjimba Creek as shown in Figure 2.5 (Sinclair, 1992).

Tailings were deposited sub-aqueously in the tailings dam from the commencement of milling in September 1981 until late 1986. This practice was required to minimize the potential radiological hazards arising from radon gas emanation. During this period approximately 6.3 million tonnes of tailings were deposited into the dam at an overall density of around 0.8 t/$m^3$ (Sinclair, 1992). After it was determined that radon emanation from beached tailings was not a significant hazard, sub-aerial deposition of tailings commenced in the 1987 dry season. A ring main spigot system was installed to discharge tailings from the perimeter of the tailings embankment. This method of deposition resulted in the formation of freely draining beaches that increased the dry bulk density of the tailings to around 1.2 to 1.8 t/ $m^3$ (Fiedler and Pilgrim, 1993).

Particle size distributions of both sub-aqueously and sub-aerially deposited tailings (Fordham et al., 1993) showed significant sorting with materials grading from coarse silt/sand (20% passing 0.02 mm) in the upper tailings beaches to fine silt/clay (80% passing 0.02 mm) below the water level of tailings pond. These trends were also supported by *in situ* permeability measurements (Richards et al., 1990; Peter et al., 1994) which showed permeabilities ranging from $1 \times 10^{-8}$ to $4 \times 10^{-7}$ m/s with the higher values generally being observed in the coarse sub-aerially deposited tailings.

The main minerals in the solid phase of the tailings at Ranger are quartz, chlorite, muscovite and gypsum. Large particles of hematite have been observed by SEM, and other minerals found in minor amounts are pyrite, chalcopyrite, galena, uraninite and brannerite (Fordham, 1993). Sulfur is predominantly speciated as acid-soluble sulfates (Fordham, 1993).

Lime was used to neutralise acid tailings, usually to a pH just above neutral. Magnesite was also used as an alternative to lime (Fordham and Beech, 1989). The deposition of acid tailings was also trialled to improve settled density (Fordham and Beech, 1989; Richards and Peter, 1990; Fordham et al., 1992; Fordham, 1993; Fordham et al., 1993).

The chemical properties of pore waters recovered from within the tailings dam have been reported by Fordham (1993). Pore water samples were taken during installation of piezometers in the tailings during 1989. The main findings were that:

- Salt concentrations were high, being composed mainly of magnesium, ammonium and manganous sulfates;

- Salt concentrations in the pore waters near the base of the dam are relatively low, and gradually increase towards the surface, to a depth of 3 to 4 m (Fordham, 1993). Fordham attributed the gradient in salt concentrations to differences between the "mixing and diffusion from sediments" during aqueous and sub-aerial deposition. There is also an effect due to progressive evaporative concentration of exposed water on the surface of the tailings, recycling of water through the processing plant and entrapment or incorporation of that water within the pore spaces as the tailings rose;

- Vertical gradients in concentrations are sustained because rates of diffusion within the tailings are very low;

- A number of anaerobic microbiological processes may influence the composition of pore waters. Changes of pH with depth are directly related to microbial activity, specifically the production of $CO_2$ and $H_2S$ and subsequent precipitation of Fe and Mn carbonates and metallic sulfides; and

- There is an apparent decrease in temperature from 1 to 4 m below the surface of the tailings, but temperatures are relatively constant below that depth.

LeGras (1992) has also provided chemical analyses for one surface sample and seven pore water samples at 1 m depth intervals within the tailings. For most species, there was a marked decline in concentration with depth. The existence of gradients is seen as evidence of low vertical seepage rates and an indication that measured concentrations may be representative of those at the time of deposition.

## *Chapter 3:    Experimental Methodology and Materials*

### 3.1.    Field Sampling Program and Measurements

Access to sub-aqueously deposited tailings within the Ranger tailings dam was facilitated by a purpose built pontoon (Figure 3.1) mounted with a soft sediment piston core sampler. This device utilises a jack to provide sufficient downward thrust to continuously push a 500 mm long by 42 mm diameter class 12 UPVC tube through soft sediment to the desired sampling depth (Figures 3.2 and 3.3).



**Figure 3.1:   Sampling pontoon on the Ranger Dam 1998**





**Figure 3.2:   Pontoon and piston core sampler**          **Figure 3.3:   UPVC sampling tube**

The sampling tube is equipped with an internal piston that is retracted to draw tailings into the tube. Surface tension holds the tailings in the sampling tube as the core is withdrawn from the tailings pile. Schematics of the sampling rig are shown in Figure 3.4(a) and 3.4(b).
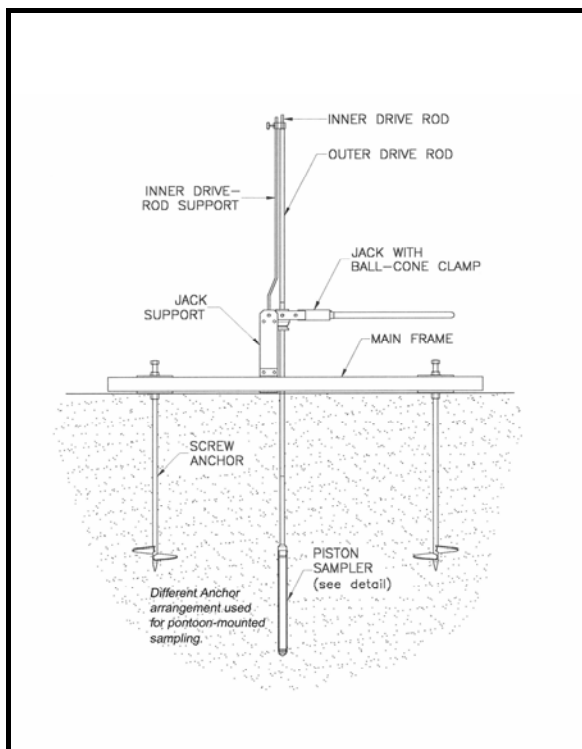


