University of Venda
Creating Future Leaders

SCHOOL OF ENVIRONMENTAL SCIENCES

DEPARTMENT OF MINING AND ENVIRONMENTAL GEOLOGY

ECONOMIC POTENTIAL OF GOLD MINE WASTE: A CASE STUDY OF CONSOLIDATED MURCHISON MINE WASTE

BY

RAVELE REMBULUWANI SOLLY
STUDENT NUMBER: 11624350

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

SIGNATURES:

STUDENT: MR. R. S. RAVELE ……...

SUPERVISOR: PROF. J. S. OGOLA ……....

CO-SUPERVISOR: MS. H. R. MUNDALAMO ……

JULY 2019

© University of Venda

DECLARATION

I, **Ravele Rembuluwani Solly**, hereby declare that this dissertation submitted to the Department of Mining and Environmental Geology, at the University of Venda, for Master of Earth Sciences in Mining and Environmental Geology, is my original work and 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.

Undersigned,

Student's signature: ……... Date: …….

© University of Venda

DEDICATION

I would like to dedicate this dissertation to my grandfather Mr. Ndivhudzannyi Gerson Ravele for being more than just a grandfather, but also playing a father figure in my life and always advising me about the importance of school. May your beloved soul rest in perfect peace papa. I am dedicating this work to my mother Rudzani Jane Ravele who passed away when I was still young. May your beloved soul rest in perfect peace mom.

© University of Venda

ACKNOWLEDGEMENT

First of all I would like to thank God who afforded me this opportunity to further my studies this far. My special gratitude to Consolidated Murchison Mine that allowed me to conduct my research on their mine wastes. I would like to thank Mr. Vincent Mashoene, Mr. Antonia Mduduzi and Mr. Trinity who provided me with relevant information for my study and assisted in conducting field work. I would take this opportunity to express my special gratitude to my Supervisors, Prof. J.S. Ogola and Ms. H. R. Mundalamo who did a delightful job in assisting me from the beginning to the end of this work by giving me the instructions, guidelines and corrections when writing this dissertation. I would like to acknowledge NRF-CIMERA for funding my research work. I would like to thank my grandmother, Mrs. Shonisani Rossie Ravele and my aunts, Tshifhiwa Judith Ravele, Fulufhelo Irene Ravele, Lufuno Olive Ravele and Matodzi Daphney Masevhe, for assisting me financially and emotionally to further my studies. My special appreciation to Mr. S. Mukatuni and Mr. V. L. Netshimbupfe for their assistance with fieldwork and laboratory work, your exertion was noticed and highly appreciated. Last but not least, I would like to thank all my friends and colleagues Mr. K. I. Mphephu, Mr. P. Mudzanani, Mr. M. P. Mavhona and Ms. P.G. Munyai, just to mention a few, for their delightful support and words of encouragement.

© University of Venda

ABSTRACT

The increase in the demand and market price of gold has led to reprocessing of gold tailings in many parts of the world. Mines are recently closing down due to depletion of resources and increasing mining costs leading to the reprocessing of old tailings dams. The cost of rehabilitation is high, and therefore a more convenient way of rehabilitation is required. The most convenient strategy identified here was to reprocess tailings for gold and use waste rocks as construction materials. The tailings residues (waste remaining after reprocessing) will be relocated to a more convenient place to avoid pollution. Gold reprocessing from tailings dams has gained momentum in South Africa especially in the Witwatersrand Basin where there are large volumes of tailings. Gold is being reprocessed from tailings in this area using hydraulic monitors.

This study focused on the evaluation of gold and heavy metals within the tailings at Consolidated Murchison Mine tailings in Gravelotte, Limpopo province. Augering was conducted over the tailings up to a depth of 8 m along four sampling Profiles. The first profile had two sampling points, the second profile with three sampling points, the third and fourth profiles consisted of four and five sampling points respectively. Samples were collected at 1 m interval, therefore a total of 112 samples were collected and analysed for heavy metals using X-Ray Fluorescence spectrometry and 84 samples were analysed for gold using fire assaying.

Tailings sampling was accompanied with tailings logging, taking note of colour, texture and moisture content. Based on this, the oxidation status of the tailings dam was determined. Oxidation zone of this tailings dam was mainly from top down to a depth of 3 m. The transitional zone was not identified, hence after the oxidation zone, the rest was unoxidized zone. This study established that gold was erratically distributed within the tailings dam with the lowest and highest values of 200 mg/kg and 1880 mg/kg respectively and the average was 670 mg/kg. The tonnage of tailings within the dam was found to be 13 280 310 tons with a total gold amount of 8 897. 81 kg. At the current world market, this interprets to US$ 306 932 396.00 (R 4 281 706 924.20). It was concluded that this tailings dam is economically viable for reprocessing, although previous studies have indicated that it is not possible to extract gold from tailings dams completely. The heavy metal content of Pb, Ni and Cr were found to be high with average values of (ppm); 5631.5, 2062.6 and 1345

© University of Venda

respectively. The metals with the lowest values were Cd, Co and Cu, averaging (ppm); 0.01 ppm, 19.8 ppm and 42.1 ppm respectively. Heavy metal content in soil around the tailings dam was gradually decreasing with distance from the tailings dam.

Waste rocks have been used in some parts of the world as sub-base material for engineering construction, hence in this study, a total of 6 waste rock samples were collected using grab sampling method for geostatistical investigation. Such samples were subjected to various geotechnical tests which included particle size distribution analysis (sieve analysis), Atterberg limit tests and laboratory compaction test to determine their suitability for construction. The waste rock material was found to be suitable for road construction as it was classified under Group A-1-a using the AASHTO classification system. The material consisted mainly of rock fragments, gravel and sand material with minor silt/clay. In general, Consolidated Murchison mine waste was found to be suitable for road construction.

Keywords: Tailings dams, gold reprocessing, heavy metals, waste rock, engineering construction

© University of Venda

University of Venda
Creating Future Leaders

TABLE OF CONTENT

© University of Venda

© University of Venda

© University of Venda

APPENDICES

© University of Venda

LIST OF FIGURES

© University of Venda

© University of Venda

© University of Venda

LIST OF TABLES

© University of Venda

LIST OF ACRONYMS AND SYMBOLS

AAC	Anglo-American Corporation of South Africa
AASHTO	American Association of State Highway and Transport Official classification
ALS	Australian Laboratory Service
AMD	Acid Mine Drainage
As	Arsenic
Au	Gold
BAW	Beach above water
BGB	Barberton Greenstone Belt
CB	California Bearing Ratio
Cd	Cadmium
CIL	Carbon-in-Leach
CIP	Carbon-in-Pulp
Co	Cobalt
Cr	Chromium
Cu	Copper
ERGO	East Rand Gold Company
FeS_2	Pyrite
GPS	Global Positioning System
JCI	Johannesburg Consolidated Investment
JMS	Joint Metallurgical Scheme
LL	Liquid Limit
MGB	Murchison Greenstone Belt
Ni	Nickel

© University of Venda

OPC	Portland cement
Pb	Lead
PGEs	Platinum Group Elements
PH	Potential Hydrogen
PL	Plastic Limit
PMC	Phalaborwa Mining Company
RMMM	Rand Mines Milling and Mining Company
TTP	To The Point
USCS	Unified Soil Classification System
VMR	Village Main Reef
WHO	World Health Organization
XRF	X-ray fluorescence
Zn	Zinc

© University of Venda

CHAPTER ONE: INTRODUCTION

1.1 Background

The Murchison Greenstone Belt (MGB) is one of the Archaean greenstone belts situated on the Kaapvaal Craton of Southern Africa known for producing gold and antimony. Gold was discovered in the early 1900s where mining activities began (Willson and Viljoen, 1986). Geologically, the belt is located on the northeast section of the Kaapvaal Craton which is approximately 200 km north of the Barberton Greenstone Belt (BGB) and about 40 km away from the Limpopo Belt.

The price of gold has been increasing over the past years in the world market and thus tailings material that was considered as waste in the past can now be valuable (Viljoen, 2009). Reprocessing of tailings dams has gained momentum because of the low operational costs and low labour requirements involved. Reprocessing of tailings dams also lead to rehabilitation of such dams through relocating tailings residues into a more convenient place or using tailings residues as construction material. It is important to ascertain the values of gold within the tailings before reprocessing to determine the economic potential of such tailings.

Tailings are associated with environmental problems such as Acid Mine Drainage (AMD) (Xinyi, 2012). Tailings contain heavy metals and sulphides which end up in the environment through wind erosion, soil erosion or being carried by water either through runoff or infiltration and bioaccumulate in food chain. This means that both plants, animals and human beings are affected by such heavy metal pollution. It is also important to identify a more convenient area were tailings residues will be relocated/deposited after reprocessing for gold to avoid such environmental impacts. Waste rock rocks are used for construction purpose to minimize the large volumes of waste from underground operations.

Gold and antimony have been mined at Gravelotte by Consolidated Murchison Mining Company for more than 80 years, living large volumes of tailings and waste rock in place. No work has been done on the tailings dam and waste rock dump of this mine. This study focused on the evaluation of tailings and waste rock at Consolidated Murchison Mine to determine their economic viability and the extent of tailings impact on the environment.

© University of Venda

1.2 Study area

The study area is described fully in terms of geographical location, climate, topography, soil and land use.

1.2.1 Location

Consolidated Murchison mine is located approximately 10 km east-north-east of Gravelotte which is approximately 50 km away from the Phalaborwa town in Limpopo province, South Africa (Fig. 1.1). Geographically, the mine lies at approximately 23°54'04'' S and 30°41'15'' E. It is located within the Ba-Phalaborwa municipality in Mopani district.

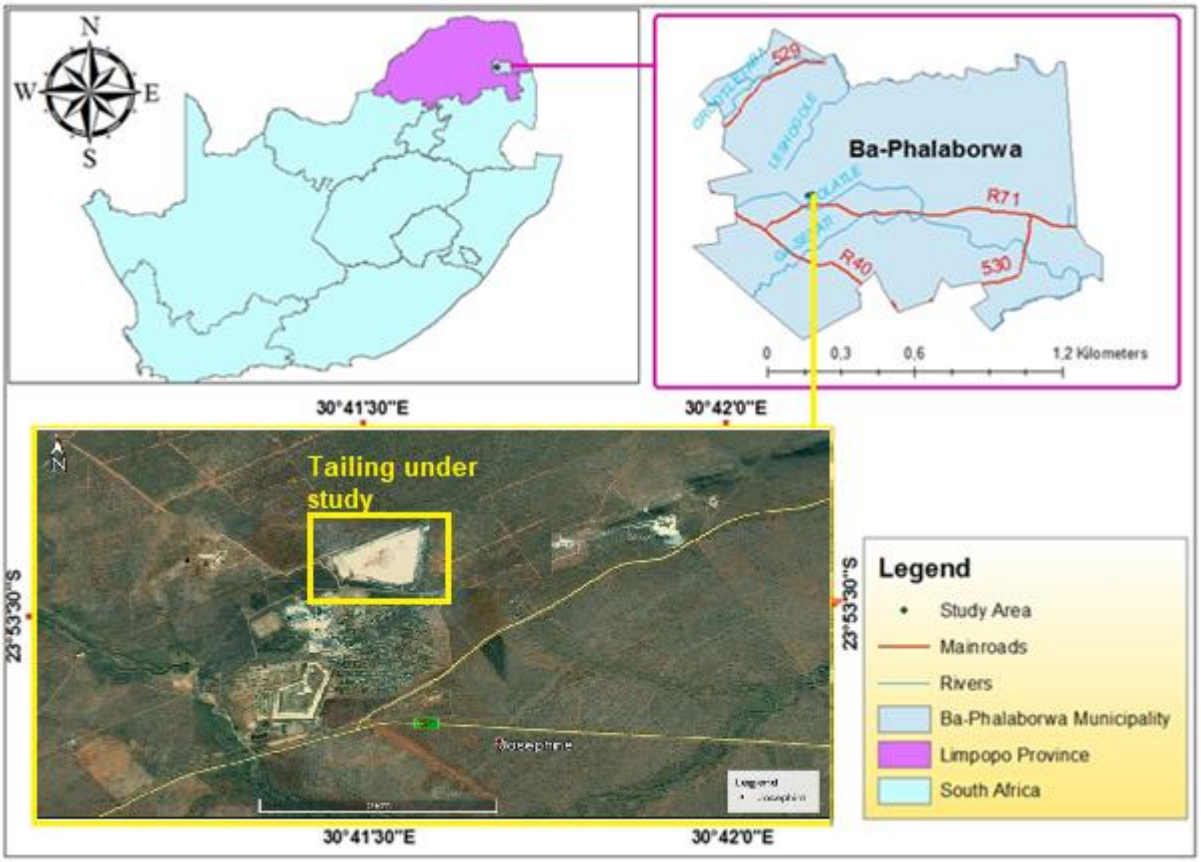

Figure 1.1: Map showing the location of Consolidated Murchison Mine tailings dam (Esri, 2018).

1.2.2 Climate

This is a sub-tropical climate region with temperatures ranging from 23°C to 35°C and even higher. Gravelotte normally receives rain mainly during mid-summer which is approximately 429 mm of rain per year. Lowest rainfall (0 mm) in this area occurs in June with highest rainfall (94 mm) occurring in December. The average midday temperatures in Gravelotte from June to January range from 23.2°C to 30.3°C

© University of Venda

respectively. Coldest temperatures in this region are experienced in July which can drop to 7.1° C.

1.2.3 Topography and drainage

This area is situated about 405 m above sea level. It consists of an undulating topography with the occurrence of some natural kopjes and drainage features that makes the development to be unsuitable. It is within the Olifants river primary catchment area and the tertiary catchment water which is shed between the Letaba river and Ga-Selati river taken along the topographical ridge line. Surface hydrology with some of the flood plains of small drainage systems poses a risk for development. This area consists of low groundwater yields of poor quality.

1.2.4 Soil

Generally, this area consists of sandy soil and is poor in nutrients and not fairly suitable for crop production. All land developments require appropriate geotechnical investigations to determine recommended foundation specifications. Soil forms occurring within this area are Glenrosa and Mispah soil forms. The high lying areas at the western part also consists of some red-yellow apedal soil forms.

1.2.5 Land-use

Land-use in this area is mainly mining. This include underground copper mining operations at Phalaborwa by Phalaborwa Mining Company (PMC) and the expansion of rock phosphate production by FOSKOR. This also include the mining of gold and antimony at Gravelotte by Consolidated Murchison Mining Company. Small scale mining also exists in the Murchison sequence near Gravelotte where antimony and emerald is being mined. This area also consists of game reserves such as Kruger National park and Selati Game reserve.

1.3 Problem statement

Gold has been mined at the Gravelotte deposit in the past, resulting in large volumes of tailings and waste rocks. No work has been done on such mine wastes to determine their impacts on the environment, hence there is need to ascertain the amount of gold present within the tailings and determine the possibility of using such mine waste as construction material. Tailings dams and waste rock dumps are associated with various environmental impacts which requires special attention to manage and prevent such impacts from occurring.

© University of Venda

1.4 Justification

Reprocessing of tailings is more economic because of the relatively low operational costs involved as compared to mining the reef. Recent improvements in processing technology together with the increase in the price of gold justify the need to reprocess such tailings. Tailings dams are associated with environmental hazards such as acid mine drainage. Using tailings, residues and waste rocks as construction material will help minimize these large volumes and hence rehabilitation will be taking place.

The reprocessing of the tailings has gained momentum in many parts of the world. This shows that tailings reprocessing is the future of the mining industry. Tailings reprocessing has been proven to be productive in the East Rand and is now being adopted in the West Rand (Nel, 2008). Reprocessing of gold tailings has been found to be productive at Moutech gold mine in Iran, where it has been estimated that 778.5 kg of gold (0.5 g/t) exist within the tailings (Dehghani *et al.,* 2009). The average assay of gold in the feed to the plant was found to be 2.5 g/t. Reprocessing of gold tailings was also found to be productive at Ariab area where these tailings contained 2 500 000 ton with gold values ranging between 1.1 and 1.4 g/t (Mohammed, 2015).

An abandoned lead mining in Missouri has waste rocks that have been used for many years for bituminous paving (Collins and Miller, 1979). This material has been used in St Francois country and sold to the city of St. Louis for use in street paving. A highly skid-resistant aggregate has been produced from waste rock from Bethlehem Steel Company's Grace iron ore mine in Berks country (Collins and Miller, 1979).

1.5 Research questions

- What are the values of gold and heavy metals within the tailings?
- What is the economic potential of Consolidated Murchison mine waste?
- How is gold and heavy metals distributed within the tailings dam?
- To what extent do tailings impact on the environment?

1.6 Objectives

The main objective of the study was to ascertain the economic viability of tailings and waste rock of Consolidated Murchison Mine.

© University of Venda

Specific objectives were to:

- Undertake auger drilling, sampling and logging of tailings from the top to the bottom of the tailings dam using manual auger drilling tool;
- Analyze samples for gold using fire assaying method;
- Analyze samples for heavy metals using X-ray fluorescence (XRF) spectrometry;
- Establish the distribution and dispersion of heavy metals within and around Consolidated Murchison tailings dam;
- Conduct sieve analysis to classify waste rock for construction purpose;
- Conduct Atterberg limit tests to classify waste rocks for construction purpose; and
- Conduct compaction test to determine the compaction strength of the waste rock.

© University of Venda

CHAPTER TWO: LITERATURE REVIEW

2.1 Mine waste

Mine waste refers to the valueless material extracted with the ore material from the surface or beneath the earth's surface which cannot be processed at a profit. Waste material can either be discarded at a suitable site or used to fill void spaces left during mining. Waste material varies in the amount produced, properties of the material and the type of the material depending on the resources being mined, the geology and ore mineralisation and the technology used during mining and processing operations. It is therefore essential for the mining company to handle and manage the waste material with proper care to avoid any environmental contamination. This can be done by proper selection and design of a suitable site for waste storage, strategies to handle toxic waste and long-term stabilization of waste material as part of mine closure.

Mining can be classified as the removal of rock and/or soil material to gain access to the ore body and the valueless commodities (either solids, water or gases) left behind after processing to separate valuable mineral from gangue. Waste material can be considered as waste for that moment, but with change in market conditions and processing technologies, waste can be considered to be valuable material. There are several cases where material that was once considered as waste has become a resource for modern mining operations (Rankin, 2011).

2.2 Tailings dams

Tailings dams are dispersal facilities for tailings from the processing plant after separating valuable minerals from the ore.

2.2.1 Tailings characteristics

Characteristics of tailings vary greatly and depend on the mineralogy of the ore together with the chemical processes used to extract the economic mineral. Tailings may be of different mineralogy though they are of the same type, thus they will have different chemical and physical characteristics (Ritcey, 1989). It is important to determine the characteristics of tailings before deposition to know the tailings behaviour. This is done to know and avoid the environmental impacts that may arise after the deposition of tailings and the potential short and long-term liabilities.

© University of Venda

Water is liberated from the tailings after discharging the material into a storage facility. The physical properties of the tailings have an influence in this liberation and it is estimated through laboratory testing of the tailings at different solid concentrations. This can minimize seepage and evaporation losses and can also prevent the discharge of water to a tailings storage facility. The following characteristics of tailings are considered when designing a tailings dam (Engels, 2006):

- erosion stability;
- physical composition and stability;
- chemical composition;
- hard pan behaviour;
- behaviour under pressure and consolidation rates; and
- settling, drying time and densification behaviour after deposition.

The degree of thickening and the method of deposition of the tailings influence the engineering characteristics of tailings. It is important to investigate the tailings properties, physical characteristics and the material parameters such as particle size segregation that can occur because of deposition techniques (SANS, 1998). The process of deciding on which storage method to use can begin after identifying the potential site parameters, for example, geotechnical and environmental as well as costs.

2.2.2 Production of tailings

Tailings are produced when ore is crushed and milled to produce small particles (Fig. 2.1). The extraction method used to remove the economic product from the ore determines the optimum degree of grinding. This can also be used to identify any other economic minerals present in the ore and the type and quantities of chemical reagents used in separating the economic mineral from the ore (Ritcey, 1989). The final design of the tailings dam is always conditional and can only be approved once the tailings are being produced (Blight, 1998).

The process of separating the economic mineral from the crushed ore is known as ‘concentration’ and tailings are the waste material from this process. Most mines use the froth floatation concentration method. This is normally the initial stage in the mineral processing sequence where chemical reagents are introduced (Vick, 1990).

© University of Venda

The froth floatation uses five basic types of reagents namely: frothers, activators, collectors, depressants and modifiers. Gravity and magnetic methods of separation are also used to get the economic product from the ground ore. Coarser particles are recovered by the gravity method, whereas leaching is used to recover finer particles. Pressure oxidation, bioleaching and roasting can be used to process refractory ores.

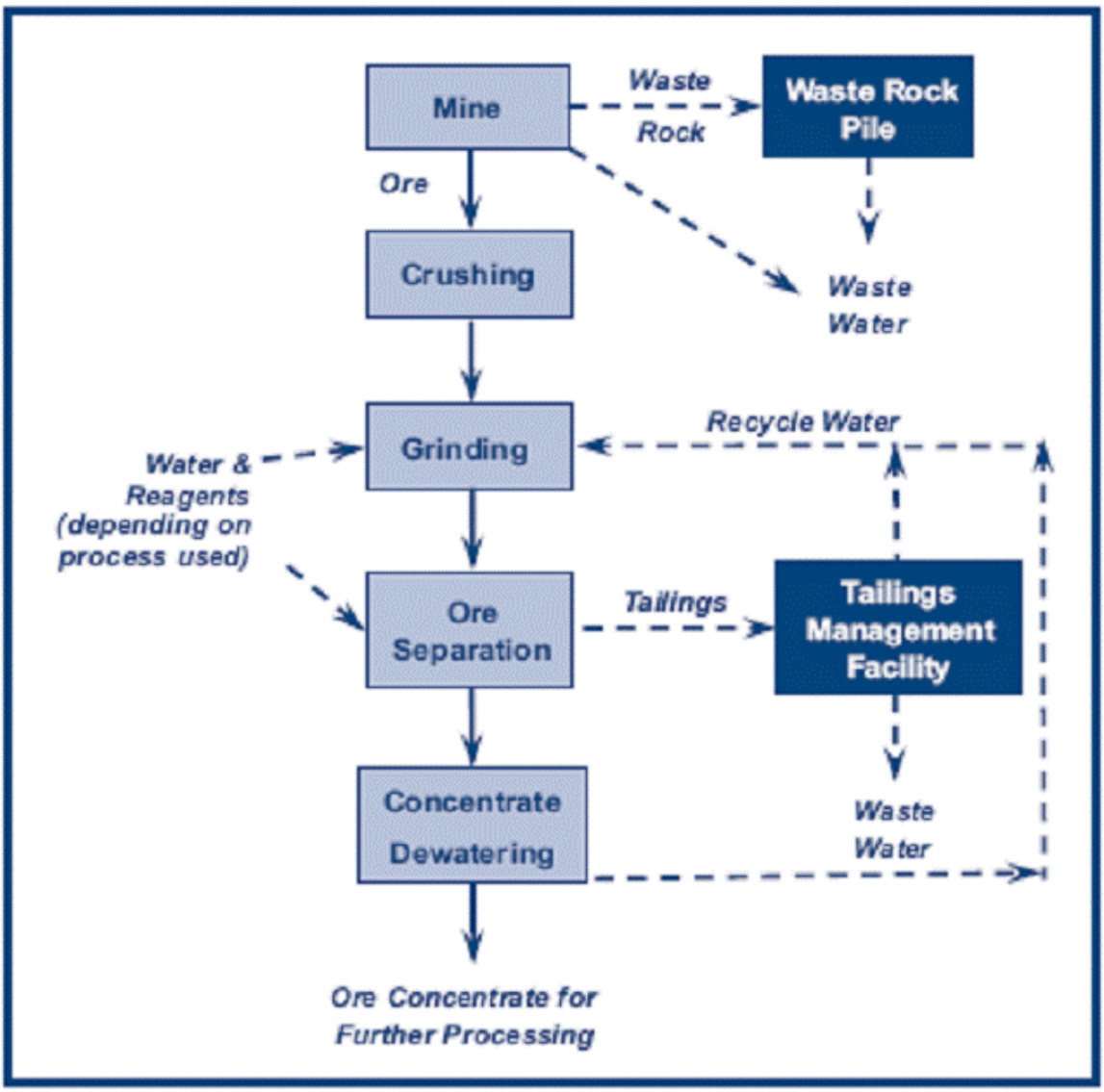

Figure 2.1: Ore processing and generation of tailings and management (Environment and Climate Change Canada, 2017).

Tailings dams and waste rock dumps are identified as the most significant source of environmental impact in many mining operations (Vick, 1990). In some instances, the volume of tailings that requires storage at a particular area can be more than the in-situ total volume of ore that might be mined and processed. This may lead to dams over-flowing and dam failure thus leading to environmental problems. In the

© University of Venda

1960's, more than ten thousand tonnes of tailings had been produced daily in South Africa and this increased to about 100's of thousands of tonnes in 2000 (Jakubick *et al.,* 2003). Today, more than 200 000 tonnes of tailings are produced every day (Jakubick *et al.,* 2003).

2.2.3 History of tailings storage methods

Tailings in the past were discharged directly into the nearest surface water body (Vick, 1990). This is still practiced in some parts of the world in areas of steep and unstable terrain with high rainfall. Common examples include Grasberg mine in Indonesia and the OK Tedi mine in Papua New Guinea (Vick, 1990). This leads to vast environmental problems and could also lead to high costs during remediation and reclamation (Jakubick *et al.,* 2003). The impact of uncontrolled tailings disposal was recognised in the early 1900's (Engels, 2006). Downstream areas were contaminated and plugging of irrigation ditches became a concern, thus creating conflicts between land and water use.

2.2.3.1 Deposition methods of tailings

Tailings can be discharged using either subaerial or subaqueous techniques (Engels, 2006). Tailings characteristics influence the behaviour of tailings after they have been discharged. Natural segregation can occur after the deposition of tailings after flowing away from an outfall. The pulp density of the slurry together with the range of particle size of the tailings influences the degree of segregation (Vick, 1990). There will be less slurry to carry coarse fraction if the thickening of tailings increases (Robinsky, 2000). This leads to tailings staking closer to the discharge point, thus increasing the tailings beach slope.

Thickening of tailings lead to a non-segregated slurry because of high pulp density of the depositing tailings. After this stage, fines fill up voids in the coarse fraction resulting in a homogenous mixture. Paste tailings have these characteristics.

Five deposition techniques generally applied are (Engels, 2006):

- subaerial technique;
- subaqueous technique;
- spigots technique;
- single point technique; and

© University of Venda

- underground disposal technique.

Subaerial technique

The subaerial deposition technique forms a beach above water (BAW) sloping gently towards the supernatural pond. Tailings form shallow low velocity braided streams after being discharged (Fig. 2.2). This allows the tailings to settle and segregate (DME, 1999). The deposition of tailings can rotate between different locations around the tailings dam to allow newly deposited tailings to bleed, dry and consolidate. The tailings are then allowed to discharge to other zones of the tailings dam. Climate, tailings drying characteristics, tailings production rate and the tailings dam shape influence the number of deposition zones and the frequency of discharge point rotation (Engels, 2006). The subaerial technique is more commonly used as compared to subaqueous technique.

Figure 2.2: Shallow low velocity braided streams of subaerial technique of tailings deposition (Engels, 2006).

Subaqueous technique

This technique is suitable for tailings which contain sulphides that would oxidise, produce acid mine water and mobilise metals (Engels, 2006). Oxidation is prevented by placing tailings underwater to restrict oxygen to the tailings (Fig. 2.3). This also

© University of Venda

minimise the environmental problems associated with Acid Mine Drainage (AMD). This technique can be practiced in conventional impoundments. This deposition can create significant steeper slopes while discharging tailings below water than that of subaerial deposition (Robertson and Wels, 1999).

Figure 2.3: Tailings being stored under water in subaqueous technique of tailings deposition (Holden, 2012).

Spigots technique

In spigot disposal technique, tailings are discharged around the perimeter of the tailings dam (Ritcey, 1989). By so doing, a beach between the supernatant and the embankment is created. Beached tailings surround the pond completely (Fig. 2.4). A deposition plan of the tailings must be established during the design stage, implemented and managed throughout the entire operation. Deposition trials can be used to determine the ideal spigot spacing. Incorrect spigot spacing has the potential to cause undulating beaches between spigots. This also reduces the effectiveness of tailings deposition. Blockages and ruptures that could occur can be managed by using multiple spigots (Engels, 2006). Tailings deposition must be monitored and maintained to ensure that tailings are being deposited to the desired areas and to ensure that tailings are deposited in intended quantities.

The main delivery lines from the plant which are larger in diameter are fed by small diameter Pipes (multiple spigots) that feed the main configurations (distribution Pipes). Plugging of the lines and sanding can occur as a result of incorrect flow velocities and Pipeline size reduction ratios (Engels, 2006). Sanding can be prevented by introducing flushing lines and valve stations where multiple spigots are

© University of Venda

used. The discharge velocity of the tailings being pumped to the tailings dam can be reduced by multiple spigotting.

Figure 2.4: Multiple spigots depositing tailings in spigots technique of tailings deposition (Engels, 2006).

Single point technique

This technique places the tailings in fairly thick layers. Tailings are discharged into tailings dams through a single Pipe with a large diameter opening (Fig. 2.5). Tailings remain saturated for a long period of time if they are not dried before depositing new layers (Norman, 1998). Single point is suitable in downstream embankment and sometimes on centreline embankment design. This technique requires that the discharge Pipes should have irregular movement. A single beach deposit of tailings or deltas are normally formed (Vick, 1990). The possibility of seepage erosion is increased by collecting the slimes at the lower end of the impoundment (Ritcey, 2005). This deposition cannot be employed where the slimes and/or the pond are to be kept away from the embankment (Engels, 2006).

© University of Venda

Figure 2.5: Large diameter pipe depositing tailings in single point technique of tailings deposition (Engels, 2006).

Underground disposal technique

Waste from underground can be taken back to underground openings. This is done to reduce the amount of environmental impacts that may occur due to the exposed material on the surface of the earth. Another reason for returning the waste material is to improve the support on mine operations. Copper and gold mines with underground operations may produce a large underground space after mining. After comminution process, the volume of tailings become larger than their original volume, normally there is insufficient room to store all the waste (Xinyi, 2012).

2.2.3.2 Conventional impoundment

Conventional impoundments are surface disposal methods which are the most commonly used disposal methods world-wide. These impoundments are used to store both water from the processing plant and tailings. Water can be reclaimed for use in the processing plant. Water-retention and raised embankments are the two classifications of embankment of surface disposal.

Water retention

Water retention type is constructed to its full height before it is used (Vick, 1990). Water retention tailings dam is used where high volume of water should be stored.

© University of Venda

This water can be reused in the processing plant during dry seasons where surface water inundation can occur.

Raised embankment designs

A raised embankment begins with a starter dyke and when the tailings approach the height of that dyke, a second one is built above the first (Xinyi, 2012). The available volume to store tailings and water is increased by raising the embankment at certain time intervals. Raised embankment has lower capital costs as compared to water retention dams. This is due to fill material and placement costs are phased over the life of the impoundment (Vick, 1990).

There are basically three principal designs (Engels, 2006) namely:

- upstream design;
- downstream design; and
- centre-line design.

Upstream design

Primarily, a small starter dam (Starter dyke) is placed at the extreme downstream side (stage 1) (Fig. 2.6). A beach that becomes the foundation for future embankment raises is created by discharging tailings from the top of the dam crest. Finer material settles at furthest away from spigots discharging tailings whereas coarse material settles closest to spigots. This particle segregation can be accelerated by cyclones. The conventional method of upstream raises relies on no compaction of the spigoted beach that forms the embankment shell (Martin, 1999).

The wall of the dam is progressively raised on the upstream side. To discharge the tailings, the top of the starter dyke is spigoted off and then the initial pond will be nearly filled. The cycle is repeated after the dyke is raised. Various methods used to raise the dyke include: material being taken from the dry surface of the previously deposited tailings and the cycle being repeated, or building the wall from the coarse fraction of the tailings separated out by spigots or cyclones and the fines can be directed into the pond. This method was named upstream since the centre line moves upstream to the pond (Xinyi, 2012).

© University of Venda

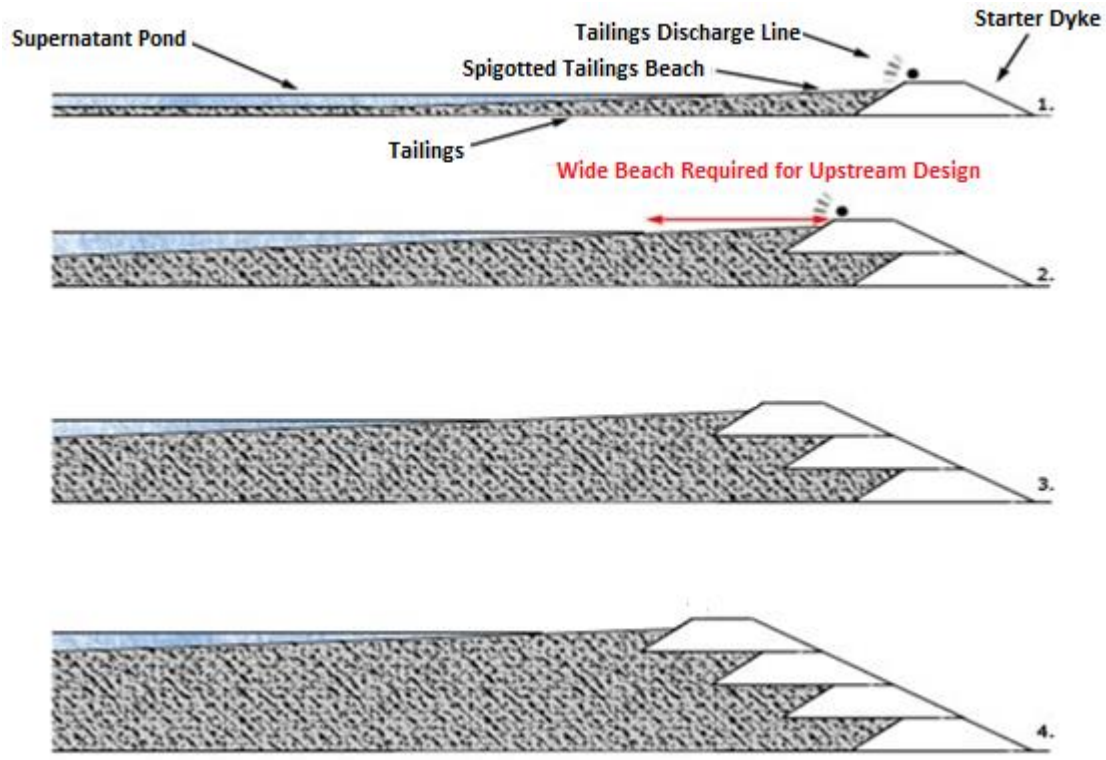

Figure 2.6: Stages in upstream design of tailings dam construction (Engels, 2006).

