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DEPARTMENT OF MINING AND ENVIRONMENTAL GEOLOGY

EVALUATION OF ECONOMIC POTENTIAL OF GOLD TAILINGS DAMS: CASE STUDIES OF THE KLEIN LETABA AND LOUIS MOORE TAILINGS DAMS, LIMPOPO PROVINCE, SOUTH AFRICA

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A DISSERTATION SUBMITTED TO THE DEPARTMENT OF MINING AND ENVIRONMENTAL GEOLOGY, SCHOOL OF ENVIRONMENTAL SCIENCES, UNIVERSITY OF VENDA, IN FULFILMENT OF THE REQUIREMENTS FOR THE DEGREE OF MASTER OF EARTH SCIENCES IN MINING AND ENVIRONMENTAL GEOLOGY

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FEBRUARY 2017

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Declaration

I **Nemapate Ndivhuwo,** hereby declare that the work contained in this master's dissertation submitted to the Department of Mining and Environmental Geology at the University of Venda, for the degree of Master of Earth Sciences in Mining and Environmental Geology, is my original work in design and implementation and it has not previously been submitted at this or any other institution of higher learning for a degree and that all reference materials contained therein have been fully acknowledged.

Student's signature.....Date...15/05/2017.....

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Co-Supervisor's signature.....Date.....16/05/17.....

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Dedication

I dedicate this research work to my parents; Dr. M. A. Nemapate and Mrs. S. P. Nemapate.

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Acknowledgement

First and foremost, I would like to give thanks to God for His everlasting love, strength and protection and favour in my life and during this research work. Secondly, I would like to express my gratitude to my supervisors Prof. J.S. Ogola and Prof. G. E. Ekosse for the supervision and advice and allowing me to be part of this project. I would like to give thanks to Mr. C. Muzerengi not only for his assistance during laboratory work, but also with advice, guidance and patience with me throughout the whole project. To my colleagues Mushiana K., Ramuada T, Shavhani T and my partner Mamaregane Beatrice Sedutla Mosema, I thank you for your assistance and encouragement during fieldwork and data collection. To my parents and my sisters, I thank you for the love, support, encouragement and having faith and never giving up on me throughout my life. You have always been a pillar of strength and hope, I thank you.

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Abstract

The tailings dams emanating from mining activities in the Giyani Greenstone Belt have high potential of containing gold in them. These tailings have since been left abandoned when the mines ceased operation 30 years ago. The ever increasing mining cost coupled with the recent surge in the gold price has necessitated the need to rework mine tailings. This can be achieved by determining the concentration and distribution of gold and heavy metals such as Ag, As, Cd, Co, Cu, Cr, Mn, Ni, Pb and Zn within the Klein Letaba and Louis Moore tailings dams.

Sampling of the tailings dam was undertaken with the collection of samples at a meter interval using a hand held auger drill up to a depth of 8 m at Klein Letaba and 4 m at Louis Moore. Samples collected were prepared at the University of Venda and heavy metals were analysed using atomic absorption spectrometry, whilst samples for gold analysis were sent to Set Point Laboratories in Johannesburg and were analysed using fire-assaying method. For major oxides, samples were sent to the Department of Geology, University of Pretoria, for XRF analysis. The Atterberg limit tests were also conducted on tailings at the Soillab in Pretoria.

The study established the abundance of As, Cr, Ni and Mn contained in the Klein Letaba tailings which had maximum values of 3943 mg\kg, 759 mg\kg, 731 mg\kg and 383 mg\kg respectively. Gold had a mean value of 468 µg\kg and the highest value was 740 µg\kg. At Louis Moore tailings dam, gold was found to have a mean value of 314 µg\kg and highest value of 480 µg\kg. The tonnage of gold at the Klein Letaba tailings dam was 699 kg. This was found to be economic although it is not technically possible to extract gold completely. At the Louis Moore tailings dam, it was not possible to evaluate the gold tonnage due to the extensive erosion of the tailings dam, however, the mean gold value at this dam was found to be 314.50 µg\kg with the maximum value of 480 µg\kg.

Analysis of the Atterberg test results revealed that the tailings are composed mainly of sandy and silty material and is lacking in clay. Hence tests conducted on the material to produce bricks revealed that the best combination is that of adding clay and coal waste

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to tailings in the proportion of ; tailings - 10%, clay - 80% and coal waste - 10%. The study thus, recommends the reprocessing of gold and the use of tailings to produce bricks with the addition of clay and coal waste. This would go a long way in solving environmental impacts caused by tailings erosion and dispersion of heavy metals to the environment. While recovering gold from tailings at the Klein Letaba tailings dam.

Keywords: *Tailings dams, Economic potential of gold, heavy metals, bricks production*

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List of Acronyms and Symbols

AAS	Atomic Absorption Spectrometer
DWAF	Department of Water Affairs and Forestry
GGB	Giyani Greenstone Belt
ppm	Parts per million
Ag	Silver
As	Arsenic
Cd	Cadmium
Co	Cobalt
Cr	Chromium
Cu	Copper
Mn	Manganese
Ni	Nickel
Pb	Lead
Zn	Zinc
Au	Gold
AASHTO	American Association of State High Way and Transportation Official
ASTM	American Society for Testing and Materials
OPC	Ordinary Portland Cement
ML	Inorganic and organic silts

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CHAPTER 1: INTRODUCTION

1.1 Background

The Giyani Greenstone Belt (GGB) is situated in the north-eastern part of Polokwane town and is well known for its gold mineralisation. Gold had been previously exploited at the Klein Letaba, Louis Moore, Birthday, Golden Osprey and Fumani mines. These mines were operational about 30 years ago and were not rehabilitated properly, leaving heavy metals within the tailings dams. This has led to the area being prone to negative environmental impacts of mining (Maarten and Lewis, 1997).

With the ever rising cost of mining and processing of precious metals and the increasing gold price in the world market, reprocessing of gold mine tailings dams is gaining momentum as it is associated with low operating costs in the recovery of gold and sometimes leads to the rehabilitation of such tailings dams. It is important to investigate the concentration of gold and heavy metals associated with gold in order to determine whether or not the tailings dams will be economically viable to reprocess. The common heavy metals occurring within gold mine wastes are lead (Pb), zinc (Zn), copper (Cu), arsenic (As), cadmium (Cd), cobalt (Co), nickel (Ni), manganese (Mn) and chromium (Cr).

Some tailings impoundments are hydraulically mined by just re-processing the tailings and re-locating them to environmentally friendly sites. This is an effective method and can take a shorter time than mechanical excavation, depending on the density and particle size of the tailings. For example, some of the old tailings dams around Johannesburg in South Africa have been reprocessed using hydraulic monitors.

Historically, mineral dressing processes have been somewhat inefficient because after crushing, grinding and metallurgical treatment, some amount of valuable metals are left behind in the residue. Ever-increasing price of gold and other precious metals in the marketplace have opened the opportunity for mining companies to purchase old mine tailings dams and metallurgically treat the tailings in order to recover residual gold (Baumgart, 2006).

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South African company Mintails Limited is currently focused on exploring, evaluating, and reprocessing hundreds of millions of tons of surface gold-bearing tailings in the Witwatersrand Basin of South Africa, a region that has produced more than 40% of all the gold ever mined from our planet *(http://www.epa.gov, 2008).*

1.2 Study Area

1.2.1 Location

The study focused on the two tailings dams, Klein Letaba and Louis Moore, in the Giyani Greenstone Belt, Limpopo Province (Fig.1.1). Klein Letaba tailings dam is located about 26 km west of Giyani town along the R578 road to Elim. The dam is situated within latitude 23° 17' and longitude 30° 33'. The Louis Moore tailings dam is located about 10 km north of Giyani town off R81 road and within Latitude 23°13' and Longitude 30° 41'.

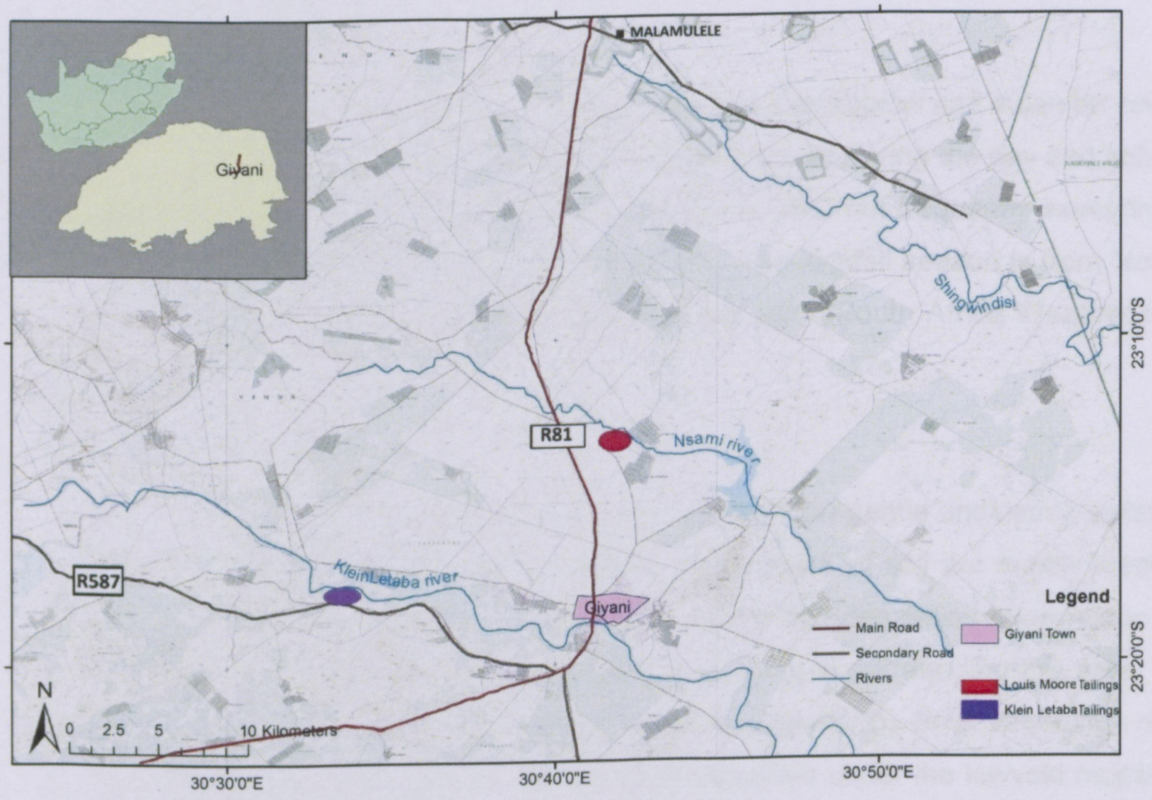


Figure 1.1: Location map of the study areas (Map of South Africa, by South Africa Channel, http://www. South Africa.com/maps).

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1.2.2 Topography and Drainage

The Giyani area is characterised by steep slopes along its western and southern boundaries which gradually dissipate into undulating hills in the central areas, to Mangombe and Khavagari mountains which are located to the eastern and western parts of the town respectively.

There are four major non-perennial rivers in the area, namely; the Shingwidisi, Nsama, Klein Letaba and Malotsi. The gross surface water availability in the Klein Letaba sub-area is estimated at 27 million m^3 per annum, most of which is derived from the yield of the Middle Letaba Dam and the smaller dams upstream (DWAF, 2004). There is also a relatively large groundwater resource in this catchment. Groundwater was used to supply most of the rural population in the sub-area, but much of this has now been replaced by reticulated supply from the middle Letaba Dam (DWAF, 2004).

1.2.3 Climate

The Giyani area is generally warm, dry, frost-free and sub-tropical and it can be very hot in summer during the day and night, whereas winters are mild during the day and cold during the night. The summer average temperature is about 35°C but frequently exceeding 40°C and the winter average temperature is about 20°C. The rainfall season is from November to March with average precipitation of 386 mm per year (South Africa Weather Service, 1980-2003).

1.2.4 Soil and Vegetation

The soil is loamy sand and clayey and is identified with the gentle undulating valley of the Soutpasberg Mountain range (Acocks, 1988). Deep strata of soil are mainly identified in low lying areas and at the base of mountains where soil deposition takes place due to transportation from high elevation of mountainous terrains surrounding the area. In high elevated areas, the soil depth is shallow though it supports the thick vegetation cover at mountain summits.The vegetation in the area is classified under the lowveld mopane veld savannas (Rutherford *et al.*, 2009), characterised by a mixture of trees, shrubs and grasses. Mopane tree occurs in abundance in the area together with many other species such as *Acacia, Commiphora, Maroela* and occasionally *Baobab species.*

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1.3 Problem Statement

In South Africa an extensive study of the Witwatersrand Basin tailings dams has led to gold reprocessing of these tailings dams. However, the Giyani tailings dams lack prerequisite data to support the potential of tailings as a construction material and gold reprocessing of the tailings dams, the subsequent rehabilitation of such dams. Evaluation of such tailings dams would lead to their economic viability.

1.4 Justification

The purpose of this study was to evaluate the economic potential of gold tailings dams at the Klein Letaba and Louis Moore in the Giyani greenstone belt. The establishment of gold values and potential of tailings as a construction material within the tailings dams would lead to tailings dams reprocessing and recovery of gold. This would further lead to tailings dams rehabilitation, thus eliminating the negative potential effects of tailings on the environment and on human health. Rising costs of mechanical mining and rising price of gold in the world market make the reprocessing of gold mine tailings a better opportunity to recover residual gold that remained within tailings during mineral dressing.

1.5 Research Questions

- Which heavy metals are associated with gold tailings dams?
- What are the quantities and distribution pattern of such heavy metals within the Klein Letaba and Louis Moore tailings dams?
- What is the economic potential of the Klein Letaba and Louis Moore gold tailings dams?
- What is the quality of the Klein Letaba and Louis Moore tailings as a construction material?

1.6 Objectives

The main objective of the study was to establish the occurrence and quantity of gold and associated heavy metals within the Klein Letaba and Louis Moore tailings dams in the Giyani Greenstone Belt, hence establish the potential for gold reprocessing and use of tailings as a construction material.

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Specific objectives were:

- undertake auger drilling, logging and sampling of Klein Letaba and Louis Moore tailings dams;
- conduct geochemical analysis of the collected samples for Pb, Zn, Cu, As, Cd, Co, Ag, Ni, Cr, Mn and Au using atomic absorption spectrometry and fire assaying for gold;
- determine the distribution pattern of the heavy metals within the tailings dams;
- evaluate the economic potential of gold for reprocessing; and
- determine the suitability of tailings as a construction material by determining the atterberg limits and pilot production of bricks from tailings.

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CHAPTER 2: LITERATURE REVIEW

2.1 Geological Setting of Giyani Greenstone Belt

The Giyani Greenstone Belt (GGB) which was previously considered to be part of the Sutherland greenstone belt is located in the Limpopo Province of South Africa (Fig. 2.1). The surrounding rocks are the Swazian age Goudplaats gneisses. The GGB is approximately 70 km long and up to 17 km wide (Maarten and Lewis, 1997). It is composed of metamorphosed volcano-sedimentary succession comprising schists and gneisses, intruded by the Kaapvaal Craton. To the southwest the greenstone belt is divided into two arms, the Khavagari and Lwaji arms (Fig. 2.1).

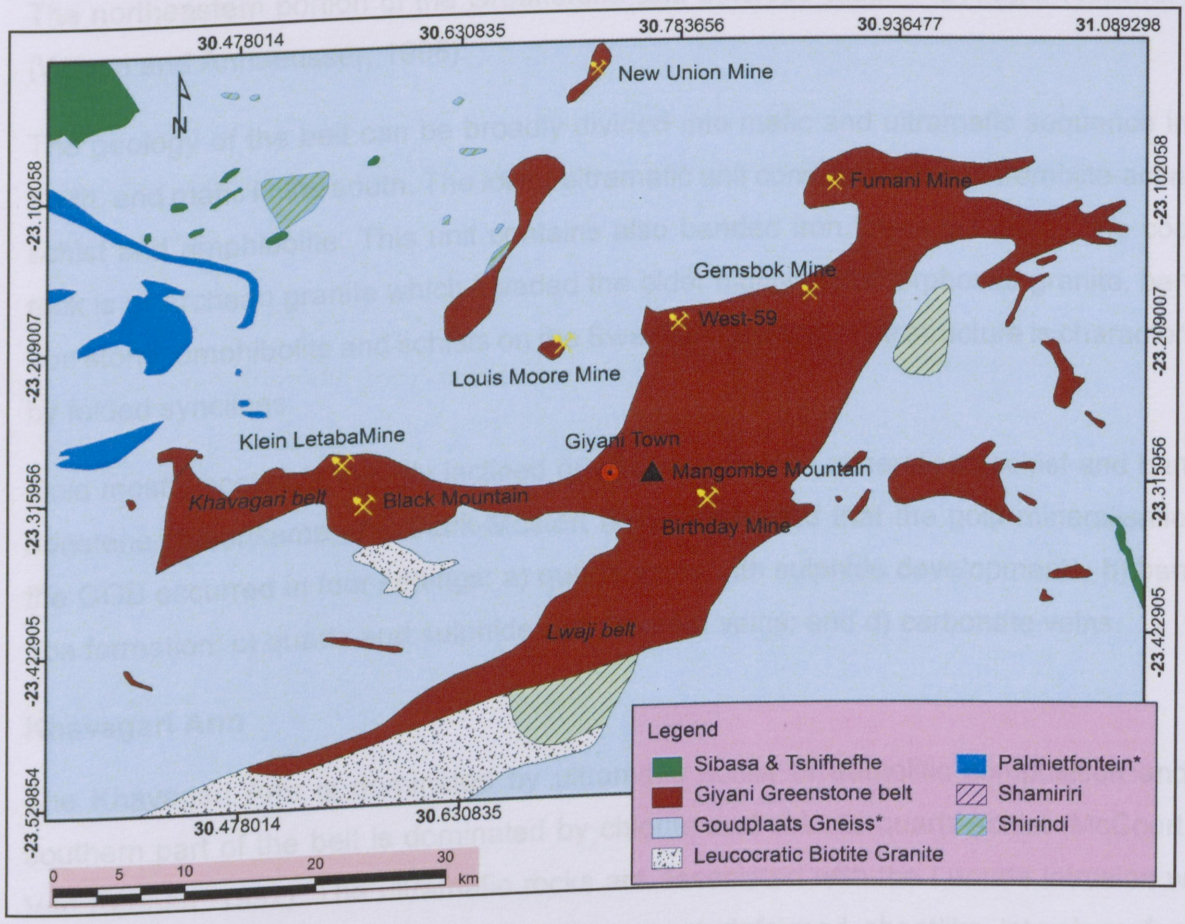


Figure 2.1: Generalised geological map of the Giyani Greenstone Belt (Ward and Wilson, 1998).

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The northen portion of the GGB is dominated by ultramafic and mafic sequences, while the southern part is dominated by mafic and a surbodinate of felsic units which is termed the Giyani Group. The GGB contains an interlayer of chemical sediments in the form of chert and banded iron formation (Wilson and Anhaeusser, 1998). The GGB also contains dolomitic sedimets in the Mangombe Hill. Zircron dating of a felsic metavolcanic from the Giyani Group yielded an age of 3202 Ma (Wilson and Anhaeusser, 1998).

Metamorphism of the GGB varies from greenschist to granulite facies. The metamorphic grade increases from the southwest to northeast. Evidence of hydrothermal activity is demonstrated by the presence of large amounts of magnesite, talc and fibrous tremolite. The northeastern portion of the Greenstone Belt outcrops within the Kruger National park (Wilson and Anhaeusser, 1998).

The geology of the belt can be broadly divided into mafic and ultramafic sequence in the north, and mafic in the south. The lower ultramafic unit consists mainly of tremolite-actinolite schist and amphibolite. This unit contains also banded iron formation (BIF). The country rock is of Archean granite which invaded the older highly metamorphosed granite, banded iron stone, amohibolite and schists on the Swaziland System. The structure is characterized by folded synclines.

Gold mostly occurs in steeply inclined quartz veins and in mineralised schist and banded ironstone. Steenkamp and Clark-Mostert (2012) indicated that the gold mineralisation in the GGB occurred in four settings: a) quartz veins with sulphide developments; b) banded iron formation; c) quartz and sulphide replacement veins; and d) carbonate veins.

Khavagari Arm

The Khavagari Arm is dominated by ultramafic schist of tremolitic composition and the southern part of the belt is dominated by chlorite and chlorite-quartz schist (McCourt and Van Reenen, 1992). The ultramafic rocks are associated with the Leonde intrusion which was interpreted by Prinsloo (1977) to represent deformed sheetlike intrusions forming disconnected hills of serpentinite covered to a large extent by birbirite. The serpentinite hosted in tremolite-actinolite, talc-tremolite, tremolite-chlorite and chlorite schists also contains magnesite which has been mined historically (van Zyl *et al.*, 1942).

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The Khavagari arm is overlain by a succession of sedimentary rocks including iron-formation, ferruginous and vitreous quartzite and politic schists with mafic tuff. At the top of the succession an ultramafic unit contains two thin layers of spinifex-textured komatite interlayered with coarse and fine-grained amphibolite (Prinsloo, 1977).

Lwanji Arm

Along most of the southern contact of the Lwanji arm, two large bodies of granite are present, termed the Pade and Shamiriri granites (Vorster, 1979). In the Lwanji arm, ultramafic rocks are represented, in varying proportions, by tremolite, talc, chlorite or hornblende (Vorster, 1979). The mafic units are massive and are pillowed hornblende-bearing amphibolites. The iron-formation is composed of lamina of goethite with haematite grading upwards into grunerite and magnetite rock (Brandl and Kroner, 1993). The structural base is formed by ultramafic schists which pass upwards into a succession of metasediments including phylite, quartz-sericite-chlorite schist and quartzite (Kroner *et al.,* 2000).

2.2 Gold Mineralisation in the Giyani Greenstone Belt

The Giyani Greenstone Belt contains a number of mineral deposits including gold, magnesite and various pegmatite related minerals such as muscovite, feldspar, beryl and tantalite (Brandl *et al.,* 2006). The association of gold and banded iron formation has long been recognised as important in gold exploration in greenstone belts of Zimbabwe and South Africa. The abundance of mafic to ultramafic rocks in the Giyani greenstone belt with associated banded iron formation makes these units good exploration targets for gold mineralisation (Minnitt, 1980).

There are 44 known gold occurrences in the Giyani greenstone belt, though six of these, namely; the Klein Letaba, Louis Moore, Golden Osprey (New Union), Fumani, Franke and Birthday mines, have produced in excess of 97% of the more than 10 t of gold recovered from the belt (Wilson and Anhaeusser, 1998).

The deposits are structurally controlled with mineralisation at the Klein Letaba, Fumani, and Birthday deposits as well as several abandoned deposits unequivocally linked to the

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occurrence of the north-dipping oblique to reverse ductile shear zones (McCourt and Van Reenen, 1992). Mineral assemblages in auriferous metapelites at Klein Letaba and Fumani suggested that gold mineralisation was late and related to the retrograde metamorphism of amphibolite and granulite facies and mineralised fluids moved along ductile shear zones (Pretorius *et al.,* 1988).

Pretorius *et al.* (1988) noted that the more significant gold occurrence was concentrated along the north-western margin of the belt and can be linked to the north-dipping ductile shear zones. Gold mineralisation was closely associated with sulphide minerals of which pyrrhotite show the highest frequency of occurrence (Van Reenen *et al*., 1988). Microprobe analysis of sulphide minerals from Franke mine indicated that sulphides comprise pyrrhotite, chalcopyrite; arsenopyrite and pyrite, while gold occurred as inclusions and discrete grains in quartz and sulphides (Van Reenen *et al*., 1988). An assessment of gold mineralisation by Gain *et al*. (1986) was able to ascertain that in the Giyani greenstone belt gold is associated with shearing along with pyrrhotite, chalcopyrite, pyrite and arsenopyrite being the main sulphide minerals.

2.3 Mining History in the Giyani Greenstone Belt

The Murchison and Sutherland Greenstone Belts goldfields were discovered in 1870 by the prospectors, Button and Sutherland. Gold was first recovered from the Letaba and Shingwedzi rivers in the Sutherland range. The gold rush to this area only started in 1886 and the area was declared a public digging by the Zuid-Afrikaansche Republic. This also resulted in the founding of the town of Leydsdorp. Mining was interrupted by the Second Anglo-Boer War and after the war the rich goldfields of the Witwatersrand Basin had surpassed the smaller deposits of the Eastern Transvaal. The Gold ore was in both Murchison and Sutherland greenstone belts that were found to be refractory at depth (Swash, 1988) and by 1928 most of the mines in the area had ceased operations (Davis *et al.,* 1988).

Shortly after the discovery of gold in the Murchison and Sutherland Greenstone Belts several mines were opened, including the Eclipse (now part of the Klein Letaba holdings), Louis Moore, Ellerton, Tombstone and Birthday mines. In 1894 the Klein Letaba gold field

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was proclaimed, but erratic yield, heat and malaria soon drove most prospectors away. In 1934 gold mineralisation was discovered at the New Union deposit. A comprehensive reappraisal of many of the prospects was undertaken by the Klein Letaba Mining Company during 1940s and 1950s (Lombaard, 1956). By 1968, when the Klein Letaba mine closed down, the Louis Moore was the only producing mine in the area and that eventually closed in 1978. However, Increase in the gold price stimulated renewed interest and the Giant Reefs mine was reopened in 1976 and renamed the Fumani Mine. In 1980, the Louis Moore Mine was reassessed by Mining Corporation and the dams were reprocessed between 1981 and 1985. The New Union Mine (renamed Golden Osprey in 1987, then Northfield Gold Mine in 1993) reopened in 1982. Between 1986 and 1995 the Klein Letaba mine resumed operation as Gazankulu Mining Company Ltd but went into liquidation in mid-1995 and by then 1, 75 t of gold had been produced from the Klein Letaba, Franke and Birthday mines.

2.3.1 Klein Letaba Mine

This mine, discovered in 1887, was the largest producer in the GGB with an estimated total yield of more than 3 t of gold (Lombaard, 1956). Exploration carried out in the late 1940s revealed high grade gold mineralisation in nine quartz sulphide lenses (arsenopyrite or pyrrhotite dominating), arranged in an en-echelon array, within a shear zone (Lombaard, 1956). Lower grade gold mineralisation was discovered in garnetiferous quartz-amphibolite schists which formed as replacement lodes within the major shear zones. The steep northerly dipping ore zone which was hosted in massive amphibolite, had a length of 240 m and a width of 60 m. During the 1960s the Klein Letaba Mine was the Northern Province's largest gold producer, but it was closed in 1968 following a rock burst. After reopening in 1986, lower grade reefs were exploited down to a depth of 275 m and efforts were underway to locate the extension of the main ore body on the other side of a major fault when the mine went into provisional liquidation in mid-1995.

2.3.2 Louis Moore Mine

This, second most productive gold mine in the GGB, consistently yielded gold grade of between 11 and 31 g/t over a width of 2 m, in the lower portion of the mine (Gan *et al,* 1986). The mine was situated on a greenstone remnant 8 km north of the main belt and it

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operated intermittently from 1905 to 1978, during which time at least 2,5 t of gold were produced (Gain *et al,* 1986). The underground operation ceased when it was found that the ore body was severed by a pegmatitic intrusion at a depth of 183 m. During 1953/4/5 the Klein Letaba Mining Company drilled 55 boreholes revealing six auriferous, en echlon veins.

2.3.3 New Union (Golden Osprey) Mine

This mine is also situated on a detached greenstone remnant north of the main greenstone belt, on the farm Kirsten 212 LT. Mining commenced in 1935 and by 1986 an estimated 2,5 t of gold had been recovered (Gain *et al*. 1986). The mine reopened, and was listed under the name Golden Osprey in 1987. From that time until it went into liquidation in 1991, more than 550 kg of gold were produced. In 1993 the mine became the Northfileds Gold Mine and since then has only produced an average of 20 kg annually from surface material. Gold was hosted in two main mineralised shear zones known as the eastern and western ore bodies. In the eastern ore body, where mineralisation had been drill-indicated to a depth of 360 m, gold occur and in association with pyrrhotite and pyrite, in a fine quartz stockwork associated with deformed lenses of a BIF (Gain *et al*. 1986). In the western ore body gold was hosted in quartzose shears at the contact between amphibole and feldspathic unit. The largest working, the Golden Glory, was situated on the western ore body and was mined down to level 12 during the 1930s. The Manville operations produced small quantities of gold from surface deposits on the northern extension of the Northfields Gold Mine.

2.3.4 Fumani Mine

Operation at this mine, situated on the farm Plange 221 LT near the northern end of the GGB, is believed to have commenced in 1933 and continued until 1961. When the mine reopened as Fumani in 1977, gold ore had been drill-indicated to a depth of 600 m. From 1980 until the mine went into liquidation in 1991, 280 000 t of ore were treated with an average recovery grade of 4 g/t. At least 1, 1 t of gold had been extracted and potential still exists below the 270 m level (Portgieter and De Vililers 1986). Gold was concentrated within the fold closure of a northerly dipping, westerly plunging, isoclinally folded anticline, composed of a meta- sedimentary succession of BIF, micaceous quartzite and amphibolite

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(Portgieter and De Vililers 1986). The BIF unit changes from predominantly silicate facies near surface, to sulphide facies with depth (Portgieter and De Vililers 1986). In the latter, gold is genetically associated with arsenopyrite in garnetiferous, biotite-rich bands.

2.4. Mine Wastes and Disposal Methods

The ore is crushed and mixed with water and lime then cyanide is used to leach the gold and silver. The metals are adsorbed onto activated carbon from which gold and silver are recovered. The waste material from the processing plant, tailings, occur in a semi-fluid state. During mineral processing, the mine produces large quantities of fine-grained rock particles, ranging from sand-sized down to as low as a few microns (US EPA, 1994).

The chemical composition of the tailings depends on the geochemical composition of the ore and the chemicals added during the processing stage. Tailings are the milled materials formed after mineral processing of the ore when the valuable mineral has been extracted. Mine waste can be divided into two, which is historic mine waste and modern mine waste. The historic mine waste comprises waste rock and mine tailings. Most metallic mines produce tailings with high concentration of metals which when oxidized and hydrated can cause acid mine drainage (AMD). The main types of tailings include the Ring-dike, In-Pit, Specially Dug Pit, and variations of the Valley design (US EPA, 1994).

The design depends on a number of factors such as topography, economic factor, and site conditions.

In the past tailings were discharged directly onto land or into streams. Today's open pit mine discharges tailings to areas which are close to the processing plant. There are a number of waste disposal methods which are used and the most common is tailings disposal dams. Other disposal methods include landfill, backfill, riverine, submarine, deep sea tailings placement and thickened tailings.

Tailings dams are designed to perform a number of functions, including treatment and removal of suspended solids by sedimentation, containment of tailings, water recycling, balancing of storm water (US EPA, 1994). Lead (Pb), zinc (Zn), copper (Cu), arsenic (As), cobalt (Co) and cadmium (Cd) are the most common heavy metals in gold tailings dams.

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Heavy metals occur naturally in the environment, however, higher concentrations are increased due to mining, weathering and erosion (Ogola, 2010).

2.5 Tailings Dams

2.5.1 Tailings Dams Design and Constraction

A tailings dam is an earth structure that stores tailings as well as the water used to transport the tailings to the tailings facility. The outer structure of the tailings dam may be constructed by using the tailings itself or earth or rock-fill as in the case of water storage dam (Williamson, 1994).

As with water-retaining dams, each tailings dam is an individual, dependent on its detailed design on the site conditions, the type and the rate of delivery of the tailings, availability of other waste materials from mining or industrial processes and climatic conditions (Penman, 1994). Despite individual details, tailings dams can be classified by the material from which they are constructed, hence the types of tailings dams are (Penman, 1994):

- those built from borrowed earth-fill or earth/rock wastes from a mining or industrial process;
- pure tailings dams using only the tailings themselves for construction; and
- those that use both the tailings and other fills.