**Figure 3.4(a):  Schematic of soft sediment sampling rig**
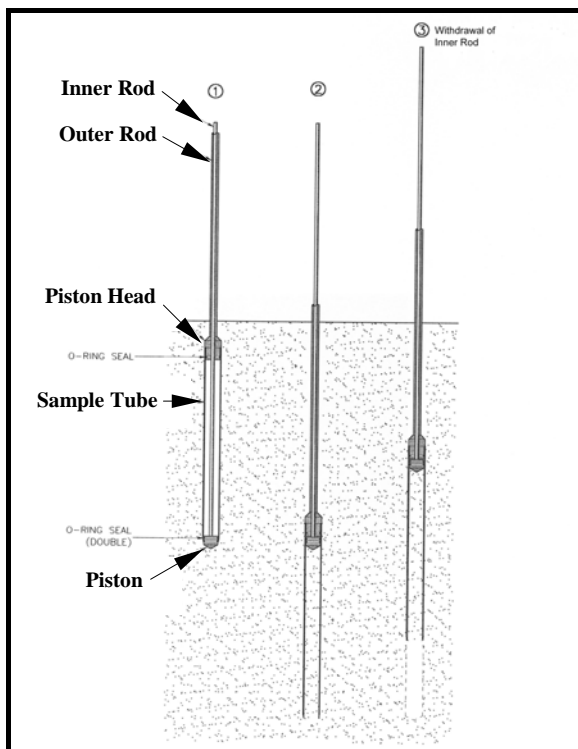


**Figure 3.4(b):  Schematic of piston core sampler.  Steps 1 to 3 show withdrawal of inner rod to affect sample collection.**

Immediately following sampling, the cores were sealed with refrigeration putty and rubber bungs for transportation to the laboratory. The seals and bungs held the cores firmly in place thereby reducing disturbance and the potential for exposure to the atmosphere. Porewater recovery from each of the tailings core tubes was achieved by pressurising the core to 250 kPa with high purity nitrogen. Following the extraction step, porewater samples were measured for pH, electrical conductivity (EC) and redox potential (Eh). Porewater pH was measured with a combination electrode calibrated with pH 4 and pH 7 buffers. These buffers were used for both field and laboratory samples as they best represented the range of anticipated pH values. Porewater Eh was also measured with a combination electrode standardised against Zobell's redox buffer (Light, 1972). Electrical conductance was measured with a conductance meter and probe following determination of the cell constant with KCl standards and was converted to $25^{o}C$ equivalent values. Porewaters were then prepared for cation, trace metal, radionuclide and anion/nutrient analyses in accordance with APHA methods (APHA, 1995) as summarised in Table 3.1.

**Table 3.1:   Sample preparation procedures**

| Parameter | Preparation |
|---|---|
| Trace metals/radionuclides | Filtered < 0.45 µm and acidified with $HNO_3$ to pH 1 to 2 |
| Cations | Filtered < 0.45 µm and stored at $4^o$C |
| Nutrients and anions | Filtered < 0.45 µm and frozen |

Following porewater extraction, the cores were split longitudinally with the aid of a circular saw and a 0.5 mm stainless steel wire. Each core was photographed, logged for texture and wet colour (McDonald et al., 1984) and sub-sampled for physical, geochemical and mineralogical examination. A composite sample of tailings solids was taken over the length of the core to enable direct correlation between the tailings geochemistry and pore water chemistry as shown in Figure 3.5. Solid sub-samples were also collected at discrete core depths, particularly where distinctive mineral phases and/or fresh precipitates were observed.



Groove remaining in the core after the composite sample was removed

**Figure 3.5: Composite sample removed from tailings core – Site 4 (9-9.5 m)**

The soft sediment sampler was also used to install standpipe piezometers to record the piezometric surface of the tailings pile and to collect in situ porewater samples. The design of the piezometers was adapted from criteria reported by Richards et al. (1989) and included a 16 mm diameter PVC tube (stand pipe) connected to a sintered stainless steel filter and cone tip. The stand pipe was threaded to allow the piezometer to be extended in the field to the desired depth. A 20 micron filter was used to minimise the potential for screen blinding and to maximise the ingress of porewater into the piezometer.

In-situ porewater was sampled by sealing the stainless steel filter and inserting a dip tube below the standing water level. A nitrogen purge of 10 to 20 psi forced the entrained

porewater up the dip tube to a collection bottle at the surface. Porewaters collected from the piezometers were immediately measured for pH, EC, Eh, and preserved for subsequent major ion, trace metal and radionuclide analyses. Standing water levels were measured with a conventional down hole electrical resistance probe. A detailed description of physical, mineralogical, chemical and analytical methods is provided in Appendix 1.

## 3.2   Solid State Speciation Test Work

A six step sequential extraction scheme was used to selectively dissolve and liberate radionuclides associated with the various mineralogical phases within the tailings. Extractants were primarily chosen for their ability to inhibit solute re-adsorption following dissolution of the target phase.  The scheme is summarised in Table 3.2 and is based on studies by Tessier et al. (1979), Edis et al. (1992) and Susksi et al. (1996).

**Table 3.2:   Sequential extraction scheme**

| Sequence | Extractant | Procedure | Target Phase |
|---|---|---|---|
| 1 | *De-ionised water | 7 g sample in 220 mL solution, shake 4 h @ 25°C  (2 extractions) | Evaporite Salts |
| 2 | Morgan's Solution | Residual from (1) in 220 mL of  1 M NaOAC adjusted to pH 5.0 with acetic acid, 4 h @ 25°C  (1 extraction) | Exchangeable ions and carbonate minerals |
| 3 | Tamms Acid Oxalate | Residue from (2) in 200 mL of 0.11 M Ammonium oxalate and 0.09 M oxalic acid, pH 3, in dark 4 h | Amorphous oxides of Al, Fe and Mn, secondary U minerals |
| 4 | CBD | Residue from (3) in 200 mL of 0.3 M Na-Citrate, 1.0 M $NaHCO_3$ and $Na_2S_2O_4$. 2 h @ 80°C. Boil liquor with $HNO_3$ and HCl to prevent sulfide ppt | Crystalline iron oxides, goethite and hematite |
| 5 | EDTA | Residue from (4) in 200 mL 0.11M $Na_2EDTA$ + 1.7 M $NH_4OH$, 4 h @ 95°C | Alkaline Earth Sulfates (Barite) |
| 6 | HCl | Residue from (5) in 200 mL 6 M HCl 2 h @ 85°C | Clay Minerals, primary U-oxides |
| 7 | Total Digestion | Residue from (6) in borax fusion | Resistant minerals, Quartz, Muscovite |

*Except for steps 1 and 2 the sample was rinsed with 100 mL of de-ionised water prior to each extractant.