The upstream method has the advantage of low cost and speed with which the dam can be raised by each successive dyke increment. It is, however, disadvantaged by the fact that the dam wall is to be built on the top of the previously deposited unconsolidated slimes retained behind the wall. The upstream method is rarely employed because of the limiting height to which the dam can be built before it fails and the tailings flow out. This design is suitable in arid climate areas (Engels, 2006). Rapid water accumulation is impossible and negligible amounts of water require storage in the impoundment. This prevents frequent water level deviations that can alter pond geometry, freeboard and phreatic surface within the impoundment area (Engels, 2006).

Downstream design

In downstream embankment design, an impervious starter dyke is implemented unlike the upstream design which starts with a pervious starter dyke (Engels, 2006). The first tailings are deposited behind the dyke. The new wall is raised progressively with raising the embankment and the wall is supported on top of the downstream slope of the previous section (Fig. 2.7). The centreline of the top of the dam

© University of Venda

downstream will be shifted as the embankment stages are progressively raised (Vick, 1990).

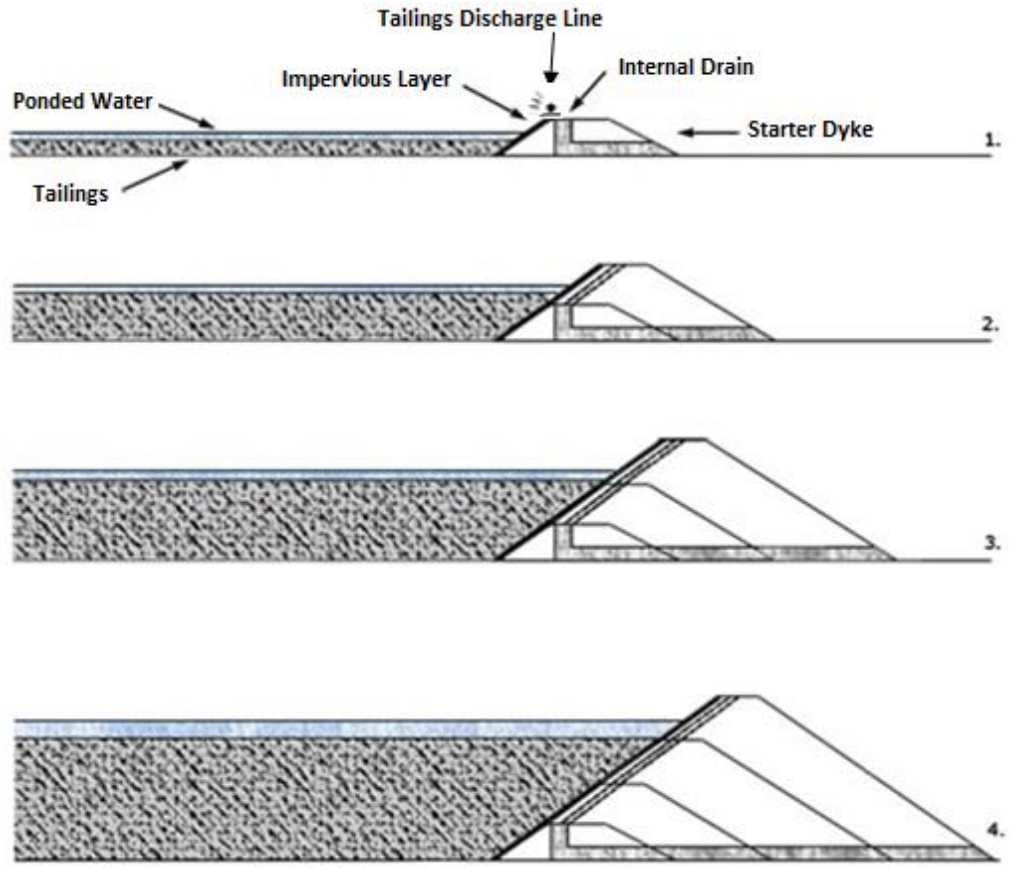

Figure 2.7: Stages in downstream design of tailings dam construction (Engels, 2006).

The downstream method is the reverse of upstream method. Sand for construction of the dam is produced using cyclones in most operations. Acceptable tailings dam by the engineering standards can only be constructed and built by downstream method. The efforts to build larger and safer tailings dam evolved the need for downstream method. To raise the dam wall, large amount of sand is required and it is difficult to maintain the crest of the tailings dam above the rising pond levels. This is more common on early stages of the operation. The sand supply can be augmented with borrowed fill or employing a high starter dam and this will increase the cost of tailings disposal.

The main advantage of this design is that it is not restricted to height because each raise is structurally independent of the tailings (Engels, 2006). The main disadvantage is that it is expensive to raise the embankment as it requires large volumes of fill material which increases exponentially as the embankment height

© University of Venda

increases (Engels, 2006). When more raises are added, a large area around the dam is required as the toe of the dam progressively moves out.

Centre-line design

The wall is raised and the crest remains in the same horizontal position (Fig. 2.8). Tailings are discharged by spigots from the downstream design. Raising the crest to any given height requires smaller volume of sand-fill and the dam can be raised rapidly. This design is a compromise between both the downstream and upstream design (Benckert and Eurenius, 2001). During the early stages of construction, fewer troubles are encountered. Centre-line method is more stable than the upstream method and does not require as much construction material as the downstream method.

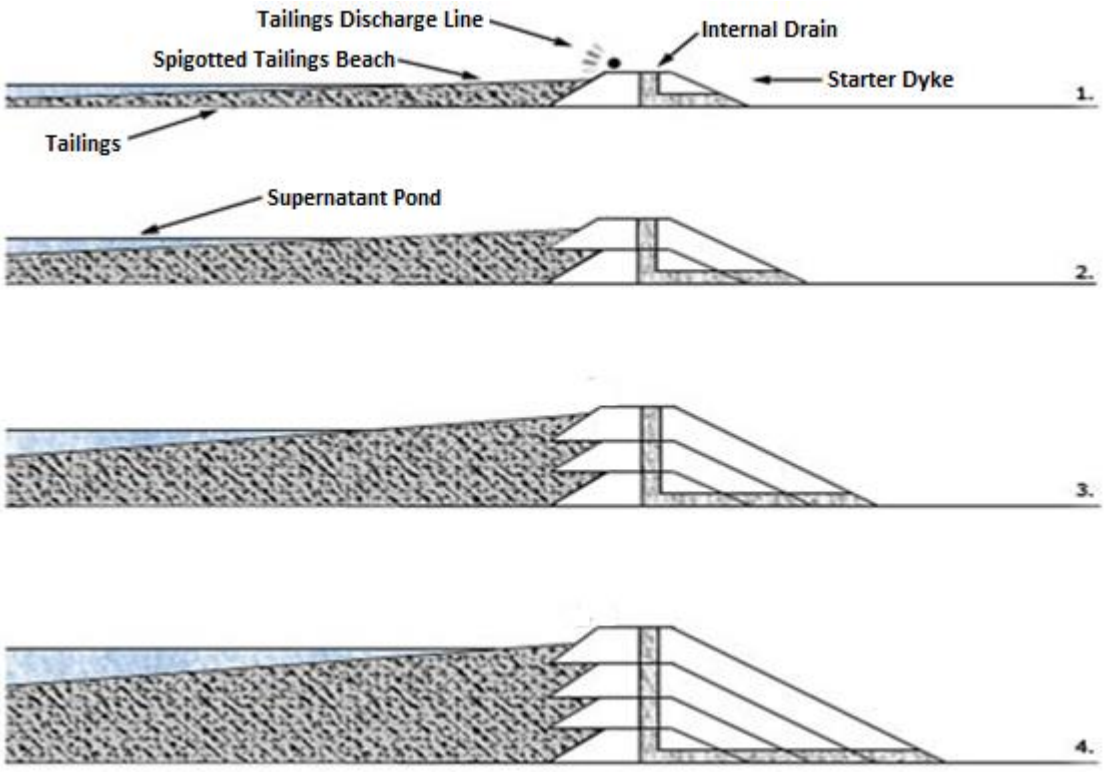

Figure 2.8: Stages in centre-line design of tailings dam construction (Engels, 2006).

When raising the upstream face of the dam, care must be implemented to ensure that unstable slopes do not develop temporarily. This method cannot be used as a larger water retention facility since the subsequent raises are built on consolidated tailings. Prevention of free water from submerging the beach around the dam crest is done by installing suitable decant system.

© University of Venda

2.2.4 Tailings dams and their impacts

Mine tailings dams have the potential to cause various environmental problems if they are not managed or sustained. Rehabilitation of tailings dams is of importance in reducing the potential environmental impacts that may be caused by such dams. The environmental problems caused by tailings can either be physical problems or chemical problems (Xinyi, 2012).

Tailings dams have a risk of failure if not well built and managed. Tailings dams that are not well constructed may result into stability failure. This may damage the local terrestrial features around the tailings dam due to erosion that may result from stability failure. On average, globally, one or two failures of tailings dams have happened each year since 1960 with over 100 of these failures being disastrous or catastrophic resulting in human death (Wise-Uranium, 2011). Environmental impacts are caused by water which is in the waste within the tailings dam. Tailings dam's failure is mainly caused by two major factors which are hydraulic design and inadequate water management (Xinyi, 2012). Rain water coupled with earth-quakes and other natural causes also have the potential to cause stability failure of tailings dams. Hydraulic failure in the form of overtopping by flood discharge accounts for nearly 40% of all the earth dam disasters in the world (Xinyi, 2012). For example, the failure of the Marriespruit tailings dam in South Africa in 1992 was due to overtopping (Fig. 2.9).

Figure 2.9: Marriespruit tailings dam failure in 1992 due to overtopping (www.tailings.info/casestudies/marrespruit.html).

© University of Venda

This was due to heavy rainfall that caused flowslide (static liquefaction) of part of the embankment (Davies, 2002). The dam failure occurred due to an embankment failure during heavy rains. A total of 17 people were killed due to this catastrophic event and scores of houses were demolished (Ulrich and Fourie, 2003).

2.2.4.1 Acid mine water generation

The outflow of acidic water from metal or coal mines is referred to as AMD. The most significant source of water pollution originating from mines is iron pyrite (FeS_2) which is also known as 'fool's gold'. Pyrite gets exposed to water, air and oxidizes resulting in the formation of sulphuric acid, iron oxides and hydroxides (Baker and Banfield, 2013). This causes leachate pH to drop to 4 or lower. This oxidation reaction is accelerated and extended by bacteria known as 'Thiobacillus ferrooxidans'. Sulphuric acid reacts with the host rock or residue deposit to formulate salts and mobilize heavy metals contained in the host rock or residues.

The acidity is usually neutralized during this reaction. The resultant drainage contains high levels of salts such as calcium and magnesium sulphates and metals (mainly iron). The oxidation reaction is proceeded by prolonged contact between water and pyrite and the bacteria also speeds it up and produces more acid. Coal discard with exposed pyrite and old workings produces elevated concentrations of AMD (Fuggle and Rabie, 1992).

The rate of acid mine drainage is determined by the following factors (Fuggle and Rabie, 1992):

- low pH;
- temperature;
- chemical activity of Fe^{3+};
- oxygen content of the gas phase;
- degree of saturation of water;
- surface area of exposed metal sulphide;
- chemical activation energy required to initiate acid generation;
- oxygen concentration in the water phase; and
- presence of certain bacteria (Thiobacillus ferrooxidans) that promote sulphur and/or iron oxidation.

© University of Venda

Sources of acid mine drainage from mining operations include (Fuggle and Rabie, 1992):

- tailings deposits;
- waste rock dumps;
- surface runoff from open Pit mine faces and Pit workings;
- ore stockpiles and spent ore Piles from heap-leach operations; and
- drainage from underground workings.

Waste rock and tailings are exposed to precipitation, runoff and possibly seepage. The greatest source of acid mine drainage is waste rock that contains sulphide minerals, especially pyrite.

The potential for AMD is increasing because the quantities of waste rock from underground mine workings from earlier operations were less than the recent large open Pit mining operations (Sullivan *et al*., 1988). AMD can be generated long after the mine has ceased operation from the tailings and waste rock dumps that are not maintained.

2.2.4.2 Heavy metals pollution

Heavy metals vary in their chemical properties and biological functions. They are of a heterogeneous group of elements. Heavy metals have toxic effects on human beings, plants and animals, thus they are categorized under environmental pollutant category. Heavy metals are exposed to the environment by anthropogenic and natural activities. Anthropogenic activities include mining, smelting operations and agricultural activities. Anthropogenic sources have increased the level of heavy metals (such as Pb, Zn, Cu, As, Cd, Co, Ni, and Cr) on the environment up to a hazardous level. Heavy metals accumulate in soils and plants because they are persistent in nature. This affects human health through food chain because human beings end up consuming such plants causing long term detrimental effects (Environment Canada, 2012). Aquatic organisms get affected through the movement of such pollutants from various diffuse or point sources. This gives rise to coincidental mixtures in the ecosystem. This causes a great threat to aquatic fauna, fish in particular, since they are the significant source of protein to human beings.

© University of Venda

Heavy metals pollution does not biodegrade and has harmful effects on the biological systems. Pb, Co, Cd, and Hg are toxic and cannot biodegrade but accumulate in living organisms causing various diseases and disorders on human lives even at lower concentrations (Rajeswari and Sailaja 2014). These metals have been extensively studied and their effects on human health regularly reviewed by international bodies such as the World Health Organization (WHO) (Rajeswari and Sailaja, 2014). Human beings get exposed to heavy metals through air inhalation, diet and being physically exposed (Ogola, 2002). This can lead to serious health effects. A common example includes the Minamata disease where 60 people (excluding 34.2% under ten years old) were diagnosed of this disease (Harada *et al.,* 2011). Fetal Minamata disease can cause serious psychiatric disorders as well as neurological signs and motor disturbance (Harada, 1964).

2.2.5 Geo-environmental modeling

Geo-environmental models are simply natural extensions of mineral deposits. Geo-environmental models can be defined as compilation geologic, geophysical, geochemical, hydrologic and engineering information pertaining to the environmental behavior of geologically similar mineral deposit prior to mining and resulting from mining, mineral processing and smelting (Plumlee and Nash, 1995; Seal *et al.,* 2002). A geo-environmental model provides information about natural geochemical variations associated with its mining effluents, waste and mineral processing facilities. The key elements of geo-environmental models include deposit type, host-rock, wall-rock alteration, mining and ore processing method, deposit trace element, geochemistry, primary and secondary mineralogy, topography and physiography, hydrology and climate effects.

Geo-environmental models are used at a site as a guideline for potential range of environmental impacts (Seal *et al.,* 2002). Streams, soil and sediments are polluted by rocks with particular element of enrichment and such enrichments may even bring about adverse effects on local and regional ecosystems (Lottermoser, 2010). It is important to have a solid understanding of the environmental geology of a mineral deposit to any mining operation. This helps in providing knowledge about the development of effective prediction, prevention and remediation tools necessary for the successful environmental management of the contaminated area. Models are

© University of Venda

used to highlight likely and less-than obvious potential environmental impacts that arise from geological attributes that are unique to a specific deposit.

Geo-environmental models are basically used to establish the cause-and-effect linkages among the geological attributes of a deposit, its environmental setting, mining history and/or future and its environmental behavior. These models are beneficial to environmental scientists interested in remediating existing problems at un-reclaimed tailings dams and mitigating potential problems associated with tailings dams, land-use planners that are involved in reclaiming tailings and reprocessing of the tailings dams. It can also be used by industries interested in mineral exploration, mine planning. Environmental engineers and scientists use geo-environmental models to highlight potential problems associated with mining and waste management.

2.2.6 Tailings reprocessing technologies

There are generally three major approaches to reprocess tailings impoundments (Xinyi, 2012):

- hydraulic reprocessing;
- mechanical excavation; and
- dredging.

Hydraulic reprocessing

Old tailings can be reprocessed through the use of hydraulic mining. It is also used for mining of soft rock/material. A flow-sheet below (Fig. 2.10) shows a typical hydraulic mining process. High pressure monitor guns are used to wash the tailings downstream. The prevention of large objects from entering the stream is done by collecting material in a sump with a screen to prevent them from blocking the flow (Muir *et al.,* 2005). The slurry is pumped to a thickener to achieve the required pump density. The overflow returning for reuse by the monitors is processed with the underflow.

Mechanical excavation

Sand material is better reclaimed by mechanical excavation, though, it is expensive for slime materials. This is caused by the tendency of transfer points chocking up

© University of Venda

(Muir *et al.,* 2005). Equipment such as bulldozers, trucks to load, front-end loaders and haulage trucks are used to load and haul the material using this approach.

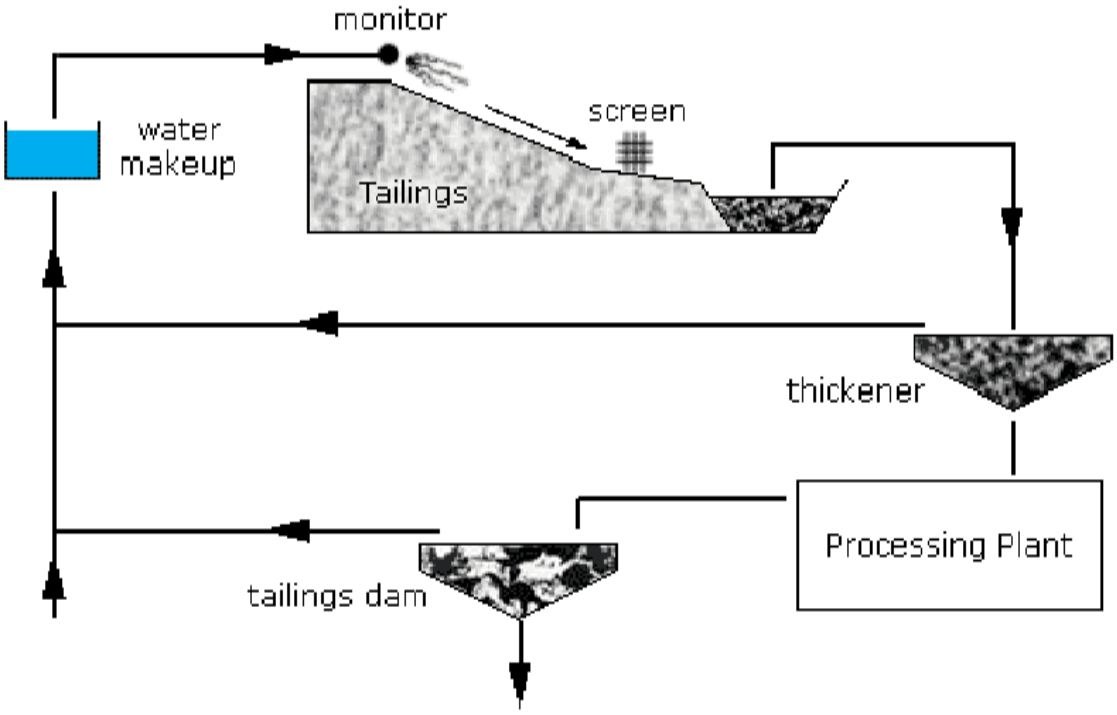

Figure 2.10: Simplified diagram showing the hydraulic re-mining of tailings (Engels, 2006).

Dredging

The high costs of the dredging equipment make this approach to be the most expensive method of reprocessing. This approach can only be employed where mechanical excavation and hydraulic reprocessing cannot be used. This method is generally employed in situations such as water-logged marsh areas.

2.2.7 Tailings dams retreatment

Environmental issues associated with tailings dams can be dealt with by reprocessing tailings dam. This adds profits to the mine as well as increasing the life span of the mine. An increase in metal price coupled with significant improvements in technology makes the retreatment of old tailings dams to be effective and productive. Other tailings dams can be well-thought-out as potential ore deposits due to high metal value. For example, Codelco's El Teniente mine in Chile which is the largest underground copper mine in the world has tailings that are being reprocessed by Amerigo Resources (Xinyi, 2012). TriAusMin have planned to retreat 10 million

© University of Venda

tonnes of copper tailings from three dams at the Woodlawn mine in Australia (TriAusMin, 2010).

Considerable amount of valuable minerals has been lost to the tailings dam in the past due to the limitations in the past mineral processing technologies. For this reason, tailings dams contain considerable amount of valuable minerals that can be reprocessed at a profit. In British Columbia, Carolin mine which produced gold and silver in the past is an ideal case (Xinyi, 2012). It had drilling tests and fire assaying results which had proven reserves of almost 800 000 tonnes with gold values of 1.74 grams per tonne (Xinyi, 2012). This tailings dam has drawn significant attention due to the high grades and current increased gold price (Daniel and Dowing, 2011).

It is remarkably simpler and more economical to reprocess tailings due to reduced exploration, mining, processing and closure costs in the mining process. The creation of new underground openings and/or open Pits is not required when reprocessing tailings. It also does not require drilling and blasting activities. The tailings can be exploited by the use of hydraulic mining with high pressure hoses and simple launders or ditches to the new improved processing plant. The reduction in grinding requirements reduces processing costs and energy since the material being reprocessed had been milled in previous operations. Apart from retaining valuable minerals from the tailings, retreatment also decreases the responsibility of sorting on surface in-perpetuity (Xinyi, 2012).

2.2.8 Tailings reprocessing for gold

The strategy of reprocessing tailings dams has been adopted in many parts of the world with old tailings dams. Mining activities in the Witwatersrand Basin commenced from 1887 and by 1984 and a total of approximately 4.2 billion tonnes of gold ore had been milled (Bosch, 1987). Metallurgists have conducted researches on the extraction of gold from these tailings dams and several developments have been combined to make reprocessing profitable (Bosch, 1987). Towards the end of the seventies, at about the time of the Central Rand Mine closures, it was realized that a valuable resource of gold and other commodities was present in the tailings dams of the Witwatersrand mines' dumps (Viljoen, 2009). In 1978, the East Rand Gold Company (ERGO) started producing gold, uranium and pyrite from re-treated dams (Viljoen, 2009). A similar programme of gold recovery commenced at about the

© University of Venda

same time on the Central Rand on tailings dams averaging about 0.4 g/t of gold and sand dumps averaging about 0.6 g/t (Viljoen, 2009).

The working costs of reprocessing old tailings dams are much lower than conventional mining costs. Tailings dams in the East Rand are being reprocessed for gold, uranium and sulphuric acid. These dams are being reprocessed by means of diverting high-pressure jets of water at the working phase. Approximately 1.5 million tonnes per month are being processed with an average gold value of 0.53 ppm, 40 ppm of uranium and 1.04% of sulphuric acid (www.miningreview.com/news/ergo-to-be-reborn/). ERGO had commissioned one million tonnes per month Carbon-in-Leach (CIL) plant in 1986 at the East Daggafontein mine site (Bosch, 1987).

A treatment plant was established a few kilometres southwest of Central Johannesburg by the Rand Mines Milling and Mining Company (RMMM) to reprocess 50 million tonnes of sand and 20 million tonnes of slime tailings delivered from the Old Crown Mines (Bosch, 1987). This plant was commissioned in 1982 to treat 370 000 tonnes per month for the recovery of gold and pyrite and included the largest Carbon-in-Pulp (CIP) circuit yet built (Laxen, 1984). A similar plant was commissioned by RMMM in 1987 at the old City Deep mine to reprocess approximately 42 million tonnes of tailings (Bosch, 1987). 300 000 tonnes of tailings were reprocessed for gold at Blyvooruitzicht Gold Mine using mechanical methods. In this area, a bucket wheel excavator was used for tailings recovery (Bosch, 1987).

Joint Metallurgical Scheme (JMS) is the first large reclamation operation and it is an arrangement amongst those gold mines in the Orange Free State managed by the Anglo-American Corporation of South Africa (AAC) (Bosch, 1987). Several small operations have occurred in both the West and East Rand. 70 million tonnes of tailings on the East Rand is owned by a less well-known company, Egoli (Bosch, 1987). A plant which incorporates one of South Africa's earliest CIP circuits is being used to reprocess 55 000 tonnes per month of sand and slime in Modderfontein 74 (Bosch, 1987). A plant on the West Rand, managed by this company, is reprocessing sand from several old Randfontein Estates dams. One of the well-known mine in the Central Witwatersrand, Village Main, was saved from closing down by retreating surface residues. Fairview mine in the eastern Transvaal reclaims

© University of Venda

and treats 25 000 tonnes of pyite per month of accumulated flotation tailings (Bosch, 1987; Dehghani *et al.,* 2009).

Moutech Gold Mine, the main producer of gold in Iran located 272 km to the south-west of Tehran has an annual production of 300 kg of gold (Dehghani *et al.,* 2009). Studies in this area show that the content of gold in tailings is more than expected (0.1 g/t) (Dehghani *et al.,* 2009). It has been estimated that the Moutech gold tailings dam contains approximately 778.5 kg of gold with values ranging around 0.5 g/t and the average assay of gold in the feed to the plant is 2.5 g/t (Dehghani *et al.,* 2009). Advancement of processing technologies have added potential benefits such as providing extra gold and reducing environmental impacts. In this work, investigations on a mixture of sulphide and non-sulphide minerals were conducted through flotation, roasting and leaching (Dehghani *et al.,* 2009). This was done to effectively recover gold from the Moutech tailings.

Different processes of gold reprocessing were used where roughly 87.79% of gold was recovered using floatation concentration for gold associated with sulphide minerals (Dehghani *et al.,* 2009). Regrinding, roasting and cynidation of the floatation concentrate was used to recover about 87.8% to 98.4% of gold and about 98% of gold was recovered using the carbon-in-column method (Dehghani *et al.,* 2009). For the floatation method, active carbon was separated using crude oil as the collector and Aeorofloat 39 as the frother. Potassium ethyl xanthate was used as the collector for conditioning the pulp and Sacsol 95 was used as the collector and Aeorofloat 39 as the frother to recover pyrite. A Taguchi design for four variables at two levels was employed.

In regrinding, roasting and cynidation method, samples were roasted at 620°C for an hour the cynidation tests were conducted on roasted samples and ground samples. Fine grinding of concentrate was conducted in a ball mill (200 mm diameter X 200 mm) (Dehghani *et al.,* 2009).

Gold tailings reprocessing studies were also conducted at an artisanal gold mining in Nicaragua where findings had an average of value of 3.82 g/t. Gold values in this tailings dam varied from 1.94 g/t to 5.6g/t (Annicaert, 2013). It has been concluded that gold values from all collected samples were above the cut-off level of 1.75 g/t (Murthy *et al.,* 2003).

© University of Venda

2.3 Grain size distribution

The particle sizes of soil, especially granular soils, has effects on the engineering behaviour of such soil, hence, there is need for soil classification (Holtz and Kovacs, 1981). Soil particle size range is very tremendous and can range from boulders/ cobbles or several centimetres in diameter down to ultrafine-grained colloidal material. Particle size distribution is obtained by a process known as 'Gradation test'. Gradation test is basically a procedure used to assess the particle size distribution of granular material by mechanical shaking the soil material through a series of sives of progressively smaller mesh size.

2.3.1 Grain size analysis (sieve analysis)

Sieve analysis is basically a test that is performed on soil to determine the percentage of different grain sizes contained in soil. This test is required when classifying soil and it provides the grain size distribution within soil (Krishna, 2002). The distribution of sizes of soil particles is of critical importance depending on the way the material performs in use. Sieve analysis can be performed on organic or non-organic granular material including crushed rocks, sands, coal, soil, clays, granite, feldspars, grain, seeds or a wide range of manufactured powders to a minimum size depending on the exact method (Mcglinchey, 2005).

2.3.2 Soil classification

Different soils with similar properties may be classified into groups and sub-groups according to their engineering behaviour. Two classification systems are used by engineers to classify soil taking into account the particle size distribution and Atterberg limits. Soil can either be classified by the American Association of State Highway and Transport Official (AASHTO) classification or Unified Soil Classification System (USCS). The AASHTO classification system is mostly used by the state and country highway departments and the USCS system is used by geotechnical engineers.

American Association of State Highway and Transport Official (AASHTO)

This classification system was developed in the late 1920 for the USA Bueau of Public Roads by Terzaghi and Hogentogler (Holtz and Kovacs, 1981). This system is basically applied in all spheres of engineering designs. This classification system has undergone several revisions. The present AASHTO classification used is given in

© University of Venda

table 2.1 where soil is classified into seven major groups: A-1 through A-7. Granular materials of which 35% or less of the material passing through the No. 200 sieve are classified under A-1, A-2 and A-3. Soils classified under groups A-4, A-5, A-6 and A-7 are mostly silt and clay-type materials of which more than 35% passes through the No. 200 sieve.

Table 2.1 Revised AASHTO system of soil classification (Amadi *et al.,* 2015)

General Classification	General Materials (35% or less passing 0.075 mm)							Silt-clay materials (more than 35% passing 0.075 mm)			
Group Classification	A-1		A-3	A-2				A-4	A-5	A-6	A-7
	A-1-a	A-1-b		A-2-4	A-2-5	A-2-6	A-2-7				A-7-5 A-7-6
Sieve Analysis % passing 2.00 mm (No10) 0.425 mm (No40) 0.725 mm (No200)	 50max 30max 15max	 50max 25max	 51min 10max	 35max	 35max	 35max	 35max	 36min	 36min	 36min	 36min
Characteristics of fraction passing Liquid limit Plastic Index	6max		N.P	40max 10max	41min 10max	40max 11min	41min 11min	40max 10max	41min 10max	40max 11min	40min 11min
Usual types of significant Constituent material	Stone fragment Gravel and sand		Fine Sand	Silty or clayey Gravel and sand				Silty soils		Clayey soils	
General rating	Excellent to Good							Fair to poor			

Unified Soil Classification System (USCS)

This classification system was first proposed in 1942 by Casagrande during the World War II. This system was introduced for the use in airfield undertaken by the Army Corps of Engineers. This system was revised in 1952 by the Corps in cooperation with the U.S. Bureau of Reclamation. It is widely used by engineers and the following should be taken into consideration (Das, 2006):

- The classification is based on material passing a 75 mm (No. 3) sieve.
- Coarse fraction = percent retained above No. 200 sieve = 100 – F_{200} = R_{200}.
- Fine fraction = percent passing No. 200 sieve = F_{200}.
- Gravel fraction = percent retained above No. 4 sieve = R_4.

This classification system divides soil into two major categories:

1. Coarse-grained soils that are gravelly and sandy in nature with less than 50% passing through the No. 200 sieve (that is, $F_{200} < 50$).The group symbols start with prefixes of either G (gravel or gravelly soil) or S (sand or sandy soil); and

© University of Venda

2. Fine-grained soils with 50% or more passing through the No. 200 sieve (that is, $F_{200} \geq 50$). The group symbols start with prefixes of M (inorganic silt), C (inorganic clay) and O (organic sitls and clays). The symbol Pt is used for peat, muck and other highly organic soil.

2.3.3 Atterberg limits

Atterberg limits are a basic measure of the critical water contents of fine-grained soil. This basically includes its *liquid limit, plastic limit and shrinkage limit*. Soil may appear as solid, semi-solid, plastic or liquid state depending on its water content (Fig. 2.11). The engineering properties, consistency and the behaviour of soil are different from one state to another. The boundary between each state can be well defined by the changes in soil behaviour and atterberg limits can be used to distinguish between silt and clay. As water is added to any dry plastic soil, the remoulded mixture will eventually have characteristics of a liquid. The material will change from solid to semi-solid and as water is continuously added, it turns to liquid state. The point at which soil changes from one state to another is known as the atterberg limits (Das, 2010).

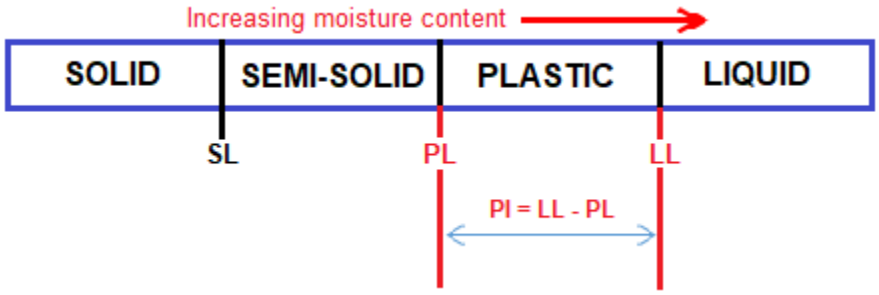

Figure 2.11: Illustration of Atterberg limits.

Liquid limit

Liquid limit is the water content where the behaviour of a soil changes from plastic to liquid. At this point, soil starts to lose its shear strength as liquid does not have its shear strength (Atkinson, 1993). This transition is gradual over a range of water contents and the shear strength of the soil is not actually zero at the liquid limit.

Plastic limit

Plastic limit is the water content at which a thread of soil crumbles when it is carefully rolled out to a diameter of 3 mm (Karlsson, 1977). A thread of soil is at its plastic limit

© University of Venda

when it is rolled and begins to crumble. Plastic limit is determined by rolling out a thread of soil on a flat, non-porous surface. If the thread of soil can be rolled out to a diameter that is less than 3 mm, then the soil is too wet (above the plastic limit). The sample can be remoulded and the test may be repeated. if it crumbles before reaching the 3 mm diameter, then the plastic limit has been passed (Hansbo, 1957). Soil is considered to be plastic if a thread cannot be rolled out down to 3.2 mm at any moisture possible (Das, 2006).

Shrinkage limit

The shrinkage limit is the water content where further loss of moisture will not result in any more reduction of volume (Seed and Idriss, 1967). Shrinkage limit is less used than the liquid and plastic limits. Soil shrinks as moisture is gradually lost from it and the continuous loss of moisture content results in a stage of equilibrium where more loss of moisture will result in no further volume change (Das, 2006).

2.4 Production of construction materials from mine waste

Gold tailings are associated with serious environmental problems such as acid mine drainage. The management of such tailings dams is expensive thus there is need to find alternative use of tailings to minimize large volumes of tailings. The population in South Africa has increased to about 52.98 million, increasing the number of houses required to accommodate people (Lehohla, 2013). Tailings can be used for brickmaking to reduce large volumes of the waste. Extensive research has been conducted on the production of bricks using waste material (Ahmari and Zhang, 2012; Zhang 2013). A research has been conducted in utilizing copper mine tailings and cement kiln dust to manufacture geopolymer bricks (Mathew *et al*., 2013). Copper mine tailings bricks have been found to have good physical and mechanical properties such as being water absorption (17.7%) compressive strength (260 kg/cm2) and density of 1.8 g/cm3 (Sharp, 2012).

Gold tailings were mixed with Portland cement (OPC), red soils and black cotton soils in different proportions to make bricks (Roy *et al*., 2007). The compressive strength of the cement-tailings bricks was determined by immersing them in water for different periods of time. Bricks which were cured for 14 days with 20% of cement were found to be suitable. Gold mine tailings were used to produce autoclaved calcium silicate bricks (Jain *et al*., 1983). Saturated steam was used to cure these

© University of Venda

bricks. During this process, lime reacted with silica grains to form a cementing material consisting of calcium silicate hydrate. The idea of making bricks from tailings material has been adopted by mining companies such as Bharat Gold Mine in India (Malatse and Ndlovu, 2015).