In South Africa's mining sector, it is a common practice for the outer retaining structure of the tailings dam to be formed by using a portion of the mine waste itself with the rest of the tailings being deposited inside this outer impoundment. A small earth wall is first placed to define the toe of the dam (Williamson, 1994).

Most of the tailings dams are built in two stages (Messrs *et al,* 1989): a) **Stage I**, the initial starter dam is constructed before the mining operation starts and provides the starting point for the construction of the ultimate tailings dam. The type of starter dam selected usually depends on the method of construction to be used for the remainder of the tailings dam. b) **Stage II** involves the construction of the remainder of the tailings dam. This phase of construction constitutes the major portion of tailings dam construction. It is a continuous

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operation which begins with the start-up of mining operation and continues until the mining operation ceases, or the tailings dam is filled.

Although the function of a tailings dam is to dispose the waste solids from the processing plant, it also invariably stores a portion of water used to transport the tailings slurry to the dam. Excess water is generally removed from the dam by means of a specially designed outlet or decanting ,structure and returned to the plant by pipeline for re-use. Whereas a water storage dam is convectionally built to its final design height and water is then stored behind the dam structure, a tailings dam is different as it only stores the wet tailings solid fraction at the rate it is generated from the mineral processing. The outer retaining structure of the tailings dam is therefore, raised progressively and may take ten or twenty years or even longer to reach full height and storage capacity.

A water storage dam is most often constructed across a river valley so as to intercept and store the water flowing along the river. Tailings dams are therefore most often constracted on flat ground, on a hill slope, or in the upper reaches of a valley where they fill the valley and where stromwater flow would not be affected by the presence of the tailings dam. The general impoundment layout type include (Vick, 1990):

- **Ring dikes**: They are suited on a flat terrain and require a relatively high quality of embackment fill in relation to the storage volume produced and are laid out with regular geometry (Fig. 2.2a).
- **Cross-valley impoundment**: The impoundment is confined by a dam extending from one valley wall to another (Fig. 2.2b). It can be nearly universally applied to almost any natural topographic depression.
- **Sidehill impoundment**: The impoundment is enclosed by embankments on three sides; they are best suited for sidehill slopes of less than 10% grade (Fig. 2.2c). On steeper slopes, fill volumes may become exessive in relation to storage volume.
- **Valley-bottom impoundment**: This represents a compromise between cross-valley and sidehill layout; it is well suited for cases where the drainage catchment area would be too large for cross-valley layout, but hillslope are too steep for practical application of the sidehill option (Fig 2.2d).

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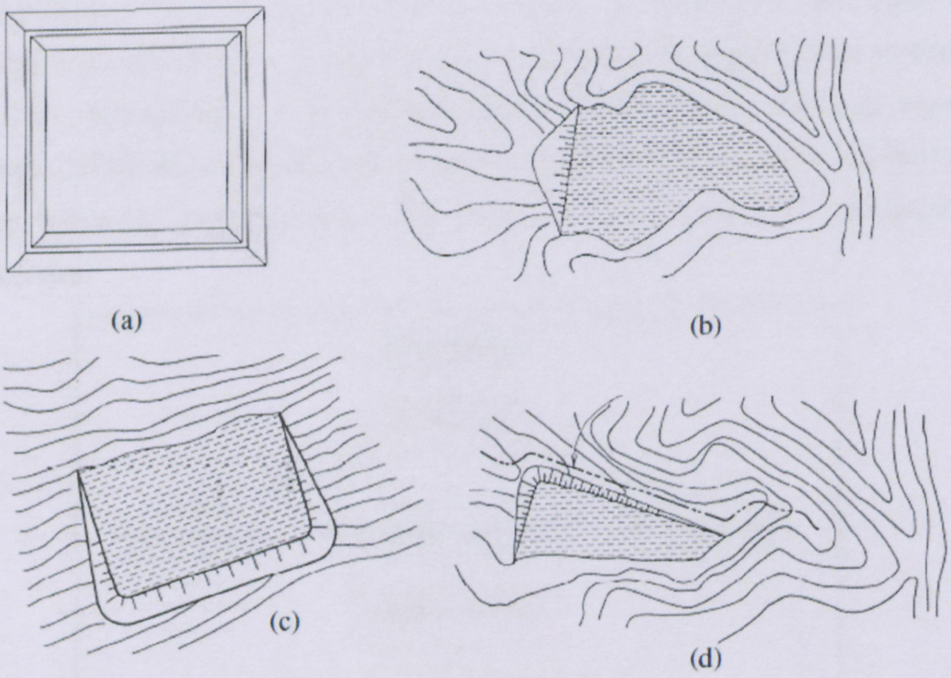


Figure 2.2: Impoundment layout types: (a) Ring dike, (b) Cross-valley impoundment, (c) Sidehill impoundment, and (d) Valley-bottom impoundment (Vick, 1990).

2.5.2 Tailings Production

In order to better understand tailings and tailings dams, some background information is necessary on how tailings are produced. Some fundamental steps in the process of ore extraction that results into tailings production are presented in Figure 2.3 (Vick, 1990):

- **Crushing and grinding:** Crushing is performed in stages with the aim of reducing ore fragments from mine-run size to a size that can be accepted as feed to grinding machine. Grinding further reduces the size of the material produced by crushing.
- **Concentration:** The purpose of concentration is to separate those particles with high mineral value (concentration) from those with lower value (tailings). Methods for concentration vary according to ore type, but three general classes are in use: gravity separation, magnetic separation and froth flotation.
- **Leaching or heating:** Optional processes following concentration may include leaching or heating.

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- **Dewatering:** The final stage in the process is recovering excess water from the tailings in preparation for pumping the tailings slurry to the disposal impoundment. The term dewatering in a mill process context refers not to complete drying of the tailings, but rather to the process of removing some of the water in the tailings-water slurry following concentration. The most common means of dewatering is by thickeners.

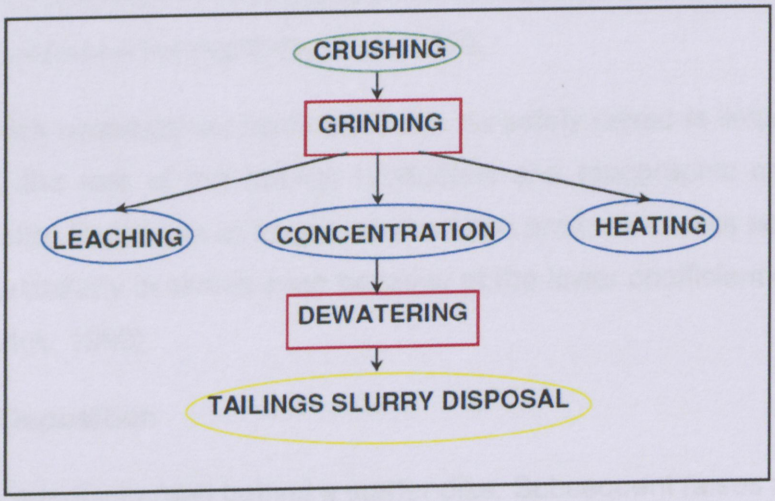


Figure 2.3: Procedures in tailings production (Vick, 1990).

2.5.3 Tailings Transportation and Deposition

Transportation

The tailings are collected after the dewatering process and are almost universally transported in slurry form to the tailings impoundment. The transport of tailings slurry is sometimes by gravity flow through open launders, but more commonly through pipes, either with or without pumping depending on relative elevation (Vick, 1990).

Deposition

Deposition adopted to form the outer tailings impounding wall generally consists of one of the following types (Vick, 1990):

- Upstream deposition
- Downstream deposition

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- Centreline deposition

Upstream Deposition

The major advantages of the upstream method are cost and simplicity. Only a minimal volume of mechanically placed fill are necessary for construction of the perimeter dikes and large embankment height can be attained at very low cost. Factors that constrain the application of the upstream method include phreatic surface control, water storage capacity and seismic liquefaction susceptibility (Vick, 1990).

The rate at which upstream embankments can be safely raised is limited. Raising rate is determined by the rate of mill tailings production and topographic configuration of the impoundment site. Rapid rate of height increase can produce excess pore pressure within the deposit, particularly in slimes zone because of the lower coefficient of consolidation of this material (Vick, 1990).

Downstream Deposition

Initially tailings are discharged behind a starter dike. Subsequent raises are constructed by placing embankment fill on the downstream slope of the previous raise. This method is amenable to the incorporation of structural measures within the embankment for positive control of the phreatic surface. Downstream raising methods are well suited to conditions where significant storage of water along with tailings is necessary. Because the phreatic surface can be maintained at low levels within the embankment and because the entire body of the fill can be compacted, downstream raising methods are liquefaction resistant and can be used in areas of high seismicity. Unlike upstream embankments, raising rates are essentially unrestricted because the downstream raises are structurally independent of the spigotted tailings deposit (Vick, 1990).

Downstream raising methods require careful advance planning because the toe of the dam progresses outward as its height increases sufficient space must be left during layout of the starter dike to prevent encroachment of the dam toe on property lines, roads, utilities etc.

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The major disadvantage of downstream raising method is the comparatively large volume of embankment fillment and the corresponding high cost. The volume of fill required for each successive downstream raise often increases exponentially as the embankment increases in height.

Centreline Deposition

Centreline deposition is a compromise between upstream and downstream deposition in many respect. As a result, it shares to a degree the respective advantages of two methods, while mitigating their disadvantages. Initially, a starter dike is constructed, and tailings are peripherally spigotted from the dike crest to form the beach. Subsequent raises are constructed by placing fill onto the beach and onto the downstream slope of the previous raise (Vick, 1990).

The centreline of the raises are coincident as the embankment progresses upward, giving rise to the method's name. Because internal drainage zones can be provided with the embankment, control of the phreatic surface is not as sensitive to the location of ponded water as it is for upstream deposition. Unlike downstream embankment, the centreline method cannot be used for permanent storage of large depths of water (Vick, 1990).

The water can be allowed to rise temporarily during floods, however, without adversely affecting the stability of the structure provided that proper internal impervious and or drainage zones are incorporated. The overall raising rate is not generally restricted by consideration related to pore pressure dissipation. Because the main body of the embankment fill can be compacted and saturated levels controlled by internal drainage, the centreline method has generally good seismic resistance (Vick, 1990).

2.5.4 Environmental Problems Associated With Mine Tailings Dams

Mining on its own is a temporal project because the mineral deposits are depleteable, but the environmental impacts caused by mining such as tailings, open pits and undermined workings last for a long time. The environmental impacts of mining are seen by comparing the state of the environment after mining and rehabilitation has been done with the condition that existed before mining (Wilson and Anhaeusser, 1998). Heavy metals are individual

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and metal compounds that can impact negatively on human health (Sabine and Wendy, 2009).

Historically, the location of mine dams was chosen considering the economic factor only and this made the old tailings dams be a threat to the environment and human health. However, the location of mine dams nowadays is controlled by law which looks into a variety of factors such as environmental impacts, visual impacts and permanent change of the land-use. Gold mining in the Giyani greenstone belt, for example, took place when there were no environmental laws to deal with impacts of mining. Tailings have the potential to pollute the environment as a result of acid mine drainage and wind erosion. The acid produced dissolves salts and transports the heavy metals from mine workings or tailings to pollute the nearby environment and water bodies (Wilson and Anhaeusser, 1998). Both surface and groundwater can be affected by the leachate from the tailings dams, this includes the seepage of acid mine drainage into groundwater polluting the water-table which is difficult and very expensive to treat.

Tailings contain high concentrations of metals like Pb, Zn, Cu, As, Cd, Cr, Co, Ag, Ni, Mn and Cr. These metals are non-degradable, thus they stay for a long time in the environment and can cause a serious health problem to human and the environment. The Gold mine tailings are a threat to the environment because the tailings have the ability to produce acid when oxidized depending on the mineralogical properties of the ore or due to the presence of cyanide used to recover gold and the structural stability of the tailings facility. These impacts can remain for many years after the mine has closed (Sabine, 2009).

As, Zn, Cu and Cr which can be dispersed from tailings have got the potential for poisoning through dust inhalation and food chain if their concentrations are above the recommended standards. When acid mine drainage has been generated due to the oxidation of sulphides-bearing minerals, especially pyrite and the addition of water which may contain bacteria that act to catalyze the oxidation reaction, this could affect the environment for hundreds of years (Sabine and Wendy, 2009).

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Tailings dams are permanent fixtures which restrict future land-use. The poor design, construction, and maintenance of the dams together with the lack of treatment facilities are contributing factors to the effects of the dam to the environment.

2.6 Reprocessing of Gold Tailings Dams

Hydraulic mining monitors used today were first developed back in the early sixties by English China Clays, Cornwall, England. The technology has been used on soft rock applications and on old tailings dams to either move or reprocess the stored material. The monitor uses high pressure water to erode the tailings in sections, washing the material downstream which is collected in a sump (*http://www.e-goldprospecting.com, 2007)*.

Tailings dams are generally segregated by the coarseness of the material (coarse fraction near the spigot) and if a particular area of a dam is too coarse for pumping then blending is required (Adamson, 1972). Once the required density is obtained in the sump and screening has prevented large objects from passing, the slurry is then pumped to thickeners and the underflow is reprocessed in the plant. The tailings are then stored in a tailings impoundment (Adamson, 1972). A typical flow sheet for the reprocessing of tailings is shown below (Fig. 2.4).

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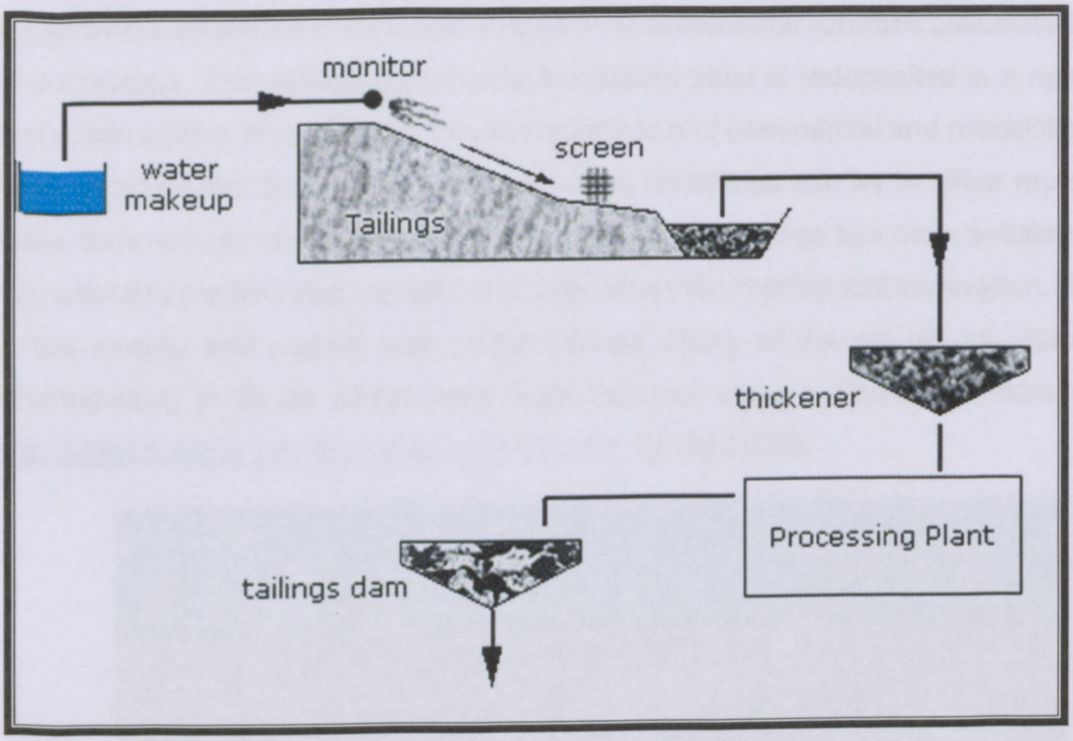


Figure 2.4: Simplified diagram of the hydraulic reprocessing of tailings (Adamson, 1972).

2.7 Reprocessing of Gold Tailings Dams within the Witwatersrand Basin

South Africa is a mineral-rich country with metals such as gold, copper, and platinum group metals being exploited to a significant extent in the country's mining history. Mining generates large volumes of tailings, with consequent disposal and environmental problems. By far most gold that has been mined in South Africa (98%) has come from the Witwatersrand goldfields (Messner, 1991). The gold mines in this area are situated around an ancient sea (over 2700 million years old) where rivers deposited sediments in the form of sand and gravel that became the conglomerate containing gold (Messner, 1991). The extensive exploitation of the gold resources has led to numerous mine tailings dams scattered within the Witwatersrand Basin. As long as mining contributes significantly to the economic development of South Africa, generation of these tailings is inevitable.

The technical and economic feasibility of re-processing hundreds of millions of tons of tailings in a number of historic gold tailings dams in the Witwatersrand area of South Africa is being investigated. The objective is to recover residual gold from the tailings, as well as

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to oxidize the sulphides to eliminate long term environmental liabilities associated with acid mine drainage. The tailings from the re-processing plant is redeposited in a new area far from urban centre, thus allowing for the construction of commercial and residential estates. It can be noted that the use of hydraulic mining for tailings can be to either reprocess the waste, thus recover valuable metals and to relocate the tailings to a more suitable site. This is an effective method and can take a shorter time than mechanical excavation, depending on the density and particle size of the tailings. Many of the old tailings dams around Johannesburg in South Africa have been remined using hydraulic monitors (Fig. 2.5) (http://www.tailings.info/technical/hydraulic.htm, 20-06-2016).



Figure 2.5: Hydraulic reprocessing of gold (http://www.tailings.info/technical/hydraulic. htm, 20-06-2016).

2.8 Gold Tailings as Construction Material

South Africa is currently faced with challenges resulting from AMD and the government and mining companies are under pressure to find viable solutions to this problem. This, coupled with the increasing landfill costs, and stricter implementation and enforcement of environmental legislation, has caused the scientific community to focus on finding innovative methods of utilizing mine tailings. Even though some applications of the generated tailings have been exploited, such as in the building of slimes dams and backfill in underground mines, these uses do not take up more than a fraction of the total amount of tailings in the Witwatersrand region. There is therefore a significant need to develop other

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long-term, commercially viable uses for mine tailings in order to minimize the disposal costs and the impact on the environment.

According to Statistics South Africa (2013), South Africa has a population of about 52.98 million. This population is growing, and this consequently results in an increasing demand for housing, which places severe stress on the natural resources used for construction materials. Conventional bricks are produced from clay fired in high-temperature kilns or from Ordinary Portland Cement (OPC) concrete. Clay, the common material used for brickmaking, is usually mined in quarries. Quarrying operations are energy-intensive, adversely affect the landscape, and generate a high level of waste (Zhang, 2013; Bennet *et al.*, 2013). Furthermore, in many areas of the world, there is already a shortage of natural resource material for the production of the conventional bricks (Zhang, 2013). To conserve the clay resources and the environment, some countries such as China have started to limit the use of bricks made from clay (Zhang, 2013). Thus the depletion of these natural resources has created a need for alternative sources of construction materials in order to sustain development.

Extensive research has been conducted on the production of bricks using waste material (Zhang 2013; Saeed and Zhang, 2012). These waste materials include mine waste, construction and demolition waste, wood sawdust, cotton waste, limestone powder, paper production residues, petroleum effluent treatment plant sludge, kraft pulp production residue, cigarette butts, waste tea, rice husk ash, crumb rubber, cement kiln dust, and coal fly ash (Zhang 2013; Bennet *et al.*, 2013; Saeed and Zhang, 2012). The mining and mineral processing waste includes overburden, waste rock, mine tailings, slags, granulated blast furnace slag, mine water, water treatment sludge, and gaseous waste (Zhang, 2013; Saeed and Zhang, 2012; Koumal, 1994; Dean *et al.*, 1968; Bennet *et al.*, 2013). Gold tailings dams at Kolar Gold Fields, Karnataka, and South West India are creating environmental problems. One of the solutions to these problems is to put the gold tailings for some useful purpose, for example, in bricks making. In a study conducted by Roy *et al.* (2005), some samples of the gold tailings and the additives were analysed for particle size distribution, Atterberg limits and specific gravity; from that it was observed that the plasticity index of

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the tailings turned out to be zero, owning to the fact that the additives used had no plasticity or some binding properties in it. Portland cement, black cotton soil and red soil were selected as additives. Each of the additives was mixed separately with the gold tailings in different proportions by weight and a large number of bricks were prepared using metallic moulds. The bricks were termed as cement-tailings bricks or soil-tailings bricks, depending on the additives used. The cement-tailings bricks were cured for different periods and their corresponding compressive strengths were determined. The bricks with 20% of cement and 14 days of curing were found to be suitable for construction. The soil-tailings bricks were sun-dried and then fired in a furnace at different temperatures. Hence cost analysis revealed that cement-tailings bricks would be uneconomical whereas the soil-tailings bricks would be quite economical.

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CHAPTER 3: MATERIALS AND METHODS

This chapter presents the methods and procedures used during the course of the study (Fig. 3.1).

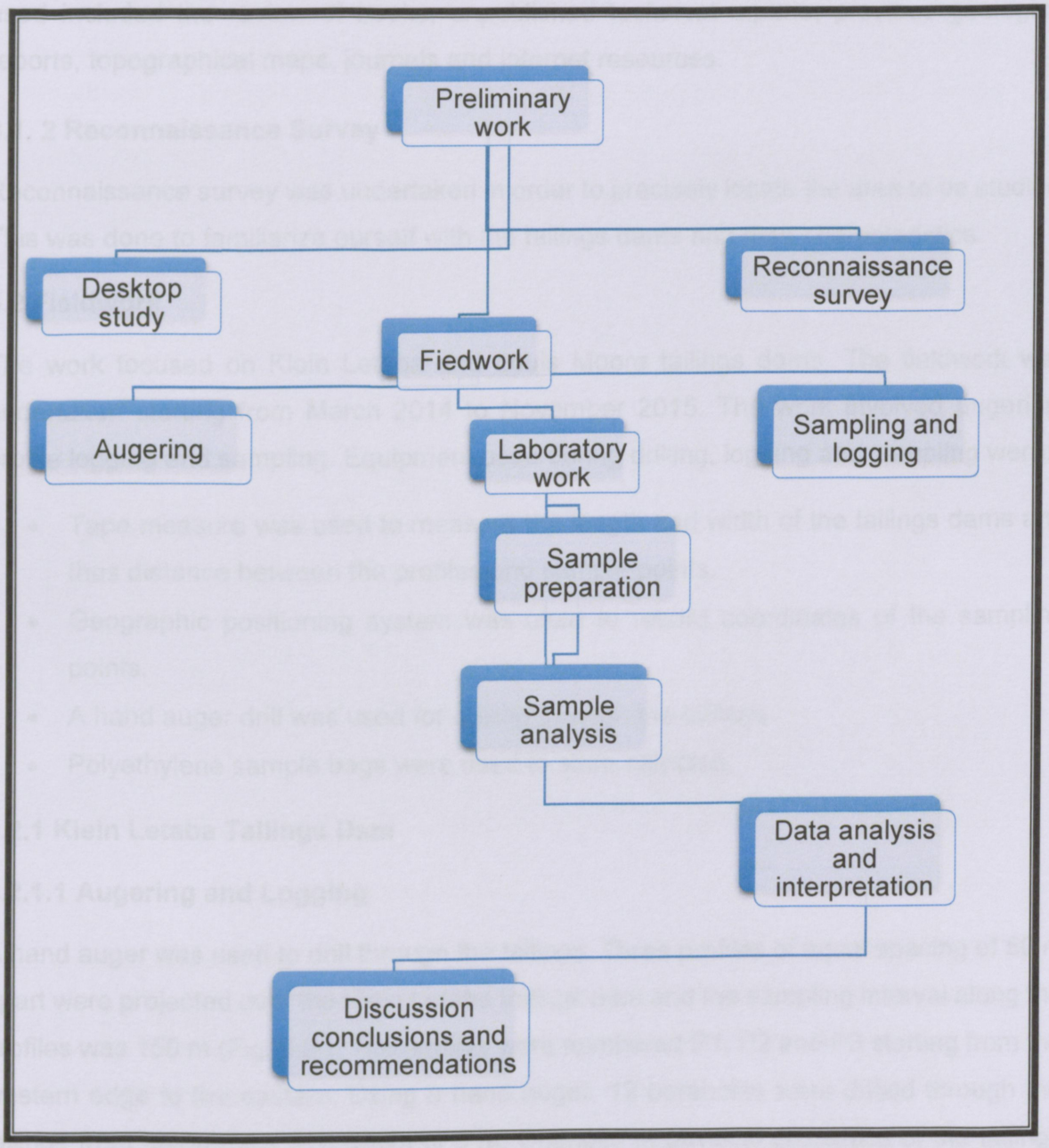


Figure 3.1: Flow chart indicating the sequence of work undertaken.

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3.1 Preliminary Work

3.1. 1 Desktop Study

This was conducted prior to the fieldwork. The various sources of information which were used included the review of books, unpublished technical reports, previous geological reports, topographical maps, journals and internet resources.

3.1. 2 Reconnaissance Survey

Reconnaissance survey was undertaken in order to precisely locate the area to be studied. This was done to familiarize ourself with the tailings dams and their characteristics.

3.2 Fieldwork

The work focused on Klein Letaba and Louis Moore tailings dams. The fieldwork was undertaken starting from March 2014 to November 2015. The work involved augering, profile logging and sampling. Equipment used during drilling, logging and sampling were:

- Tape measure was used to measure the length and width of the tailings dams and thus distance between the profiles and sample points.
- Geographic positioning system was used to record coordinates of the sampling points.
- A hand auger drill was used for drilling through the tailings.
- Polyethylene sample bags were used to store samples.

3.2.1 Klein Letaba Tailings Dam

3.2.1.1 Augering and Logging

A hand auger was used to drill through the tailings. Three profiles of equal spacing of 50 m apart were projected over the Klein Letaba tailings dam and the sampling interval along the profiles was 150 m (Fig. 3.2.). The profiles were numbered P1, P2 and P3 starting from the western edge to the eastern. Using a hand auger, 12 boreholes were drilled through the tailings from the surface to a depth of 8 m. Changes in physical properties of the tailings such as colour, moisture content and hardness with depth were carefully recorded in a logbook (Appendix A). This information was used to correlate logs along the profiles.

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3.2.1.2 Tailings Sampling

Tailings samplings was conducted simultaneously as augering and logging were taking place. Each borehole was sampled at an interval of 1 m interval up to 8 meters from which 95 samples were collected, each weighing about 5 kg. The samples were put in a clean and properly marked polyethylene sample bags. The samples were transported to the Department of Mining and Environmental Geology Laboratory, University of Venda.



Figure 3.2: Orientation of three profiles (P1, P2 and P3) over the Klein Letaba tailings dam.

3.2.2 Louis Moore Tailings Dam

3.2.2.1 Augering and Logging

A hand auger was used to drill through the tailings. The tailings dam was divided into three profiles, the first profile had 5 drill holes, the second profile had 4 drill holes and the third profile with 2 drill holes (Fig. 3.3). This was because some of the sections of the dam had been strongly weathered resulting into deep gulleys especially in the north-western part of the dam. Sampling points were chosen at places not affected by erosion in order to obtain

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fresh samples. Each hole was drilled up to a depth of 5 meters. Changes in the physical properties of the tailings such as colour, moisture content and hardness with depth were carefully recorded in a logbook (Appendix B). This information was used to correlate logs along the profiles.

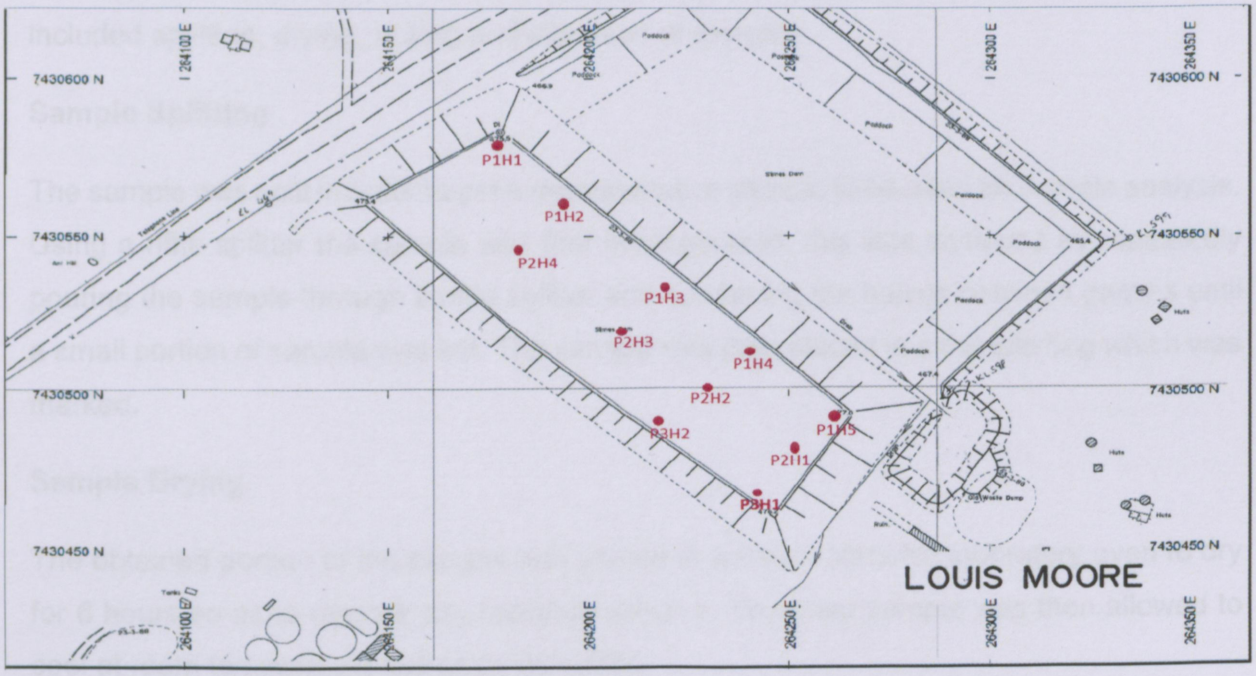


Figure 3.3: Location of sample points over Louis Moore tailings dam.

3.2.2.2 Tailings Sampling

A total of 55 samples were collected from the 11 drill holes, the 11 sampling points used for logging as explained above were used for sampling the tailings dam. Each borehole was sampled at an interval of 1m up to 5m or 7m depending on the thickness of the tailings dam. A total of 55 samples were collected, each weighing about 5 kg. Samples were put in a clean and properly marked polyethylene sample bags. The samples were transported to the Department of Mining and Environmental Geology Laboratory, University of Venda for laboratory analysis. Other samples were sent to Set Point Laboratory for Gold analysis using Fire Assey method and other to soillab Pretoria for Atterberg limits.

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3.3 Laboratory Work

3.3.1 Sample Preparation

A total of 150 samples were prepared at the Department of Mining and Environmental Geology laboratory, School of Environmental Sciences, University of Venda. Preparation included splitting, drying, milling and digestion of samples.

Sample Splitting

The sample was split in order to get a representative sample to be used for sample analysis. Using a riffle splitter the sample was first homogenized, this was achieved by repeatedly pouring the sample through a riffle splitter and combining the halves between passes until a small portion of sample was left. The sample was then placed in a sample bag which was marked.

Sample Drying

The obtained portion of the sample was placed in a Bench Vacutec laboratory oven to dry for 6 hours so as to remove any moisture within it. The dried sample was then allowed to cool at room temperature before being milled.

Samples Milling

The oven dried sample was transferred into a milling pot and placed in a Retsch Model RS 200 milling machine wherein it was milled for 5 minutes at 700 rpm to reduce the particle size. Quartz was used to clean the milling pot after every sample was milled. The milled samples were then transferred into sample bags which were marked, ready for weighing and digestion.