At the end of the extraction procedure, the samples were centrifuged and the extractant decanted and analysed for trace metals and radionuclides. The final extraction residue together with the original tailings sample (pre-extraction) was subjected to a borax flux fusion to facilitate a trace metal/radionuclide mass balance.

Phase selectivity of each extractant was assessed by differential X-Ray Powder Diffraction (DXRD) analysis. DXRD patterns were obtained from samples after extraction treatments by subtracting the digital XRD reflections from samples measured before and after extraction.

## 3.3.    Kinetic Column Leach Studies

### 3.3.1    Columns

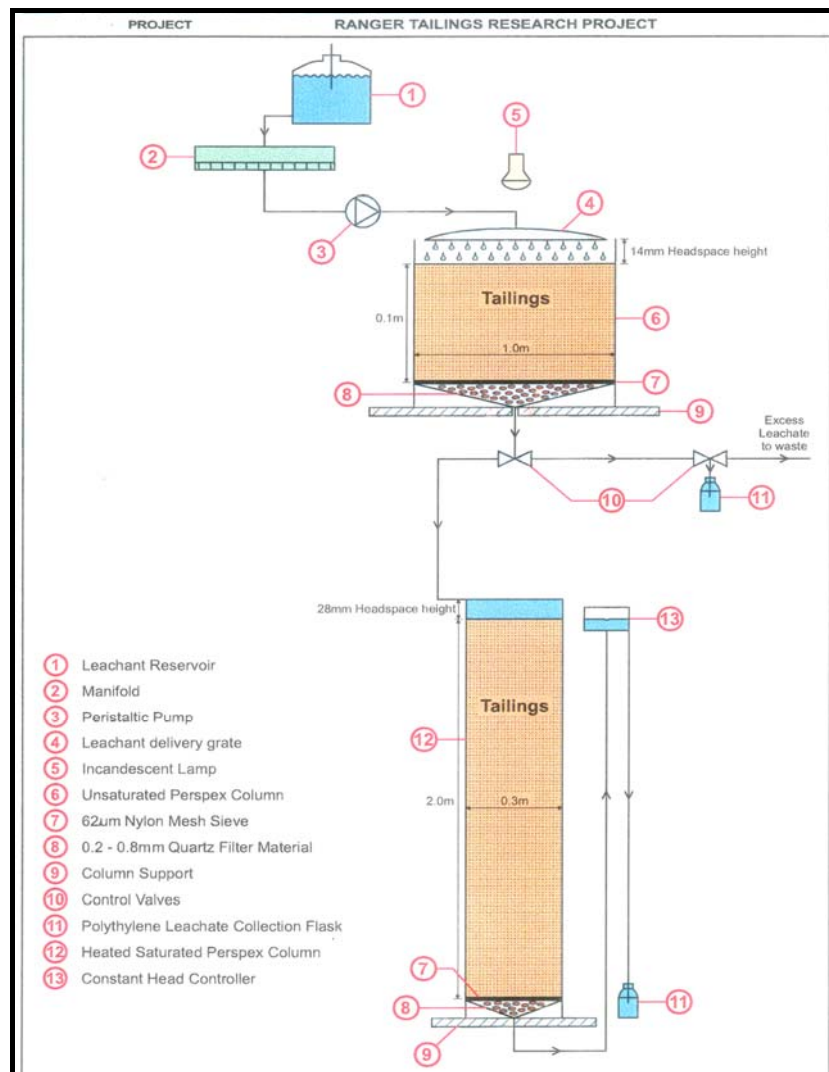Figure 3.6 shows a schematic representation of the leach column set-up.



**Figure 3.6:   General arrangement of leach column**

The columns were designed, constructed and operated within the Environmental Science Division of the Australian Nuclear Science and Technology Organisation (ANSTO) facility at Lucas Heights, Sydney. The design of the kinetic column study has attempted to account for climatic hydrological, chemical and depositional parameters that have a significant influence on the in-situ weathering processes in the tailings.

To achieve the desired design outcome, two columns were constructed. The upper column is designed with a high surface area per unit mass of tailings solids to ensure unsaturated leach conditions. The purpose of this column is to simulate leaching and exposure conditions similar to those that occur in the upper unsaturated zone (beaches) of the tailings dam. An understanding of unsaturated geochemical processes is necessary as the tailings were deposited sub-aerially over a ten year period (1986 to 1994). In the longer term, it is also conceivable that unsaturated zones may develop in the tailings repositories following final decommissioning and close-out.

The lower column is completely encapsulated to maintain a constant state of water saturation and to prevent atmospheric contact. The purpose of this column is to simulate leaching conditions deep within the tailings dam and Pit #1 repositories where the tailings are stored below the water table and isolated from the atmosphere.

The split column configuration was designed to primarily account for the depth of saturation in the tailings dam. Geotechnical surveys of the tailings dam by Richards et al. (1990) indicate that the standing water level lies approximately 0.5 – 1.0 m below the surface of the tailings. The depth of tailings at the northern or deepest end of the dam was approximately 18 m (as at June 1997); therefore the zone of saturation extends over a depth of 17 to 17.5 m.

In an attempt to closely approximate the depth ratio of unsaturated to saturated zones in the dam, a 2 m long column with a relatively narrow diameter (0.3 m) was selected to represent the saturated zone. In contrast, the upper unsaturated column was designed with a proportionately shallow flowpath (0.10 m) and a wide diameter (1 m). This configuration provided a high degree of surface exposure that was conducive to oxidative weathering processes.

Climatic design criteria were based on local meteorological records and included: the duration of the wet season within an annual seasonal cycle, the annual average rainfall, diurnal variation, ambient air temperature and rainwater quality (Table 3.3).

**Table 3.3:  Climatic data for Jabiru Airport**
**(Australian Bureau of Meteorology)**

| Parameter | Value |
|---|---|
| Period of record | 18 years (1971 to 1989) |
| Average annual rainfall | 1480 mm |
| Average annual rain days | 104 days |
| Average daily maximum temperature | 34.0 $^{o}$C |
| Average daily minimum temperature | 22.4 $^{o}$C |
| Average daily sunshine | 8.8 hours |

Based on the surface area (1.0 m$^2$) of the upper column, the annual volume of leachant required to simulate annual average rainfall is 1.480 m$^3$ or 1480 L. To account for the wet/dry seasonal cycle, the periodicity of leachant delivery closely approximated the number of rain days. Over the period of record, the average number of rain days is 104 per year. Accordingly, in an average year, 1,480 mm of rainfall occurred over a period of 104 days or 28% of the time. Therefore, to simulate an average annual wet/dry cycle, leachant was delivered at a rate of 22.3 L per week over a two-day period.

