

CHAPTER 2

LITERATURE REVIEW

2.1 INTRODUCTION

The objective of this chapter is to present a literature review of tailings dam design and construction in general, as well as a critical review of the engineering properties of gold tailings. A brief history of the South African gold mining industry is given with emphasis on the geology and mineralogy of the two principal gold producing areas: the Witwatersrand Gold Fields and the Barberton Gold Fields. This is followed by a detailed examination of the physical and chemical processes employed to recover the gold valuables, which produce the tailings by-product to be disposed of. The design and construction of tailings impoundments are considered, highlighting the differences in philosophy between the mining companies and regulatory authorities. Alternative layouts and construction methods are discussed and it will be seen that the most popular arrangement used in South Africa, although economical, is also the least desirable from a safety point of view. The chapter ends with a comprehensive review of the mechanical properties of gold tailings, laying the groundwork for the rest of the thesis.

2.2 A BRIEF HISTORY OF THE GOLD MINING INDUSTRY IN S.A.

2.2.1 Introduction

The earliest documented discoveries of gold were made in river beds or shallow excavations in ancient Egypt, Russia and later during the gold rushes in California, Australia and Alaska. However, it was South Africa that became the scene of the greatest gold find of all - the gold-bearing Witwatersrand reefs which constitute the largest known deposits of gold in the world. The discovery of gold in South Africa has had a profound effect on the development of this country, not only in sustaining a precious metal based economy, but also in providing job opportunities for millions and supporting a vast range of subsidiary industries. South Africa has been the world leader in gold production for many years and at stages produced more gold than all the other countries combined. The following presents a short chronology of significant events in the gold mining industry in South Africa as reported by Stanley (1987):

- 1806 The Secretary Governor of the Cape announces the discovery of gold between the Witwatersrand and the town of Magaliesberg, 154 years after Jan van Riebeeck and the V.O.I.C. "Vereenigde Oost-Indische Compagnie" sailed around Cape Point.
- 1836 Reports are heard of gold mining activities in the Soutpansberg in the far north of the country.
- 1850 Cape farmers discover gold nuggets in the Gamka river.
- 1868 Gold is discovered in the Olifants river and Murchison range in the Transvaal.
- 1871 Organised mining of the Natalia reef starts on the farm Eersteling in the northern town of Pietersburg.
- 1872 Low veld alluvial gold deposits are found in the Sabie-Pilgrim's Rest area. A town rises out of the dust and becomes a mecca for gold diggers, panners and swindlers alike.
- 1874 Alluvial gold is found near the town of Magaliesberg on the West Rand.
- 1875 Alluvial and vein gold is discovered near Barberton on the East Rand and leads to the Barberton Boom from 1877 to 1883.
- 1881 First discovery of gold quartz veins at Kromdraai.
- 1884 Vein gold in the Confidence Reef west of Johannesburg on the Central Rand leads to the discovery of the greatest gold strike in history - the legendary Witwatersrand Gold Reefs.
- 1886 The highly profitable Main Reef of the Witwatersrand is first discovered in March 1886 as an exposed conglomerate.
- 1891 Mining is established along the outcrop reefs of the Witwatersrand.
- 1894 East Rand Proprietary Mines (ERPM) the oldest operating mine, declared gold in September 1894 and has produced roughly 1,500 tons of gold at an average grade of 7,23 grams per metric ton to date.
- 1932 Discovery of the continuation of the Witwatersrand Reefs on the West Wits Line leads to the establishment of the country's wealthiest and deepest mines on the far West Rand.
- 1939 Basal Reef becomes a major contributor of gold in the Orange Freestate.
- 1952 By this time the broad outline of the Witwatersrand Basin had been established as outcrops and "Shallow" Reefs stretching from Germiston to Randfontein comprising the East, Central and West Rand gold fields, from which the reef dips south and stretches to the "Deep" Reefs of the West Wits Line near Carltonville, and eventually into the Freestate goldfields.
- 1977 With the development of more effective metallurgical processes, the recycling of old slimes dams and waste dumps becomes popular.

A number of large mining houses have been established in South Africa to co-ordinate mining activities with co-operative organisation from the Chamber of Mines of South Africa,

which also functions as a spokesman for the industry. The first Chamber of Mines was formed on 7 December 1887 in the Central Hotel in Johannesburg and was succeeded by the modern Chamber on 5 October 1889. The establishment of a number of significant mining houses soon followed:

1887 Gold Fields of South Africa (GFSA), operates mainly in the far West Rand.

1893 Rand Mines, specialises in deep level mining and once controlled 15% of world richest mines.

1895 Durban Roodepoort Deep, was formed to exploit the first payable gold deposit discovered on the farm Langaagte on the western side of the Witwatersrand in 1886 near what is now called the town of Roodepoort.

1895 General Mining Corporation (GENCOR), widely diversified today.

1917 Anglo American Corporation, active in the East Rand, West Rand and Orange Freestate, has recently split into a number of subsidiaries of which Anglo Gold is responsible for gold mining operations.

1933 Anglovaal Ltd, has active gold mining interests in the Free State.

2.2.2 Terms and Definitions

Terms and definitions used in the mining industry for the various components associated with hard rock mining and tailings, which are useful for the purposes of this study, include (Truscott, 1923; Gilchrist, 1989, Cowey, 1994):

- **Country Rock:** The rock bordering and containing the gold bearing reef, typically quartzite, sandstone, granite, rhyolite, andesite, etc. In South Africa the Country Rock is mostly shales and clays.
- **Effluent:** Liquid fraction of the tailings slurry or pulp with soluble chemicals.
- **Gangue:** Gangue, consisting of minerals associated with the gold in the ore and country rock, constitutes the valueless portion to be removed and disposed of. Gangue minerals associated with gold ore are generally non-metalliferous and may include quartz (SiO_2), calcite (CaCO_3), silver (Ag) alloyed with the gold, pyrites (FeS_2), arsenopyrite (FeAsS) and chalcopyrite (CuFeS_2).
- **Ore:** Metalliferous rock from which metal or metallic compounds are extracted as valuables. The Witwatersrand gold ore, or Banket as it is known, consists of consolidated pebbles and gravels fast cemented in a quartzite matrix, which is hard and tough.
- **Ore-mineral:** The valuable portion of the ore typically gold, uranium, platinum, coal etc. In all cases in South Africa, gold is disseminated extremely finely within the ore, nuggets being very rare.
- **Pulp density or Solids Concentration (S_c):** The ratio of the mass of the solids to the mass of the total slurry, or $S_c = 1/(1 + w)$, where the moisture content, w , represents

the ratio of the mass of water to the mass of solids. Pulp densities can range between 15 and 55% in tailings, but is usually between 40 and 50%.

- **Rate-of-Rise (ROR):** Term used to describe the rate of increase in height with time as deposition proceeds on a tailings impoundment, usually in meters per year. Rate-of-rise is the single most important construction related factor controlling impoundment stability.
- **Tailings Sands:** Fraction by weight coarser than 75 μm , but for the purposes of this thesis measured at 63 μm .
- **Tailings Slimes:** Fraction by weight finer than 75 μm , but for the purposes of this thesis measured at 63 μm .
- **Tailings:** The by-product of the extraction process, tailings consist of finely ground and chemically treated rock flour in a slurry with process water. Tailings in South Africa are usually disposed of in perimeter dyke surface impoundments or slimes dams. Occasionally it is put to use as underground backfill for mined out areas.
- **Waste Rock:** Mostly country rock extracted in developing access to the reefs. These rocks do not enter the metallurgical works and are usually disposed of in large rock dumps.

2.2.3 Geology and Mineralogy of the Witwatersrand gold fields

The sediments of the Witwatersrand Goldfields or Triad were laid down between 2.7 and 3 billion years ago in a large basin south of Johannesburg and are derived from the surrounding Archaean granite-greenstone terrains (Stanley 1987). The deposits lie in an oval area of approximately 42,000 km² in Gauteng, North-West Province and the Free State Province. The gold was originally introduced in the deposit as detrital particles that underwent the low grade metamorphism which led to re-crystallisation. The Witwatersrand reefs are the largest contributor of gold in the world. More than 40 million kilograms of fine gold have been recovered up to 1985, with an average grade in all reefs mined until 1962 of 8.74 g/t. Undoubtedly, many more kilograms have been recovered to date, but at reduced yields, as the richest areas have become mined out.

Throughout the Witwatersrand, gold ores occur in sheets or reefs originally deposited horizontally under water. The reefs were subsequently covered by material up to thousands of meters deep. Following the consolidation and cementation of these layers, geological movements transformed it into tilted and faulted strata. The thickness of the reefs ranges between a line of grit to several meters, with an average of 300 mm. The sediments were also intersected by dykes and sills of dolerite, diabase and syenite intruding existing faults.

The reefs can be in the form of either coarse conglomerates or, less frequently, greyish metamorphosed sedimentary rock formations. In the conglomerates, rock pebbles are cemented in a silicate matrix. Pebbles, usually derived from vein quartz, may also consist of quartzite, chert jasper and quartz porphyry and vary in composition, size and colour. The matrix consists of pure silica, but also contains minute flakes of muscovite and pyrophyllite as well as visible pyrite and other sulphides. Table 2-1 summarises the mineral composition of a typical gold reef on the Witwatersrand.

Table 2-1: Mineral composition of a typical Witwatersrand gold reef (Stanley, 1987).

Mineral	Abundance
Quartz (SiO ₂), primary and secondary	70 - 90%
Muscovite and other phyllosilicates	10 - 30%
Pyrites	3 - 4%
Other sulphides	1 - 2%
Grains of primary minerals	1 - 2%
Uraniferous Kerogen	1%
Gold	~45 ppm in the Vaal Reef

The gold is usually confined to the matrix and tends to be concentrated along bedding planes. Visible gold is relatively rare. Table 2-2 presents a summary of the major types of gold to be found in the Witwatersrand reefs.

Table 2-2: Major types of gold occurrences in the Witwatersrand reefs.

Type of Gold	Relative Abundance
Detrital gold	Up to 90% of total gold in some reefs occur as clusters of grains
Biochemical redistributed gold	An important source if present
Metamorphosed gold	5 - 40% depending on local conditions
Primary gold in allergenic sulphides	Mainly pyrite, less than 2%
Gold in secondary quartz veins	Extremely rare but highly concentrated
Surface outcrops	Highly weathered, oxidised and friable
Other minerals of economic interest	Silver, Pyrite (Sulphuric acid), Uranium, Platinum and Sulphides

2.2.4 The Barberton gold fields

The Barberton gold fields are situated in a volcanic sedimentary greenstone belt of Archaean granite-greenstones in the Mpumalanga and Swaziland low-veld. The gold is largely found in refractory ores containing significant amounts of sulphide minerals, such as pyrite. Gold is also recovered from quartz veins and weathered and oxidised outcrops that have been enriched by the removal of the sulphides. In the refractory ores gold extraction is adversely affected by cyanide-consuming sulphide minerals, the presence of coarse gold particles which require extended contact for dissolution as well as by impervious coatings (iron oxides) on the gold particles which hinder contact with the leachate. Economical extraction of gold from these ores requires very fine grinding and roasting (calcining) to oxidise the iron sulphide minerals and thus expose the gold.

2.3 THE GOLD EXTRACTION PROCESS

2.3.1 Winning mineral ores

The winning of mineral ores can be summarised as: drilling, blasting and moving broken ore to the mill. When gold reserves are discovered, usually by diamond core drilling, a shaft is sunk in an optimal location. From the main shaft horizontal development tunnels are advanced at different levels to intercept the dipping ore bearing reefs. Winzes (connecting tunnels) are then created in the reef dip between the horizontal drives. From these winzes the reef is mined by drilling and blasting in stopes or working areas extending on the reef dip. Actual stoping widths are usually of the order of 1m. The broken ore is shovelled into the paths of mechanical scrapers, which move the ore down the stopes to be collected in "koekepanne" or underground rail-mounted hoppers. From the stopes the ore is transported to central, near vertical, shafts known as ore-passes, which collect the material from several levels before hoisting it to the surface in ore-cages. Reef and development waste is handled separately. Some sorting of waste rock and reef is sometimes employed in the stopes, where the broken ore ranges in size from about 300 mm to very fine fractions. Once at the surface the material is transported to the mill either by conveyer belt, train or truck.

2.3.2 Ore-dressing

Ore dressing comprises the mechanical preparation of ore rocks by crushing, grinding and separation, in order to liberate, expose and concentrate the valuable mineral for metallurgical extraction (Ritcey, 1989). The objectives of ore-dressing include: reducing the ore to a proper size for the metallurgical extraction, removal of the gangue as far as possible, increasing the concentration of valuable minerals for transport or extraction,

removal of impurities that can hinder metallurgical processes and separation of different ore-minerals that might occur together. All this has to be done in the most economical way, i.e. using the least number of machines of the largest capacity. Liberation of the mineral is governed by factors such as grain size and complexity of mineralisation. The gangue can often provide clues as to the possibility of excessive reagent consumption during leaching, which would require calcining prior to the metallurgical processes to convert the ores to a reduced or oxidised state for enhanced metal recovery. On the Witwatersrand 5 - 10% of the gold is encased in sulphides, but in Barberton deposits up to 100% is encased in the refractory ores. Calcining is thus required with the Barberton refractory ores to reduce the iron sulphide minerals.

Many properties of the ore minerals make separating it from the gangue possible. Colour, lustre and general appearance have often been used in the past to hand-pick valuable pieces from the waste. Density and surface energy effects influence the rate of sedimentation of the different constituents in water and other fluids. Magnetic permeability and electric conductivity may also be used to concentrate the ore mineral by magnetic and electrostatic separation. Other useful properties include fusibility, friability, fracture, hardness, texture, aggregation etc.

The ore-dressing program on a gold mine may consist of the following, (Truscott, 1923):

- (a) **Washing and Sorting:** Washing cleans the crude-ore for subsequent sorting and crushing, and is sometimes sufficient to release some of the mineral-ore from the gangue. Separation of different constituents or classes of ore is usually done by hand. The operators pick out pieces of ore clean enough for direct metallurgical treatment, whilst sorting out worthless pieces of gangue. Removal of minerals and foreign objects such as tramp metal and wood chips that interfere with the rest of the process is a major advantage. However, hand-sorting is limited to a certain size fraction by practical and economical factors.
- (b) **Comminution (Crushing):** Comminution denotes the whole operation of reducing crude ore in successive stages to the fine slurry or pulp required for metallurgical treatment. Crushing is often the most expensive part of ore-dressing because of the fineness required. The old workhorses of the reduction works, gravity stamps and steam stamps, have since been replaced by more efficient machines, which reduce the ore to expose the valuable mineral in the following stages:
 - *Preliminary Comminution or Breaking:* Jaw and/or Cone Breakers, Figure 2-1 and Figure 2-2, are used to break the rocks by applying direct pressure without adding any water. Breaking rarely releases or exposes the mineral but reduces the size of the material for subsequent crushing. With jaw breakers

the ore is broken between fixed and moving jaws actuated by a pitman lever within a massive iron or steel frame. In the cone breakers the ore is caught and broken inside a outer concave ring by the eccentric gyration of an inner conical cylinder. In the 1950's Run-of-Mine (ROM) milling was introduced negating the need for preliminary breaking by sending the mined ore directly to the mills. Nevertheless, breakers are still being used widely in preliminary comminution today.

- *Primary Comminution or Crushing:* In the beginning the stamp mill reigned supreme as the primary mill, fed from gyratory crushers, but was eventually replaced by primary Tube Mills and Ball Mills, Figure 2-3. These mills, fed from the breakers or directly by hoisted material (ROM), crush the ore by shear and impact caused by the rolling and falling of metal balls or rods within a revolving cylinder. Crushing results in a substantial release and exposure of the mineral as the ore is completely slimed with very little sand fraction remaining. Crushing can also be done using Pressure Rolls, Pendulum Rollers and Edge Runners.
- *Secondary Comminution or Grinding:* Secondary comminution completes the release or exposure of the mineral and is basically a continuation of primary comminution.

Table 2-3 gives an indication of the size reduction ratios and capacities of different plant used for comminution in the gold mining industry.

- (c) **Sizing and Classification:** In dressing it often becomes necessary to eliminate variability in size in order to improve staged comminution. Sizing and classification comprise the division and separation of broken, crushed or ground material into classes according to the average diameter (sizing) and density (classification). Sizing can be done efficiently by screening, and classification by separating particles in a rising column of water. Depending on whether mechanical or chemical processes follow, flocculants or de-flocculants may be added to the slurry. Flocculants are added in small amounts to promote settlement in preparation for chemical treatments such as cyanidation. Flocculants used include inorganic acids, sulphuric acid, ferrous sulphate, lime, neutral salts, calcium chloride, magnesium sulphate, alum and iron salts, with lime being most popular. Mine water is usually a flocculant in itself and inhibits mechanical separation by trapping ore-minerals in the gangue. De-flocculation is then brought about using dispersants such as alkalis and alkali salts, sodic carbonate, organic acids and tannic acid.
- (d) **Concentration:** Optimisation of ore-dressing techniques require maximum release of the ore-mineral but at the same time avoiding excessive comminution. In addition, gold

Table 2-3: Ratings of crushing plant used commonly in the gold industry.

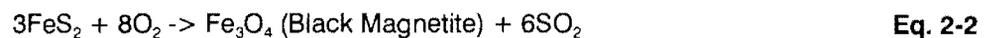
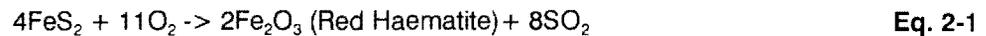
Machine	Size Reduction			Crushing Rate	Daily capacity
	Feed	Out	Factor		
	(mm)	(mm)		(kg/kW.hr)	(Tons)
Jaw and Cone Breakers	..	50	4	1200	120-1200
Gravity Stamp	50	4	12	200	8-20
	50	1	50	125	5-8
	50	0.4	120	75	2-4
Steam Stamp	100	12.5	8	150	250-400
Tube-mill, short	4	0.2	20	125	50-250
Tube-mill, long	1	0.1	10	50	25-100
Ball Tube-mill	50	1.25	40	250	50-400
	50	0.3	160	100	40-200
Ball-mill	75	0.5	150	100	40-100
Rolls, coarse	50	12.5	4	600	100-400
Rolls, fine	12.5	1.25	10	350	100-400
Pendulum, wet	19	0.8	24	100	30-60
Pendulum, dry	38	0.4	90	50	10-20
Edge-runner, slow	50	0.4	120	100	20-40
Edge-runner, fast	9	0.6	15	100	50-80

values and sulphides (pyrite) have to be extracted efficiently from the gangue after the mineral ore has been detached or liberated. Water concentration (gravity), flotation and other methods attempt to separate the valuable ore-mineral from poor gangue, thus avoiding wasteful and useless comminution and providing a concentration of the valuable ore-mineral.

- Water concentration requires the mineral to be of a different density than the gangue, allowing the division of ore-mineral from gangue by density if already sized, or by size if classified. Gold ores lend themselves to gravity extraction as the gold and pyrite are much denser than the silica carrier.
- Flotation Concentration relies on the wetting and non-wetting properties of the ore-mineral by water and other contaminants (oils). Oil has an affinity for mineral ores and floats on water to be collected as overflow - oil flotation. Flotation can also be effected by rising gas bubbles, where the non-wetting property of the mineral allows it to be carried off by the bubbles - gas-froth flotation. Flotation concentration is mostly used to treat refractory ores for roasting (calcining) prior to cyanidation, but has grown in popularity elsewhere.

- Other separation techniques include: magnetic separation, electrostatic separation, pneumatic separation and centrifugal separation. These, however, are not commonly used in the gold mining industry.

(e) **Heat Treatments:** Ores may be heat treated during the dressing stages to remove excess water, or to expose the ore-mineral in refractory ores. This process is known as calcining. During calcining the ore is heated to between 450 and 800°C. The iron sulphides (typically pyrite) decompose into porous haematite, rendering the gold amenable to chemical treatment as follows:



Several intermediate materials are formed and the sulphur dioxide may be used in the manufacturing of sulphuric acid. Roasting also volatilises cyanides (cyanide consuming agents) such as antimony and arsenic.

2.3.3 Metallurgical extraction

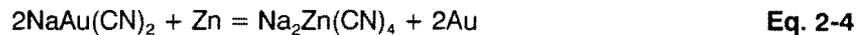
Once the valuable ore-mineral has been liberated, exposed and concentrated by ore-dressing techniques, it may be extracted chemically through the processes of amalgamation or cyanidation, Stanley (1987).

- (a) **Amalgamation:** The first step in the amalgamation process is to bring the gold into contact with mercury, whereby the gold dissolves in the mercury. This is done by either passing the pulp over mercury laden copper plates, or by tumbling the pulp with mercury in an amalgam drum with the addition of caustic soda to help oxidation of the sulphides. The resultant amalgam is washed clear of sulphides. Excess mercury may be pressed out through canvas in an amalgam press followed by retorting. Retorting comprises the separation and recovery of mercury from pressed gold amalgam through distillation and condensation. The retorted gold, spongy in appearance, is melted with flux and poured into bullion bars. The drawbacks of this process are the obvious health and safety hazards it poses, together with labour intensiveness and large floor space requirements. The process has largely been superseded by cyanidation.
- (b) **Cyanidation:** During cyanidation the gold is leached from a thickened pulp with soluble cyanide salts (sodium cyanide, NaCN or calcium cyanide, CaCN) according to the following equation:



Other essentials are oxygen and calcium hydroxide, $\text{Ca}(\text{OH})_2$, to stabilise the cyanide radical, control alkalinity and maintain a minimum pH of 10 to 11. The pulp is usually prevented from sedimenting by mechanical or compressed-air agitation to facilitate leaching. Depending on the grade of the ore up to 98% gold can be extracted in the leaching process. Cyanide leaching results in a solution containing anionic metal cyanide complexes from which the gold complexes must be recovered. Two major routes for recovery may be followed. The first, zinc precipitation, requires the liquid (gold in solution) to be separated from any insoluble material before recovery; the second, carbon-in-pulp (CIP) process, is able to recover the gold directly from the leached pulp. Separation of liquid and solids is usually done through continuous vacuum filtration followed by filtrate clarification, to remove the remaining fine suspended solids.

- *Zinc Precipitation or Cementation:* Vacuum de-aeration is applied to remove dissolved oxygen, since this greatly improves the efficiency and economics of the process. Soluble lead salts (lead nitrate) are then added together with zinc shavings or dust, whilst satisfactory cyanide and alkalinity levels (using calcium oxide, CaO) have to be maintained for efficient operation. Mixing can take place prior to precipitation in a small emulsifier tank. Zinc precipitates the gold due to its high electro-negative charge in relation to gold's electro-positive charge, according to the following formula:



The slime of gold and zinc salts produced in this way is recovered by filtration and then treated with dilute sulphuric acid to remove as much zinc and other impurities as possible. The resulting precipitate is then calcined to dry the pulp and to oxidise impurities such as lead and zinc prior to smelting with a flux of borax and silica. Smelting oxidises the base metals in the gold slime and combines them with silica to form a slag, which separates from the gold and silver due to its lower density. Gold smelts at 1063°C , requiring the ovens to work at between $1200 - 1400^\circ\text{C}$. The bullion is poured into bars, cleaned, numbered and assayed before dispatch to the Rand Refinery. The slag is also dispatched to the Rand Refinery as a by-product.

- *Carbon-In-Pulp Process:* Activated carbon is used to recover gold by adsorption directly from the leachate. The carbon must be coarser than the pulp (usually $> 1 \text{ mm}$) so that it can be readily screened from the pulp in the end. Adsorption occurs as the pulp flows continuously by gravity through a series of tanks and inter-stage screens. The inter-stage screens ensure that the carbon remains in the tanks but allows the pulp to pass. The pulp loses

gold progressively down the train, with barren value in the last tank. The carbon, removed once a day by counter current from the last to the first tank, is then washed by dilute hydrochloric acid (HCl), followed by washing with water to remove tramp material (wood chips etc.) as well as slime. During washing the system is raised to the desired elution temperature of 110 - 120°C. Eluates such as potassium cyanide, sodium sulphide or caustic cyanide (NaCN + NaOH) are then used to release the gold and bring it into solution. Gold is recovered from the eluate either by electro winning or zinc precipitation. A similar technique called the carbon-in-solution (CIS) process may be used to scavenge gold from other solutions that arise from existing mine circuits.

- *Carbon-in-Leach (CIL) Process:* The leach and adsorption circuits are occasionally combined. The carbon enhances the leach efficiency by removing surface coatings and has advantages from a capital cost point of view. However, CIL presents considerable operational problems.

Other methods employed include: resin-in-pulp (RIP) using ion-exchange resins instead of carbon, solvent-in-pulp (SIP), reverse osmosis and membrane technology, electro winning, pressure elution and electro elution.

2.3.4 Refining

The Rand Refinery, based in Germiston near Johannesburg, was established in 1920 to provide a gold refining service to the South African mining industry (Cowey, 1994). It is the largest and most modern gold refinery in the world, processing South Africa's entire production of newly-mined gold. In addition to gold refining services, the Rand Refinery also provides analytical services, produces silver and gold granules of various caratages and delivers high purity export products and gold chemicals, such as gold potassium cyanide used for gold plating.

At the refinery bullion is upgraded to at least 99.5% pure by either the Miller chlorination process or electrolytic refining. Miller chlorination involves chlorine gas being blown through the molten bullion, which converts base metals and silver to chlorides or slag. Electrolytic refining may be used to extract the remaining impurities from the chlorination process to render 99.99% pure gold by removing the platinum group metals. This process is also used to recover gold from mine by-products such as sweepings, dust, clothes, etc.

2.4 TAILINGS IMPOUNDMENTS

2.4.1 Introduction

It is generally recognised that mining industries provide an essential input into the economy of various regions around the world. It is equally well known that mines can also be a source of immense environmental concerns. More than 5 t of solid and liquid wastes have to be generated in the mining and milling processes to obtain about 10 g of gold, (Ripley et al., 1982). Only a small percentage of these wastes can potentially be backfilled into worked-out areas underground and the bulk of the waste has to be stored in large surface impoundments. However, point of view can often colour one's perceptions of these waste impoundments, as illustrated by two quotes from the same era:

*"Many Massive 'Manhill's that Mark the Mortals' Map
Make Magnificent and Mighty Monuments to
- the Miner's Muscle
- the Metallurgists Magic and
- the Meaningful Meditations of the Master
of Mud Mechanics, the Mill Manager"*

Smith (1972)

"In the view of conservationists, there is something special about dams, something - as conservation problems go - that is disproportionately and metaphysically sinister. The outermost circle of the Devil's world seems to be a moat filled mainly with DDT. Next to it is a moat of burning gasoline. Within that is a ring of pinheads each covered with a million people - and so on past phalanxed bulldozers and bicuspid chain saws into the absolute centre of hell on earth, where stands a 'Tailings' dam."

McPhee (1971)

Tailings, the by-product of ore-dressing and metallurgical extraction processes, contain the waste fraction of the processed mineral-ores, usually in the form of a fine grained slurry or pulp. Until the mid-1800's mining and smelting technologies were primitive and only the highest grade ores could be mined profitably with the result that little waste was produced. As mining, smelting and transportation technologies improved, lower grade ores could be mined, leading to increased volumes of tailings as well as a finer grind, (USCOLD, 1994). During 1900 - 1930 the first dams were constructed specifically to retain tailings in the USA. Before that, tailings were disposed of directly into rivers and other water bodies. The first dams, designed by trial and error by mine operators, were often constructed across stream channels employing upstream methods and did not survive long. After the 1928 Barahona

dam failure in Chile, the downstream method was employed for the first time, as were cycloned sands for constructing the embankment walls. In the 1940's earthmoving equipment made conventional water retention dams viable, and by 1950 engineering principles of seepage and foundation stability were applied. The 1965 seismic failures in Chile gave impetus to liquefaction research in tailings and issues related to groundwater contamination were introduced in the 1970's.

Tailings impoundments are arguably the largest structures built by man (Hoare & Hill, 1970), with significance for engineers, regulatory authorities and the public alike. They can and often do have a greater environmental impact than any other aspect of the mine, including, for example, the mine excavation. Tailings structures are primarily geotechnical structures; built on soils and rocks, with soil and rock materials to contain the tailings product that behaves essentially as a soil. The amount of tailings produced in the world, 5×10^9 t/yr in 1994, greatly exceeds the amount of fill handled by the civil engineering profession in the construction of embankment dams, motorway embankments, and all other earthworks (Penman, 1994). By far the greatest amount of tailings is produced by the processing of metal ores. South African mines were producing in excess of 250 million tons of mine tailings waste annually in 1987, with an average mine producing roughly 100,000 tons of tailings per month over an expected lifetime of 25 years (Donaldson, 1965). This much waste requires impoundment structures that pose huge risks of environmental destruction and possible disaster in the event of a failure. Fortunately such disasters have been relatively infrequent. Nevertheless, more than 1,000 fatalities have been reported since 1917 as a result of incidents on all types of tailings impoundments. These and similar incidents have caused countless millions of rand in damages to property, not to mention devastating the environment.