Waste rock has been used for construction, particularly for highways. Waste rock was used in Colorado in the United States during the 1930s where waste rock from gold mining was used for road construction (Collins and Miller, 1979). A million dollar highway road in the U. S. Route 550, which extends from Durango to Silverton, was built from these waste rocks. In the south-eastern part of the United State, waste rock from fluorspar mining in Illinois has been used as coarse aggregate. Poor rock from copper mining in the Upper Peninsula of Michigan has been used for various construction purposes (Collins and Miller, 1979). Coarse waste rock from Missouri (underground iron mine) has been sold to an aggregate producer (Collins and Miller, 1979). This producer crushes and sells aggregates of approximately 110 000 tonnes per year which is used as skid-resistant aggregate for bituminous paving.

An abandoned lead mining in Missouri has waste rocks that have been used for many years for bituminous paving (Collins and Miller, 1979). This material has been used in St Francois Country and sold to the city of St. Louis for use in street paving. A highly skid-resistant aggregate has been produced from waste rock from Bethlehem Steel Company's Grace iron ore mine in Berks Country (Collins and Miller, 1979). This aggregate has been used for several years in the bituminous resurfacing of the Pennsylvania Turnpike from Morgantown to Valley Forge. Approximately 75 000 tonnes of waste rock from two slate producers in Buckingham Country, Virginia has been used each year as a stone base aggregate by the Commonwealth of Virginia (Collins and Miller, 1979). The amount of flat and elongated particles through more exacting crushing methods has to be controlled to produce satisfactory aggregates.

2.5 Gravelotte gold deposit

The Murchison Greenstone Belt hosts the Gravelotte gold and antimony deposit. This belt consists of three formations, La France and Weigel Formations that occupy the central part of the belt, and MacKop Formation which occurs south of the Baderoukwe gneiss pluton in the Bawa Schist Belt (Fig. 2.12). The Murchison range

© University of Venda

comprises rock types that are highly metamorphosed and are members of the Swaziland System consisting of the oldest known rocks in Southern Africa (Consolidated Murchison Limited information brochure, 1991). The range is flanked on three sides by Archaean granite, which is the source of mineralisation. Mineralisation strikes in the general direction of the Murchison range and consists of five lines from north to south (Minnitt and Anhaeusser, 1992):

- titanium from iron ore line;
- copper-zinc line;
- antimony line; pyrite-gold line; and
- emerald line.

Operations at Consolidated Murchison mine are confined to the antimony line, which also carries gold, cinnabar and tetrahedrite. The antimony ore, mainly consisting of stibnite (Sb_2S_2), with some beberthierite ($FeSSb_2S_2$), is a medium temperature hydrothermal deposit characterized by steeply dipping reefs (Consolidated Murchison Limited information brochure, 1991). It consists predominately of phyllites, talc schists and talc carbonate schists occurring mainly as an envelope engulfing the more competent core of schistose and massive grey and green quartz-carbonate rocks. The ore body is relatively shallow, Pinching and swelling both laterally and vertically and attaining a boundinage structure (Consolidated Murchison Limited, 1991).

The ore body has a general dip ranging from 60° to 85°. Stibnite deposits are separated from each other with the furthest separation of approximately 55 km apart and mineralisation is not continuous along the line. The numerous ore minerals that have been identified in the Consolidated Murchison's claims testify to the rich mineralisation of the region (Consolidated Murchison Limited Information Brochure, 1991).

The antimony line hosts major occurrence of antimony. The antimony line occurs within the Weigel Formation (Davis *et al.*, 1988). Quartz-carbonate are host rocks which reveal high concentrations of SiO_2, FeO, MgO, and Al_2O_3 with low CaO-to-MgO ratio. According to Davis *et al.*, (1988), quartz-carbonate host rocks were originally magnesium-rich basalts or, more particularly, peridotitic komatites. These host rocks contain dolomite, magnesite and quartz with minor talc, chlorite, fuchsite

© University of Venda

and some sulphides. There is a gross foliation between the quartz-carbonate and the chromium-rich mica fuchsite which imparts a bright-green colour. A sharp contact occurs with the surrounding talc rocks, especially where the quartz-carbonate bodies are boudinaged. A good cleavage is developed where the host rocks are schistose and the contact may be gradational. The contact zone is generally conformable with the cleavage (Davis *et al.*, 1988).

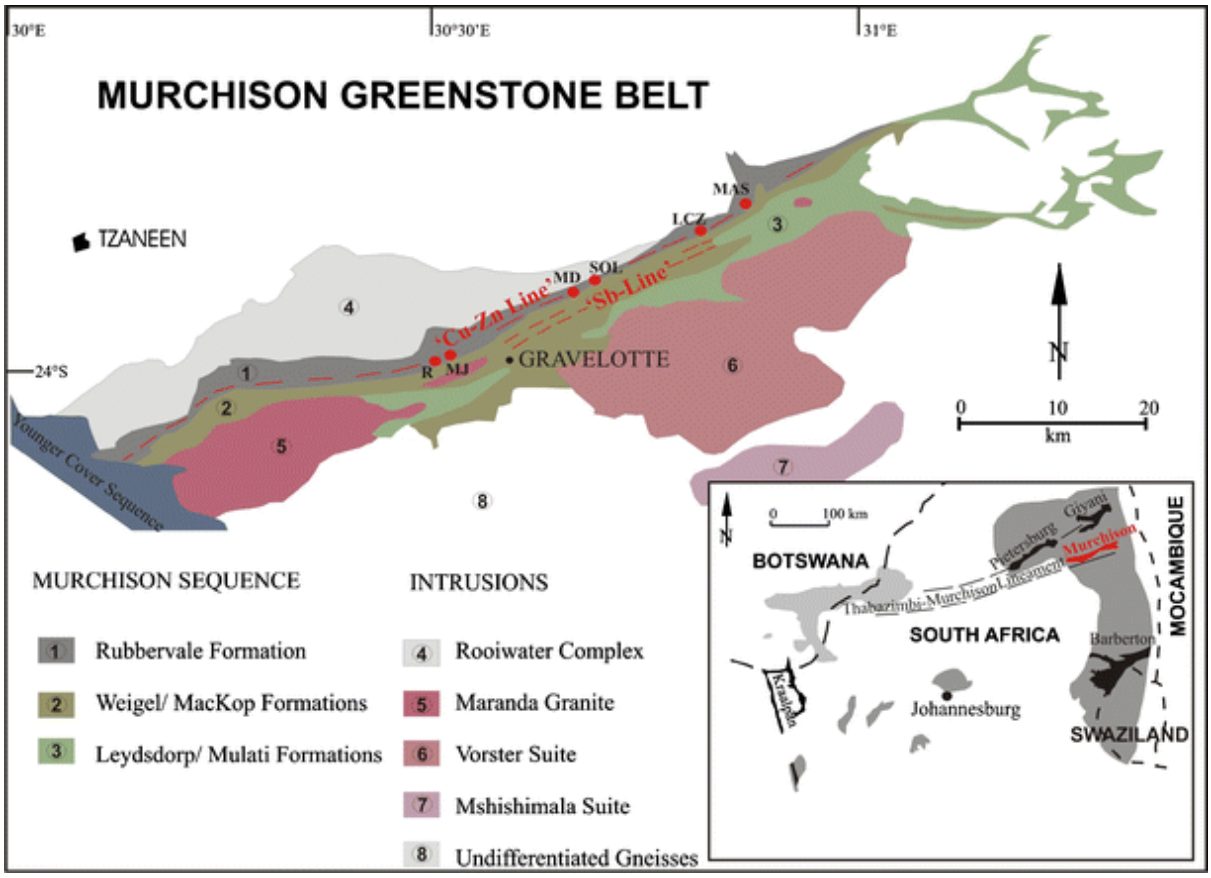

Figure 2.12: Geological map of Murchison Greenstone Belt (Schwarz-Schampera *et al*., 2010).

2.6 Ore mineralisation

The main ore bearing rock type is the massive and generally fine-grained quartz-carbonate rock. It generally displays a brecciated texture and it is highly fractured in many places. These fractures are filled with quartz, stibnite and lesser amounts of carbonate. They are erratic and discontinuous. Gold is more common in veins which are predominately rich in stibnite and gold specks occur erratically in such veins (Willson and Viljoen, 1986). There is no correlation between gold and antimony as proven by the statistical analysis of assay results of the ore intersections, however,

© University of Venda

mineralogically, there is an association of gold, stibnite and quartz (Willson and Viljoen, 1986).

Gold occurs in three different forms. It can either occur as coarse visible gold, as finer disseminations or in close association as sub-microscoPic intergrowths within the sulphides (Davis *et al.*, 1988). The arsenopyrite horizon which forms discrete lenses at or near the antimony mineralisation contains additional gold (Davis *et al.*, 1988). The most common minerals throughout the ore zone are pyrite, arsenopyrite berthierite. The occurrence of gersdorfite is indicated by the microscoPic examination of the mill feed. Quartz-carbonate rock contains disseminated pyrite and is often biotitic in areas richer in iron (Willson and Viljoen, 1986). Veins are zoned with berthierite and pyrite occurring at the margins of areas richer in iron and stibnite occurring in the center.

Gold and antimony economically occur within the antimony line. Various known iron, nickel, copper, lead and some antimony sulphides are present in lesser amounts together with some minor arsenopyrite (FeAsS) and berthierite (FeS, Sb_2S_3) (Davis *et al.*, 1988).

2.7 Mining history

Mining activities started in 1934 when large scale production of gold began (Consolidated Murchison Limited, 1991). Antimony was found as a by-product of gold and was mined in small scale. Large scale mining began in 1937. In 1972, the name of the mine changed from Consolidated Murchison (Transvaal) Goldfields to Consolidated Murchison Limited. In 1991, Johannesburg Consolidated Investment (JCI) acquired the company and disposed its shareholding and Metorex acquired the mine in 1997 (Consolidated Murchison Limited, 1991). Trackless mining was introduced in the mine in 2008 (Consolidated Murchison Limited, 1991). To The Point (TTP) took over operational management in 2009/2010 and secured the offer to purchase the mine. In 2010, the name was changed to Consmurch Mine (Consolidated Murchison Limited, 1991).

Village Main Reef (VMR) acquired the mine in 2011 and put it up for sale in 2013. From 2014 towards early 2015, VMR was going through business administration and eventually provisional liquidation (Consolidated Murchison Limited, 1991). Liquidators placed Consmurch Mine under care and maintenance in 2015 and

© University of Venda

employees were retrenched. Over 200 employees were recalled during mid-2015 due to the management deal that Stibium striked with liquidators. From 1937 to 1989, the total amount of gold recovered was 26 854 kg with an average production of 516 kg per year (Consolidated Murchison Limited, 1991). From 1937 to 1989, the average concentration of gold from the ore was 1.89 g/t and the highest and lowest concentration were 10.55 g/t in 1940 and 0.21 g/t in 1971 respectively (Consolidated Murchison Limited, 1991)

© University of Venda

CHAPTER THREE: MATERIALS AND METHODS

This chapter deals with the methods and procedures that were applied during the study (Fig. 3.1).

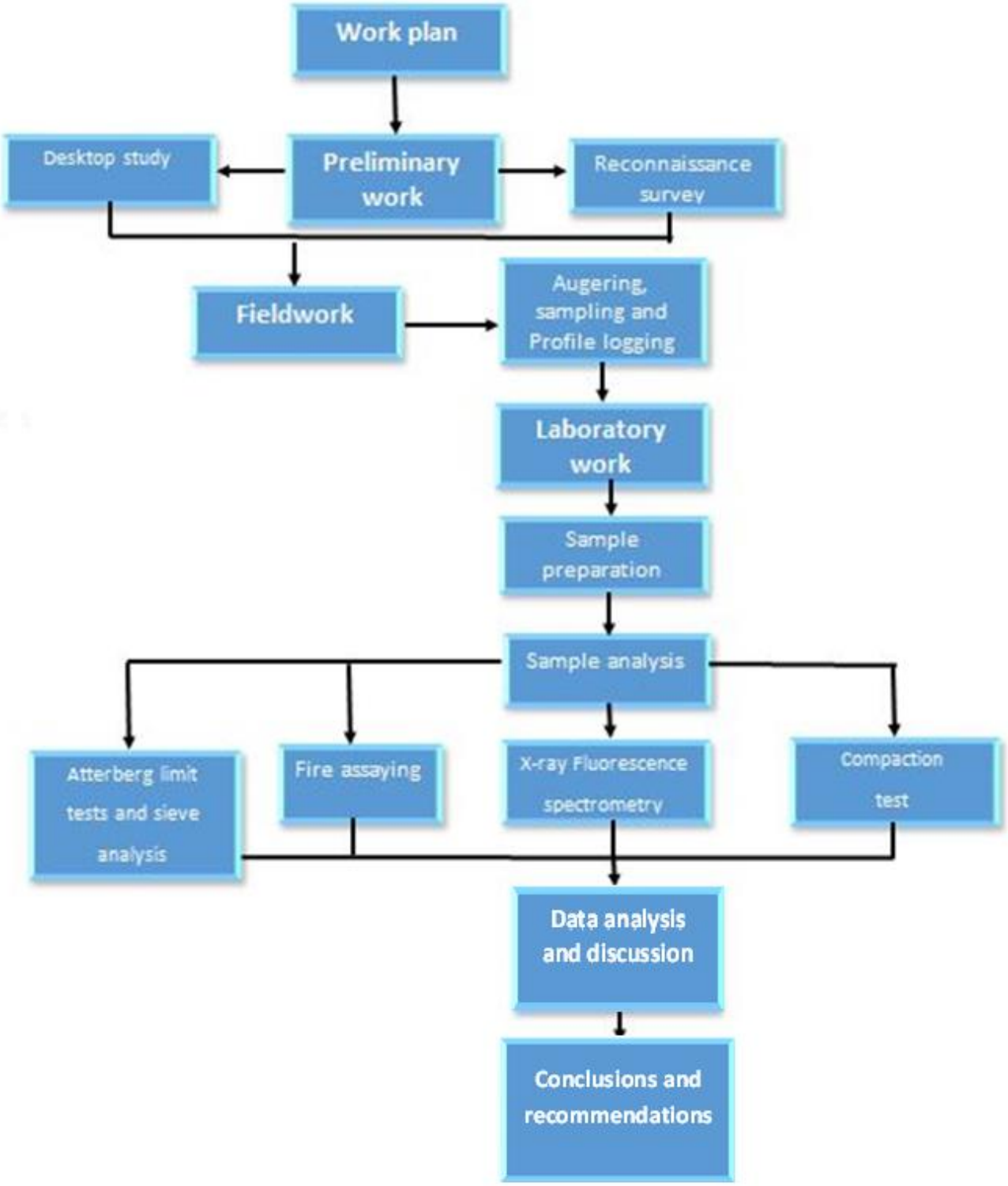

Figure 3.1: Flow chart showing the methods and procedures applied in the study.

© University of Venda

3.1 Preliminary work

3.1.1 Desktop study

For desktop study, materials related to the research topic were reviewed. Journals, topographic and geological maps, books, published and unpublished technical reports and information retrieved from world-wide website were used to collect secondary information and data. Information about the characteristics of the area such as climate, topography, vegetation cover, and tailings were acquired during this phase.

3.1.2 Reconnaissance survey

Reconnaissance survey was necessary to determine the accessibility of the study area and to also plan how the work was to be undertaken. Sites to be sampled were identified and this enabled the researcher to precisely gain information about the study area. Reconnaissance survey provides first-hand information and feel about the study area.

3.2 Fieldwork

This work was conducted at Consolidated Murchison mine tailings dam. Auger drilling was used to collect samples from the tailings dam to a depth of 8 m and logging of tailings was done simultaneously. Tailings samples were collected for gold analysis to ascertain whether the tailings can be reprocessed for gold and determine the values of heavy metals within such tailings. Soil samples around the tailings dam were collected for heavy metal analysis to determine the extent of pollution (if any) by heavy metals from the tailings dam. Waste rocks were collected from the waste rock dump for the purpose of geotechnical testing of waste rock in order to determine the usability of such rocks as sub-base layer for road construction or other foundations. The following equipments were used during fieldwork;

- tape measure was used to measure the length, width and distance between the Profiles and sampling points;
- GPS was used to locate sampling points;
- shovel was used to clear each sampling point before auger drilling was conducted and to collect soil samples;
- hand auger was used to drill through the tailings dam to collect tailings samples; and

© University of Venda

- polyethylene sampling bags were used to collect samples.

3.2.1 Augering and sampling of tailings

A total of four sampling Profiles with a spacing of 75 m between them were projected over the tailings dam. The first Profile had two sampling points due to the irregular shape of the tailings dam. The second Profile had three sampling points, the third Profile had four sampling points and the fourth Profile had five sampling points (Fig. 3.2). For every Profile, sampling interval was 130 m. Augering was conducted from the top of the tailings dam to a depth of 8 m. Every sampling point was cleared off the top material with a shovel prior to drilling. Samples were collected at 1 m interval with depth. A total of 112 tailings samples were collected with each sample weighing approximately 5 kg.

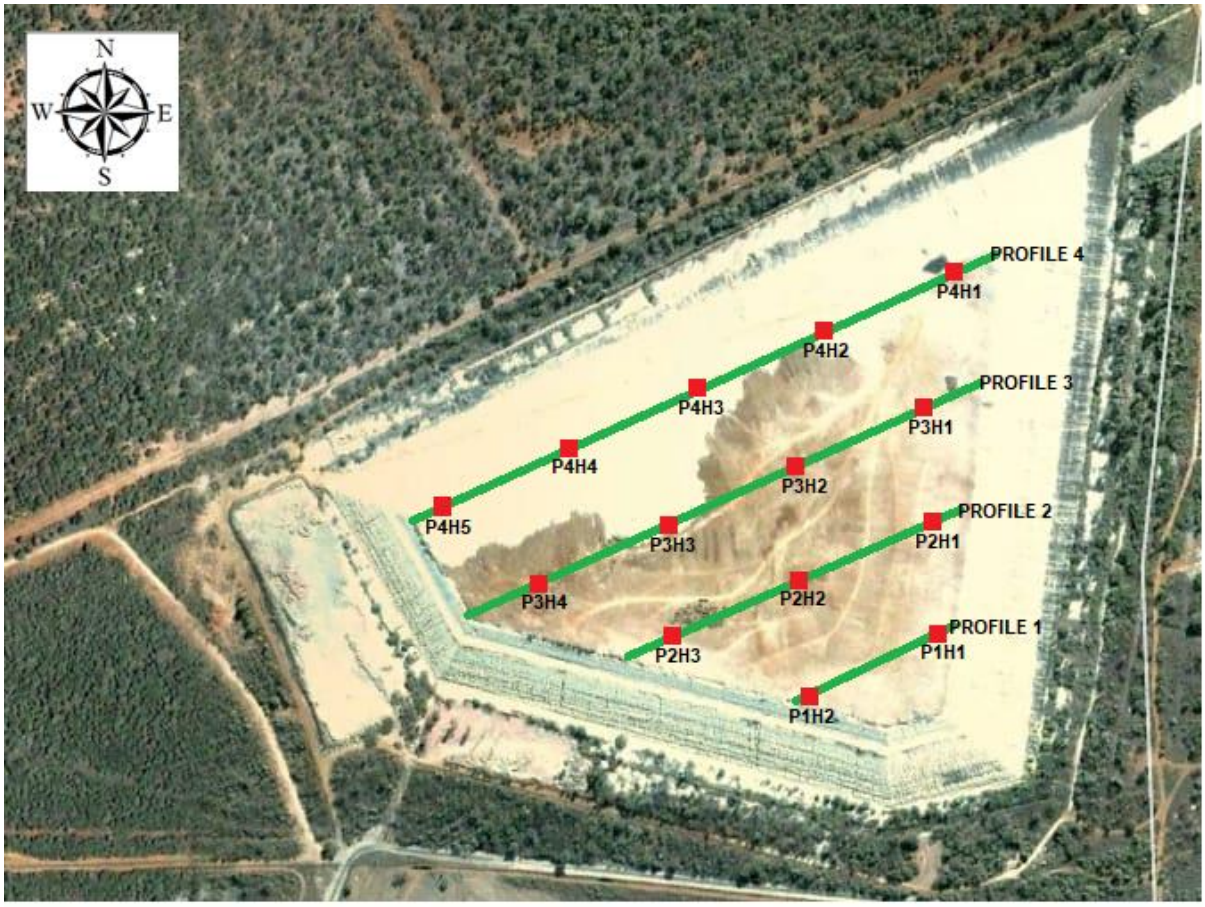

Figure 3.2: Positions of profiles and sampling points over the tailings dam.

3.2.2 Profile logging

Profile logging of tailings was done simultaneously with augering of tailings. Changes with depth in physical properties of the tailings such as colour, moisture content and hardness were carefully recorded in a logbook. This information was used to

© University of Venda

correlate logs along the Profiles. This information was used to determine the oxidation, transitional and unoxidized zones of the dam.

3.2.3 Soil Sampling

Soil samples were collected in a systematic manner around Consolidated Murchison tailings dam to a distance of 5 km away from the dam, to the north, south, east and west (Fig. 3.3). Samples were collected at a sampling interval of 250 m for the first 2.5 km away from the tailings dam. A control point was established at a distance of 5 km away from the tailings dam at the northern, eastern and western part of the dam. Six instead of eleven samples were collected on the southern part of the tailings dam due to lack of access to the area as the farm owner refused access. A shovel was used to clear the identified sampling points as well as to collect the soil samples.

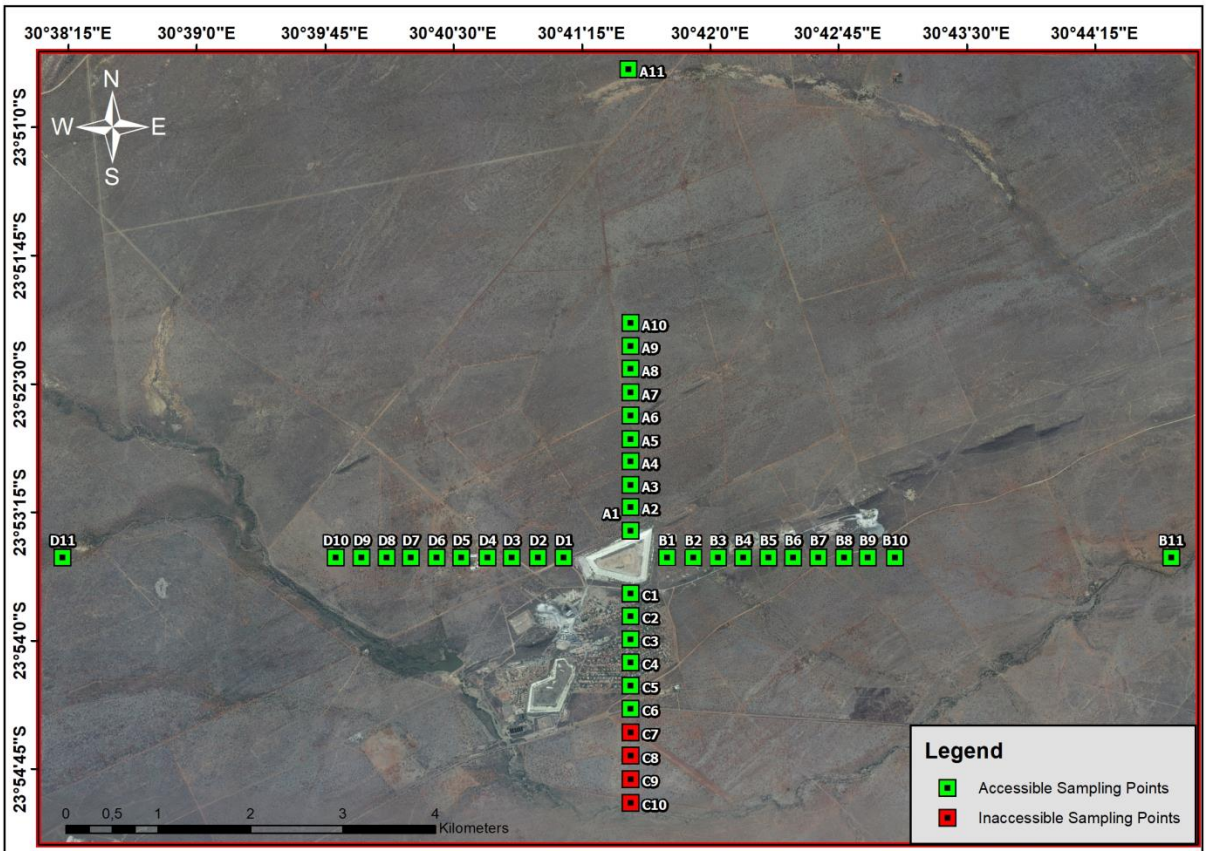

Figure 3.3: Sampling points around the tailings dam.

A total of 39 samples were collected. Soil samples of approximately 2 kg each were collected at a depth of about 30 cm. All samples were stored in marked and labelled polyethylene sampling bags. To avoid contamination, the sample bags were tied tightly with a string.

© University of Venda

3.2.4 Waste rock collection

Grab sampling method as illustrated in Figure 3.4 was used to collect waste rock from the rock waste dump. This method was used because waste rock dumps do not contain a uniform size of rocks and rocks are piled up on each other.

Figure 3.4: Collection of waste rock.

3.3 Laboratory work

3.3.1 Sample preparation

Collected samples were taken to the Department on Mining and Environmental Geology Laboratory at the University of Venda for sample preparation. This involved drying, milling and weighing of the samples prior to geochemical analysis of such samples.

© University of Venda

Drying of samples

A total of 112 tailings and 39 soil samples were dried. Each of the collected samples were split into four equal parts at the laboratory. About 1 kg of each sample was placed inside a drying oven to dry the samples, using the Vacutec oven (Fig. 3.5). The samples were dried for a period of 12 hours at 110°C. After drying, the samples were removed and allowed to cool and ready for milling.

Figure 3.5: Bench Vacutec laboratory drying oven.

Milling of samples

Samples were milled using Restsch model RS 200 milling machine (Fig.3.6). Milling of the samples was done based on the following procedure to ensure homogeneity of samples for analysis (Tan, 2005):

- milling machine was cleaned to remove any dust particles before the milling process begin;
- cleaning of the milling pots was done by quartz fragments for 5 minutes;
- pots were dusted to clean and to remove the milled quartz;
- samples were placed in milling pots and milled at 700 rmp for 5 minutes;
- samples were removed from the pots and transferred to sample bags; and
- milling pots were cleaned after the completion of each sample and milling continued.

© University of Venda

Figure 3.6: Restsch model RS 200 milling machine used to prepare samples.

Pelletisation of samples

After the milling process, samples were subjected to pelletisation using a 40-ton pressing machine. In this process boric acid was used as a binding agent and acetone used for cleaning the die-sets to avoid contamination from one another. The following procedure was followed in preparing such samples (Tan, 2005):

- approximately 15-20 g of milled powder was added in a set of 40 mm diameter die-set and about 30-40 g of boric acid was added as the binding agent;
- samples were pressed using about 30 tons of pressure on the manual operated pressing machine for about a minute each (Fig. 3.7);
- after pressing, the pellet was ready then removed from the die-set and placed in a container which was labelled (Fig. 3.8);

© University of Venda

Figure 3.7: Pelletisation using a 40-ton pressing machine.

Figure 3.8: Prepared pellets ready for analysis.

Weighing of samples

A weighing balance, Redwag model AS 220/C/2 (Fig. 3.9) was used by precisely weighing 100 grams of each tailings sample and transferring it into a clean paper sampling bag ready for analysis.

© University of Venda

Figure 3.9: Redwag model AS 220/C/2 used to weigh the milled samples.

3.3.2 Fire assaying

Approximately 100 g of each milled tailings sample was sent to Australian Laboratory Services in Johannesburg for gold analysis by fire assaying method. In this method, an aliquot of powdered sample (approximately 30 g) was mixed with soda ash, flour, borax (sodium borate), litharge, silica and nitrate. Silver and palladium are used as collectors which were added in a solution or as a foil. The material was fired after mixing at a temperature of 1100°C. Gold was then scavenged from the melt by the settlement of silver and lead in the melt to the bottom of the crucible. The hot molten mixture cooled after being poured into a mould. The lead button was placed in pre-treated cupels. Silver which was used as a collector for gold and other Platinum Group Elements (PGEs) was left and absorbed into the cupel. Gold was separated from silver bead by dissolving in nitric acid (process known as partying). The entire bead dissolved in acid and gold was determined by atomic absorption spectrometry. A total of 84 tailings samples were analysed for gold.

3.3.3 X-ray Fluorescence spectrometry

After the pelletisation process, the end products (pellets) were taken for heavy metals analysis using X-ray fluorescence spectrometry. In this process, each sample was analysed for Pb, Zn, Cu, As, Cd, Co, Cr, and Ni.

© University of Venda

XRF analysis is non-destructive elemental analysis which can qualitatively and quantitatively measure periodic table elements which are found in the sample. Samples are comprehensively mixed to make sure of its homogeneity. The pressed pellet was loaded in the sample port of the X-Ray Fluorescence analyser which was connected to a computer system and the assembly left for about six to twelve minutes after which the values of elements concentration were displayed on the monitor (Fig 3.10).

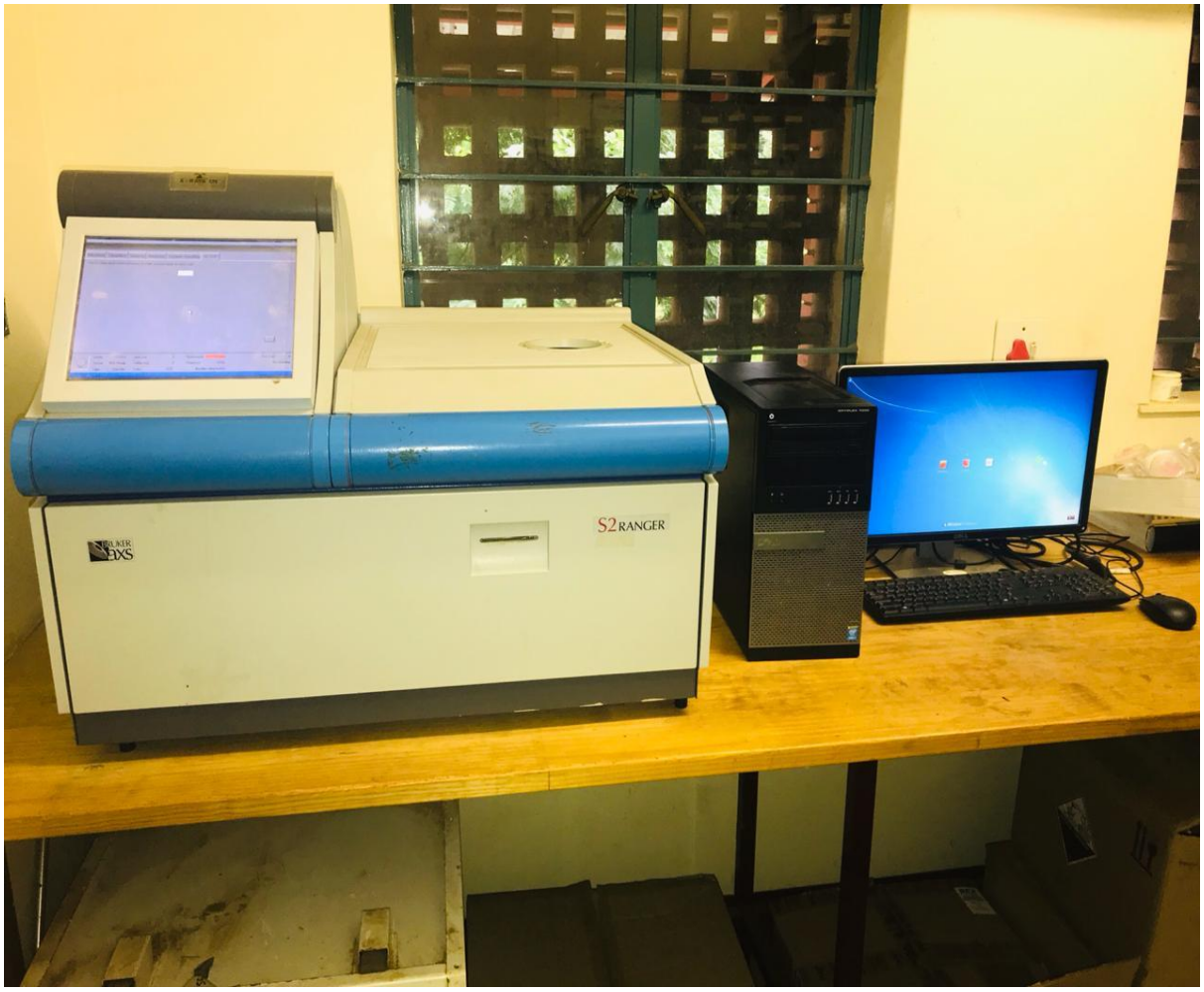

Figure 3.10: Analysis of heavy metals using X-Ray Fluorescence Spectrometry.

The XRF results were saved directly on the computer system and then printed out. Prior to each cycle of analysis, the spectrometer was calibrated using a copper disk. A sample was designated as a check sample to ensure repeatability of the analysis, and the sample was analysed after a cycle of analysis. The standardised sample SARM 42 was also used to ensure precision of the instrument for quality control. A total of 151 tailings and soil samples were analysed.

© University of Venda

3.3.4 Sieve analysis

Dry sieve analysis was carried out to determine the particle size distribution of waste rock. This is a simple laboratory test which was performed on the waste rock material to determine the particle size distribution. This test is important because the particle size composing the material ultimately determines the behaviour of the material under the various conditions and thus dictates its applicability for various geotechnical projects, for example, soil constituting more clay and silt material is undesirable for geotechnical projects. This test was carried out as stipulated by the Standard Methods of Testing Road Construction Material (1986); TMH1 Method A1.

A stack of standardized sieves were used for this procedure which quantitatively separates gravel and soil particles according to their grain size. The sieves used had the following apertures: 75 mm, 50 mm, 40 mm, 31.5 mm, 20 mm, 10 mm, 5 mm, 4 mm, 3.15 mm, 2 mm, 1 mm, 0.425 mm, 0.25 mm, 0.125 mm, 0.075 mm and the pan. All these sieves together with the pan were cleaned by washing them with running water from a tap to avoid contamination of the material. Sieves were weighed on the scale balance before loading the material and their masses were recorded in the notebook.

Each dried sample was emptied onto the stack of sieves, that is the upper most sieve on the stack with the aperture of 75 mm, and then the sieve was closed with a lid to contain all the material. The stack of sieves containing the waste material was placed on the mechanical shaker as shown in Figure 3.11, which facilitated the sieving process. The mechanical sieve operated at an amplitude of 40 and shook the stack sieves for 60 minutes. The stack of sieves was then removed from the mechanical shaker and the mass of each sieve and retained material was weighed and recorded in the notebook. The mass of the retained material per sieve was calculated by subtracting the mass of the sieve initially recorded from the mass of the sieve recorded after sieving and the results used to plot a gradation curve.

© University of Venda

Figure 3.11: Stack of sieves with samples on the mechanical shaker.

3.3.5 Atterberg limit tests

Atterberg limit tests basically measure the critical water content of a fine-grained soil, its shrinkage, plastic and liquid limits. Variation in water content in soil makes soil to appear in four stages namely; solid, semi-solid, plastic and liquid. Engineering properties of soil depend on these stages; thus, soil behaves differently from each stage. Atterberg limits are used to distinguish soil between silt and clay. These tests were conducted on all six waste rock samples. The finer material that passed through the 0.075 mm sieve to the pan was used to conduct these tests.