Weighing and Digestion of samples

Samples were weighed using an analytical balance (Redwag Model AS220/C/2). The weighed 5 g of each sample was placed in a decomposition beaker for acid digestion upon adding 2-5 ml of deionized water to moisten the sample. Two grams of certified reference material from MINTEK were also weighed. Aqua-regia composed of 15 ml nitric acid and 45 ml hydrochloric acid was added to the sample in a decomposition beaker. The beaker was then placed on the hot plate for at least 90-120 minutes for decomposition. The

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samples were then removed from the hot plate and allowed to cool to room temperature. The samples were then transferred to a 100 ml volumetric flask and accurately made up to the mark with deionized water before shaking vigorously and allowed to settle for 2 hours ready for analysis using, the Atomic Absorption Spectrometry.

3.3.2 Sample Analysis

Atomic Absorption Spectrometry

The flame AAS technique is primarily used for chemical analysis of metallic elements in a variety of materials. The quantitative analysis of these elements can be accomplished by using calibration curves. A calibration curve is used to determine the unknown concentration of an element in a solution. The instrument is calibrated using several solutions of known concentrations. A calibration curve is produced which is continually rescaled as more concentrated solutions are used. The more concentrated solutions absorb more radiation up to a certain absorbance. The calibration curve shows the concentration against the amount of radiation absorbed. The sample solution is fed into the instrument and the unknown concentration of the element is then displayed on the calibration curve. Some of the advantages of this technique include (Haswell, 1991).

- fast analysis (10-15 second per sample per element);
- very good precision (repeatability);
- no or moderate interferences that can be easily corrected;
- easy automation of the measurement; and
- relatively low cost.

A total of 150 samples, 95 Klein Letaba and 55 Louis Moore were analysed at the School of Environmental Sciences, Department of Mining and Environmental Geology laboratory using the atomic absorption spectrometry. The analysis was conducted for Pb, Zn, Cu, As, Cd, Cr, Co, Ag, Ni, Cr and Mn using an AASPerkin Elmer AAnalyst 400, whilst Au samples was analysed at Set Point Laboratory using the fire assaying technique 60 Klein Letaba and 20 Louis Moore (Appendix C).

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Fire Assaying

Fire assaying is the quantitative determination in which a metal or metals are separated from impurities by fusion processes and weighed in order to determine the amount present in the original sample (http://www.mine-engineer.com/mining/assay2.htm July 2015).

Metals recovered by fire assaying are the metals that emerge from the fusion of the material in an assay furnace, for gold, usually at temperatures of about 2,000°C. The Fire Assay Method is centuries old, but it is still one of the most reliable method for performing assays (to determine the metal content of an ore) of ores that contain precicus (noble) metals, for example, gold, silver and platinum. The ore is scientifically sampled using a statistically accurate method fitting the desired accuracy, it is then prepared by crushing, splitting and pulverizing. This is a process referred to as sample preparation. Since the original sample may be a hundred or more kilograms, a scientific method of obtaining a representative sample for testing of usually 30 grams is used, and it includes reducing the sample to a predetermined particle size and then splitting the sample using a statistically accurate method, to obtain the 30 grams representative sample for fire assaying (http://www.mine-engineer.com/mining/assay2.htm July 21, 2015).

3.3.3 Atterberg Limit Tests of Tailings

The Atterberg limit is a basic measure of the critical water contents of a fine-grained soil, such as its shrinkage, plastic and liquid limits. As a dry, clayey soil takes on increasing amounts of water, it undergoes dramatic and distinct changes in behaviour and consistency. Depending on the water content of the soil, it may appear in four states: solid, semi-solid, plastic and liquid. In each state, the consistency and behaviour of a soil is different and consequently so are its engineering properties. Thus, the boundary between each state can be defined based on a change in the soil's behaviour. The Atterberg limits can be used to distinguish silt and clay, and it can distinguish between different types of silts and clays. These limits were created by Albert Atterberg, a Swedish agriculturist 1913. They were later refined by Arthur Casagrande, 1930s. The Atterberg tests include sieve analysis, hydrometer, liquid and plastic limit tests.

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Sieve Analysis and Hydrometer Test

The sieve analysis and hydrometer test were conducted in order to determine the particle size of the Klein Letaba and Louise Moore tailings.

Sieve Analysis

This test is performed to determine the percentage of different grain sizes contained within a soil (Terzaghi *et al.,* 1996). It is performed to determine the distribution of the coarser, larger-sized and fine particles. The distribution of different grain sizes affects the engineering properties of soil. Grain size analysis provides the grain size distribution, and it is required in classifying the soil.

Sieve analysis was done to determine the grain size distribution of the tailings. A set of sieves of various sizes were cleaned, weighed and individual sieve mass recorded. The sieves used were; 63.0 mm, 53.0 mm, 37.5 mm, 26.5 mm, 19,0 mm, 13.2 mm, 4.75 mm, 2.00 mm, 0.425 mm, and 0.075 mm in their decreasing order (Tables 3.1 and 3.2). A pan was placed at the bottom of the stack of the sieves to collect the finest fraction of the tailings.

Three samples of Klein Letaba and 3 samples of Louise Moore tailings dams were weighed each being 400 g. The retained fraction of the sample in each sieve was oven dried and allowed to cool to room temperature before being weighed to determine the mass. The outcome of the experiment is presented in Table 3.1 and 3.2.

Hydrometer Test

The hydrometer test is used to determine the distribution of the finer particles (Krishna, 2002). In this test, the tailings samples were treated with sodium hexametaphophate to prevent silt and clay particles from binding. The apparatus used for hydrometer test were Hydrometer ASTM 151H, Sedimentation Cylinder (1L graduated cylinder), shaking bottle, sample mixer with dispersing cup, balance sensitive to 0.01 g, a thermometer, rubber stopper and stopwatch.

A 50 g of oven dried sample was weighed and transferred into a 500 ml beaker, 250 ml of distilled water was added into the beaker to form tailings sample solution. A 5 g of calgon

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dispersing agent sodium hexametaphosphate $[Na_6(PO_3)6]$ was added to a separate beaker and 50 ml of distilled water was added. The mixture was then stirred until the dispersing agent was dissolved. The solution was then transferred into a shaking bottle. The bottle was shaken for 15 minutes to form a suspension.

The suspension was transferred to a sedimentation cylinder and water was added to 1000 ml mark. The temperature of the suspension was immediately taken. To mix suspension, the cylinder was inverted upside down, the mouth being covered with a rubber stopper. Immediately after mixing, the cylinder was placed upright on a table. The timer was started at the same time the hydrometer was inserted into the suspension.

The reading of the hydrometer was taken after 30 seconds (this was chosen so that sand particle; i.e.≥0.05 mm would have settled at the bottom of the cylinder) (Tables 3.1 and 3.2).

The hydrometer was calibrated at 20°C and the hydrometer readings were corrected for the variation in temperature. This is because of the viscosity of water and, to a lesser extent, the density of water change with changes in temperature. The composite correction for temperature was made by taking hydrometer readings in distilled water at different temperatures.

Table. 3.1: Sieve and Hydrometer Analysis for Klein Letaba Tailings

Sieve analysis	Passing percentage
63.0 mm	100
53.0 mm	100
37.5 mm	100
26.5 mm	100
19.0 mm	100
13.2 mm	100
4.75 mm	99
2.00 mm	99
0.425 mm	98
0.075 mm	79
Hydrometer analysis	Passing percentage

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0.040 mm	63
0.027 mm	49
0.013 mm	34
0.005 mm	0
0.002 mm	0

Table. 3.2: Sieve and Hydrometer Analysis for Louis Moore Tailings

Screen analysis	Passing percentage
63.0 mm	100
53.0 mm	100
37.5 mm	100
26.5 mm	100
19.0 mm	100
13.2 mm	99
4.75 mm	99
2.00 mm	99
0.425 mm	98
0.075 mm	80
Hydrometer analysis (% passing)	
0.040 mm	63
0.027 mm	49
0.013 mm	32
0.005 mm	0
0.002 mm	0

Liquid limit, Plastic Limit and Plasticity Index of Soils

These test are is used as an integral part of several engineering classification systems to characterize the fine-grained fractions of soils and to specify the fine-grained fraction of construction materials. The liquid limit, plastic limit and plasticity index of soils are also used extensively, either individually or together, with other soil properties to correlate with engineering behaviour such as compressibility, permeability, compactibility, shrink-swell and shear strength (Sharma *et al.,* 2003).

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Liquid limit Test of Klein Letaba and Louis Moore Tailings

The liquid limit property of the fine-grained fraction of the samples was tested at Soillab, Pretoria, using a liquid limit device (Casagrande), grooving tool, water bottle, mixing dishes, spatula, and a drying oven. The procedure of testing the liquid limit of the samples begins with calibration and cleaning up of the Casagrande equipment. The calibration involved setting of the drop of the cup to a consistency drop height of 10 mm. Water was added to the dry sample specimen in the dish to increase its moisture level.

The moist sample was then placed on the Casagrande cup and smoothened to maximum depth of 8 mm. A groove was cut at the centre line of the sample in the cup and the device was cranked at 2 revolutions per second until the two halves of the sample came into clear contact of about 13 mm long at the bottom of the groove (Fig. 3.4).

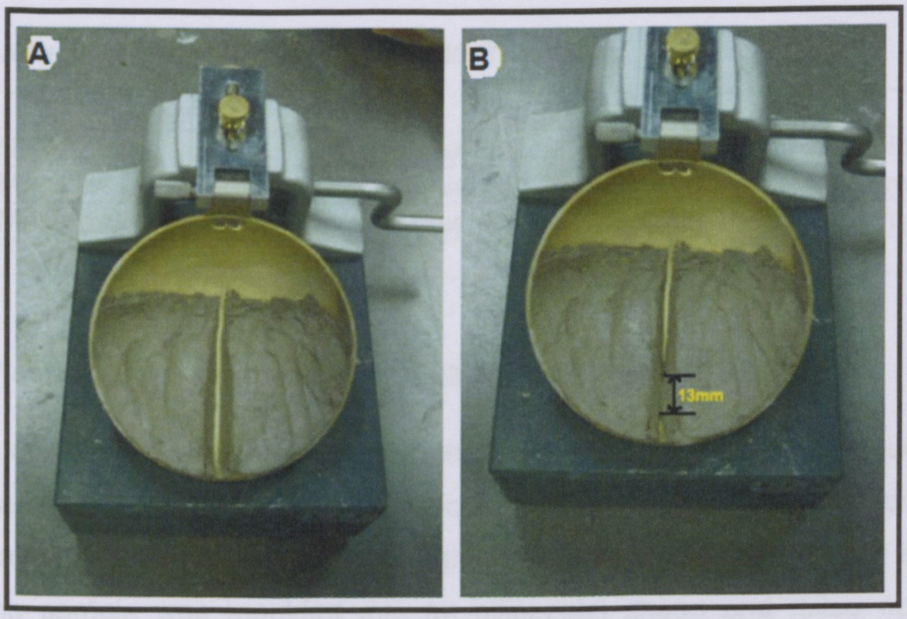


Figure 3.4: Liquid limit test using Casagrande, showing a groove at the centreline of the samples; a) Opened centre line; b) Closed centre line.

The number of blows (N) that caused the closure were recorded. The sample in the pat was collected, weighed together with the can, labelled and dried in the oven at 110°C for more than 6 hours. The weight of the dried sample was then determined. The test was repeated 3 times at different moisture content, producing successively lower numbers of

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blows to close the groove, resulting in blows of 1 to 8, 1 to 4 and 1 to 4 for Klein Letaba tailings and 1 to 8, 1 to 4 and 1 to 3 for Louis Moore (Tables 3.3 and 3.4).

The number of blows in each trail were plotted against the water content and the best-fit straight line through the plotted points was constructed for which the liquid limit (LL) was determined as the water content at 25 blows to close the groove over a distance of 13 mm (Casagrande, 1945).

Table 3.3: Klein Letaba Liquid Limit Test

Trial 1		Trial 2		Trial 3	
No of blows	8	No of blows	4	No of blows	4
Mass wet	41, 26 g	Mass wet	44, 63 g	Mass wet	31, 47 g
Mass dry	32, 45 g	Mass dry	34, 88 g	Mass dry	24, 39 g

Table 3.4: Louis Moore Liquid Limit Test

Trial 1		Trial 2		Trial 3	
No of blows	8	No of blows	4	No of blows	3
Mass wet	52, 34 g	Mass wet	45, 89 g	Mass wet	41, 89 g
Mass dry	41, 08 g	Mass dry	35, 92 g	Mass dry	32, 33 g

Plastic Limit Test of Klein Letaba and Louis Moore Tailings

A dry sample of about 20 g was mixed with water using a plastic squeeze bottle to make a paste. An ellipsoidal shaped mass of tailings was prepared. The specimen was rolled, using palms and fingers on the ground glass plate. When the diameter of the thread was 3 mm, the sample was moulded into an ellipsoidal shape and rolled out. This process was continued until the crumbling of the sample prevent the formation of the thread of 3 mm in diameter. This means that the crumbling occurred when the thread had a diameter slightly greater than 3 mm, but this was regarded as a satisfactory end-point provided that just before that, the sample had been rolled into 3 mm in diameter. The broken threads were placed into weighing trays and weighed before drying in the oven and weighed again after drying. The procedure was done three times in order to have three trials (Tables 3.5 and 3. 6).

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Table 3.5: Plastic Limit of Klein Letaba

Mass of wet tailings	10, 07 g
Mass of dry tailings	8, 17 g

Table 3.6: Plastic Limit of Louis Moore

Mass of wet tailings	6, 27 g
Mass of dry tailings	5,10 g

3.3.4 Production of Bricks

Soil were excavated and then laid on leveled ground. Then the soil were cleaned of impurities such as vegetation matter, stones or pebbles etc. After removing these it was exposed to weather for few months. This is called the process of weathering. After completion of weathering process the soil were blended with tailings and coal waste to prepare good brick. Then the mixed soil was tempered by being thoroughly broken up, watered and kneaded. The temperings were usually done in pug mill

The mixture of clay coal waste and tailings were forced in the mould in such a way that it filled all the corners of the mould. Extras mixture were removed either by wooden strike or frame with wire.

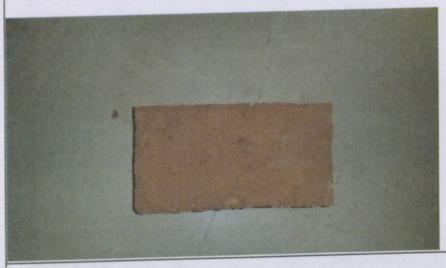
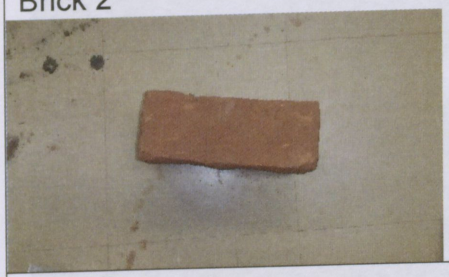
Drying was done by placing the bricks in sheds with open sides so as to ensure free circulation of air and protection from bad weather and rains. The bricks (Table 3.3) were allowed to dry till they are left with 5 to 7 percent moisture content. The drying period usually varied from 7 to 14 days. The moulded bricks were dried because of the following reasons.

- If damp bricks or green bricks are directly taken to burning then, they are likely to be cracked and distorted.
- To remove maximum moisture from the brick so as to save time and fuel during burning.
- To increase the strength of raw bricks so that they can be handled and stacked in greater heights in the kiln for burning without damage.

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Clamps method were used, one batch of green bricks was heaped along with firewood and sealed with clay. It was then fired slowly to intense heat for one week or more.

Table 3.3: Klein Letaba and Louis Moore tailings dams bricks

Klein Letaba (tailings)	Bricks composition (%)	Colour	Comments
Brick 1 	10% coal waste 10% tailings 80% clay	Red brick	Very strong brick
Brick 2 	20% coal waste 20% tailings 60% clay	Red brick	Strong brick
Brick 3 	18% coal waste 22% tailings 60% clay	Brownish brick	Moderately strong brick
Brick 4 	30% coal waste 30% tailings 40% clay	Brownish brick	Poor brick

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Brick 5 	100% tailings	Brownish brick,	Very poor brick
Louis Moore (tailings dam)	**Bricks percentage**	**Colour**	**Comment**
Brick 1 	10% coal waste 10% tailings 80% clay	Red brick	Very strong brick
Brick 2 	20% coal waste 20% tailings 60% clay	Red brick	Strong brick
Brick 3 	18% coal waste 22% tailings 60% clay	Brownish brick	Moderately strong brick
Brick 4 	30% coal waste 30% tailings 40% clay	Brownish brick	Poor brick

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Brick 5 	100% tailings	Brownish brick	Very poor brick

3.3.5. Major and Minor Oxide Analysis of Klein Letaba and Louis Moore Tailings

The main aim of conducting major oxide geochemical analysis of tailings was to establish the amount of major oxides of the material.

Sample Preparation

Samples were milled in a tungsten-carbide milling pot to achieve particle sizes < 75 micon, dried at 100°C and roasted at 1000°C to determine loss of ignition (LOI). Sample weighing 1 g were mixed with 6 g Lithumteraborate flux and fused at 1050°C to make a stable fused glass bead.

Sample Analysis

A total of 8 samples were analysed at the Geology Department, Faculty of Natural and Agricultural Sciences, University of Pretoria. The analysis was conducted for SiO_2, Al_2O_3, Fe_2O_3, MgO, CaO, K_2O, P_2O_5, Cr_2O_3, NiO, V_2O_5, TiO_2, MnO, ZrO_2, CuO, Na_2O using XRF Spectrometry method (Tables 3.4 and 3.5).

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Table 3. 4 Major and Minor Oxides of Klein Letaba tailings

Oxides / Samples	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	K_2O	P_2O_5	Cr_2O_3	NiO	V_2O_5	TiO_2	MnO	ZrO_2	CuO	Na_2O	LOI	Total
S1	52.77	6.92	21.15	6.39	4.16	0.57	0.12	0.12	0.07	0.02	0.42	0.17	0.01	0.01	0.88	7.02	100.8
S2	42.63	4.03	10.19	17.83	10.1	0.83	0.04	0.18	0.08	0.01	0.13	0.25	0.01	0	0.43	12.64	99.39
S3	56.99	5.91	10.54	9.91	6.91	0.7	0.08	0.12	0.06	0.02	0.31	0.17	0.01	0	0.84	6.86	99.45
S4	57.5	7.37	13.09	8.15	5.3	0.81	0.1	0.11	0.05	0.02	0.44	0.15	0.01	0.01	1.01	5.01	99.23
Average	52.47	6.06	13.74	10.57	6.62	0.73	0.09	0.13	0.07	0.02	0.33	0.19	0.01	0.01	0.79	7.89	99.72

Table 3.5 Major and Minor Oxides of Louis Moore tailings

Oxides / Samples	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	K_2O	P_2O_5	Cr_2O_3	NiO	V_2O_5	MnO	TiO_2	ZrO_2	CuO	Na_2O	LOI	Total
S1	54.32	5.21	9.97	11.62	7.9	0.65	0.06	0.11	0.05	0.02	0.18	0.23	0.01	0	0.71	8.35	99.39
S2	51.2	4.14	8.02	14.48	9.29	0.62	0.04	0.014	0.06	0.01	0.2	0.13	0.01	0	0.56	9.93	99.42
S3	60.1	6.78	13.48	6.49	5.31	0.77	0.1	0.07	0.03	0.02	0.15	0.44	0.01	0.01	1.02	4.77	100.94
S4	51.38	6.91	20.68	7.35	4.09	0.6	0.12	0.1	0.06	0.02	0.19	0.41	0.01	0.01	0.85	8.18	100.94
Average	54.25	5.76	13.03	9.96	6.65	0.66	0.08	0.07	0.05	0.02	0.18	0.30	0.01	0.01	0.79	7.80	100.17

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CHAPTER 4: DATA ANALYSIS AND INTERPRETATION

4.1 Correlation of Tailings Logs

4.1.1 Correlation within the Klein Letaba Tailings Dam

Correlation along Profile P1

Correlation along Profile 1 was done in order to determine the oxidation, transitional and unoxidized zones along the profile within the Klein Letaba tailings dam (Fig. 4.1). It was observed that along each borehole, the colour of the tailings changed from orange and grey to brown up to a depth of 2.2 m along borehole P1A1 to the east. The oxidation of P1A2, P1A3 and P1A4 were up to a depth of 2.2 m, 1 m and 1.1 m respectively.

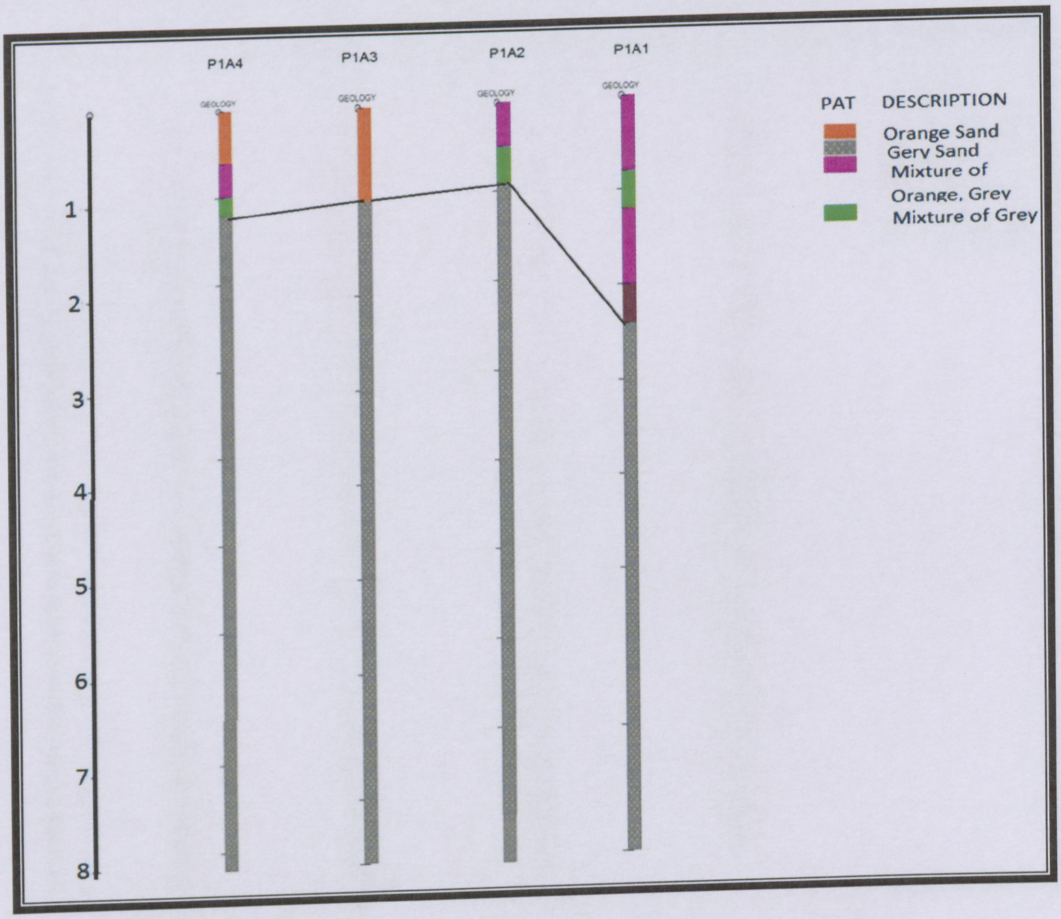


Figure 4.1: Correlation of drill hole logs of Profile 1 within Klein Letaba tailings dam depicting the oxidized and unoxidized zones.

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Correlation along Profile P2

As was the case with Profile 1, correlation was done to understand the continuity of layers with depth throughout Profile 2 over the Klein Letaba tailing dam (Fig. 4.2). It was observed that along each borehole the colour of the tailings changed from orange and grey to brown up to a depth of 3.2 m for P2A1, 2.9 m, for P2A2, 2 m for P2A3 and 1 m for P2A4. Variations indicate that the tailings were not being oxidized uniformly within the dam. The oxidation zone, however, was up to a depth of 3.2 m to the east and 1.0 m to the west. The transitional zone was not identified. As was observed in the previous case, there was uneven oxidation of the tailings with depth, a phenomenon that is attributable to uneven topography of the tailings dam and the retention of water on the tailings dam surface.

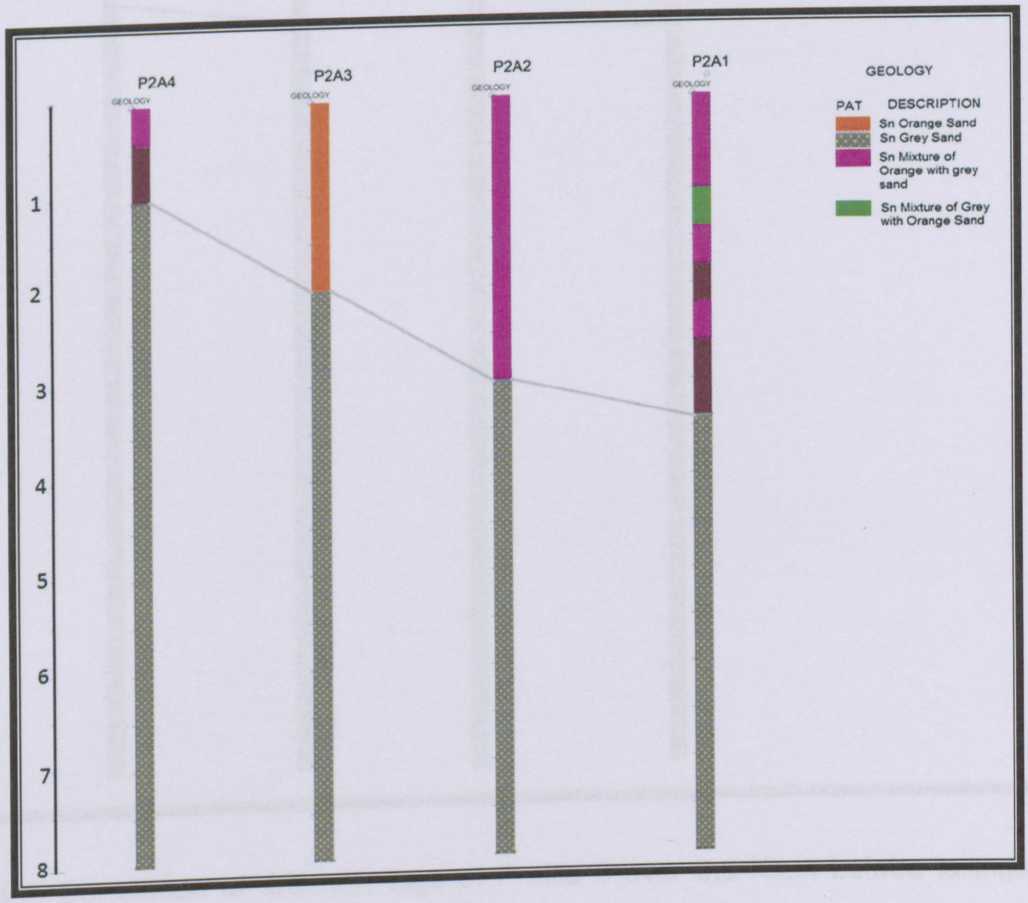


Figure 4.2: Correlation of drill hole logs along profile 2 over the Klein Letaba Tailings dam showing the oxidized and unoxidized zones.

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Correlation along Profile P3

Along the Profile 3 of the dam, the oxidation was restricted to a shallow zone between 1. 2 m in the east and 0. 9 m in the west. This profile was located to the southern margin of the dam (Fig. 4.3). Judging from the level of tailings erosion of this dam, it can be concluded that there was surface run-off, thus there was limited water infiltration through the tailings to support oxidation of the material.

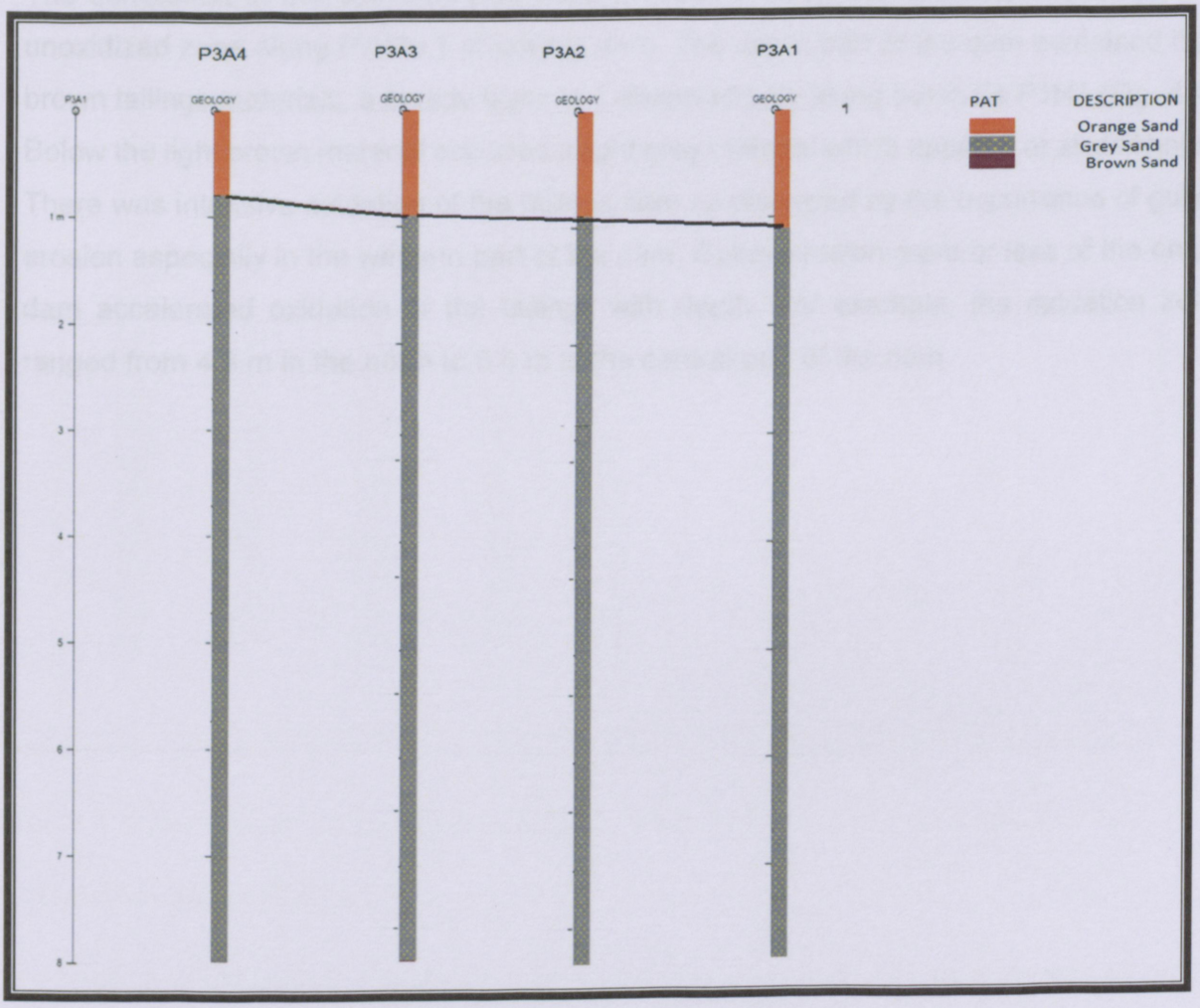


Figure 4.3: Correlation of drill hole logs of Profile 3 over the Klein Letaba tailings dam showing the oxidized and unoxidized zone

The Klein Letaba tailings dam was not being oxidized uniformly within it. The oxidation zone ranged from 0.9 m on the western part of the dam to 3.2 m in the eastern part. The variation

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of the oxidation zone may be attributed to the morphology of the dam, resulting in water settling longer at the same parts of the dam, especially in the eastern part. Below the oxidation zone was the unoxidized zone as the transition zone was not identified.