The main components of a residue disposal system according to McPhail and Wagner (1989) are (see Figure 2-4):

- *Delivery system*: Pipe work, valves and discharge points to convey and deposit the tailings pulp onto the impoundment.
- *Containment wall or Embankment*: Either as a raised wall composed of deposited tailings and a small compacted earth starter wall, or as a conventional retaining earth and/or rock dam wall.
- *Under-drainage system*: System of filter drains to control seepage from within the dam.
- *Decant system*: Penstock or floating barge type drainage facilities for the removal of clarified slurry water and storm water from the pond.
- *Stormwater diversion system*: Trenches and/or bunds around the dam to divert storm water runoff.

- *Stromwater catchment system*: Paddocks arranged around the dam to control water and tailings sediment from the dam slopes.
- *Return water system*: System of dams, sumps and pipes for handling effluent from the dam.
- *Ancillary systems*: Access roads, power supply, etc.

2.4.2 Siting and Layout

As with conventional water-retaining dams, each tailings impoundment constitutes an individual project, dependent for its detailed design on the site conditions, the type and rate of delivery of the tailings, availability of other waste materials from the mining or industrial processes, climatic conditions and many other factors.

The most important restraint in siting a proposed impoundment is storage capacity, i.e. the ability of the structure to accommodate daily demands for tailings disposal with rate-of-rise (ROR) being the restraining factor. Should the structure rise too quickly, the development of pore pressures in excess of equilibrium levels can adversely affect the stability of impoundments constructed of low permeability tailings products. For gold tailings, of relatively high permeability, the ROR becomes critical in controlling densification of embankment material through desiccation. Environmental aspects such as air and water pollution have become major issues recently. Blight (1987) estimates the rate of surface erosion on a typical South African gold ring dyke dam to be as much as 500 t/ha per year.

Prior to actual commencement of the design of any tailings impoundment structure, other studies must be developed. These include a preliminary decision regarding the volume of ore to be treated daily and yearly, the general order of the size of the ore body to estimate the total volume of tailings, metallurgical studies to assess the probable tailings gradation and recovery process and potential plant layout. In South Africa it is common practice to start the design with an estimated final height of the impoundment in the order of 35 - 40 m. Rates-of-rise commonly employed range between 1 and 3 m/yr as a function of pulp density and foundation conditions (Wates, 1983) resulting in a life expectancy of at least 10 years. Measurements of settled dry densities range between 1250 and 1650 kg/m³ as a function of depth, so a value of $\rho_d = 1450$ kg/m³ is used in general for design purposes. These values can be used to estimate the required area of the impoundment, which will influence the possibilities of siting and layout. Siting considerations, after Cowey (1994), should also take consideration of:

- *Transportation costs*: Distance and elevation relative to the mill.
- *Topography*: Layout, fill volume requirements and diversion feasibility.
- *Hydrology*: To minimise inflow or diversion requirements.

- *Geology*: Foundation stability, seepage and requirements for borrow fill material.
- *Groundwater*: Seepage contamination.

In addition to the area required for the actual tailings fill, area has to be allocated for stormwater cut-off trenches, access roads, water and pulp services, catchment paddocks as well as area for the handling of returnwater. Usually a 50 m zone around the perimeter is sufficient, but this can make up as much as 25% of tailings dam area.

Areas obviously not suited for tailings dams are those that

- are too far from the metallurgical plant,
- are of a too steep topography,
- pose difficulties with access,
- comprise too large a water catchment area,
- include unsuitable geology or mineralisation,
- include important land use zones and
- are in a sensitive ecological zone.

In defining alternative solutions for a tailings impoundment the following aspects of materials selection and internal arrangement have to be taken into consideration (Caldwell & Stevenson, 1984):

- (a) **Embankment type**: The choice of embankment type is governed by mill related factors such as type and output of tailings and effluent as well as site related factors including seismicity, climate, water handling and available materials.
- (b) **Phreatic surface**: The free water level in a tailings impoundment governs to a large degree the overall stability of the structure and has to be kept as low as possible near the embankment faces. Seepage breakout on the embankment could lead to mass instability as well as seepage related problems of piping, erosion and sloughing. The phreatic surface is governed by the arrangement of internal permeabilities, which should ideally decrease towards the decant facility, as well as by the permeability of the foundation material relative to the tailings. The only aspect influencing the level of the phreatic surface that can be controlled during operation is the location of the surface pond relative to the embankment walls. Structural aspects that are used to control the phreatic surface in the embankment include low permeability cores and high permeability zones (drainage chimneys, blanket drains or finger drains) to extract water near the embankment face. Filters should be properly designed and constructed and often constitute a disproportionately large fraction of the total cost of construction.

(c) **The starter dike:** By providing a starter dike more pervious than the tailings, an extra measure for phreatic surface control is provided. Typical construction materials for starter dikes include:

- *Natural soils:* Maximum use should be made of natural accessible material, where borrow areas inside the impoundment is most advantageous. Factors which influence selection are: moisture content, permeability and compactibility.
- *Mine waste other than tailings:* Using mine waste may present a major cost advantage, but should balance production rates with rate-of-rise requirements. These materials require careful attention to filter design and construction.
- *Cycloned tailings:* The use of tailings offers major cost advantages over borrowed material, but requires less than 60% non-plastic slimes in the feed. Cyclone construction is rarely used on gold tailings impoundments due to its low sand content.

According to a parametric study by Nelson et al. (1977) the starter dyke permeability is the most important factor influencing the phreatic surface location in upstream constructed dams, with blanket drains having a lesser effect.

(d) **Foundation conditions:** The design must be compatible with foundation conditions. Problem materials include organic material, normally consolidated soft clays and collapsible soils. If these cannot be removed prior to construction, embankment slopes have to be adjusted accordingly. Notably it is the internal structures that suffer most through settlement damage.

Mining companies and regulatory agencies have fundamentally different objectives in choosing a design alternative for a tailings impoundment. Whereas the mine is concerned with economics, regulatory agencies are concerned with environmental factors, often of equal or greater importance than economic issues, on behalf of the public. The gold market is not controlled by mining companies, but by a free market system of supply and demand. The extra cost of mine waste disposal cannot be passed on to the consumer but must be absorbed within the economic constraints of the project (Van Zyl, 1993). The disposal of mine waste represents, therefore, capital and operating costs to a mining operation. For this reason, mine waste has historically been disposed of at the lowest cost, often with considerable environmental impact. However, any failure in the disposal system could shut down the processing plant causing loss of profit plus payment to unemployed staff, as well as cost of repair and any claims that could arise from the failure. Compounding the economics of these structures is the fact that they have to provide for cost-effective reclamation when the dam is decommissioned (Caldwell & Robertson, 1986). Tailings

structures become part of the environment after they have served their purpose and must be left in a durable and safe state.

The Systems Approach (Roth, 1991) of generating and selecting alternatives has been successfully implemented to meet the interests of all groups concerned in tailings dam design. The final selection of a proposed site and layout is ultimately dependent on:

- visual impact,
- land use,
- airborne dust release,
- seepage water release,
- stability,
- operational requirements and
- cost.

2.4.3 Statutory Requirements in South Africa

Prior to 1998 the main acts controlling the design, construction and operation of tailings impoundments were the Mines and Works Act (Act 27 of 1956), the Atmospheric Pollution Prevention Act (Act 45 of 1965), the Water Act (Act 54 of 1956, as amended from time to time), the Health Act (Act 63 of 1977) and the Soil Conservation Act (Act 54 of 1956). Although the Water Act (as amended by Act No. 96 of 1984) was used to classify a tailings dam structure on the basis of size and hazard potential, a Government Notice in 1986 exempted the Mines from the provisions of the regulations (except regulation 15 regarding registration) for a period of 5 years. This was extended in 1991 for a further period of 2 years, and again for a further period of exemption of 2 years in 1993. No license or permit was required to operate gold residue disposal facilities, but permits regarding slimes dams had to be acquired from the Department of Mineral and Energy Affairs in terms of Section 8(1)(a) and Section 6B/4(2) of the Physical Planning Act (Act 88 of 1967) as well as from the Department of Water Affairs in terms of Section 12,21 and Section 30(5) of the Water Act. In addition, an environmental impact assessment was required as a formal approach to evaluate the effects of the project on the environment and to minimise environmental degradation in the most cost effective manner. Regardless of legislative requirements, emphasis on environmental aspects had the additional benefits of establishing a good public image by preservation or actual improvement of the environment, and demonstrated that industrial development could go hand-in-hand with environmental protection.

After the 1994 Merriespruit disaster (Wagener et al., 1998), work started on the drafting of an obligatory Code of Practice for the design, operation management, rehabilitation and closure of tailings dams. After a trial period in 1997 the draft code was finalised in 1998 and

encoded under South African mining legislation as SABS 0286. Under this code a tailings dam is classified in terms of its hazard potential, requiring certain mandatory procedures depending on the hazard classification. A professional engineer, experienced in tailings dam engineering, is required to review and audit each tailings dam on a regular basis including the monitoring and reporting on rates of rise, piezometer levels, drain outflows, freeboard and pond control. Emphasis is also placed in the code on ensuring that the owner of a tailings dam appoints appropriately experienced and trained operators. In addition to SABS 0286, construction and operation of tailings impoundments are currently controlled by the National Water Act (Act No. 36 of 1998) and regulations incorporating the Water Services Act (Act No. 108 of 1997) Volumes 1 and 2, the Mine Health and Safety Act (Act No. 29 of 1996) and the Occupational Health and Safety Act (Act No. 85 of 1993).

2.4.4 Embankment layouts

When deciding on an embankment layout the designer aims to optimise embankment height and fill area to accommodate daily production rates and final storage capacity, without compromising the safety and serviceability of the structure in the most economical manner. Other factors that have to be taken into account are: earthworks requirements, availability of construction materials, reclamation and seepage control measures.

Typical layout designs that are used worldwide include:

- (a) **Perimeter or Ring Dyke**, Figure 2-5a: As the name implies the ring dyke impoundment requires an embankment wall to be constructed around the perimeter of the dam prior to or during filling. This type of layout is used extensively in South Africa due to the flat topography in the gold producing areas, which affords little opportunity for valley type impoundments. Ring dykes require relatively high quantities of embankment fill in relation to their storage capacity and are therefore ideally suited to constructing the embankment wall from the tailings product that is produced over the operational life of the dam. Another advantage of this type of layout is that runoff inflow to the surface is eliminated.
- (b) **Cross Valley Impoundments**, Figure 2-5b: The cross valley impoundment requires only one embankment to cross a valley or depression from wall to wall downstream of the storage area. The storage area should ideally be located near the head of the drainage basin to minimise flood inflow quantities that must be handled either by storage, spillways or separate upstream water control dams.
- (c) **Side-hill Impoundments**, Figure 2-5c: Best suited to side-hill slopes of less than 10%.

- (d) **Valley Bottom or Incised Impoundments**, Figure 2-5d: A compromise between cross valley and side-hill layouts, valley bottom impoundments must be provided with a diversion channel to carry flood inflow around the impoundment.

- (e) **Multiple Impoundments vs. Single Impoundment**: A large tailings storage facility can be constructed either as a single impoundment or as a number of separate, but connected impoundment units or compartments. Multiple compartments demand greater volumes of embankment fill material and may require excessive flood control measures in a small upstream section. However, they allow better phreatic surface and seepage control in ring dike systems and may be the only viable option in the restricted space at valley bottoms. Multiple impoundment units can also be used to spread initial construction costs if sequentially constructed.

2.4.5 Transportation and Discharge

In the early gold mining days, the stamp mills produced a coarse sand tailings that was dumped dry using an endless rope cocopan method and more recently belt conveyors (Gowan & Williamson, 1987). Today's reduction product is extremely fine, with a high water content, and does not allow free stacking or dumping. It requires engineered impoundment structures. The most economic materials used to construct a tailings impoundment are often the coarse (sand) fractions of the tailings. The sands can be separated from the fines in the tailings either gravitationally following discharge onto a beach, or centrifugally by hydraulic cycloning.

The tailings slurry is usually pumped at the highest concentration practicable to reduce the bulk of material and the cost of transportation. Pulp concentration is increased at the metallurgical plant by classifiers, cyclones, screens, filters or thickeners, the cost of which should be offset against the benefits of higher rates-of-rise attainable, the value and re-use potential of recovered water as well as reduced costs of return water recovery. The maximum concentration is limited by metallurgical plant processes or by the pumping plant. In a typical gold tailings pulp the ratio of water to solids varies between 2.8 and 1.0 by mass, which relates to a moisture content of between 280 and 100%. For underground filling purposes pulp concentration is further increased to a moisture content of 50% to enhance the dewatering and consolidation characteristics of the fill.

The delivery system comprises rubber lined centrifugal pumps, pipelines - sometimes rubber lined to reduce corrosion and wear, and valves. The options available for discharge

onto the tailings impoundment are sub-aqueous deposition and sub-aerial deposition, either using open ended pipe discharge, spigotting or cycloning.

(a) **Sub-aqueous Deposition:** Usually in the form of uncontrolled discharge within a body of water. The tailings settle as a soft bottom sediment, or are transported and dispersed over a large area under water. The resulting low density material is generally very soft and may require special construction techniques and reclamation procedures. The soft sediments can also retain excess pore pressures for a very long time under self-weight consolidation and can be expected to be under-consolidated¹ in many cases.

(b) **Sub-aerial Deposition:**

- *Open Ended Pipe Discharge:* Open ended pipe discharge is often used in South Africa as an inexpensive delivery system. Deposition is typically accomplished by discharging the slurry from one or more open ended pipe outlets or stations around the perimeter of the dam area. The discharge sequence is cycled to allow a freshly deposited layer time to dry and increase in density before the next layer is deposited. This has been standard practice in South Africa since the early 1900's (Ruhmer, 1974) and results in a significant improvement in the mechanical properties of the deposit. The discharged slurry leaving the pipe forms a deep hole or plunge pool at the base of the pipe outlet from where the slurry breaches and flows towards the central area containing a semi-permanent pond. Flow is fast and concentrated on the beach (sloped area above water level) before fanning out in a delta as it approaches the pond area. The idea is for segregation to take place on the beach, leaving coarser particles closer to the embankment and the finer fraction in the central area. Although there is a general trend of increasing fineness towards the pond area, flow on the upper beach tends to concentrate in meandering channels with flow velocities high enough to pass most of the material to the pond with little opportunity for segregation and deposition on the beach (Penman, 1994).
- *Spigotting:* Similar to open ended discharge, but with the objective of reducing the velocity of the deposited stream and encouraging sheet flow across the beach to aid segregation. This form of deposition effectively separates the coarse sandy fraction from the slimes by gravity segregation on the beach and is therefore generally used when the tailings have a wide grading and high percentage of clay sized material. Spigotting is

¹ Under-consolidation, in the context of this document, implies that primary consolidation and therefore the process of excess pore pressure dissipation has not yet been completed.

accomplished by partly covering the delivery pipe orifice with a faceplate, leaving only a small half-moon opening. This results in a spraying of the discharged slurry over a large area on the beach with more uniform sheet-like flow towards the pond. The same effect can be accomplished by punching holes into a ring-type tailings delivery pipe. With delivery points spaced at 1 to 5 m intervals and operated in groups of 20 to 40 spigots at a time. A ring main around the toe of the dam feeds the spigot pipe, which together with supports and extension pipes has to be raised periodically as the dam rises.

- *Cycloning*: Cyclones mechanically separate the feed pulp into coarse and fine fractions providing competent sands at a reduced water content for embankment building and high water content slimes for interior filling. Operation of a cycloning system requires planning and a significant amount of management if the system is to function properly. It is only practicable when the tailings product has a wide grading with sufficient coarse underflow for embankment wall construction. The Witwatersrand uniformly graded gold ores and general flat terrain makes this type of construction prohibitively expensive. Under special circumstances it might, however, find application in short valley impoundments, where seismic activity is of particular concern or in small operations of limited life.

2.4.6 Methods of Construction

Methods of impoundment construction vary considerably from one tailings product to another as a result of the grading, solids content and solids density of the slurry. These aspects control the tendency of the material to segregate or separate into size fractions following discharge onto the impoundment structure. Copper and platinum tailings being well graded between 2 mm and 1 μm , segregate readily and disposal methods can be designed to optimise the effects of this segregation (McPhail & Wagner, 1989). Gold tailings, on the other hand, are fairly uniformly graded between 200 μm and 2 μm with a reduced tendency to segregate and the design can not rely on the benefits of gravity segregation in the method of disposal (Wates et al., 1987; McPhail & Wagner, 1989). The most important factor controlling segregation of gold tailings slurries is the density of the pulp discharged, and to a lesser extent variations in grading.

In addition to the segregating properties of the slurry, the choice of method of constructing the impoundment structure will be determined by the following factors:

- Capital and operating costs.
- Previous experience and preferences.
- Site topography.

- Climatic conditions which affect drying characteristics and freeboard requirements.

Direct Disposal into a Water body

Historically tailings were discharged from the mill directly into the nearest water course or body. Eventually farmers supporting the mining communities started to complain about plugged irrigation ditches as well as contaminated farmlands. Today, the environmental impact or pre-treatment costs of this method of disposal virtually eliminates the option.

Offshore Disposal

Where the mill effluent is relatively innocuous, sub-aqueous offshore disposal outside of biological productive or sensitive zones, may be considered. However the tailings product should be coarse or flocculated to settle without causing excessive turbidity. Offshore disposal may be the only viable method of disposal in coastal areas with extreme high precipitation, steep terrain and high seismicity.

Conventional Earth or Rock-fill Dam Wall

Tailings impoundments can be constructed in the same way as a conventional water retaining dam using natural materials such as rock-fill with a clay core to construct the retaining wall.

Advantages: Suitable for any type of tailings.

Any discharge procedure may be used.

Suitable for high water storage capacity.

Good seismic resistance.

An engineered upstream filter can readily be included in the wall.

Disadvantage: Construction of entire embankment wall usually in one stage.

Requires natural soil borrow for construction.

High cost of embankment construction.

This method is best suited to impoundments with:

- high water storage requirements and high runoff inputs,
- no re-circulation of tailings water required,
- high clay content tailings requiring large evaporation areas and
- toxic leachate tailings, utilising natural or artificial seepage barriers.

Raised Embankments (Brawner & Campbell, 1972; Mittal & Morgenstern, 1977; Vick, 1983)

To minimise the cost of a tailings impoundment structure the most economical design must be employed. This frequently requires consideration of the use of tailings to construct the dam wholly. Staged construction may then be utilised with the height of the embankment structure maintained minimally above the storage level required. This procedure minimises initial capital investment and spreads the cost of construction over many years.

Construction starts with a small starter dyke of natural materials designed to impound the initial 2 to 3 years of tailings with some flood protection. Thereafter the embankment is raised periodically using tailings material to keep ahead of the main impoundment level with sufficient freeboard for flood control. In areas of low to no seismic activity, tailings embankments can be placed to acceptable densities by carefully controlled hydraulic procedures without additional mechanical compaction. However, in areas of moderate to high seismic activity, additional densification must be achieved by mechanical compaction (Mittal & Morgenstern, 1977). Relative small reductions in the slope of a raised embankment can result in large increases in the dam height and storage capacity. Embankment slope is normally restricted by erosion and stormwater control requirements rather than stability concerns, resulting in quite steep slopes. These are effectively reduced for stability purposes, generally to 25°, by using step-back berms.

There are fundamental differences between conventional earth-fill dams and raised embankment tailings dams that should be taken into account when designing a tailings impoundment (Kealy & Busch, 1979). Tailings from the reduction plant have different characteristics to materials used in natural earth-fill dams, and the method and sequence of construction are entirely different. Consequently embankments built with tailings have the following unique features:

- Unlike earth and rock-fill dams, tailings embankments are active and have an average annual growth rate of 1 to 3 m/yr, depending on the tonnages produced and areas available for disposal. Thus each embankment is under continuous construction until abandonment.
- Due to continuous construction, the seepage regime takes on a different geometry compared with that of a standard water-retention dam.
- Because of the method of hydraulic deposition, each sector of a cross section of the impoundment is highly layered and can have different properties. Furthermore, the methods of deposition and rate-of-rise can change with time and climatic seasons.
- Unlike conventional dams, construction takes place without the use of certified inspectors. Extended field monitoring programs are, therefore, required to provide the data necessary to increase confidence levels in the factor of safety, or to amend

construction techniques. This method of continuous assessment lends itself to the Observational Method of construction as discussed by Peck (1969).

- Finally there is a finite life planned for a tailings embankment, after which land reclamation and re-use should be considered.

Raised embankments may be constructed using one of the following techniques:

- Upstream.
- Downstream.
- Centreline.
- Daywall-nightpan paddock.
- Cyclone.
- Thickened discharge.

(a) **Upstream Raised Embankment (Figure 2-6a):** The upstream technique is the oldest method of constructing tailings dams and is a natural development from considerations of disposing of tailings as cheaply as possible. An initial starter wall is constructed at the downstream toe. This starter wall should be sufficiently pervious to pass seepage water, but designed to resist piping. Tailings are then discharged from the top of the starter dam by spigotting or cycloning to develop a dyke composed of the coarser fraction. The centreline of the top of the embankment is shifted towards the pond area as the height of the dam increases. The downstream toe of each subsequent dyke is supported on top of the previous dyke, with the upstream portion placed over the finer tailings. As the height of the dam increases, the potential failure surface is located at an increasingly greater distance from the downstream face and possibly through layers of soft slimes deposited under backponding due to careless earlier discharge practices. As a result, the outside competent shell contributes less to stability as the height increases.

Harper et al. (1992) label upstream raised embankments as most susceptible to excessive post earthquake deformations and liquefaction (Dobry & Alvarez, 1967; Finn 1982). They stress the importance of water management (Kealy & Busch, 1971; Abadjiev, 1976; Nelson et al., 1977), especially pond location relative to the embankment crest and construction quality control as crucial to the safety of these impoundments. As most historical dams have been built this way, the authors recommend the following methods of improving or stabilising upstream dams: provision of drainage under and within the embankment, reduction of slope angle, use of compaction to improve the density and construction of a rock-fill toe berm against the downstream face to increase stability. Nevertheless, Casagrande and McIver (1970) describe an upstream dam built to a height of 92 m with a slope of 1 in 0.58.

Advantages: Simplest and least costly method of construction.
Smaller quantities of coarse tailings required.
Can use natural soil borrow or tailings in the embankment.
Large initial dam area and high storage capacity.

Disadvantages: Phreatic surface control requires well controlled pond location.
Low pulp density required to promote segregation after deposition.
Low shear strength results from low in-situ densities.
Does not provide for internal drainage facilities.
Limited safe rate-of-rise.
Seismic liquefaction susceptibility - inappropriate in seismic areas.
Cannot be used for water retention.
More prone to cracking due to differential settlements.

(b) **Downstream Raised Embankment (Figure 2-6b):** The downstream method of construction is similar to the upstream method with the centreline of the top of the embankment shifting downstream as the dam is raised. Each subsequent stage of dyke construction, is therefore supported on the top of the downstream slope of the previous section. This method of construction eliminates the possibility of a slip surface passing through earlier backponded slimes as might be the case with the upstream method. If the dam is located in a seismic active area the downstream extensions must be compacted to increase in-place density and minimise the risk of liquefaction. To minimise seepage through an embankment constructed with tailings, the upstream face may be sealed with a layer of impervious soil.

Advantages: Suitable for any type of tailings.
Can use natural soil borrow or tailings for embankment construction.
Rate-of-rise essentially unrestricted.
Can readily incorporate internal drainage systems.
Storage of significant water volumes possible.
Embankment compaction is no problem.
Liquefaction resistant, can be used in areas of high seismicity.
The embankment is not built over previously deposited slimes.

Disadvantages: High initial costs.
Large volume of embankment fill required; increases exponentially.
High cost of embankment construction.
Small initial dam area, but increases with time.
Erosion protection cannot be applied until the dam is completed.

(c) **Centreline Raised Embankment (Figure 2-6c):** The centreline method is a compromise between the upstream and downstream methods. In the centreline

method, the crest of the dam is maintained at the same horizontal position as the height of the dam increases. The dam is raised by spreading and compacting additional coarse tailings on the top, on the upstream shoulder and on the downstream slope. If the upstream slope of the embankment relies on the buttressing effect of the impounded slimes and the slimes undergo large post deposition settlement leaving the upstream face unsupported, localised slope instability may occur. A coarse gradation of the tailings is necessary to afford rapid drainage to provide access for construction and compaction equipment. This method shares advantages with both the upstream and downstream methods, while mitigating many of the disadvantages.

Advantages: Requires less embankment fill than the downstream method.
Moderate cost of embankment construction.
Can use natural soil borrow or tailings for embankment construction.
Acceptable seismic resistance.

Disadvantages: High initial costs.
Requires more sand or coarse fraction than upstream methods.
High restrictions on the rate-of-rise.
Not recommended for permanent water storage.
Erosion protection cannot be applied until the dam is completed.

- (d) **Daywall-Nightpan Paddock System (Figure 2-7):** This form of construction relies on a temperate to semi-arid climate and low water table conditions or acceptable evaporation rates to improve the properties of the deposited material. Upstream, downstream and centreline configurations may be employed. The upstream paddock system, developed empirically over the last 100 years, is used almost exclusively in South Africa with its prevailing dry and hot climate in the mining areas. On the Witwatersrand evaporation generally exceeds precipitation and dams up to 60 m high have been built successfully (Penman, 1994). Desiccation suctions of up to 1 MPa have been measured (Blight, 1969), however, these will be destroyed upon re-wetting of the material, and the only reliable increase in strength is thought to be due to densification as a result of the high suctions (Donaldson, 1965).

Compacted earth starter walls and under-drainage are constructed initially so that the rate-of-rise does not exceed 2.5 m/yr by the time the starter wall is overtopped. The tailings then take over from the starter wall as the main embankment wall. The dam is divided into two sections, the perimeter wall or daywall and the interior or night area also known as the nightpan or floor. The daywall is designed to provide sufficient freeboard to retain the accumulated water from deposited tailings and that from the design storm. The daywall is generally 10 - 30 m wide (Wagener & Jacobsz, 1999), at an average slope (Section 2.4.7) of 35° (Blight, 1988) and is sectioned into paddocks around the perimeter, each paddock being filled from its midpoint by a delivery station.

Wall raising is done by building up the rim of a paddock using the deposited tailings, either by tractor-plough or by labour-intensive shovel packing. During the day-shift pulp is delivered into these daywall paddocks to a depth of about 150 - 200 mm and distributed by gravity. Delivery stations are fed from a ring-main around the toe of the dam and are usually not more than 400 m apart. Excess or supernatant water is decanted into the night area via decant pipes through the inner wall of the daywall. The daywall is left to dry, consolidate and crack for up to 3 weeks before the next lift, thus improving the mechanical properties of the material as a result of densification (Blight & Steffen 1979). The need for supervision and close control of the pulp depth makes daywall raising entirely a daytime procedure, hence the name. The daywall is not compacted in South Africa and is approximately 1 - 2 m higher than the interior of the dam.

During the night, tailings are discharged into the interior of the dam (night area or nightpan) from delivery stations located just inside the daywall. Supernatant water is drawn off the next day by penstock decant or barge pump. A natural beach forms in the night area from the delivery point towards the pond surrounding the decant facility. The location of this semi-permanent pond in the night area is controlled by the spacing, sequence of operation and duration of delivery of the internal delivery stations.

Paddock deposition is successful when the tailings product is fairly uniform in grading, remains well in suspension at a constant density until placed, and when the rate-of-rise matches the drying time of the tailings (Gowan & Williamson, 1987). These conditions result in a deposited slurry that is reluctant to segregate and therefore does not result in too weak a region in the low point between delivery stations on the daywall. The method is particularly successful in South Africa for most gold, tin, calcine and gypsum products, aided by the highly evaporative climate. The paddock system results in a daywall which is normally well consolidated and reasonably firm, while the material in the night area and especially in the pond area is significantly less consolidated and soft (Wagener & Jacobsz, 1999).

- (e) **Cyclone Construction (Figure 2-8):** Cyclone construction methods can be applied in an upstream, centreline, downstream or combination configuration, depending on the pulp grading and topography. The feed pulp is separated into sand underflow and slimes overflow by hydro-cyclone. The low permeability, fine, overflow contains most of the water and is used to fill the interior of the dam, this is ideal to limit seepage losses to the foundation. The underflow contains the coarser particles with significantly less water content, which greatly reduces its susceptibility to liquefaction (Watermeyer & Williamson, 1979). The improved strength and permeability properties of this fraction makes it the ideal material for constructing the embankment wall.

The percentage of coarse material in the feed pulp is important as it dictates the split percentages and therefore the ratio of coarse building material to fine impound material. Coarse splits of 15 - 25% are normally required for embankment construction and can be achieved by proper choice of cyclone. South African practice is to use the cyclones on site, connected to a manifold pipe maintained along the crest of the dam, where the discharged underflow forms cone shaped piles. The number, size and positions of cyclones are governed by operator preferences, ease of moving and handling, quantity of tailings and the need to distribute the overflow around the perimeter for pond control. Growth of the coarse embankment must exceed the growth of the inner fines beach. This requires careful design of the dam shape and size to ensure adequate embankment fill material.