As the leachant rate for the column system is based on the surface area of the upper unsaturated column (1.0 m$^2$) there is potential to leach the lower saturated column (surface area 0.071 m$^2$) at a rate far in excess of the average annual rainfall. To overcome this problem, a three-way valve was installed between the two columns to control the leach rate into the lower saturated column. A leach rate of approximately 2.0 L per week over a two day period was required to simulate average annual rainfall. Although the exact infiltration rate within the unsaturated zone of the dam is not known, the proposed leach rate is the minimum volume required to satisfy the analytical program. The need to strike a balance between emulating field variables and experimental design criteria is a constraint with the column leach experiments.

The columns were kept in a temperature controlled room set at around 25°C. This closely approximates the average field temperature. An incandescent lamp was installed directly above the unsaturated column to simulate diurnal cycling. The lamp operated for 8.8 hours

per day. Both columns were fabricated from clear Plexiclas or Perspex to allow for visual inspection of the tailings throughout the leaching program. Specific details on the various column design parameters and material specifications are summarised in Table 3.4.

**Table 3.4:   Column design parameters and material specifications**

| Parameter | Unit | Upper Column | Lower Column |
|---|---|---|---|
| Length | m | 0.1 | 2.0 |
| Diameter/width | m | 1.0 | 0.3 |
| Surface Area | $m^2$ | 1.0 | 0.071 |
| Volume of tailings | $m^3$ | 0.1 | 0.142 |
| [1] Dry Bulk Density | $t/m^3$ | 1.1 | 1.1 |
| Specific Gravity | _ _ | 2.7 | 2.7 |
| Dry weight of tailings | kg | 110 | 156 |
| Porosity | _ _ | 0.57 | 0.57 |
| Pore Volume | L | 57 | 80.4 |
| [2] Leachant Volume | L/wk | 28.5 | 2.0 (increased to 22.5) |
| Headspace to contain daily leachant volume | mm | 14 | 28 |
| Annual Leachant Volume | L | 1482 | 105 (increased to 1170) |
| Ratio of leachant to mass of tailings | L/kg | 13.5 | 0.67 (increased to 7.5) |
| Annual Pore Volumes | _ _ | 26 | 1.3 (increased to 14) |

1.   Average dry bulk density reported for tailings dam (Ranger pers comm., 1996)

2.   Leachant delivered over a two day period, in the case of the lower column leachant rate increased to provide sufficient sample for analytical requirements

### 3.3.2   Leachant Quality

The chemical composition of the leachant was based on the composition of rainwater at Jabiru East (Noller et al., 1990). Rainfall in the Alligator Rivers Region has a pH consistently between 4.0 and 5.0 and also exhibits local concentrations of ions and free acidity associated with the presence of weak acids (Noller et al., 1990). Table 3.5 summarizes the chemical constituents of Jabiru East rainwater.

**Table 3.5:   Chemical composition of Jabiru East rainwater (1983-1984)**

| Parameter | Mean Concentration (µg/L) |
|-----------|---------------------------|
| pH | 3.5 – 4.9 (no units) |
| H | 54 |
| Na | 230 |
| K | 59 |
| Ca | 44 |
| Mg | 37 |
| $NH_4$ | 100 |
| Cl | 490 |
| $SO_4$ | 380 |
| $NO_3$ | 370 |
| $PO_4$ | 20 |
| $HCO_3$ | 12 |

To achieve these solute concentrations, a 1 L stock solution of artificial rainwater concentrate was made up by dissolving in Milli-Q water the salts of weak acids as summarised in Table 3.6. The concentrate was transferred to a 1 L polyethylene bottle and stored at 4°C until ready for use. A new concentrate was made up on a bi-monthly basis. On the day of the leaching event, a 10.5 mL aliquot of the rainwater concentrate was added to 14.2 L of Milli-Q water in a Nalgene bottle. This solution was allowed to equilibrate to 25°C and represented the simulated rainwater leachant. The pH of the leachant was adjusted to a range of 4 – 5 by the addition of $H_2SO_4$ and formic acid (Reid et al., 1993).

**Table 3.6: Quantities of salts and acids for the preparation of artificial rainwater**

| Chemical Composition | Quantity for 1 L Concentrate |
|---|---|
| $NaHCO_3$ | 0.0230 g |
| $NaH_2PO_4 \cdot 2H_2O$ | 0.0425 g |
| $NH_4NO_3$ | 0.6549 g |
| NaCl | 0.7652 g |
| KCl | 0.1525 g |
| $CaCl_2 \cdot 2H_2O$ | 0.2205 g |
| $MgSO_4 \cdot 7H_2O$ | 0.5042 g |
| $H_2SO_4$ (18 N) | 175 µL |
| $HCO_2H$ (concentrated) | 2.06 mL |

### 3.3.3 Sample Collection, Preparation and Analysis

Representative samples of aged and fresh tailings were collected from the tailings dam and Ranger mill, respectively. The ore type being milled and process conditions were recorded at the time of sampling. Aged tailings were collected from the centre of the dam. All tailings were

transported to ANSTO in polyethylene lined 250 L drums. The tailings were homogenised and loaded into the columns via a slurry pump. A sub-sample of the homogenised tailings was collected and tested for:

- Physical properties (water content, particle size distribution, bulk density, permeability);

- Mineralogy (XRD, SEM-EDX);

- Geochemistry including major and minor elemental composition (XRF), Radionuclides (U, Ra); and

- Porewater chemistry including major ions (Na, K, Mg. Ca, Mn, $SO_4$) by ICPAES, anions and ammonia ($Cl^-$, $NO_3^-$, $NH_4^+$) by ion chromatography, trace elements (Si, Al, Fe, Ba,

As, B, Be, Cd, Co, Cr, Cu, Li, Mo, Ni, Sb, Sr, Tl, U, Zn, Pb, Ga, Y, Th, Sc, Ge, Rb, Cs, La, REE) by ICPAES/ICPMS and radionuclides ($^{226}$Ra, $^{210}$Pb and $^{230}$Th) by alpha and beta spectrometry.