4.1.2 Correlation within the Louis Moore Tailings Dam

Correlation along Profile 1

The correlation of the drillholes was done in order to show the oxidation, transition and unoxidized zone along Profile 1 of tailings dam. The upper part of the dam contained light brown tailings materials, a muddy layer was observed only along borehole P1H1 (Fig. 4.4). Below the light brown material occurred a light grey material which appears at all drill holes. There was intensive oxidation of the tailings dam as displayed by the occurrence of gulley erosion especially in the western part of the dam. Gulley erosion more or less of the entire dam accelerated oxidation of the tailings with depth. For example, the oxidation zone ranged from 4.3 m in the north to 5.4 m in the central part of the dam.

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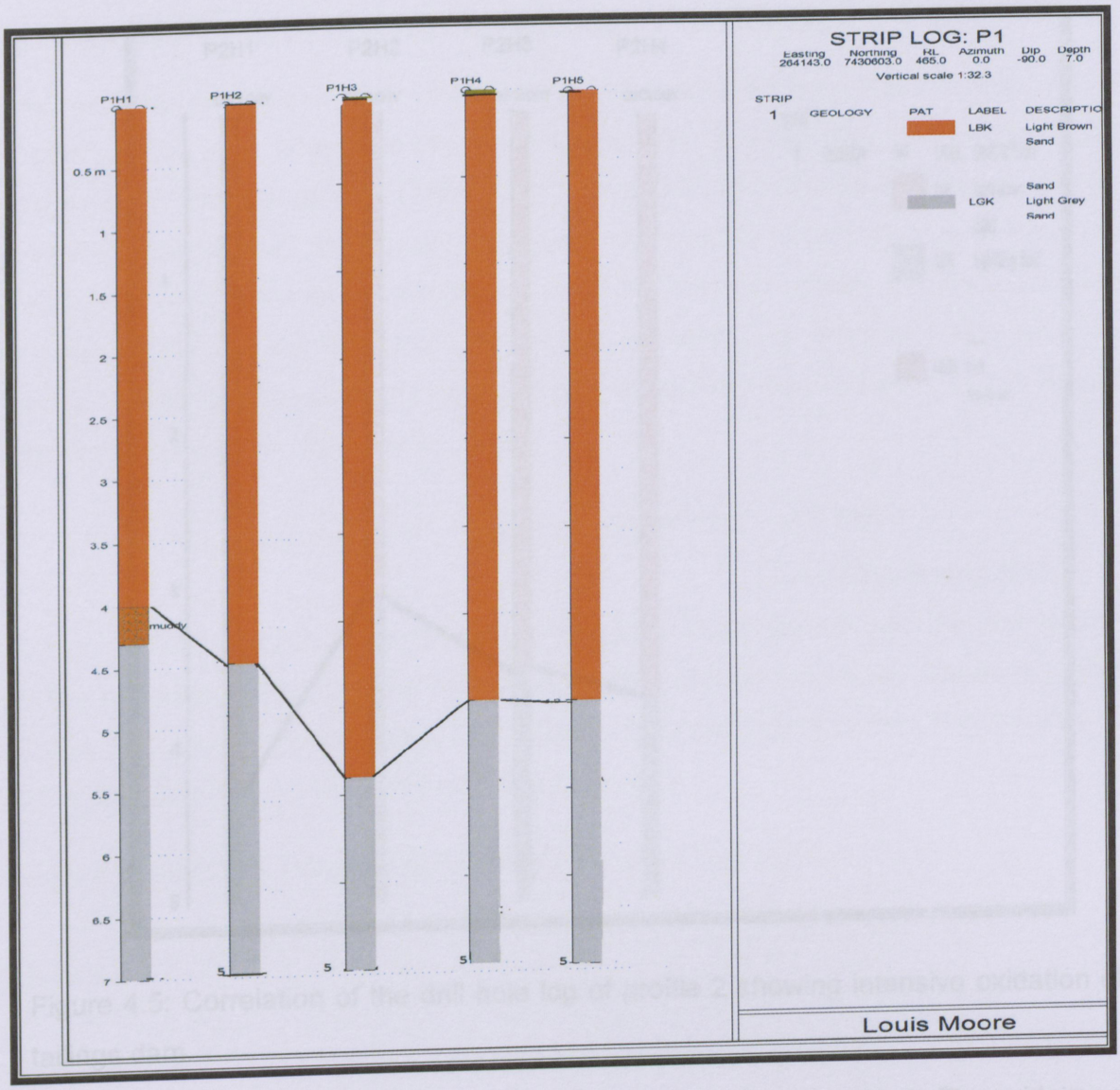


Figure 4.4: Correlation of the drill hole logs of profile 1 over Louis Moore Tailings dam

Correlation along Profile 2

The upper part of the dam contains light brown tailings, the muddy layer was observed only P2H1 (Fig. 4.5), and below the light brown material a light grey material that extended up to the bottom of the dam. The oxidation zone extended from a depth of 3.2 - 3.4 m in the middle to 4.3 m in the western part of the dam.

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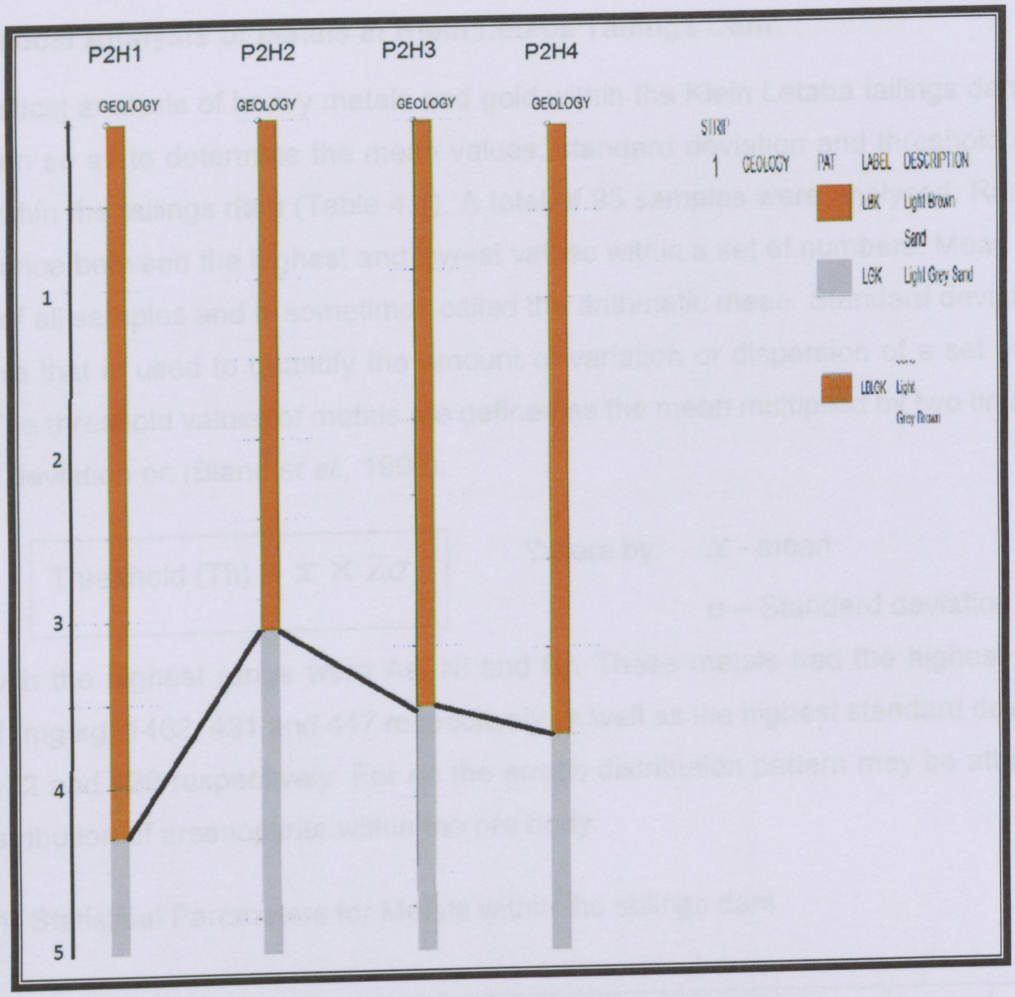


Figure 4.5: Correlation of the drill hole log of profile 2 showing intensive oxidation of the tailings dam.

The Louis Moore tailings dam was not being oxidized uniformly within it. The oxidation zone ranged from 4.3 m in the north to 5.4 m in the central part of the tailings dam and the oxidation zone of Profile 2 extended from a depth of 3.2 m to 3.4 m in the middle to 4.3 m in the western part of the dam. The gulley erosion were more or less of the entire tailings dam accelerated oxidation of the tailings with depth.

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4.2 Statistical Analysis of Metals at Klein Letaba Tailings Dam

The statistical analysis of heavy metals and gold within the Klein Letaba tailings dam was undertaken so as to determine the mean values, standard deviation and threshold of the metals within the tailings dam (Table 4.1). A total of 95 samples were analysed. Range is the difference between the highest and lowest values within a set of numbers. Mean is the average of all samples and is sometimes called the arithmetic mean. Standard deviation is a measure that is used to quantify the amount of variation or dispersion of a set of data values. The threshold values of metals are defined as the mean multiplied by two times the standard deviation or: (Bland *et al.*, 1996).

$$\text{Threshold (Th)} = x \times 2\sigma$$

Where by: x - mean

σ – Standard deviation

Metals with the highest range were As, Ni and Cr. These metals had the highest mean values of (mg\kg) 1462, 431 and 447 respectively as well as the highest standard deviation of 890, 172 and 129 respectively. For As the erratic distribution pattern may be attributed to the distribution of arsenopyrite within the ore body.

Table 4.1: Statistical Parameters for Metals within the tailings dam

Sample ID mg\kg	Pb	Zn	Cu	As	Cd	Cr	Co	Ag	Ni	Mn
Number of samples	95	95	95	95	95	95	95	95	95	95
Minimum	3.18	3.48	41	335	0.20	133	10	0.39	63	114
Maximum	127	113	138	3943	1.58	759	80	2.78	731	383
Range	124	109	97	3609	1.38	625	70	2.38	668	270
Mean	20	50	74	1462	0.57	447	46	0.65	431	205
Median	16	36	73	1261	0.40	457	48	0.60	422	186
Standard deviation %	17	26	20	890	0.23	129	15	0.31	172	55
Threshold	54	101	114	3242	1.03	706	75	1.27	774	315

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4.3 Distribution of Heavy Metals within the Klein Letaba Tailings Dam

Borehole P1A1

The data obtained from geochemical analysis of metals within the tailings dam were used to analyse the distribution of the metals with depth (Figs 4.6 - 4.17). The metals showed different distribution pattern with depth (Fig. 4.6). For example, Pb, Zn, Cu, As, Co, Ni, Mn and Ag showed erratic distribution with depth with elevated values either between 2 – 3 m for Cu and Ag and between 2 – 3 m for Pb, As, Zn, Co and Ni. Cu and Mn also had elevated values between 3 – 4 m while Ni was between 5 – 6 m. The only metal that showed consistent increased values with depth was Cr. Cd however showed more or less uniform values between 0 – 2 m the uniform increase between 2 – 5 m followed by a uniform decrease.



Figure 4.6: Distribution of metals along borehole P1A1.

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Borehole P1A2

The distribution of metals along this borehole was erratic in most cases. As, Ag, Zn and Pb showed an almost smooth trend from the first meter to the bottom (Fig. 4.7). Cd and Ni had a similar trend from the 2 m of the dam, but gradually decreased with depth although for Ni there was a gradual increrase again from a depth of 4 m downwards while for Cd there were erratic increases and decreases with depth, Mn, Co and Cu had an erratic distribution throughout the depth.

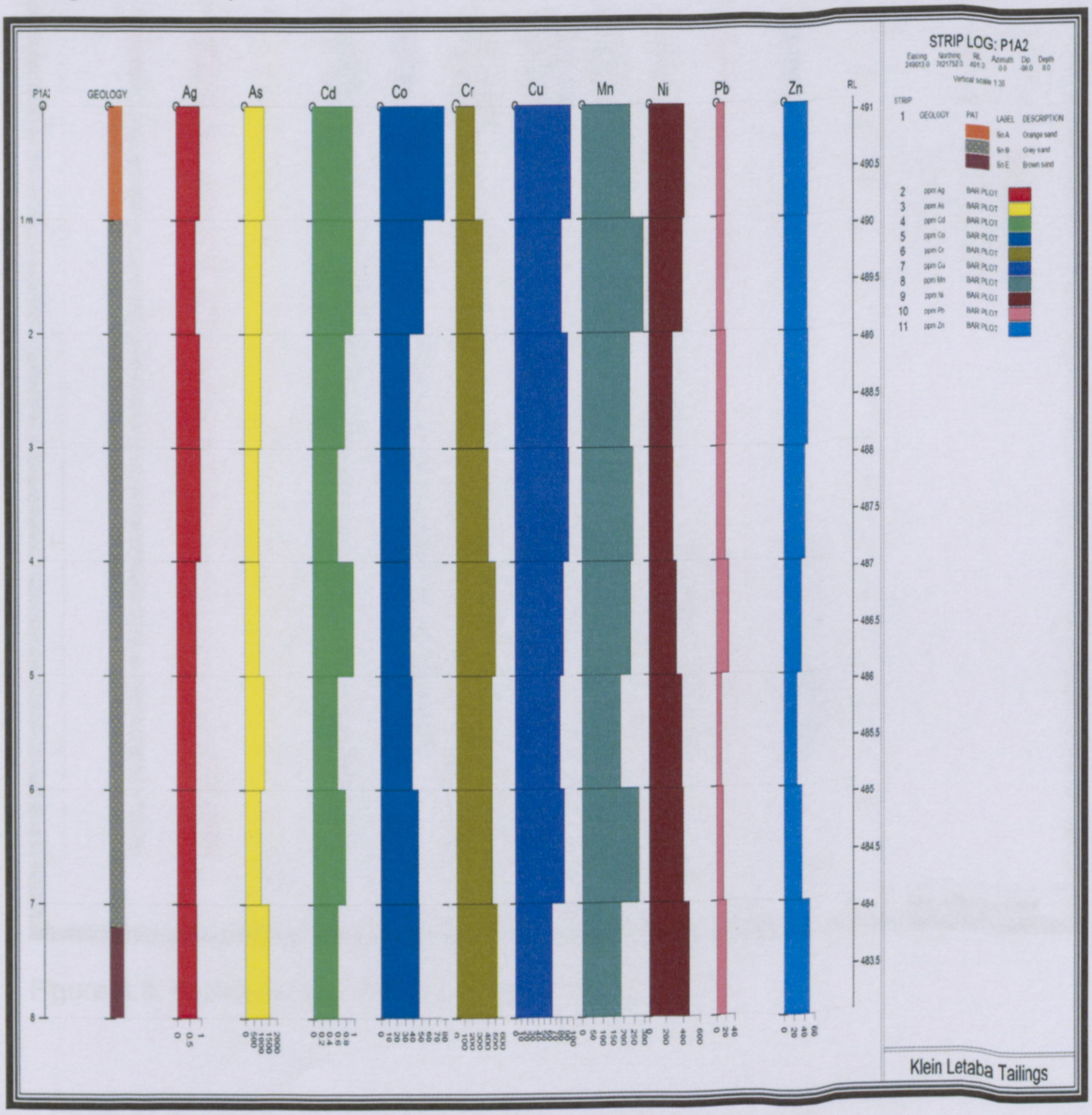


Figure 4.7: Distribution of metals along borehole P1A2.

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Borehole P1A3

Along this borehole Zn, Ni, and Co had an almost similar trends throughout the depth (Fig. 4.8). From the first meter to the seventh meter there was a general increases metal values, whilst from level 7 there was a sharp decrease in values except for Pb and Zn and Co. The distribution pattern of Ag, Cd, Cr, Cu and Mn were rather erratic throughout the 8 m column.

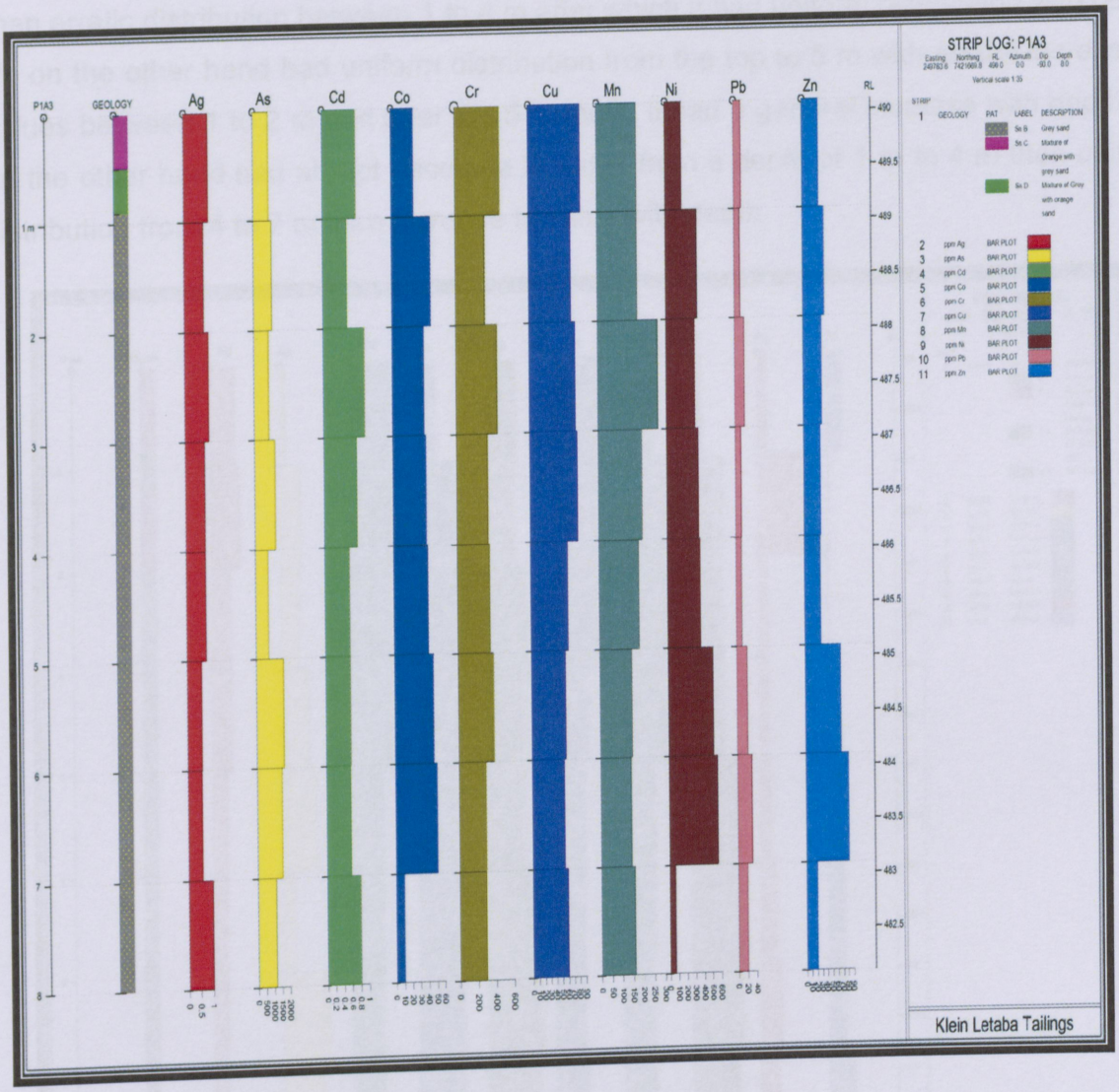


Figure 4.8: Distribution of metals along borehole P1A3.

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Borehole P1A4

Ag and Cu showed a general decrease with depth where as Co, Cr and Ni indicated the opposite (Fig. 4.9) Cd also registered a general decrease with depth except between 1 – 2 m. As showed a general increase with depth except at an interval of 5 – 6 m that there was an abrupt decrease in values. Mn showed a relatively uniform distribution from the first 1 m then erratic distribution between 1 to 4 m after which it had uniform distribution with depth. Pb on the other hand had uniform distribution from the top to 5 m with abrupt increase of values between 1 to 2 m and after the 5 m mark, it had a general increase with depth. Zn on the other hand had abrupt decrease in value from a depth of 1 m to 4 m then uniform distribution from 4 to 7 m then increase in value with depth.

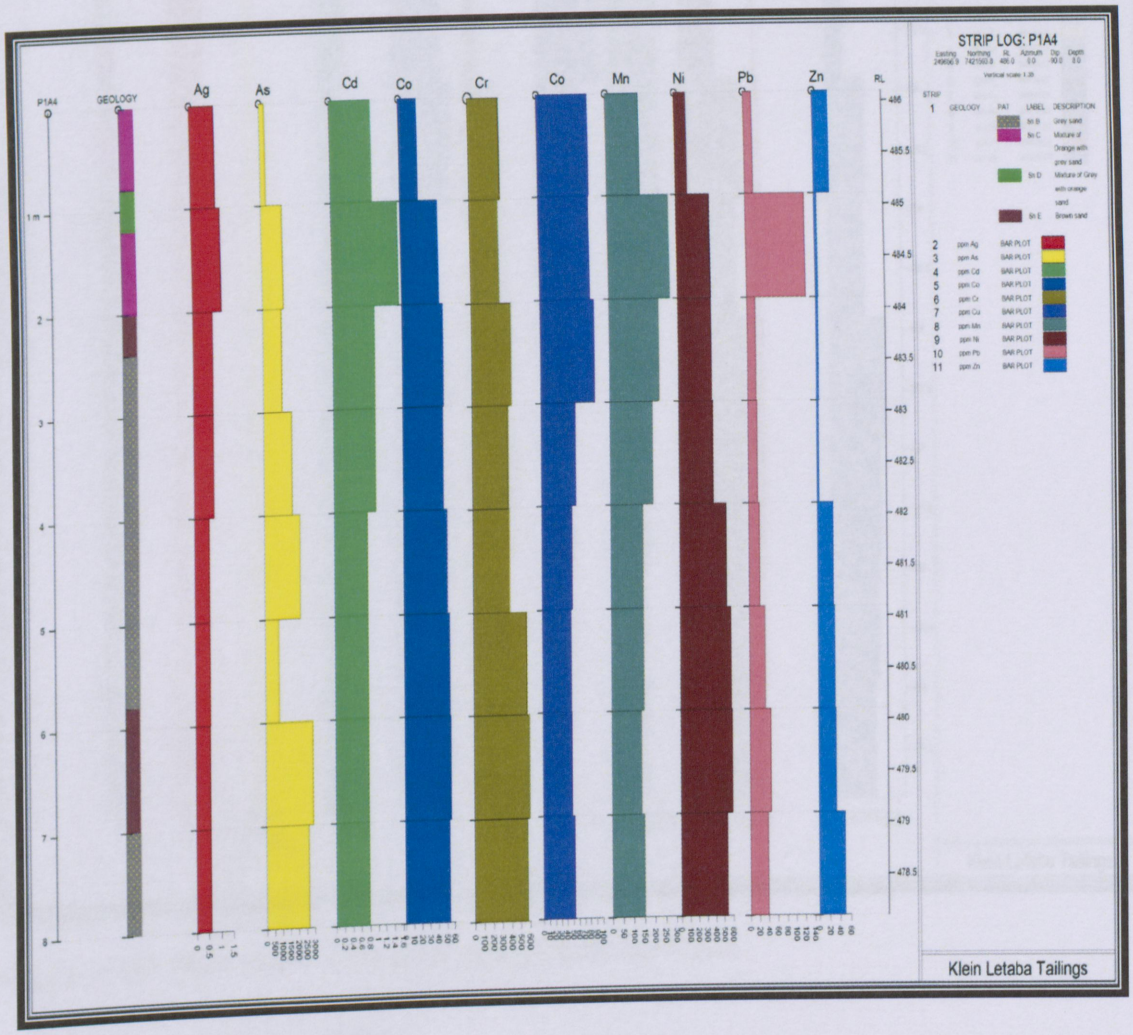


Figure 4.9: Distribution of metals in Borehole P1A4.

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Borehole P2A1

Along this borehole, Ag, Cd, Cu and Mn showed a general decrease with depth, whereas Co, Ni and Zn had a general increase with depth (Fig. 10). The values of Pb were low and mainly uniform, whereas Cr values were erratic. As showed decrease with depth up to 2 m and from 2 m up to 8 m showed increase and Mn showed decrease with depth.

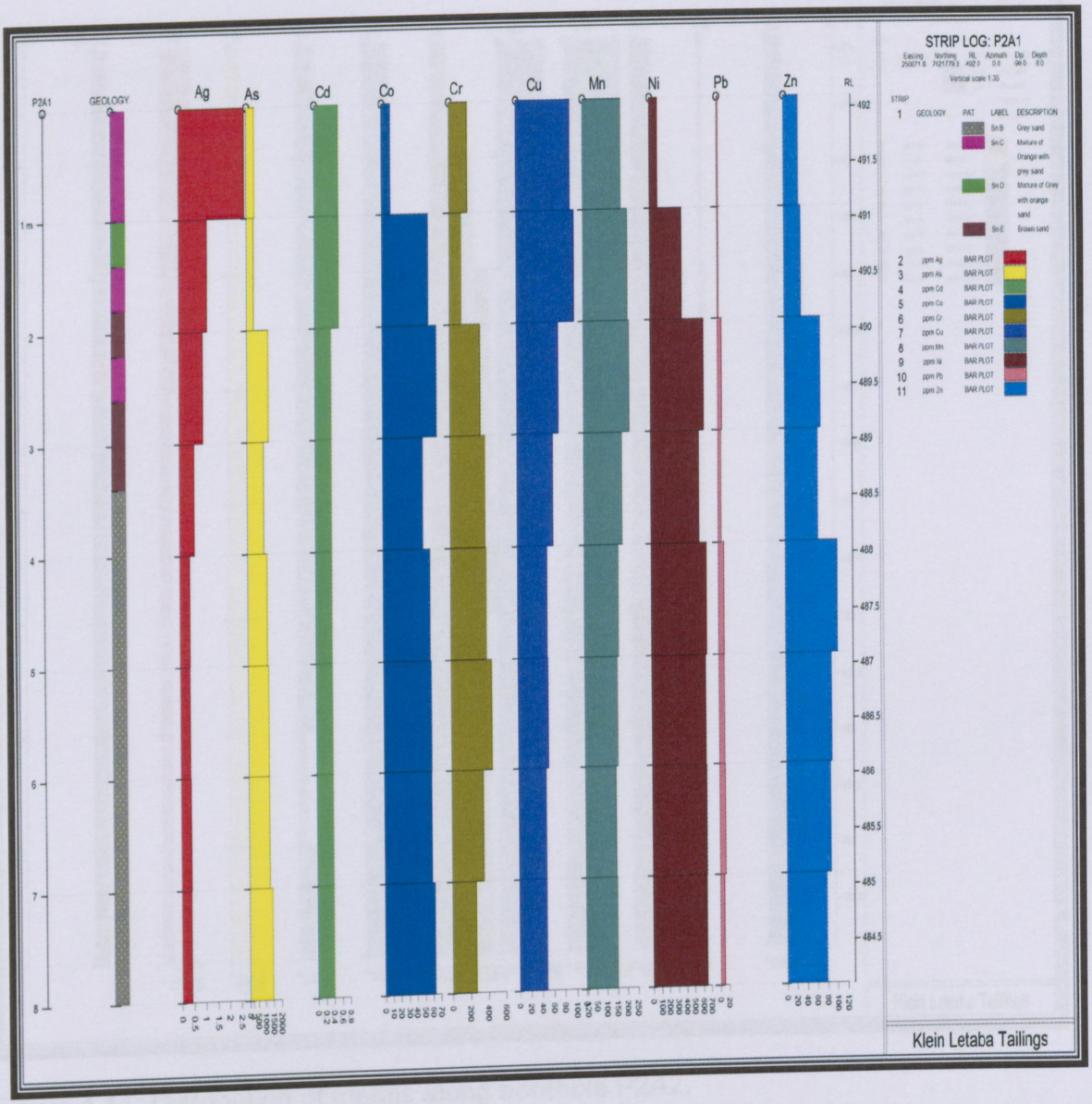


Figure 4.10: Distribution of metals along borehole P2A1.

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Borehole P2A2

Along borehole P2A2, Ag, Cr and Mn decreased with depth whereas As, Co, Cr, Ni and Zn increases with depth (Fig. 4.11). The values of Cd were more or less uniform except between 6 to 7 m. Pb registered low values that were relatively uniform.

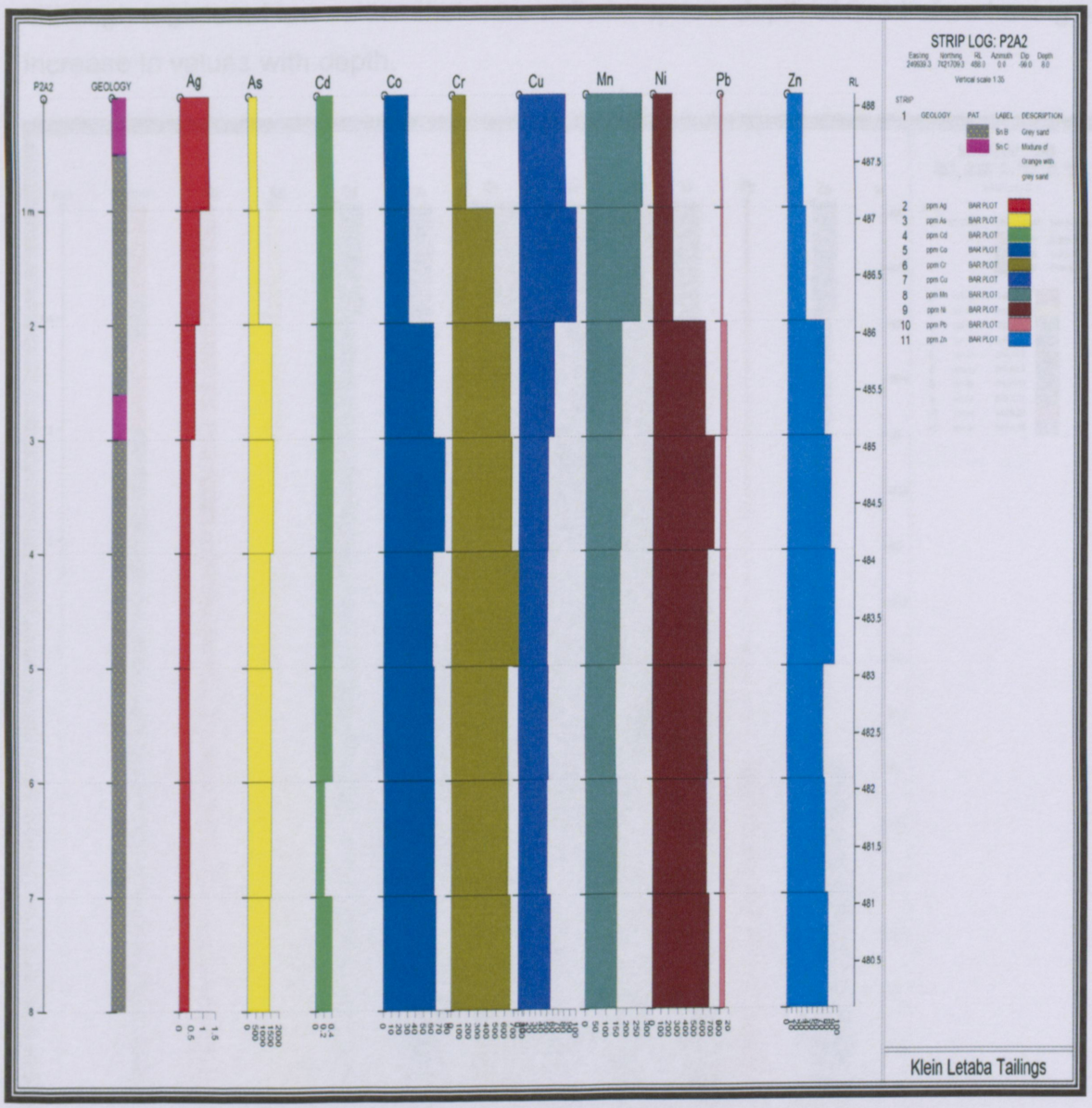


Figure 4.11: Distribution of metals along borehole P2A2.