Cycloned walls usually perform better under seismic loading conditions and are able to handle higher rates-of-rise at the expense of operating costs. These advantages are not as great with the more uniform gold tailings with its poor split. Cyclone construction is therefore used only in specialised applications where high rates-of-rise, steep topography, large freeboard requirements and high rainfall conditions warrant the extra expense.

- (f) **Thickened Discharge Method:** First proposed by Shields (1974) and later developed by Robinsky (1975; 1978) the thickened discharge method comprises the dumping of de-watered tailings slurry into a conical pile. Seepage is collected in a small dam downstream of the pile. Palmer and Krizek (1987) developed a flow prediction model for the thickened slurry to determine the thickness and lateral extent of a deposited layer and thus the shape and volume of the pile.

Advantages: No risk of static failure.

Cost of embankment construction virtually eliminated.

Smaller surface area required.

Moderately resistant to liquefaction (Jeyapalan, 1982a).

Seepage contamination drastically reduced.

Less time for consolidation.

Disadvantages: High cost of thickener construction and operation.

Pumping of thickened slurry difficult and costly.

Susceptible to erosion from runoff water, best suited to flat areas.

Underground Disposal

Underground disposal of tailings is primarily used for backfill and open pit infill purposes. Disposal purely for storage is of secondary importance. Only the coarse tailings sands are used for underground disposal. The slimes need to be disposed of on the surface and may

cause additional problems not offset by the benefits of underground storage. Artificially cemented and thickened tailings slurries have been used successfully to infill the rock-skeleton of stiff underground support systems (Blight, 1988). These support systems are mainly used to augment highly stressed pillars. Another use of underground tailings disposal is to provide point supports in the form of yielding, horizontally reinforced, tailings packs. The yielding properties of these packs are ideally suited for this purpose.

- Advantages: Can be used to provide underground support or a working floor.
Maximises ore recovery by allowing pillars to be mined in backfilled areas
- Disadvantages: Requires coarse tailings of high permeability and low compressibility.
May require the addition of cement when used as structural support.
Requires surface disposal of slimes.

2.4.7 Water control in and around surface impoundments

Water plays a leading role in any impoundment failure whether by overtopping, slope stability failure, seepage related failure, liquefaction or erosion of the embankment wall. The Water Balance or difference between inflow and outflow is used to predict long term accumulation of excess water in an impoundment. Water inputs to a tailings impoundment originate from three major sources, mill water discharge (process water), direct precipitation and external runoff or flood water. Water leaves the impoundment either as mill reclaim water, evaporation, seepage, direct discharge by overtopping, or might be stored in inter-particle voids on the dam. The water balance is controlled by implementation of flood handling and inflow control mechanisms of which adequate freeboard is the most important, as well as surface water decant facilities and seepage water drain systems. After balancing the water storage needs, a further 0.5 m freeboard is usually required by regulation (McPhail & Wagner, 1989). Van Zyl and Harr (1977) provide some guidelines as to the sizing of blanket drains to aid in the control of the phreatic surface near the embankment wall. Water balance is becoming a key issue in environmental and regulatory evaluations so that control measures should be selected to minimise environmental impact through contamination. To this extent seepage barriers, seepage return systems and effluent modification prior to deposition may be required.

The position of the phreatic surface in a tailings impoundment (Figure 2-10b) is determined by the water balance (Figure 2-10a) and plays a major role in controlling slope stability of the embankment. It is essential that the pond remains located over the settled tailings in the pond and kept at a safe distance from the dam crest, thus maintaining as wide a beach as possible. This helps to prevent the phreatic surface from moving too close to the downstream slope of the embankment and assists the task of any drains that have been provided to limit the downstream advance of the phreatic surface. Keeping the phreatic

surface well below the beach and embankment wall not only helps to control the phreatic surface, but also improves densification in these areas due to desiccation suctions. This leads to increased strength and stability, as well as a reduction in the volume of the impounded mass leading to increased storage capacity.

Originally (pre 1960) tailings dams did not provide for drainage systems at the toe of the embankment. For these dams seepage erosion on the embankment slopes are critical in undermining slope stability. Fortunately the rate-of-rise and foundation drainage provided stability in most cases with feasible dam heights in the order of 15 to 20 m. Since 1959, toe and blanket under drains have been incorporated in dam design, which provides a control point well inside the structure for drawing down the phreatic surface (Wagener & Wates, 1982), and to intercept seepage water percolating through the upper soil horizons. The drainage system may consist of granular blankets, strip drains or drainage pipes located beneath the downstream slope of the embankment. Water entering the blanket and toe drains is transported to a solution trench (Figure 2-4) by drain outfall pipes. The solution trench is usually an open, trapezoidal, unlined channel located around the perimeter of the dam. The word solution indicates that it collects contaminated water from the impoundment, either as surface runoff or seepage water. The solution trench usually drains into the penstock drain channel (Figure 2-10) towards a water reservoir or evaporation dam. Depending on the depth of the channel the solution trench may in addition function as a cut-off trench.

Drainage layers must be carefully designed if they are to function satisfactorily on a long term basis. The drain must be more permeable than the adjacent soil to allow free drainage, but should be graded to prevent clogging of the drain by passage of soil particles into the drainage layer. With drainage systems in place feasible dam heights have increased to 60 m and more. However, poor deposition practice, rapid rate-of-rise and poor pond control, negates this advantage. Seepage erosion at the toe of an embankment has in the past been remedied by rock-fill buttressing with a proper filter, at considerable expense.

In a discussion on the effect of drainage layers below the embankment, Van Zyl (1993), agrees that, during initial stages of deposition, the drainage layer enhances dewatering of the deposited tailings. After the drain is covered with tailings, however, the permeability of the tailings directly above the drainage layer can reduce to as low as 5×10^{-7} cm/s through consolidation, effectively forming a seepage barrier. This process is dependent on the fineness and clay content of the tailings immediately above the drain. If the area below the pond becomes "lined" in this way, it can effectively reduce seepage losses and contamination of groundwater. Kealy and Busch (1979) have found that pond fines, in contact with the natural soil foundation, can be consolidated to a greater extent than the

material immediately above it due to suctions in the natural soil, thus reducing its permeability and forming a similar liner to that over blanket drains.

Penstock decant facilities (Figure 2-10a) and floating barge pumps are commonly used to extract supernatant tailings water and stormwater from the surface of an impoundment. The number and size of these structures are dictated by minimum freeboard requirements. A penstock consists of a series of inlet boxes with vertical risers (concrete rings or steel pipe sections typically 500 mm in diameter) connecting to a horizontal steel or concrete pipe outlet in the foundation of the dam. The outlet discharges decant water for reclamation outside the dam area. After a storm, one or more riser rings may be removed to accelerate de-watering of the pond and to maintain freeboard, but management of the riser rings is hazardous due to the suction effect of the inflowing water. Hydrological studies indicate that a freeboard of at least 0.5 m is sufficient to accommodate a 1:100 year storm of 24 hour duration (Smith et al., 1987). As the geometry of the basin can change during the operational life of an impoundment, it is common practice to maintain a minimum vertical freeboard of 1.0 m between the storage area and the embankment wall. The majority of minor incidents on tailings impoundments are from penstock failures (Wagener & Jacobsz, 1999). These failures require the installation of a floating barge type decant system or a floating penstock system to keep the dam operational.

With barge systems, a primed pump is located centrally in the pond area with the decant pipe floated across the pond to the embankment. Barge systems are considered structurally more reliable but require a lot more maintenance, and are only practicable where there is a large year-round pond.

Where penstock drains are used, access is provided by pond walls and catwalks. Pond walls are extensions of the daywall into the night area, which are raised with the daywall. These walls stretch from the closest external wall to near the penstock. Only a limited length of catwalk then connects the pond wall with the penstock inlet. Catwalks consist of gumpole uprights and wooden plank walking surfaces. These have to be raised periodically by extending the uprights. Pond walls can be useful to maintain pond control over the penstock and increase the effective length of the beach, providing longer flow paths to the penstock and more time for segregation to take effect.

Storage policies under return water management dictate that water should not be stored on the surface of a tailings impoundment unless the structure is specifically designed for this purpose. The pond area should be kept at minimum and just deep enough to ensure water clarity for decant purposes. A return water system should discharge, treat or store water at

rates above the rate of decant. Return water should either be recycled or allowed to evaporate, thus minimising loss of recycle water and the risk of discharge pollution.

In addition to internal and surface drainage measures, ground level catchment paddocks are provided around the perimeter of the dam to intercept runoff from the side slopes of the impoundment. These paddocks are designed to hold and eventually evaporate the runoff water. It is common practice on many operational dams not to store water in the paddocks, which is drained into the solution trench by penstock drain systems similar to that of the main dam. The paddocks also serve to limit siltation pollution by material eroded from the slopes. Side slopes of the embankment wall are protected against stormwater erosion by providing a stepped profile with horizontal berms. The berms are drained after a storm to the catchment paddocks by berm decant penstocks if necessary. Water should never be retained or stored on the berms since this can lead to instability problems of the embankment wall. Factors influencing the frequency and size of step-back berms include stability constraints of the intermediate sections (i.e. between step-backs), erosion potential of the material on the side slopes and sufficient width to permit access to operations tractors and vehicles on the berm (Smith et al., 1987).

2.4.8 Design Considerations and Stability Analyses

Tailings dams are generally not designed with the same conservatism as conventional dams. The aim is to balance safety (stability) and economics (storage capacity) with embankment slope. The design of tailings structures must recognise that stability is required not only during the operational life of the structure, but also for generations after mining has been completed. Most embankment failures, with the exception of liquefaction related failures, occur as flow slides, triggered by a rotational type slide. In line with this, most stability calculations rely on common slope stability methods based on limit equilibrium. Invariably these procedures will identify a shallow critical failure, approaching the infinite slope condition. The designer has to be careful and experienced to interpret the results from these classical procedures.

According to Vick, (1983), errors result mainly from incorrect input parameters and not from analysis procedures. A typical analysis can be broken down into the following steps:

- (a) **Phreatic Surface:** A critical step in the analysis of embankment stability is determining the location of the phreatic surface and pore pressure distributions. In a simplified model the pore pressure distribution can be derived from assumptions of steady state gravity seepage in an homogeneous and anisotropic (typically $k_h/k_v \approx 10$, Pettibone & Kealy, 1971; Vick, 1983) material. However, this does not account for permeability variations due to segregation across a section through the dam. For more accurate

results non-steady, transient flow in an inhomogeneous and anisotropic material has to be modelled and account be taken of unsaturated capillary flow in addition. Other aspects that are of importance include variations of void ratio and permeability with depth and increased confinement stress as well as consolidation related seepage for rapid rate-of-rise situations.

- (b) **Stress Distribution:** Total stresses are calculated from the vertical overburden pressures based on in-situ densities. The effective stress distribution can then be determined as the difference between the total stress distribution and the pore pressure distribution and estimates of the earth pressure coefficient.
- (c) **Failure Criterion:** A Mohr-Coulomb type failure envelope is developed from laboratory triaxial or shear box tests on reconstituted samples. Stress levels correspond to the calculated in-situ effective stresses. Stress relief and recompression cycles may be employed to simulate field loading conditions as well as overconsolidation due to suction pressures.
- (d) **Embankment Stability Analysis - Static:** Design for static stability requires a design shear strength from the assumed failure criterion, usually Mohr-Coulomb, and is carried out for the following scenarios:
- *End of construction:* Applied to starter dikes and conventional earth dam walls approaching fast undrained loading conditions following construction.
 - *Staged construction:* Especially useful for raised embankments that are constructed to allow dissipation of excess pore pressures. Vick (1983) reports that if the rate-of-rise is less than 5 to 10 m/yr, complete dissipation of excess pore pressures is usually assumed with a corresponding drained effective strength. Regardless, pore pressure dissipation appears to be faster than rates predicted from one-dimensional consolidation theory, principally due to the presence of thin sand seams and horizontal consolidation. Ladd, in the 1991 Terzaghi Lecture, emphasises the need for stability analyses to use a shear strength consistent with the most likely drainage condition during a potential failure. Many assumed drained failures have shown no dissipation of excess pore pressures during shear. Loading conditions that can lead to undrained instability include: normal operation at too high a rate-of-rise in low permeability tailings preventing excess pore pressures from fully dissipating, excessive foundation strains and slope oversteepening caused by rapid internal or surface erosion (Lo et al., 1988). For these cases a conservative, undrained, failure strength should be chosen based on in-situ effective stresses taken as equal to the consolidation stresses at the time of failure.

Ladd argues that such an undrained strength method of analysis should be used for staged construction problems where full dissipation of excess pore pressures have not occurred. Another point for consideration is the strain history and stress-strain behaviour of the material in terms of peak vs. steady state strengths coupled with conditions of strain hardening vs. strain softening. Peak strengths can only be relied upon if the designer can be certain that the material has never been strained beyond the level of peak shear stress; on the other hand post peak softening in a collapsible soil structure can lead to liquefaction.

- *Long term steady seepage*: Intended for raised embankments at their maximum height. Excess pore pressures are assumed to be fully dissipated and redistribution of internal stresses completed. The assumption of steady state seepage is a conservative measure and the structure is assumed not to fail in the long term. Therefore, shear induced pore pressures can be ignored.
- *Rapid draw down*: Only applicable to impoundments designed to retain substantial amounts of flood water, where the water breaches the embankment or is released to induce a rapid draw down situation. This method can also be applied to investigate the stability of an embankment wall following liquefaction of the impounded slimes during a seismic event.

(e) **Embankment Stability Analysis - Dynamic (Seismic)**: According to Lo et al. (1988), undrained instability and liquefaction of a tailings impoundment may be brought about by dynamic loading from earthquake shaking and blasting shocks. Often processes responsible for ore genesis are associated with seismic activity. Cyclic loading of this nature may result in a loss of strength caused by excess pore pressure build-up, strain softening and remoulding. Liquefaction of the impounded slimes alone may lead to embankment instability by increasing the external load on the upstream slope of the embankment (Klohn & Maartman, 1973; Finn & Byrne, 1976; Klohn et al., 1978). Liquefaction of the internal slimes can amount to 25% increase in load on the embankment wall as a result of hydrostatic conditions in the liquefied pond. Even if a liquefaction type flow failure is not likely, excessive deformations resulting from the seismic loads can cause overtopping of free water on the dam and result in a failure (Harper et al., 1992). Blasting shocks on the other hand are of too high a frequency to influence a dam of typical low natural frequency.

Seismic risk at a specific location can be estimated from seismic coefficient charts based on historical data, probabilistic methods which in addition take into account probabilities of larger events, or from deterministic methods which predict the maximum credible acceleration based on site geology and give no regard to seismic history. The design acceleration should be based on the hierarchy of accelerations

given by the above methods. For low level activity the historic approach may be enough. In more active areas the probabilistic methods should be used and in areas of high seismicity with active faults, both the probabilistic and deterministic predictions should be considered.

Factors influencing the liquefaction susceptibility of tailings during a seismic event are reported to be:

- *Stress related factors:* Initial effective consolidation stress and stress anisotropy, magnitude and direction of cyclic applied stresses and the number of cycles of stress reversal.
- *Material related factors:* These include, grain characteristics and grading, method of deposition, ageing, strain history, and the state of consolidation or relative density (Harper et al, 1992). Vick (1983) states that tailings are more susceptible to liquefaction than natural soils due to the fact that in most cases tailings structures consist of loose, uncompacted, fine sands with excess pore pressures likely to be induced by stress reversal.

Design approaches are either empirically based on the histories of other similar tailings dams or on analytical procedures using simplified liquefaction analyses (steady state analysis), pseudo static methods, total stress equivalent linear dynamic analyses or complete effective stress dynamic analyses using sophisticated numerical techniques.

Klohn et al., 1978 propose a pseudo static analysis with seismic coefficients corrected for dynamic response as a conservative approach using the following assumptions:

- *Dilatancy during shear:* Strength recovery (Castro & Christian, 1976) due to dilatancy for uncompacted fills is neglected; with compacted fills strength recovery is a definite possibility.
- *Principal stress ratio:* Assume the stress ratio (σ'_h/σ'_v) on the failure plane to be equal K_0 , the earth pressure coefficient at rest.
- *Pore pressure migration from a liquefied pond:* If beach and embankment zones are significantly saturated, excess pore pressures will migrate from the pond into these saturated outer zones as well as into saturated foundations. Pore pressure migration can be ignored for dams with proper drainage.
- *Redistribution of stresses:* As sections fail due to rising dynamic pore pressures, stress redistribution follows and may result in instability of previously stable areas.

In seismically active areas, the risk cost of seismic failure has to be added to construction cost to evaluate alternative design alternatives, (Vick et al., 1985).

2.4.9 Seepage and Contaminant Transport Analyses

The objectives of seepage analysis are to determine the amounts of seepage losses and the risk of groundwater contamination. The design of drainage structures is determined by dam geometry, tailings properties and the nature of the foundations, which influence the seepage regime (Wates, 1991). Design takes account of prior geologic, hydrologic and geochemical conditions, as well as the physical and chemical characteristics of the tailings, effluent and natural soils, to predict the need for seepage control and to pose alternative measures that will minimise environmental impact at a reasonable cost. Seepage quantities can be estimated using closed-form solutions for simple geometries and seepage assumptions, or from more complicated numerical techniques in critical situations. The buffering effect of chemical reactions with natural soils can play a major part in controlling contamination of groundwater systems, especially if these soils are partially saturated and above the groundwater table.

Seepage Analysis

Flow quantities from a tailings impoundment can be estimated quite accurately from approximate methods based on analytical equations or flow net solutions (Van Zyl, 1987). In these cases it is standard to assume:

- steady state unconfined flow with respect to the boundary conditions,
- Darcy's law to be valid,
- saturated flow, neglecting any unsaturated flow and
- spatially varied isotropic or anisotropic hydraulic conductivity.

However, the calculations are most sensitive to changes in hydraulic conductivity. Mittal and Morgenstern (1976) show that due to rapid rates-of-rise, slimes can be under-consolidated in the pond, locking in high excess pore pressures and violating the conditions of steady state seepage.

Barrett (1987), gives a summary of commonly used closed form seepage analysis methods:

- *Flow net*: Although well developed and able to predict seepage quantities accurately, the method is cumbersome to use and results in unreliable estimates of the pore pressure distribution.
- *DuPuit*: The DuPuit solution assumes vertical equipotential lines between sections and an impervious foundation, which results in a small inclination of the phreatic surface.
- *Kozeny (1931)*: Solution to Kozeny's equations prove accurate for a parabolic phreatic surface assuming an impervious foundation and a continuous drainage blanket on the downstream side of the embankment.

- *Kealy and Busch (1969)*: By solving the LaPlace and Richards' equations for phreatic surface location and axisymmetric plane flow in a porous medium this method is able to account for spatial variation and anisotropy in permeabilities.
- *Abadjiev (1976)*: Abadjiev attempts to account for variations in vertical permeability due to stress increase with depth and provides solutions that are accurate within the assumption of an impervious foundation.

Van Zyl and Harr (1977) present a number of standard solution charts for homogeneous isotropic seepage conditions in tailings impoundments. These charts are based on assumptions stated to model the most conservative condition including homogeneous and isotropic material properties, steady state flow and full discharge of all water entering the drainage systems at atmospheric pressure. Solutions are given for the following cases:

- *Material extends to infinite depth with a drainage layer at finite depth*: Nelson-Skornyakov (1949), gave a closed form solution to this problem for horizontal and near vertical upstream faces by applying conformal mapping techniques. Equations are derived for the phreatic surface and the effective length of the drain. Harr also presents an equation for the factor of safety against piping.
- *Horizontal drain underlain by impervious material of infinite extent*.
- *Impervious base*: Solution follows directly from Kozeny's equations.

Special conditions that are not easily analysed by simplified models include: consolidation which has the effect of decreasing hydraulic conductivity as the void ratio decreases and the effect of evaporation on seepage characteristics in the unsaturated zone. In addition, the pore pressure regime is not as easily predicted with approximate methods as the flow quantity. The phreatic surface is usually conservatively placed for stability calculations.

For more advanced analyses techniques reference should be made to:

- (a) **Consolidation Effect** - Gibson et al. (1967 & 1981): Gibson and his co-workers made use of a one-dimensional, finite non-linear strain, numerical method in modelling the effect of consolidation on seepage quantities from sub-aqueously deposited material. All theories of one-dimensional consolidation are special cases of this method and no closed-form solution has yet been found. Numerical procedures have been developed and are discussed by the authors. The method requires as input, relationships between void ratio and effective confinement stress as well as between void ratio and permeability. These relationships can be established from standard laboratory tests. They found permeability reductions of up to one order of magnitude or more as a result of consolidation. These reductions can show the blinding effect on under drains that can push up the phreatic surface; on the other hand they may be of benefit in modelling the reduced seepage to the groundwater below the pond.

- (b) **Evaporation Effect (Sub-aerial deposition)** - Bartlett and Van Zyl (1984): Unsaturated flow in the flat beach areas can either be vertical downwards seepage of water bleeding from subsequent depositions, or vertical upwards flow due to evaporation from surface. Evaporation has a greater influence on reducing water content in fine tailings than in coarse tailings, although the fine tailings reach equilibrium at a higher water content. The authors show that neglecting unsaturated flow can lead to underestimates of seepage losses.

Contaminant Transport Analysis

Methods that are commonly used in simplified contaminant transport analysis include the following:

- (a) **Lumped Parameter Methods:** Used for preliminary design and to check the results of other methods. This method incorporates: water and salt balances, storage in partially saturated zones, neutralisation capacity and attenuation distances.
- (b) **Analytical Methods:** These methods are employed when preliminary estimates indicate a high contamination potential, or if the tailings contain a high degree of toxic compounds. In some cases it is required by regulatory agencies. One-dimensional saturated and partially saturated flow according to Darcy's law is assumed together with a uniform geometry and uniform material properties to allow direct calculations.
- (c) **Numerical Methods:** Highly sophisticated numerical techniques are only justified in cases where it may provide significant cost advantages such as with very complex dam geometry and geology, or if detailed analysis of two dimensional partial saturated flow is required. These methods attempt to model simultaneous and interactive processes of dilution of contaminant concentration, changes in contaminant solubility due to pH, contaminant adsorption, etc.

2.4.10 Tailings Dam Disasters

According to mining folklore, no tailings dam has ever been completed without at least one failure occurring during construction (Smith, 1972). Although the term failure is rarely defined in these stories, it can be assumed to include everything from a slight nonconformity with the design to complete collapse. It has been estimated that over 70% of large Canadian mining operations have experienced waste dump failures of some kind or the other (Hoare & Hill, 1970). Numerous waste embankment failures have occurred worldwide but were not reported as they did not involve any fatalities. The consequences of

failure have three major components: property damage costs, environmental damage and loss of life. By far the most common cause of distress is the control of the phreatic surface and maintaining adequate freeboard for stormwater confinement. Past failures indicate that designers often underestimate the probable maximum rainfall, and that the pond is operated with less than the required minimum freeboard or is allowed to encroach on the embankment wall due to poor deposition practices or too high a rate-of-rise.

Failure of a tailings impoundment can arise from a number of mechanisms, including:

- Foundation failure.
- Liquefaction.
- Slope instability from local sloughing to massive circular arc slides.
- Overtopping by floodwaters.
- Erosion of the face - Blight (1989).
- Piping in either the dyke or foundations - Wates et al. (1999).
- Failure in the decant facility - Wagener and Jacobsz (1999).

Except for liquefaction, all types of failure give some warning signs. Liquefaction events occur rapidly with little or no warning and the consequences and impact they have on their surroundings, the environment, and on human lives can be catastrophic, (Papageorgiou et al., 1997). For other types of distress, signals such as cracking, wet-spots on the downstream embankment face, critical settlement, and piezometric trends all indicate deficiencies in the structure. Without proper instrumentation and supervision it may be difficult to interpret accurately the extent of the problem.

The United States Committee on Large Dams has recorded failures, accidents and groundwater-related incidents for tailings dams, USCOLD (1994). The compilation of 185 incidents (21 in gold tailings) is intended to emphasise those potential failure modes of greatest significance in design. The major causes of tailings dam incidents and failures are reported to be:

- Slope instability of the embankment wall.
- Seepage instability of the embankment.
- Foundation failure.
- Overtopping of the dam crest.
- Structural deficiency of spillways, decant structures or discharge pipes.
- Earthquake (seismic) shaking.
- Mine subsidence caused by underground workings.
- Erosion damage.

Most failures involve active dams with only a small percentage of incidents on inactive dams. Slope instability is reported to be the leading cause of incidents, especially due to earthquakes. Other than that, the majority of failures have been due to erosion by water, either overtopping the dam or piping through some weakness (Penman, 1994). Upstream dams dominate failure records but may be disproportionately represented. Upstream dams are especially prone to slope instability and excessively vulnerable to earthquakes. Table 2-4 provides a list of some of the incidents mentioned in the USCOLD report.

The most important lessons learned from a study of past failures in tailings impoundments are the need for a comprehensive monitoring program and continuous detailed analysis of all data, backed up by rapid remedial action on identifying possible problems.

2.5 TAILINGS AS AN ENGINEERING MATERIAL

This section presents a summary of the mechanical properties of gold tailings as reported in the literature. It is intended as a basis for interpreting experimental test results and to augment such results for the purposes of defining the state and composition of gold tailings. Knowledge of the composition and state should enable the engineer to predict the engineering behaviour of tailings impoundments under typical load conditions. Where applicable, publications on tailings other than gold are listed separately.

2.5.1 The Nature of Gold Tailings Slurries

Following comminution and chemical extraction of the gold valuables the by-product of the reduction plant, tailings, are transported to the impoundment in the form of a slurry or pulp for disposal. It is the properties of this slurry, governing its soil forming behaviour on the impoundment, which are the subject of this section.

Tailings Slurries

Gold tailings have been classified as a fine, hard and angular rock flour, with 0 - 15% fine sand, 80% silt and 0 - 10% clay sized particles slurried with process water, (Vick, 1983; McPhail & Wagner, 1989). Cowey (1994), breaks down the mineral constituents of gold tailings into non-sulphide components (silicates, oxides, carbonates, etc.) and sulphide minerals (pyrites, etc.). In a typical South African gold tailings slurry pulp density varies between 25 and 50%, which relates to a moisture content of between 280 and 100% (Wagener & Wates, 1982).



Table 2-4: Summary of some significant tailings dam incidents since 1917.

Year	Location	Tailings	Cause	Fatalities	Reference
1917	South Africa	Gold			White, 1917
1928	Barahona, Chilly	Copper	EQ ²	54	Dobry & Alvarez, 1967
1937	Simmer & Jack, South Africa	Gold	SI		Donaldson et al., 1976
1938	Fort Peck, USA	Sand	SI	80	Casagrande, 1965
1944	Hollinger, Canada	Gold	FN		Blackshaw, 1951
1956	Grootvlei, South Africa	Gold	SI		Donaldson et al., 1976
1961	Jupille, Belgium	Fly-ash	SI	11	Bishop, 1973
1965	El Cobre, Chilly	Copper	EQ	200	Dobry & Alvarez, 1967
1966	Aberfan, UK	Coal	SI	144	Bishop et al. 1969
1970	Mulfulira, Zambia	Copper	MS	89	Brawner, 1979
1972	Buffalo Creek, USA	Coal	SI	118	Jeyapalan et al., 1982a & b
1974	Bafokeng, South Africa	Platinum	SE	12	Jennings, 1979
1978	Mochikoshi, Japan	Gold	EQ	1	Okusa & Anma, 1980
1978	Hirayama, Japan	Gold	EQ		Okusa et al., 1980
1978	Norosawa, Japan	Gold	EQ		Okusa et al., 1980
1978	Arcturus, Zimbabwe	Gold	SI	1	Shakesby & Whitlow, 1991
1980	Phelps-Dodge, USA	Copper	SI	54	
1981	Freestate, South Africa	Gold	SI		Sully, 1985
1983	Golden Sunlight, USA	Gold	GR		
1983	Grey Eagle, USA	Gold	GR		Hutchinson et al., 1985
1984	Battle Mt. Gold, USA	Gold	SI		
1985	Olinghouse, USA	Gold	SE		
1985	Stava, Italy	Fluorite	SI	268	Berti et al., 1988 Chandler & Tosatti, 1995
1987	Montana Tunnels, USA	Gold	GR		Clark et al., 1989
1994	Merriespruit, South Africa	Gold	OT/SI	17	Wagener, 1997 Wagener et al., 1997a, 1998
1995	Omai, Guyana	Gold	ER		Vick, 1996
1995	Golden Cross, New Zealand	Gold	O		WISE Uranium Project, 1998
1995	Placer, Philippines	Gold	FN	12	WISE Uranium Project, 1998
1995	Simmergo, South Africa	Gold	SI		Blight, 1997

² SI - Slope instability, SE - Seepage, FN - Foundation, OT - Overtopping, EQ - Earthquake, MS - Mine subsidence
GR - Groundwater, O - Other.