Determining the liquid limit

Liquid Limit (LL) refers to the moisture content at which it will take 25 blows to close a grove distance of 13 mm. This is the moisture content at which the soil starts behaving as liquid. It is important because it assists in planning geotechnical project such as road construction or selecting the most appropriate material for subgrade since the material which acts as a liquid at low moisture content makes the road

© University of Venda

being more susceptible to damage during moist or slightly moist condition. The equipment or apparatus used to determine the Atterberg limits were the Cassagrande tool, a spatula, soil specimen, moisture cans and a grooving tool.

The soil collected from the pan was transferred into a clean dish and distilled water was added. Soil was mixed with distilled water in a bottle until the soil becomes a paste. Distilled water was used to prevent any reaction between ions in tap water and those in the clay soil. A small portion of the paste was placed on a clean Casagrande's cup. The height of the cup was levelled to ensure that the distance at which it was dropped to the hard rubber base was roughly 10 mm. The grooving tool was used to cut a groove of 13 mm at the centre of the paste in the Casagrande's cup thus dividing the paste into two as shown in Figure 3.12.

Figure 3.12: Determining Liquid Limit using Cassagrande apparatus.

The crank was used to lift its cup and drop it several times onto the hard rubber base at 2 drops per second. The aim was to acquire the moisture content at which the groove in the centre closes after 25 blows. The cup was dropped and when the groove closed, a slice of the paste was collected and placed onto the weighing trays and weighed. The weight as indicated by the balance scale was recorded and the paste sampled was placed into the oven to dry over 24 hours at 110°C. The paste in

© University of Venda

the weighing trays was then removed and weighed again. The moisture content at which the groove closed was determined by subtracting the dried mass from the mass weighed before the paste dried. Three trials were carried out for each sample.

Determining the Plastic Limit

The Plastic Limit (PL) refers to the moisture content in percentage, at which the soil will crumble when it is rolled in threads of 3.2 mm. The plastic limit is the lower limit of the plastic stage of soil (Das, 2008).

The ellipsoidal size soil mass was rolled repeatedly on the glass plate by hand. In so doing, the water evaporated and the moisture content in the soil decreased, causing the soil to eventually crumble (Fig. 3.13). The crumbled Pieces of soil were collected and placed into the weighing trays. The crumbled soil was placed on the scale balance and their weight was recorded on a notebook and then they were placed into the Vacutec oven for drying over a period of 24 hours at 100°C.

Figure 3.13: Determining plastic limit by repeatedly rolling moist soil on glass plate.

It was then removed and weighed again. The dry mass of the tray with the crumbled Pieces was subtracted from the moist mass to find the moisture content. A total of 3 trials were conducted for each sample and the average moisture content was recorded to be the plastic limit of the soil mass. The difference between the Plastic Limit (PL) and the Liquid Limit (LL) results in the Plasticity Index (PI) as shown below (Das, 2010):

$$PI = LL - PL$$

© University of Venda

3.3.6 Determination of California bearing values

The California bearing ratio (CBR) test was conducted on all 6 waste rock samples. This test was done to determine the bearing capacity of sub-grade soil for design of flexible roads. The following apparatus were used; a Jack used as an extruder, Mould and standard hammer, Lubricating grease, California Bearing Ratio base plate and a steel cutting collar (Fig. 3.14).

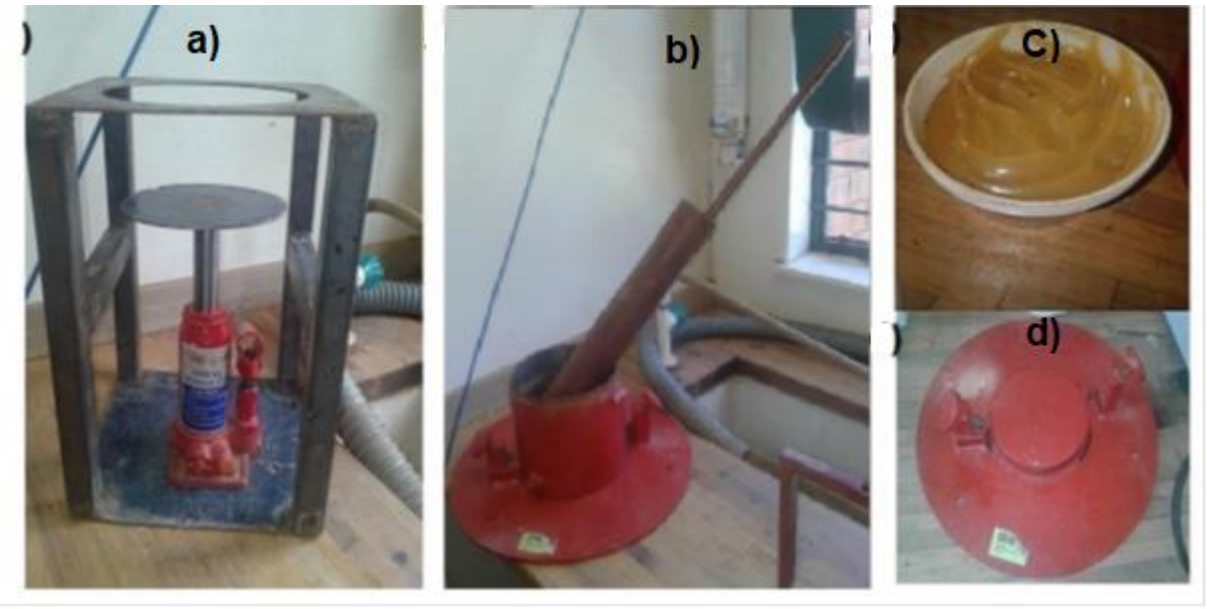

Figure 3.14: Apparatus for Laboratory Compaction test: a) Jack used as an extruder; b) Mould and standard hammer; c) Lubricating grease; and d) CBR Base plate.

The representative sample of material passing through 9.5 mm sieve was taken from the sample bag containing waste rock from the field. The sampled material was equally divided into three basins/containers. To ensure an equal division of material, a balancing scale was used. Measuring cylinders were used to add water into the three basins containing the soil samples, however, the water was in varying quantities. The soil was mixed with water as it was added, and then water was allowed to soak in for 15 minutes and the basins with samples were covered with plastic to preserve the moisture content in the soil samples.

The 150 mm mould was weighed and the mass was recorded. The mould was then placed on a base plate, screws on the base plate were used to tighten the mould and hold it in place. The material was placed into the mould in five layers and a hammer was dropped on each layer for 55 blows (equally distributed throughout the mould) from a height of 300 mm. The mould with compacted soil specimen was weighed

© University of Venda

and the mass was recorded. The mass of the mould resulted from the difference between the two masses. The diameter and the height of the compacted soil were measured and the volume was calculated using the formula (Bronshtein *et al*., 2007):

Volume = Area X Height

And the density of the soil specimen was calculated using the formula (Das, 2013):

$$\gamma\text{bulk} = \frac{Weight\ of\ Solids + Weight\ of\ Water}{Total\ Volume}$$

The compacted soil specimen was removed from the mould using a jack. The moisture content for the compacted soil specimen was determined by sampling the compacted soil extruded from the mould and weighed. The sample was left to dry overnight in the oven thermostatically controlled at 110°C and was then weighed. The difference gave the pore water weight and moisture content was calculated using the formula (Das, 2013):

$$moisture\ content = \frac{weight\ of\ water}{weight\ of\ solids} \times 100$$

© University of Venda

CHAPTER FOUR: RESULTS AND DISCUSSION

4.1 Correlation of tailings logs

Logging of tailings was done simultaneously with sample collection and this was done to delineate the oxidized zone, transitional zone and the unoxidized zone of the tailings. The extent of tailings oxidation with depth was recorded and tailings logs were correlated along each Profile (Appendix A).

Correlation along Profile 1

Tailings along Profile 1, located along the southern part consisted of dark brown and light brown material. The demarcation between the oxidized and unoxidized zone was clear due to the difference in tailings colour (Fig. 4.1). The top 0 m to 2 m of Profile 1 was mainly the dark brown material whereas between 2 m and 8 m tailings were light brown in colour. It was concluded that the oxidation of P1H1 and P1H2 extended to a depth of 1.6 m and 2 m respectively. The transitional zone within this Profile was not easy to identify.

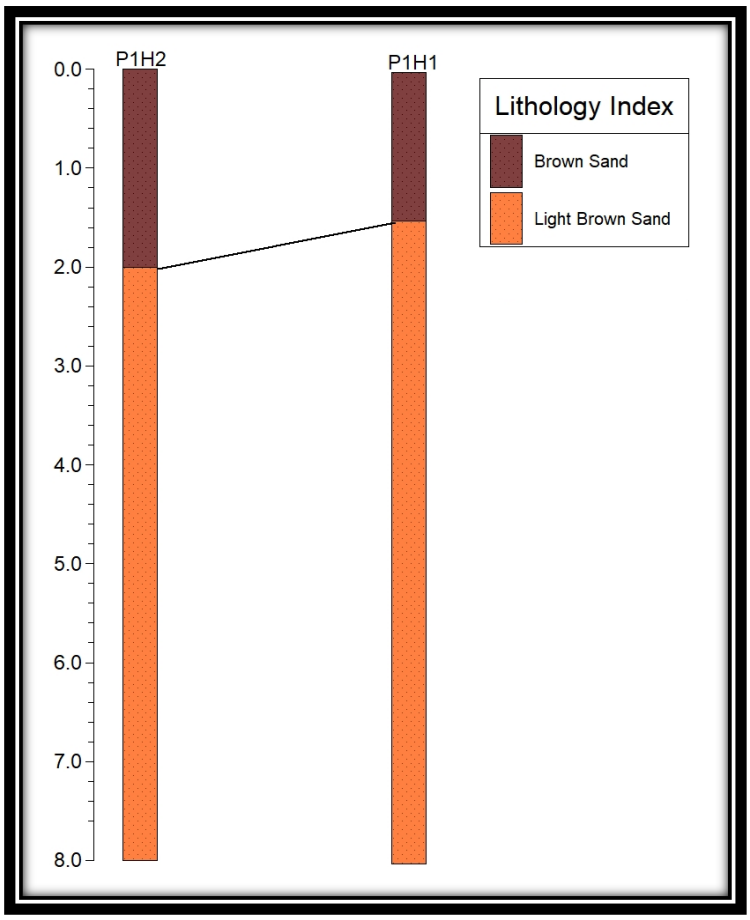

Figure 4.1: Correlation of drillhole logs along Profile 1.

© University of Venda

Correlation along Profile 2

Correlation was conducted along Profile 2 and similar characteristics of tailings as Profile 1 were observed. The colour of the tailings changed from dark brown to light brown along each borehole. The oxidized zone extended to a depth of 2 m along hole P2H2 (Fig. 4.2). The oxidized zone in holes P2H1 and P2H3 extended to a depth of 1.7 m and 1.8 m respectively. The oxidized zone was easy to auger as compared to the unoxidized zone which was quite competent and difficult to auger. The transitional zone was not easy to identify as was the case in Profile 1. The variation in oxidation zone with depth may be due to either the uneven topography of the tailings dam and water retention.

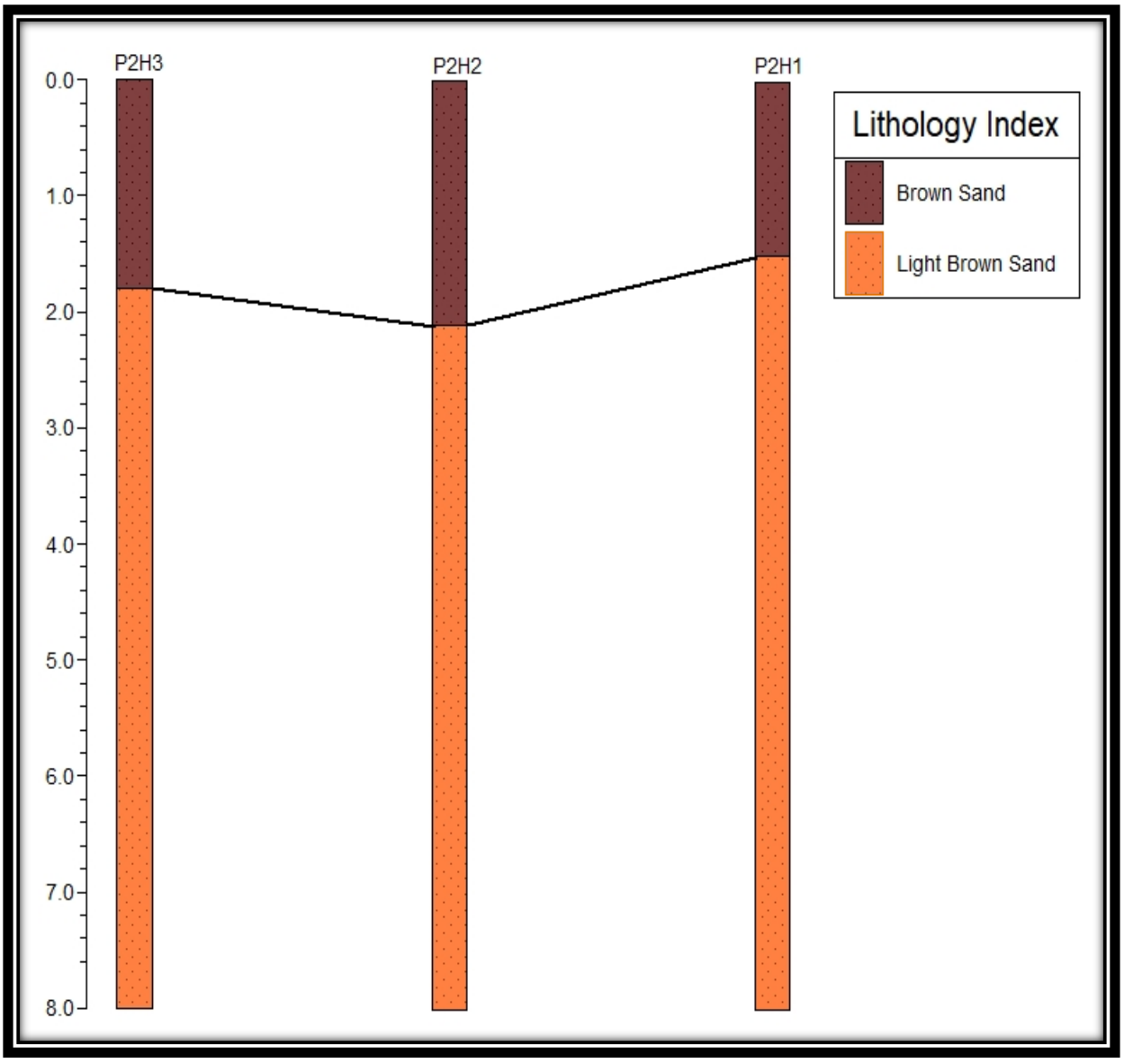

Figure 4.2: Correlation of drillhole logs along Profile 2.

© University of Venda

Correlation along Profile 3

The oxidized zone along Profile 3 was quite extensive up to a depth of 3 m along hole P3H4 and 2.7 m along hole P3H1. The oxidized zone of P3H2 and P3H3 extended to depths of 1.7 m and 2 m respectively (Fig. 4.3). The transitional zone was not distinctive along this Profile. Drillhole P3H2, P3H3 and P3H4 consisted of some sticky greyish material. This was encountered at depths of 3.6 m, 5.4 m and 6.2 m respectively. This material was grey sticky mud and was quite competent to auger and contained higher moisture content. This material extended for 0.3 m at P3H2, 0.5 m at P3H3 and 0.8 m at P3H4.

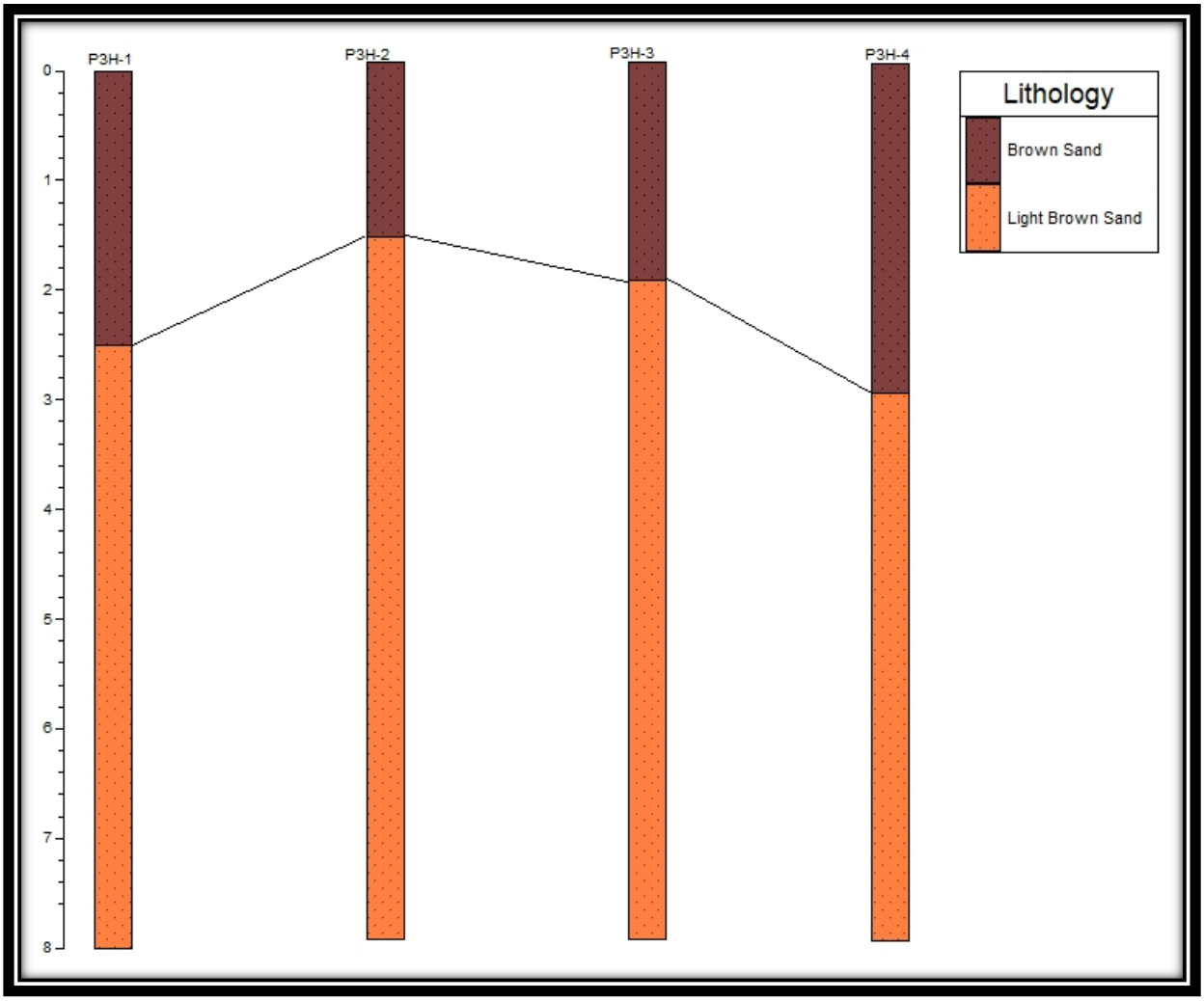

Figure 4.3: Correlation of drillhole logs along Profile 3.

Correlation along Profile 4

Profile 4 was located at the northern part of the tailings dam. The oxidized zone along Profile 4 was relatively extensive as in the case of Profile 3 where it extended

© University of Venda

to a depth of 3 m in hole P4H5. The oxidized zone of P4H1, P4H2, P4H3 and P4H4 extended to depths of 1.6 m, 2.0 m, 2.6 m and 2.4 m respectively. Like in Profile 1 and 2, the transitional zone was not distinctive. The greyish sticky material was encountered in holes P4H1, P4H2 and P4H4 at a depth of 4.2 m, 4.4 m and 6 m respectively (Fig. 4.4). This material extended for 0.3 m in hole P4H1, 0.7 m in hole P4H2 and 0.6 hole P4H4.

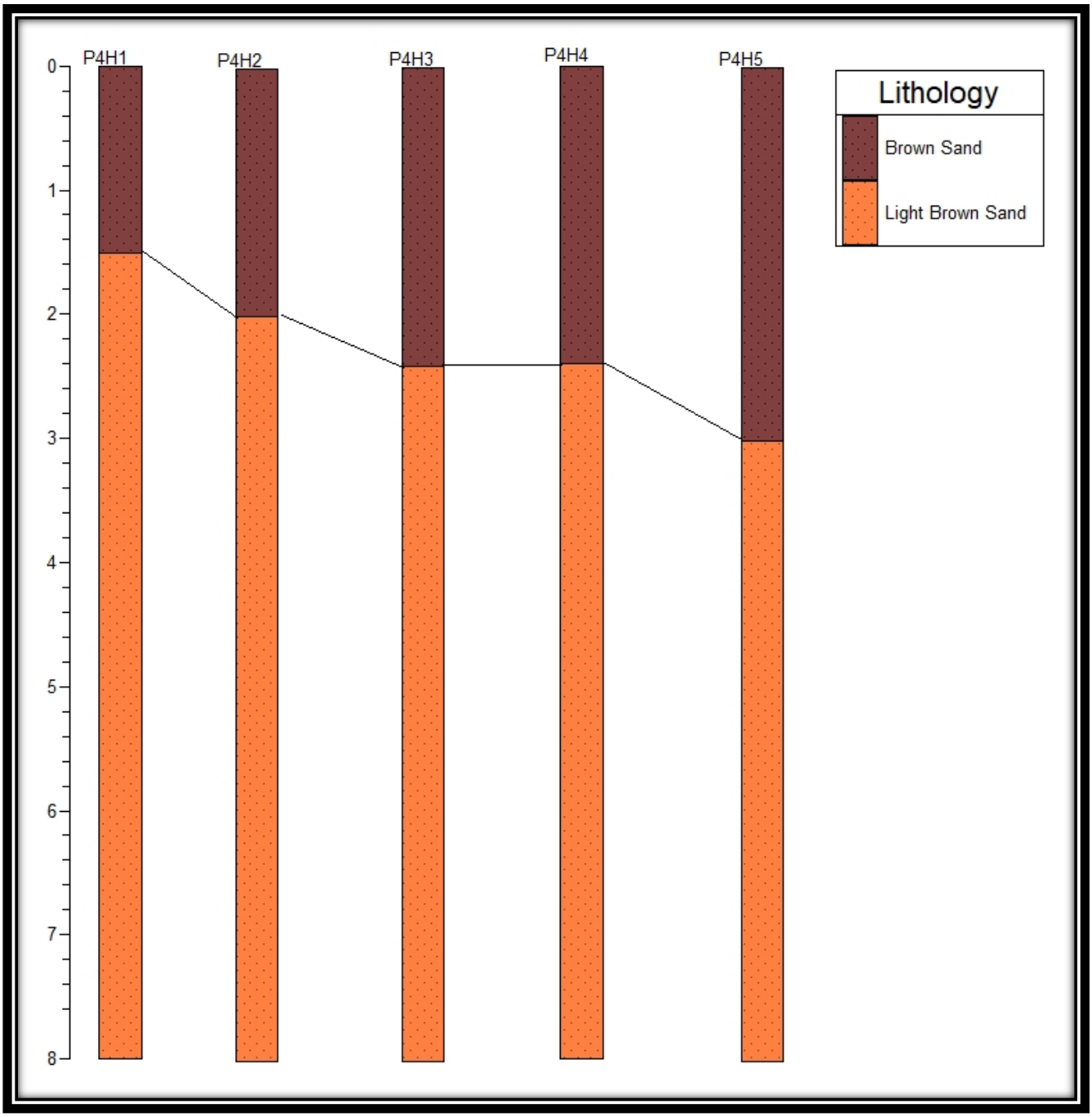

Figure 4.4: Correlation of drillhole logs along Profile 4.

© University of Venda

In general, the oxidation of tailings at Consolidated Murchison mine tailings dam was uniform. The oxidation zone ranged from 1.6 m in hole P1H1 to 3 m in holes P3h4 and P4H5. Limitation of oxidation zone to the first 3 m of the dam can be due to the fact that moisture and oxygen was only limited to the topper part of the dam. Oxidation was generally extensive on the western part of the tailings dam. This may be due to the uneven topography of the tailings dam and uneven infiltration of water along the tailings dam. The transitional zone was generally not distinctive although a layer of sticky material was identified along Profile 3 and 4 that are attributed to this zone

4.2 Evaluation of gold values within the tailings dam

Statistical analysis of heavy metals was conducted using the method of (Utochkin, 2015):

Range

Range refers to the difference between the largest and the smallest observed value of each sample. The range was determined using equation 1.

Range (r) = Max. Observed – Min. Observed ……. Equation 1

Mean

Mean was determined by summing the observed values of the sample and divide by the total number of samples. Equation 2 was used to determine the mean.

$$\text{Mean } (m) = \frac{\sum(X_1 + X_2 \ldots + X_n)}{n} \quad \text{..... Equation 2}$$

Where; $\sum$ = sum of all observed values

X = observed value of each sample

n = total number of samples

Median

Median is the middle value of a set of numbers after rearranging such numbers in numerical order.

© University of Venda

Variance

Variance is used to measure how far a set of numbers are spread out. Variance was determined by using equation 3.

$$S^2 = \frac{\sum(X_1 - \bar{x})^2}{n - 1}$$ …….. Equation 3

Where; S^2 = Variance

$\sum$ = sum of all observed values

X_1 = sample observation

$\bar{x}$ = the mean

n = total number of samples

Standard deviation

Standard deviation refers to the measure that is used to quantify the amount of variation or dispersion of a set of data values. Standard deviation was determined by using equation 4.

$$s = \sqrt{\frac{\sum(x - \bar{x})^2}{N - 1}}$$ ……..……Equation 4

Where; S = the standard deviation

X = sample observation

$\bar{x}$ = the mean

N = total number of samples

Threshold

Threshold refers to the mean multiplied by two times the standard deviation (Bland and Altman, 1996). Threshold was determined using equation 5.

Threshold (Th) = X x 2S.....Equation 5

© University of Venda

Where: $\mathcal{X}$ = Mean

S = Standard deviation

Samples from the first 6 m of each hole were analysed for gold, thus a total of 84 samples were analysed and results were presented in Appendix B. The lowest and highest gold values were found to be 200 mg/kg and 1880 mg/kg respectively with an average value of 670 mg/kg (Table 4.1). Based on the distribution pattern of gold as shown on figure 4.5 to figure 4.8, the distribution of gold within the tailings dam was generally erratic.

Table 4.1: Statistical analysis of gold within the tailings dam

	Au (mg/kg)
Number of samples	84
Minimum	200
Maximum	1880
Range	1680
Mean	670
Median	660
Variance	40
Standard deviation	200
Threshold	270

4.3 Distribution of gold within Consolidated Murchison mine tailings dam

Distribution of gold along Profile 1

The distribution of gold along Profile 1 was erratic although along P1H2 there was a general decrease with depth (Fig. 4.5). Gold values ranged from 200 mg/kg to 900 mg/kg. The lowest value of gold was observed at a depth of 2-3 m along hole P1H2 and the highest value was recorded along hole P1H1 at a depth of 1 m. The average gold value along this Profile was 611 mg/kg.

© University of Venda

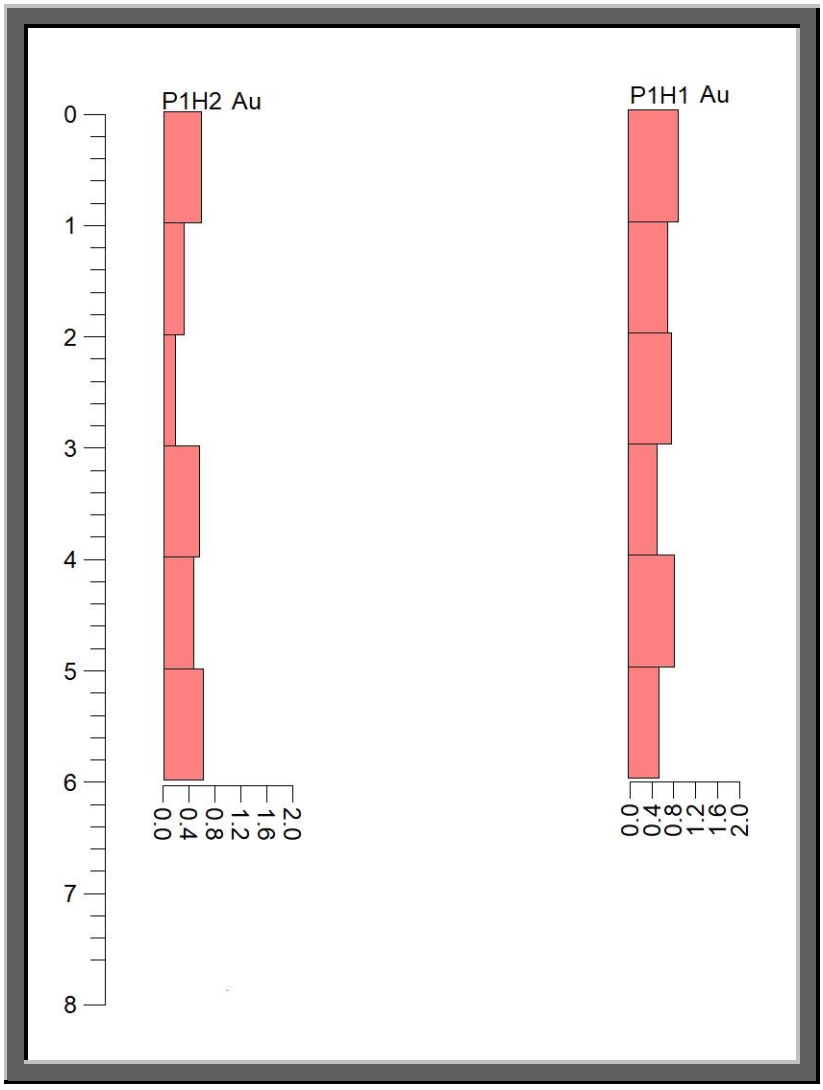

Figure 4.5: Distribution of gold along Profile 1.

Distribution of gold along Profile 2

The distribution of gold along Profile 2 was generally erratic within hole P2H1, P2H2 and P2H3. Gold values were increasing and decreasing from one meter to the next along P2H2 and P2H3 (Fig. 4.6). In hole P2H1, gold values decreased with depth from the surface to a depth of 5 m from which there was an increase. The lowest and highest values of gold along Profile 2 were recorded to be 410 mg/kg and 910 mg/kg respectively. The lowest value was recorded at a depth of 4-5 m in hole P2H1 and the highest value was recorded at 1 m depth in hole P2H1. The average value of gold along Profile 2 was found to be 656 mg/kg.

© University of Venda

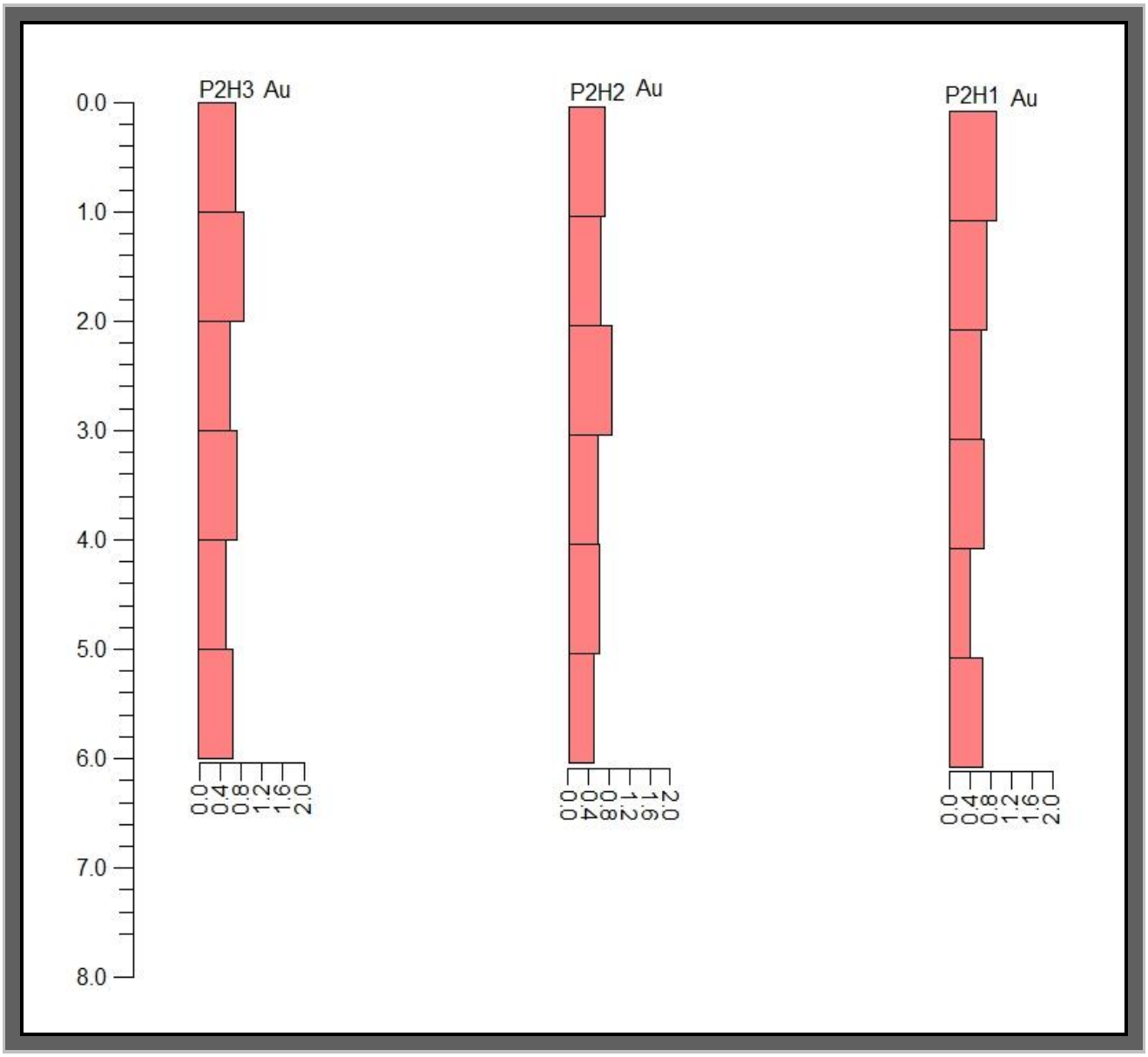

Figure 4.6: Distribution of gold along Profile 2 within the three boreholes.

Distribution of gold along Profile 3

The distribution of gold along this profile was decreasing with depth along hole P3H2 and P3H3. In hole P3H1, gold values were generally increasing with depth (Fig. 4.7). Gold distribution in hole P3H4 was found to be erratic. The average value of gold was calculated to be 747 mg/kg with the lowest and highest values found to be 450 mg/kg and 1880 mg/kg respectively. The lowest value occurred in hole P3H2 at 5-6 m depth and the highest value was found in hole P3H4 at 3-4 m depth.