The mineralogical and geochemical test program was repeated on the leached tailings (unsaturated and saturated) following column decommissioning at the end of the 520 day study period.

Leachate from the tailings and blank column was collected in polyethylene containers in equilibrium with the atmosphere. Following a leaching event, the samples were immediately measured for pH, EC, Eh, temperature and dissolved oxygen. The volume and weight of each sample were also recorded. The sample was then split into two fractions, unfiltered and filtered. The unfiltered samples were refrigerated at 4°C and used to assay major ions (Na, K, Mg, Ca, NH$_4$, Cl, SO$_4$, PO$_4$, NO$_3$, HCO$_3$).  The second fraction was filtered through a 0.45 µm membrane filter and the filtrate acidified with HNO$_3$ to pH 1.5 to 2.0. This fraction was assayed for dissolved trace metals and radionuclides. The non-filterable residue was weighed and stored at 4°C. To estimate the geochemical mass balance of the column system, total metals/radionuclides in the leachate were assayed for composite samples.

## 3.4    Geochemical Modelling

The geochemical speciation program HARPHRQ was used to model and describe the complex physico-chemical reactions arising from the interaction of porewater and minerals within the tailings pile. HARPHRQ was developed by Brown et al. (1991) and is based on the geochemical code PHREEQE that was written by the U.S. Geological Survey (Parkhurst et. al., 1980). HARPHRQ uses both the measured solute concentrations and thermodynamic data to predict the distribution and activities of aqueous species at chemical equilibrium. The code also calculates the degree of saturation of the aqueous phase with respect to associated mineral phases.

The kinetic modelling code STEADYQL was used to describe the weathering of phyllosilicates and oxidative dissolution of metal sulfides in unsaturated tailings.

Details of the equilibrium and kinetic models are discussed in Chapter 7.

## *Chapter 4: Field Investigations and Summary of Key Observations*

### 4.1 Field Sampling Campaign

This chapter summarises the key aspects of the field investigation program with respect to site layout, observations, sample descriptions and measurements. Detailed analyses and interpretation of both field and laboratory mineralogical, geochemical and hydrochemical data are provided in Chapter 5.

Two sampling campaigns were conducted in the tailings dam, one in January 1998 to collect undisturbed tailings cores and the second in June 1999 to install and collect in-situ porewater samples. Data obtained from these campaigns allowed for a detailed evaluation of vertical and lateral hydrogeological and hydrochemical profiles within the tailings impoundment. Sample locations for both campaigns are shown in Figure 4.1. Vertical cross-sections of core profiles and piezometer installations are shown in Figures 4.2 and 4.3, respectively.
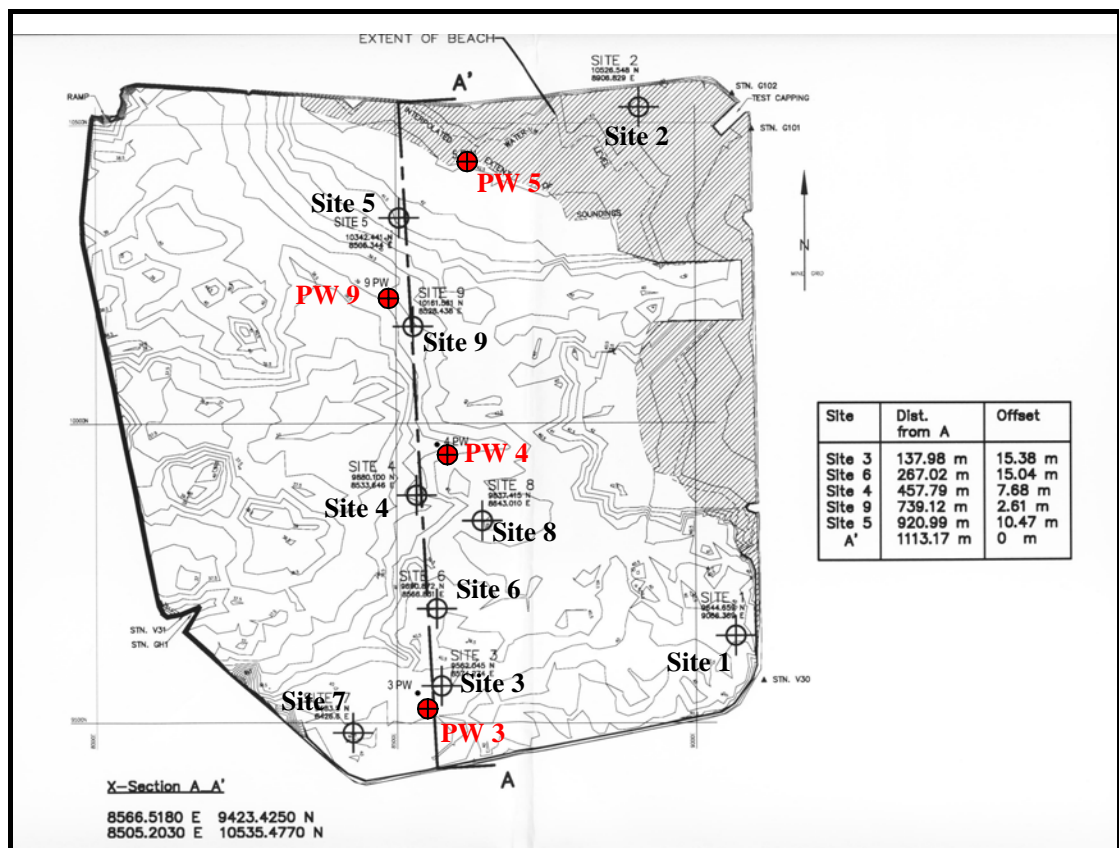