It appears that gold tailings fines generally exist in a flocculated state in a typical delivery slurry. When subjected to sedimentation tests, significant differences are noticed in the results of tests, whether performed with a dispersing agent or not. Non-dispersed slurries immediately form an interface between clear solution above and a shrinking mobile mass below the interface. It is as if the flocculated particles, in the absence of a dispersant, sweep the water clear of colloids as it "consolidates" in an homogeneous shrinking mass of low density, Truscott (1923). If treated with dispersant, the fines segregate and settle to form a dense sediment at the bottom of the sedimentation cylinder in the normal manner. There is also a significant time difference between the two processes, with non-dispersed sedimentation taking only a fraction of the time of sedimentation in a dispersed state. Hamel and Gunderson (1973) also report the differences in settlement behaviour between dispersed and non-dispersed tailings slurries. Having studied the effect of flocculants and dispersants on the settled density of Australian high clay content gold tailings, Fell (1988) concluded that the settled density could be improved by adding small quantities of dispersants. Fell also concluded that the settling properties did not alter with time, provided the material did not dry out. Furthermore, the density was not affected by initial water content, provided it was high enough.

A great deal of research has been carried out at the University of the Witwatersrand by Blight and his co-workers on the shear strength properties of tailings slurries (Blight & Bentel, 1983; Blight, 1988; Blight, 1994). These studies were aimed at modelling the flow properties and friction losses of tailings slurries in pipes, channels and on dam beaches, as well as predicting the extent of post failure debris flows from breached tailings impoundments. Tailings slurries have been found to behave somewhere between a Bingham plastic and a Newtonian fluid. The shear strength of a Newtonian fluid can be expressed by,

$$\tau = \frac{d\gamma}{dt} \eta \quad \text{Eq. 2-5}$$

where τ = shear strength
 γ = shear strain
 η = Newtonian or 'true' viscosity
 t = time

and that of Bingham plastics by,

$$\tau = \tau_0 + \eta \frac{d\gamma}{dt} \quad \text{Eq. 2-6}$$

where τ_0 = strength when the rate of shear strain $d\gamma/dt$ is zero, a type of yield stress.

Using a variable speed coaxial-cylinder type viscometer the apparent viscosity, η_a , can be determined and used to approximate the rheological behaviour of a tailings slurry using the relationship:

$$\eta_a = \frac{\tau_o}{dy/dt} \quad \text{Eq. 2-7}$$

Eq. 2-7 in effect approximates the tailings slurry to a Newtonian fluid. The approximate viscosity of water is one, and that of a mixture of soil and water is greater than one, depending on solids content. The dilute mixtures of sediment and water in rivers still have apparent viscosities close to one, whereas in gold tailings slurries the apparent viscosity ranges between 2 and 7. Changes in water content have a major influence on the apparent viscosity and shear strength of the slurry. Shear strength, detectable by the viscometers, only develops at a water content of less than 30%, where the slurries behave more like Bingham plastics than Newtonian fluids. At lower water contents a threshold shear strength starts to dominate the viscosity.

Hungr (1995) also developed a continuum model to simulate post failure debris flow, based on a Lagrangean solution of the equations of motion and a semi-empirical numerical model of unsteady flow. The model allows selection of a variety of material rheologies as well as variances in rheologies along the slide path, or within the sliding mass. Effects of lateral confinement are accounted for in a simplified manner. The model compares favourably with results from laboratory experiments and other analytical methods. Han and Wang (1996) used the assumption of a Bingham plastic to successfully model a breach failure in China.

Effluent

Gold tailings water is often slightly alkaline with high concentrations of soluble salts. It often contains soluble sulphates, chlorides, sodium, calcium and low concentrations of organic chemicals (Vick, 1983). The effluent chemistry is the result of a combination of the ore minerals and chemical treatments in the reduction plant.

Oxidation, Ageing and Cementing

In the presence of water and oxygen, the reactive sulphide minerals in tailings can produce an acid leachate that favours the solution of heavy metals, thus becoming a severe threat to local ecosystems. In pyrite-rich tailings some crusting of the dried out exposed surfaces on a dam may occur. This crusting, however, is purely superficial.

The iron sulphides are mostly pyrite (FeS_2) in gold tailings slurries, and are especially susceptible to oxidation in the presence of water and oxygen or by anaerobic biological oxidation to ferrous and ferric oxides, ferric hydroxide and sulphuric acid (Cowey, 1994). The products of oxidation may result in volume changes as well as acidic conditions in the tailings, which are conducive to the solution and leaching of toxic heavy metals. Under saturated conditions with no access to free oxygen the oxidation of the sulphides can only take place by means of anaerobic bacterial oxidation through *Thiobacillus ferrooxidans* and *Thiobacillus thiooxidans* (Kleinmann et al., 1981). Stanley (1987) argues that the interior of a tailings dam is more or less impenetrable to air and free oxygen and that the slime is preserved in the laid down condition. However, where air and water penetrate the beaches and embankment slopes rapid oxidation of the surface sulphide minerals will take place, within one month, up to a depth of 0.5 to 1 m.

A number of workers have commented on the self cementing properties of tailings (Patton, 1952; Thomas, 1971; Pettibone & Kealy, 1971). Cementation may take the form of:

- Chemical bonds between clay mineral as a function of their pore water chemistry.
- Thixotropic hardening caused by clay minerals.
- Crystal bridges between grains formed by precipitating soluble salts.
- Hard cementing between grains on precipitation of sulphates
- Precipitation cementing by silicates, metal oxides or calcium carbonate following pH changes as a result of pyrite oxidation.

Solubility of natural soil bonding agents such as silica, iron and carbonate, known to be present in gold tailings slurries, are affected by the pH, oxidation potential, temperature and pressure of the solution. Changes in these parameters caused by pyrite oxidation, can lead to precipitation bonding. Silica is highly soluble in an alkaline environment but relatively insoluble in neutral to acidic conditions. Iron precipitation usually follows neutralisation of acidic conditions. Carbonates are soluble in acidic conditions.

For work published on tailings slurries other than gold, reference can be made to: Copper: Finn et al. (1978), Volpe (1979), Chen et al. (1988), Mlynarek et al. (1991), Mlynarek et al. (1994); Iron: Guerra (1972); Uranium: Matyas et al. (1984); Larson and Mitchell (1986); Lead-Zinc: Mabes et al. (1977); Molybdenum: Klohn (1984); Lignite: Kotzias et al. (1984); Coal: Siriwardane and Ho (1985); Canadian Oil Sands: Suthaker and Scott (1997); non-ferrous: Abadjiev (1985), Abadjiev (1988).

2.5.2 Deposition and Sedimentation of Tailings

The formation of a hydraulic fill is very much like the formation of natural sediments; transportation is followed by deposition, sedimentation and eventually by consolidation due

to self-weight and external loading (Imai, 1981; Schiffman et al., 1988). The difference is age; compared with natural sediments, hydraulic fills are young with a very recent stress history. The study of the sedimentation and consolidation behaviour of tailings following deposition is of major concern in the design of tailings impoundments, not only from a stability point of view but also in determining the storage capacity and allowable rate-of-rise on these structures.

Transportation of sediments, sedimentation and consolidation are simultaneous parts of the tailings deposition process (Consoli & Sills, 2000). It is important to note that much of the work on sedimentation-consolidation of natural sediments and tailings slurries has thus far been restricted to the solution of one dimensional vertical settling problems. The study of the sedimentation of tailings should not be limited to the modelling of vertical sedimentation only, but should also include the effects of horizontal transportation. Tailings particles should be regarded as moving through and being carried along by water following deposition.

From a stress history point of view the profile in a tailings impoundment should be soft, normally consolidated, with high void ratios and low effective stresses. In reality, desiccation suctions developed from evaporation on the beach surfaces and embankment walls can build in large pre-consolidation stresses with heavily overconsolidated states existing below these surfaces. Benefits from sub-aerial deposition and desiccation include increased storage capacity, reduced permeability and seepage, reduced compressibility and increased shear strength due to densification. Another consequence of desiccation is the development of drying cracks. These cracks, even if filled in by successive depositions, remain weak surfaces within the dam, presenting potential piping channels and preferential paths for pyrite oxidation as the dam ages (Blight & Bentel, 1983).

A consistent feature of tailings impoundments is the highly layered nature of the profile as a result of depositional practices, soil forming processes and variations in milling consistencies, (McPhail & Wagner, 1989). Often this layering takes the form of alternating fine and coarse deposits, with up to 50% variation in fineness over a depth of 10 to 200 mm.

Sedimentation Behaviour

Compared with natural sediments, predicting the sedimentation behaviour of tailings is complicated by a number of factors, (Lappin, 1997):

- Small depth of flow.
- High sediment load.
- Turbulent flow close to the discharge point and in scour channels on the beach.

- Particles being trapped in the bed load or in flocculated structures.
- Coarse particles held in suspension or rolled along the bed.
- Suctions in the bed trapping fine particles.

In addition to these difficulties, account has to be taken of both the slurry properties including particle size, shape and specific gravity, water content and viscosity as well as deposition-related flow conditions on the beach (sub-aerial), compared with those in the pond (sub-aqueous).

Under sub-aerial conditions on the beach, sedimentation is more or less instantaneous followed by consolidation effected by desiccation, and saturated or unsaturated compression due to overburden pressures (Senevirante et al., 1996). Under sub-aqueous conditions in the pond, sedimentation is initially rapid forming a loose soil of no strength, which gradually changes into a very low strength loose soil structure consolidating under its own weight and overburden pressures. Sub-aqueous sedimentation can be modelled using Kynch's theory of hindered settling by considering continuity of the solid phase together with the particle velocities as a function of porosity. This leads to Kynch's "flux-plot" used in the design of sedimentation processes (Kynch; 1952, Been, 1980; Imai, 1980 & 1981; McRoberts & Nixon, 1976; Schiffman et al., 1986). Ulrich and Valera (1995) point out that, being a ground product, tailings particles are usually quite angular. In addition the fines seldom exhibit cohesion. This results in high void ratio packing arrangements with little particle-to-particle contact following sub-aqueous deposition, which must be susceptible to liquefaction.

Consolidation Behaviour

Consolidation involves primary consolidation as a result of the dissipation of excess pore pressures and secondary consolidation or creep as a function of the viscosity of the soil skeleton. Both occur simultaneously but primary consolidation, together with sedimentation, is believed to dominate in tailings materials.

The first coherent formulation of the consolidation phenomenon was developed by Terzaghi (1923), based on the conservation of momentum (equilibrium) and mass (continuity) for infinitesimal strain and one-dimensional flow and settlement. It is doubtful whether small strain theory can be applied adequately to the consolidation of tailings with the large reductions in void ratio under self-weight consolidation (Carrier et al., 1983; Van Zyl, 1993). Still considering a one-dimensional theory, Gibson et al. (1967) added the consequences of non-linear finite strain using the Lagrangean co-ordinate system. The large strain non-linear consolidation theory, using Lagrangean co-ordinates has been applied successfully to tailings by Gibson et al. (1967) and specifically to gold tailings by Caldwell et al. (1984) and

Senevirante et al. (1996). Horizontal permeability can dominate the consolidation process as a result of the severe anisotropy in permeability with k_h typically an order of magnitude larger than k_v . However, global anisotropy can significantly be affected by desiccation cracks and the properties of their infill material. For example, a desiccation crack filled with coarser material can vertically connect two free draining layers across a relatively low permeability layer. Biot (1941) formulated the first general and coherent multidimensional consolidation theory, taking into consideration the interdependency between deformation of the soil skeleton and the flow of interstitial fluid. With the development of finite element and finite difference numerical methods, the solution of realistic boundary problems with the three-dimensional Biot coupled consolidation theory became possible.

Recently, there has been a move towards the development of unified sedimentation-consolidation models for tailings. Experimental and theoretical studies linking sedimentation and consolidation of settling tailings have been proposed by Pane (1985) and Toorman (1996). Been (1980) and Been & Sills (1981) pointed out the relation which existed between the theories of hindered settling developed by Kynch (1952) and finite deformation consolidation developed by Gibson et al. (1967). Schiffman et al. (1984) and Pane (1985) presented a single theoretical basis for sedimentation and consolidation processes of solid-water mixtures. Studying phenomena concerning the end of sedimentation and beginning of consolidation phases, Been and Sills (1981) concluded that the basic difference between a suspension (sedimentation phase) and a solid skeleton consisting of the same particles (consolidation phase) is the presence of effective stresses (see also Carrier et al., 1983). The maximum density at which a soil slurry can exist as a suspension without the presence of effective stresses is termed the structural density. One way of determining this separation is by measuring the state (density) at which the suspension starts to exhibit shear stiffness. Another way is to take the void ratio that exists at the top of the sediment bed after a considerable period of self-weight consolidation has occurred. Although both these values correspond by definition to zero effective stress, Been and Sills have shown that there can be a significant difference in their magnitudes as a result of creep at the surface of the sediment bed. Carrier et al. (1983) proposed the following equation to calculate the void ratio at the end of sedimentation,

$$e_{sed} = 7G_s \frac{LL}{100} \quad \text{Eq. 2-8}$$

where e_{sed} = void ratio at the end of sedimentation start of consolidation
 G_s = specific gravity
 LL = liquid limit as a %

Swarbrick and Fell have worked on a unified model which includes the effects of desiccation and re-wetting for high clay content tailings at the University of New South Wales in Australia (Swarbrick & Fell, 1990; Swarbrick & Fell, 1991; Swarbrick, 1992; Swarbrick & Fell, 1992; Swarbrick, 1993; Swarbrick, 1994). Input parameters for the model are derived from large column settling tests and lysimeter drying tests. Tailings properties that influence its depositional behaviour were found to be particle density, net negative charge on the clay particles, soil water diffusion and thermal conductivity. Their model has been comprehensively evaluated against laboratory and field tests for iron, coal and bauxite tailings. However, the authors warn against the use of this and similar models for tailings with high salt concentrations, where surface crust formation will inhibit evaporation and result in erroneous predictions, see also Fahey and Fujiyasu (1994).

Hydraulic Sorting

Sub-aerial deposition and the progression of flow and sedimentation of the tailings slurry on a typical South African impoundment are well described by Bentel (1981). The slurry, discharged from an open ended pipe forms a plunge pool where the stream hits the beach. In the plunge pool the slurry swirls around and deposits tailings, dissipating much of the energy as it does so. Where the pool-rim is breached, a stream of tailings discharges onto the beach. Flow is supercritical on the upper beach and concentrated in individual scoured rivulets or channels with deposition of material taking place as a result of energy losses at hydraulic jumps. Further down the beach there are fewer hydraulic jumps spaced further apart and a gradual decline in the slope of the profile. The individual rivulets widen, meander and eventually coalesce at the end of the beach. As the slurries approach the beach-pond interface the rivulets fan out in deltas with slow sub-critical sheet flow and ripple bedding, resulting in a low profile gradient. The sudden reduction in energy at the edge of the pond leads to increased deposition.

The in-place physical properties or state of tailings, especially permeability, compressibility and strength, are often assumed to be related to the beach profile or surface geometry of the impoundment. Many authors agree that there is a tendency for decreasing permeability towards the pond as a result of increasing fineness (Jerabek & Hartman, 1965; Kealy & Busch, 1971). It is the particle segregation and hydraulic sorting processes that are responsible for this increase in fineness. Figure 2-9 shows the reduction in the normalised median particle diameter, $D_{50}/D_{50(max)}$, with distance from the point of deposition on a diamond tailings dam. The resulting beach geometry, discussed in detail in the next section, is a result of particle sorting processes, which in turn are a function of the properties of the tailings slurry and of the depositional conditions. On gold tailings dams, the beach profile is more a function of pulp density than of the slurry properties due to the generally uniform

grading of these materials. The lower the pulp density the more effective the sorting processes.

In addition to horizontal sorting on a beach, size sorting also occurs vertically resulting in the highly layered structure and anisotropy of the tailings profile (Blight & Bentel, 1983).

Deriving equations for the unhindered settling velocity of suspended sediment in a quiescent fluid undergoing laminar flow, Blight and Bentel (1983) predict the grading properties of a tailings beach as follows:

The vertical settling velocity, v_v , is calculated from,

$$v_v^2 = \frac{4(\rho_s - \rho_w)g \cdot D}{3C \cdot \rho_w} \quad \text{Eq. 2-9}$$

where: ρ_s = density of the particle

ρ_w = density of the slurry liquid

D = diameter of the particle

C = coefficient of drag, constant for Reynolds numbers greater than 400

and the horizontal settling velocity v_h , from,

$$v_h = C(\delta \cdot i)^2 \quad \text{Eq. 2-10}$$

where: δ = depth of flow

i = hydraulic gradient

thus,

$$D = \frac{C \cdot \delta^{3/2} \cdot \sqrt{i}}{K \cdot x} \quad \text{Eq. 2-11}$$

$$\text{where: } K = \frac{4(\rho_s - \rho_w)g}{3C \cdot \rho_w} \quad \text{Eq. 2-12}$$

Eq. 2-11 gives the maximum diameter, D , of a particle expected at a distance x along the flow path down a beach.

Alternatively, a similar equation can be derived following Stokes' Law, as:

$$v_v = \frac{(\rho_s - \rho_w)g \cdot D^2}{18\eta} \quad \text{Eq. 2-13}$$

where η = viscosity of the slurry

then

$$D = \left[\frac{C \cdot \delta^{3/2} \cdot i}{K' \cdot x} \right]^{1/2} \quad \text{Eq. 2-14}$$

where $K' = \frac{(\rho_s - \rho_w)g}{18\eta}$ Eq. 2-15

These equations are approximate and do not account for:

- Interference of adjacent particles.
- Variations of the hydraulic gradient with beach gradient.
- The change in depth of flow (δ) with distance.
- Transportation of material after settlement or rolling sorting.
- Scour and re-deposition of previously deposited material.
- Deposition of fine material high up on the beach due to turbulent flow and hydraulic jumps.
- The flocculated nature of tailings.

Blight (1987 & 1994) suggests using an exponential ratio between the median particle at a distance down the beach compared with that of the whole tailings mix to predict particle size sorting:

$$D_{50(at\ x)} = D_{50(Total)} \cdot \exp\left(-B \frac{x}{L}\right) \quad \text{Eq. 2-16}$$

where: D_{50} = median particle diameter

B = constant as a function of tailings type and the rate of deposition

L = total length of the beach.

Morris (1993) derives a similar sediment sorting equation based on the assumption of constant specific gravity, G_s , as:

$$D = D_{50(Total)} \exp(-\alpha \cdot x) \quad \text{Eq. 2-17}$$

where D = expected particle diameter at a distance of x down the beach

α = constant, so that $\alpha = \varepsilon/(1 + b)$

ε = constant defining the local slope of the beach

b = transport constant relating the bed shear stress to particle properties

For a sediment with variable G_s , particle sorting by specific gravity results in:

$$G_s - 1 = (G_s - 1)_{Total} \exp(-\beta \cdot x) \quad \text{Eq. 2-18}$$

where β = constant and is related to α and ε by,

$$\varepsilon = \alpha(1+b) + \beta \left(1 + \frac{b}{3}\right)$$

Eq. 2-19

Morris is of the opinion that the specific gravity, G_s , of gold tailings is not constant and is actually related to the particle sizes. Abadjiev (1997) stresses that not only D_{50} , but the whole grading curve should be investigated for its influence on the particle sorting processes.

Beach Profiles

A review of tailings literature soon reveals the significance of predicting beach profiles on impoundment structures. Being able to predict the beach slope or profile allows not only better management of the pond, but also improved estimates of storage volumes and freeboard. As a result of difficulties with sampling and testing, a number of researchers have also sought indirect measures of estimating the settled physical properties of tailings based on the beach profile.

It has been shown that particle-size sorting processes, the strength of the settling slurry and requirements of continuity and conservation of energy are responsible for the shape of a hydraulic fill beach. As the slurry flows from the discharge point down the beach coarser particles will settle out, increasing the moisture content of the remaining slurry and lowering the pulp density, which results in a general flattening of the beach slope. Beach slopes on tailings impoundments range typically from up to 2% at the discharge point down to possibly 0.1% at the pond (Vick, 1983). Gold and uranium tailings, however, possess relatively uniform gradings and a narrow range in particle sizes, which virtually eliminate horizontal segregation. Changes in the grading of these materials, therefore, have little effect on the beach slope. Moisture content or solids concentration dominates the beach profile on gold and uranium tailings impoundments.

As the solids concentration of the feed pulp, S_c , increases and gravitational sorting becomes less effective, the overall slope or steepness of the beach also increases (Blight, 1994). As long as gravitational sorting is effective the slurry will be in a particle sorting regime. At some point, $S_c > 55\%$ in gold tailings, a mudflow regime commences, where there is no horizontal sorting and the slurry moves as a homogeneous viscous fluid or mud to the pond. Following the onset of a mudflow regime the overall beach slope will start to decrease. The rate of deposition does not appear to have a large effect on the beach profile provided deposition is on a wet beach. When deposition takes place on a dry beach, the slurry loses water to the beach, which increases the pulp concentration and results in a more curved profile and steeper overall beach slope. Increasing the number of discharge points around the perimeter of a dam also results in a steeper beach slope.

The first empirical attempt at predicting hydraulic fill beach profiles was made by Melent'ev et al. (1973), on beached natural alluvial soils and mine tailings in Russia. Melent'ev observed that geometrically similar beaches are formed for a specific type of particulate material, solids content of the slurry at placing and rate of placement. This Master or Melent'ev Profile applies within limits regardless of the length of beach and the difference in elevation between the point of deposition and the pond.

For conditions of sub-aerial deposition the master profile, Figure 2-11a, is expressed as,

$$y = i_{av} \cdot L(1 - x_o)^n \quad \text{Eq. 2-20}$$

where y = vertical height of the beach measured upwards from the pond edge

i_{av} = overall slope of the beach or H/L

H = total height of the beach

L = total length of the beach

$x_o = x/L$

x = horizontal distance along the beach measured from the discharge point

n = beach parameter

The beach parameter, n , is a function of the particle size distribution and solids concentration of the slurry and is constant for a particular tailings type, grading and pulp density, (Blight, 1987). It controls the shape and curvature of the beach profile so that for a flat beach n is equal to one, for a concave-up beach n is greater than 1, and n is less than 1 for a convex-up beach. Values of n between 1.33 and 1.66 was found by Melent'ev for fine to coarse tailings. Blight (1994) quotes a value of $n = 4.0$ on a gold tailings beach.

In a non-dimensional or normalised form Eq. 2-20 becomes,

$$\frac{y}{H} = \left(1 - \frac{x}{L}\right)^n \quad \text{Eq. 2-21}$$

For conditions of sub-aqueous deposition the master profile, Figure 2-11b, is expressed as,

$$y = H \left(1 - \exp^{-a \cdot x/L}\right) \quad \text{Eq. 2-22}$$

where y = vertical height of the beach measured down from the pond water level

H = total depth of the pond

L = beach length

x = horizontal distance measured negative from the edge of the pond

a = dimensionless coefficient

Bentel (1981) in studying the beach profiles of sub-aqueously deposited gold tailings, found the beach parameter, $n = 2$, in other words concave-up.

Melent'ev and his co-workers also derived an expression for predicting the overall slope of the beach, H/L ,

$$\frac{H}{L} = a \cdot S_c^{1/3} \left(\frac{D_{50}}{h^*} \right)^{1/6} \quad \text{Eq. 2-23}$$

where a = constant

S_c = solids concentration of the slurry

D_{50} = median particle size

h^* = stream depth associated with the scour velocity of the water

Since the groundbreaking work of Melent'ev, many researchers have investigated, or tried to improve on the simple master profile concept:

- (a) Smith et al. (1986) experimented with an exponential function, which best fit the geometries of the beaches on gold tailings impoundments, but found the Melent'ev power function to be more generally applicable.
- (b) Using scaled laboratory tests Blight et al. (1985), Wates et al. (1987) and Fan and Masliyah (1990) found the simple Melent'ev master profile to predict successfully flume profiles, which were representative of field beach profiles. In addition they also showed that:
 - Increased solids content of the slurry resulted in steeper beach slopes close to the discharge point and flatter slopes at the pond.
 - Increasing the coarseness of the pulp had the same effect.
 - Increased solids content resulted in an increase in the rate-of-rise of the beach.
 - Total slurry discharge rate at constant solids content had no significant effect on the slope of the beach, but higher flow rates lead to an increased rate-of-rise.
 - Shear strength decreased towards the pond corresponding to the decreasing slope angle of the beach profile.
- (c) Morris (1993), used river-transport dynamics with engineering hydraulics as a theoretical basis to solve the three fundamental differential equations for two dimensional flow over a mobile bed. The resulting exponential relationship seemed promising for modelling the profiles on coal and platinum beaches, but lacks the simplicity of the master profile.

- (d) More recently McPhail (1995) and McPhail and Blight (1997) used the stream power function and entropy maximisation for predicting the large scale beaching characteristics of a tailings slurry. The resulting exponential integral is dependent on the hydraulics of the plunge pool at the discharge end, as well as a variable, μ , as a function of the initial slope of the stream power curve.

Publications on the deposition and sedimentation behaviour of other tailings materials include: Coal, Iron and Bauxite: Fourie (1988); Uranium Chen et al. (1988), Emerson et al. (1994); Consoli (1997); Consoli & Sills (2000).

2.5.3 Sampling and Testing

The sampling and testing of tailings present a number of challenges. As a geo-material, tailings exhibit behaviour somewhere between that of a sand and a clay, depending on the state and composition of the material. This causes difficulties in the interpretation of in-situ tests, where assumptions of fully drained or fully undrained shear cannot be relied upon. The low density and low effective stress levels, which result from hydraulic deposition processes, make sampling and specimen preparation for laboratory testing extremely difficult. If undisturbed samples cannot be recovered, representative specimens have to be prepared artificially using techniques that simulate the soil formation processes on the actual impoundment. Once a test specimen is prepared, the actual testing is often assisted by the quick draining characteristics of the material. However, quite large strains may be required to develop the full stress-strain response. Another difficult aspect to address is the behaviour of the partially saturated material on the embankment wall and beach areas. Although not critical to the safety and serviceability of the dam, these zones play a defining role in determining the state and composition of material which may subsequently become critical. This section highlights some of the aspects concerned with the sampling and testing of tailings material.

Sampling

The properties of tailings in an impoundment are highly dependent on location, ranging from semi-dry coarse tailings near the discharge point to normally consolidated soft liquefiable fines in the penstock pond. This may call for different and specialised sampling techniques depending on location and whether bulk or undisturbed samples are required.

Collecting bulk samples for basic tests can be as simple as shovelling tailings into a bag from a dry beach, or virtually impossible in trying to separate and collect a specific layer, which might be only 10 mm thick, from the pond area below the water surface. In collecting

bulk samples, the highly layered nature of the tailings profile has to be taken into account. In some cases a mixed sample of material spreading over several layers of alternating fine and coarse tailings may be collected. In others, a specific layer has to be isolated and carefully sampled.

With undisturbed sampling it is important to retain the soil structure and prevent liquefaction disturbance during handling and transportation. The most difficult part of undisturbed sampling in tailings is recovery from below the water table. Liquefaction and withdrawal suctions make recovery of samples extremely difficult, if not impossible. Sampling is usually done by simple piston sampler (Mittal & Morgenstern, 1976; Carrier et al., 1983), thin-walled tube sampler (East et al., 1988) or coring techniques (Plewes et al., 1988). Donaldson (1965), using the shoestring method after Collins (1954), was able to extract 76 mm undisturbed samples with a thin walled tube sampler. Kealy and Busch (1979), working from a floating barge in the pond, used a plastic air line attached to a Shelby tube sampler to break suction when withdrawing the tube from the hole. Follin et al. (1984) used the Delft stocking net sampler to recover successfully 18 m continuous samples with virtually no sampling disturbance evident. They concluded that the Delft sampler is an excellent tool for sampling tailings, but that it is not effective in dry sands of medium to high relative density due to excessive forces on the nylon stocking. Controlled ground freezing techniques have been employed by Yoshimi and Goto (1996), but disturbance caused by the formation of ice-lenses must be of concern. Roe and Zahl (1986) recovered undisturbed samples from the beach surface by impregnation with a low viscosity resin. The method is only viable above the water table, to allow impregnation by the resin. After the resin has set the sample is extracted and prepared for microscopic analysis of the structure.

Testing in the Laboratory

Tests in the laboratory are performed on either undisturbed or reconstituted samples. As a result of difficulties in obtaining quality undisturbed samples, most laboratory tests for determining compressibility, consolidation and strength of tailings have been performed on reconstituted specimens.

Provided that a high quality undisturbed sample can be delivered to the laboratory, due care has to be exercised in cutting and trimming the test specimen to prevent physical disturbance and moisture loss. Alternatively, representative reconstituted specimens have to be artificially prepared. There are numerous methods for preparing reconstituted tailings samples including compaction (Hamel & Gunderson, 1973), wet-tamping (Highter & Tobin, 1980), pluviation (Kuerbis et al., 1988), and slurry sedimentation (Donaldson, 1965), but these should be carefully tailored and calibrated with field measurements to ensure that it is

representative. Donaldson (1965) has described a method of preparing triaxial specimens by pouring the tailings slurry directly into a triaxial split mould and applying a small initial consolidation pressure with the split mould still attached. After 2 hours of consolidation the mould could be successfully removed and the test continued.