© University of Venda

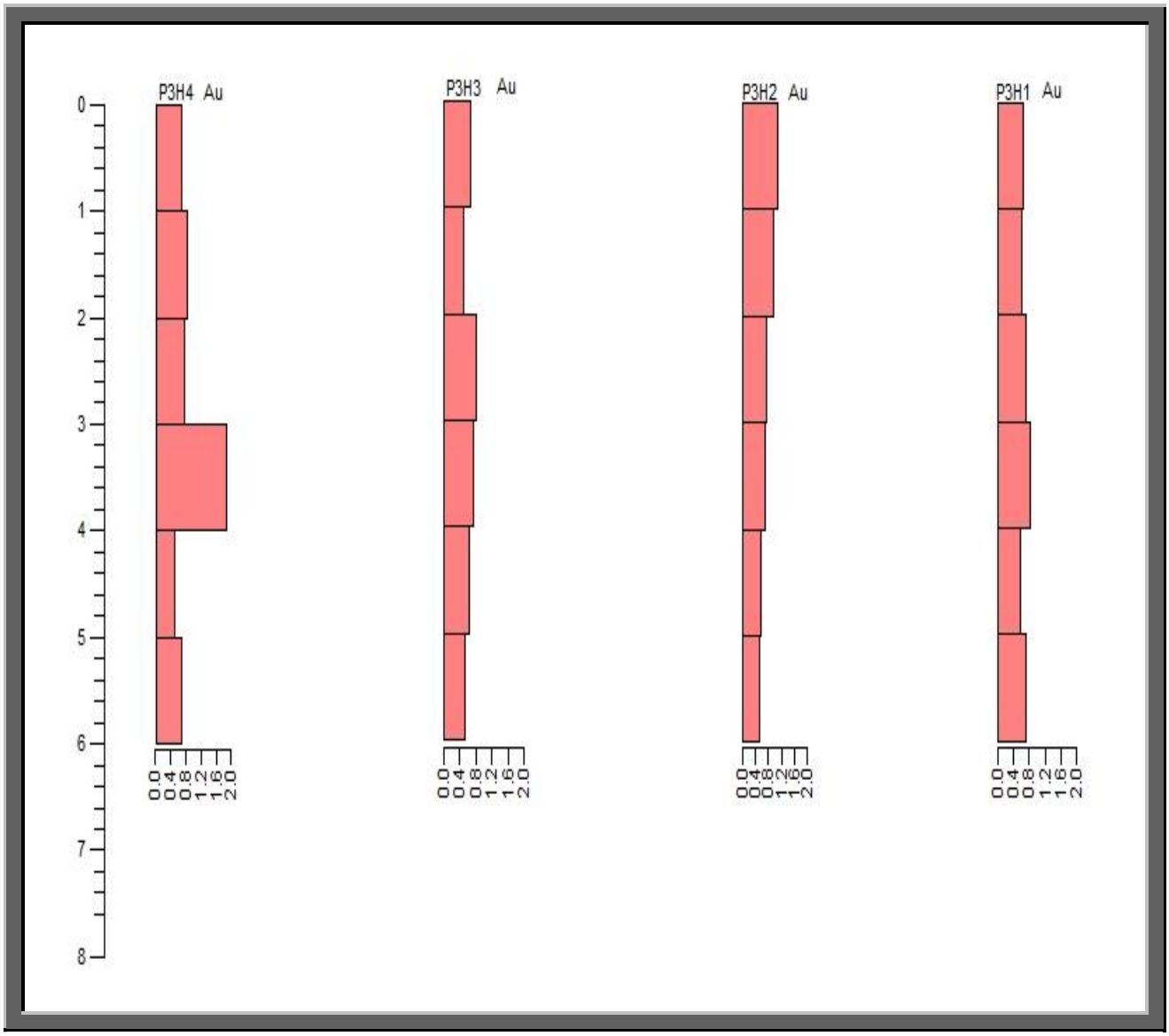

Figure 4.7: Distribution of gold along Profile 3 within boreholes P3H1-P3H4.

Distribution of gold along Profile 4

Along boreholes P4H1 and P4H4, gold values were generally increasing with depth, whereas there was a general decrease in values with depth along borehole P4H2 (Fig. 4.8). Gold values along boreholes P4H3 and P4H5 were more or less uniform with depth. The lowest and highest values were observed along hole P4H1 and P4H4 respectively. Gold values along this Profile ranged from 280 mg/kg in P4H1 to 950 mg/kg in hole P4H4. The average value was calculated to be 639 mg/kg.

© University of Venda

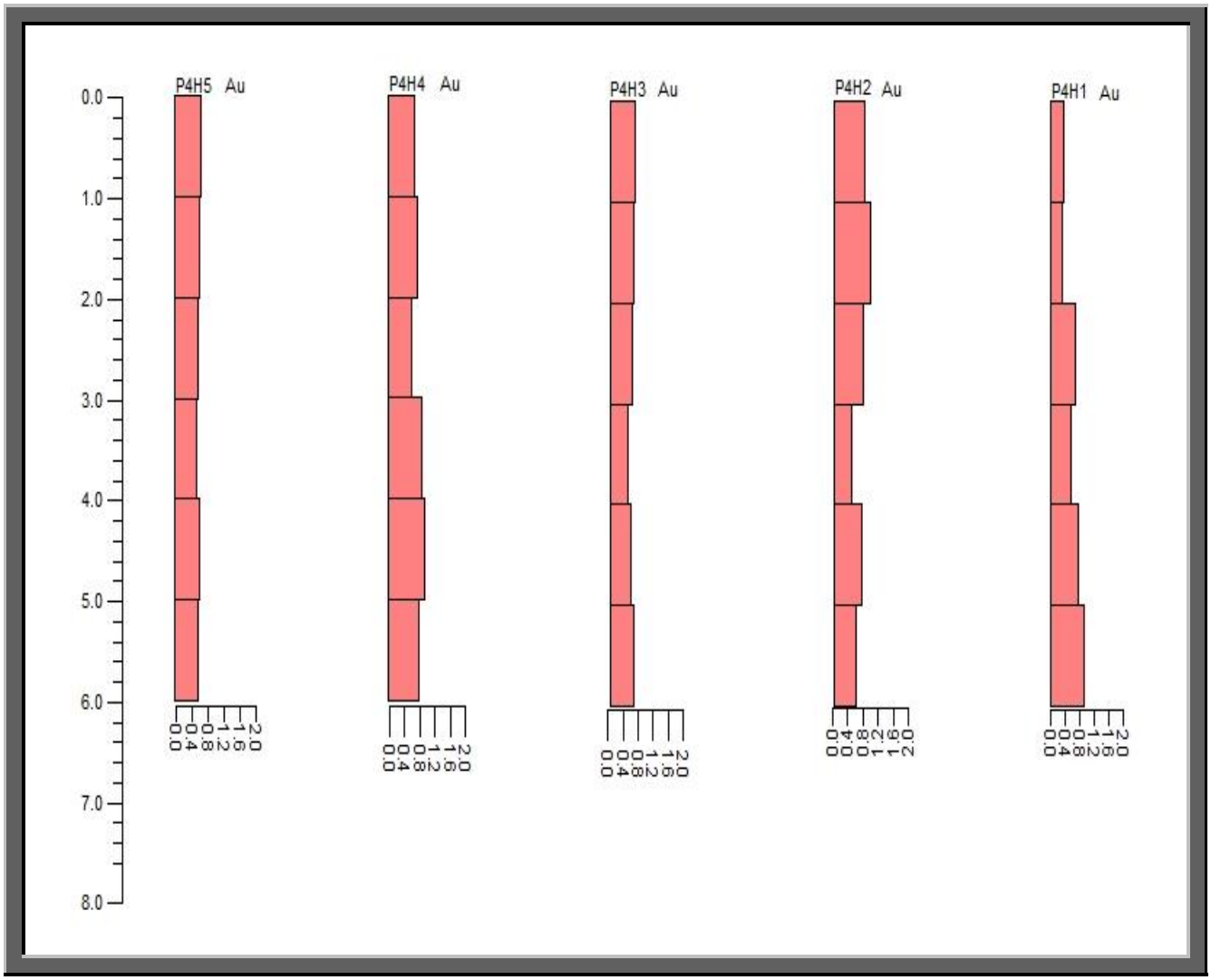

Figure 4.8: Distribution of gold along Profile 4 within boreholes P4H1-P4H5.

4.4 Estimation of gold content within the tailings dam

The geochemical data analysis for gold that was obtained was used to estimate the total amount of gold within the tailings dam. To ascertain the total gold content within the tailings dam, it was necessary to determine the area of the entire tailings dam, the height, volume, tonnage and the value of gold within the tailings dam

The total area and height of the tailings dam were determined to be 250 100 m^2 and 30 m respectively. The volume of the tailings dam was calculated by multiplying the total area with height, with the assumption that the dam was regular:

Where V= $a X h$

= 250 100 m^2 X 30 m

= 7 503 000 m^3

© University of Venda

The tonnage of the tailings dam was determined by multiplying the volume by density of the material. The average density of the material was considered to be 1.77 kg/m^3 (Ramuada, 2014).

Therefore: Q = Vd, where Q = Tonnage,

V = Volume, and

d = Density

= 7 503 000 X 1.77

= 13 280 310 tons

The value of gold within the tailings was calculated using the following formula:

P = QC, Where P = metal value in grams,

Q = tailings tonnage, and

C = average gold value

$$= \frac{13\ 280\ 310 \times 0.67}{1000}$$

= 8 897. 81kg

Gold values within the Consolidated Murchison tailings dam were high since the highest value was 1880 mg/kg. The average value of gold within this tailings dam was 670 mg/kg. This value is high being compared to the average value observed at the tailings of East Rand Gold and Uranium Company (ERGO) which was found to be 530 mg/kg (www.miningreview.com/news/ergo-to-be-reborn/). A similar programme of gold recovery commenced at about the same time at the Central Rand on tailings dams averaging about 400 mg/kg gold and sand dumps averaging about 600 mg/kg (Viljoen, 2009). The lowest Au value was found to be 200 mg/kg. In general, there was an erratic distribution of gold with depth. It was assumed that the erratic behaviour of gold with depth continued to the bottom of the tailings dam.

ERGO is reprocessing tailings dams using hydraulic reprocessing method which can also be effective in reprocessing Consolidated Murchison tailings. The total amount of gold within the Consolidated Murchison tailings dam was determined to be 8 897. 81 kg. The total amount of tonnes within this tailings dam were 13 280 310 tons. In

© University of Venda

British Columbia, Carolin mine which produced gold and silver in the past is an ideal case (Xinyi, 2012). It had drilling tests and fire assaying results which had proven reserves of almost 800 000 tonnes with gold values of 1 740 mg/kg. The tonnage of gold within the Carolin mine tailings dam was found to be lower than those of Consolidated Murchison mine which makes tailings at this mine reprocessible. High values of 583 mg/kg were found by Matshusa *et al*. (2012) in South Africa. High values of 11, 200 mg/kg were recorded in Tanzania by Bitala *et al*. (2009).

The amount of gold within this tailings dam was found to be 8 897. 81 kg. According to Dehghani *et al*. (2009) microscopic analysis of the Moutech gold tailings dam revealed that sulphide minerals and active carbon are the main sources of gold in the tailings residues. They conducted floatation and cyanidation tests on tailings and were able to recover about 87.79% of gold, which was associated with sulphide minerals were, active carbon was separated using crude oil as the collector and Aeorofloat 39 as the frother. Potassium ethyl xanthate was used as the collector for conditioning the pulp and Sacsol 95 was used as the collector and Aeorofloat 39 as the frother to recover. The dominant suphide ore minerals hosting free gold and gold telluride within the ore were noted to be pyrite and chalcopyrite (Dehghani *et al.,* 2009). Similar methods may be applied at Consolidated Murchison mine tailings dam to reprocess gold.

Assuming that similar method of floatation and cyanidation used at Moutech gold tailings dam is to be applied at the Consolidated Murchison tailings dam at gold recovery of 87.79%, approximately 7 811.37 kg of gold can be recovered from the Consolidated Murchison tailings dam. Currently, the price of gold is $39 293.03/kg, thus a total of US$ 306 932 396.00 (R 4 281 706 924.20) in monetary terms can be reprocessed from this tailings dam.

4.5 Statistical analysis of heavy metals within the tailings dam

Calculations of statistical summary of heavy metals within the tailings dam of the Consolidated Murchison mine were done and the same formulae for gold analysis were used to get such statistical summary. Heavy metal data from XRF spectrometry were presented in Appendix C. From these data, the highest values of heavy metals were recorded for Pb, Ni, Cr and As with values of (ppm); 9471.8, 2817, 1824.2 and 1016.6 respectively. Cd was found to have the lowest values below 1 ppm alongside

© University of Venda

with Cu, Co and Zn with minimum values of (ppm); 2.3, 16.7 and 49.6 respectively (Table 4.2). Variance, Standard deviation and threshold were calculated and presented in Table 4.2 below. This was done to quantify the dispersion of heavy metals within the tailings dam.

Table 4.2: Statistical analysis of heavy metals within the tailings dam

	Cr (PPM)	Co (PPM)	Ni (PPM)	Cu (PPM)	Zn (PPM)	As (PPM)	Cd (PPM)	Pb (PPM)
Number of samples	112	112	112	112	112	112	112	112
Minimum	934.8	16.7	1690.9	2.3	49.6	432.9	0	4033
Maximum	1824.2	25.1	2817	261	91.7	1016.6	0.1	9471.8
Range	889.4	8.4	1126.1	258.7	42.1	583.7	0.1	5438.8
Mean	1345	19.8	2062.6	41.9	70	604.4	0.01	5631.5
Median	1340.8	19	1850.4	37.25	10.4	80.8	0	5228.6
Variance	2052245.5	176	23293.07	72952	5634.4	738651	0.06	64125715
Standard deviation	1432.6	13.3	171.2	270.1	75.1	859.4	0.3	8007.9
Threshold	3853694	526.68	706234.2	22634.4	10514	1038842.7	0.006	90192977.7

4.6 Distribution of heavy metals within the tailings dam

Data attained from X-Ray Fluorescence Spectrometry of heavy metals in Appendix C were used to analyse the distribution of heavy metals with depth of the tailings dam. The outcomes of the analyses are presented in Figures 4.9-4.22.

© University of Venda

Distribution of heavy metals along borehole P1H1

Heavy metals within borehole P1H1 generally showed erratic distribution with depth as shown in Figure 4.9. Cd showed a uniform distribution with depth but elevated values between 1 to 2 m depth (Fig. 4.9). However, some metals, for example, Cr, Co and Zn had relatively even distribution with depth. The values of Cu and Cd were extremely low and erratic.

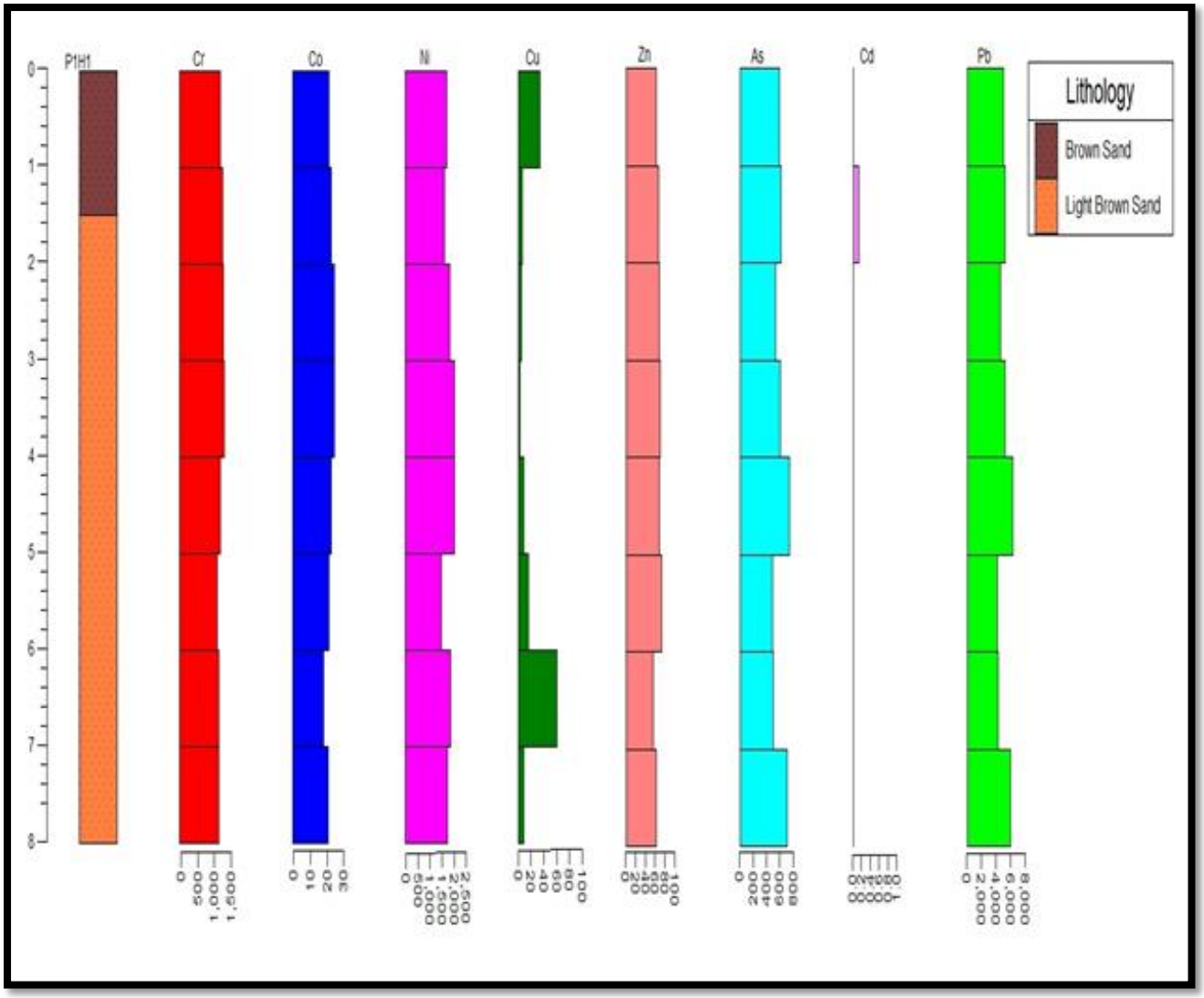

Figure 4.9: Distribution of heavy metals along borehole P1H1.

Distribution of heavy metals along borehole P1H2

Heavy metals within borehole P1H2 were generally changing with depth as illustrated on Figure 4.10. The distribution of Cr, Co and Ni were more or less uniform with depth. Cd within this borehole was below the detection limit of 0.1

© University of Venda

(Appendix C) throughout the entire borehole. The distribution of Cu was rather erratic. However, As and Pb were generally decreasing with depth and Zn was found to be generally increasing with depth. The highest value recorded was 8686.2 ppm in Pb at a depth of 2-3 m.

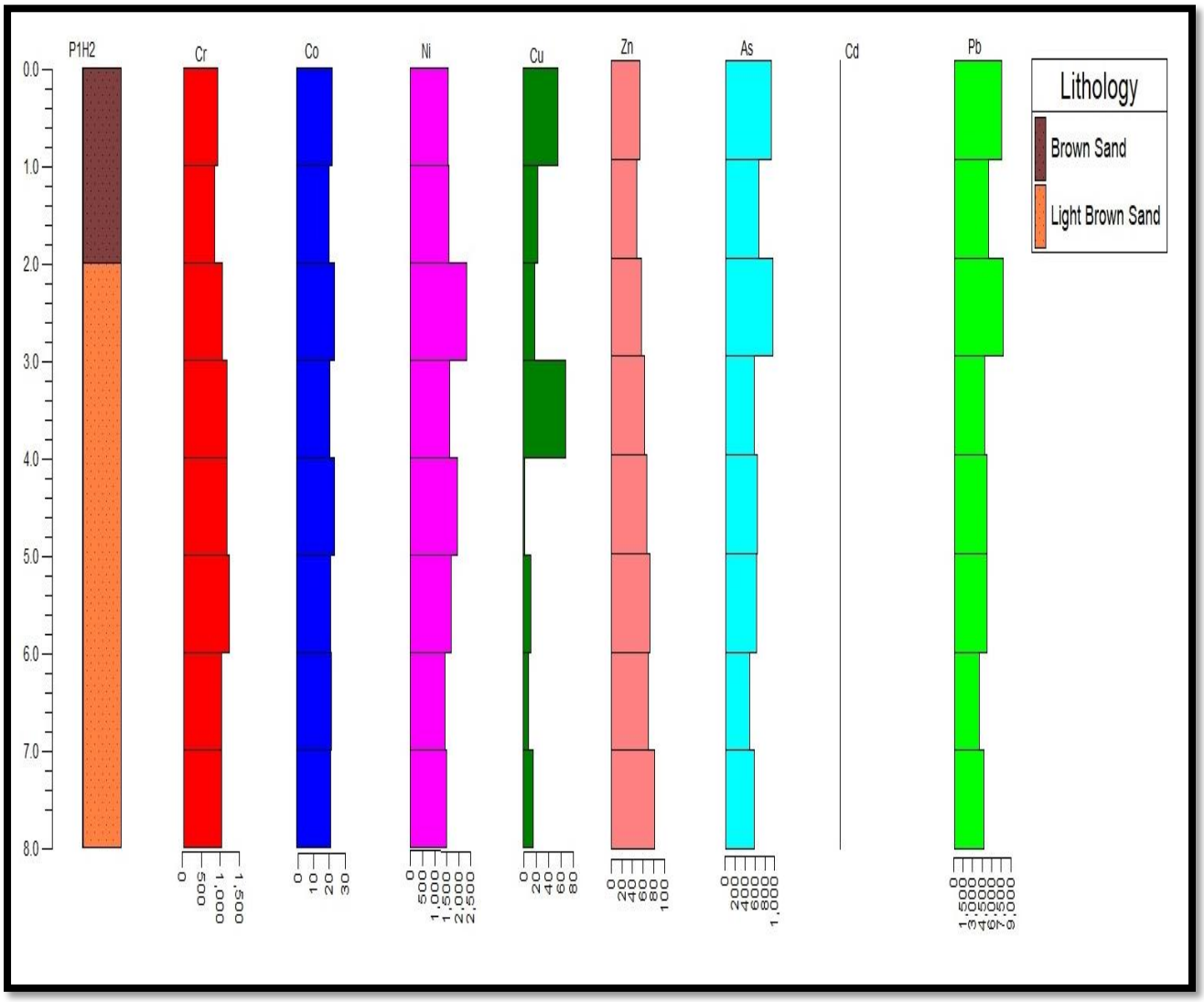

Figure 4.10: Distribution of heavy metals along borehole P1H2.

Distribution of heavy metals along borehole P2H1

The behaviour of heavy metals within borehole P2H1 were well illustrated on figure 4.11 below. Cr, Co and Ni were more or less. As and Pb were more or less uniform apart from the first one meter and minor variations.. Cd was below the detection limit throughout the entire borehole as was the case in borehole P1H2. In general, Zn was found to be increasing with depth. Cu was found to be decreasing with depth

© University of Venda

from 0-5 m but increasing from 5-8 m (Fig. 4.11). The lowest and highest values recorded were 0 ppm and 7508.3 ppm in Cd and Pb respectively.

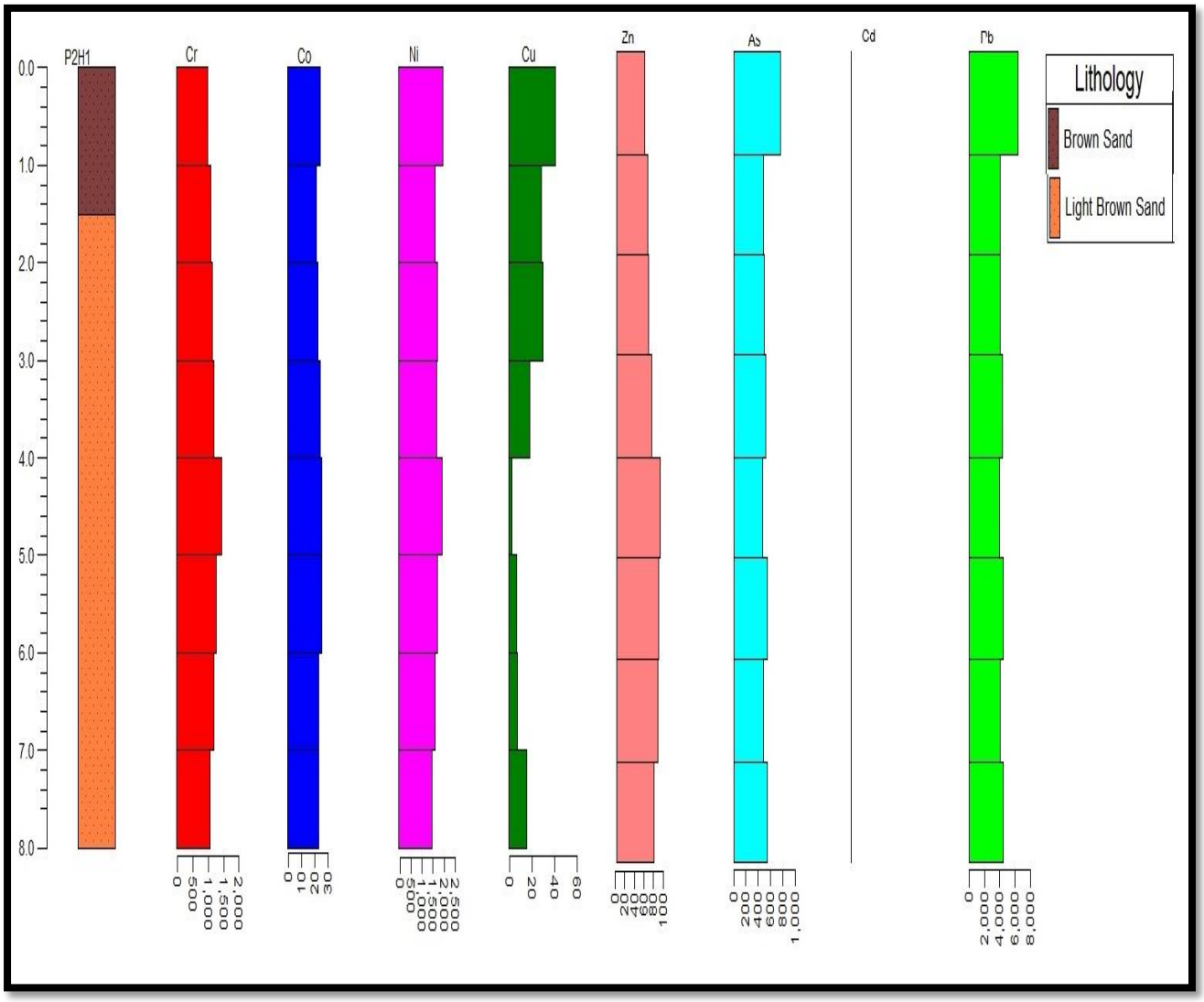

Figure 4.11: Distribution of heavy metals along borehole P2H1.

Distribution of heavy metals along borehole P2H2

The second borehole (P2H2) along Profile 2 showed different trends of heavy metals distribution with depth. Cr, Co and Ni were more or less uniform throughout the entire borehole. Zn was perfectly increasing with depth from 0-8 m. However, As and Pb had a similar erratic distribution with depth throughout the entire borehole and Cu was rather erratic (Fig. 4.12). Cd was below the detection limit except at 4-7 m.

© University of Venda

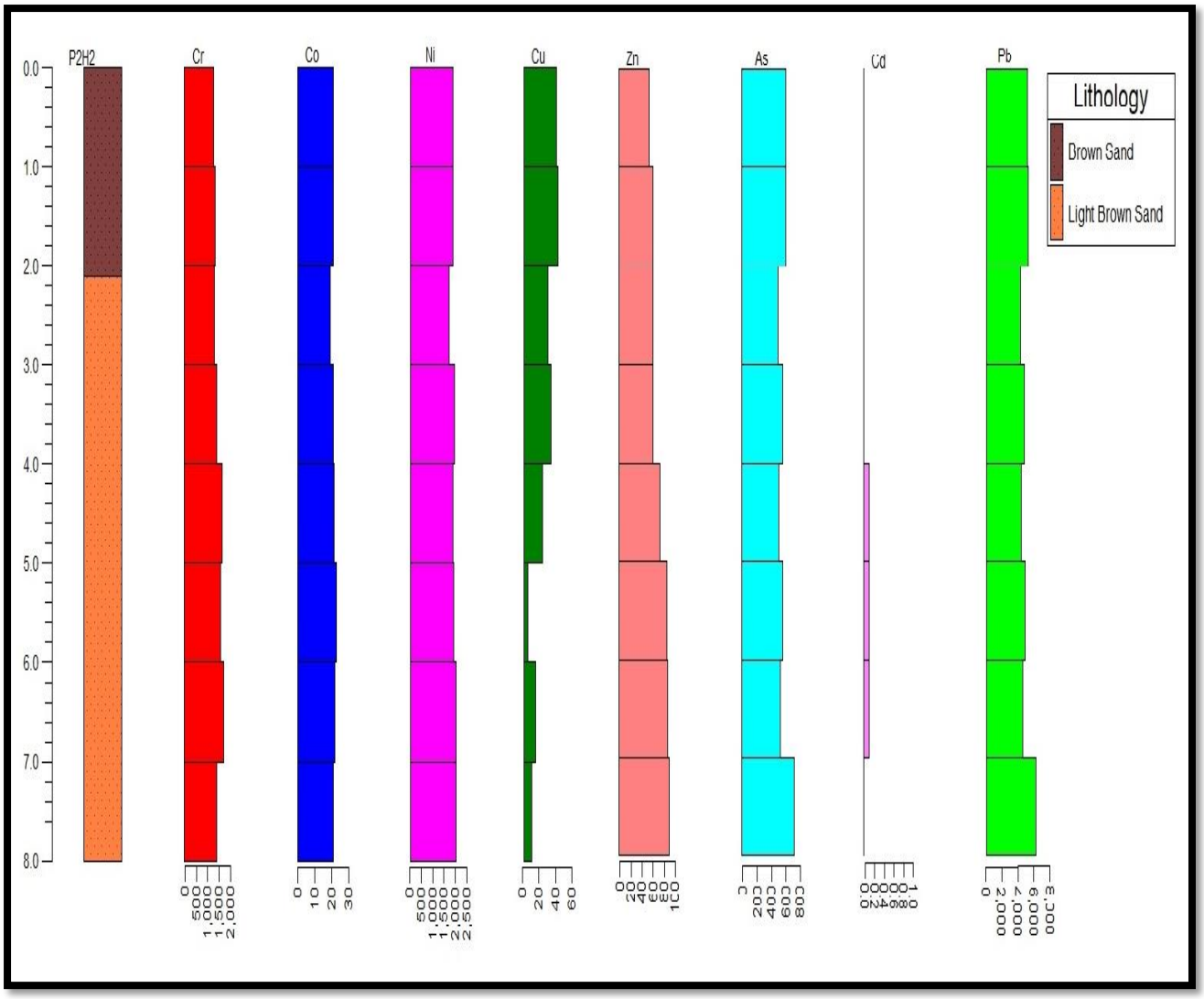

Figure 4.12: Distribution of heavy metals along borehole P2H2.

Distribution of heavy metals along borehole P2H3

In borehole P2H3, Cr and Cu were generally erratic and Zn was generally increasing with depth. Co and Ni were more or less uniform, however, As and Pb appeared to be more or less uniform with minor variations (Fig. 4.13). Cd was below the detection limit except at a depth of 5-6 m where a value of 0.1 ppm was detected. The highest value of 6234.5 ppm was recorded in Pb at at a depth of 0-1 m.

© University of Venda

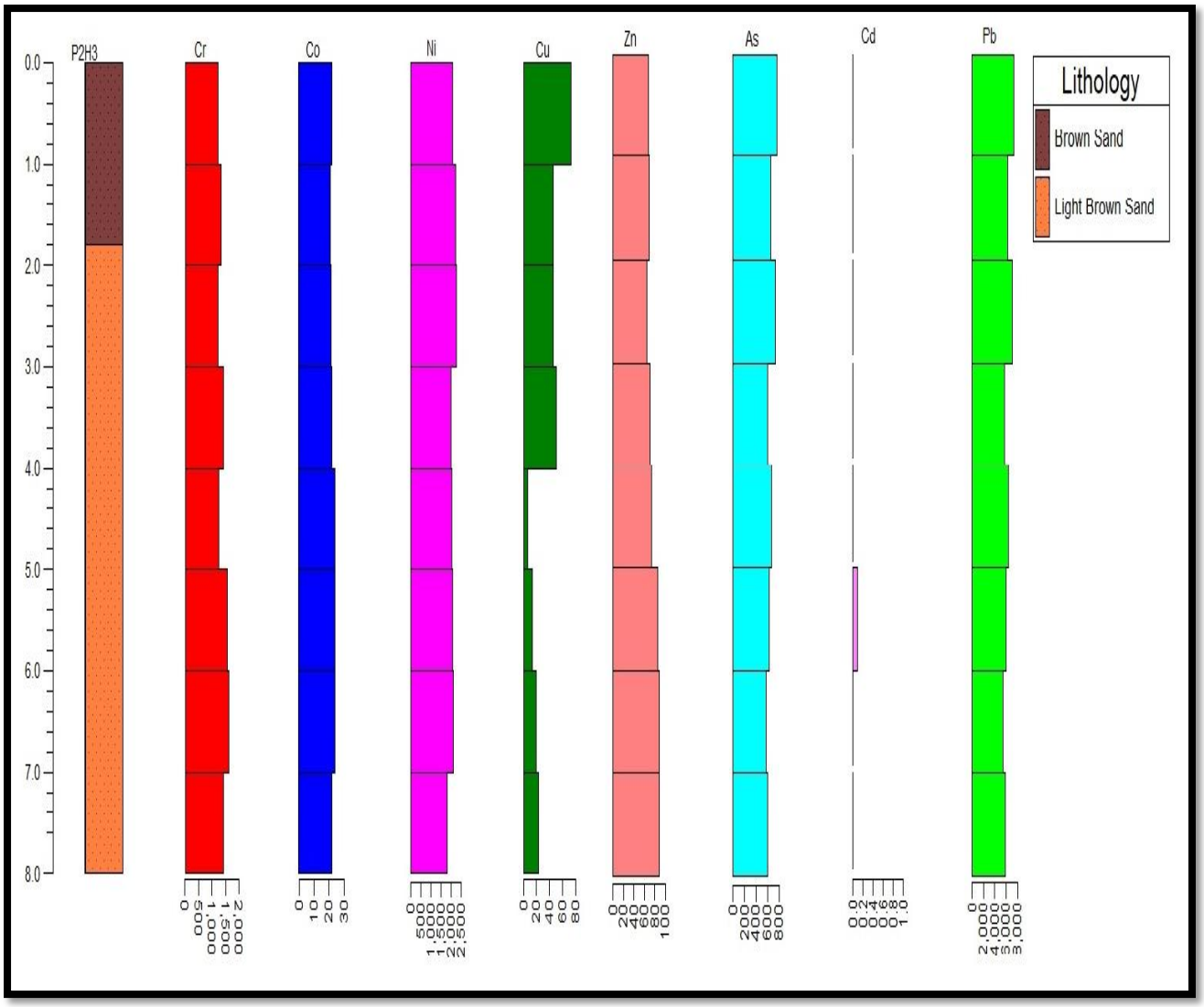

Figure 4.13: Distribution of heavy metals along borehole P2H3.