**Figure 4.1: Tailings dam sample profile locations**
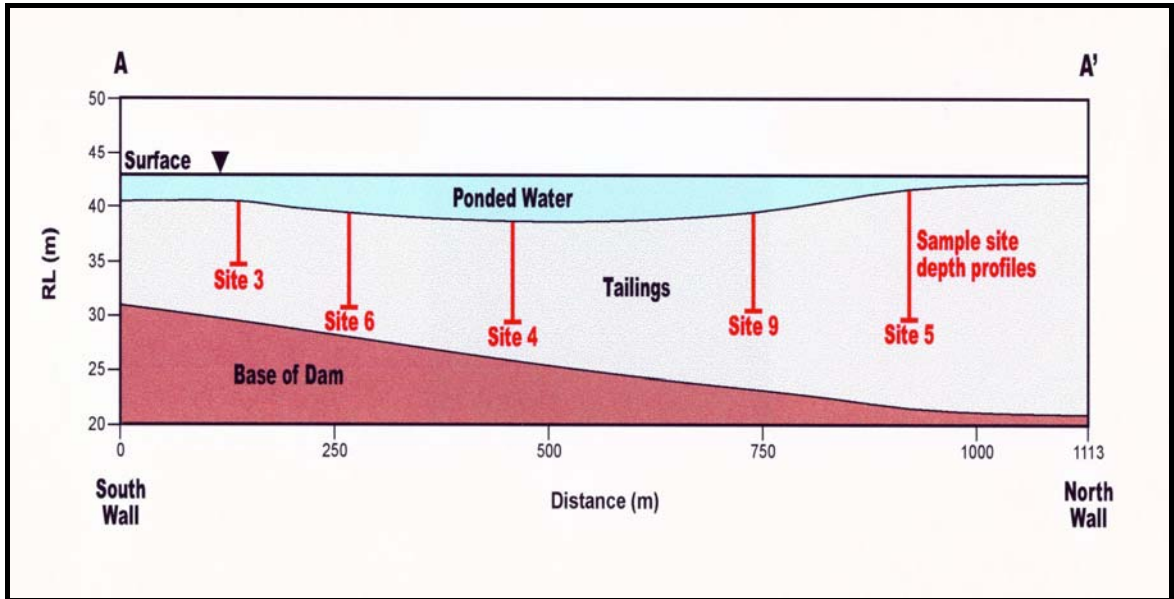
**Figure 4.2:   Cross section A-A' tailings core profiles January, 1998**
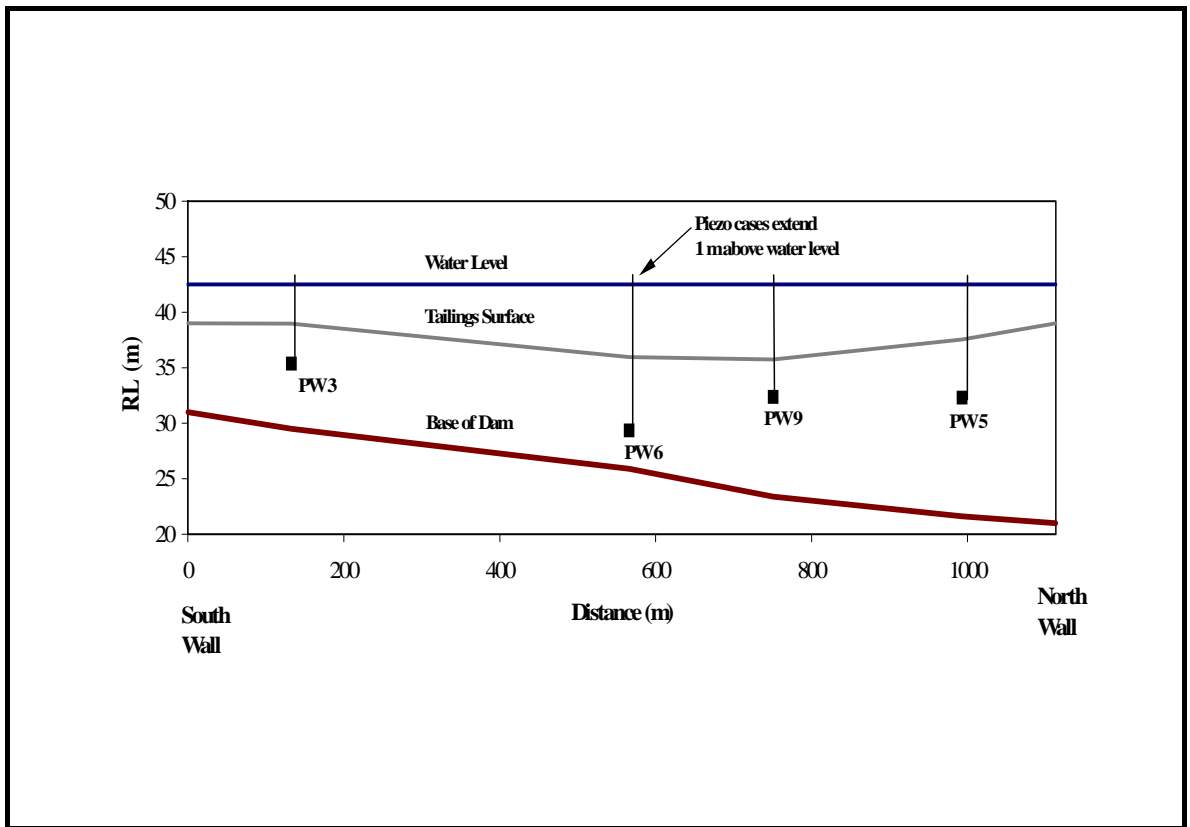


**Figure 4.3: Cross section A-A$^{'}$ piezometer profiles June, 1999**

Relative levels (RL) for the pond water surface, piezometer/core profile depths and tailings surface were obtained from on-site surveys conducted at the time of sampling and from pre-