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Borehole P2A3

The distribution of metals along this borehole was rather erratic, although Cu and Mn showed a general decrease with depth whereas, Co and Ni indicated a general increase with depth (Fig. 4.12). Ag and Cd indicated more or less uniform distribution with depth. Pb although registered low values that were uniform up to a depth of 5 m before having slight increase in values with depth.

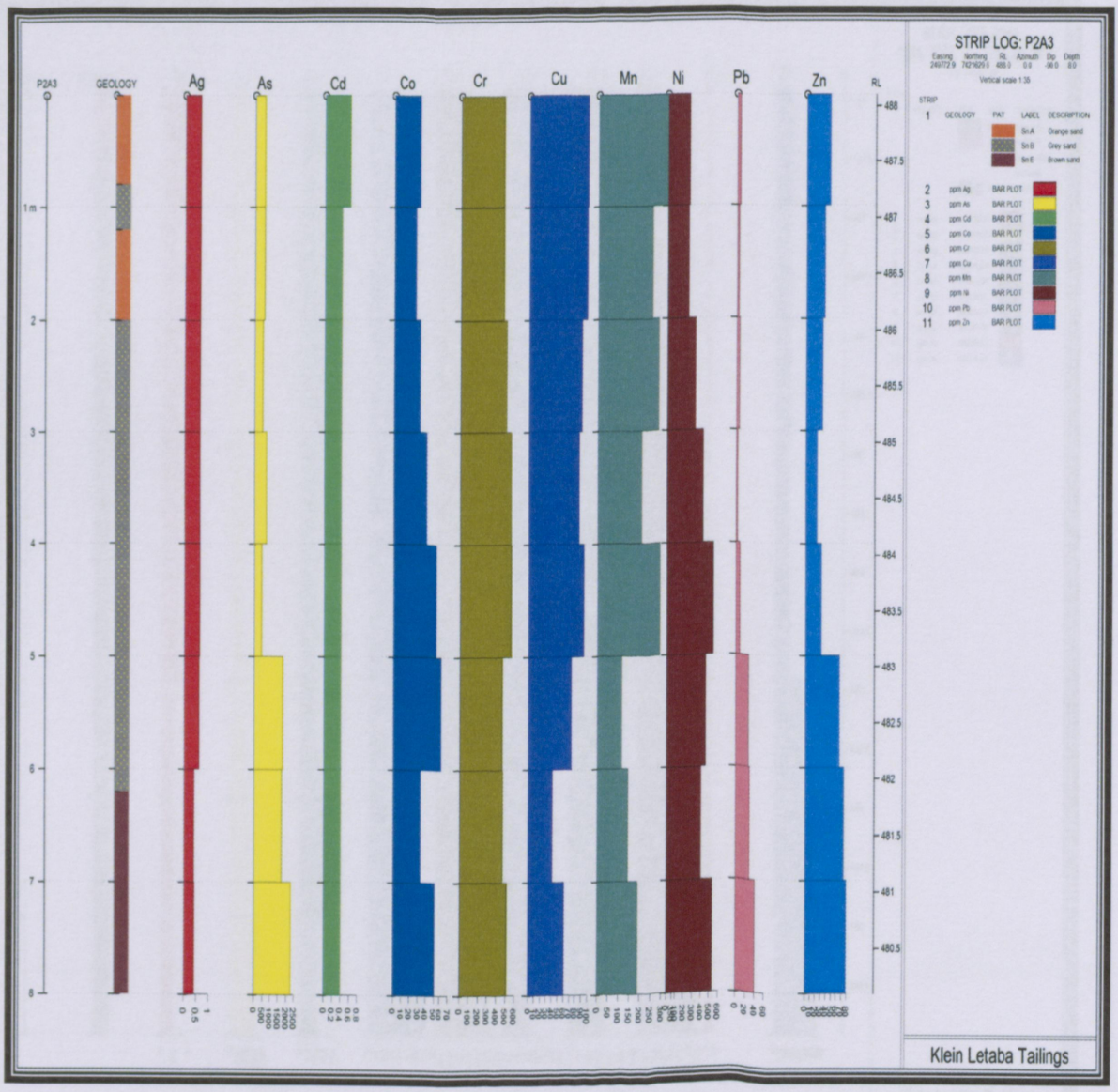


Figure 4.12: Distribution of metals along borehole P2A3.

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Borehole P2A4

Borehole P2A4 showed more or less metal distribution pattern that were either decreasing with depth or increasing with depth (Fig. 4.13). For example, Ag, Cu and Mn decreased with depth, whereas Co, Cr, Zn and Ni increased with depth. The exceptions to these trends were As that registered more or less uniform distribution except from level 5 m downwards that showed a slight increase.

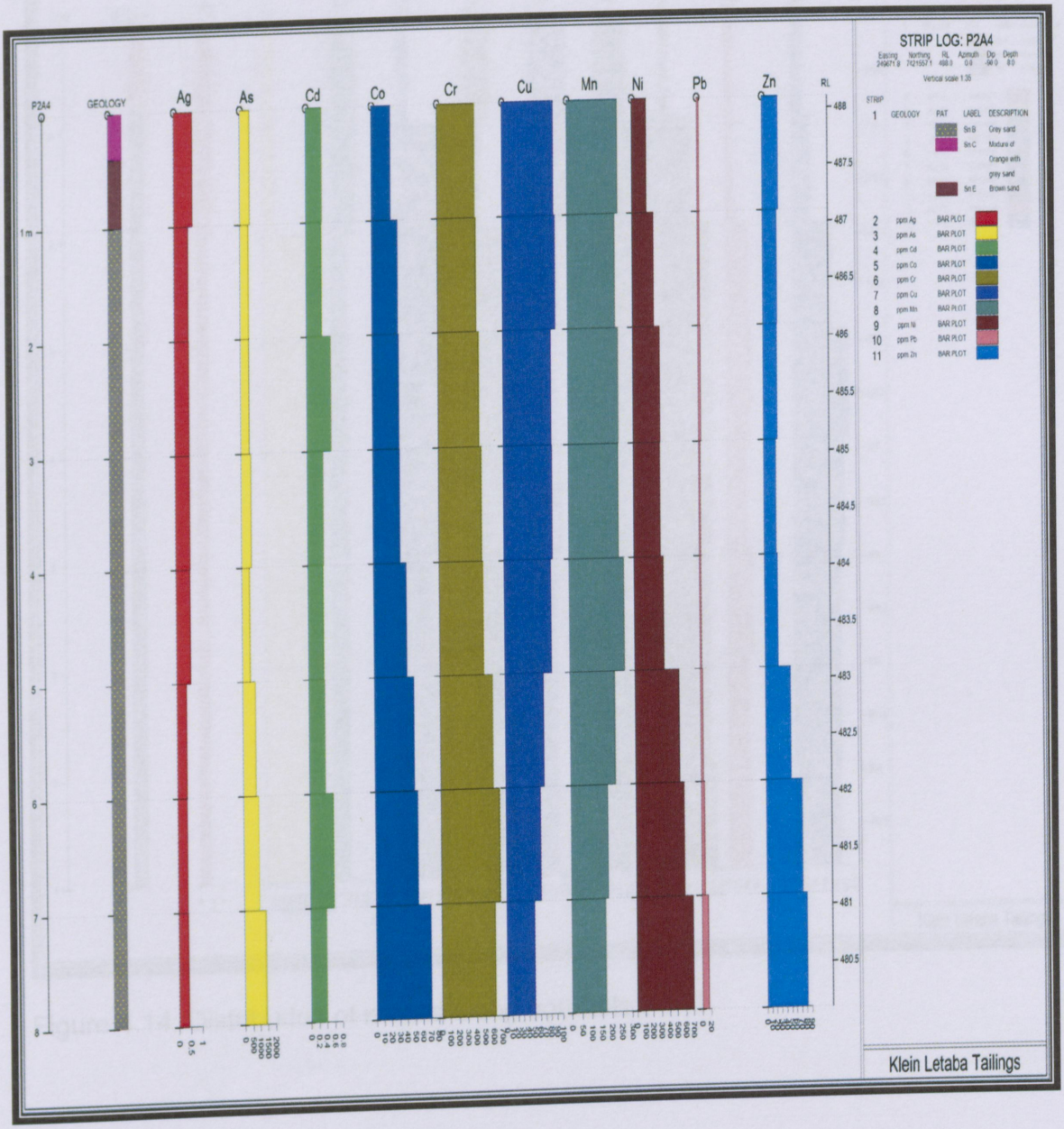


Figure 4.13: Distribution of metals along borehole P2A4.

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Borehole P3A1

The distribution of metals along borehole P3A1 increase or decrease with depth (Fig. 4.14). For example, either As, Co, Cr, Pb and Zn generally increased with depth, whereas Ag, Cd, Cu and Mn decreased with depth.

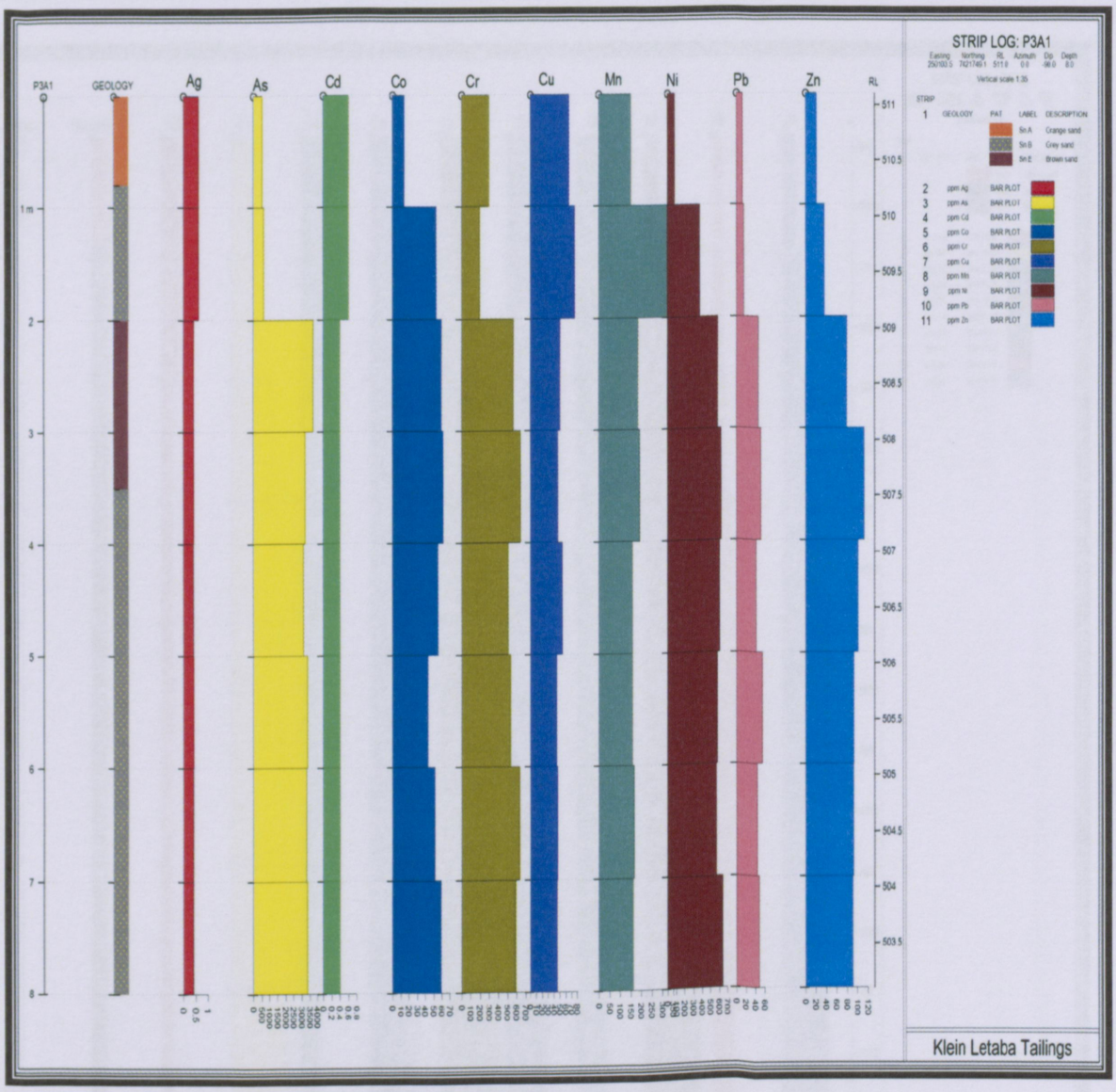


Figure 4.14: Distribution of metals along borehole P3A1.

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Borehole P3A2

A similar trend was observed along borehole P3A2 (Fig. 4.15). As, Co, Cr, Ni, Pb and Zn showed a general increase with depth whereas Ag and Cd indicated uniform distribution between 0 to 2 m there is a decrease that was uniform downwards. Cu however displayed erratic distribution.

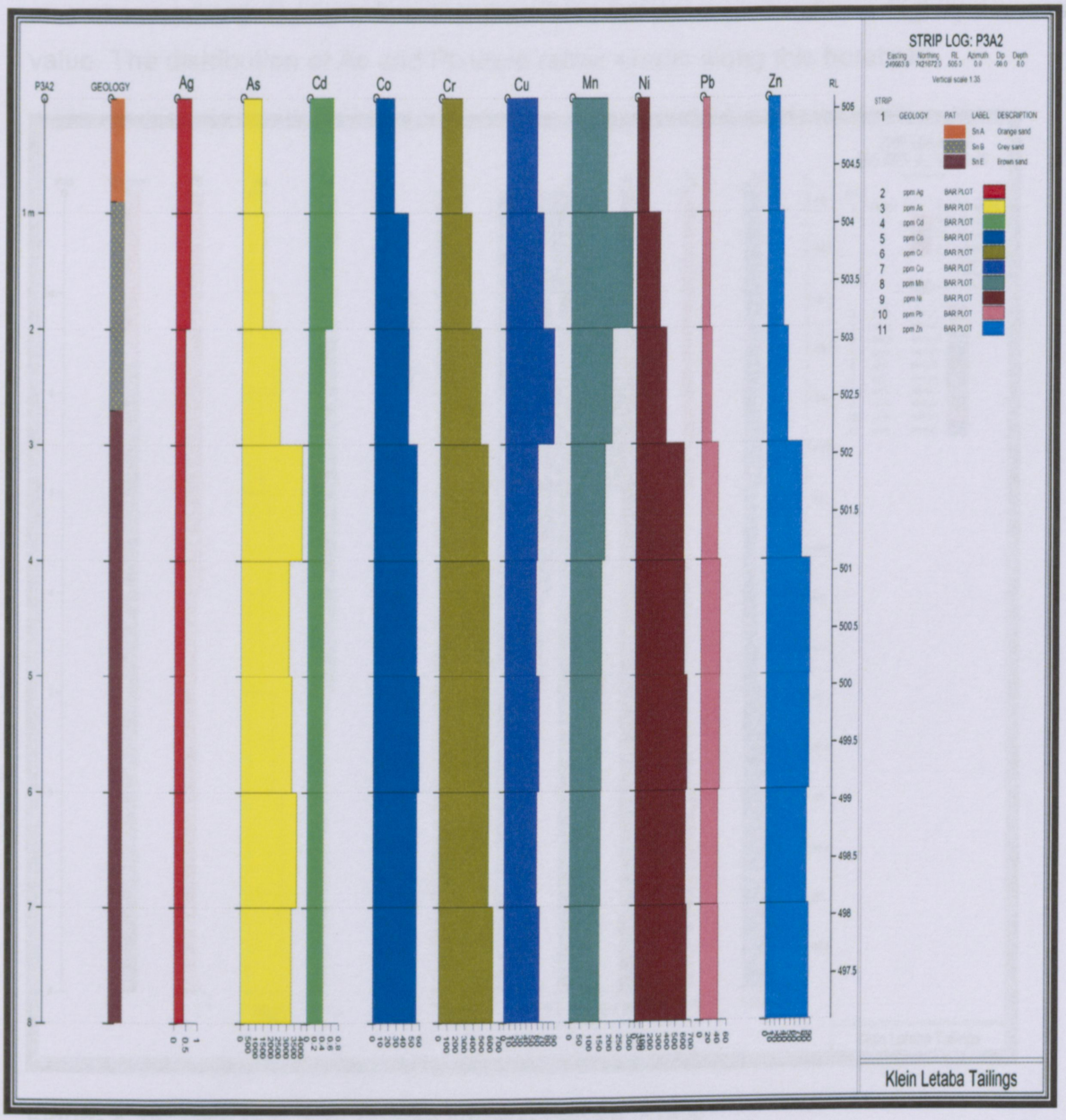


Figure 4.15. Distribution of metals along borehole P3A2.

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Borehole P3A3

Distribution of heavy metals along borehole P3A3 showed different trend (Fig.4.16). Co, Cr, Ni and Zn showed a general increase with depth. Whereas Cu and Mn showed a decrease with depth. Ag showed uniform distribution between 0 to 4 m and from the 4 m level, there was a slight decrease that persisted with depth. Uniformity is distribution was also observed with Cd from 0 to 5 m then from 5m downwards after a slightly decrease in value. The distribution of As and Pb were rather erratic along this borehole.

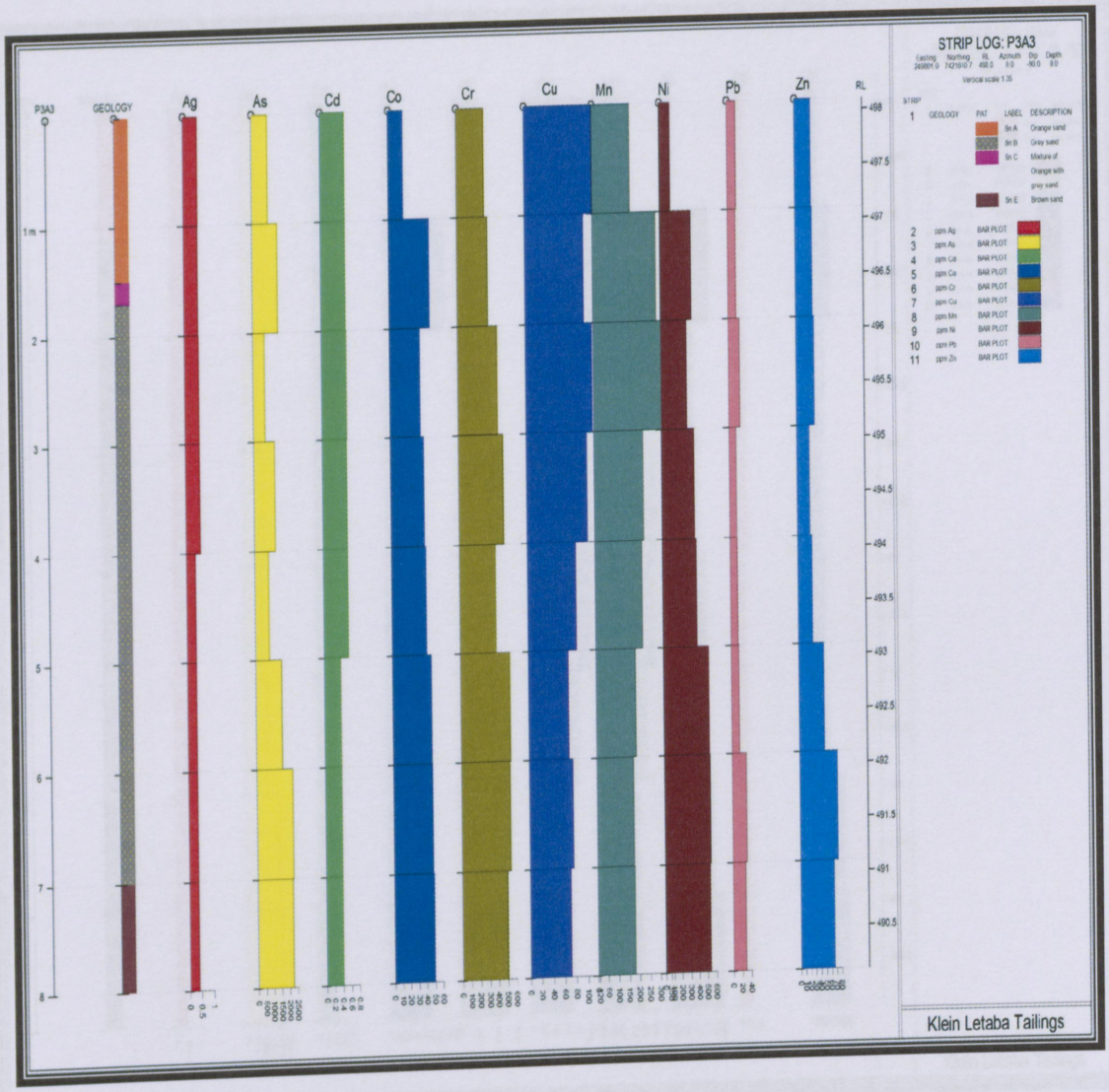


Figure 4.16: Distribution of metals along borehole P3A3.

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Borehole P3A4

The distribution of metals along this borehole was variable (Fig.4.17). For example, Cr, Cd, Ni and Zn increased value with depth although with abrupt increase between 1 to 2 m for Co, Ni and Zn and between 3 to 4 m for Cr. Cu and Mn decreased in values with depth except for the abrupt increase between 1 to 2 m. Cd and Pb were rather uniform in distribution except for that showed a slight increase between 1 to 3 m. Ag and As displayed erratic distribution.

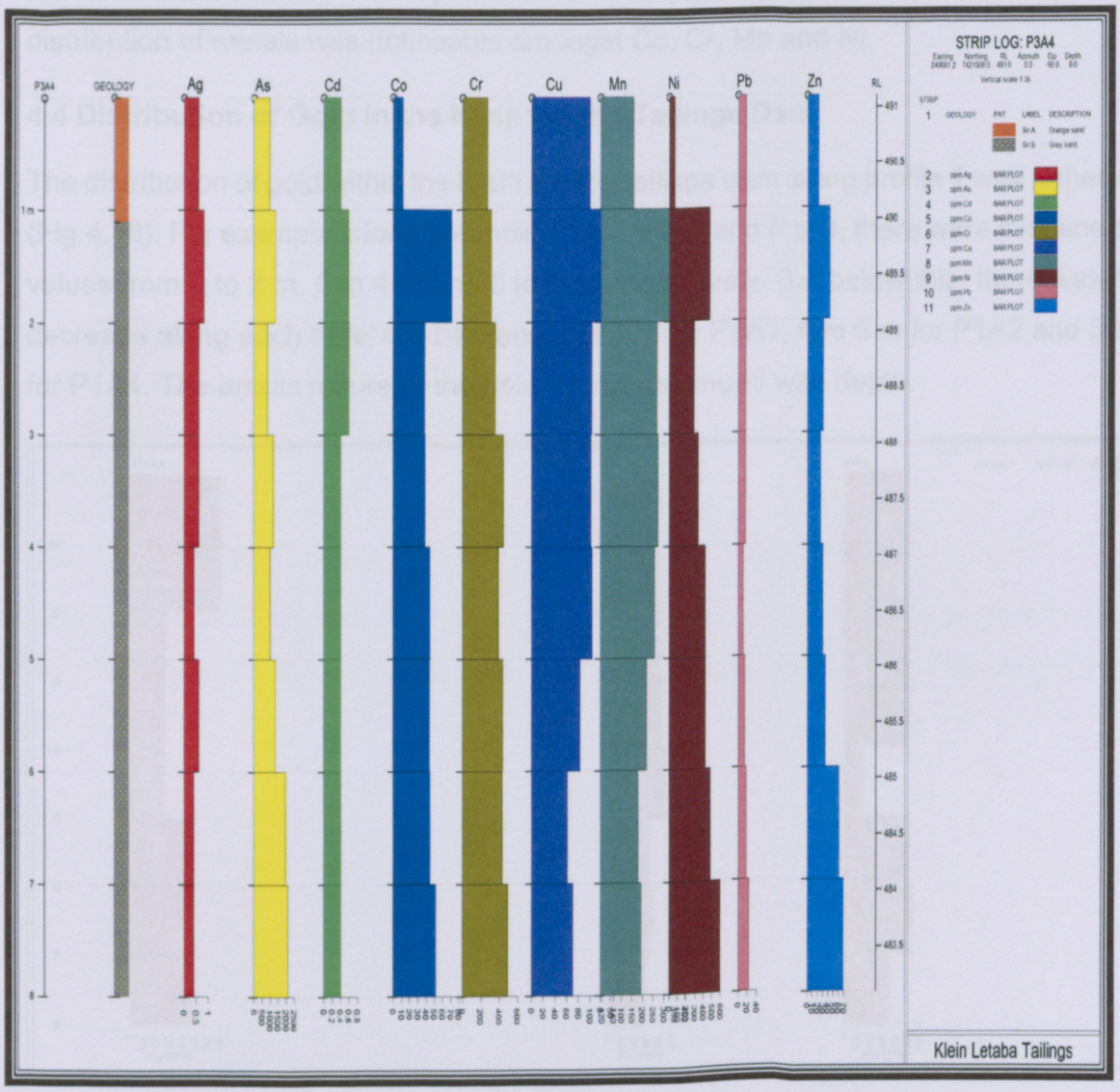


Figure 4.17: Distribution of metals along borehole P3A4.

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The overall assessment of the distribution pattern of heavy metals at the Klein Letaba tailings dam showed erratic distribution that may be attributed to the nature of tailings at the initial stages of tailings disposal. The tailings dam has not matured enough to produce acid mine drainage, hence there was no distinct oxidation boundary between the reducing zone. The distribution of the metals literally varied from one drill hole to the other, although a general trend was noted with some metals decreasing in the value, whereas others increase in values with depth. For example, along a number of drill holes, Ag, Cd, Cu and Mn decrease in values with depth, whereas As, Co, Cr and Ni increase with depth. Erratic distribution of metals was noticeable amongst Co, Cr, Mn and Ni.

4.4 Distribution of Gold in the Klein Letaba Tailings Dam

The distribution of gold within the Klein Letaba tailings dam along profile 1 was rather erratic (Fig.4.18). For example, along borehole P1A1, P1A2 and P1A4, there were sustained good values from 2 to 2 m. 0 to 4 m and 0 to 5 m respectively. But below this, there was abrupt decrease along each borehole between 2 to 5 m for P1A1, 4 to 5 m for P1A2 and 5 to 6 m for P1A4. The erratic nature of the gold values continued with depth.

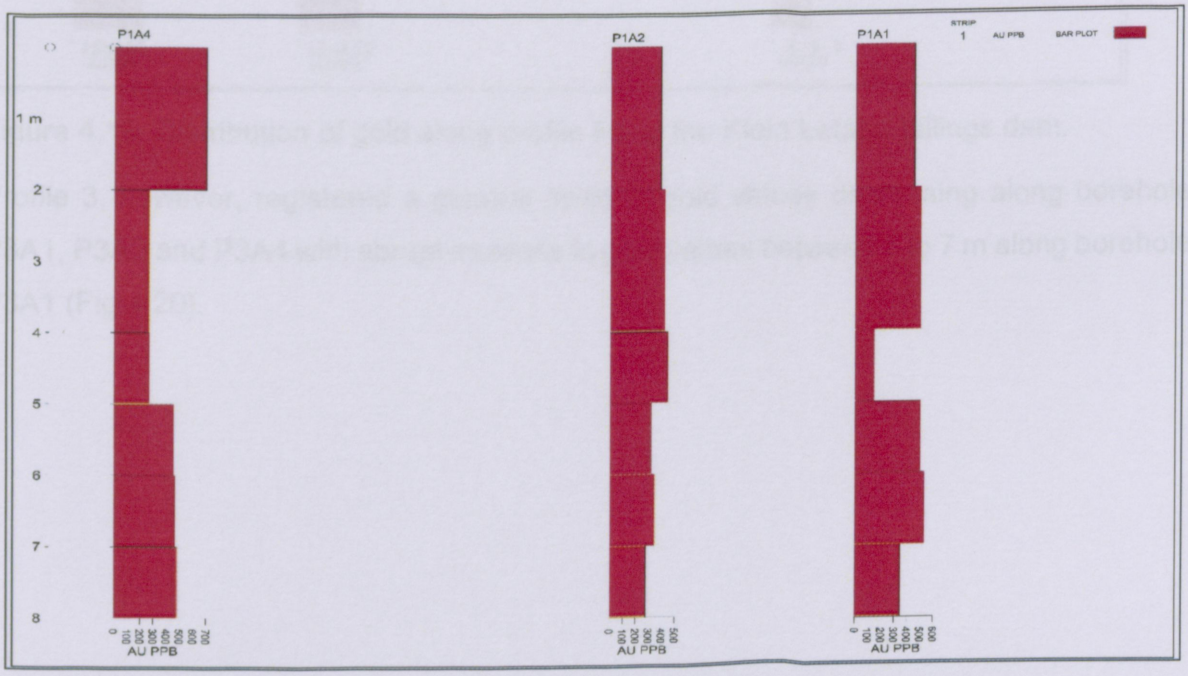


Figure 4.18: Erratic distribution of gold along profile P1 at the Klein Letaba tailings dam.

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Along profile 2, the gold values decrease with depth along borehole P2A1 (Fig 4.19), whereas there was general increase of gold values along profiles P2A3 and P2A4, although for profile P2A4, there was abrupt decrease of gold values between 5 to 6 m the values to increase downwards from the 6 m mark.

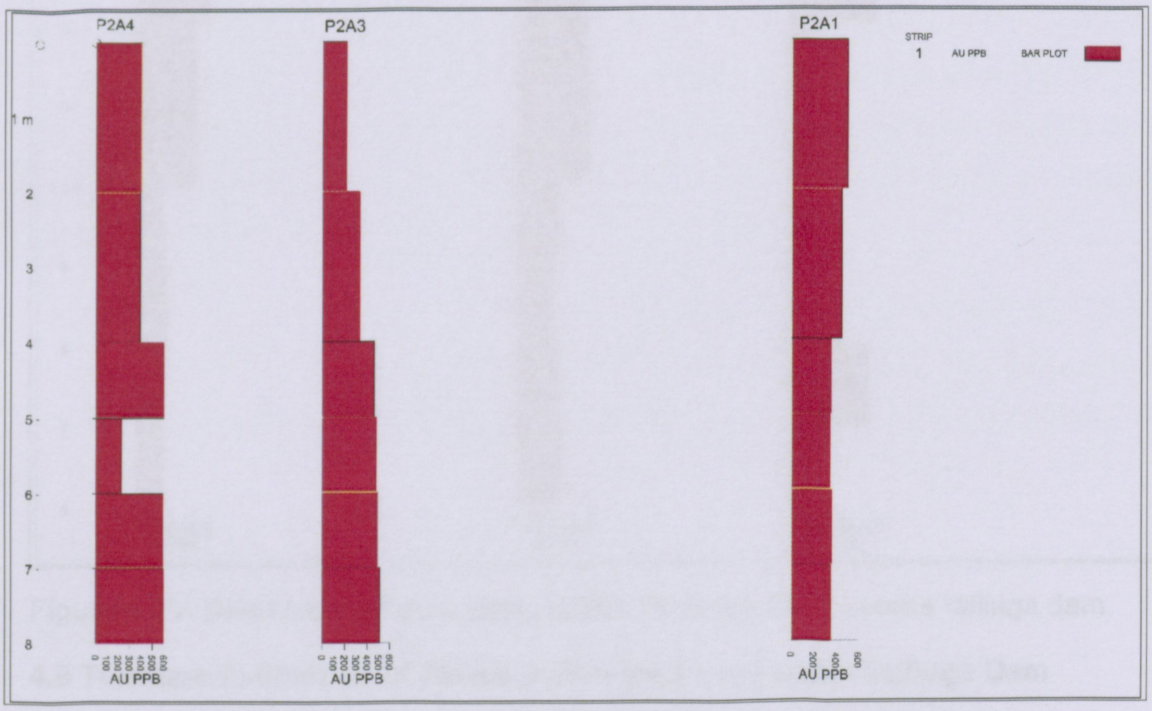


Figure 4.19: Distribution of gold along profile P2 at the Klein Letaba tailings dam.