Distribution of heavy metals along borehole P3H1

Heavy metals within borehole P3H1 had different behaviour with depth. The distribution pattern of Cr, Co and Zn was generally uniform with a slight increase in Zn at a depth of 5-6 m. Ni showed a more or less uniform trend except between 1-3 m where there was an increase with depth (Fig. 4.14). Pb was more or less uniform except from 1-3 m from the surface. Cu was generally erratic with depth. A general decrease with depth was observed in As, however, Cd was below the detection limit with depth except at 2-3 m where a value of 0.1 ppm was detected. The highest metal value of 6866.8 ppm was observed in Pb at a depth of 3 m.

© University of Venda

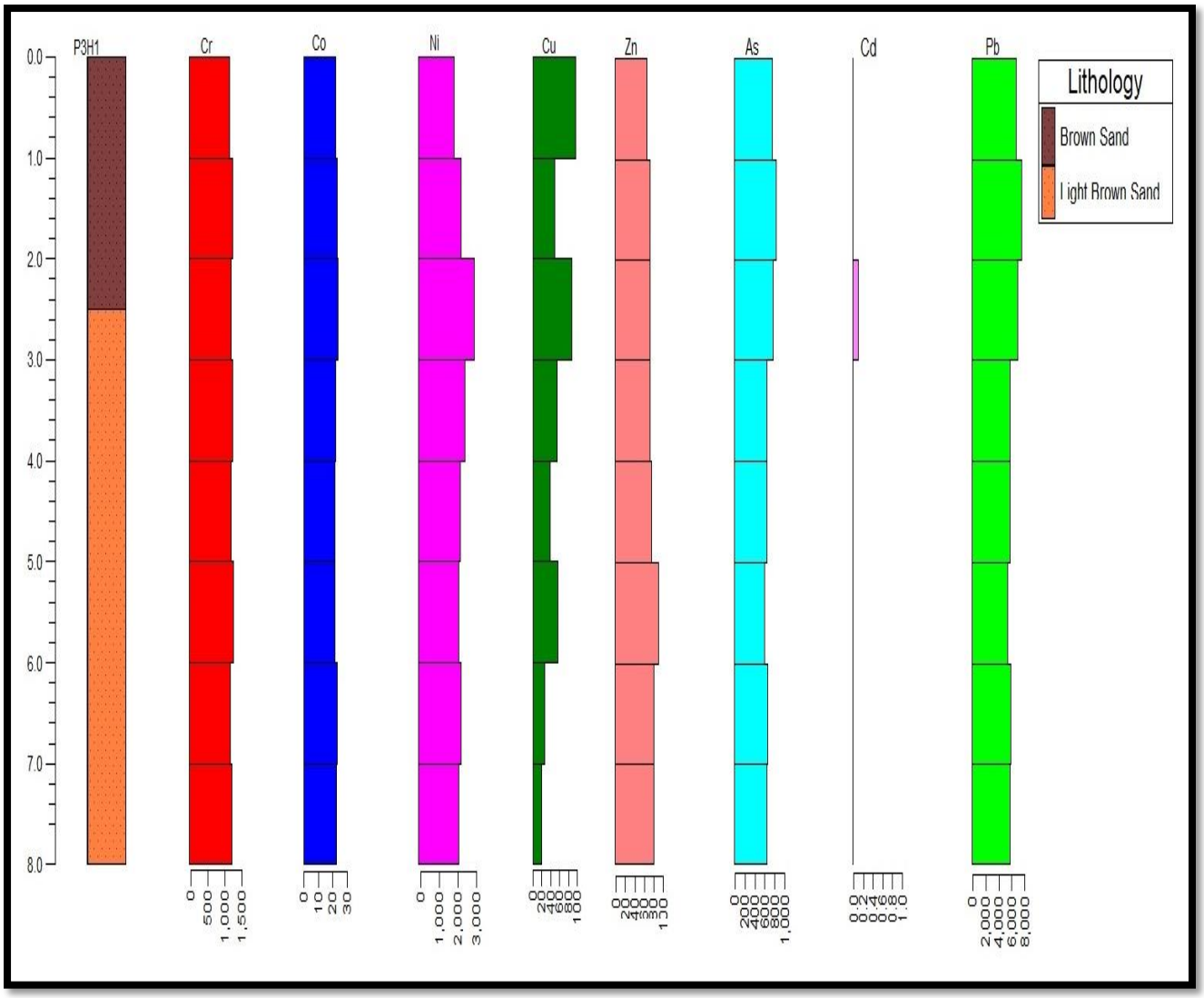

Figure 4.14: Distribution of heavy metals along borehole P3H1.

Distribution of heavy metals along borehole P3H2

In borehole P3H2, Cr, Ni, Cu, Zn and As showed a more or less erratic behaviour with depth. Cd was below the detection limit throughout the borehole except at 4-5 m. However, Co and Pb were more or less uniform except the first one meter where the highest value of 6524.4 ppm was recorded in Pb (Fig. 4.15).

© University of Venda

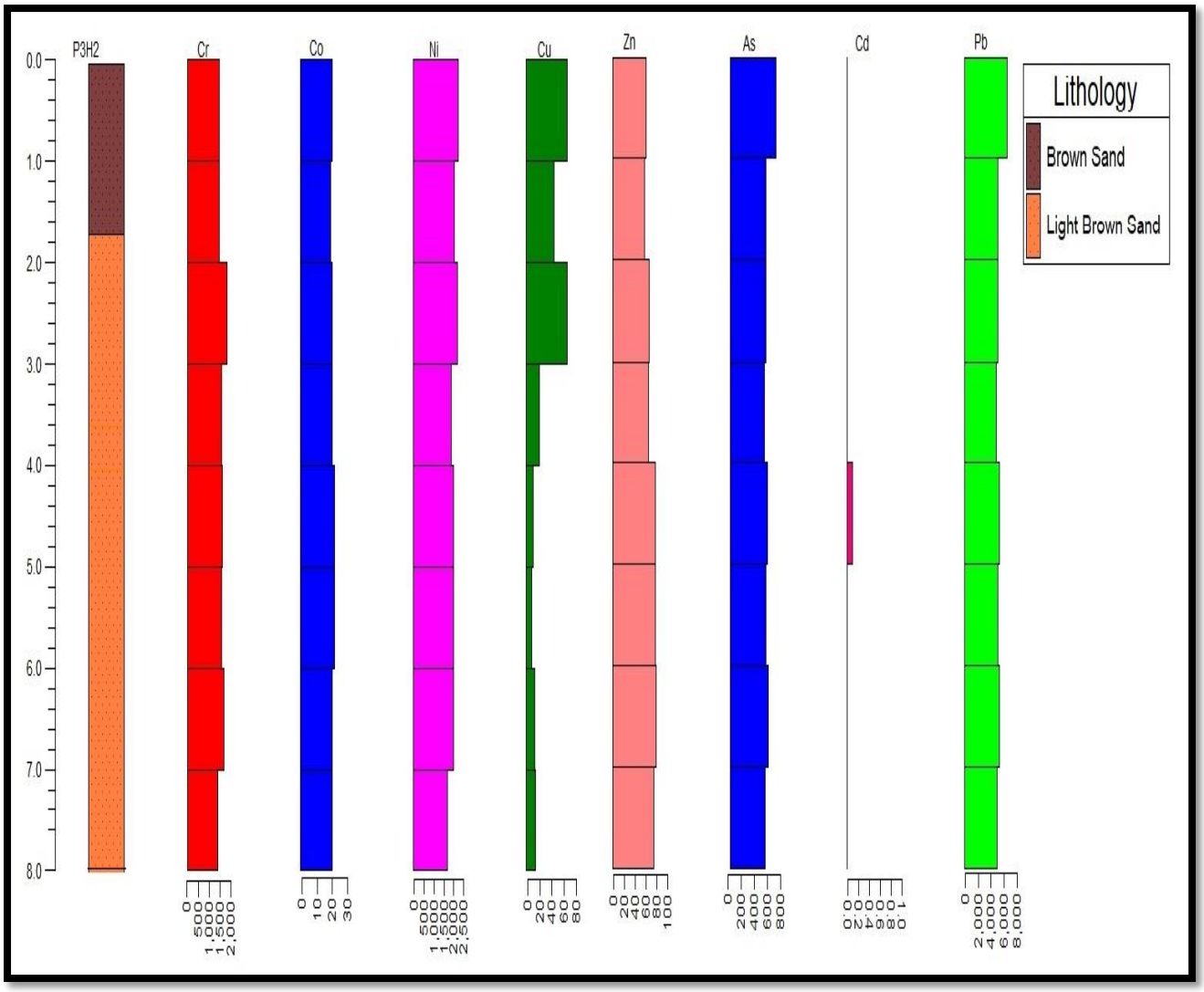

Figure 4.15: Distribution of heavy metals along borehole P3H2.

Distribution of heavy metals along borehole P3H3

In borehole P3H3, most of the metals were erratically distributed with depth. Cr, Ni, Cu, Zn, As and Pb were erratically distributed (Fig. 4.16). A more or less uniform trend was observed in Co, however, Cd was undetected except between 3-4 m and 5-6 m. The highest value of 6505.7 ppm was observed in Pb at a depth of 3-4 m.

© University of Venda

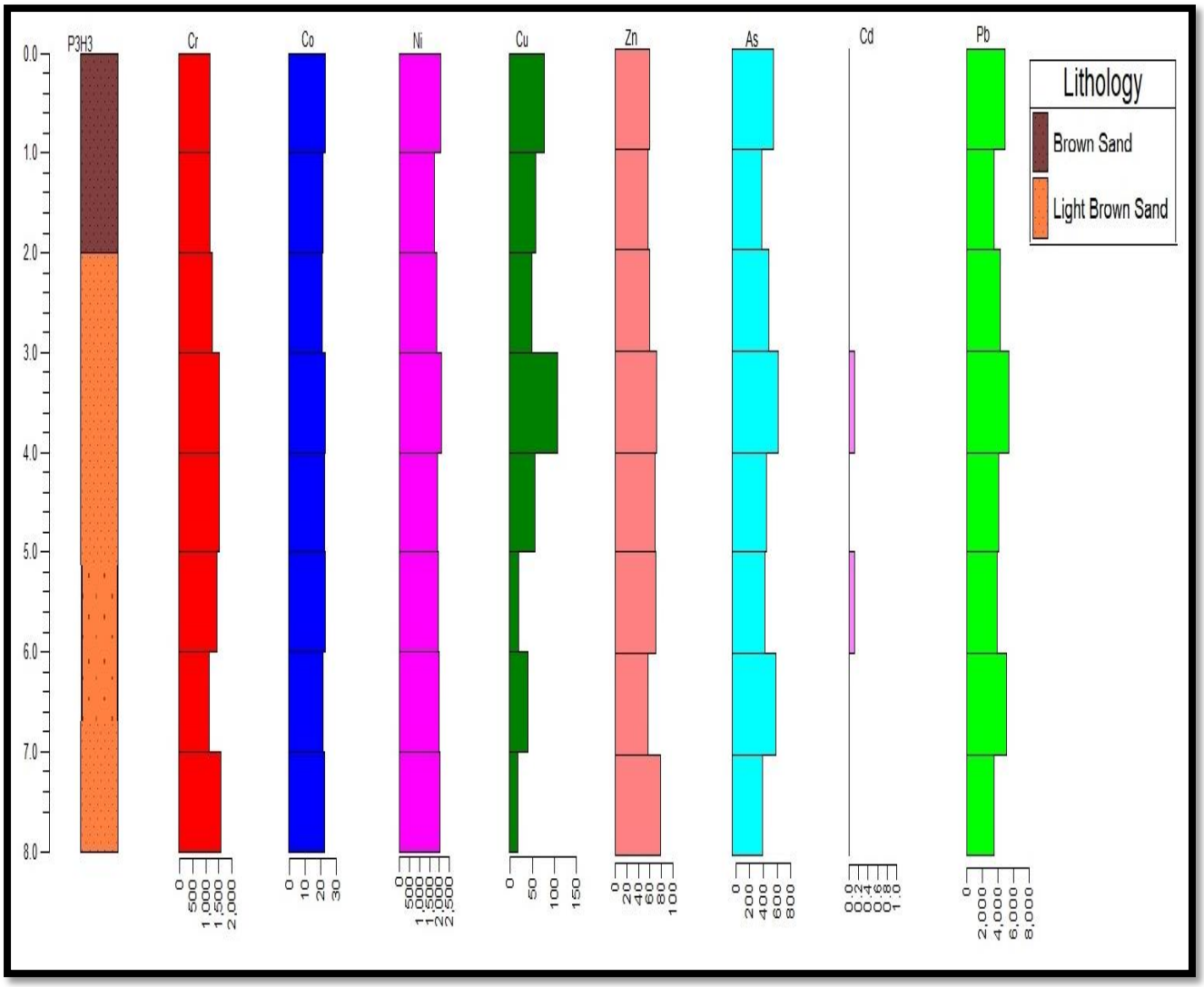

Figure 4.16: Distribution of heavy metals along borehole P3H3.

Distribution of heavy metals along borehole P3H4

Here, most of the metals were more or less uniform with depth. It was observed that Cu was the only metal that was erratically distributed with depth (Fig. 4.17). A more or less uniform distribution trend with depth was observed in Cr, Co, Ni, Zn, As and Pb. Cd, like at most cases was undetected throughout the entire borehole.

© University of Venda

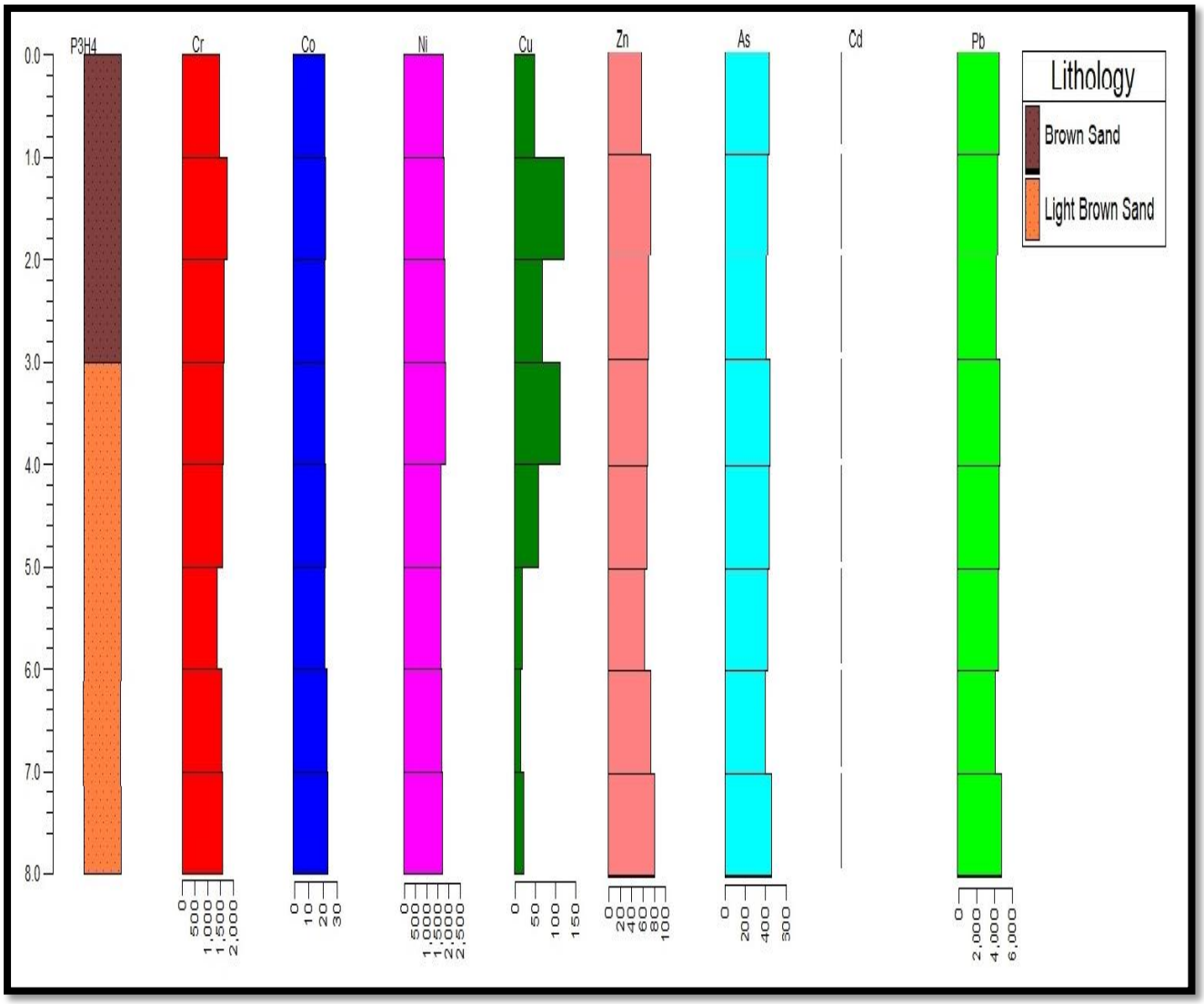

Figure 4.17: Distribution of heavy metals along borehole P3H4.

Distribution of heavy metals along borehole P4H1

In borehole P4H1, a more or less uniform trend with depth were observed in Co and Zn. Cr, Ni and Cu were erratically distributed from the top to the bottom. Similar general decreasing trend in metal values were observed in As and Pb (Fig. 4.18). Cd was undetected throughout the entire borehole except between 0-1 m and 2-3 m where a value of 0.1 ppm was detected.

© University of Venda

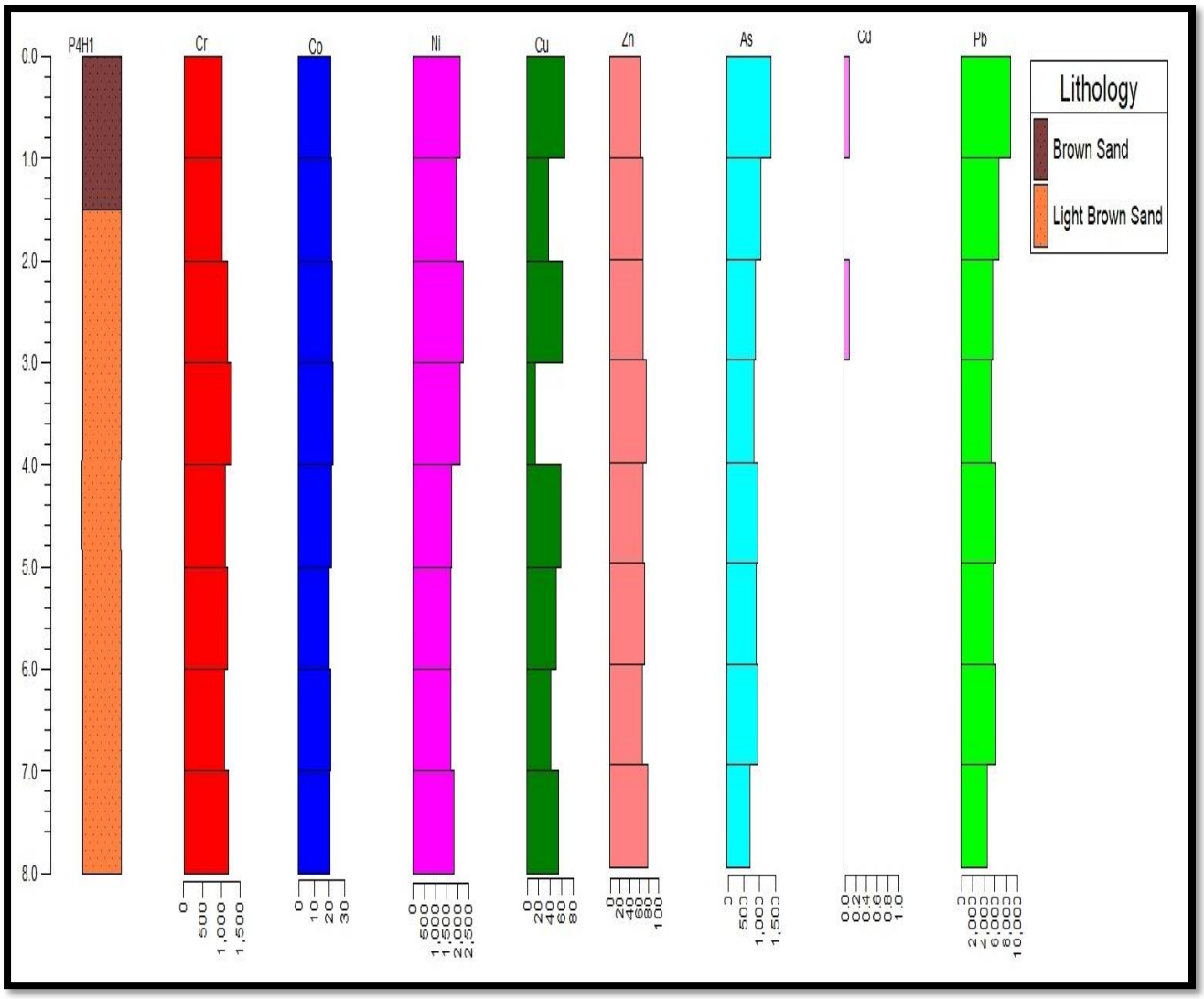

Figure 4.18: Distribution of heavy metals along borehole P4H1.

Distribution of heavy metals along borehole P4H2

In borehole P4H2, Cr, Cu, As and Pb were erratically distributed with depth. More or less uniform distributions with depth were observed in Co and Ni. Like at most cases, Cd was not detected except at 1-2 m where a value of 0.1 ppm was detected (Fig. 4.19). Zn was generally increasing with depth.

© University of Venda

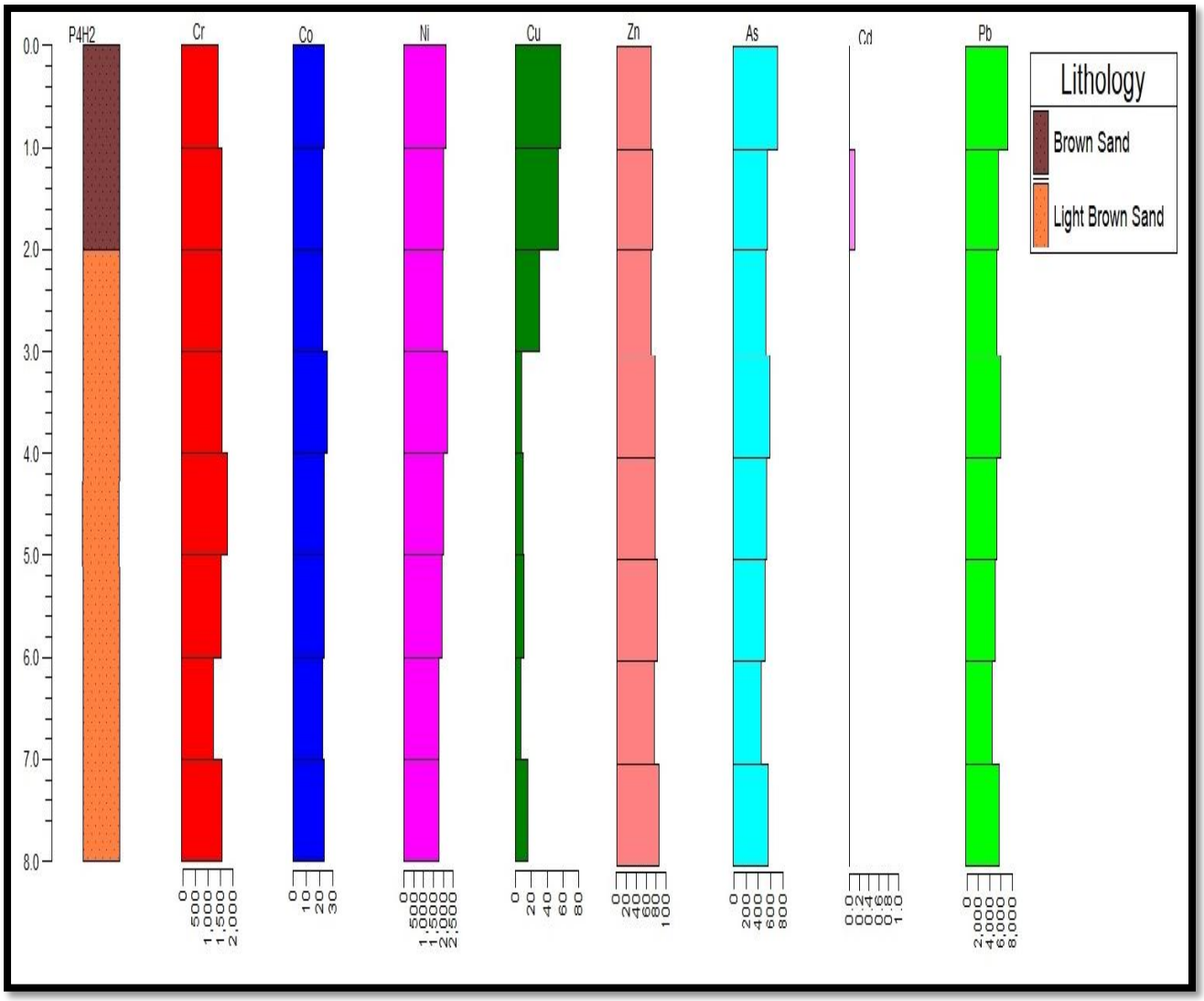

Figure 4.19: Distribution of heavy metals along borehole P4H2.

Distribution of heavy metals along borehole P4H3

Heavy metals within borehole P4H3 were either erratically distributed with depth or more or less uniform with depth. Cr, Ni and Cu were erratically distributed with depth, whereas, Co, Zn, As and Pb were more or less uniform with depth (Fig. 4.20). Though the distribution of Zn was more or less uniform, there was a slight increase with depth. Cd values were only detected between 0-2 m deep and could not be detected from 2-8 m deep.

© University of Venda

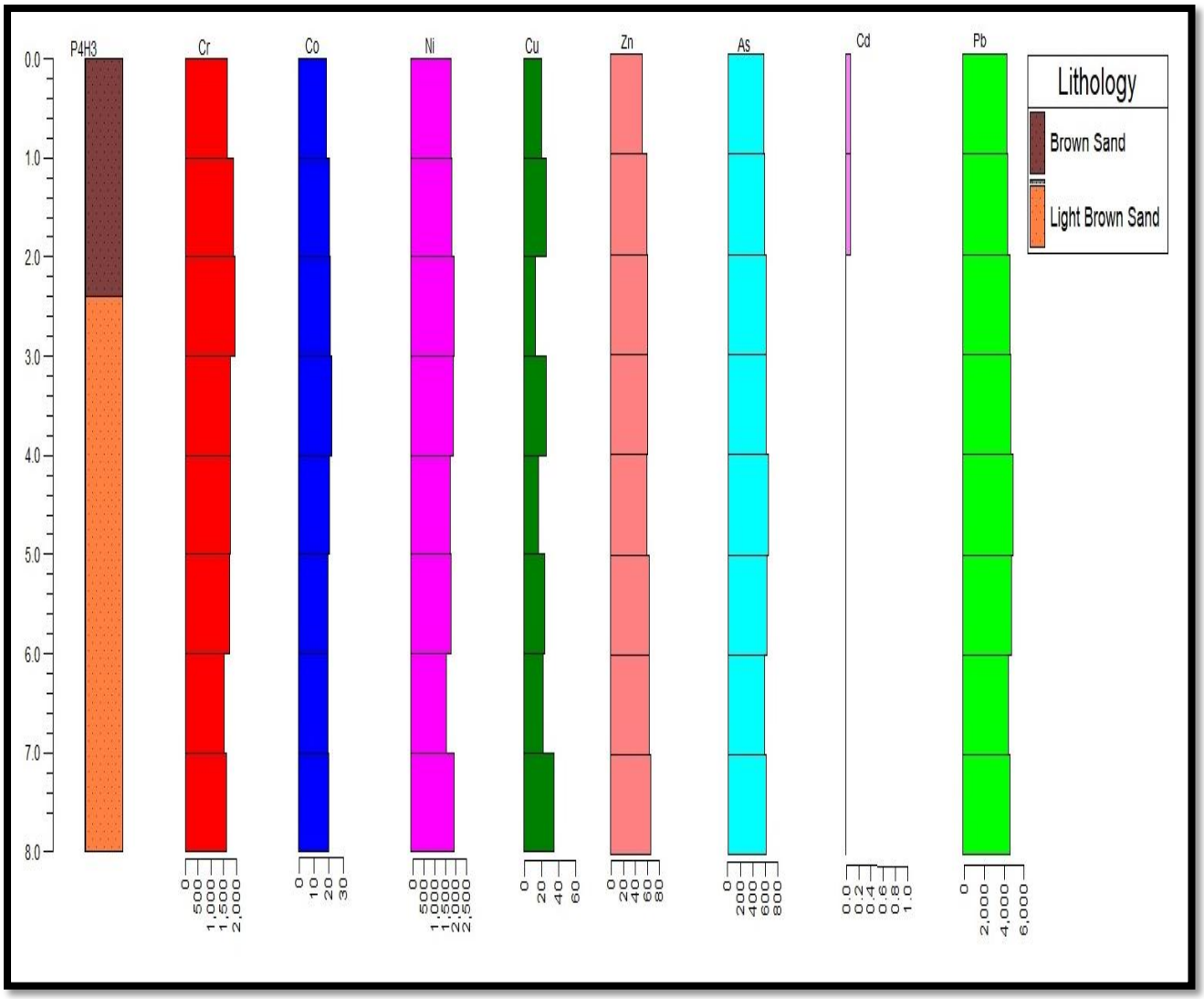

Figure 4.20: Distribution of heavy metals along borehole P4H3.

Distribution of heavy metals along borehole P4H4

Heavy metals within borehole P4H4 were more or less erratic though a uniform behaviour was detected in some metals. Erratic distribution of metals with depth were observed in Cr, Zn, As and Pb. However, a more or less uniform trend was observed in Co and Ni with minor variations. Cu was generally increasing with depth from 0-7 m and suddenly decreased at 7-8 m (Fig. 4.21). Cd was only detected at 0-1 m and 6-8 m

© University of Venda

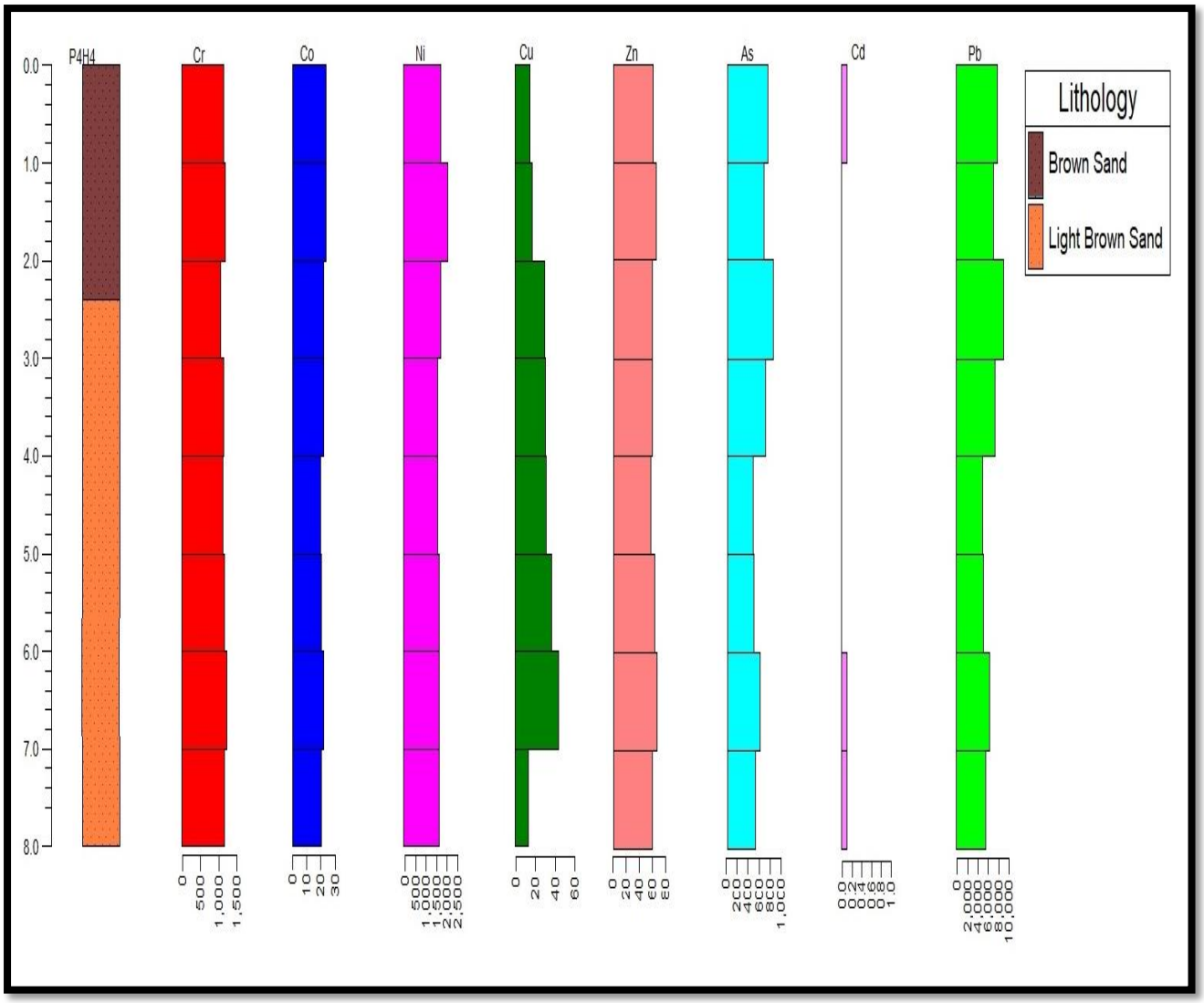

Figure 4.21: Distribution of heavy metals along borehole P4H4.

Distribution of heavy metals along borehole P4H5

Heavy metals within borehole P4H5 were either erratically distributed with depth or more or less uniform. Like other previous cases, Cd was not detected except at 0-2 m and 5-6 m. The distribution of Cr, Cu, As and Pb was rather erratic and the distribution of Co, Ni and Zn were more or less uniform with depth.