tailings dam contour data (Figure 4.4). The relative levels for the base of the dam were interpolated from contour plans that were developed prior to construction of the tailings dam.
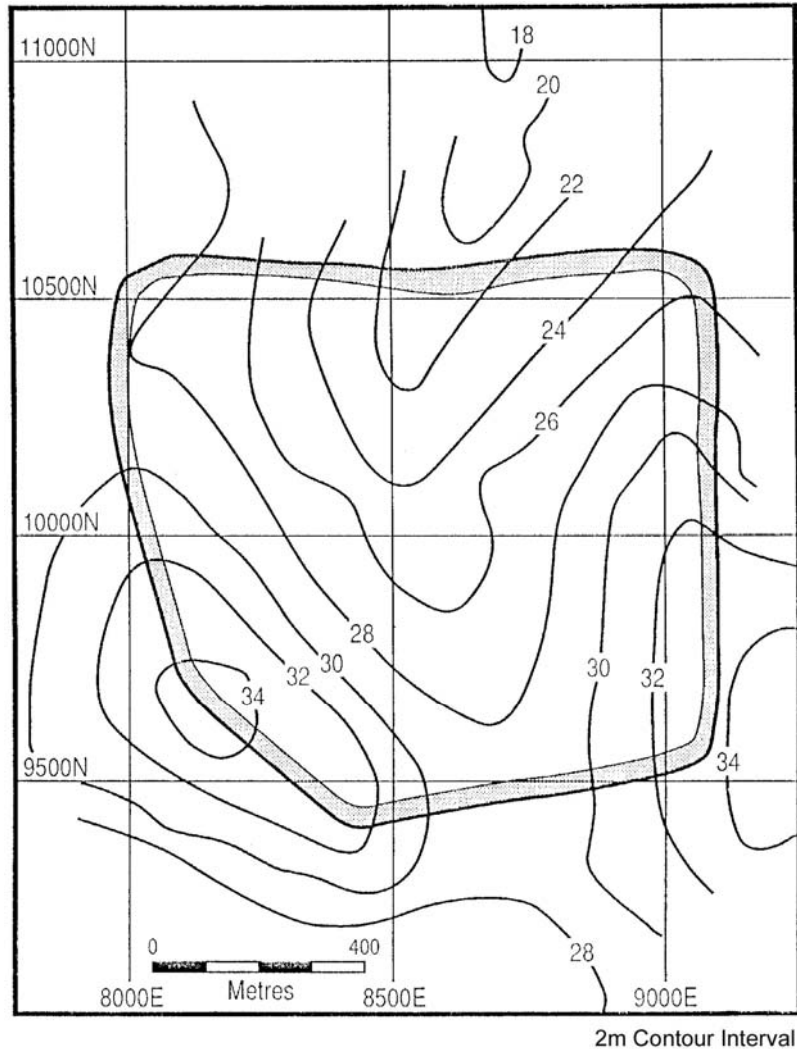


**Figure 4.4:   Original ground contours - pre-tailings dam**

## 4.2    Core Profiles

Tailings cores were successfully retrieved from five sampling sites (3, 4, 5, 6, and 9) located along a 1.1 km N-S traverse as represented by section A-A' in Figures 4.1 and 4.2. Table 4.1 provides a summary of  the tailings cores retrieved from each site and their respective sample depths (RL). All sites are representative of subaqueously deposited tailings that were collected in January 1998, following a major dredging campaign in the tailings dam during 1997. During this period, tailings solids were transferred via dredge from the tailings dam to Pit #1 to maintain statutory free board criteria.

**Table 4.1:  Summary of sample depths and relative level (RL) for each core profile**

| Site # | Core depth below water level (m) | RL Core midpoint (m) | Site # | Core depth below water level (m) | RL Core midpoint (m) |
|--------|-----------------------------------|----------------------|--------|-----------------------------------|----------------------|
|        |                                   |                      |        |                                   |                      |
| 3      | 4-4.5                             | 39.5                 | 6      | 4-4.5                             | 38.6                 |
|        | 5-5.5                             | 38.5                 |        | 5-5.5                             | 37.8                 |
|        | 6-6.5                             | 37.5                 |        | 6-6.5                             | 36.9                 |
|        | 7-7.5                             | 36.5                 |        | 7-7.5                             | 36                   |
|        | 8-8.5                             | 35.6                 |        | 8-8.5                             | 35.2                 |
|        | 9-9.25                            | 34.6                 |        | 9-9.5                             | 34.2                 |
|        |                                   |                      |        | 10-10.5                           | 33.4                 |
| 4      | 7-7.5                             | 37.6                 |        | 11-11.3                           | 32.6                 |
|        | 8-8.5                             | 36.4                 |        | 12-12.5                           | 31.7                 |
|        | 9-9.5                             | 35.2                 |        | 12.5-13                           | 30.8                 |
|        | 10-10.5                           | 34                   |        |                                   |                      |
|        | 11-11.5                           | 32.8                 | 9      | 6-6.5                             | 38.3                 |
|        | 12-12.5                           | 31.7                 |        | 7-7.5                             | 37.2                 |
|        | 13-13.5                           | 30.5                 |        | 8-8.5                             | 36.1                 |
|        | 14-14.5                           | 29.3                 |        | 9-9.5                             | 34.9                 |
|        |                                   |                      |        | 10-10.5                           | 33.8                 |
| 5      | 2-2.5                             | 40.6                 |        | 11-11.5                           | 32.6                 |
|        | 3-3.5                             | 39.8                 |        | 12-12.5                           | 31.5                 |
|        | 4-4.5                             | 38.9                 |        | 13-13.5                           | 30.3                 |
|        | 5-5.5                             | 38                   |        |                                   |                      |
|        | 6-6.5                             | 37.1                 |        |                                   |                      |
|        | 7-7.5                             | 36.2                 |        |                                   |                      |
|        | 8-8.5                             | 35.2                 |        |                                   |                      |
|        | 9-9.5                             | 34.4                 |        |                                   |                      |
|        | 10-10.5                           | 33.4                 |        |                                   |                      |
|        | 11-11.5                           | 32.5                 |        |                                   |                      |
|        | 12-12.5                           | 31.6                 |        |                                   |                      |
|        | 13-13.5                           | 30.7                 |        |                                   |                      |
|        | 13.5– 14.0                        | 29.8                 |        |                                   |                      |

Sample sites 1 and 2 were part of an earlier reconnaissance survey to establish sampling methodology and to test the prototype piston core sampler. Cores were logged but given the relatively shallow sampling depth they were not considered to be representative of the tailings profile and thus excluded from the research program.

Sample sites 7 and 8 were abandoned following poor core recoveries. Foreign objects in the tailings dam such as discarded 200L drums or pipes are thought to have obstructed core insertion and recovery at these locations. For the remainder of the sample sites (3, 4, 5, 6 and 9) volume based core recoveries were typically on the order of 85-95%. Sample sites 4 and 6 were cored to within 3 m of the base of the dam (Figure 4.2) and as such provide a representative cross section of the tailings pile. Although the point of refusal varied for each site, it was generally found that coring beyond a depth of 14.5 m from a floating pontoon was beyond the design limitations of the piston core sampler.