The effect of pre-treatment procedures should be determined when performing all laboratory tests; particularly the effect of changing the pore water chemistry and the make-up of the dissolved constituents when using tap or distilled water for specimen preparation and saturation (Emerson & Self, 1994). On the other hand, drying out of samples may cause precipitation of the dissolved constituents as hydrate solids, which will influence many test results. The flocculated nature of gold tailings slurries has already been mentioned and should be taken into account when specifying and performing grading analyses and indicator tests (Hamel & Gunderson, 1973). Duplicate tests should be specified at all times to give an indication of the effect of pre-treatment procedures on the test results and material behaviour.

The use of laboratory flumes in studying the depositional processes and profiles on tailings beaches has already been mentioned in Section 2.5.2. Flume tests have also been used to examine the post failure flow behaviour of breached tailings by simulating overtopping and pond encroachment failures by Papageorgiou et al. (1997).

Knowledge of the location of the phreatic surface in a tailings impoundment is arguably the most valuable piece of information for design and auditing purposes. Unfortunately, permeability, as the defining parameter controlling the seepage regime, is also one of the most difficult to determine in the laboratory, either directly using permeameters or indirectly based on consolidation data. Barrett (1987) summarises some of the shortcomings of permeability testing in the laboratory:

- (a) **Sampling related problems:** Difficulties with access and sampling conditions can lead to sampling bias, and physical disturbance, pore pressure changes and stress relief all have an influence on the measured permeability. Generally, permeability samples must be big enough to allow for the influence of macrostructure (cracks, layering etc.). On a gold tailings impoundment desiccation cracks typically form in grid pattern at roughly one meter intervals. These cracks and the properties of their infill material greatly influence the anisotropy of permeability. The seepage regime in a tailings dam is further complicated by the highly layered nature of the deposit, which makes the choice of preferred drainage direction very difficult during testing. However, it might be worthwhile determining the permeabilities of individual representative layers as well as the macro-permeability.

- (b) **Equipment related problems:** Seepage against the sides of permeameters, stress levels during testing and seepage force induced consolidation (Suthaker & Scott, 1996) are some of the problems that have to be addressed during testing.

Suthaker and Scott (1996) reviewed a number of methods for determining hydraulic conductivity in the laboratory including constant and falling head permeameters, flow pump tests (Olsen, 1966), restricted flow tests (Sills et al., 1986) and seepage tests, but found conventional measurement techniques not suited to measure permeability, especially in the fine tailings slimes. The authors subsequently designed a slurry consolidometer to prevent seepage induced consolidation by locking the top cap of the consolidometer during permeability measurements.

Mittal and Morgenstern (1976) and Lappin (1997) argue that compressibility and consolidation characteristics should preferably be tested in Rowe cell type oedometers. The Rowe cell allows variations of the boundary conditions together with pore pressure control and measurement, which allows constant head permeability tests between loading stages. However, it is doubtful whether the drainage facilities of the Rowe cell are permeable enough to allow free drainage of tailings. The Rowe cell also allows only limited settlement of the loading diaphragm and excessive frictional effects and low sensitivity restrict the use of the apparatus at very low stress levels. Compressibility and permeability, for the early stages of self weight consolidation, can better be determined from density and pore pressure measurements during sedimentation column tests (Lappin, 1997). These tests can also be used to determine the rate of sedimentation.

Sully (1985), suggests that permeabilities should rather be determined from a large number of in-situ infiltration tests, to account for macrostructure.

Stress-strain and strength measurements are usually performed in the direct shear box or triaxial apparatus, Vick (1983). The shear box test suffers from many limitations (Atkinson & Bransby, 1978):

- Stresses and strains are non-uniform
- States of stress and strain cannot be determined completely
- The specimen is forced to shear more or less along a horizontal plane

The apparatus is better suited for the determination of stresses that cause failure on a particular plane, and is particularly suited for finding the strength of pre-existing failure surfaces in a soil specimen. Even with the more sophisticated simple shear device (Bjerrum & Landva, 1966; Roscoe, 1970), either the stress distribution or boundary deformations are not sufficiently controlled (Airey & Wood, 1987).

The shear device that is most commonly used both for design and research is the triaxial apparatus, as described originally by Bishop and Henkel (1962), with all its modern adaptations. However, the isotropic consolidation conditions in the triaxial may not conform to the field loading situation, and drainage conditions are limited to being either fully drained or fully undrained. Typical results of triaxial tests on tailings indicate, (Vick, 1983; East et al., 1988; Van der Berg et al., 1998):

- Relatively high strains to failure.
- Zero effective cohesion with undrained effective friction angles ranging between 32 and 42°. The high angles of friction are likely the result of the angularity of the tailings particles. Both coarse and fine tailings appear to give similar results in terms of friction angle.
- At densities and stress levels relevant to in-situ saturated conditions, dilation seldom occurs prior to failure with no strain softening or strength reduction following peak shear strength.
- At low stress and high density states, typical of samples recovered from desiccated beaches and subsequently saturated in the laboratory, dilation does occur and strength reduction may be a possibility as a result of the overconsolidated states of these specimens.
- In undrained shear, the coarser material is particularly prone to phase transfer dilation as described by Ishihara et al. (1975), followed by strain hardening at the critical state.

Feasibility studies of using physical simulation techniques, especially centrifuge modelling, to simulate failure mechanisms and phreatic surface development on tailings structures have been published for coal tailings by Al-Hussaini et al. (1981), Sutherland and Rechar (1984) and Stone et al. (1994). These studies indicated good performance in modelling staged filling, Lappin (1997).

In-situ Testing

In-situ testing offers an attractive alternative to sampling and laboratory testing, but difficulties in defining boundary and drainage conditions hamper the interpretation of these tests.

The in-situ vane shear test (VST) has been used extensively in tailings investigations, but there is no assurance regarding drainage conditions during shear. Usually attempts are made to ensure either fully undrained (rapid rate of shear) or fully drained (very slow rate of shear) conditions. However, these attempts are not necessarily successful (Blight, 1970; Blight & Steffen, 1979; Vick, 1983).

The cone penetration test and piezocone (CPT & CPTU) suffer from the same limitations concerning drainage conditions during shear as the vane shear test, but have the advantage of providing excellent information on sub-surface layering and, in the case of the piezocone, the seepage regime within a tailings impoundment (East et al., 1988a). Stratigraphy from the piezocone provides almost an historic record of mine operations over the life of the tailings dam. The material type changes, the rate of deposition and any major stoppages are reflected in the penetration record. The use of friction ratios is limited in tailings due to the highly layered nature of these deposits, where the friction sleeve can be located across a number of different layers simultaneously.

Jones et al. (1981) used the piezocone extensively on platinum and gold tailings dams and developed the first version of the Jones and Rust soil classification chart for the piezocone using normalised cone resistance and pore pressure parameters, Figure 2-12. Subsequent development of this chart is discussed in Jones and Rust (1982, 1983). Classification is based on the principle that penetration through coarse material results in high penetration resistance together with low or negative generated excess pore pressure. In fine material, penetration resistance is low with large positive excess pore pressure. The identification chart proposed by Robertson and Campanella, (Robertson, 1990), is also popular for classification purposes in tailings. For more information and other identification systems, reference can be made to Campanella et al. (1983); Larson and Mitchell (1986), Mlynarek et al. (1991) and Tschuschke et al. (1995).

Standpipe piezometers, preferably with water sampling capabilities, are widely used on tailings dams to monitor the pore water regime, phreatic surface and in-situ permeabilities (Kealy & Busch, 1979; Follin et al., 1984; Wagener et al., 1997). Piezometers are either installed in boreholes or are of the push-in type, but care must be taken to ensure that the measuring tip is sealed off properly after installation. The accuracy of a standpipe piezometer is of the order of ± 150 mm provided it is properly installed, however, due to the time lag of response of the water level inside the piezometer, accuracies of better than ± 300 mm cannot be expected (Wagener et al., 1997). In order to determine the level of the phreatic surface accurately a number of piezometers have to be installed at a specific location to account for the effect of vertical flow. The error resulting from assuming the phreatic surface to be located at the level of the water inside a single standpipe is not always conservative in a tailings impoundment. When seepage flow has a vertical downwards component the water level in a piezometer will be below the actual phreatic surface and vice versa.

The piezocone, in turn, gives reliable estimates of the equilibrium or ambient pore pressure distribution as well as the location of the phreatic surface, from dissipation tests (Rust et al.,

1984). By comparing the ambient pore pressure increase with depth to that resulting from hydrostatic water pressures the following conclusions may be drawn:

- If equilibrium pore pressures are hydrostatic, then there is no component of vertical flow at the location of the sounding.
- If equilibrium pore pressures are below hydrostatic, then a vertical downwards flow component exists.
- If equilibrium pore pressures are above hydrostatic, then a vertical upwards flow component exists.

A non-hydrostatic profile of equilibrium pore pressures in the pond area is often the result of ongoing consolidation resulting in vertical movement of the dissipation water.

Coefficients of consolidation and a good indication of permeability can also be derived from piezocone dissipation data based on theories of consolidation, plasticity and cavity expansion (Rust et al., 1995; Rust, 1996). Van der Berg (1995) noted the unique relationship between the slope of the phreatic surface, the rate of increase in ambient pore pressure with depth and anisotropy in permeability. Piezocone dissipation data enables the determination of the slope of the phreatic surface, α , in the horizontal direction, and the slope of the equilibrium pore pressure increase with depth, β , in the vertical direction. These parameters can then be used to calculate the anisotropy in permeability at the phreatic surface as,

$$\frac{k_x}{k_z} = \frac{\tan \beta}{\tan^2 \alpha} \quad \text{Eq. 2-24}$$

Permeability ratios of 7 to 22 and even as high as 25 (Wagener et al., 1998), have been found in this way, and take into account in-situ structure and macro permeability.

The density state or relative density is a critical parameter in determining the stability and liquefaction potential of a tailings impoundment. Relative density, D_r , can be estimated from cone penetration results with reference to Schmertmann (1978) using,

$$D_r (\%) = \frac{100}{2.91} \ln \left(\frac{q_c}{12.31 \sigma'_{vo}{}^{0.71}} \right) \quad \text{Eq. 2-25}$$

where q_c = measured cone resistance

σ'_{vo} = effective vertical overburden pressure

Mlynarek et al. (1995) suggest that the density state in a tailings dam can only be determined from cone results if site specific correlations are first established using in-situ void ratio's from undisturbed sampling and estimates of the maximum and minimum void ratio's from laboratory tests; see also Jones et al. (1981), Baldi et al. (1982), Klohn (1984) and Matyas et al. (1984). Alternatively interpretation of relative densities from cone results

should be accompanied by calibration chamber tests (East et al., 1988a). However, tailings data lie in areas with high uncertainty compared with typical chamber sands.

According to Mittal (1974), penetration tests are unreliable tools for in-situ density measurements. As an alternative down-hole nuclear density logging may be used, which is reported to give reliable results (Mittal & Morgenstern, 1975 & 1976; Carrier et al., 1983; Klohn, 1984; Tjelta et al., 1987; Plewes et al., 1988).

Jones et al. (1981) derived an equation for profiling the in-situ variations in the angle of internal friction, ϕ' , in tailings by assuming zero cohesion and making use of bearing capacity relations after Harr (1977) and Parez et al. (1976),

$$\frac{q_c + \Delta u((q_c/\sigma'_{vo}) - 1)}{\sigma'_{vo}} = (1 + \tan \phi') \tan^2 \left(45 + \frac{\phi'}{2} \right) \exp(\pi \tan \phi') \quad \text{Eq. 2-26}$$

where q_c = measured cone resistance, adjusted according to Parez et al. (1976)

Δu = excess pore pressure

σ'_{vo} = effective vertical overburden pressure

Other empirical attempts at relating shear strength parameters in cohesionless tailings to cone penetration results include:

(a) Sugawara and Chikaraishi (1982) - gold tailings:

$$\frac{q_c}{\sigma'_{vo}} = 1.5 \exp \left[2\pi \tan \left(\frac{\sin \phi'}{(1 - 2A_p \sin \phi')} \right) - 1 \right] \cos \left(\frac{\sin \phi'}{(1 - 2A_p \sin \phi')} \right) \quad \text{Eq. 2-27}$$

where $c' = 0$, no cohesion is assumed

A_p = pore pressure coefficient given as a function of the pore pressure response

(b) Robertson and Campanella (1983) - natural soils:

$$\tan \phi' = 0.1 + 0.38 \log \left(\frac{q_c}{\sigma'_{vo}} \right) \quad \text{Eq. 2-28}$$

East et al. (1988) in using this bearing capacity type formulation on the Homestake gold tailings dam found friction angles ranging between 20 and 34° and as low as 10°. East et al. (1988a) conclude that interpretation of cone results in this way is not sufficient to describe the strength of the material and stress the importance of relative density in controlling the behaviour of tailings.

(c) East and Ulrich (1989) - tailings in general:

$$\tan \phi' = 0.105 + 0.161 \ln \left(\frac{q_c}{\sigma'_{vo}} \right) \quad \text{Eq. 2-29}$$

Eq. 2-29 is a modification of Eq. 2-28, specifically adapted for use in tailings.

(d) Mlynarek et al. (1991) - non-cohesive tailings:

$$\phi' = 10(2a_2 + b_2 \cdot D_r) \quad \text{Eq. 2-30}$$

where a_2 & b_2 = constants given as a function of friction ratio or tailings grade.

(e) Tschuschke et al. (1992) - copper tailings:

$$\tan \phi' = -1.547 + 1.574\gamma_d + 0.012w(1 - 0.083w) \quad \text{Eq. 2-31}$$

$$\text{where } \frac{\gamma_w}{\gamma_d} = 1.04 - 0.113 \frac{\log q_c}{Pa} + 0.025 \frac{\log \sqrt{\sigma_v}}{Pa} \quad \text{Eq. 2-32}$$

γ_d = effective unit weight of the tailings

w = moisture content

γ_w = unit weight of water

Pa = reference pressure (atmospheric)

σ_v = vertical overburden pressure

These equations were used by the authors to calculate the friction angle of tailings on partially saturated beach areas.

East et al. (1988) prefer the self-boring pressuremeter for determining effective friction angles at selected depths in tailings impoundments, and in applying the interpretation proposed by Robertson and Hughes (1986) with correction for looseness, found the friction angles in the Homestake gold tailings dam to vary between 30 and 40°.

The undrained shear strength parameter, c_u , is often used in the design of tailings dams for calculating the factor of safety against undrained slope instability. Undrained shear strength is usually derived from cone penetration data using a cone factor, N_{kt} , in the following equation,

$$c_u = \frac{q_e}{N_{kt}} \quad \text{Eq. 2-33}$$

Values for the cone factor quoted in the literature for tailings are:

- Larson and Mitchell (1986) - uranium tailings, $N_{kt} = 15$ to 19
- East and Ransone (1988) – gold tailings, $N_{kt} = 9$ to 12
- East and Ulrich (1989) - tailings in general, $N_{kt} = 10.4$

- Mlynarek et al. (1994) – copper tailings, $N_{kt} = 15.5 \pm 4.5$

Ishihara et al. (1990) derived correlations for empirically estimating the residual strength of tailings dams and other slopes from cone data using a cone factor of 15 for silty sands. However, the scatter is great.

Deformation and stiffness parameters are usually related to cone penetration data in the form of Eq. 2-34 (Senneset et al., 1989).

$$M = \alpha \cdot q_c \quad \text{Eq. 2-34}$$

where M = drained constrained secant modulus

α = constant

Mlynarek et al., (1995) found α to range between 1.9 and 4.6 as a function of grain size and in-situ stress level for a variety of tailings types, whereas Tschuschke et al. (1994) quote the value of $\alpha = 18.4$ on average in copper pond tailings.

Assessment of liquefaction potential with the CPT is usually done by correlations with Standard Penetration Test (SPT) data. In assessing the liquefaction potential of a tailings dam, Ulrich and Valera (1995) found very low SPT blow counts typically ranging between 0 to 5 up to a depth of 10 m. Due to the insensitivity of the SPT in this range it was decided to use the cone penetration test, which is not as operator dependent as the SPT, does not require a borehole and gives a lot more information, much faster, even at great depth. The cone penetration data can be transformed into equivalent SPT numbers, which are then used to determine the liquefaction potential after Seed and De Alba (1986). Unfortunately the guidelines do not extend to low enough q_c values, and extrapolation is required when considering the data in tailings. However, the CPT gives additional information on the equilibrium pore pressure distribution and location of the water table, which can be used to calculate vertical effective stresses from overburden pressures and serve as guidelines to the possibility of pore pressure generation during a seismic event.

A number of papers have been published on the use of in-situ tests in tailings other than gold. Coal: Wahler and Associates (1973); Copper: Campanella et al. (1983), Campanella et al. (1984), Tschuschke et al. (1992), Tschuschke et al. (1993), Vidic et al. (1995), Tschuschke et al. (1994), Tschuschke et al. (1995); Uranium: Larson and Mitchell (1986).

2.5.4 Basic Engineering Properties of Tailings Deposits

A tailings impoundment consists of multiple layers of sediment formed by the processes of deposition, sedimentation and consolidation. The engineering characteristics of these layers

ultimately control the safety and serviceability of the structure as a function of the in-situ composition (particle properties) and state (density, stress level, stress history, etc.). The remainder of this chapter is dedicated to a review of the engineering properties of gold tailings deposits with the aim of defining composition and state .

For publications on the properties of tailings other than gold, reference can be made to: Uranium: Keshian and Rager (1988); Hard rock tailings: Aubertin et al. (1996), Aubertin et al. (1998).

Fundamental Particle Properties

The mineral constituents of tailings particles include quartz, chlorite, biotite/mica, talk, pyrite and arsenopyrite (Donaldson, 1965; Hamel & Gunderson, 1973), quartz being by far the most abundant mineral in the coarser grains (Mlynarek et al., 1995). Considering the mineral shapes of these constituent minerals it becomes clear why tailings particles are mainly angular in shape (Pettibone & Kealy, 1971):

- Quartz : Angular grains.
- Pyrite : Cubic grains.
- Mica : Platy shaped grains.
- Talc : Platy shaped grains.

A recurring theme in the published literature on tailings is the similarity in engineering properties between coarse and fine tailings grades. Ulrich and Valera (1995) ascribe this phenomenon to the absence of clay minerals in gold tailings, although it will be shown in this thesis that gold tailings can have significant amounts of clay minerals especially in the slimes.

Quoted values of the specific gravity of gold tailings particles are listed in Table 2-5

Table 2-5: Specific gravity of gold tailings.

Reference	G_s
Pettibone and Kealy (1971)	2.5 to 3.5
Hamel and Gunderson (1973)	3.1
East et al. (1988a)	3.02

The sands or coarser fraction of gold tailings range in shape from very angular to sub-angular with sharp edges (Mittal & Morgenstern, 1975; Lucia et al., 1981; Garga & McKay, 1984; Mlynarek et al., 1995). The fines are invariably angular, sometimes needle shaped, with very sharp edges and resemble shards of broken glass under the microscope (Hamel

& Gunderson, 1973). Papageorgiou et al. (1999) mention, in addition to the irregular shapes of the particles, also harsh surface textures.

Gradings and Atterberg Limits

The particle size gradings of gold tailings are generally limited to and uniformly distributed in the silt size range with small percentages, of the order of 10%, in the sand and clay sized ranges (Pettibone & Kealy, 1971; Van Zyl, 1993).

Figure 2-13 represents a summary of grading curves for gold tailings as found in the literature.

Gold tailings exhibit very little plasticity and no cohesiveness (McPhail & Wagner, 1989), and classify, based on their Atterberg limits, as low to high plasticity silts on the Casagrande chart, see Figure 2-14 (Carrier et al., 1983). Mlynarek et al. (1995) advises the use of the fall cone test to determine the liquid limit and plasticity properties of gold tailings. The Atterberg limits for gold tailings generally fall within the following ranges (Wagener et al., 1998):

- Liquid Limit : 23 - 43
- Plastic Limit : 22 - 35
- Plasticity Index : 1 - 8
- Linear Shrinkage : 2.7 - 4.7

Despite this Wates et al. (1999) quote a linear shrinkage of gold tailings of up to 22%.

Density

The density of the material on a tailings impoundment is controlled by both the properties of the slurry and the depositional conditions. Depending on the specific gravity of the solids the sediment in the pond of a tailings impoundment settles to a dry density of approximately 1000 kg/m^3 at a moisture content of 60%. On the beach above the water table the solids settle to a density of approximately 1450 kg/m^3 at a moisture content of 20 - 50%. There is usually an increase in density with depth in an impoundment, as consolidation takes effect, but with considerable scatter. Vick (1983) argues that the density state in a tailings impoundment is better described by in-situ void ratio than dry density; void ratios do not include the effect of specific gravity and are only a function of the tailings particle properties and stress level.

Quoted values of the densities of gold tailings deposits are listed in Table 2-6.

Table 2-6: In-situ densities and void ratios for gold tailings.

Reference	Description	Density (kg/m ³)	Void Ratio
Donaldson (1965)	Dry density sub-aerial	1750	
	Dry density sub-aqueous	900	
Blight (1969)	Void ratio after deposition		1.7
	After evaporation		1.25
	After sun drying		0.5
Hamel & Gunderson (1973)	Standard Proctor	1700	
	Modified Proctor	1860	
	Minimum dry density	1000	
Blight & Steffen (1979)	Void ratio		1.1 - 1.2
Blight (1981)	In-situ dry density	1835	
Vick (1983)	Tailings sands		0.6 - 0.9
	Low plasticity slimes		0.7 - 1.3
	High plasticity slimes		5 - 10
East et al. (1988a)	In-situ dry density	1340 - 1740	
	Average dry density	1650	
Van Zyl (1993)	In-situ dry density	1000 - 1450	

Permeability

Permeability, in general, is a function of the particle properties, density state and structure of a soil (Pettibone & Kealy, 1971). According to Kealy and Busch (1979) the spatial variation of permeabilities through a dam cross section, from embankment wall to decant pond, is by far the most significant factor in determining the location of the phreatic surface. Anisotropy in permeability and foundation permeability only have minor effects in controlling the location of the phreatic surface. Material is generally coarser towards the embankment wall as a function of hydraulic sorting processes, Section 2.5.2. The resulting increase in permeability towards the embankment helps lowering the phreatic surface. However, this may be counteracted by severe anisotropy in permeability (Blight et al., 1985). Permeability is dependent on the coarseness of the tailings, but McPhail and Wagner (1989) report a maximum permeability difference of only one order of magnitude or less between gold tailings sands and slimes.

The permeability of tailings, excluding the effects of structure, can be calculated by the well known empirical relationship proposed by Hazen in 1892,

$$k = c \cdot D_{10}^2 \quad \text{Eq. 2-35}$$

where k = permeability in cm/s

D_{10} = particle diameter for 10% passing

c = constant varying between 1.0 and 1.5

Mittal and Morgenstern (1975), from laboratory studies, confirm the use of Eq. 2-35 with $c = 1$ for tailings sands at a relative density of about 40 to 50%. Blight et al. (1985) also recommends the use of this equation in tailings.

Other attempts at defining the mass permeability of tailings have been made by:

(a) Bates and Wayment (1967):

$$\ln(K_{20}) = 11.02 + 2.912 \ln(e \cdot D_{10}) - 0.085 \ln(e) \ln(CU) + 0.194e \cdot CU - 56.5D_{10}D_{50} \quad \text{Eq. 2-36}$$

where K_{20} = permeability at 20°C in in/hr

e = void ratio

D_{10} = particle diameter for 10% passing

D_{50} = particle diameter for 50% passing

CU = coefficient of uniformity

This equation is reported to be conservative by Mittal and Morgenstern (1975), who recommend the use of Hazen's formula.

(b) Carrier et al. (1983):

$$k = \left(95.2G_s \frac{PI}{100} \right)^{-4.29} \frac{e^{4.29}}{1+e} \quad \text{Eq. 2-37}$$

where k = permeability in m/s

G_s = specific gravity

PI = plasticity index as a percentage

e = void ratio

(c) Sherard et al. (1984; 1984a):

These authors propose an expression similar to Hazen with,

$$k = 0.35D_{15}^2$$

where k = permeability in cm/s

D_{15} = particle size for 15% passing

or using the characteristics of the beach profile,

$$k = a \cdot \exp(-bx)$$

where a & b = beach constants

x = horizontal distance from the discharge point.

Quoted values for the permeability of tailings in the literature are summarised in Table 2-7.

Table 2-7: Permeability values for gold tailings.

Reference	Description	Permeability (m/yr)
Blight (1980)	Range	1 - 50
Blight (1981)	General	3
Sully (1985)	For tailings sands	
	50 kPa effective stress	0.31
	100 kPa effective stress	0.23
	150 kPa effective stress	0.22
	200 kPa effective stress	0.21
Aubertin et al. (1998)	Hard rock tailings	1.5 - 60

The depositional processes on a tailings impoundment result in a highly layered, and anisotropic profile, as mentioned previously. Measured anisotropies, k_h/k_v , in gold tailings range from 5 - 10 (Pettibone & Kealy, 1971), 10 (Kealy & Busch, 1979), 2 - 10 (Vick, 1983), 7 - 22 (Van der Berg, 1995), 25 (Wagener et al., 1998) and even as high as 100 at the beach pond interface (McPhail & Wagner 1989). McPhail and Wagner (1989) point out that desiccation cracks and their preferential filling with coarse material reduce the effects of layering and consequently the macro anisotropy to only some 1.5 to 3.

The literature gives conflicting reports on the reduction of permeability with depth and increasing stress levels. Sully (1985) showed little reduction in the permeabilities of undisturbed block samples of coarse gold tailings for stress levels ranging between 50 and 400 kPa, whereas McPhail and Wagner (1989) mention reductions in permeability by a factor of 5 to 10 as a function of compressibility.

Engineers have become increasingly aware of the importance of unsaturated flow above the phreatic surface in modelling the seepage regime in a tailings impoundment. Aubertin et al. (1998) studied the water-retention properties of hard rock tailings to estimate the unsaturated hydraulic conductivity and capillary rise of these materials. Plate extractor tests (ASTM D3152) and Tempe cells (ASTM D2325 & D3152) were used to find the moisture retention curve (MRC) on remoulded tailings specimens, densified to void ratios between 0.5 - 0.9. The MRC, characteristic of the hydraulic properties of unsaturated porous media, provides a relationship between volumetric water content, θ , and matric suction, ψ . Popular models for the MRC include those proposed by Brooks and Corey (1966) and Van Genuchten (1980), where model parameters are empirically related to the grading

characteristics and porosity of the material. Another model proposed by Kovacs (1981) has physical significance that is lacking in the others. Aubertin et al. found saturated permeabilities well predicted by the Kozeny-Carman equation (Chapuis & Montour, 1992). Air entry values (*AEV*), or the matric suction at which the largest pore space de-saturate, are equally well estimated using Eq. 2-38.

$$AEV = \frac{b}{e \cdot D_{10}} \quad \text{Eq. 2-38}$$

where e = void ratio

D_{10} = particle size for 10% passing

b = constant ranging between 2.5 to 4 mm²

2.5.5 Compressibility and Strength of Tailings

The compressibility and strength of tailings as a function of their composition and in-situ state, together with a knowledge of the seepage regime, will determine the stability, storage capacity, allowable rate-of-rise and other controlling factors in the design and operation of an impoundment. Compressibility and the change in compressibility with time as well as static and dynamic strength of gold tailings will be the subject of this section.

Compressibility

Tailings appear to be more compressible than similar natural soils, partly due to their grading characteristics, high angularity and loose depositional state, Vick (1983). Considering the very recent stress history of tailings sediments, many authors assume a normally consolidated state. However, when deposition is sub-aerial, desiccation suctions and capillary effects can build in some overconsolidation. Donaldson (1965) measured pre-consolidation pressures of up to 500 kPa in gold tailings using the method of Casagrande in the oedometer, as well as tensiometers. Blight (1969) reports desiccation suctions of 1 to 10 MPa in gold tailings, although 10 MPa would require an unrealistically small D_{10} of the order of 0.15 micron³. Donaldson found almost all of the material above the water table to be overconsolidated. Nevertheless, the increase in shear strength due to pore suctions is unreliable as a result of frequent re-wetting and an almost instantaneous release in suction following precipitation and seepage from subsequent depositions. The principal advantage of desiccation is the resulting increase in density and stiffness. The overconsolidation effect may also be destroyed with depth and overburden pressure as the dam rises. Under sub-aqueous deposition, in the pond, the state of the tailings can be expected to be normally

³ Calculated assuming the effective pore diameter, d , is 20% of the effective grain size, D_{10} , which can support a capillary suction of $4T\cos(\alpha)/\rho_w d$, where, T , the surface tension is taken as 73 mN/m and, α , the contact angle as zero.

consolidated or even under-consolidated during the process of ongoing primary consolidation.

The coefficient of compressibility for gold tailings, as reported in the literature, is listed in Table 2-8. However, these values were derived from reconstituted normally consolidated laboratory samples and would not be applicable to material that has become overconsolidated due to desiccation suction.