Profile 3, however, registered a general trend of gold values decreasing along borehole P3A1, P3A2 and P3A4 with abrupt increase in gold values between 6 to 7 m along borehole P3A1 (Fig.4.20).

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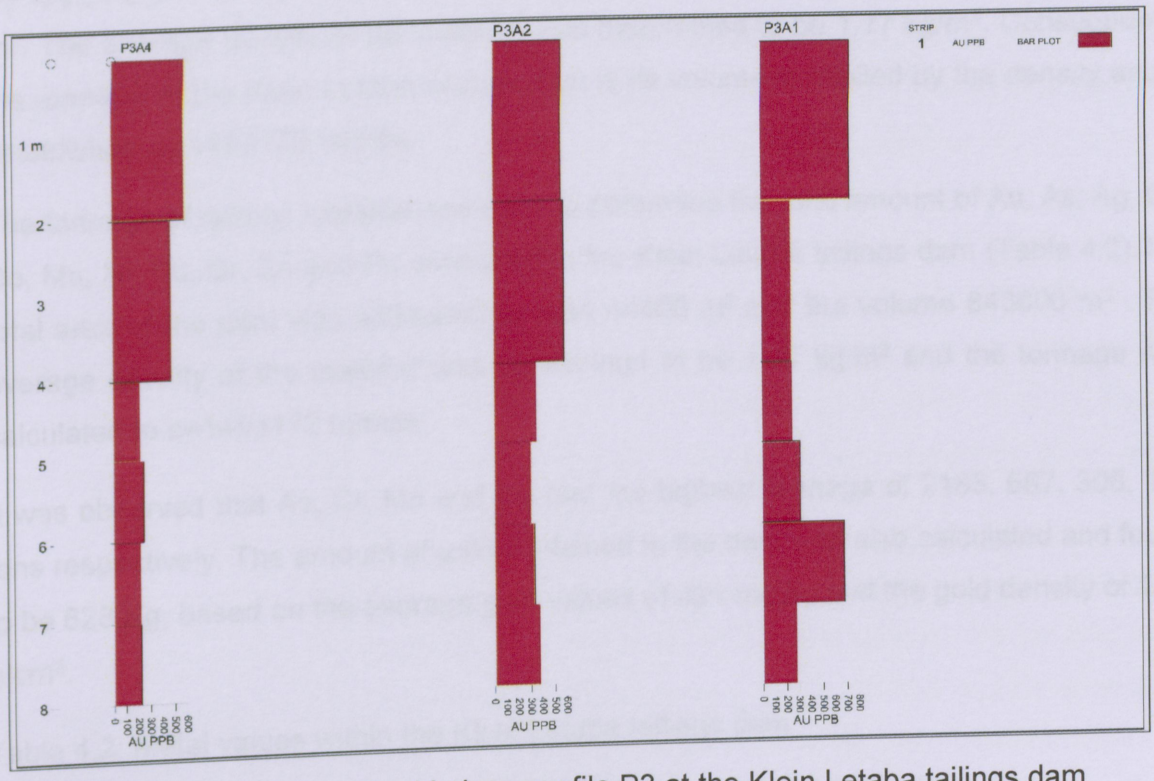


Figure 4.20: Distribution of gold along profile P3 at the Klein Letaba tailings dam.

4.5 Tonnage Estimation of Metals within the Klein Letaba Tailings Dam

The trapezoidal rule method was used to estimate the volume of the tailings material. The method is applicable since it assumes that the ground between the spot heights is joined by straight lines. In order to accurately estimate the volume of the tailings, the tailing dam was divided into three segments of width; 100m, 88 m and 100 m and the length of each segment as determined in the field was 150 m. The formula for calculating areas of the three segments of the tailings dam is given by the Trapezoidal rule:

$$\mathbf{A} = \frac{L(O_A + O_C)}{2} (O_B) \text{ and hence, the Volume} = \mathbf{A\ H}$$

Where the L = length of each segment, O_A, O_B and O_C are the three segments and H = height of the tailings dam.

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By applying the formula, the total area of the tailings is 44400 m^2 and the volume is 843600 m^3. The average density of the material was determined to be 1.77 kg/m^3. Consequently, the tonnage of the Klein Letaba tailings dam is its volume multiplied by the density and is established as 1493172 tonnes.

The tonnage of tailings material was used to determine the total amount of Au, As, Ag, Cu, Co, Mn, Ni, Cd, Cr, Zn and Pb contained in the Klein Letaba tailings dam (Table 4.2).The total area of the dam was estimated as was 44400 m^2 and the volume 843600 m^3 . The average density of the material was determined to be 1.77 kg/m^3 and the tonnage was calculated to be1493172 tonnes.

It was observed that As, Cr, Mn and Cu had the highest tonnage of 2183, 667, 306, 110 tons respectively. The amount of gold contained in the dam was also calculated and found to be 628 Kg, based on the average gold values of 421 mg\kg and the gold density of 18.3 g/cm^3.

Table 4.2: Metal values within the Klein Letaba tailings dam

Metals	Mean Concentration (mg\kg)	(Kg)
Pb	20	30
Zn	50	75
Cu	74	110
As	1462	2183
Cd	0.57	0.851
Cr	447	667
Co	46	69
Ag	0.65	0.971
Ni	431	644
Mn	205	306
Au	421	628

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4.6 Statistical Analysis of Metals at the Louis Moore Tailings Dam

As in the case of the Klein Letaba tailings dam, statistical analysis of the Louis Moore tailings dam was undertaken so as to determine the statistical characteristics of the metals. The range, mean, median, standard deviation, coefficient of variation and threshold of metals were calculated (Table 4.3). The Table shows the statistical summary of the results for Pb, Zn, Cu, As, Cd, Cr, Co, Ag, Ni and Mn. The interpretation of the results in the following sections is based on the data presented in Table 4.1. It is observed that Mn has got the highest concentration with a mean of 1306.50 mg\kg. The median value for Mn is closer to the mean and is the coefficient of variation is about 15%. This indicate the homogeneous distribution of the data. Standard deviation was computed to show how widely spaced the concentration values are spread.

Cr, Mn, Zn, Ni and As had got high standard deviation values, thus meaning their concentration values are widely spaced, however As and probable Ni are heterogeneous distributed and positively skewed which probably suggest the metals are concentrated at certain depth, whereas the other base metals are homogenously distributed throughout the tailing depth. This is confirmed by coefficient variation of As, Ni of 85% and 37% respectively and Cr, Mn, Zn have less than 20% CV. In this table it is apparent that while the concentration for some metals (Cd and Ag) are low 0.43 ppm and 0.97-ppm respectively some others (Cr, Ni and Mn) are quite high 335,71mg\kg, 415,95 mg\kg and 1306,50 mg\kg respectively.

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Table 4.3: Statistical analysis of the metals within the tailings dam

Sample ID mg\kg	Cu	Co	As	Pb	Cd	Ag	Cr	Ni	Mn	Zn
Number of samples	55	55	55	55	55	55	55	55	55	55
Minimum	16,19	0,59	7,91	37,20	-1,99	0,39	265,38	232,63	836,20	63,33
Maximum	27,77	35,46	185,88	207,65	0,80	1,58	761,09	1494,42	1848,64	149,22
Range	11,58	34,87	177,97	170,45	2,78	1,18	495,72	1261,79	1012,44	85,89
Mean	21,43	26,29	59,65	60,08	0,44	0,98	335,71	415,95	1306,50	93,21
Median	21,55	27,35	33,67	55,55	0,40	1,00	328,99	404,38	1360,17	91,18
Standard deviation	2,44	4,68	50,81	22,02	0,40	0,29	63,89	154,03	201,70	16,97
Coefficient of variation	0,11	0,18	0,85	0,37	0,91	0,29	0,19	0,37	0,15	0,18
Threshold	26,30	35,65	161,27	104,11	1,23	1,55	463,48	724,02	1709,90	127,15

4.7 Distribution of Heavy Metals within the Louis Moore tailings dam

The distribution pattern of heavy metals with depth at the tailings dam was determined (Fig. 4.21 - 4.24). This was done in order to deduce the distribution pattern of the concentration of the heavy metals with depth, see appendix D for more graphs showing the distribution pattern of metals at different sample locations around the tailings dam.

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Borehole P1H5

Distribution of heavy metals along borehole P1H5 showed different trend (Fig.4.21). As showed a low values that are more or less uniform from top to bottom and then at the other hand Cr also indicate distribution of more or less uniform behavior. Cu and Pb showed erratic trend distribution from with depth and Ni also showed more or less uniform except from 1 to 0. Co, Ag, Cd, Mn and Zn has indicated increase with depth.

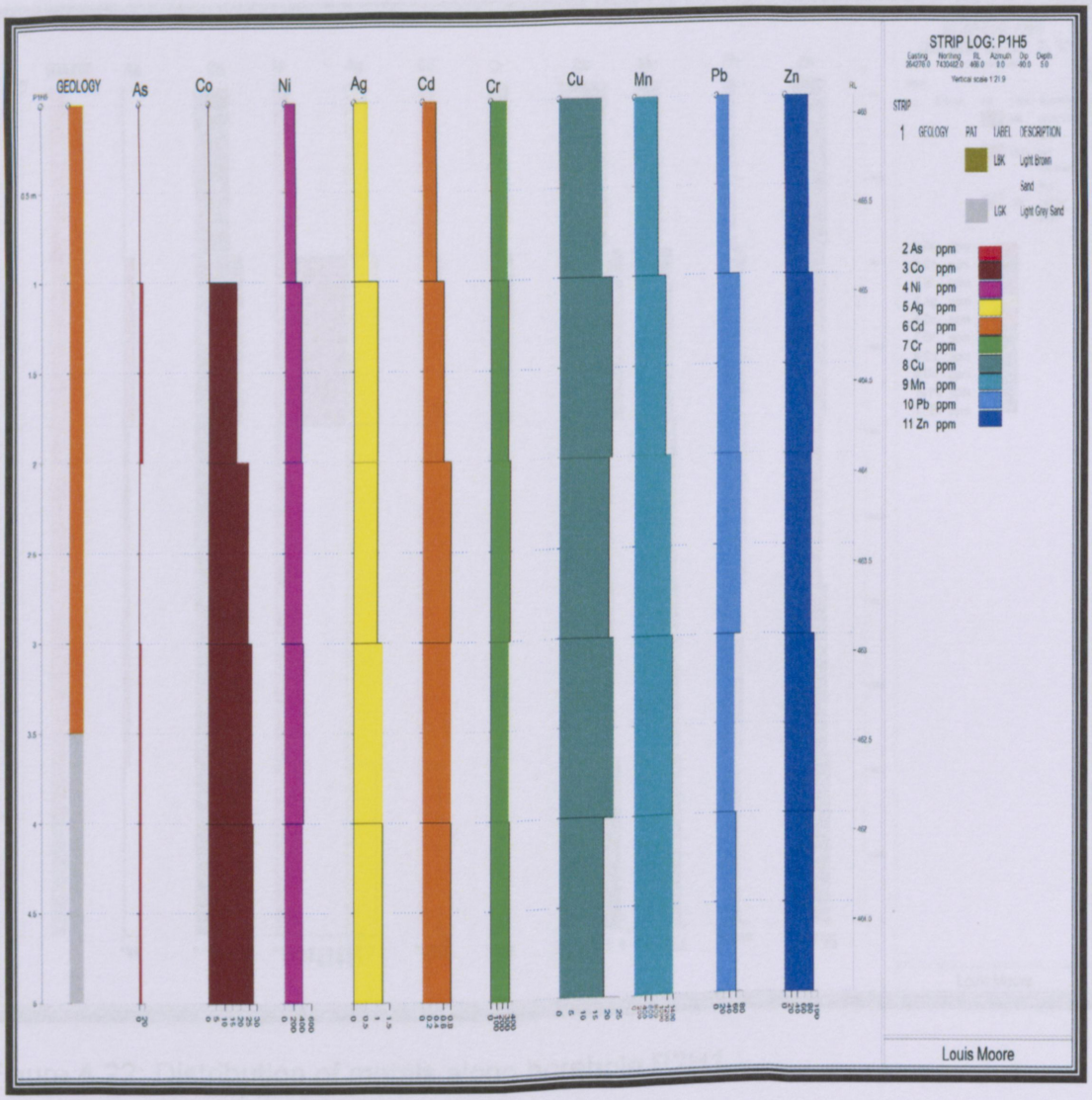


Figure 4.21: Distribution of metals along borehole P1H5.

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Borehole P2H1

Along this borehole, Cu, Pb and Zn showed erratic distribution from top to bottom (Fig.4.22), and at the other hand Co and Ag had showed more or less distribution pattern except from 1 to 2 m. Cr had indicated more or less uniform distribution and As also showed decrease distribution with depth. The increases distribution pattern with depth were being showed at Mn, Cd and Co.



Figure 4.22: Distribution of metals along borehole P2H1.

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Borehole P2H3

Borehole P2H3 showed more or less metals distribution pattern that were either decreasing with depth or increasing with depth (Fig.4.23). For example As, Ag, Mn and Cu increase with depth, whereas Cd and Cr decrease with depth. The exception to these trends were Zn, Pb and Co that registered erratic distribution. Ni showed uniform distribution pattern.



Figure 4.23: Distribution of metals along borehole P2H3.

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Borehole P3H2

Along borehole P3H2, As and Cu had showed the erratic distribution from top to bottom (Fig. 4.24). Ni and Cr has indicated more or less uniform distribution and Mn had also indicated More or less uniform except from 0 to 1 m. Co, Ag and Cd had showed increase distribution with depth whereas, Pb and Zn indicate decrease distribution with depth.



Figure 4.24: Distribution of metals along borehole P3H2.

The overall assessment of the distribution pattern of heavy metals at the Louis Moore tailings dam showed erratic distribution that may be attributed to the nature of tailings at the initial stage of tailings disposal. Louis Moore tailings dam has not matured enough to produce acid mine drainage like Klein Letaba tailings dam, hence there was no distinct boundary between the oxidation and reducing zones. The distribution of the metals literally

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varied from one drill hole to the other, although a general trend was also noted with some metals uniform in value, whereas others increase in value with depth. For example, along a number of drill holes, Mn, Co and Mn increase in value with depth, whereas Ni, Ag and Cr showed uniform with depth. Erratic distribution of metals was noticeable among Zn, Pb, Co and Cu.

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Occurrence of Au within the Louis Moore tailings

It was observed that the mean value of Au within the tailings was 314.50 µg\kg (Table 4.5). The minimum and maximum of Au within the tailings was 200 and 480 µg\kg respectively, with a range of 280 µg\kg

Table 4.4: Statistical analysis of gold within the tailings dam

	Au (µg\kg)
Number of samples	20
Minimum	200
Maximum	480
Range	280
Mean	314,50
Median	290
Standard deviation	76,81

4.8 Distribution Pattern of Au within the Louis Moore Tailings dam

The distribution of gold within the Louis Moore tailings dam was rather erratic (Fig. 4.25). For example, along the profiles P1H1, P1H3 and P1H5, there was a general increase of gold with depth, whereas along P1H2 and P1H4, gold values decreased with depth. The variation of gold values was observed at a depth of 2 m along this profile. The analysed depth, however, was within the oxidized zone of the dam. The gold distribution within the Louis Moore tailings dam along profile 2 was equally erratic (Fig.4.26). For example, there was a decrease in the value of gold below the 2 m mark along profile P2H1, P2H3 and P2H4. Au opposite trend was observed along profile P2H2. P2H2 increase, P2H3 decrease and P2H4 decrease with values.

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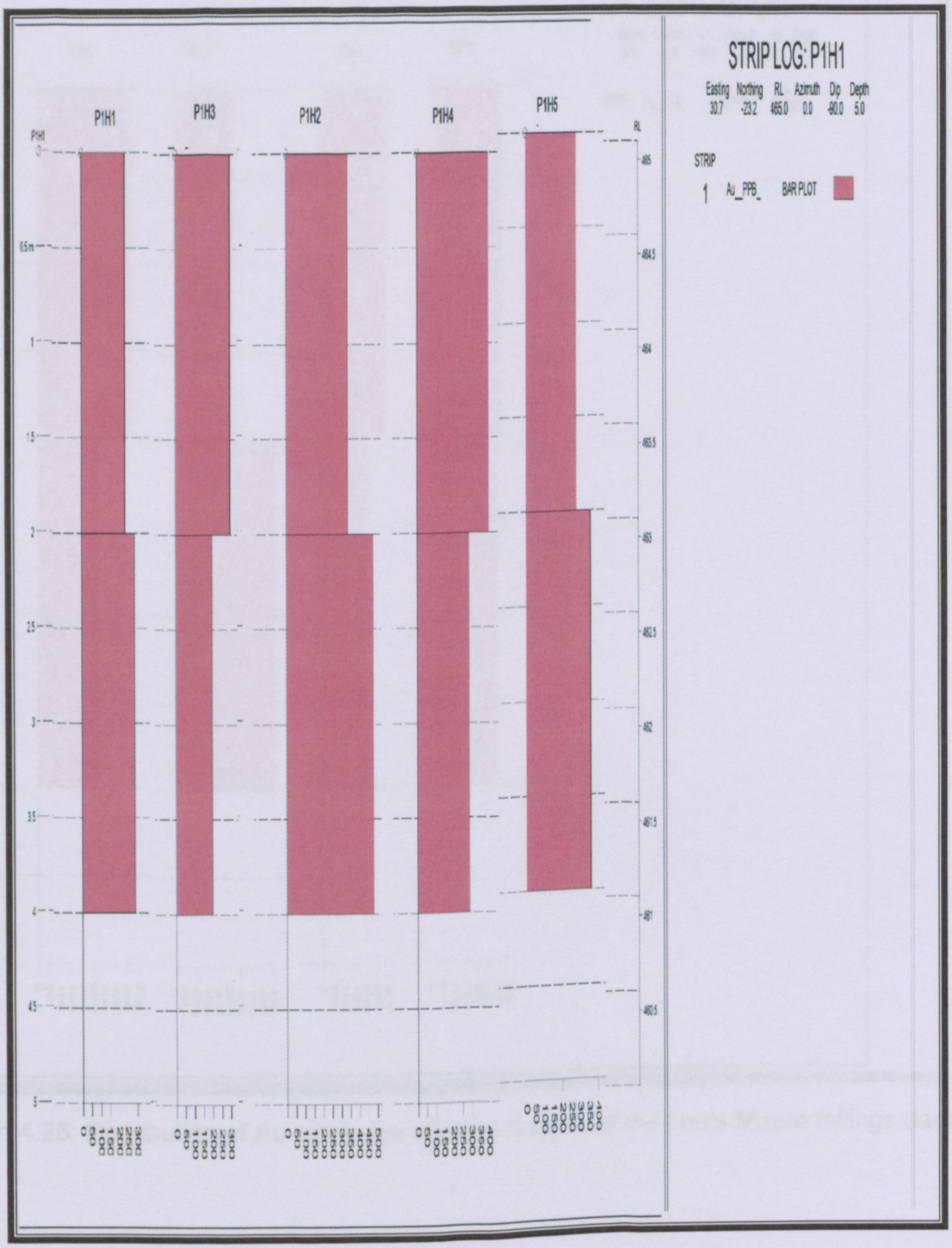


Figure 4.25: Distribution of Au within the upper 4 meters of the Louis Moore tailings dam.

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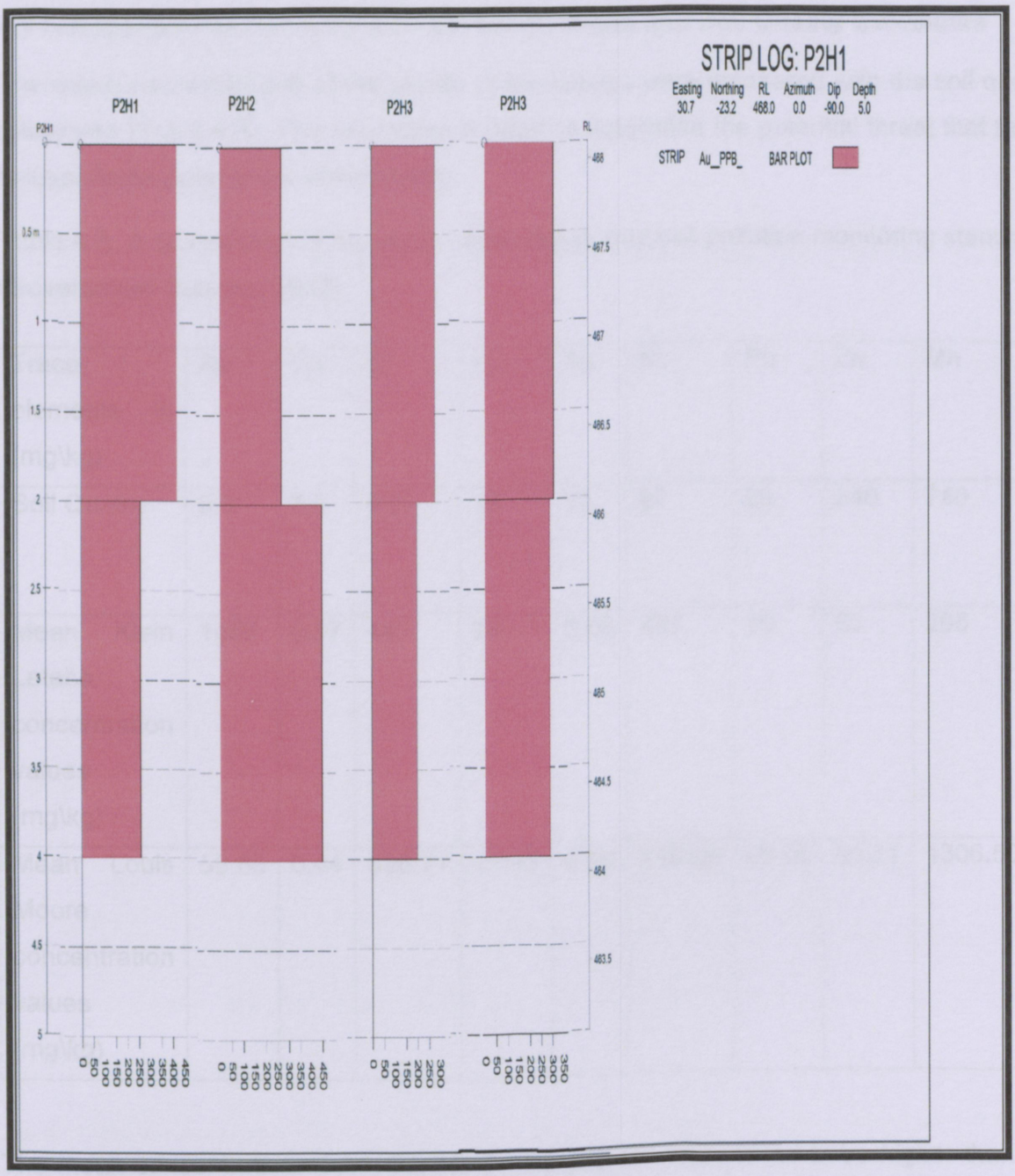


Figure 4.26: Distribution of Au within the upper 4 meters of the Louis Moore tailings dam.

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4.9 Comparison of Concentration Levels of Metals and Soil Quality Guidelines

The mean concentrations of the metals in the tailings were compared with the soil quality guidelines (Table 4.5). This was done in order to determine the potential threat that these tailings dams pose to the environment.

Table 4.5: A comparison of the mean metal values and soil pollution monitoring standards (Government Gazette, 2012)

Trace elements in (mg\kg)	As	Cd	Cr	Cu	Ag	Ni	Pb	Zn	Mn	Co
Soil Quality	5.8	7.5	6.5	16	10	91	20	240	740	300
Mean Klein Letaba concentration values (mg\kg)	1462	0.57	447	74	0.65	431	20	50	205	46
Mean Louis Moore concentration values (mg\kg)	59.65	0.44	335.71	21.43	0.98	415.95	60.08	93.21	1306.50	26.29

The mean concentration of As, Cu, Ni, Cr, Pb and Mn were found to be higher than the recommended soil pollution monitoring standards. This means that the Louis Moore mine and Klein Letaba tailings are a threat to the nearby land, farms and rivers to which the tailings are eroded through either wind or water erosion.

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4.10 Particle Size Distribution of Klein Letaba and Louis Moore Tailings

Data acquired from the sieve analysis (Tables 3.1 and 3.2) were used to determine the nature of the tailings.

Klein Letaba Tailings

Based on the particle sizes, the Klein Letaba tailings had silt, sand and gravel but without any clay (Fig. 4.27). Silt was predominant within the tailings.

The material was regarded to be cohesionless due to the absence of clay, and is expected to behave in a non–plastic manner.

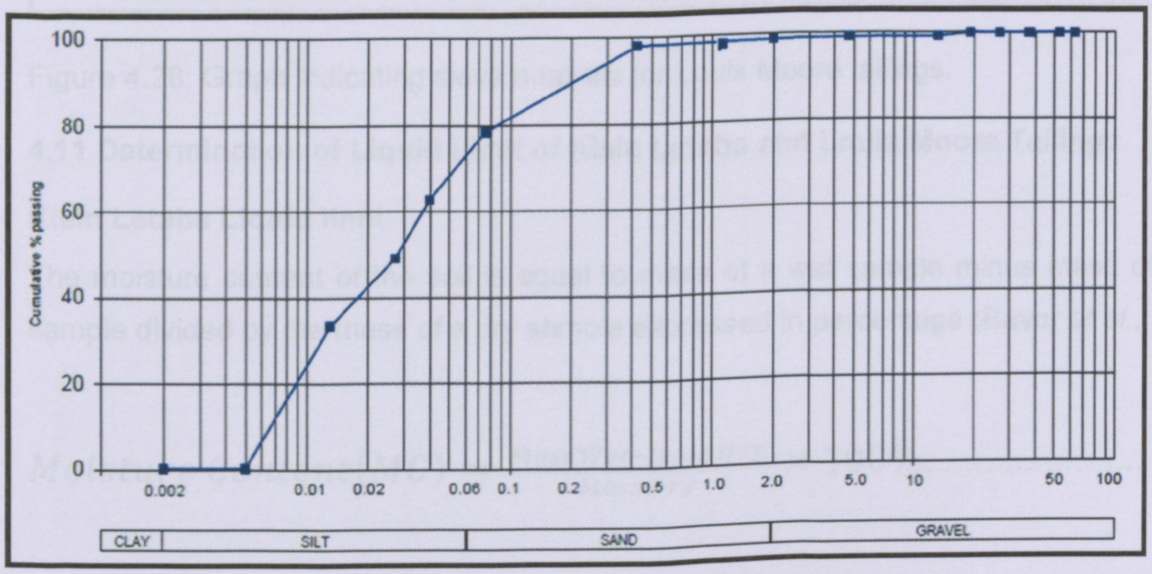


Figure 4.27: Graph indicating sieve analysis for Klein Letaba tailings.

Louis Moore Tailings

Based on the particle size, the Louis Moore tailings showed contained silt, sand and gravel but no clay (Fig. 4.28).

The material was regarded to be cohesionless due to the absence of clay, and is expected to behave in a non-plastic manner.

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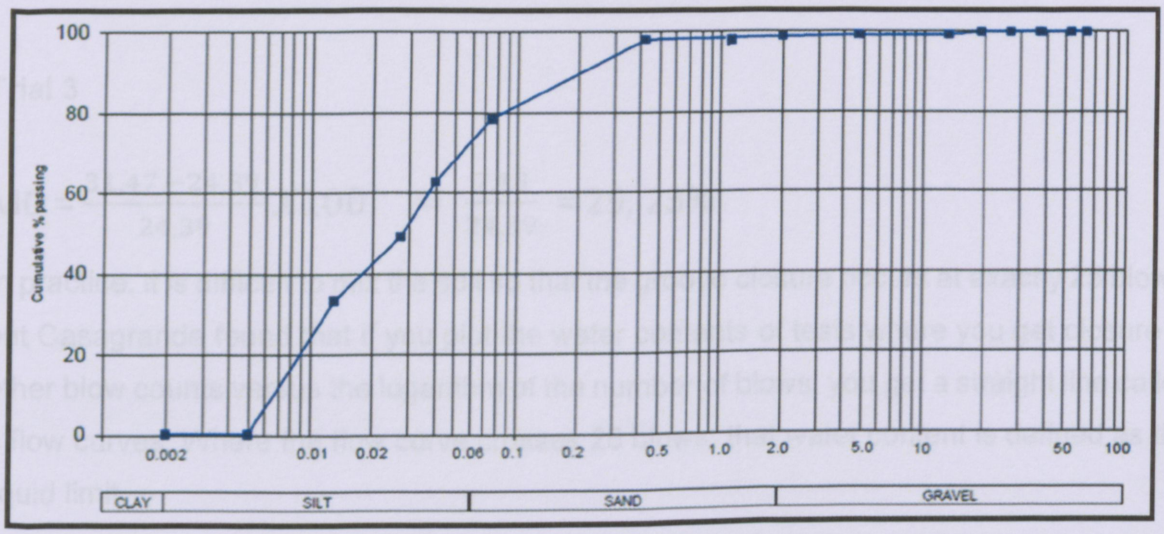


Figure 4.28: Graph indicating sieve analysis for Louis Moore tailings.

4.11 Determination of Liquid Limit of Klein Letaba and Louis Moore Tailings

Klein Letaba Liquid limit

The moisture content of the soil is equal to mass of a wet sample minus mass of a dry sample divided by the mass of a dry sample expressed in percentage (Baver *et al.*, 1972):

$$Moisture\ Content(MC) = \frac{Mass\ Wet - Mass\ Dry}{Mass\ Dry} \times 100\% \quad \text{..... (1)}$$

Determination of the, moisture content was based of three trials.

Trial 1

$$MC = \frac{41{,}26 - 32{,}45}{32{,}45} X100 \quad = \frac{8{,}81}{32{,}45} = 27,15\%$$

Trial 2

$$MC = \frac{44{,}63 - 34{,}88}{34{,}88} X100 \quad = \frac{9{,}75}{34{,}88} = 27,95\%$$

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Trial 3

$$MC = \frac{31,47 - 24,39}{24,39} X100 \quad = \frac{7,13}{24,39} = 29,23\%$$

In practice, it is difficult to mix the soil so that the groove closure occurs at exactly 25 blows, but Casagrande found that if you plot the water contents of tests where you get closure at other blow counts versus the logarithm of the number of blows, you get a straight line called a flow curves. Where the flow curve crosses 25 blows, that water content is defined as the liquid limit.

The results which were found for three trials were used to plot the liquid limit graph (Fig. 4.29). The black line representing a results of trial 1 and 2 indicate the liquid limit.