Stiff sand layers that were presumably confined between layers of lower permeability fines also significantly hindered coring. Insertion of the corer into these zones released significant pore pressures, which were observed as artesian flows from the top of the sampling rods (Figure 4.5).



**Artesian flow from depth**

**Figure 4.5:   Artesian pore pressures at Site 9,
(8 – 8.5 m)**

## 4.3    Piezometer Installations

Twelve multi-level bundle type piezometers were installed in the vicinity of the tailings core sample sites. Piezometer depths are shown in Figure 4.3.

Insertion of the piezometers to the desired sampling depth was relatively easy as the tailings liquefied around the piezometer pipe thereby providing little resistance to the force of downward thrust. Following installation, the tailings quickly settled around the piezometer standpipe.

Strict sampling protocols were developed to ensure that porewater recovered from the piezometers was representative of the tailings pile. Prior to sampling, the tailings piezometers were pumped dry to remove stagnant or aged water that had accumulated in the standpipe. The porewater sample was collected following the recovery of the water level to ensure that the sample was representative of porewater in the immediate vicinity of the piezometer screen.

The volume of recoverable porewater was generally limited to the standing water column in the piezometer due to the low permeability of the tailings and the corresponding slow recovery rate of water levels in the piezometers. For these reasons, recovered porewater volumes were variable but within the range of 100 to 800 mL.

Upon extraction, the porewaters were measured for a suite of general parameters as summarised in Table 4.2. Given the spatial paucity of piezometer porewater samples, it was difficult to delineate *in situ* hydrochemical trends and as such these data were used to augment and cross check the more spatially representative hydrochemical results obtained for the tailings core profiles. An interpretative overview of hydrochemical results and trends throughout the tailings pile is discussed in Chapter 5.

Profile temperatures ranged between 29 to $31^{o}C$, which are slightly higher than the controlled temperature ($25^{o}C$) used in the kinetic column experiments. This small temperature variation is not expected to materially impact on reaction kinetics or on the predictive capacity of the columns to simulate *in situ* porewater – solid phase interactions.

**Table 4.2:   Summary of porewater field measurements**

| Site & piezo # | *RL tailings surface | *Screen depth RL(m) | *Piezo SWL RL(m) | pH | Eh (mV) | EC (mS/cm) | Temp °C |
|---|---|---|---|---|---|---|---|
| 3PW | 39.0 | | | | | | |
| 1 | | 37.4 | 41.4 | n/a | n/a | n/a | n/a |
| 2 | | 36.4 | 41.3 | 8.04 | -58 | 29.6 | 30.4 |
| 3 | | 35.4 | 41.1 | 8.32 | -6 | 29.8 | 31.0 |
| 4PW | 35.9 | | | | | | |
| 1 | | 35.2 | 41.4 | n/a | n/a | n/a | n/a |
| 2 | | 34.4 | 41.4 | 6.97 | 29 | 22.9 | 29.4 |
| 3 | | 29.4 | 41.5 | 7.17 | 224 | 12.1 | 30.8 |
| 5PW | 37.5 | | | | | | |
| 1 | | 35.4 | 39.4 | 5.36 | -12 | 21.8 | 29.6 |
| 2 | | 33.4 | 41.3 | 5.9 | 77 | 25.7 | 30.2 |
| 3 | | 32.3 | 39.7 | 6.98 | 98 | 23.2 | 30.7 |
| 9PW | 35.8 | | | | | | |
| 1 | | 35.3 | 41.9 | n/a | n/a | n/a | n/a |
| 2 | | 34.3 | 41.8 | 7.36 | 186 | 11.32 | 31.5 |
| 3 | | 32.4 | 41.6 | 7.98 | 166 | 10.35 | 29.8 |

*RL refers to relative level and is referenced against a local mine datum

 n/a:  sample not available

RL of the ponded or tailings surface water at the time of sampling was 42.5 m

The variation in piezometric water levels with piezometer screen depth is shown in Figure 4.6, together with previous survey results reported by Richards et al. (1990).
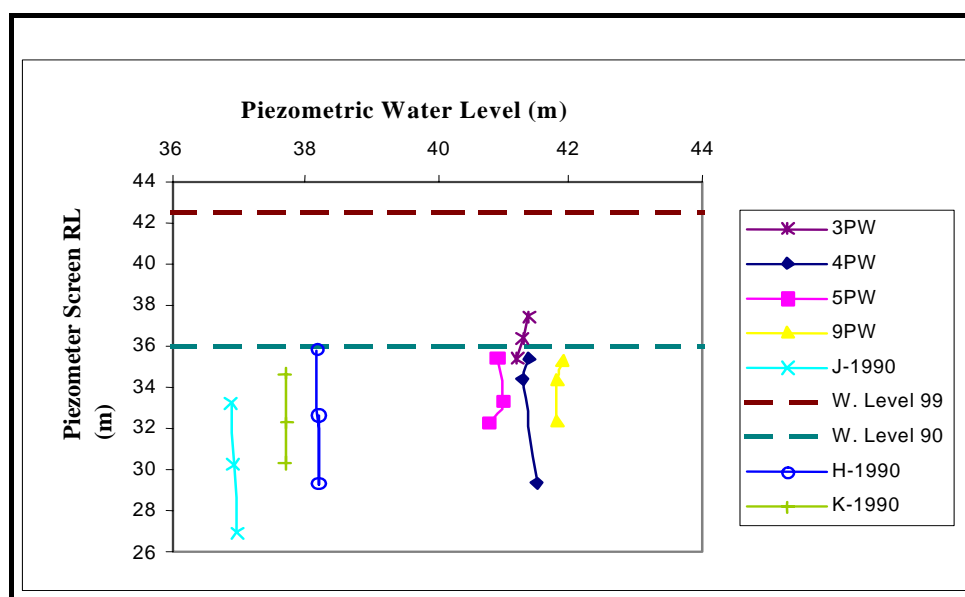


**Figure 4.6:   Comparison of 1990 and 1999 piezometric levels**

The survey results for both 1990 and 1999 (this study) are similar and confirm the existence of a hydrostatic gradient with little variation in piezometric water levels down the tailings depth profile. Combined with relatively low intrinsic permeabilities (Richards et al., 1990), these trends suggest there is little vertical movement of porewater or self weight consolidation within the tailings pile.