Table 2-8: Coefficient of Compressibility, C_c , of gold tailings.

Reference	Description	C_c
Blight & Steffen (1979)	Tailings slimes	0.35
Vick (1983)	Tailings sands	0.05 - 0.1
	Tailings slimes	0.2 - 0.3

Carrier et al. (1983) approximates the compressibility of tailings for low stress levels as a function of the overburden pressure by,

$$e = 17.7G_s \frac{PI}{100} \cdot \sigma_v^{-0.29} \quad \text{Eq. 2-39}$$

where G_s = specific gravity of the particles

PI = plasticity index as a percentage

σ_v = vertical effective overburden pressure

Consolidation Characteristics

Depositional processes and sedimentation are followed by primary and secondary consolidation, where primary consolidation is associated with the dissipation of excess pore pressures, and secondary consolidation or creep with viscous effects in the particle skeleton. Primary and secondary consolidation occur simultaneously until primary consolidation ceases with full dissipation of the excess pore pressures. Secondary consolidation is reported to be small and relatively insignificant in tailings and is attributed to continuing particle rearrangement, grain to grain slippage under the influence of constant load and continuing contact fracture propagation promoted by water, Vick (1983).

Blight and Steffen (1979) show that the coefficient of consolidation decreases with increased effective stress level and slightly with decreasing void ratio. Vick (1983) maintains that the consolidation behaviour of tailings can sometimes be dominated by permeability and at other times by compressibility.

Typical values of the coefficient of consolidation for gold tailings are listed in Table 2-9.

Table 2-9: Coefficients of consolidation, c_v , for gold tailings.

Reference	Description	c_v (m^2/yr)
Blight & Steffen (1979)	Tailings slimes	198
Blight (1980)	General	10 - 50
Blight (1981)	General	300
Vick (1983)	Tailings sands	$1.6 \times 10^3 - 0.3 \times 10^6$
	Tailings slimes	0.3 - 30
Sully (1985)	For tailings sands as a function of effective stress	
	50 kPa	112
	100 kPa	67
	150 kPa	82
	200 kPa	68
	300 kPa	55
	400 kPa	81

Shear Strength - Static

Many factors influence the shear strength of a hydraulic fill including:

- Material properties, i.e. mineralogy, grading, particle shape and surface texture, etc.
- Processes of deposition, sedimentation and self-weight consolidation and its effect on stress level, density, fabric, etc.
- Construction control, i.e. rate-of-rise, desiccation and re-wetting cycles and its influence on stress level, pore pressure distribution, overconsolidation, etc.
- Other factors including flocculation vs. dispersion, oxidation leading to precipitation bonding, etc.

Natural remoulded sands, generally, exhibit no cohesion with angles of internal friction ranging between 30° and 40° as a function of relative density and stress level (Guerra, 1972). However, at low stress levels the friction angle can be as high as 48° as reported by Bica and Clayton (1998) for Leighton Buzzard sand. Mittal and Morgenstern (1975) believe that tailings should have slightly higher friction angles than natural sands due to the highly angular nature of tailings particles.

Vick (1983) describes the shear strength properties of tailings as cohesionless with rare exceptions, and an effective angle of internal friction, ϕ , ranging between 30° and 37° . The influence of various factors on ϕ is summarised by him as:

- Very little dependence of ϕ on grading, see also Van Zyl (1993) and McPhail and Wagner (1989).
- Density has a surprisingly small influence on ϕ , 3 - 5° for tailings sands.
- Overconsolidation also has a relatively small effect.
- Effective stress level is the most important parameter controlling ϕ , with the strength envelope curved at high stress levels as a result of particle crushing.

Kuerbis et al. (1988) studied the phenomenon of nearly constant friction angles between the tailings sands and slimes by adding natural silt fines to a tailings sand. Undrained triaxial compression and extension tests demonstrated that the angle of friction at phase transformation remained constant and independent of silt content or mode of deformation. The silt fines were simply occupying void spaces, whereas the sand skeleton was controlling the shear behaviour. The addition of silt fines, however, resulted in higher settled densities following specimen preparation, and also increased dilatancy under shear.

Hamel and Gunderson (1973) performed a number of direct shear tests to determine the effect of level of saturation and density on the shear strength of Homestake gold tailings. A range from completely dry to fully saturated specimens was prepared by compaction to varying densities and subjected to direct shear in the shear box. Shear strength parameters for these specimens are listed in Table 2-10. The dry specimens (moisture content of 0.2 %) were compacted to 1344 kg/m³, loose, and 1520 kg/m³, dense; the wet specimen to 1372 kg/m³, moisture content 17%; and the dense saturated specimen to 1512 kg/m³, moisture content 34%. All results indicated contraction with no dilation during shear. The reduction in ϕ with added moisture is explained by the lubricating effect of the water on the layer lattice (clay) minerals in the tailings mix. Hamel and Gunderson argue that the addition of water reduced the charge attraction between partially hydrated surface ions on the clay minerals thus reducing the angle of internal friction (see Lambe & Whitman, 1969). The high cohesion values are ascribed to aggregate interlock resulting from compaction, increased electrical attraction which varies with the square of the distance between charges and pore water suction as a result of partial saturation. Nevertheless, the reliability of such high cohesion measurements in direct shear should be questioned.

Mittal and Morgenstern (1975) found ϕ , at peak shear strength in the shear box, to be a function of normal stress and density up to a normal stress of 400 kPa using compacted specimens. Below 400 kPa, ϕ reduces with increasing normal stress. Above 400 kPa, ϕ becomes independent of normal stress until particle crushing effects become significant. However, they also found ϕ for uncompacted loose specimens to be independent of both stress level and density. The dependency of ϕ on stress level and density for the

compacted specimens is principally the result of overconsolidation built in by the high compaction stresses. Similar results were found by Vesic and Clough (1968) for loose and dense samples of Chattahoochee River sand tested at different stress levels in the triaxial apparatus. The overconsolidated state and high density lead to peak strength behaviour and dilation during shear. Nevertheless, cohesionless soils in general behave as though heavily overconsolidated due to soil structure (Poulos, 1988). Poulos warns against the use of peak strengths, which can be up to ten times the steady state or ultimate strength, in designing for static and seismic stability. If the material has been strained beyond the strain level at peak strength, it exists in a meta-stable strain softening state which can easily lead to failure or liquefaction.

Static shear strength parameters for gold tailings are summarised in Table 2-10.

Table 2-10: Static shear strength parameters for gold tailings.

Reference	Description	Test	c' (kPa)	ϕ (°)
Donaldson (1965)	General	Triaxial	0	35
Hamel & Gunderson (1973)	Dense air-dry	Direct shear	79	38
	Loose air-dry		0	39
	Loose wet		100	28
	Dense saturated		11	24
Mittal & Morgenstern (1975)	Peak, loose	Direct shear	0	34
Blight & Steffen (1979)	Slimes		0	28 - 41
Blight (1981)	General		0	35
Vick (1983)	General			30 - 37
Sully (1985)	Average	Direct shear	5	33
Van Zyl (1993)	Sand & Slimes		0	35
Blight (1997)	Sand & Slimes	Both	0	29 - 35

Shear Strength - Dynamic or Seismic

Cyclic triaxial strength of soil is considered to be a function of relative density and stress level, grain characteristics, method of deposition, ageing effects and previous seismic history (Seed, 1976). Compared with natural soils, both the coarse sands and fine slimes of tailings are believed to have angular to sub-angular grain shapes and similar grain textures and hardness. In addition tailings gradings lie within a remarkably narrow range, mostly in the fine sand and silt size ranges. The stress history will depend on whether the material has been allowed to dry following deposition, in which case it may be heavily overconsolidated, and is not likely to include effects of any significance of seismic events. The most important

factor controlling cyclic strength in tailings, therefore, is the in-situ density (Garga & McKay, 1984). Density and seismic strength can be greatly improved by compaction during construction.

Garga and McKay (1984) performed a number of cyclic triaxial tests on undisturbed and reconstituted samples from 20 tailings and 13 non-tailings materials and concluded that:

- Reconstituted samples tend to have lower cyclic strengths than undisturbed samples (possibly the result of in-situ structure).
- Cyclic strength is sensitive to stress anisotropy.
- Cyclic strength decreases with increasing principal stress ratio.
- Materials in the fine sand range, such as tailings sands, exhibit the lowest cyclic strengths.

2.5.6 Liquefaction Potential of Tailings Deposits

Many incidents on tailings impoundments are claimed to be related to liquefaction. If failure is not caused by liquefaction, then liquefying of the mass mobilised by other types of failure can result in extensive damage as illustrated in Section 2.4.10.

Liquefaction is "the phenomenon wherein a saturated sand loses a large percentage of its shear resistance, due to monotonic or cyclic loading, and flows in a manner resembling a liquid until the shear stresses acting on the mass are as low as its residual shear resistance", Castro and Poulos (1977). Liquefaction in soils can result from either an unstable or contractile soil skeleton with collapse potential, or when effective stresses are annulled by positive pore pressure build-up, usually during cyclic loading. In the case of collapse potential (Sladen et al., 1985) the material state must lie in structurally permitted space, which will collapse to a more stable state on de-structuring. It is interesting to note that in studying the sedimentation-consolidation behaviour of natural sands, Schiffman et al. (1986) discovered that cohesionless sands and silts, deposited sub-aqueously at low to moderate relative densities (30 - 50%), may exhibit peak undrained shear strength behaviour. It must be assumed that this is a result of the particle arrangement or fabric under these conditions. Kramer and Seed (1988) found similar results on natural sand, where the peak behaviour and susceptibility to liquefaction was found to be a function of confining stress and the initial consolidation shear stress. Collapse is usually initiated by a trigger mechanism in the form of increased pore pressures due to a rise in the phreatic surface, increased shear stress due to a rise in the dam height, cyclic loading from seismic activity, loss of confining stress due to failure, etc. The work of Sasitharan et al. (1993) shows that liquefaction can follow either drained or undrained loading conditions, but that

the actual liquefaction event is undrained with a rapid loss of effective strength caused by positive pore pressure generation.

The potential for liquefaction in tailings is controlled by the level of saturation, density state, fines content and confinement stress levels (Troncoso, 1986). Liquefaction potential varies on a typical impoundment as a function of (Klohn et al., 1978):

- (a) **Permeability zoning:** The permeable embankment leads through intermediate beach zones to the relatively impermeable pond (Finn, 1982). The intermediate and pond areas are more prone to liquefaction due to more extensive saturation and higher fines content. Singh and Chew (1988) found tailings with less than 20% silt sized content to behave similar to clean sands. However, for mixes with more than 60% silt, typical of gold tailings, the behaviour was controlled by the silt fraction, which is more susceptible to liquefaction due to the reduced permeability.
- (b) **Saturation levels:** It is only under saturated conditions that liquefaction, as a result of pore pressure build-up, becomes a major concern (Klohn, 1980). However, Papageorgiou et al. (1999) warn that liquefaction as a result of structural collapse can be of importance in both partially and fully saturated soils. Use of internal drains, decant facilities and controlled rate-of-rise should prevent the phreatic surface from exiting on the downstream face of the embankment wall. In addition to improving static stability through the effective stresses, levels of saturation and hence liquefaction potential are also lowered if the phreatic surface is kept as low as possible in the embankment wall.
- (c) **Deposition densities:** Uncompacted sands are most susceptible to liquefaction, however, in low to moderate seismic areas, compaction of the embankment material may not be required.
- (d) **Fines content:** Lucia et al. (1981) allude to the fact that silt sized particles may form meta-stable honeycomb structures under sub-aqueous deposition. Similar observations have been made by Troncoso (1986), where the addition of fines to triaxial specimens led to collapse behaviour. In an extensive investigation of the effects of fines content on the liquefaction susceptibility of gold tailings, Papageorgiou et al. (1999) report the following:

- Increased fines content displaces the critical state line (CSL see Chapter 4) downwards in terms of void ratio at the same effective stress, thus increasing the density.⁴
- Addition of fines leads to a reduction in permeability, making undrained response more likely.
- More angular particles result in a steeper slope of the CSL, thus decreasing the stiffness.
- Increasing the fines content also results in a smaller range of densities attainable by a placement technique during specimen preparation.
- Higher densities are required to produce phase transfer dilation at failure, as the fines content increases.

The consequence of all these effects, on aggregate, is an increased potential for liquefaction with increasing fines content.

Seismic history and ageing are also important causes of evolution and changes in the static and dynamic stability of an impoundment structure (Troncoso, 1988).

Papageorgiou et al. (1997; 1999) investigated liquefaction of mine tailings in terms of steady state concepts. Loading conditions were identified as possible triggers of liquefaction ranging from dynamic events such as seismic shaking, piling vibrations and blasting shocks, to static loads from a sudden increase in surcharge, raise in the level of the phreatic surface or a sudden loss of confining stress on the embankment wall following liquefaction of the impounded slimes. In their 1997 paper Papageorgiou et al. report that cyclic triaxial tests on cyclone overflow (slimes), cyclone underflow (sands) and whole tailings specimens prepared using water pluviation techniques all resulted in stable dilatant behaviour. They attribute this behaviour to particle angularity, irregular particle shapes and harsh surface textures, and confirmed the observation with in-situ measurements indicating dilatant states not susceptible to collapse liquefaction. Subsequently, in 1999, using moist tamping techniques to reconstitute samples, the authors were able to induce contractile behaviour in the laboratory in fine and coarse grades, resulting in liquefaction under static load conditions. An instability trigger was identified on the stress path as a function of the effective stress state, which corresponds to the collapse potential of Sladen et al. (1985). In-situ void ratio's were measured and found to lie mostly above the steady state line developed from the moist-tamped fine specimens. The conclusion was made that the in-

⁴ This argument is only valid as long as the fines are filling void spaces within a greater sand skeleton. As soon as the sand skeleton is disrupted addition of more fines will reduce the density and lift the CSL upwards as coarser particles start to float within a sea of fines.

situ material must be susceptible to liquefaction. However, no shear tests on undisturbed samples were presented to confirm this susceptibility to liquefaction.

Hightler and Tobin (1980) use Bishop's Brittleness Index (Bishop 1967, 1973) to predict the liquefaction susceptibility and post rupture behaviour of garnet iron and zinc tailings. The brittleness index is calculated for drained loading as,

$$I_{B(Drained)} = \frac{\tau_f - \tau_r}{\tau_f} \cdot 100\% \quad \text{Eq. 2-40}$$

where I_B = Brittleness Index

τ_f = shear stress at peak strength

τ_r = shear stress at residual strength

and for undrained loading as,

$$I_{B(Undrained)} = \frac{c_{uf} - c_{ur}}{c_{uf}} \cdot 100\% \quad \text{Eq. 2-41}$$

where I_B = Brittleness Index

c_{uf} = peak undrained shear strength

c_{ur} = residual undrained shear strength.

Specimens were specially prepared with high brittleness values using wet tamping techniques, and showed pronounced peak and post peak reduction in shear stress. This behaviour is evident even at high confining pressures. Initial void ratio was found to be the most important parameter governing the undrained brittleness index with no brittleness if the void ratio is greater than 80% of the maximum void ratio, e_{max} .

There is an interesting dispute in the literature concerning the failures of the Nerlerk underwater sand berms in the Canadian Beaufort Sea up to 1985. These berms were constructed using sub-aqueous hydraulic deposition of clean sands. Some of the berms even failed under static monotonic load conditions. The failures are generally attributed to post peak contractile softening behaviour as a result of meta-stable collapsible structure following deposition (Sladen et al. 1985). Been et al. (1988) contest that the material will have a natural tendency towards a stable non-contractile state following deposition and if it proved to be contractile, then there must have been some agent interfering with sedimentation process such as,

- Partial saturation resulting in apparent cohesion.
- Deposition of sand with silt infill preventing a stable sand skeleton forming.
- High upward seepage gradients during deposition.

Been et al. (1987; 1988) maintain that none of the failures were caused by collapse of a contractile structure, but that they were due to strains in the foundation. They conclude that

flow slides may occur in dilatant sands initially below the critical state line, and that it has manifested during seismic failures and in the laboratory, where cyclic loading induces positive pore pressure build-up. Even with static loading dilatant sands usually contract first, before they dilate. This initial contraction or positive pore pressure response is more profound in mildly dilatant sands and a flow type failure may result providing a significant trigger mechanism. The trigger was provided, in the case of the Nerlerk berms, by the foundation settlements.

It therefore appears that the structure of hydraulically placed sands, and for that matter tailings, is not obvious and needs careful consideration of the in-situ composition and state. It has been mentioned that increased silt content may lead to a meta-stable structure and strain softening collapse in tailings. Vaid (1994) on the other hand was never able to induce liquefaction in a series of cyclic triaxial tests with various void ratios and silt contents. It seems that only when samples were prepared using wet tamping techniques that liquefaction in the laboratory was evident.

Publications on the liquefaction behaviour of tailings other than gold include: Copper: Scott et al. (1989); Bauxite (aluminium red muds): Poulos et al. (1985); Uranium: Dunbar et al. (1991); Zinc & Garnet: Hightner & Vallee (1980).

Special mention should also be made of the CANLEX experiment, in progress, in Canada (Phillips & Byrne, 1995; Konrad, 1997). CANLEX stands for Canadian Liquefaction Experiment and involves a large controlled field liquefaction event using oil sand tailings from Syncrude Canada and the Fraser River delta. The experiment is divided into three stages; stage one and two determine the in-situ state and stress-strain response of the material and stage three comprises a controlled liquefaction event. Both static and dynamic liquefaction will be studied and numerical predictions will be compared with field behaviour. The objectives of the CANLEX experiments are to:

- Obtain high quality undisturbed samples through freezing techniques.
- Calibrate and verify in-situ tests.
- Obtain a better understanding of liquefaction.
- Develop and evaluate liquefaction models.
- Investigate the effects of fines content, fabric, load direction, shape of the state boundary surface and triggering mechanisms on liquefaction behaviour.

2.6 CONCLUSIONS FROM THE LITERATURE

2.6.1 Summary of Conclusions from the Literature

The following list summarises the philosophies of design, construction and operation practised in the South African gold mining industry as well as relevant conclusions on the composition and state of gold tailings from a review of tailings literature.

(a) **Tailings production:**

- In the gold extraction process, mined ore is crushed and finely ground down by mechanical means, after which it is subjected to chemical treatments to dissolve, separate and precipitate the gold valuables. The by-product of this process is a fine rock flour slurred with process water, known as tailings.
- South African gold tailings slurries are generally flocculated and slightly alkaline when leaving the reduction plant.

(b) **Impoundment design, construction and operation:**

- For economic reasons, tailings dams are generally not designed with the same conservatism as conventional water-retention dams.
- South African gold tailings impoundments are almost exclusively designed and operated as perimeter ring dikes with the tailings embankment raised, up-stream, throughout the lifetime of the impoundment, using the daywall-nightpan paddock system. Deposition is sub-aerial by open ended pipe discharge or in some cases by spigotting or cycloning. The daywall is raised to provide adequate freeboard, but also to allow maximum time, three weeks on average, for evaporation and desiccation to improve the mechanical properties (especially the density) of the material. Rate-of-rise as a function of this commonly ranges between 1 and 3 m per year. The impoundment can be operated as one or more isolated compartments, each with its own delivery and decant facilities. Decant systems are almost always of the penstock type with internal drainage systems absent in all but the most recent developments. In addition to internal and surface drainage measures, ground level catchment paddocks are provided around the perimeter of the dam to intercept runoff from the side slopes of the impoundment.
- Construction of this kind relies heavily on densification through desiccation and to some extent on segregation to provide a competent embankment. Segregation is achieved either mechanically using hydro-cyclones or gravitationally along the flow path from the discharge point towards the pond area. Gold tailings do not segregate readily due to their uniform grading, and rely on the pulp density to assist segregation.

- Control of the phreatic surface from a design and operational point of view is the single most important factor controlling the stability and serviceability of a tailings impoundment. The controlled rate-of-rise and sequence of deposition should be tailored in relation to the post depositional properties of the tailings and any internal drainage facilities to ensure both adequate storage capacity and a safe structure.
- When selecting a failure criterion and design strength for static stability, careful consideration should be given to the drainage conditions prior to and during a potential failure, as well as to the strain history and stress-strain response of the material.
- Liquefaction of tailings results either from excessive pore pressure build-up during seismic shaking or from softening of a collapsible soil structure following static or dynamic loading. The fine, loose and uncompacted state of hydraulically placed tailings and the resulting low effective stress levels make tailings more susceptible to liquefaction than natural sands.
- In addition to construction related factors such as rate-of-rise and pond control, successful management of an active tailings impoundment requires a comprehensive monitoring program and continuous detailed analysis of all data, backed up by rapid remedial action on identifying possible problems.

(c) **Gold tailings slurries:**

- Gold tailings can be classified as a low plasticity, fine, hard and angular rock flour, slurried with process water in a flocculated slightly alkaline state together with soluble salts. The flocculated state of the slurry promotes low post sedimentation densities, as the material does not readily segregate.
- Pulp densities normally range between 25 and 50% (dry density of 300 to 750 kg/m³) when delivered to the impoundment site.
- The rheology of a gold tailings slurry lies somewhere between a Bingham plastic and a Newtonian fluid with the shear stress of the slurry a function of viscosity and the rate of shear strain. The viscosity of a gold tailings slurry ranges between 2 and 7 times that of water. Shear strength only develops in the slurry at moisture contents below 30%, thus marking the change between slurry and sediment with the development of effective stresses.

(d) **Soil forming processes on an impoundment:**

- Depositional practices, variations in the mill product, and soil forming processes on the impoundment result in a highly layered profile with coarse and fine layers alternating over small depths.
- The soil forming process on a tailings impoundment includes simultaneous transportation of sediments, sedimentation, consolidation and evaporation. Transportation and sedimentation are followed by primary and secondary

consolidation, where primary consolidation is associated with the dissipation of excess pore pressures, and secondary consolidation or creep with viscous effects in the particle skeleton. Secondary consolidation in tailings is usually assumed to be negligible. If a deposit is exposed to sun-drying the effects of desiccation suctions will result in additional densification.

- The state of the material on a tailings impoundment is controlled by the properties of the slurry including particle size, shape and specific gravity, water content and viscosity, as well as deposition related flow conditions on the beach (sub-aerial) compared with those in the pond (sub-aqueous). Under sub-aqueous conditions the stress history and hindered settling result in a soft and normally consolidated state. Instantaneous sedimentation under sub-aerial conditions on the beach is preferably followed by drying, which can build in large pre-consolidation stresses and result in a relative dense, heavily overconsolidated, state. The principal advantage of such overconsolidation is the resulting increase in density and stiffness and not the temporary increase in effective stresses as a result of high pore water suctions.
- The geometry of the surface profile or beach profile on a gold tailings dam is largely a function of pulp density. The lower the pulp density the more effective the sorting processes and the steeper the beach. As the solids content of the slurry increases to a critical value mudflow commences and sorting processes no longer have an effect, resulting in a flattening of the beach slope. Being able to predict the beach slope or profile of an impoundment not only allows better management of the pond, but also improved estimates of storage volumes, freeboard and the settled physical properties of the tailings. The beach profile can be modelled using the Melent'ev master profile or a similar exponential function.

(e) **Engineering properties:**

- *Mineralogy:* Quartz is by far the most abundant mineral in gold tailings with small quantities of phyllosilicates as well as pyrites and other sulphides. Specific gravity ranges between 2.5 and 3.0. Oxidation of sulphide minerals in tailings can result in self cementing of the structure, or more importantly leaching of toxic substances. Oxidation can only take place in the presence of free oxygen and water or through anaerobic bacterial processes.
- *Grading:* Gold tailings gradings are generally limited to and uniformly distributed in the silt size range with small percentages of sand and clay sized particles. The particles throughout the gradings are angular to sub-angular with sharp edges and harsh surface textures.
- *Plasticity:* Gold tailings exhibit very little plasticity and no cohesiveness and classify, based on their Atterberg limits, as low to high plasticity silts on the Casagrande chart. Typical values of the liquid limit, plastic limit and plasticity index are: 33, 28 and 5%.

- *Density*: In-situ dry density of the slimes is close to 1000 kg/m³ and for the sands range between 1250 and 1650 with an average of 1450 kg/m³.
- *Permeability*: Permeability is usually estimated using Hazen's equation as a function of the grading properties, but more advanced formulations incorporate the density state as well. Permeability ranges between 0.5 and 50 m/yr, although there is usually only one order of magnitude difference between the permeabilities of the sands and slimes of a particular tailings product. The anisotropy ratio, k_h/k_v , is approximately 10, but can be as low as 2 and as high as 20. Desiccation cracks tend to reduce the anisotropy to about 3 on average. Permeability reductions of up to one order of magnitude or more can result from self-weight consolidation.
- *Compressibility*: Tailings appear to be more compressible than similar natural soils due to their grading characteristics, high angularity and loose depositional state. Compressibility and consolidation characteristics are significantly affected by grading, with tailings slimes reported to be three times more compressible and 6 orders of magnitude slower to consolidate than tailings sands.
- *Shear Strength*: The shear strength properties for tailings are, cohesionless with rare exceptions, and an effective angle of internal friction, ϕ' , ranging between 30 and 40°. Apparent cohesion only develops under partially saturated conditions as a function pore water suction and electrical charge attraction between clay minerals, if present. Aggregate interlock following compaction can also induce some apparent cohesive strength. Apart from this, the internal friction angle can be assumed largely independent of grading, density, overconsolidation and effective stress level up to the onset of particle crushing. Under dry or partially saturated conditions the friction angle may reduce somewhat with the addition of water due to lubricating effects.
- Tailings exhibit behaviour somewhere between that of a sand and a clay, depending on the composition and state of the material. This is especially troublesome in the interpretation of in-situ tests, where assumptions of fully drained or fully undrained shear cannot be relied on. The low density and low effective stress levels, which result from depositional processes as well as the cohesionless properties of the material make undisturbed sampling for laboratory testing practically impossible on a working impoundment. It is, therefore, extremely important in preparing reconstituted or remoulded laboratory samples to use techniques that simulate field behaviour and result in a composition and state representative of in-situ conditions.
- In triaxial shear, tailings require relatively large strains for failure and seldom dilate prior to failure. They exhibit no strain softening or strength reduction following peak shear strength. In low stress and high density states, typical of samples recovered from desiccated beaches, dilation does occur and strain softening may be a possibility as a result of the overconsolidated states of these specimens. In undrained shear, the coarser material is particularly prone to phase transfer dilation followed by strain

hardening at the critical state. Only specimens prepared by methods of wet-tamping show liquefaction potential as a result of structural collapse in triaxial shear.

(f) **Liquefaction:**

- Many incidents on tailings impoundments are ascribed to liquefaction. If failure is not caused by liquefaction in itself, liquefaction is said to be induced by other types of failure.
- Liquefaction, in general, can result from either an unstable or contractile soil skeleton with collapse potential, or when effective stresses are annulled by positive pore pressure build-up, usually during cyclic loading. Cyclic pore pressure build-up may well cause liquefaction in a tailings deposit during a seismic event, but collapsible structure has never been identified outside the laboratory. The potential for liquefaction in tailings is controlled by the level of saturation, density state, fines content and confinement stress levels.

2.6.2 Specific Issues Addressed in this Thesis

With respect to the above, this study aims to investigate the composition and state of gold tailings to advance the understanding of the engineering behaviour of this material. The following aspects will be addressed:

(a) **Composition:**

A comprehensive study will be made of the composition of gold tailings including mineralogical make-up, grading properties, particle characteristics and specific gravity. While the literature reports on some of these properties, very little if any evidence is given. It is vitally important that the fundamental properties of tailings be accurately defined before any attempt is made at studying the in-situ state and mechanical behaviour of this man-made material.

To determine the composition of the material, electron micrographs, x-ray techniques and standard soil mechanics laboratory tests will be employed as follows:

- *Mineralogy:* Energy Dispersive X-ray Spectrometry and X-ray Diffraction techniques will be employed to determine the elemental and mineralogical composition of the tailings. The mineralogy, especially the clay content, is fundamental to understanding the mechanical behaviour of tailings. This aspect has not been addressed sufficiently in tailings literature.
- *Specific Gravity:* During this standard soil mechanics test, special care will be taken to minimise potential errors resulting from poor experimental practice. Accurate measurements of specific gravity should give an indication of the importance of gravity sorting, as opposed to size sorting, in individual layers on an impoundment.

- *Grading*: Gradings will be determined using standard sieve and hydrometer tests, as well as visual observations on electron micrographs. The influence of standard preparation techniques and theoretical assumptions for the sieve and hydrometer tests will also be investigated. Of special interest is the flocculated nature of the tailings fines.
- *Particle Shape and Surface Texture*: A comprehensive set of micrographs will be prepared of the various size fractions of tailings particles to accurately determine particle size, shape and surface texture properties.

(b) **State:**

With a better understanding of the composition and fundamental properties of tailings the state and behaviour of the material will be investigated using reconstituted remoulded laboratory samples in conjunction with undisturbed field samples.