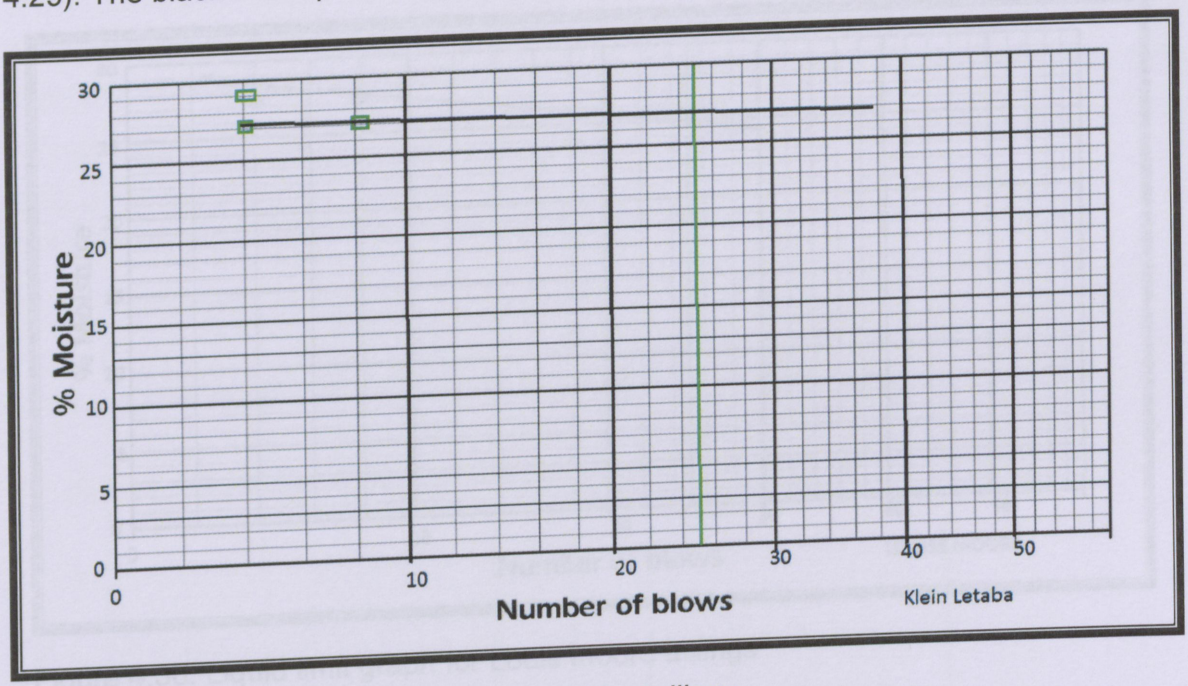


Figure 4.29: Liquid limit graph for Klein Letaba tailings.

Louis Moore Liquid limit

As in the above case, the moisture content of Louis Moore tailings was determined.

Trial 1

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$$MC = \frac{52,34-41,08}{41,08} X100 = \frac{11,26}{41,08} = 27,41\%$$

Trial 2

$$MC = \frac{45,86-35,92}{35,92} X100 = \frac{9,97}{35,92} = 27,76\%$$

Trial 3

$$MC = \frac{41,89-32,33}{32,33} X100 = \frac{9,56}{32,33} = 29,57\%$$

The results obtained from the three trial were used to plot the liquid limit graph (Fig. 4.30). In this case, however, the results from trial 1 and 3 were used to determine the liquid limit of tailings.

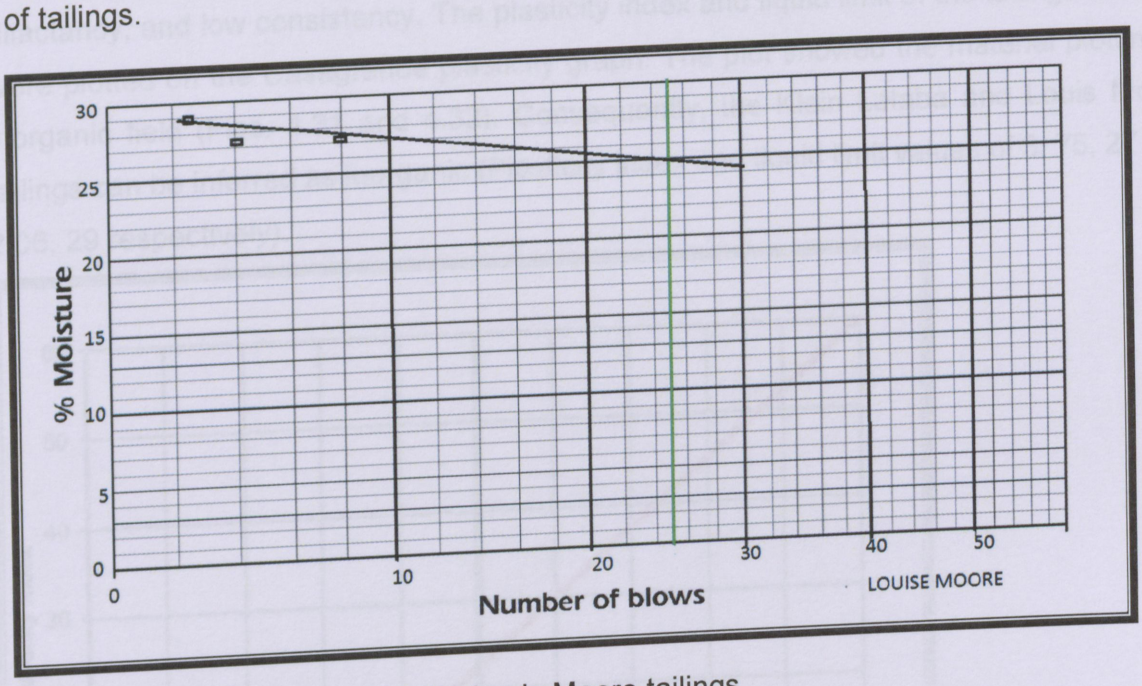


Figure 4.30: Liquid limit graph for Louis Moore tailings

4. 12 Determination of Plasticity Index of Klein Letaba and Louis Moore Tailings

Plasticity index = LL – PL, where LL is liquid limit and PL is plastic limit

Consequently, the plasticity index of Klein Letaba and Louis Moore tailings were found respectively to be:

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a) Plasticity index = 27 – 23, 25 = 3.75., and

b) Plasticity index = 25 – 22, 94 = 2.06.

The plasticity index for both tailings showed that the materials were slightly plastic.

4.13 Index Classification of Klein Letaba and Louis Moore Tailings

The Casagrande plasticity chart was used to determine the nature of tailings. (Fig. 4.33). The chart indicated that the tailings comprised inorganic silt (Casagrande, 1938), indicated as unified classification (ML). Based on AASHTO classification, the soil is classified under A-4 which is poor silty. Based on the results, material has low or no cohesive strength for the use in brick making due to the absence of clay content. According to Holtz and Kovacs (1981), this class of soil is characterized by none to slight dry strength, quick to slow dilactancy, and low consistency. The plasticity index and liquid limit of the tailings material were plotted on the Casagrande plasticity graph. The plot showed the material plotting in inorganic field (Figs. 4.31 and 4.32). Consequently, the Klein Letaba and Louis Moore tailings can be inferred as inorganic (Plasticity index and liquid limit values of 3. 75, 27 and 2.06, 29 respectively).



Figure 4.31: Plasticity graph for Klein Letaba tailings.

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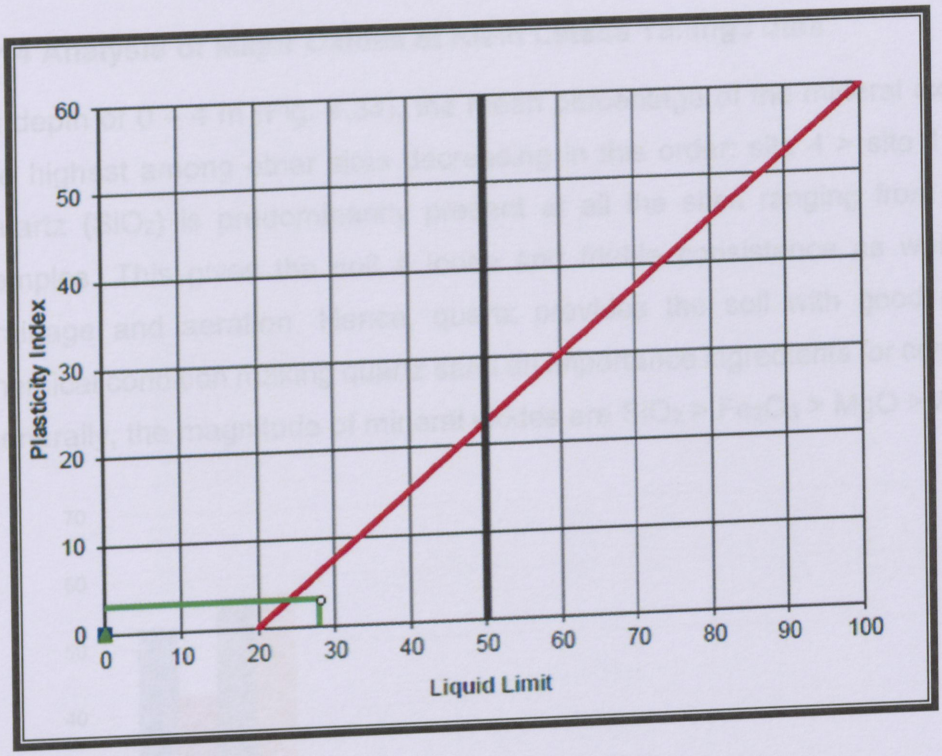


Figure 4.32: Plasticity chart for Louis Moore tailings.

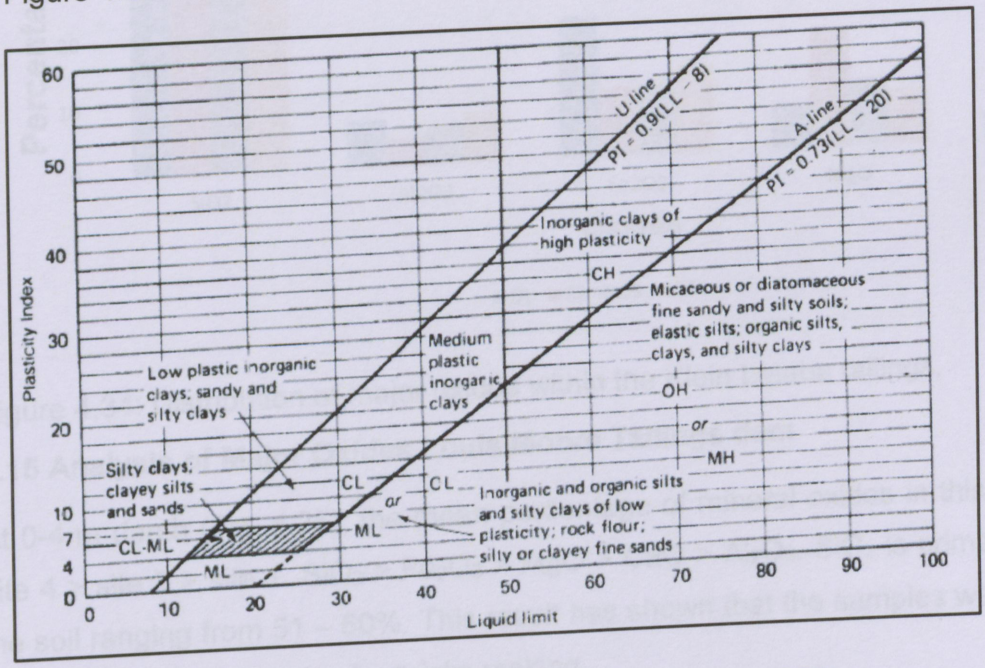


Figure 4.33: Casagrande plasticity chart. (Casagrande, 1938).

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4.14 Analysis of Major Oxides at Klein Letaba Tailings dam

At depth of 0 – 4 m (Fig. 4.34), the mean percentage of the mineral oxides at site 4 was the highest among other sites decreasing in this order: site 4 > site 1 > site 3 > site 2. Quartz (SiO_2) is predominantly present at all the sites ranging from 42 – 58% in the samples. This gives the soil a loose and friable consistence as well as permeability, drainage and aeration. Hence, quartz provides the soil with good physical but poor chemical condition making quartz sand an importance ingredients for concrete (Tan, 2009). Generally, the magnitude of mineral oxides are SiO_2 > Fe_2O_3 > MgO > Al_2O_3 > CaO.



Figure 4.34: Distribution of major oxides within the Klein Letaba tailings.

4.15 Analysis of Major Oxides Louis Moore Tailings dam

At 0-4 m depth (Fig. 4.35), the mean percentage of mineral oxides in this order; site 2 > site 4 > site 3 > site 1. SiO_2 > Fe_2O_3 > MgO > CaO > Al_2O_3. SiO_2 is primarily dominating the soil ranging from 51 – 60%. This result has shown that the samples were rich in SiO_2, which has good properties for bricks making.

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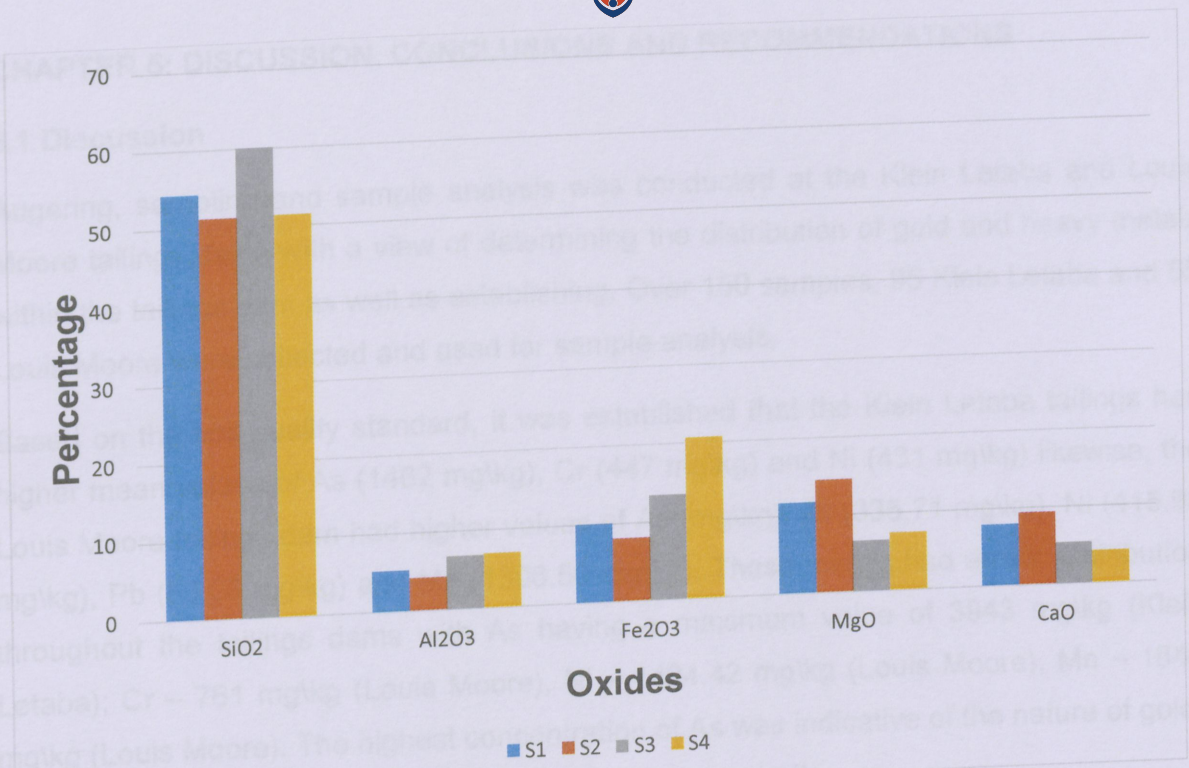


Figure 4. 35. Distribution of major oxides within the Louis Moore tailings.

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CHAPTER 5: DISCUSSION, CONCLUSIONS AND RECOMMENDATIONS

5.1 Discussion

Augering, sampling and sample analysis was conducted at the Klein Letaba and Louis Moore tailings dams with a view of determining the distribution of gold and heavy metals within the tailings dam as well as establishing. Over 150 samples, 95 Klein Letaba and 55 Louis Moore were collected and used for sample analysis.

Based on the soil quality standard, it was established that the Klein Letaba tailings had higher mean values of As (1462 mg\kg), Cr (447 mg\kg) and Ni (431 mg\kg) likewise, the Louis Moore tailings dam had higher values of As (mg\kg), Cr (335.71 mg\kg), Ni (415.95 mg\kg), Pb (50.08 mg\kg) and Mn (1306.50 mg\kg). These metals had erratic distribution throughout the tailings dams with As having a maximum value of 3943 mg\kg (Klein Letaba); Cr – 761 mg\kg (Louis Moore), Ni – 1494.42 mg\kg (Louis Moore), Mn – 1848 mg\kg (Louis Moore). The highest concentration of As was indicative of the nature of gold-sulphide mineralisation that was associated with arsenopyrite.

The tonnage of tailings was used to deduce the total amount of Au, As, Ag, Cu, Co, Mn, Ni, Cd, Cr, Zn and Pb contained in the material. The tonnage of the Klein Letaba dam was found to be 1,493172 tons, and the values of As, Cr, Mn and Cu at this dam was calculated and found to be 2183, 667, 306 and 110 tons respectively. The amount of gold within this tailings dam was 699 kg.

The statistical summary of the AAS results of Louise Moore tailings dam showed the mean values of the metals as follows: As - 59.65 mg\kg; Cr – 335.71 mg\kg; Mn – 1306.50 mg\kg; and Cu – 21.43 mg\kg. At Louis Moore tailings dam, it was not possible to calculate the gold tonnage due to the strong weathering of the material. However, the mean gold value was found to be 314.50 µg\kg with maximum value of 480 µg\kg.

The distribution pattern of the heavy metals along profiles was not always uniform for each metal. The distribution pattern of As was more or less uniform along P1H5, with low values, but was rather erratic along profiles P2H3 and P3H2 with higher values. The distribution pattern of Co, Ni, Ag, Cd, Cr, Cu, Mn, Pb and Zn was quite consistent with depth apart from minor variations. The values of Ni, Ag and Cu along profile P2H1, at a depth of 1-2 m was

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rather high. Co and Ag registered negligible values from 0 - 1 m along profile P1H5 and in the case of As also along P2H1. The erratic distribution patterns of the metals can be attributed to the nature of tailings at the time of disposal to the dam. The oxidation zone was noted to be between 0 – 4 m and it was characterised by the light brown material. The transition zone, however, was not identified as below 4 m, the tailings was grey in colour, apparently reflecting the original material from ore processing.

Sieve analysis revealed that the tailings were composed mainly of sandy material with some silt, but devoid of clay. However, the material was slightly plastic, thus was under unified classification ML which is inorganic silt.

Based on the AASHTO classification, the tailings were classified under A-4 which is poor silty material, thus was found to have no-cohesive strength for bricks making due to lack of clay. According to Holtz and Kovas (1981), this classification of material is characterized by non to slight dry strength, quick to slow dilactancy and low consistency.

5. 2 Conclusions

- The study established the abundance of As, Cr, Ni and Mn contained in the Klein Letaba tailings which had maximum values of 3943 mg\kg, 759 mg\kg, 731 mg\kg and 383 mg\kg respectively. Gold had a mean value of 468 µg\kg and the highest value of 740 µg\kg, and at Louis Moore tailings dam, gold had a mean value of 314.50 µg\kg and highest value of 480 µg\kg.
- The amount of gold within the Klein Letaba tailings dam was found to be 699 kg, but the distribution of gold was rather erratic.
- The tailings dams were found to contain mainly sandy material with some silt, hence are not good for bricks making. However, with addition of clay material and coal waste in right proportions good quality bricks were produced.

5. 3 Recommendation

The study recommends the reprocessing of gold within the dams, a process that should be accompanied with the use of tailings to produce bricks and solve environmental issues associated with tailings pollution of the environment through wind and water erosion and to

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eliminate the potential of mine water generation. For tailings to be used for bricks making further research work is recommended to investigate the possibility of Cr speciation within the tailings, especially the conversion of Cr (III) to Cr (VI). Future work should also focus on the quality of coal dust as an additive to tailings for brick making.

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Appendix A

Klein Letaba Tailings Dam Drill Holes

Tailings logging along drill hole P1A1

From (m)	To (m)	Colour	Description
0	1	Mixture of orange with grey	Tailings fine and soft with some loose material at top
1	1.2	Mixture of grey with orange	Tailings soft with orange nodules
1.2	2.1	Mixture of orange with grey	Thin, hard layer
2.1	2.4	Brown	Tailings soft and moist
2.4	8	Grey sand	Tailings uniform without colour change

Tailings logging along drill hole P1A2

From (m)	To (m)	Colour	Description
0	0.5	Mixture of orange with grey	Tailings fine and soft with loose material at the top
0.5	1.2	Mixture of grey with orange	Tailings soft with orange nodules
1.2	8	Grey sand	Tailings uniform without colour change

Tailings logging along drill hole P1A3

From (m)	To (m)	Colour	Description
0	1	Orange sand	Tailings soft
1	8	Grey sand	Tailings uniform without colour change

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Tailings logging along drill hole P1A4

From (m)	To (m)	Colour	Description
0	0.5	Orange sand	Tailings soft
0.5	1	Mixture of orange with grey	Tailings fine and soft with loose material
1	1.2	Mixture of grey with orange	Tailings soft with orange nodules
1.2	8	Grey sand	Tailings uniform without colour change

Tailings logging along drill hole P2A1

From (m)	To (m)	Colour	Description
0	1.2	Mixture of orange with grey	Tailings fine and soft with loose material
1.2	1.6	Mixture of grey with orange	Tailings soft with orange nodules
1.6	1.8	Mixture of orange with grey	Tailings fine and soft with loose material
1.8	2.4	brown	
2.4	26	Mixture of orange with grey	Tailings fine and soft with loose material
2.6	3.2	brown	Tailing soft
3.2	8	Grey sand	Tailings uniform without colour change

Tailings logging along drill hole P2A2

From (m)	To (m)	Colour	Description
0	**0.2**	Mixture of orange with grey	Tailings fine and soft with loose material
0.5	2.6	Grey	Tailings soft
2.6	3	Mixture of orange with grey	Thin, hard layer
3	8	Grey	Tailings soft and rather uniform

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Tailings logging along drill hole P2A3

From (m)	To (m)	Colour	Description
0	1	Orange sand	Tailings fine and soft
1	1.5	Grey	Tailings soft
1.5	2		Tailings fine and soft
2	8	Grey	Tailings soft

Tailings logging along drill hole P2A4

From (m)	To (m)	Colour	Description
0	0.5	Mixture of orange and grey	Tailings fine and soft with loose material
0.5	1	Brown	Tailings soft and rather uniform
1	8	Grey	Tailings soft

Tailings logging along drill hole P3A1

From (m)	To (m)	Colour	Description
0	1.2	Orange sand	Tailings fine and soft with loose material
1.2	8	Grey	Tailings soft

Tailings logging along drill hole P3A2

From (m)	To (m)	Colour	Description
0	1	Orange sand	Tailings fine and soft with loose material
1	8	Grey	Tailings soft and rather uniform

Tailings logging along drill hole P3A3

From (m)	To (m)	Colour	Description
0	0.8	Orange sand	Tailings fine and soft with loose material
0.8	8	Grey	Tailings soft and rather uniform

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Tailings logging along drill hole P3A4

From (m)	To (m)	Colour	Description
0	0.7	Orange sand	Tailings fine and soft with loose material
0.7	8	Grey	Tailings soft

Appendix B

Louis Moore Tailings Dam Drill Holes

Tailings logging along drill hole P1H1

From (m)	To (m)	Colour	Description
0	4	Light brown	Tailings dry
4	4.3	Light grey with brown	Tailings muddy with hard layer
4.3	7	Light grey	Tailings moist

Tailings logging along drill hole P1H2

From (m)	To (m)	Colour	Description
0	3.2	Light brown	Tailings dry
3.2	5	Light grey	Tailings moist

Tailings logging along drill hole P1H3

From (m)	To (m)	Colour	Description
0	3.8	Light brown	Tailings dry
3.8	5	Light grey	Tailings hard layer

Tailings logging along drill hole P1H4

From (m)	To (m)	Colour	Description
0	3.5	Light brown	Tailings dry
3.5	4.8	Light grey	Tailings muddy with hard layer
4.8	5	Dark grey	Tailings moist

Tailings logging along drill hole P1H5

From (m)	To (m)	Colour	Description
0	3.5	Light brown	Tailings dry
3.5	5	Light grey	Tailings muddy with hard layer

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Tailings logging along drill hole P2H1

From (m)	To (m)	Colour	Description
0	4.3	Light brown	Tailings dry
4.3	5	Light grey	Tailings muddy with hard layer

Tailings logging along drill hole P2H2

From (m)	To (m)	Colour	Description
0	3.2	Light brown	Tailings dry
3.2	5	Light grey	Tailings muddy with hard layer

Tailings logging along drill hole P2H3

From (m)	To (m)	Colour	Description
0	3.7	Light brown	Tailings dry
3.7	5	Light grey	Tailings with hard layer

Tailings logging along drill hole P2H4

From (m)	To (m)	Colour	Description
0	3.8	Light brown	Tailings dry
3.8	4.8	Light grey	Tailings hard layer
4.8	5	Dark grey	Tailings hard

Tailings logging along drill hole P1H1

From (m)	To (m)	Colour	Description
0	4.3	Light brown	Tailings dry
4.3	5	Light grey	Light grey

Tailings logging along drill hole P1H2

From (m)	To (m)	Colour	Description
0	3.4	Light brown	Tailings dry
3.4	5	Light grey	Light grey

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Appendix C

Gold values in Klein Letaba tailings

Sample no	Sample name	ppb Au
1	P1A1	450
2	P1A1	500
3	P1A1	143
4	P1A1	500
5	P1A1	530
6	P1A1	345
7	P1A2	390
8	P1A2	420
9	P1A2	444
10	P1A2	316
11	P1A2	347
12	P1A2	278
13	P1A3	460
14	P1A3	430
15	P1A4	700
16	P1A4	270
17	P1A4	268
18	P1A4	455
19	P1A4	466
20	P1A4	482
21	P2A1	500
22	P2A1	450
23	P2A1	356
24	P2A1	351
25	P2A1	364
26	P2A1	358
27	P2A2	450

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28	P2A2	650
29	P2A3	210
30	P2A3	330
31	P2A3	459
32	P2A3	477
33	P2A3	488
34	P2A3	512
35	P2A4	400
36	P2A4	390
37	P2A4	600
38	P2A4	529
39	P2A4	593
40	P2A4	595
41	P3A1	740
42	P3A1	250
43	P3A1	252
44	P3A1	317
45	P3A1	680
46	P3A1	276
47	P3A2	570
48	P3A2	590
49	P3A2	354
50	P3A2	306
51	P3A2	335
52	P3A2	372
53	P3A3	510
54	P3A3	500
55	P3A4	600
56	P3A4	480
57	P3A4	221

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58	P3A4	250
59	P3A4	221
60	P3A4	223

Gold values of Louis Moore

Sample number	sample I.D	ppb Au
1	P1H1S1	240
2	P1H1S2	290
3	P1H2S3	300
4	P1H2S4	200
5	P1H3S5	340
6	P1H3S6	480
7	P1H4S7	400
8	P1H4S8	290
9	P1H5S9	280
10	P1H5S10	360
11	P2H1S11	420
12	P2H2S12	260
13	P2H3S13	270
14	P2H3S14	450
15	P2H4S15	280
16	P2H4S16	200
17	P3H1S17	310
18	P3H1S18	290
19	P3H2S19	360
20	P3H2S20	270

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Heavy metals values of Klein Letaba

Sample I.D	ppm Cu	ppm Cd	ppm Pb	Ppm Cr	ppm Mn	ppm Ag	ppm Zn	ppm Ni	ppm Co	ppm As
P1A1S1	53	0.79	12	55	176	0.79	20	93	15	590
P1A1S2	82	0.79	15	144	172	0.99	33	114	18	1213
P1A1S3	53	0.99	36	16	141	0.59	65	603	56	2993
P1A1S4	87	0.99	22	1	248	0.99	36	303	41	939
P1A1S5	82	1.00	22	2	220	1.00	36	328	43	952
P1A1S6	82	0.79	10	2	202	0.79	26	451	44	1584
P1A1S7	85	0.79	12	7	207	0.79	34	391	44	1045
P1A2S8	97	0.98	17	2	233	0.98	47	423	80	1261
P1A2S9	81	1.00	17	1	297	0.80	45	394	54	1105
P1A2S10	91	0.79	20	1	231	0.98	46	274	37	1056
P1A2S11	93	0.60	19	4	245	1.00	41	283	37	862
P1A2S12	84	1.00	28	2	233	0.80	34	321	37	885
P1A2S13	78	0.59	10	2	183	0.79	26	385	39	1218
P1A2S14	85	0.79	14	2	271	0.79	35	400	47	989
P1A2S15	63	0.60	21	2	186	0.79	49	476	47	1495
P1A3S16	88	0.80	16	1	185	1.00	30	187	27	955
P1A3S17	71	0.59	13	2	186	0.79	41	362	48	1127
P1A3S18	79	0.99	21	3	283	0.99	36	332	39	922
P1A3S19	81	0.79	13	2	206	0.79	32	369	41	1267
P1A3S20	62	0.60	12	3	185	0.80	30	381	42	787
P1A3S21	58	0.60	22	2	150	0.60	67	525	48	1726
P1A3S22	55	0.59	30	2	145	0.59	82	569	51	1540
P1A4S23	59	0.80	20	1	158	1.00	21	63	10	1126
P1A4S24	86	0.99	18	4	155	0.99	30	131	21	335
P1A4S25	85	1.58	127	2	284	1.19	5	383	45	1274
P1A4S26	91	1.00	18	1	233	0.80	4	382	49	1139
P1A4S27	57	1.00	20	13	199	0.80	3	394	48	1732
P1A4S28	50	0.79	23	10	149	0.60	30	543	52	2170
P1A4S29	48	0.80	33	2	149	0.60	32	587	55	829
P1A4S30	49	0.79	46	2	138	0.59	33	598	56	2915
P1A4S31	52	0.79	39	1	150	0.59	49	539	53	2616
P2A1S32	95	0.59	3	204	186	2.78	29	93	11	486
P2A1S33	101	0.59	3	133	214	1.18	34	376	57	410
P2A1S34	73	0.40	10	347	221	0.99	71	633	68	1360
P2A1S35	62	0.40	7	393	185	0.60	66	574	50	1022
P2A1S36	52	0.40	13	400	163	0.40	103	663	58	1228
P2A1S37	52	0.40	12	457	157	0.40	90	642	59	1276