© University of Venda

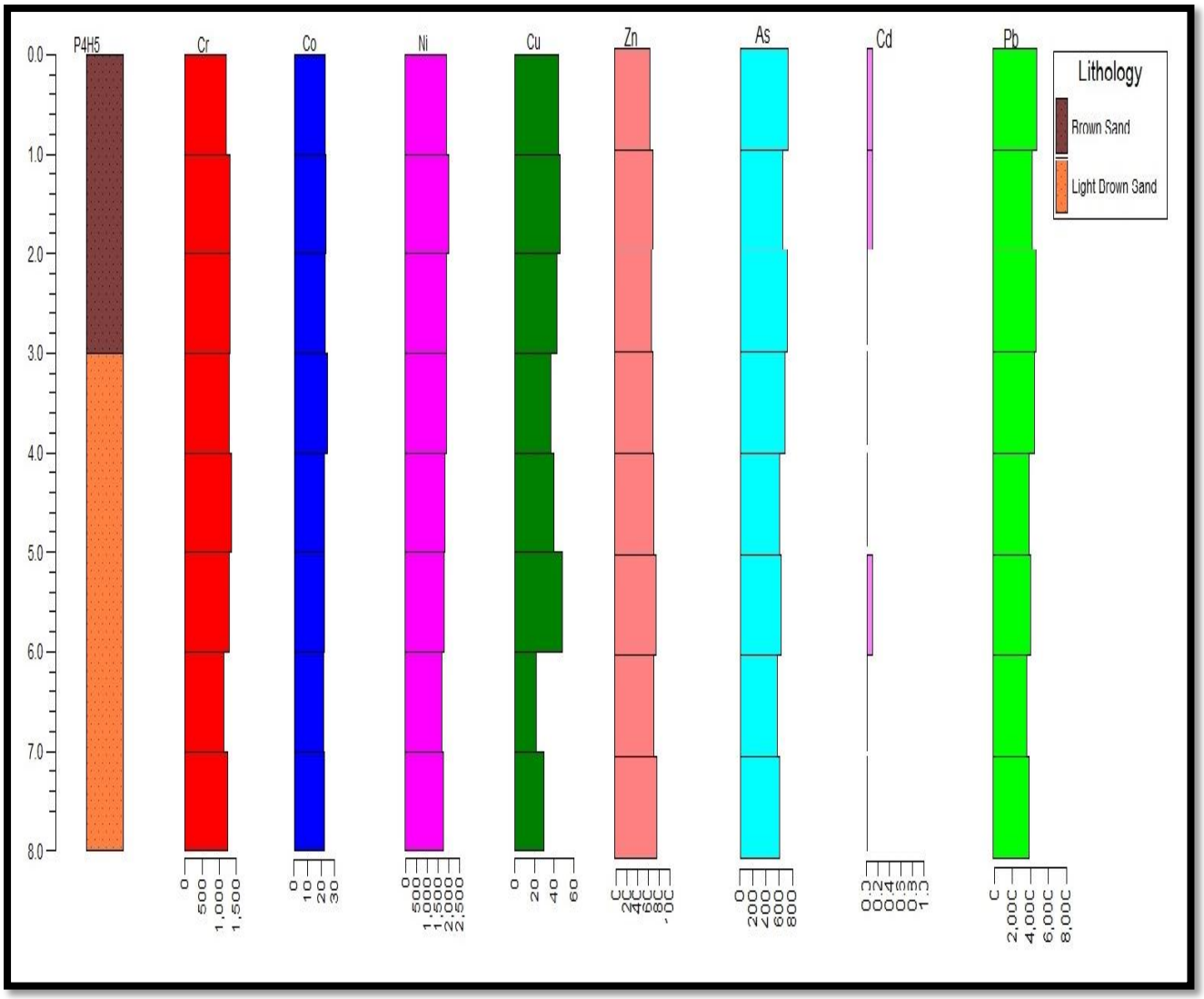

Figure 4.22: Distribution of heavy metals along borehole P4H5.

In general, the overall assessment of the distribution pattern was found to be erratic. Few metals, Cr, Co and Ni, generally revealed a uniform distribution with depth, though there were some instances were such metals were erratically distributed. Cu was the only metal that maintained an erratic distribution with depth with a slight decrease with depth in borehole P2H2. As and Pb had more or less similar distribution trends, mostly erratic, throughout the entire tailings dam. Zn was generally increasing with depth though a few erratic trend and uniform trend were detected in some cases. In general, Cd was below the detection limit in many cases and a value of 0.1 ppm was detected in few cases. This irregular trend of heavy metal distribution was due to the undulating topography of the tailings dam. The erratic distribution was due to the variation in the level of how far the oxidation level

© University of Venda

has extended. Cd was generally the heavy metal with the lowest value and Pb had the highest value in all boreholes.

High values were detected in (ppm); Pb (9471.8), Ni (2817), Cr (1824.2), As (1016.6) and Cu (261). These values in tailings can be attributed to their occurrence within the primary ore that contains sulphide mineralisation, mainly; Galena (PbS), pentlandite[$(Fe, Ni)_9S_8$], Chromatite ($CaCrO_4$) Arsenopyrite (FeAsS) Chalcopyrite ($CuFeS_2$) respectively. High values of Cu are associated with the copper-zinc line where the mineralisation strikes in the general direction of the Murchison range (Consolidated Murchison Limited Information Brochure, 1991). Cd appears in low amount of 0.1 ppm, because it usually occurs in trace quantities in sphalerite. The Cd content in sphalerite varies widely, but is normally in the range 0.02 to 1.5%, with a median value of around 0.3%, although up to 5% have been reported (Fergusson, 1990). Elevated Cr values are indicative of komatite and basalt, even in strongly-weathered environments, such as laterite (Wedepohl, 1978). Co was generally low, averaging 8.4 ppm. Co generally occurs within pyrite in trace quantities.

4.7 Distribution of heavy metals around the tailings dam

Soil data were used to determine the extent of pollution in the surrounding area due to mine tailings. Prediction maps (Figs. 4.23 - 4.30) illustrate the distribution pattern of each metal. The prediction map of each metal is made up of different colours from red through orange and yellow to lime and green that represent the metal values. Red colour indicates high values, orange indicates moderate values, yellow indicates low values, lime indicates very low values and green indicates no values. Prediction maps were drawn taking into account the heavy metal optimum allowable limits on the environment as stipulated by the Government Gazette (2012).

It can be seen from appendix D that there is a wide range of heavy metals: Cr 27-1712.2 ppm, Co 5.7-75.1 ppm, Ni 23.1-3060.6 ppm, Cu 14.6-291.8 ppm, Zn 14.3-167.3 ppm, As 4.2-649 ppm, Cd 0-0.1 ppm and Pb 38.4-6046.5 ppm. Cd (Fig. 4.29) was the only metal in the area which displayed no values at all as it was noted that Cd concentration were ranging from 0-0.1 ppm and illustrated largely by a green colour. Pb was found to be the metal with the highest values with the highest value of 6046.50 ppm (Appendix D) along the eastern part of the tailings dam.

© University of Venda

It can be concluded that the area surrounding the Consolidated Murchison tailings dam is highly enriched with heavy metals from the tailings dam to about 1-3 km away from the tailings dam, but these values decrease with the increase in distance. High concentrations were recorded in Pb, Ni, Cr and As where values of 6046.5 ppm, 3060.6 ppm, 1712.2 ppm and 649 ppm were observed respectively. The order of abundance of heavy metals observed around the tailings dam was Pb>Ni>Cr>As>Cu>Zn>Co>Cd. There was also a shift of metal values for Cr, Co, Ni and As to the east and for Cu to the west and for Zn and Pb on top of the tailings dam. The distribution pattern of Ni and As was found to be similar due to a shaft and waste rock dump appearing on the eastern part of the tailings dam.

The prediction maps summarized the statistics of heavy metals around the tailings dam of Consolidated Murchison Mine. High level of heavy metals were largely dispersed on the eastern part of the tailings dam, due to a shaft and waste rock dump located at about 1.5 km away from the tailings dam.

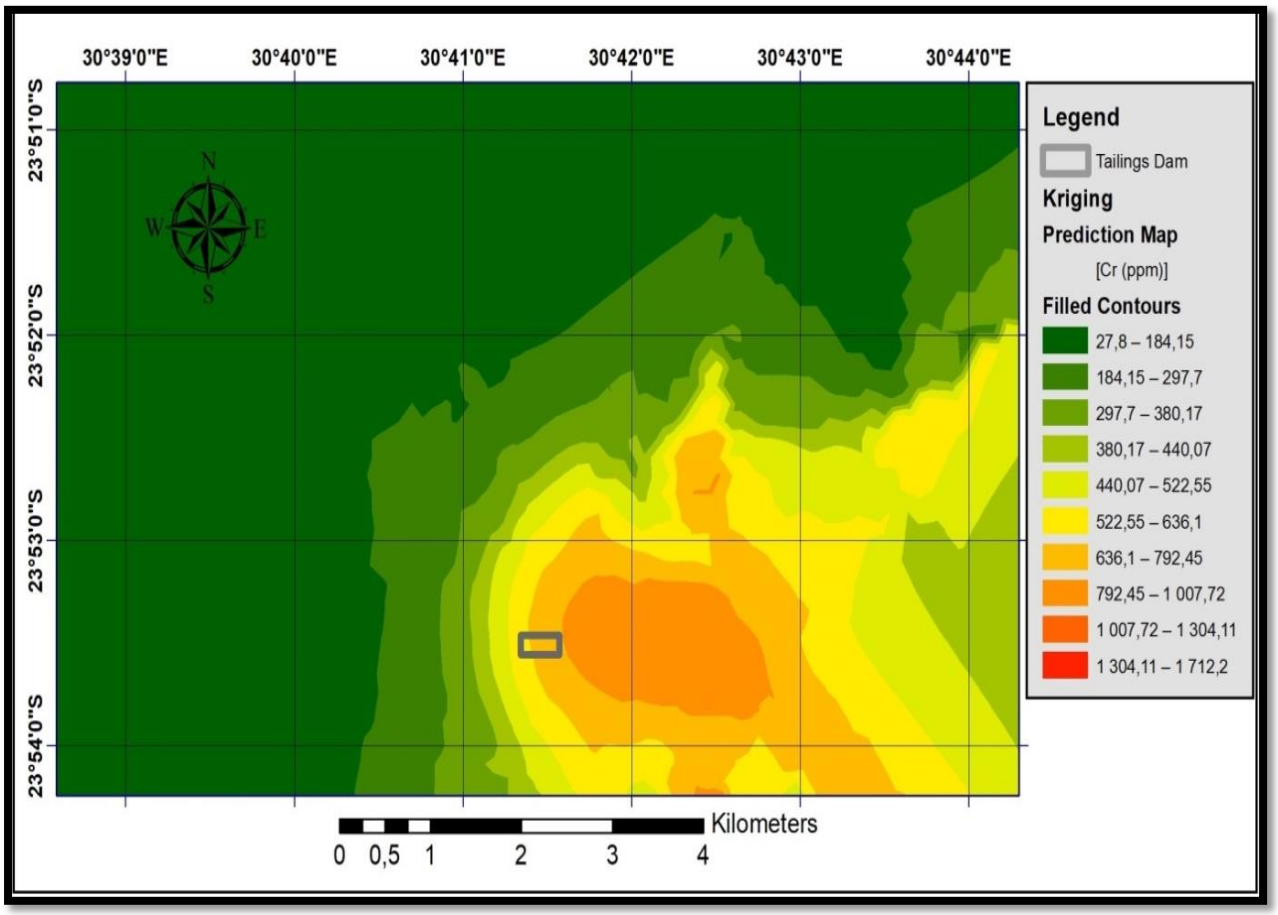

Figure 4.23: Geo-environmental model of Cr around the tailings dam.

© University of Venda

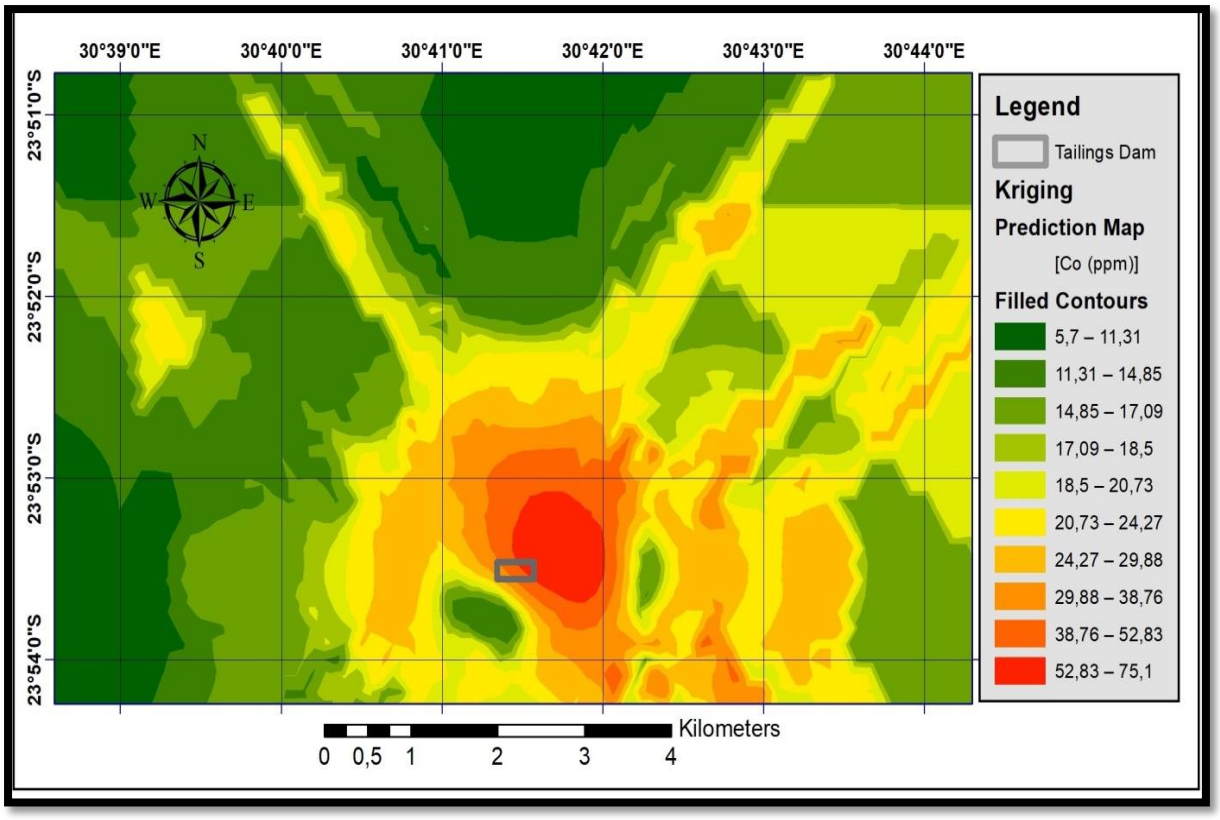

Figure 4.24 Geo-environmental model of Co around the tailings dam.

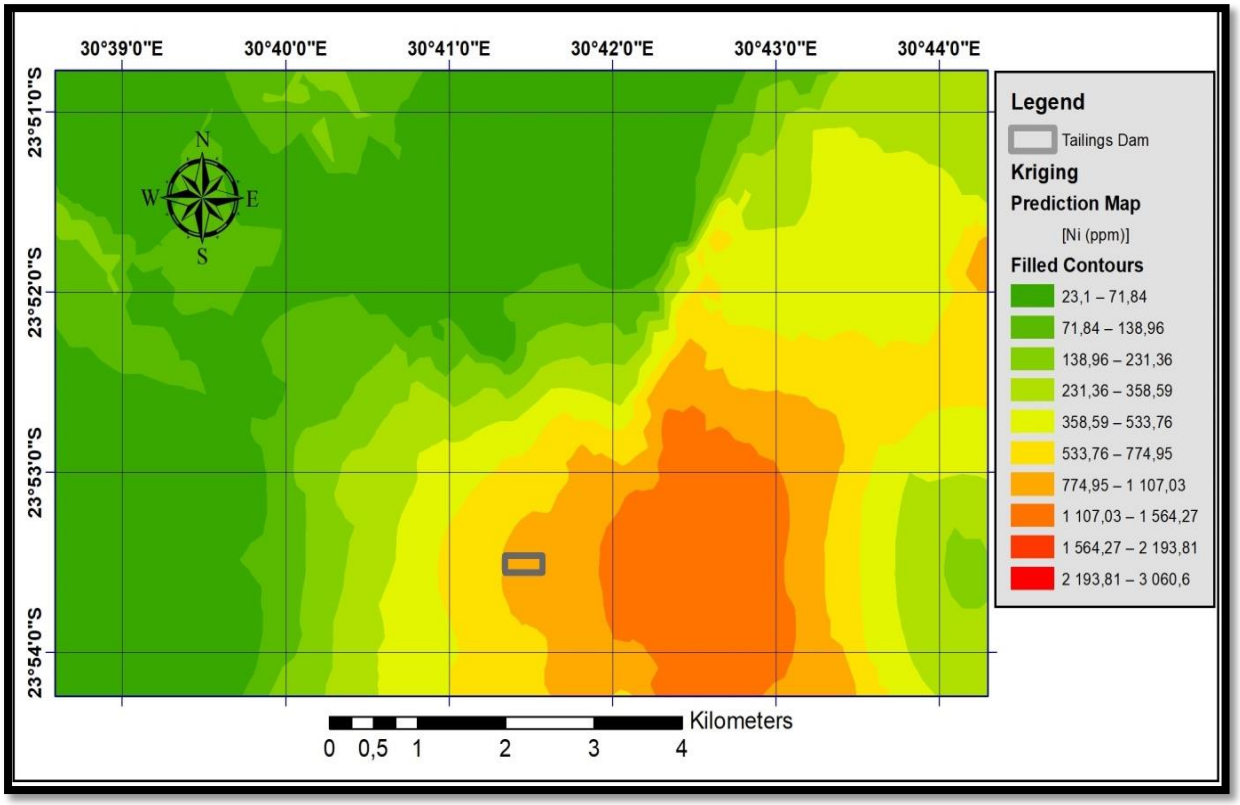

Figure 4.25: Geo-environmental model of Ni around the tailings dam.

© University of Venda

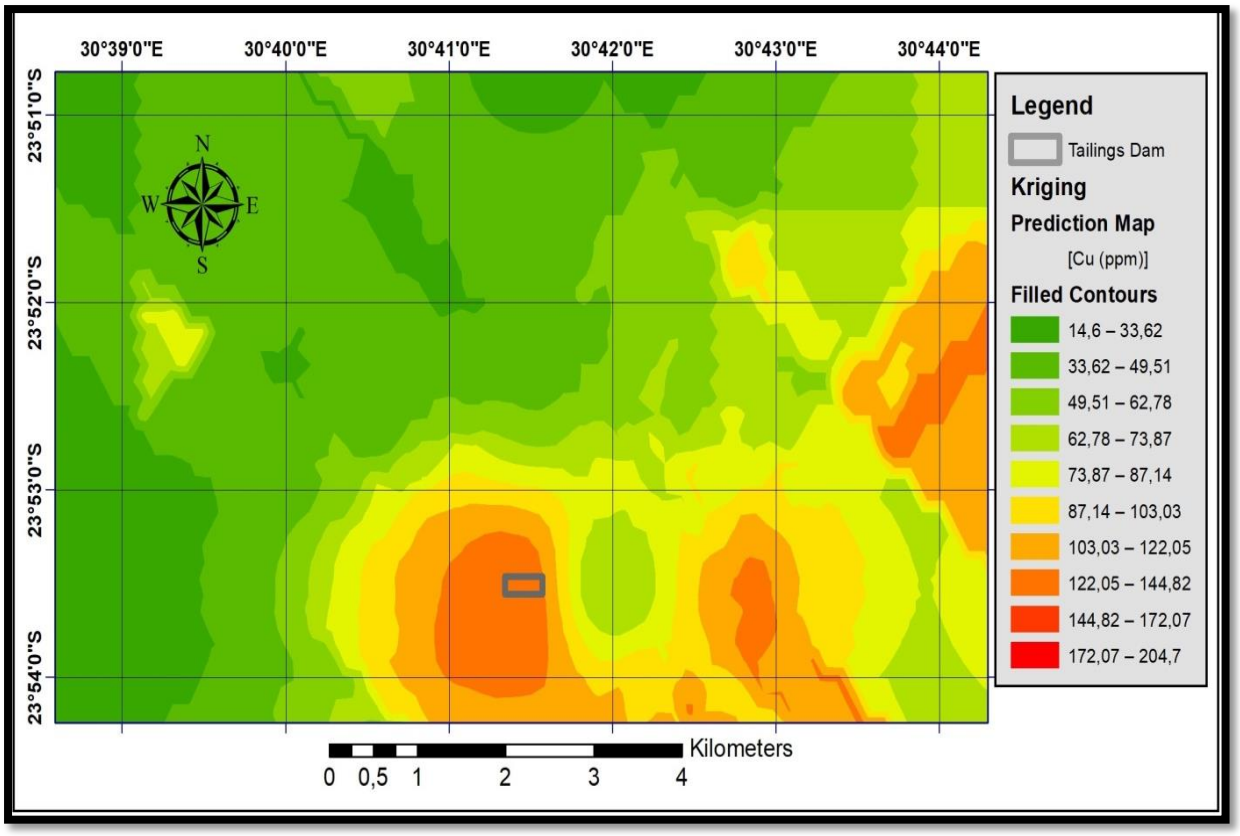

Figure 4.26: Geo-environmental model of Cu around the tailings dam.

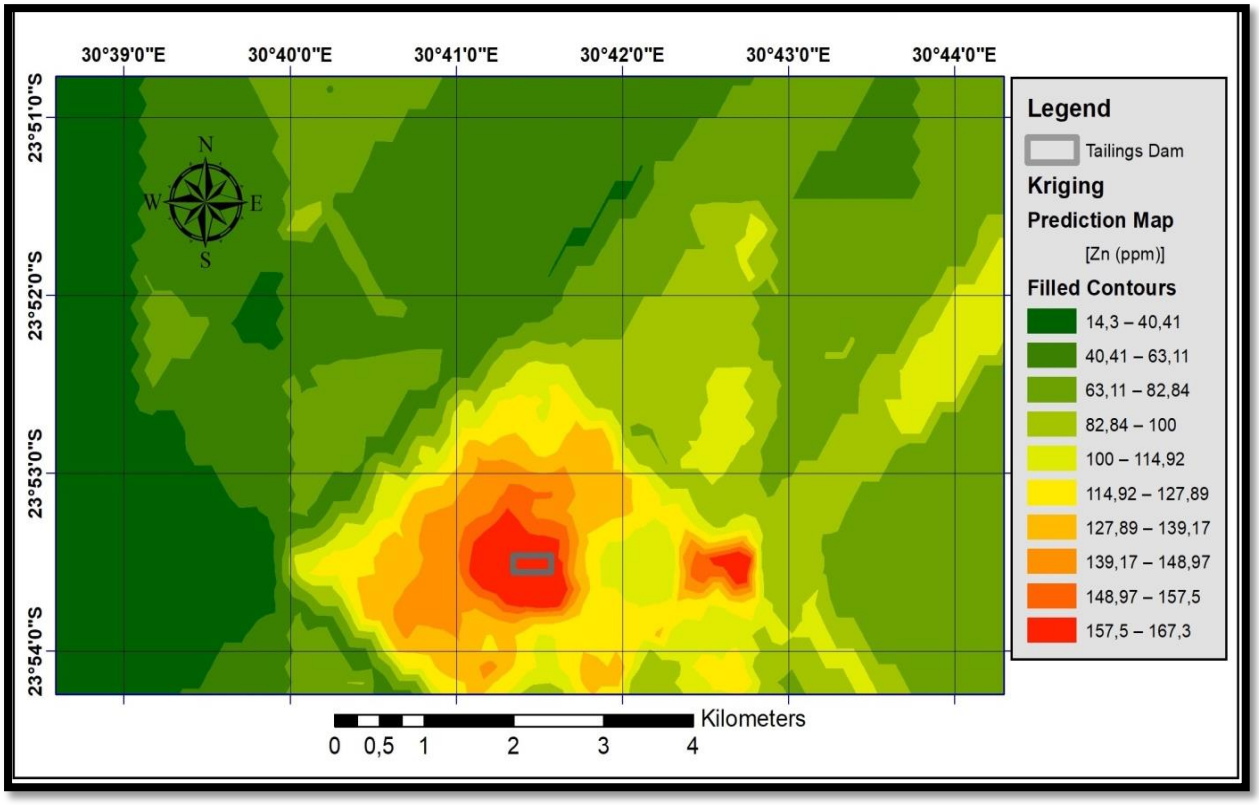

Figure 4.27: Geo-environmental model of Zn around the tailings dam.

© University of Venda

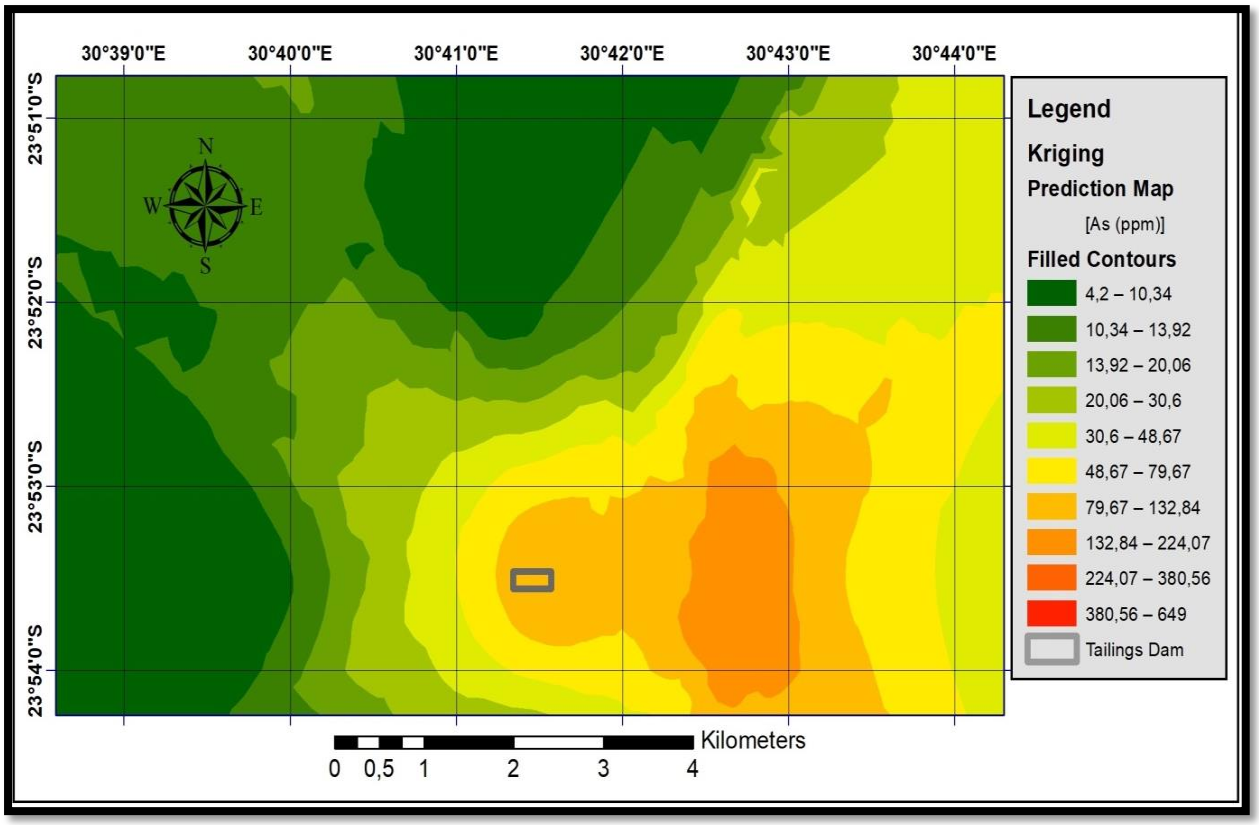

Figure 4.28: Geo-environmental model of As around the tailings dam.

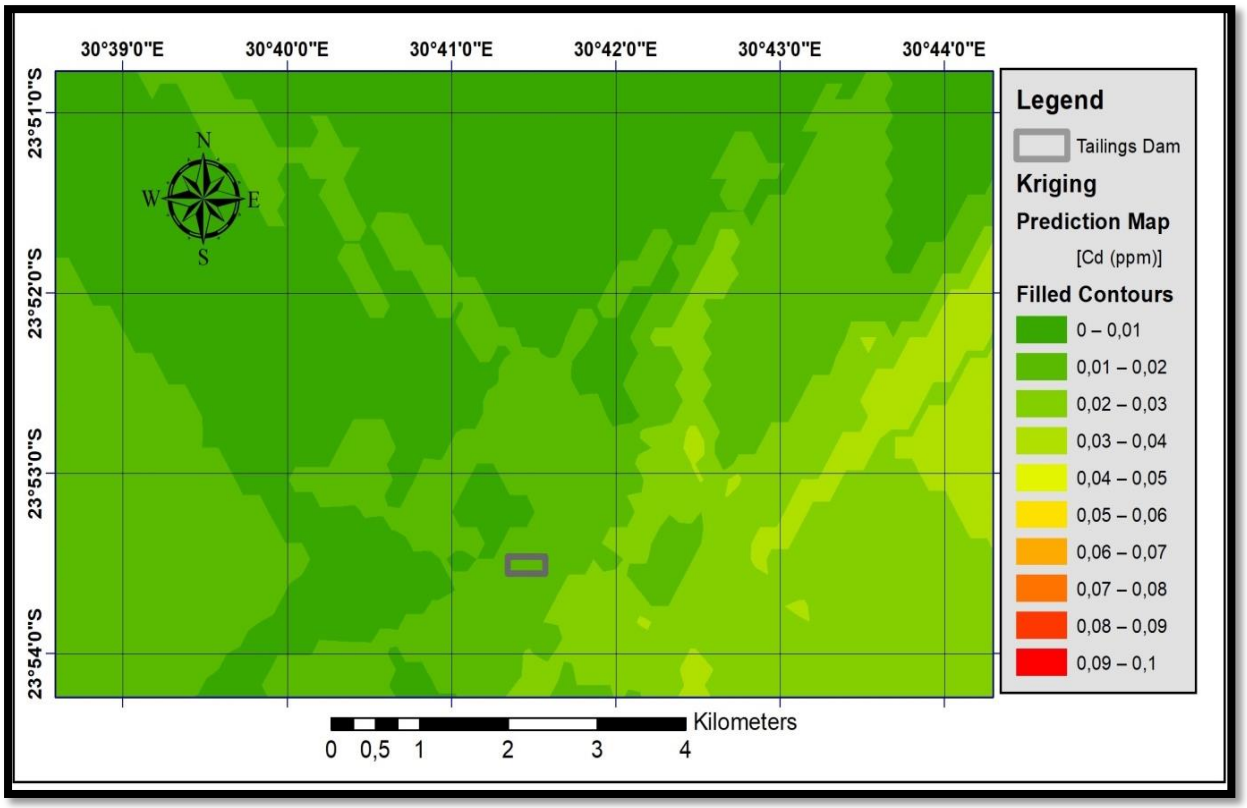

Figure 4.29: Geo-environmental model of Cd around the tailings dam.

© University of Venda

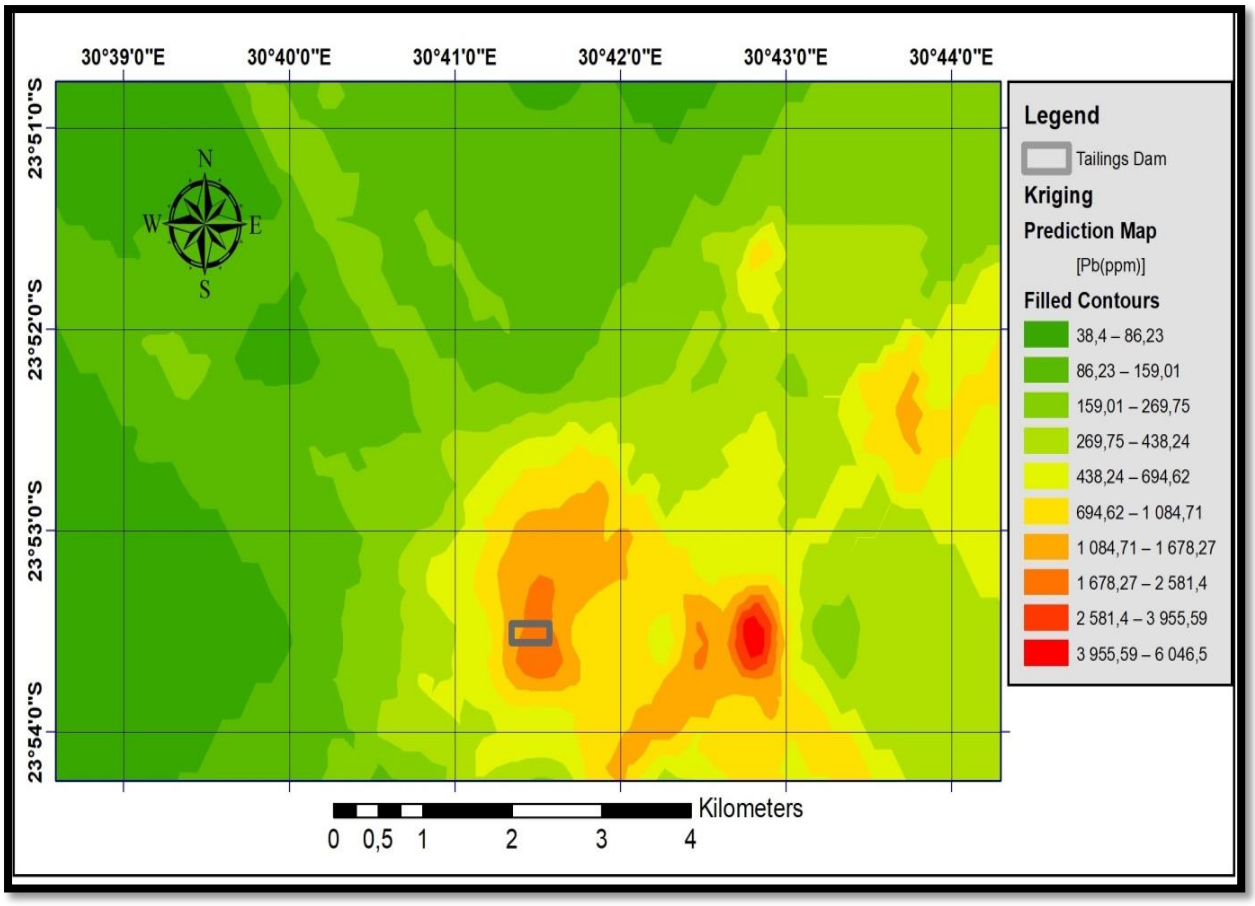

Figure 4.30: Geo-environmental model of Pb around the tailings dam.

4.8 Dispersion of heavy metals around the tailings dam

Figure 4.31 to 4.34 below are graphical presentation of how heavy metals are being dispersed from the tailings dam to the surrounding area. With the exception of the eastern direction, the dispersion of metals was decreasing with distance from the dam.

Dispersion of heavy metals along northern part of the tailings dam

Data from the soil samples that were collected in the northern direction were analysed and presented in graph form for all metals (Fig.4.31). Apart from Pb, Ni and Cr, other metals showed insignificant change with distance from the tailings dam. At the 500 m away from the tailings dam, Pb and Ni registered values that were higher than 1000 ppm with Cr ranging from 754.6 ppm to 985.3 ppm. These values decreased to less than 200 ppm at a distance of 2250 m away from the tailings dam.