- *Compressibility*: The effect of composition on the compressibility properties of gold tailings will be examined. An attempt will also be made to predict density states using normalised compression curves and some composition related parameters.
- *Strength*: It is a well documented fact that the shear strength of a particular gold tailings is governed by a single effective stress parameter, ϕ , (typically 35°) irrespective of its grading. This observation will be verified and further investigated with the aim of improving shear strength interpretation of in-situ test data.
- *Structure*: The potential for interparticle bonding as a result of precipitating agents and the existence of collapsible fabric is mentioned in the literature. Electron micrographs of undisturbed samples and triaxial test data on undisturbed samples will be examined for evidence of bonding and fabric.

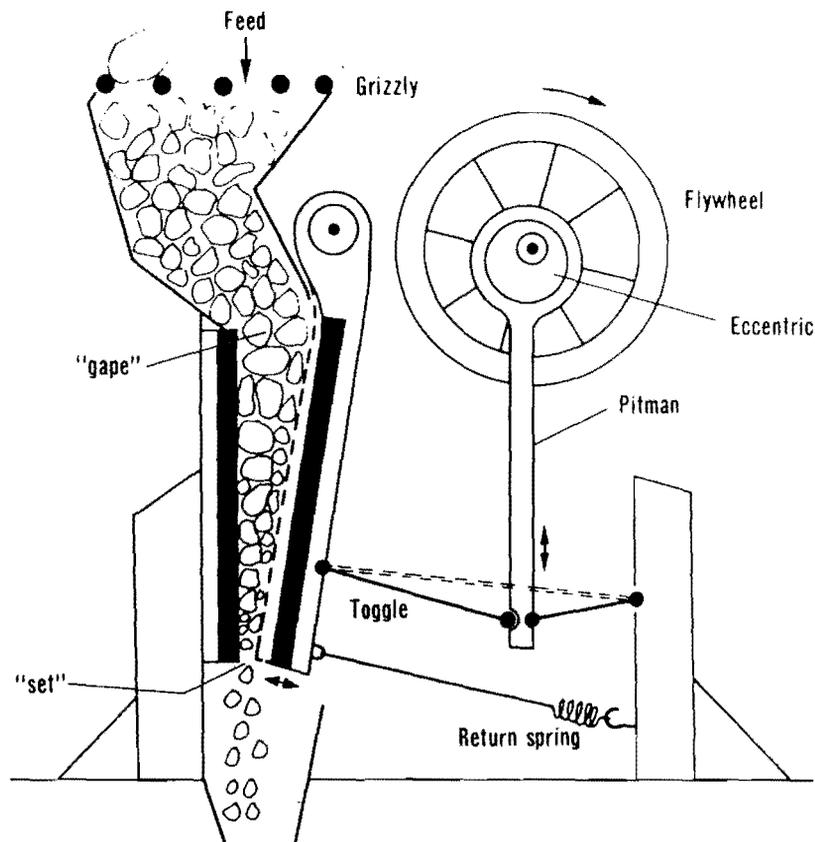
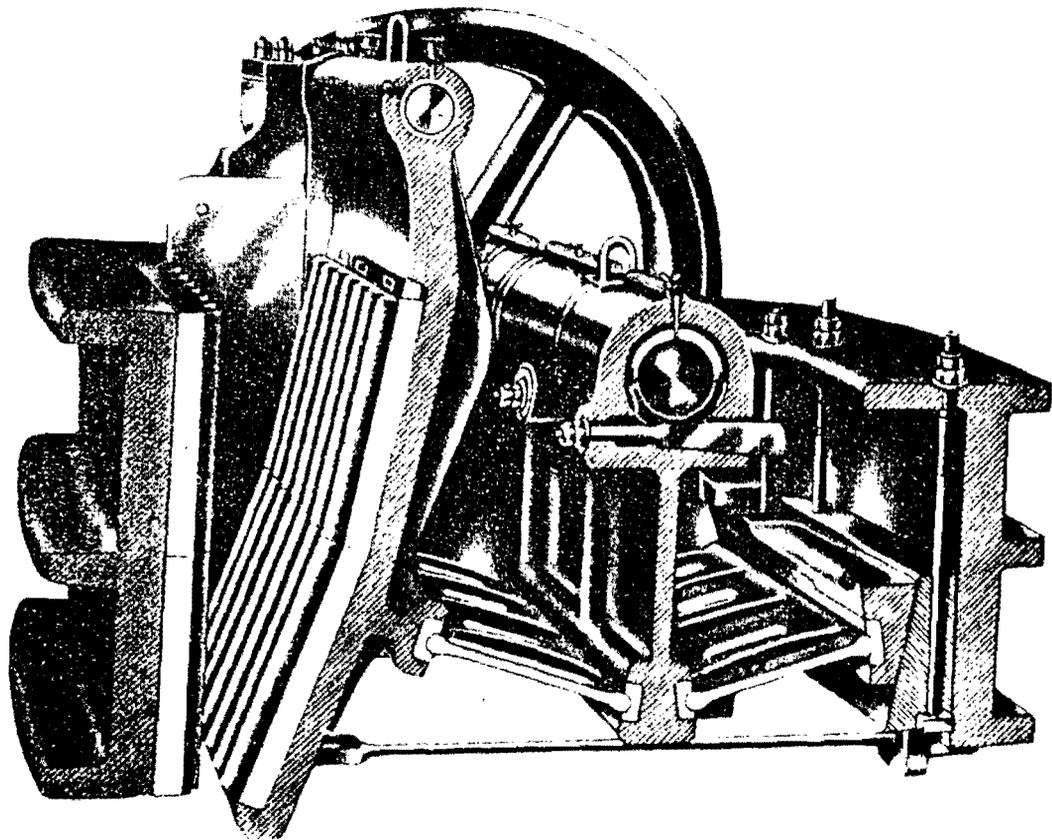


Figure 2-1: Jaw breakers (Wills, 1992; Gilchrist, 1989).

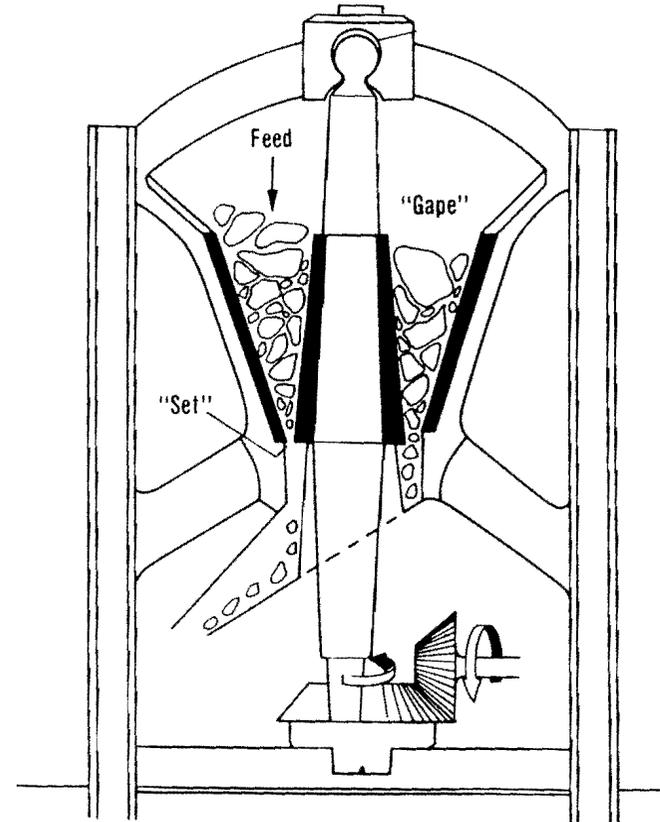
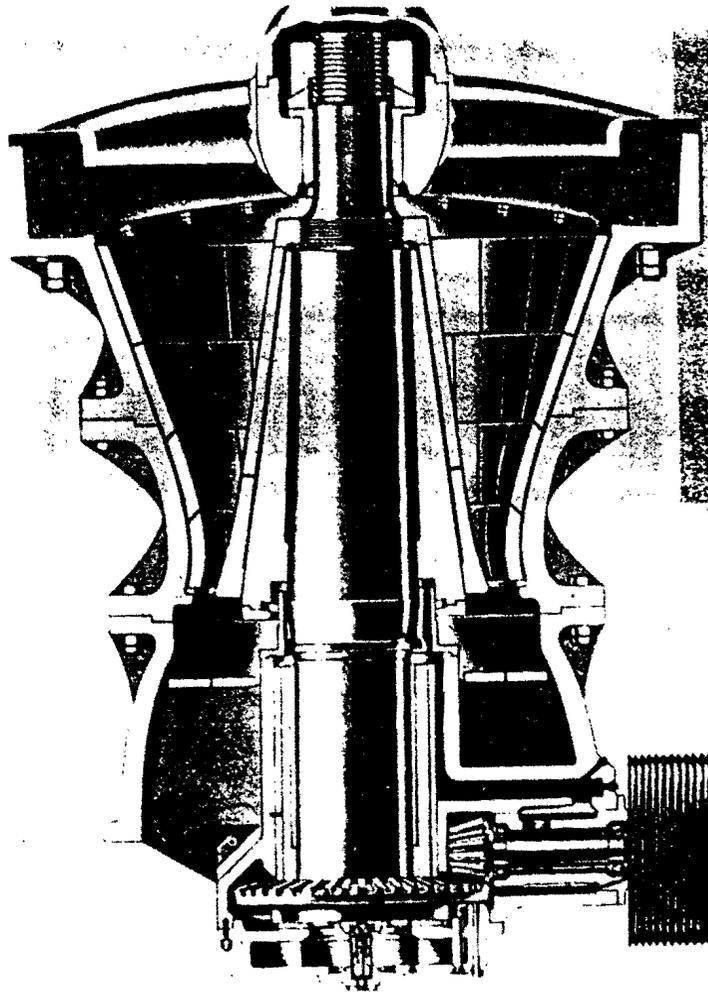


Figure 2-2: Gyratory cone breakers (Wills, 1992; Gilchrist, 1989).

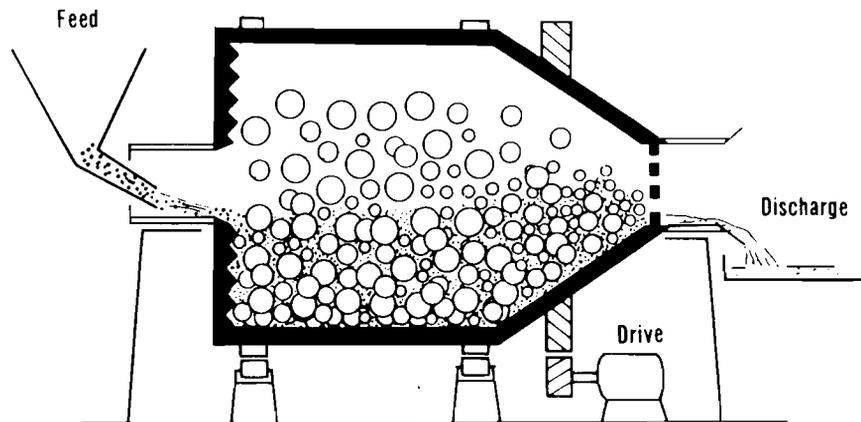
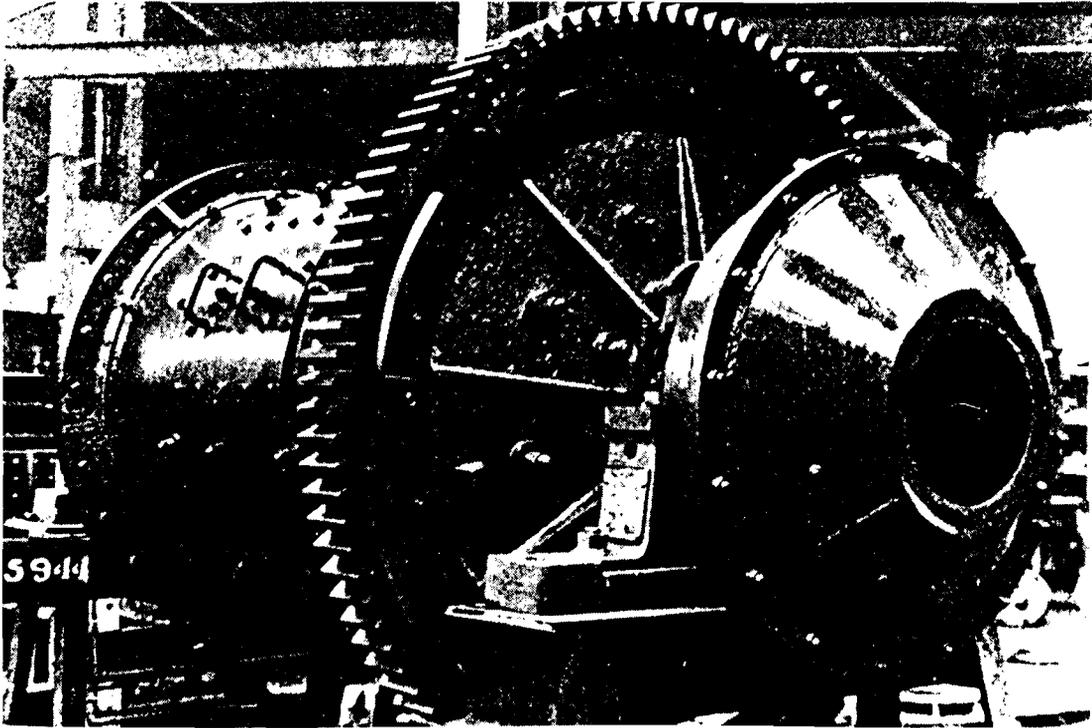


Figure 2-3: Ball Mills and Tube Mills (Wills, 1992; Gilchrist, 1989).

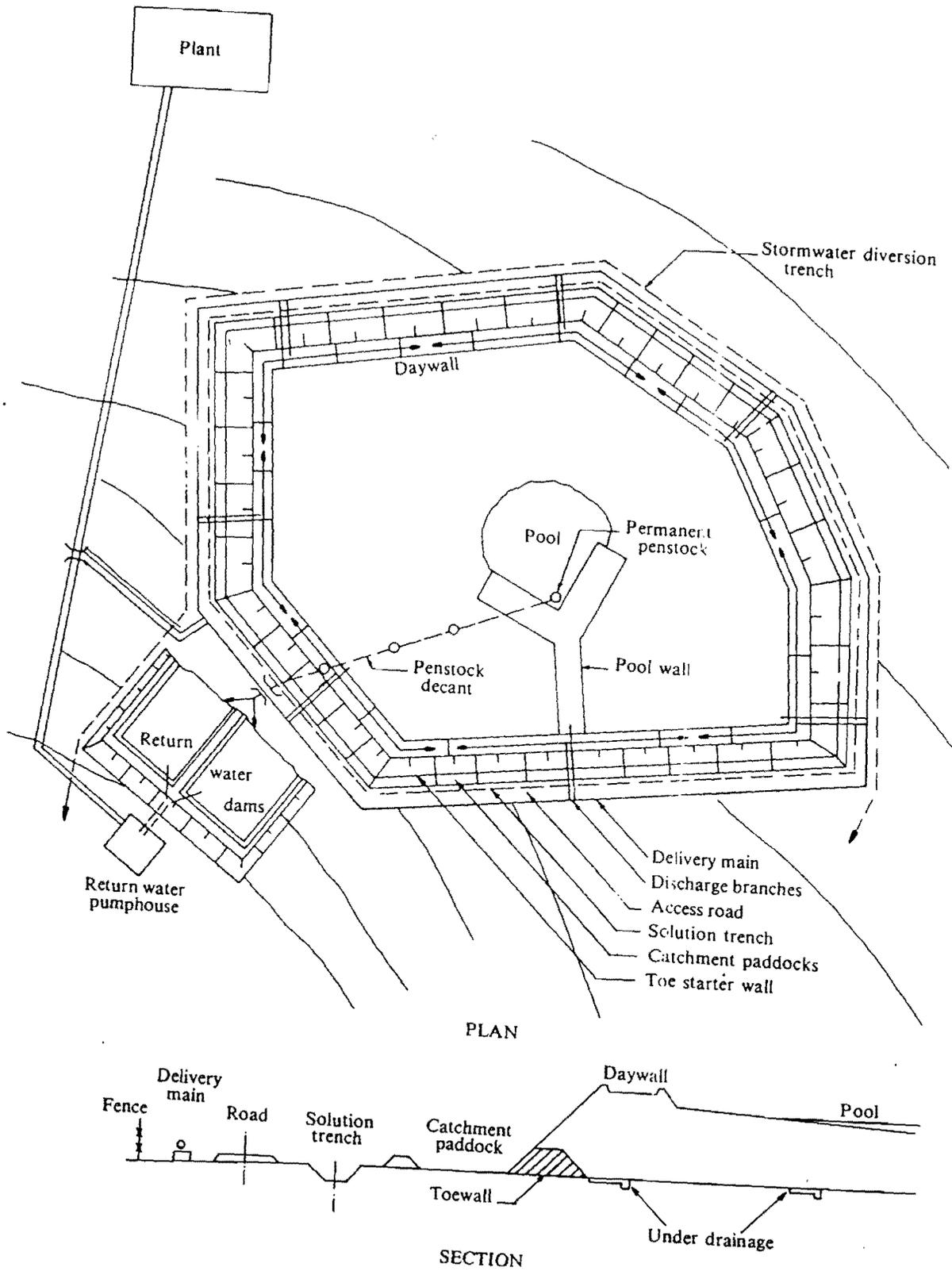


Figure 2-4: Main components of the tailings disposal system (McPhail & Wagner, 1989).

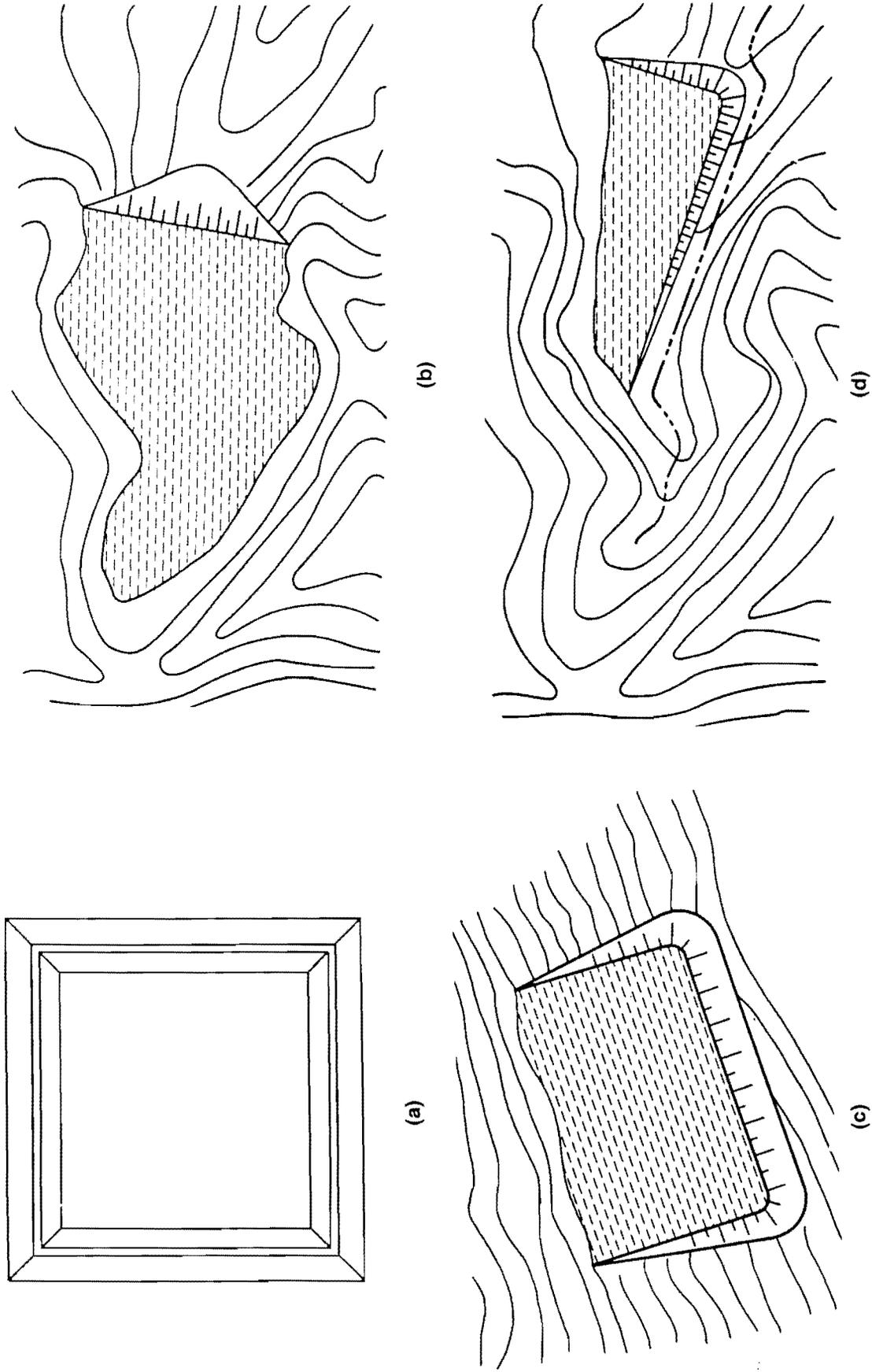
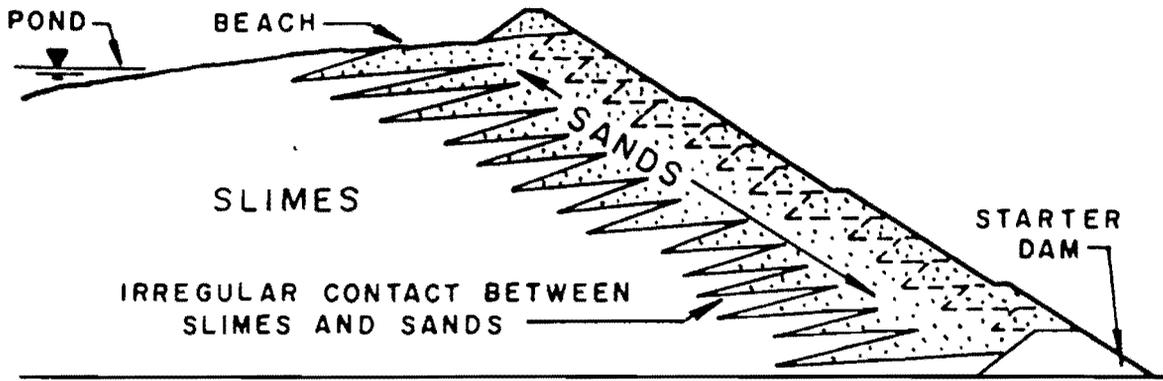
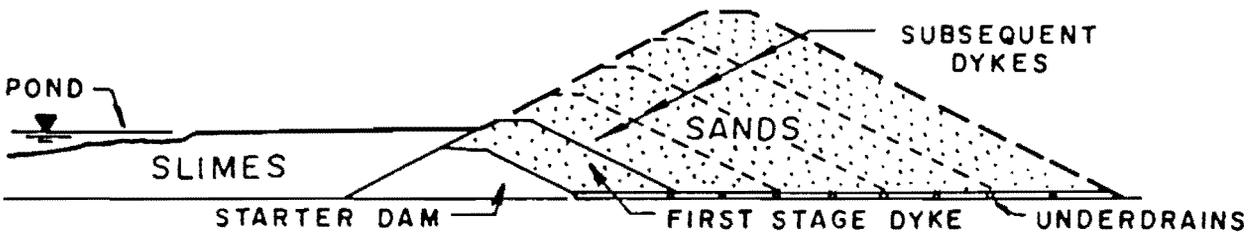


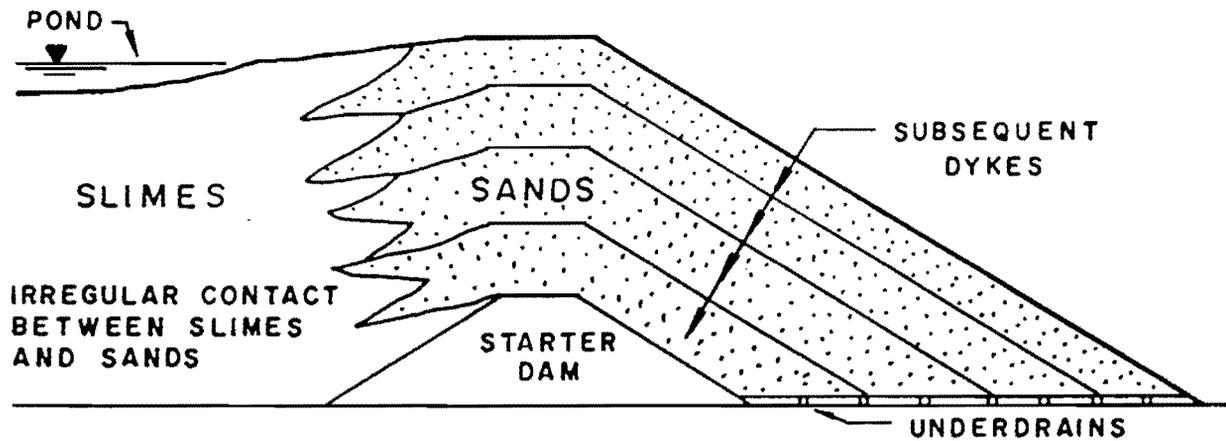
Figure 2-5: Basic impoundment layout designs: (a) ring dyke, (b) cross valley, (c) side-hill and (d) valley bottom or incised (Vick, 1983).



(a) UPSTREAM METHOD OF CONSTRUCTION

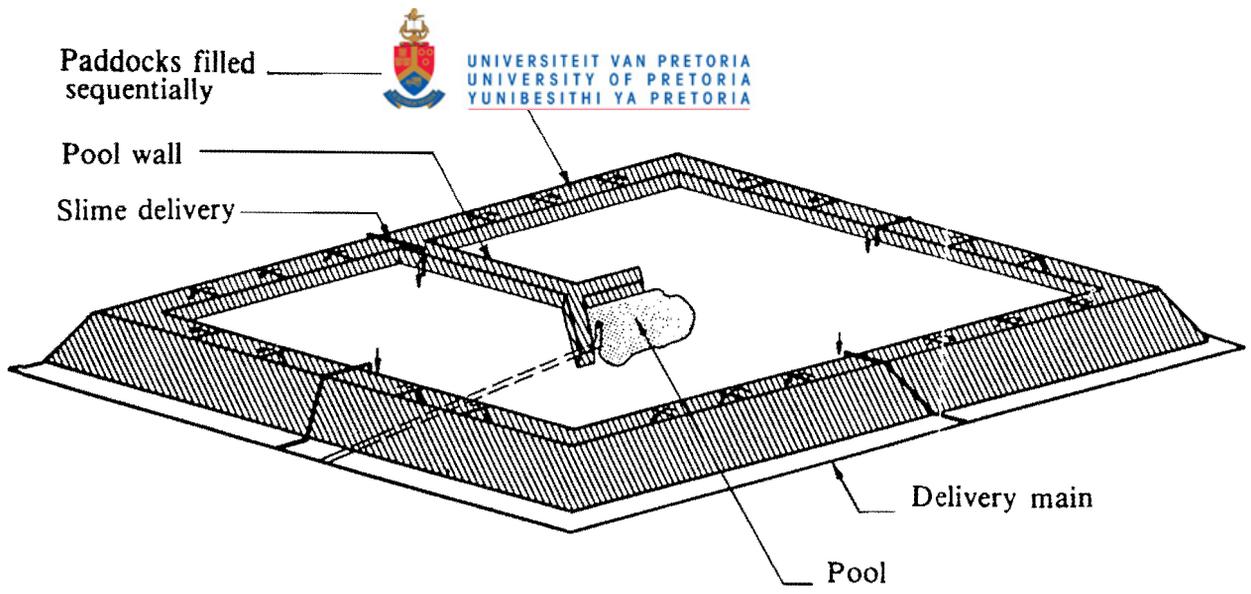


(b) DOWNSTREAM METHOD OF CONSTRUCTION

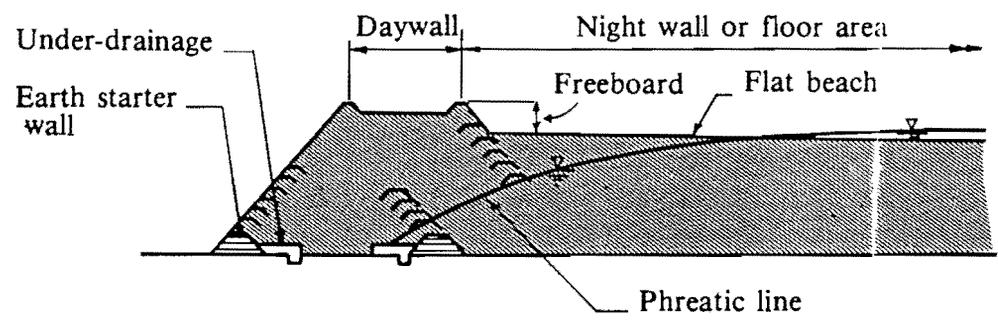


(c) CENTERLINE METHOD OF CONSTRUCTION

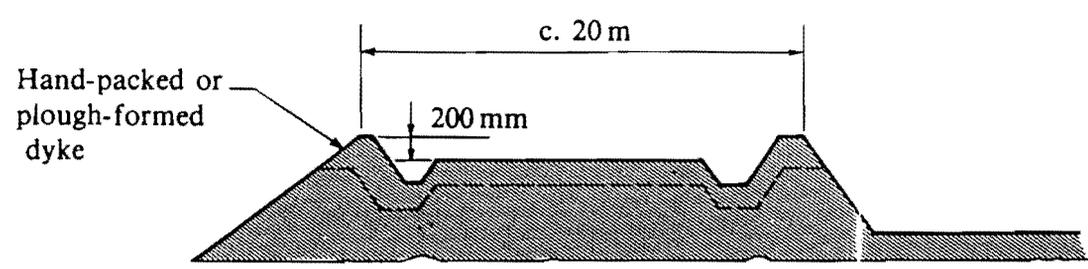
Figure 2-6: Embankment construction methods: (a) Upstream raised embankments, (b) Downstream raised embankments and (c) centreline raised embankments (Mittal & Morgenstern, 1977).