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P2A1S38	48	0.40	13	355	147	0.40	87	647	59	1279
P2A1S39	48	0.40	9	269	149	0.40	78	643	63	1447
P2A2S40	81	0.39	4	152	270	1.18	31	228	30	499
P2A2S41	99	0.40	4	460	259	0. 79	36	231	30	598
P2A2S42	61	0.39	13	646	171	0.59	72	616	61	1455
P2A2S43	51	0.40	11	669	168	0.40	85	731	76	1577
P2A2S44	50	0.40	11	759	164	0.40	94	647	61	1382
P2A2S45	51	0.40	8	632	145	0.40	72	649	63	1485
P2A2S46	50	0.20	9	632	150	0.40	74	637	63	1461
P2A2S47	55	0.40	11	658	150	0.40	81	676	65	1405
P2A3S48	99	0.60	6	474	332	0.60	45	253	31	585
P2A3S49	98	0.40	4	471	253	0.60	35	244	26	544
P2A3S50	89	0.40	5	500	282	0.60	32	325	32	479
P2A2S51	86	0.40	3	559	206	0.60	22	422	40	717
P2A3S52	94	0.40	8	564	292	0.60	30	544	55	438
P2A3S53	73	0.39	27	464	114	0.59	66	458	61	1760
P2A3S54	41	0.40	29	468	143	0.40	74	398	33	1692
P2A3S55	61	0.40	40	522	190	0.40	79	533	50	2277
P2A4S56	92	0.40	5	430	250	0.80	33	183	24	597
P2A4S57	95	0.39	4	443	235	0.59	30	261	33	537
P2A4S58	86	0.60	6	473	249	0.60	29	330	33	502
P2A4S59	87	0.40	4	496	247	0.59	25	326	34	597
P2A4S60	84	0.39	4	513	272	0.59	27	361	42	475
P2A4S61	69	0.40	6	606	224	0.40	51	540	50	773
P2A4S62	62	0.59	8	680	182	0.39	73	593	54	960
P2A4S63	50	0.40	14	617	166	0.40	82	698	71	1405
P3A1S64	65	0.60	12	302	150	0.60	22	77	15	531
P3A1S65	76	0.60	15	198	322	0.60	36	374	53	631
P3A1S66	51	0.40	46	567	185	0.40	79	584	60	3704
P3A1S67	47	0.40	52	645	195	0.40	113	622	62	3202
P3A1S68	54	0.40	42	508	156	0.40	101	604	55	3094
P3A1S69	45	0.40	55	540	157	0.40	93	581	43	3337
P3A1S70	46	0.40	44	635	165	0.40	92	569	51	3284
P3A1S71	46	0.40	48	597	160	0.40	91	638	59	3337
P3A2S72	54	0.60	15	234	174	0.60	24	140	24	1189
P3A2S73	66	0.59	17	349	303	0.59	33	286	42	1301
P3A2S74	87	0.40	23	460	205	0.40	42	362	43	2469
P3A2S75	58	0.40	38	551	169	0.40	71	586	55	3943
P3A2S76	56	0.40	44	567	155	0.40	88	596	55	3117
P3A2S77	61	0.40	44	566	153	0.40	87	632	58	3300
P3A2S78	57	0.40	40	567	150	0.40	82	614	56	3627

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P3A2S79	62	0.40	41	624	147	0.40	86	629	55	3287
P3A3S80	116	0.60	19	306	181	0.60	32	131	18	1000
P3A3S81	102	0.59	17	337	304	0.59	33	362	50	1578
P3A3S82	115	0.60	25	439	350	0.60	35	305	37	731
P3A3S83	105	0.59	15	494	235	0.59	25	388	42	1307
P3A3S84	84	0.59	16	399	225	0.40	29	406	43	878
P3A3S85	69	0.40	16	545	191	0.40	48	537	48	1595
P3A3S86	75	0.40	32	541	177	0.40	74	543	49	2260
P3A3S87	70	0.40	28	503	179	0.40	66	532	49	2226
P3A4S88	100	0.40	17	269	157	0.59	22	68	11	952
P3A4S89	138	0.59	16	308	383	0.79	42	469	71	1313
P3A3S90	103	0.60	16	352	268	0.60	31	286	38	819
P3A4S91	101	0.40	17	450	273	0.60	26	342	38	1196
P3A4S92	103	0.40	12	397	258	0.40	30	425	45	993
P3A4S93	82	0.40	15	441	218	0.60	35	423	44	1322
P3A4S94	61	0.40	18	436	179	0.40	61	477	45	1962
P3A4S95	70	0.40	24	494	193	0.40	69	598	52	2081

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Heavy metals values of Louis Moore

Sample ID	Cu	Co	As	Pb	Cd	Ag	Cr	Ni	Mn	Zn
P1H1S1	22	28	26	70	0,39	0,59	331	387	1 280	100
P1H1S2	18	28	<4	60	0,40	0,60	344	405	1 484	90
P1H1S3	22	31	18	63	0,40	0,59	350	415	1 451	106
P1H1S4	21	29	<4	55	0,39	0,59	325	394	1 619	89
P1H1S5	28	35	14	69	0,39	0,79	388	473	1 849	91
P1H2S6	21	25	34	64	0,40	0,40	363	365	1 030	79
P1H2S7	22	28	<4	51	0,40	0,60	360	381	1 376	91
P1H2S8	22	31	<4	62	0,60	0,80	377	420	1 413	102
P1H2S9	21	29	<4	53	0,59	0,79	310	426	1 444	103
P1H2S10	19	27	<4	55	0,40	0,79	330	388	1 444	77
P1H3S11	17	29	30	46	0,20	0,40	330	334	922	75
P1H3S12	20	24	20	69	0,20	0,39	346	398	1 405	74
P1H3S13	22	27	36	55	0,40	0,59	323	397	1 375	94
P1H3S14	22	25	10	54	0,39	0,59	265	368	1 087	80
P1H3S15	25	23	22	51	0,40	0,60	332	394	1 525	75
P1H4S16	20	32	<4	69	0,59	0,59	319	283	882	63
P1H4S17	22	21	14	58	0,79	0,99	333	406	1281	92
P1H4S18	20	29	20	60	0,80	1,19	327	393	1360	84
P1H4S19	19	28	<4	55	0,80	1,20	334	404	1358	92
P1H4S20	22	28	<4	51	0,79	1,18	313	371	1461	74
P1H5S21	18	1	<4	38	0,39	0,59	282	233	836	71
P1H5S22	22	17	12	70	0,59	0,99	300	373	1121	84
P1H5S23	21	24	<4	74	0,80	1,00	338	403	1306	80
P1H5S24	22	27	8	55	0,79	1,19	309	422	1373	90
P1H5S25	18	28	8	60	0,80	1,20	329	403	1382	88
P2H1S26	19	19	<4	37	0,39	0,79	327	306	845	97
P2H1S27	25	28	30	64	0,60	1,20	378	1494	1186	93
P2H1S28	21	25	14	67	0,59	1,19	331	397	1299	83
P2H1S29	21	27	16	58	0,79	1,18	349	404	1403	96
P2H1S30	24	30	<4	47	0,60	1,20	337	438	1489	105
P2H2S31	23	22	24	58	0,20	1,00	315	343	1019	112
P2H2S32	25	27	26	56	0,60	1,19	316	411	1257	101
P2H2S33	22	28	28	65	0,79	0,99	372	426	1415	95
P2H2S34	22	27	<4	51	0,79	0,99	761	398	1373	106
P2H2S35	24	28	14	53	0,79	1,19	294	412	1384	96

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P2H3S36	22	27	71	55	0,79	1,18	382	422	1126	97
P2H3S37	17	24	57	67	0,59	0,99	318	410	1285	78
P2H3S38	19	28	104	52	0,79	1,18	330	424	1375	112
P2H3S39	24	27	122	208	0,39	1,58	269	339	1409	111
P2H3S40	25	30	134	45	0,20	1,38	306	442	1404	89
P2H4S41	23	26	155	62	0,40	1,19	337	395	1213	101
P2H4S42	16	26	186	62	0,40	1,40	335	429	1318	94
P2H4S43	20	30	28	50	0,40	1,19	321	449	1381	113
P2H4S44	24	27	34	50	0,20	1,19	325	446	1430	91
P2H4S45	23	27	48	48	0,40	1,19	301	418	1379	70
P3H1S46	21	20	86	46	0,20	0,80	319	311	844	71
P3H1S47	21	26	90	52	0,20	1,00	334	381	1113	83
P3H1S48	19	25	76	66	0,20	1,19	323	418	1238	76
P3H1S49	19	27	94	52	0,39	1,18	337	415	1258	86
P3H1S50	21	28	87	56	0,20	1,19	310	426	1334	124
P3H2S51	26	24	107	70	0,20	0,99	363	386	1683	149
P3H2S52	23	27	149	71	0,20	1,19	307	424	1237	130
P3H2S53	20	27	138	70	0,39	1,18	309	424	1348	118
P3H2S54	21	28	142	56	0,20	1,20	312	438	1351	129
P3H2S55	25	29	119	46	-1,99	1,19	288	414	1397	77

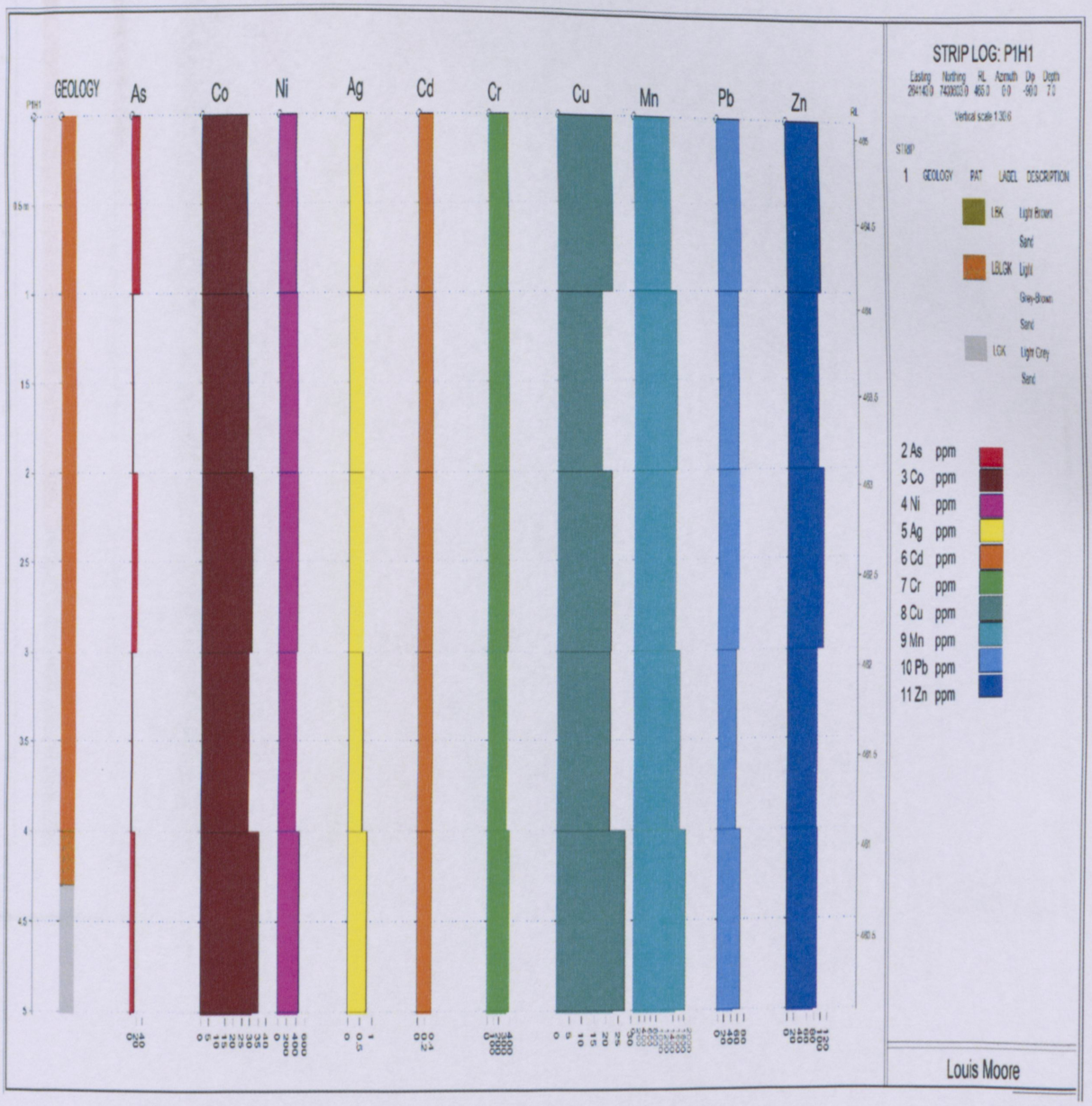
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Appendix D

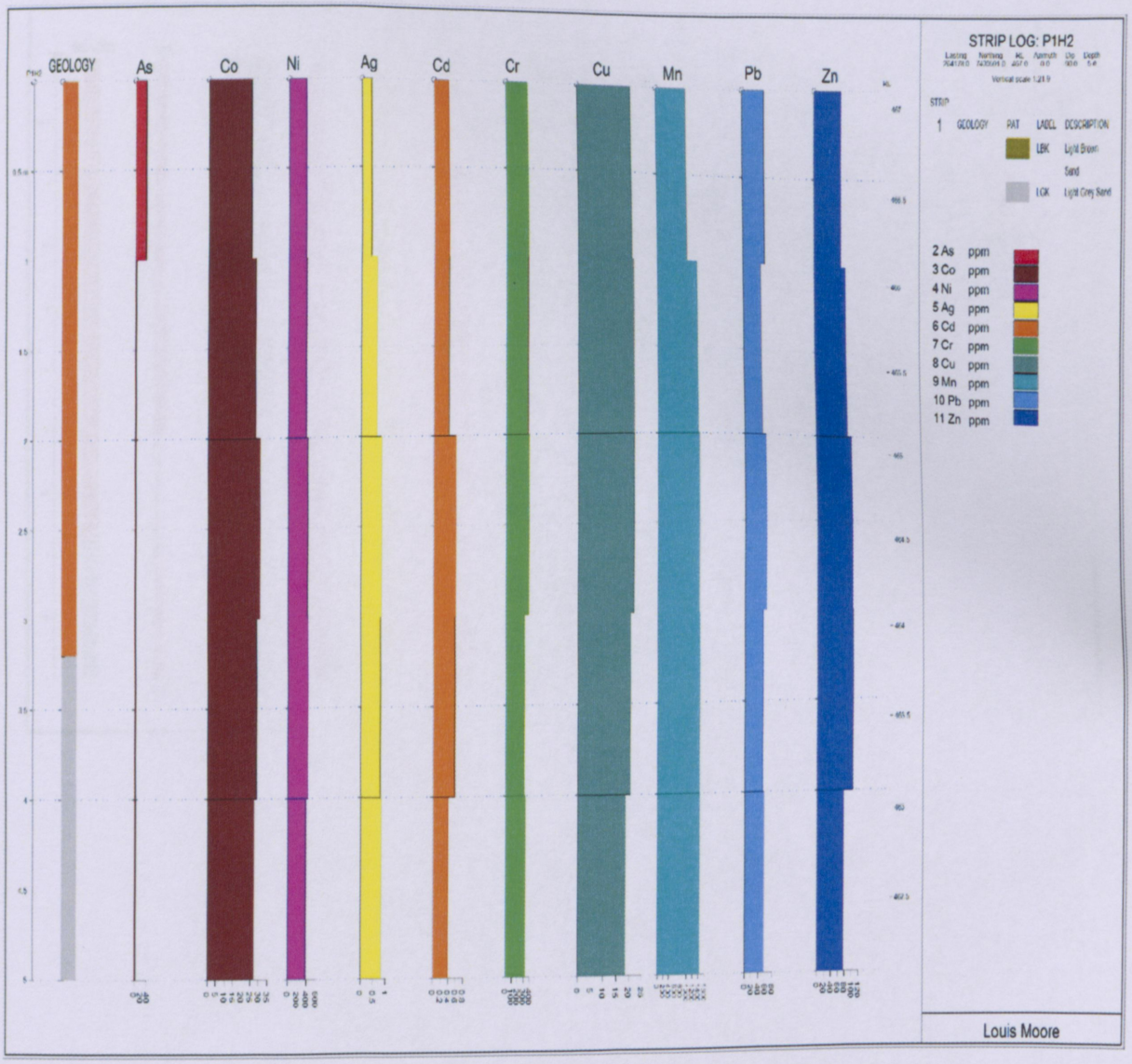
Distribution of metals with depth of Louis Moore Tailings dam.

P1H1



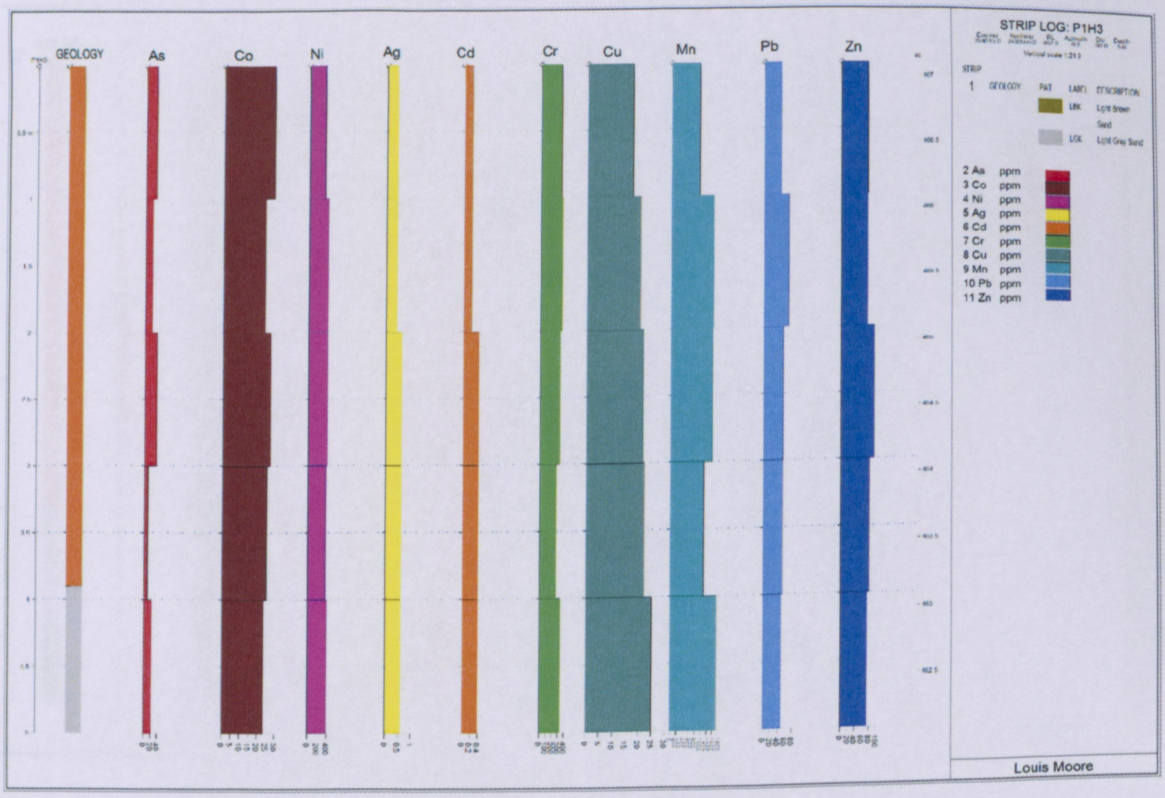
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P1H2



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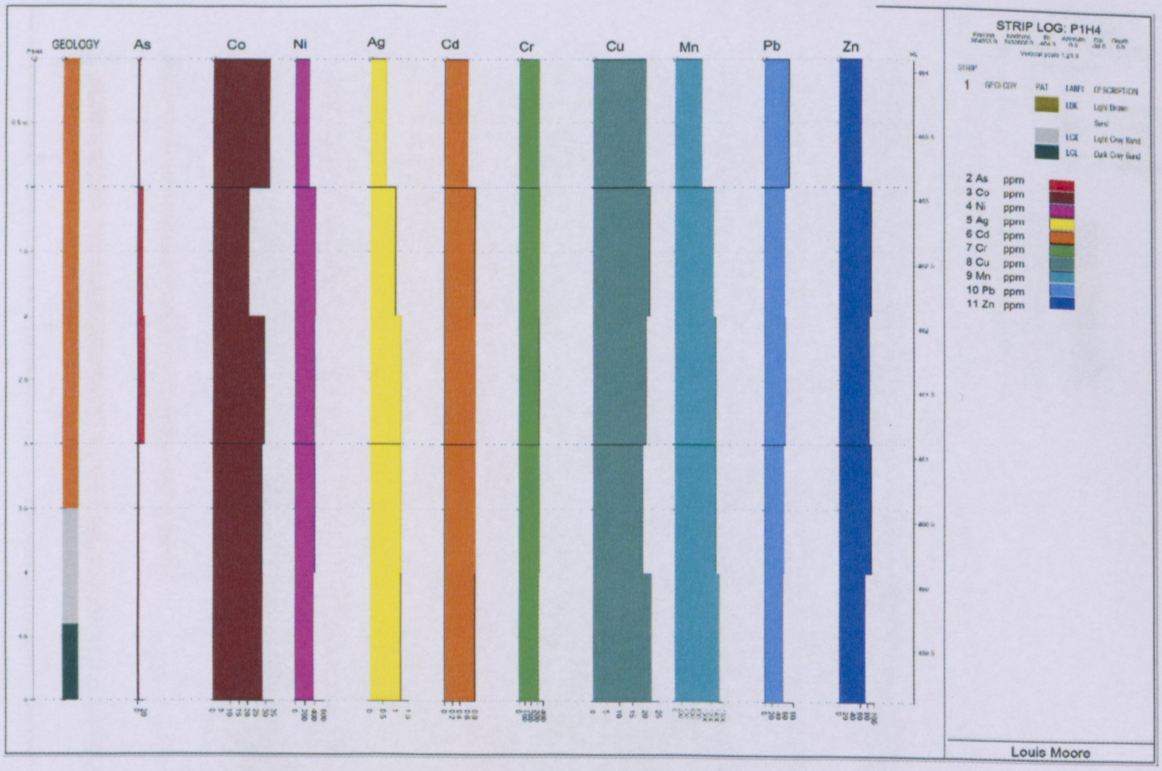
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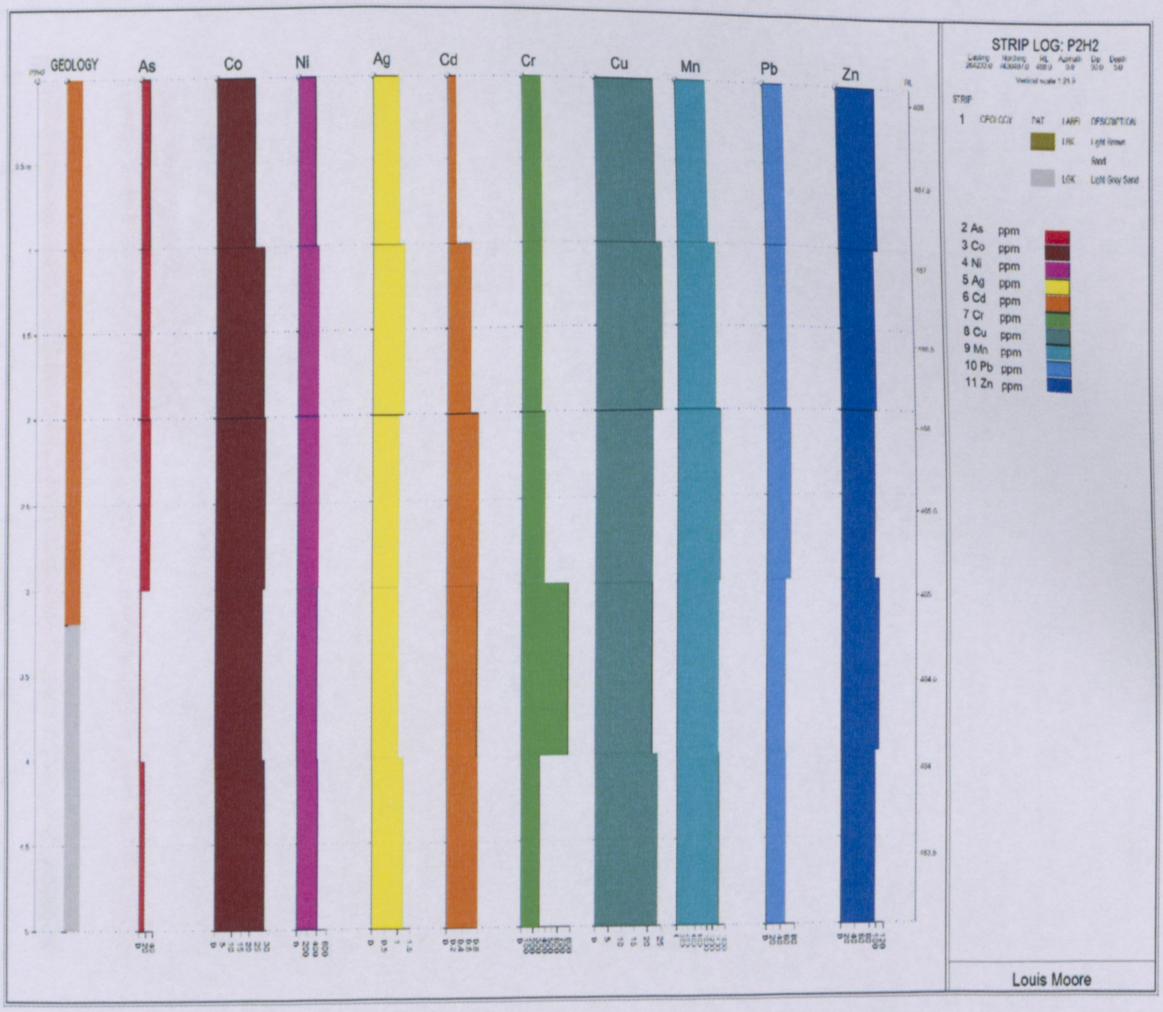
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P1H4



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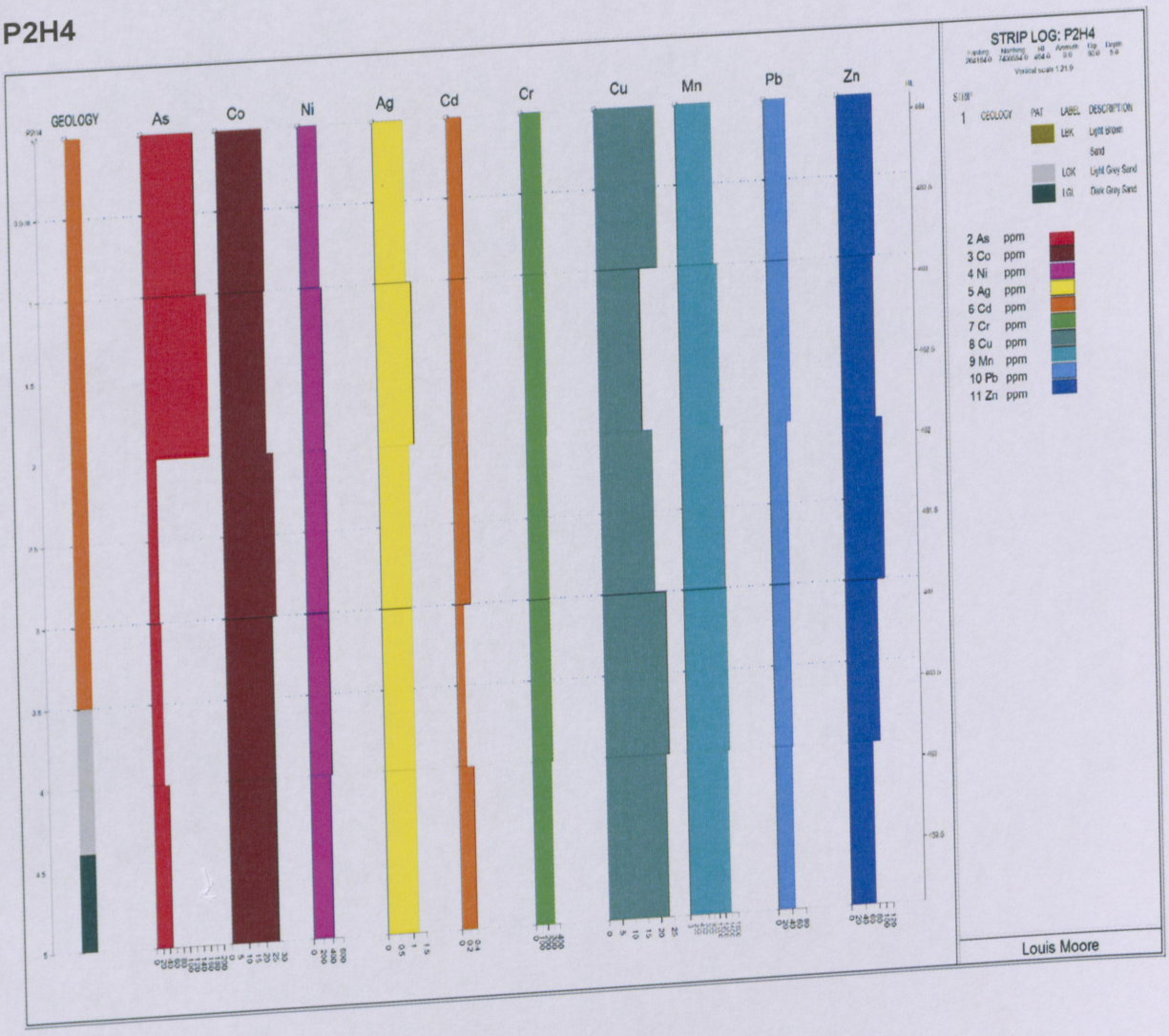
P2H2



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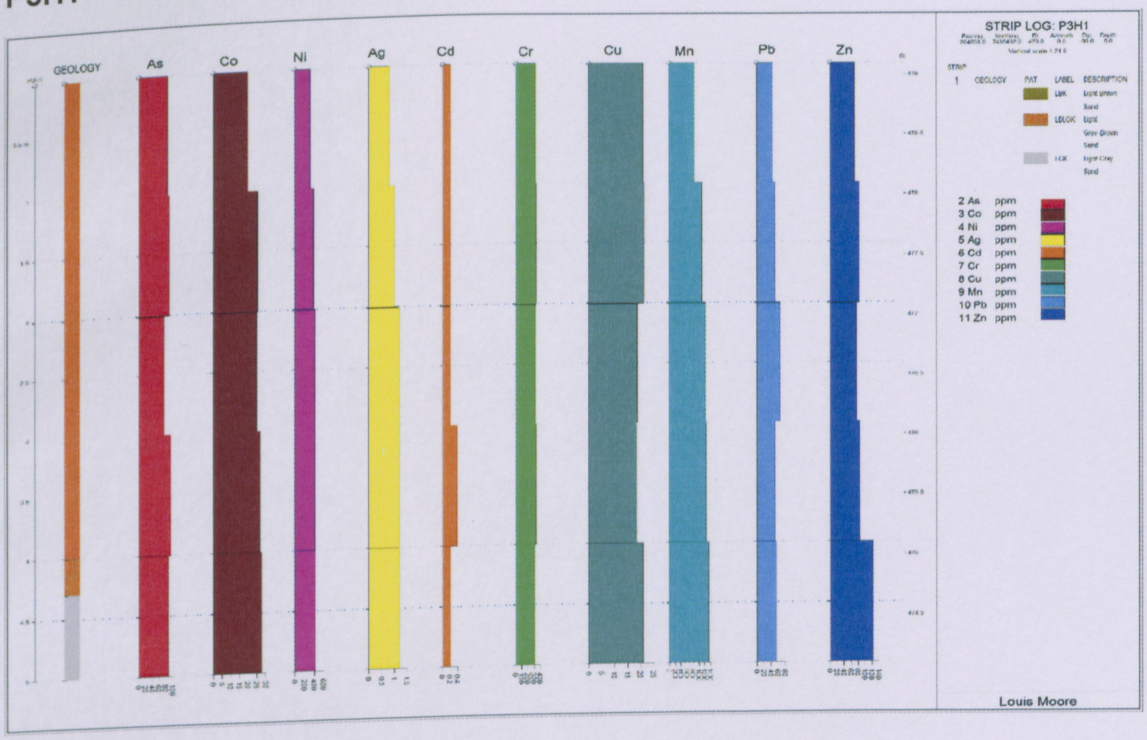
P2H4



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P3H1



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