(a) Perspective view

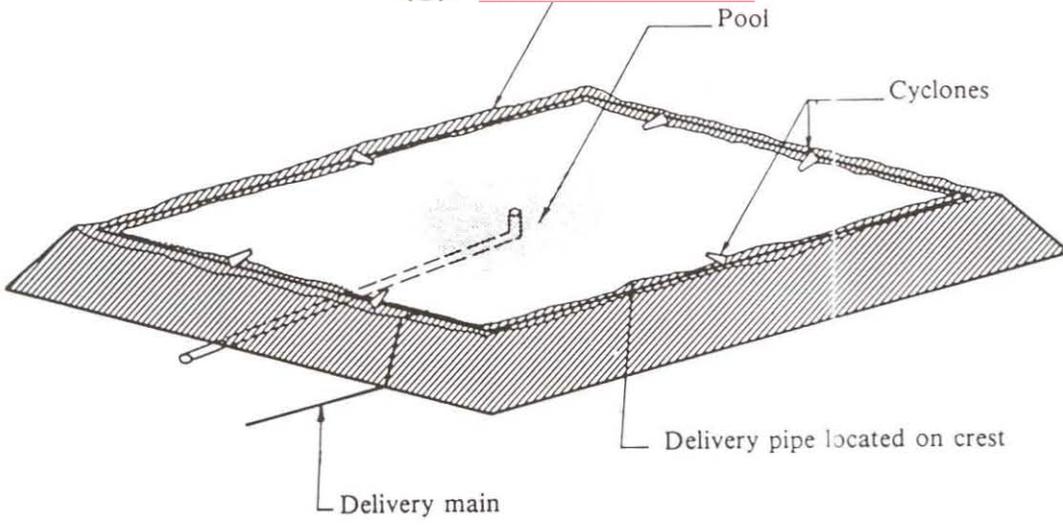


(b) Typical section



(c) Typical daywall section

Figure 2-7: Semi-dry paddock embankments (McPhail & Wagner, 1989).



(a) Perspective view

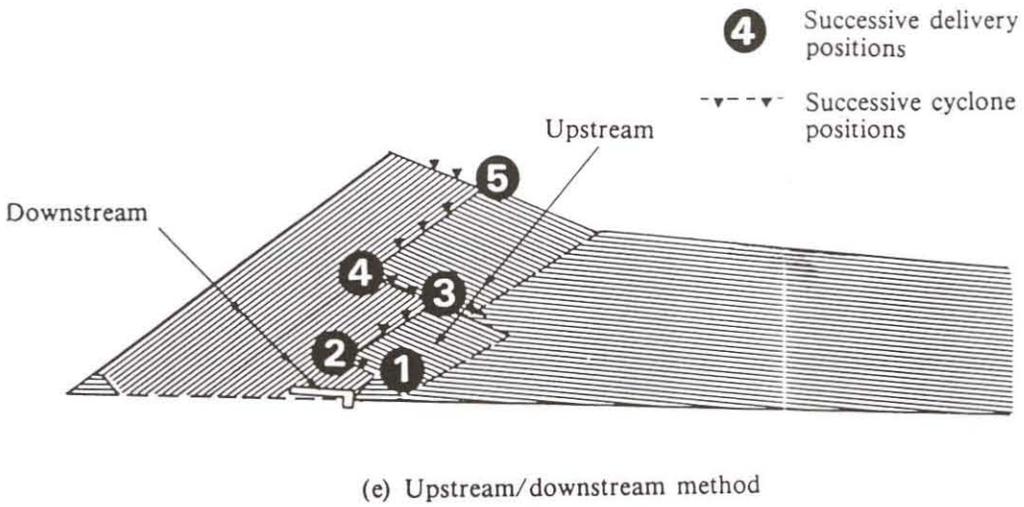
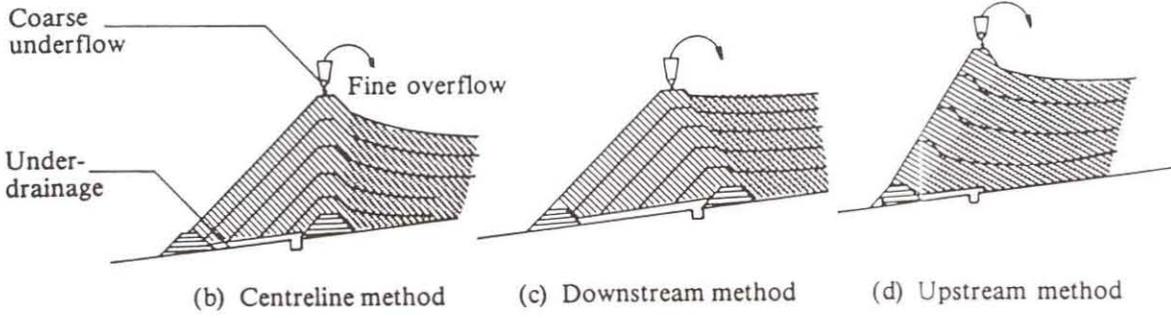


Figure 2-8: Cyclone constructed embankments (McPhail & Wagner, 1989).

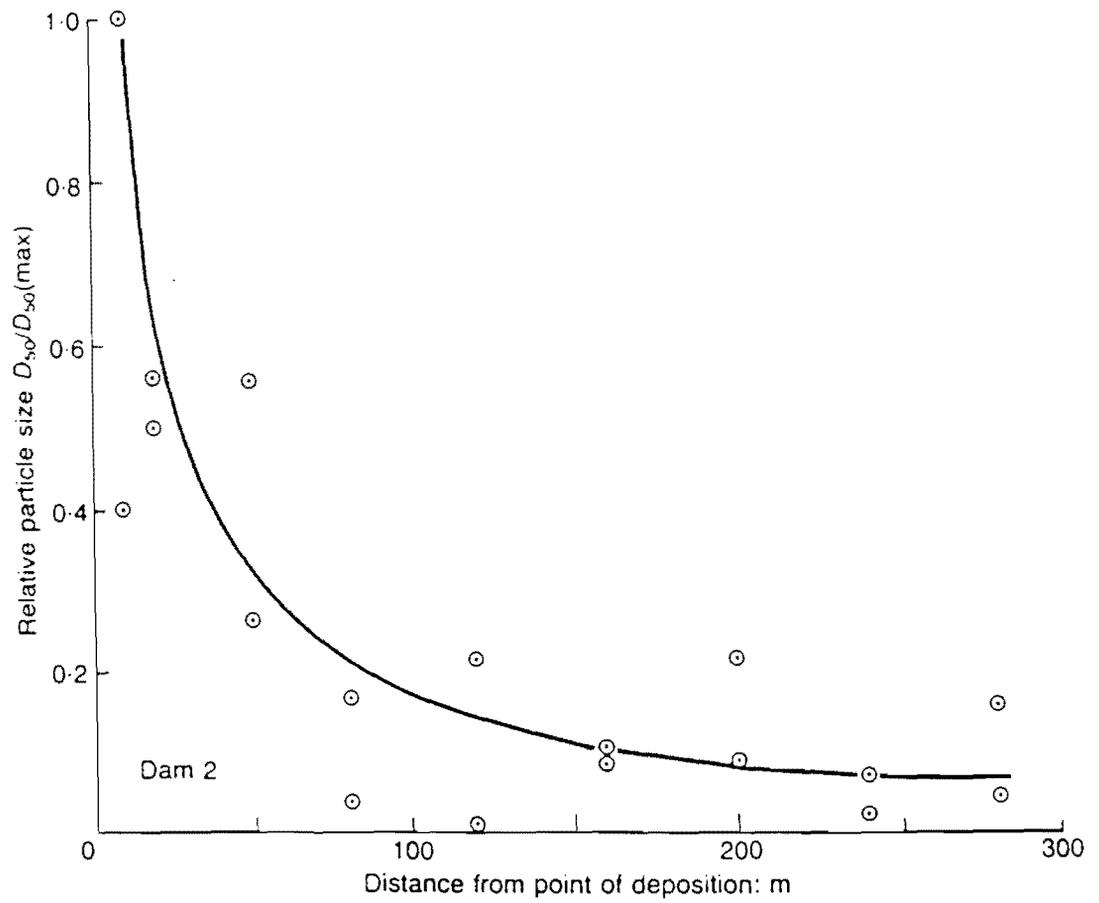
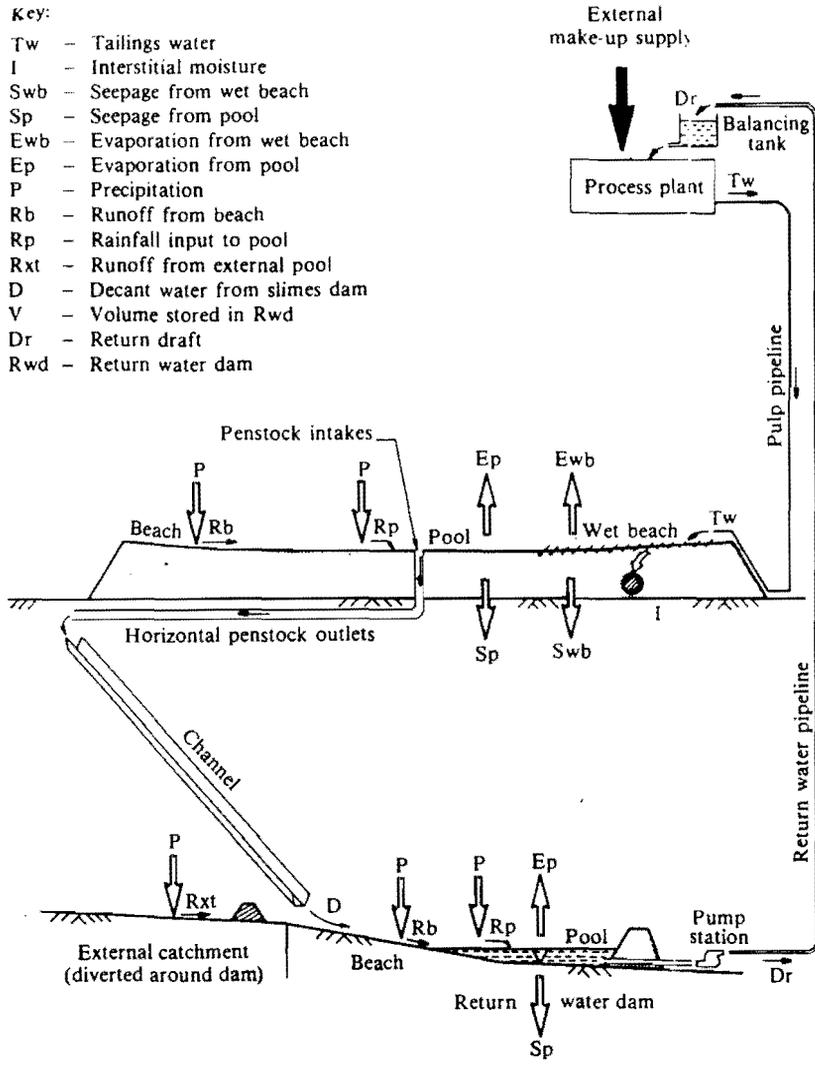
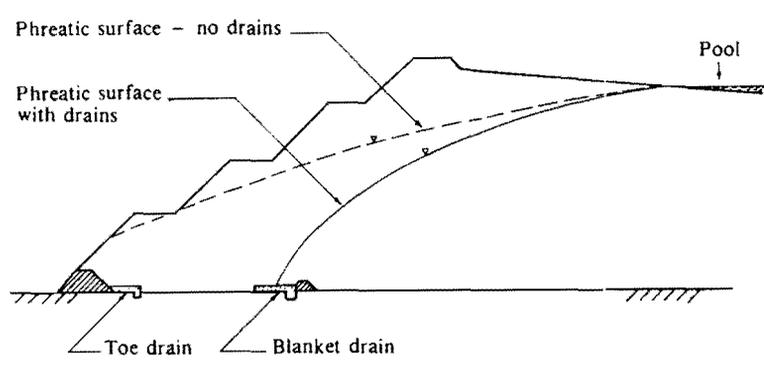


Figure 2-9: Particle size sorting observed on the beach of a diamond tailings dam (Blight & Bentel, 1983).

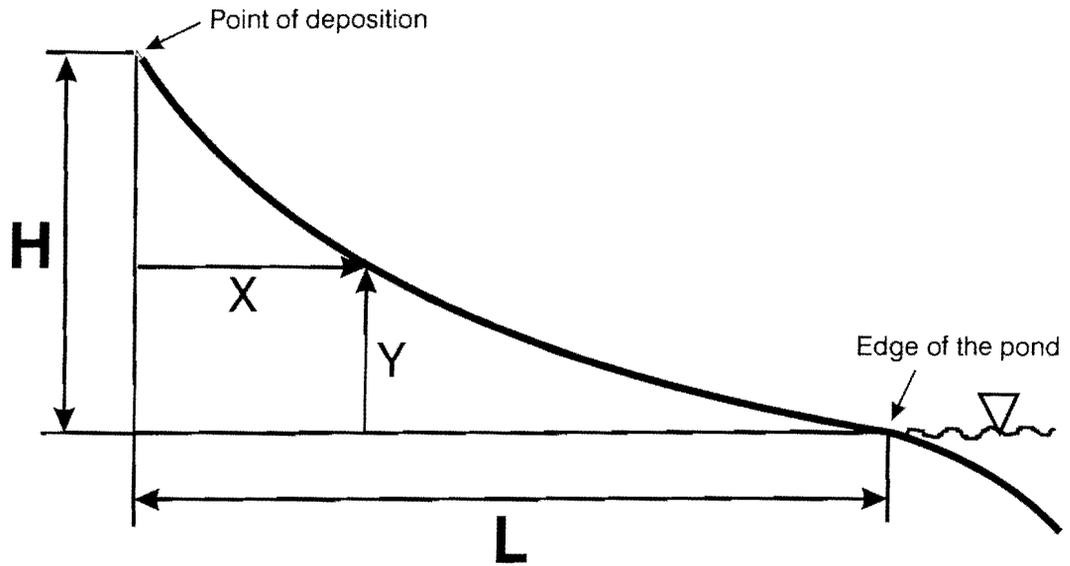


(a)

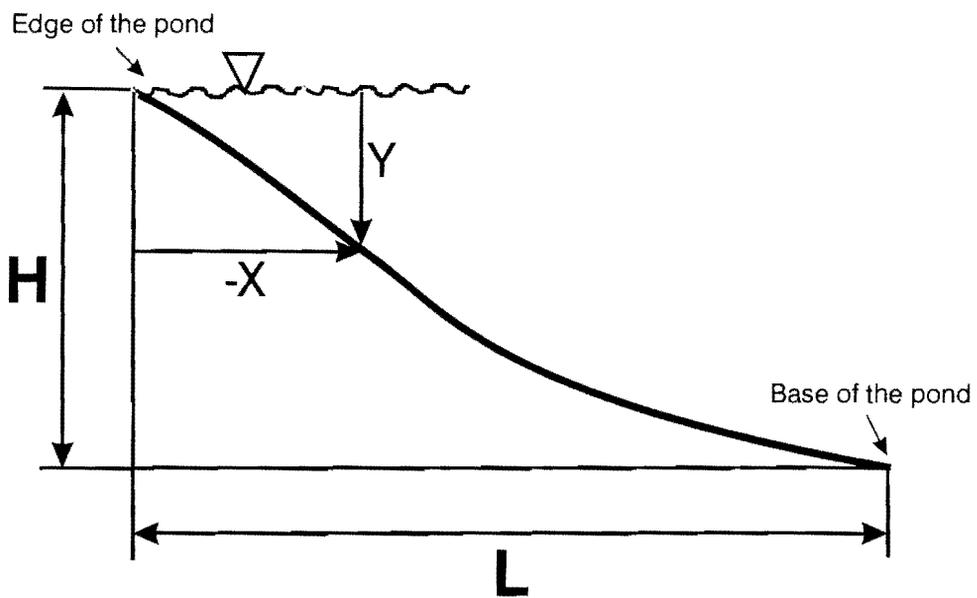


(b)

Figure 2-10: (a) Schematic of phreatic surface control in a typical tailings impoundment, (b) Influence of under wall drains on the position of the phreatic surface (McPhail & Wagner, 1989).



(a)



(b)

Figure 2-11: Schematic of the Melent'ev or Master Profile of tailings beaches for (a) sub-aerial deposition and (b) sub-aqueous deposition.

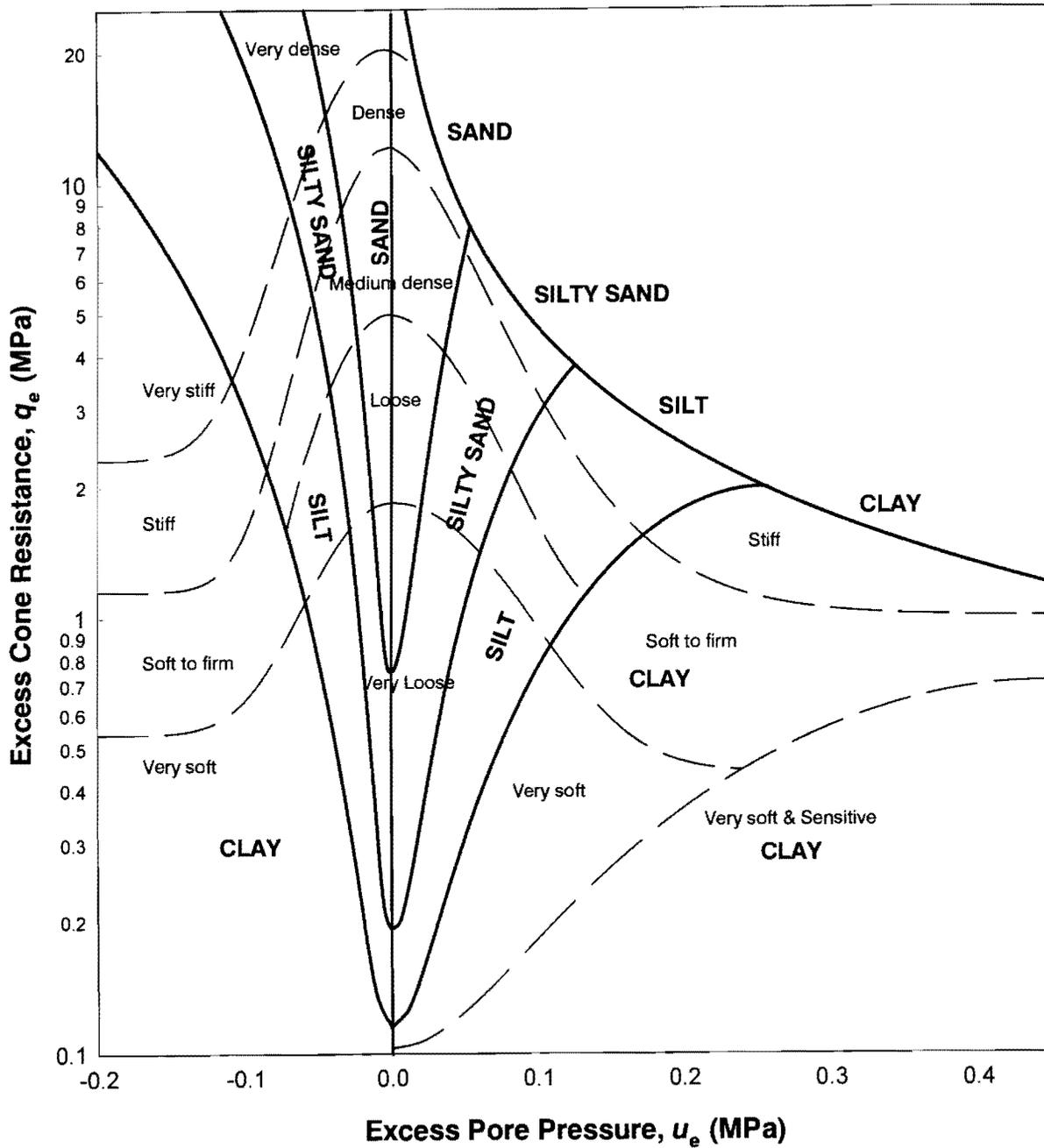


Figure 2-12: The Jones and Rust soil classification chart (Jones et al., 1981), modified by the author to emphasise the classification of soft soils.

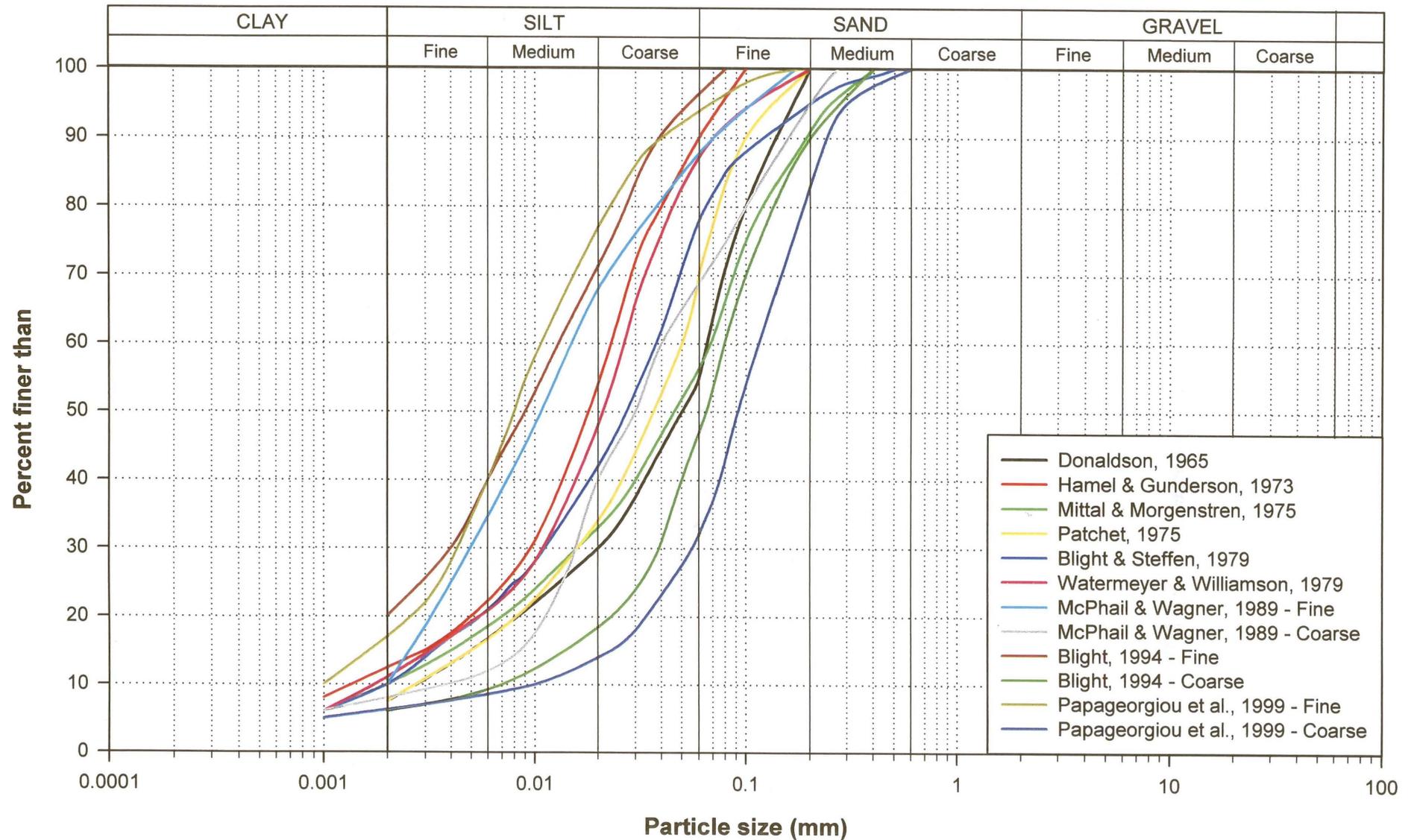


Figure 2-13: Grading curves for gold tailings.

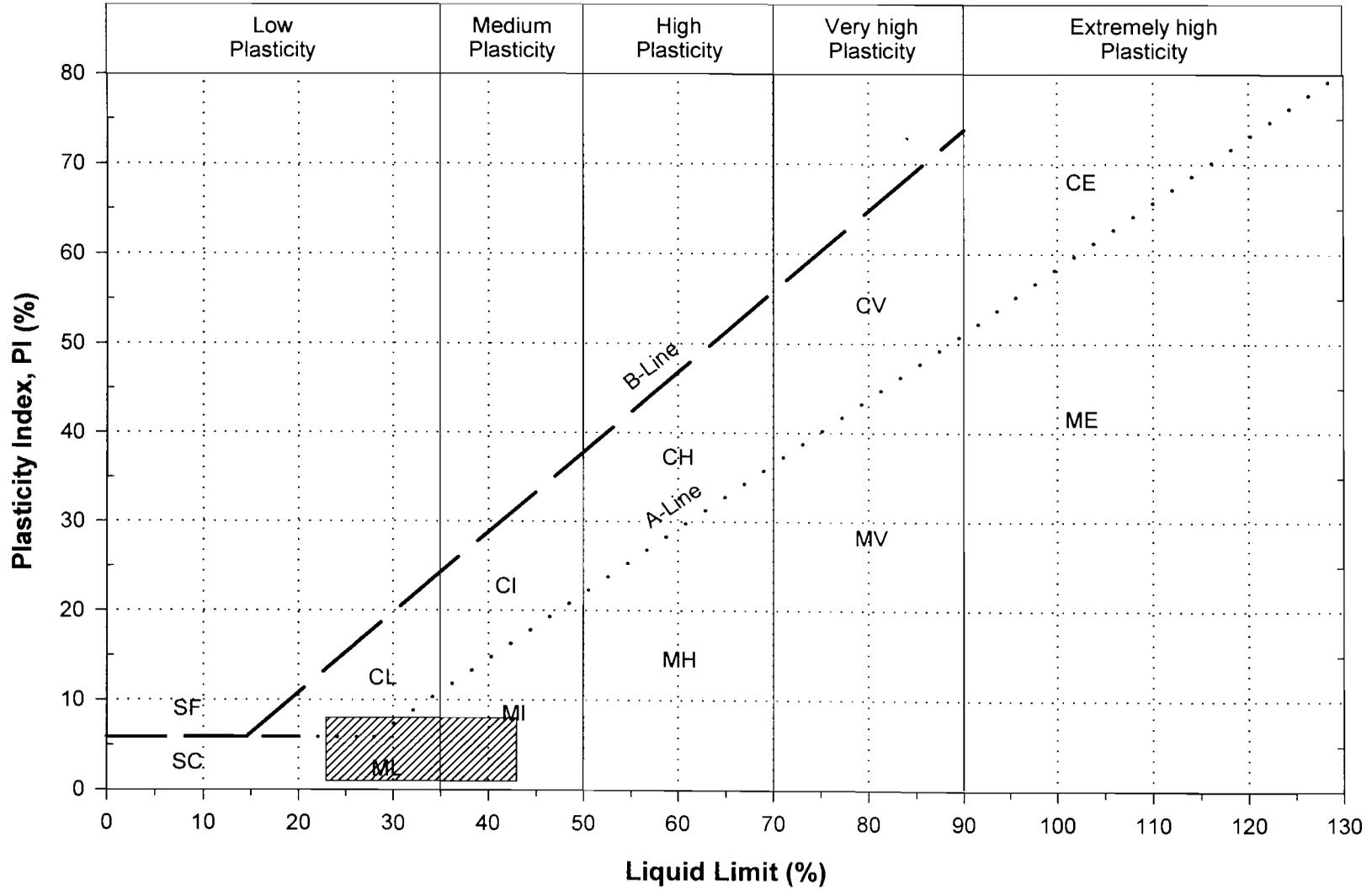


Figure 2-14: Casagrande classification of gold tailings.