© University of Venda

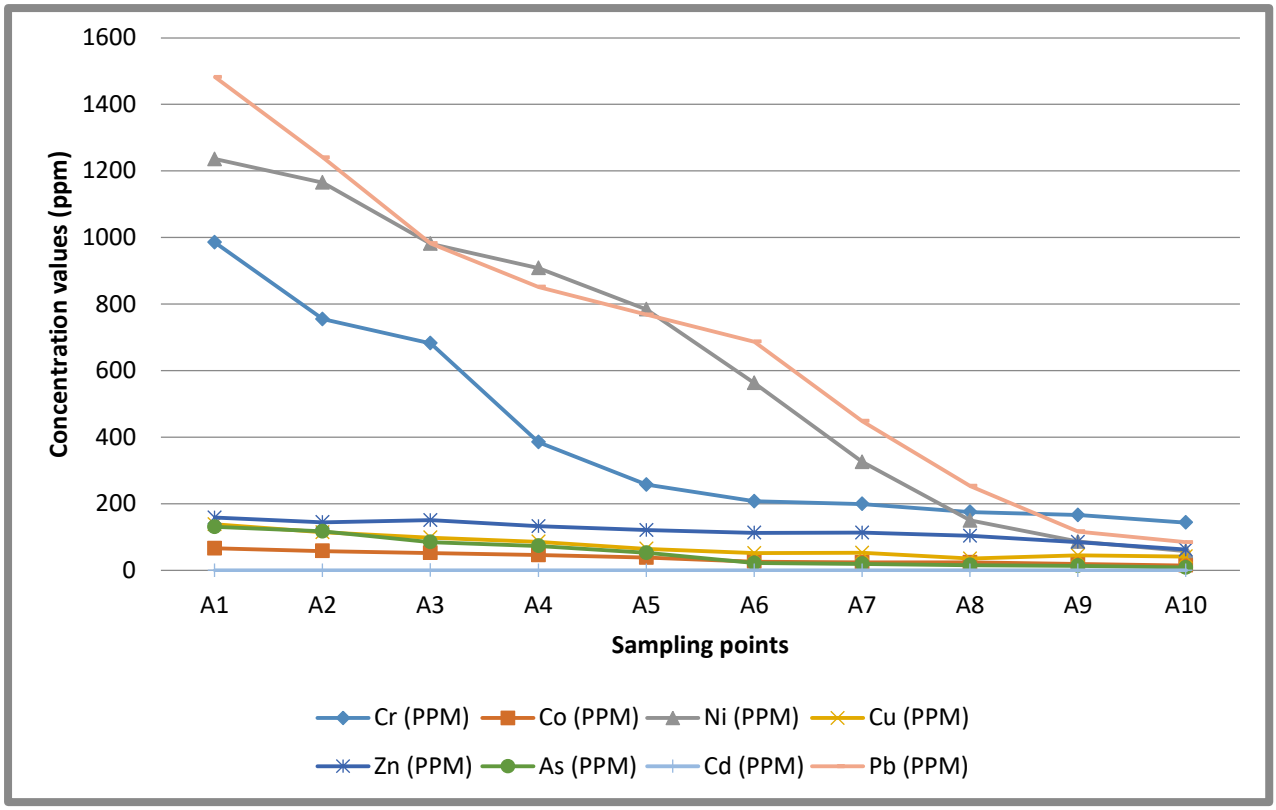

Figure 4.31: Dispersion of heavy metals along the northern part of the tailings dam.

Dispersion of heavy metals along the southern part of the tailings dam

The acquired soil geochemical data in the southern direction were used to determine the distribution pattern. As in the northern direction, the dispersion of metals in the southern direction was similar as Pb, Ni and Cr registered higher values near the tailings dam but dying out with distance from the dam. The only difference here, however, is that Pb decreased faster than Ni and Cr (Fig. 4.32). This was observed at a distance of 250-500 m away from the tailings dam. At a distance of 1500 m from the dam, Ni and Cr had values of 593.5 ppm and 427.5 ppm respectively, whereas Pb registered a value that was below 200 ppm.

Dispersion of heavy metals along the western part of the tailings dam

The acquired soil geochemical data on the western part of the tailings dam were used to determine the distribution pattern. Here, Pb, Ni and Cr were higher than other metals but lower than 1000 ppm. Pb, Ni and Cr registered 520.6 ppm, 452.5 ppm and 386.9 ppm respectively which was more or less half of what was registered at the same point along the north, east south directions. Co, Cu, Zn, As and Cd were all lower than 200 ppm as was always the case with the north and south. Heavy

© University of Venda

metal values deteriorated to less than 100 ppm at 2250 ppm with Pb registering 79 ppm (Fig. 4.33).

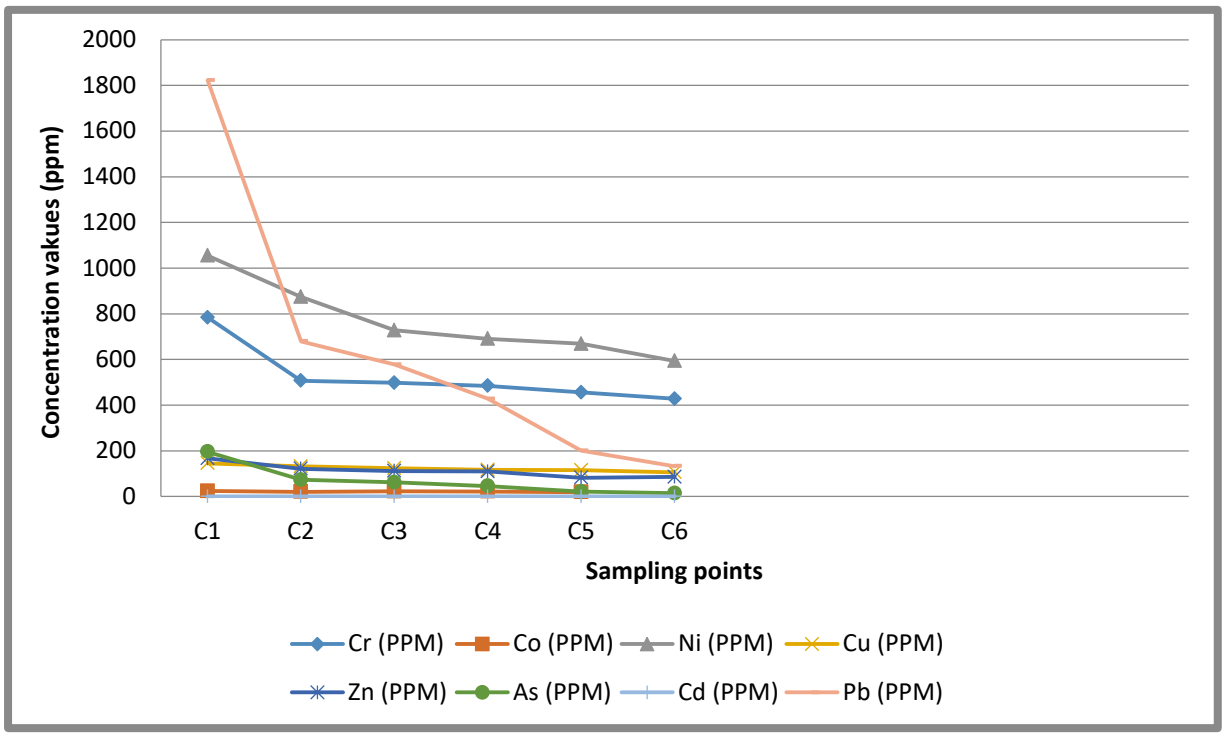

Figure 4.32: Dispersion of heavy metals along the southern part of the tailings dam.

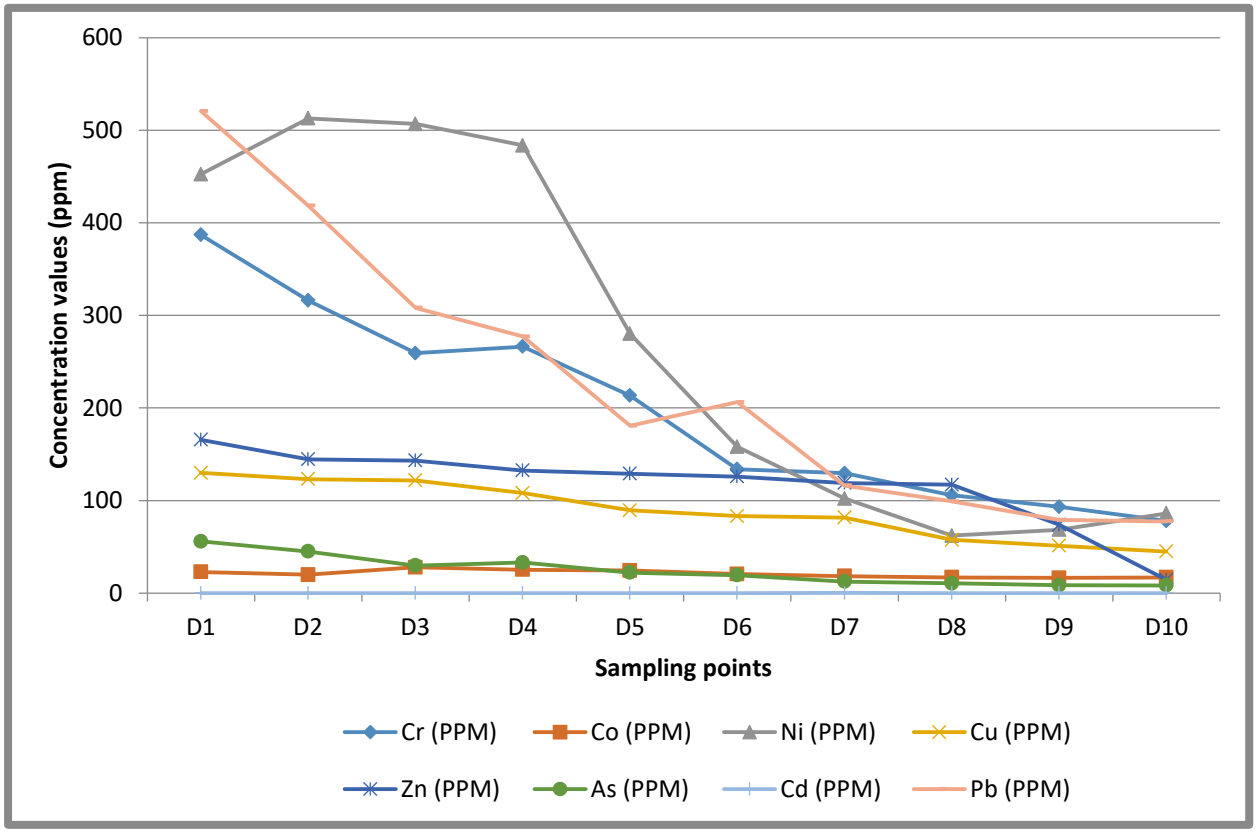

Figure 4.33: Dispersion of heavy metals along the western part of the tailings dam.

© University of Venda

Dispersion of heavy metals along the eastern part of the tailings dam

Data from the soil samples that were collected in the northern direction were analysed and presented in graph form for all metals (Fig.4.34). At this case, Pb was a little below 1000 ppm at 250 m away registering 834.7 ppm. However, Cr and Ni were both higher than 100 ppm. Co, Cu, As and Cd were lower than 100 ppm from 0-1000 m away from the tailings, with Pb, Cr and Ni deteriorating with distance. Sudden peaks of nearly all heavy metal values were observed 1750-2000 m away from the tailings dam (Fig. 4.34). Ni and Cr peaked at 1750 m away from the dam with Pb and As peaking at 2000 m away from the dam. At these points, Pb and Ni peaked to 6046.5 ppm and 3060.6 ppm respectively. The sudden peaks in metal values were due to a mine shaft and waste rock dump which was located at around this area. At 2500 m away from the tailings dam, metal values declined to less than 500 ppm besides Cr and Ni, were values of 553.9 ppm and 710.1 ppm were recorded.

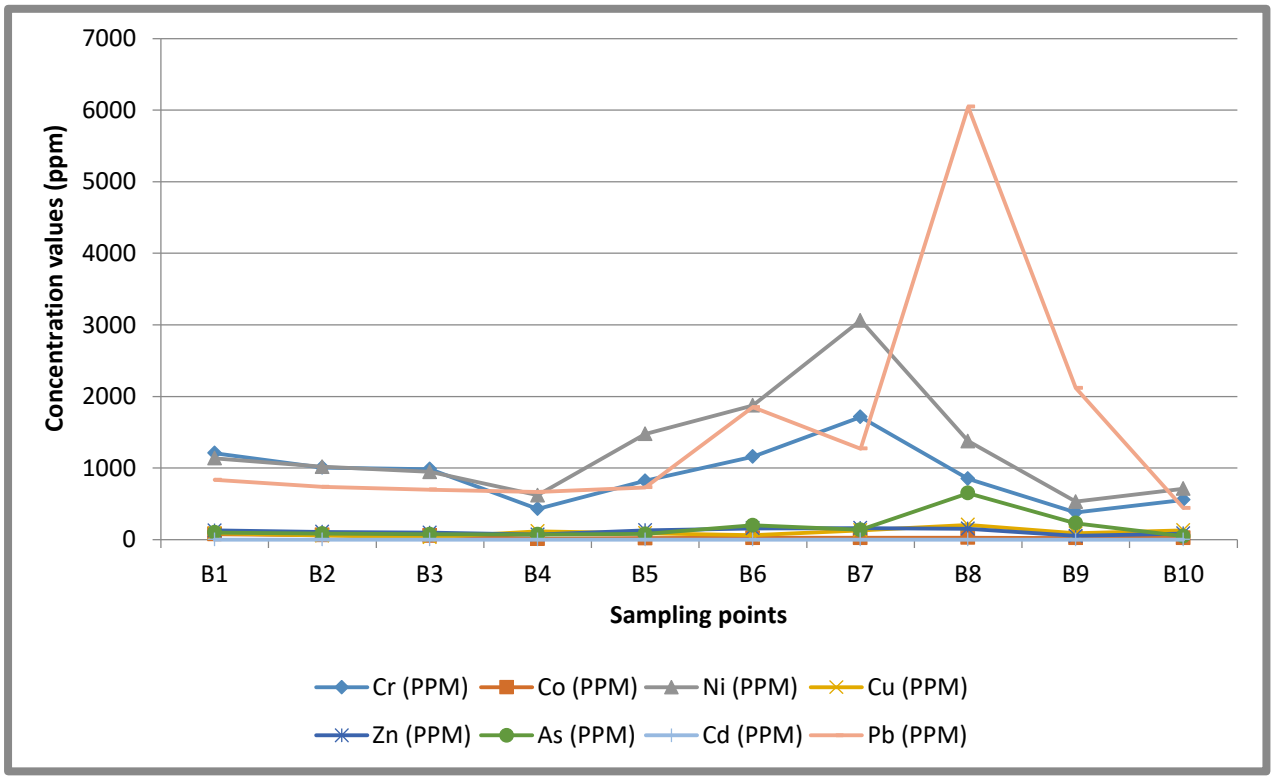

Figure 4.34: Dispersion of heavy metals along the eastern part of the tailings dam.

In general, the values of heavy metals were decreasing with distance as we move further away from the tailings dam. The values of Pb, Ni and Cr were found to be higher than other heavy metal values with Cd having the lowest value.

© University of Venda

The analysed heavy metals within this tailings dam vary in concentration with Pb having the highest value of 9471.8 ppm with an average of 5631.5 ppm. Cd could not be detected because its concentration was below the detection limit. High values were also recorded in Cr, Ni and As where the average values were found to be 1345 ppm, 2062.6 ppm and 604.4 ppm respectively. Data obtained from the analysis of Cd, Co and Zn showed low values within the tailings dam with 42.1 ppm as the highest range recorded in Zn and 0.1 ppm recorded in Cd. The concentrations of these metals were erratically distributed with depth. The main assumption of the study was that the metal values as assessed for the first 8 m from the surface was consistent with depth.

4.9 Pollution status of the soil around tailings dam

The pollution status of soil in this study was analysed and all 39 samples from four different directions (north, west, south and east) around the tailings were analysed. This analysis was done using the Single Factor Index Evaluation Method (SFIM).

Single factor index evaluation method

According to Hong-gui *et al*. (2012), the Single Factor Index Method is one of the most current methods used in evaluation of the degree of heavy metal pollution in soil. Concentrations of Cr, Co, Ni, Cu, Zn, As, Cd and Pb in soil were used to determine the pollution status of the soil. The method is denoted by the formula that was developed by Fang (Zheng *et al*., 2006);

$$Pi = C_i / S_i$$

Whereby;

Pi = Environmental quality index of the pollutant

C_i = Concentration on the spot survey of the pollutant

S_i = Assessment standard of the pollutant

If Pi ≤ 0.7, heavy metals content of samples stands at good condition (uncontaminated). If 0.7 < Pi ≤ 1, heavy metals content of soil does not exceed the environment quality standard, crops develop normally and no harm to human health, but heavy metals pollution of soil is already approaching the warning conditions (Table 4.3). If Pi >1, the heavy metals content of soils is beyond the environment

© University of Venda

quality standard, thus can affect the growth of crops and human health. If 1 < Pi ≤ 2, denotes light pollution and 2 < Pi ≤ 3 denotes moderate pollution and if Pi > 3 denotes heavy pollution (Zheng *et al*., 2006).

Table 4.3: The evaluation grading standards of the single-factor index method

Sub-index	Pi ≤ 0.7	0.7< Pi ≤1	1 <Pi ≤ 2.0	2.0 < Pi ≤ 3.0	Pi > 3.0
Quality grade	Not contaminated	Approaching pollution warning conditions	Low degree of pollution	Moderate degree of pollution	High degree of pollution

The total distance of 2.5 km of each direction (north, west, south and east) was divided into three parts. The first part, which is closest to the tailings dam, was 1 km and the last two parts were 750 m each. The mean value of each heavy metal, from each part, was determined by calculating the average concentration value of all samples that fall within the same part. Determination of the rate of pollution was done using the mean values as C_i as indicated in the above formula and maximum allowable limit of each metal (Table 4.4). The mean concentration values of Cr, Co, Ni, Cu, Zn, As, Cd and Pb in table 4.5, were determined from concentration values of soil around the tailings dam (Appendix D). Determination of the rate of pollution was done using the mean values of C_i as indicated from the formula above and maximum allowable limit of Cr, Co, Ni, Cu, Zn, As, Cd and Pb.

Table 4.4: Table showing optimum allowable limits of heavy metals (Government Gazette, 2012)

Sample	**Cr (ppm)**	**Co (ppm)**	**Ni (ppm)**	**Cu (ppm)**	**Zn (ppm)**	**As (ppm)**	**Cd (ppm)**	**Pb (ppm)**
Optimum Allowable Limits	75	30	40	75	200	15	1	100

© University of Venda

Table 4.5: Mean concentrations of heavy metals in soil around Consolidated Murchison tailings dam

	Sample points	Cr	Co	Ni	Cu	Zn	As	Cd	Pb
Northern direction (A)	**A1-A4**	701.9	55.7	1072.7	109.2	146.6	101.6	0.0	1139.2
	A5-A7	221.6	29.4	557.6	56.5	115.6	31.4	0.0	634.6
	A8-10	161.7	19.3	97.9	40.6	83.6	12.6	0.0	151.4
	A11	53.4	6.4	28.1	23.5	48.6	6.4	0.0	56.3
Eastern direction (B)	**B1-B4**	906.7	50.9	930.0	73.1	100.0	82.0	0.1	732.6
	B5-B7	1229.9	23.6	2136.4	93.8	148.8	137.7	0.0	1282.8
	B8-B10	594.6	24.7	872.2	140.3	96.3	307.8	0.0	2867.8
	B11	359.1	17.2	285.7	115.2	78.5	21.9	0.0	187.6
Southern direction ©	**C1-C3**	596.2	22.4	885.5	133.8	133.5	110.2	0.0	1026.6
	C4-C6	455.9	19.1	651.0	112.6	92.4	27.2	0.0	253.8
Western direction (D)	**D1-D4**	307.1	24.5	488.9	161.2	146.4	40.9	0.0	381.0
	D5-D7	158.9	23.4	180.1	84.7	124.6	18.0	0.0	167.6
	D8-D10	92.2	17.2	72.2	51.3	87.8	9.1	0.0	85.2
	D11	27.0	5.7	23.1	14.6	14.3	4.2	0.0	38.4

Table 4.6 below shows the extent of contamination that heavy metals posed to the environment. It was observed that at 1 km (A1-A4), away from the tailings dam, along the northern part, Cr, Ni, As and Pb contaminated the environment with high degree of pollution (Table 4.6). Cr, Ni, As and Pb had a Pi of 9.4, 26.8, 6.8 and 11.4 respectively, which is greater than 3, the value denoting high degree pollution (Table 4.3). However, Co and Cu had low degree pollution with Pi values 1.9 and 1.5 with Zn and Cd with low Pi values of 0.7 and 0.0 respectively, denoting to no contamination. At A8-A10, which is 1750-2500 m away from the tailings dam, P_i values of Cr and Ni dropped to 2.2 and 2.4 respectively, denoting to moderate

© University of Venda

degree of pollution. However, As dropped to 0.8 meaning approaching pollution warning conditions and Co, Cu, Zn and Cd had P_i values lower than 0.7 denoting no contamination. At A11, a control point which is 5 km away from the tailings dam, no contamination due to heavy metals was recorded since the Pi value of all metals was less than 0.7.

At a distance of 1 km away from the tailings dam B1-B4), along the eastern part, Pi values of Pi > 3.0 were recorded in Cr, Ni, As and Pb, denoting high degree of pollution. However, low degree of pollution was observed in Co, Zn and Cd had no contamination with Pi values of lower than 0.7 (Table 4.6). Pi values of all the heavy metals, with the exception of Co and Cd, increased at B5-B7 (1250-1750 m away from the dam) due to a shaft and a waste rock dump located within that area. At this point, Cr, Ni, As and Pb had high Pi values of 16.4, 53.4, 9.2 and 12.8 respectively. However, these values dropped at B8-B10 with Co, Zn, and Cd having Pi values of less than 1. At B11, a control point along the eastern part of the dam, which is 5 km away from the tailings dam, Cu, As and Pb had Pi values of 1.5, 1.5 and 1.9 respectively, meaning low degree of pollution. However, Cr and Ni had Pi values of greater than 3 denoting to high degree of pollution and Co, Zn and Cd had Pi values of less than one meaning no contamination.

Pi values of metals along the southern part were ranging from 0.0 to 10.3. At 250-750 m (C1-C3) away from the tailings dam, the Pi values of Cr, Ni, As and Pb were higher than 3, meaning high degree of pollution, as was the north and east. Co, Zn and Cd had Pi values of 0.7, 07 and 0.0 respectively, which is Pi ≤ 0.7 denoting no contamination. However, these values dropped at C4-C6 (1000-1500 m) away from the tailings dam. As dropped from 7.3 to 1.8 meaning low degree of contamination, Cr, Co, Ni, Cu, were still within the same range of contamination as they were at C1-C3. There was no control point within the southern part of the tailings dam due to lack of access within this area.

At the western part of the tailings dam, Cr, Ni and Pb had Pi values of greater than 3 at D1-D4 (1 km away from the dam), meaning high degree of pollution. Cu and As had a moderate degree of pollution with Pi values of 2.1 and 2.7 respectively. These values dropped with distance from the tailings dam and it was observed that at D5-D7 (1250-1750 m) away from the tailings dam, Cu, As and Pb had Pi values of 1.1,

© University of Venda

1.2 and 1.7 denoting low degree of pollution. However, Co had a Pi value of 0.8 denoting that it is approaching pollution warning conditions whereas Zn and Cd had Pi values of 0.6 and 0.0 respectively meaning that they have no contamination to the environment. At control point D11, a point which is 5 km away from the tailings dam, no contamination was recorded as the Pi values of all metals had Pi ≤ 0.7 (Table 4.6).

Table 4.6: Pollution index values of soil around Consolidated Murchison tailings dam

	Sample points	Cr	Co	Ni	Cu	Zn	As	Cd	Pb
Northern direction (A)	A1-A4	9.4	1.9	26.8	1.5	0.7	6.8	0	11.4
	A5-A7	3	1	13.9	0.8	0.6	2.1	0	6.3
	A8-10	2.2	0.6	2.4	0.5	0.4	0.8	0	1.5
	A11	0.7	0.2	0.7	0.3	0.2	0.4	0	0.6
Eastern direction (B)	B1-B4	12.1	1.7	23.2	1	0.5	5.5	0.1	7.3
	B5-B7	16.4	0.8	53.4	1.3	0.7	9.2	0	12.8
	B8-B10	7.9	0.8	21.8	1.9	0.5	20.5	0	28.7
	B11	4.8	0.6	7.1	1.5	0.4	1.5	0	1.9
Southern direction (C)	C1-C3	7.9	0.7	22.1	1.8	0.7	7.3	0	10.3
	C4-C6	6.1	0.6	16.3	1.5	0.5	1.8	0	2.5
Western direction (D)	D1-D4	4.1	0.8	12.2	2.1	0.7	2.7	0	3.8
	D5-D7	2.1	0.8	4.5	1.1	0.6	1.2	0	1.7
	D8-D10	1.2	0.6	1.8	0.7	0.4	0.6	0	0.9
	D11	0.4	0.2	0.6	0.2	0.1	0.3	0	0.4

High degree of pollution Moderate degree of pollution Low degree of pollution

© University of Venda

In general, Pi values of greater than 3, denoting high degree of pollution, were detected in Cr, Ni, As and Pb. However, these values were decreasing with distance from the tailings dam. No contamination was observed at 5 km away from the tailings dam along the northern and western part of the dam.

4.10 Grain size distribution

Sieve analysis of waste rock samples was conducted and results are well presented in Appendix E. From the gradation curve in Figure 4.35, the efficient diameter (D10), diameter corresponding to 30% passing (D30) and diameter corresponding to 60% passing (D60) were determined. The coefficient of uniformity (Cu) and co-efficient of curvature (Cc) were calculated using D10, D30 and D60. The following equations were used to calculate Cu and Cc respectively, (Das, 2010):

$$Cu = \frac{D_{60}}{D_{10}} \qquad and \qquad Cc = \frac{D_{30}^2}{D_{60} \times D_{10}}$$

Table 4.7 shows that the effect diameter ranged from 0.09 to 0.17 with an average of 0.13 mm. Sample 1 had the lowest effective diameter and Sample 3 had the highest effective diameter (0.09 mm and 0.17 mm respectively). D30 and D60 had an average of 2.39 mm and 20.55 mm respectively. The co-efficient of uniformity ranged from 80.77 to 350 with an average of 165.97. The co-efficient of curvature had an average 2.46 with the lowest and highest values of 0.99 and 5.93 respectively.

Table 4.7: Summary of key particle sizes and co-efficient

Sample ID	D10(mm)	D30(mm)	D60(mm)	Cu	Cc
Sample 1	0.09	0.90	9.10	101.11	0.99
Sample 2	0.12	2.4	42	350	1.14
Sample 3	0.17	3.15	20.53	120.76	2.84
Sample 4	0.15	4.28	20.58	137.20	5.93
Sample 5	0.13	1.7	10.5	80.77	2.12
Sample 6	0.1	1.9	20.6	206.00	1.75
Average	0.13	2.39	20.55	165.97	2.46

© University of Venda

From the gradation curve in Figure 4.35, it can be deduced that the Consolidated Murchison waste rock dump was well graded based on the general shapes of the curves and key co-efficient in Table 4.7. This is so because the material shows almost a uniform distribution of almost every particle size.

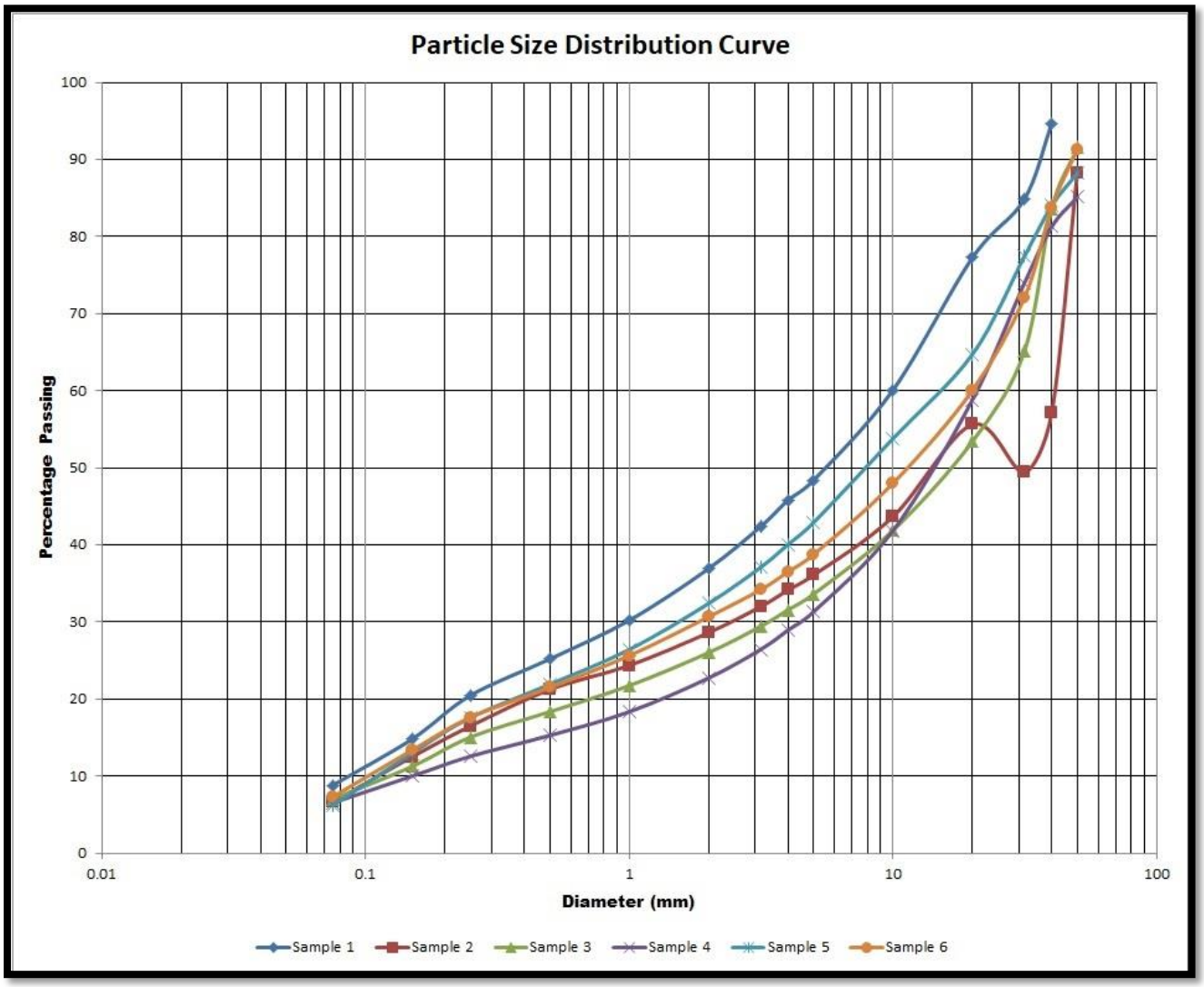

Figure 4.35: Gradation curves representing sieve analysis results.

It was observed from Figure 4.35 that, the highest (77.27%) and lowest (53.43%) amount of material passing through the 20 mm sieve was from sample 1 and 3 respectively. The highest amount (36.94%) of material passing through the 2 mm sieve was recorded in sample 1 and the lowest amount (22.75%) of material passing through the 2 mm sieve was recorded in sample 4 (Fig. 4.35). However, sample 1 had the highest percentage of material passing through the 0.075 mm sieve, where 8.81% of the material passed through this sieve and the lowest was recorded in

© University of Venda

sample 5, whereas 6.16% of the material passed through the 0.075 mm sieve. From these observations, it was concluded that the waste rock material is mainly composed of rock fragments with gravel and sand material and minor clay/silt.

Table 4.8 depicts the amount and volume of rock fragments, gravel material, sand and clay/silt material within the waste rock. The material from the waste rock dump consisted of 42.7% of rock fragments, 29.6% of gravel material, 20.6% of sand material and 7.1% clay/silt material. From these figures, the total volume of rock fragments, gravel, sand and clay/silt material within the waste rock dump were calculated. The total area of the waste rock dump was calculated to be 48 030 m^2 and the height was estimated to be 28 m.

The volume was then calculated using the following formula:

$$V = A \times h$$

Where: V = Volume

A = Area of the waste rock dump

H = Height of the waste rock dump

The total volume was then calculated to be 1 344 840 m^3. From this volume, the total volume of each particle size was calculated and presented on table 4.8 below. It was concluded that clay material had the lowest volume and percentage of 95 483.64 m^3 and 7.1% respectively with rock fragments having the highest percentage of 42.7% and volume of 574 246.68 m^3. From these observations, it can be concluded that the waste rock dump is suitable for sub-base layer for road construction since it contains all particle size in required percentages, as stipulated from Table 4.14 (revised AASHTO system of soil classification) (Amadi *et al.,* 2015).

Table 4.8: Total percentage and volume contained in each particle size

	Rock fragments	**Gravel material**	**Sand Material**	**Clay/silt material**
Percentage contained (%)	42.7	29.6	20.6	7.1
Total volume contained (M^3)	574 246.68	398 072.64	277 037.04	95 483.64

© University of Venda

4.11 Atterberg limits

Data attained from Atterberg limit tests (liquid limit, plastic limit and plasticity index) of the waste rock dump were used to analyse the suitability of waste rock material for construction purpose. The outcomes of the analysis are projected in Appendix F, Tables 4.9-4.11 and Figures 4.37 and Figure 4.38.

4.11.1 Liquid limit

The liquid limit of the waste rock material ranged from 27.49% to 34.96%. The average liquid limit was calculated to be 30.25% (Table 4.9). The lowest and highest liquid limit was found in sample 1 and sample 5 respectively and this was found to be 27.49% and 34.96% (Fig. 4.36). The waste rock material has a relatively low liquid limit which corresponds with the relatively low moisture content in the material. This is indicative of low water holding capacity.

Table 4.9: Average values of liquid limit test results

	Sample 1	**Sample 2**	**Sample 3**	**Sample 4**	**Sample 5**	**Sample 6**	**Average**
Mass of can (g)	18.04	18.01	18.02	18.33	18.03	18.35	18.13
Mass of can + moist soil (g)	28.65	32.41	32.86	30.59	29.71	31.96	31.03
Mass of can + dry soil (g)	26.34	29.53	29.43	27.91	26.83	28.88	28.15
Mass of soil solids (g)	8.3	11.51	11.40	9.58	8.77	10.50	10.01
Mass of pore water (g)	2.27	2.93	3.44	2.69	2.89	3.08	2.88
Moisture content (%)	27.49	31.51	29.79	28.29	34.96	29.45	30.25
Number of blows	38	30	34	30	37	27	33

© University of Venda

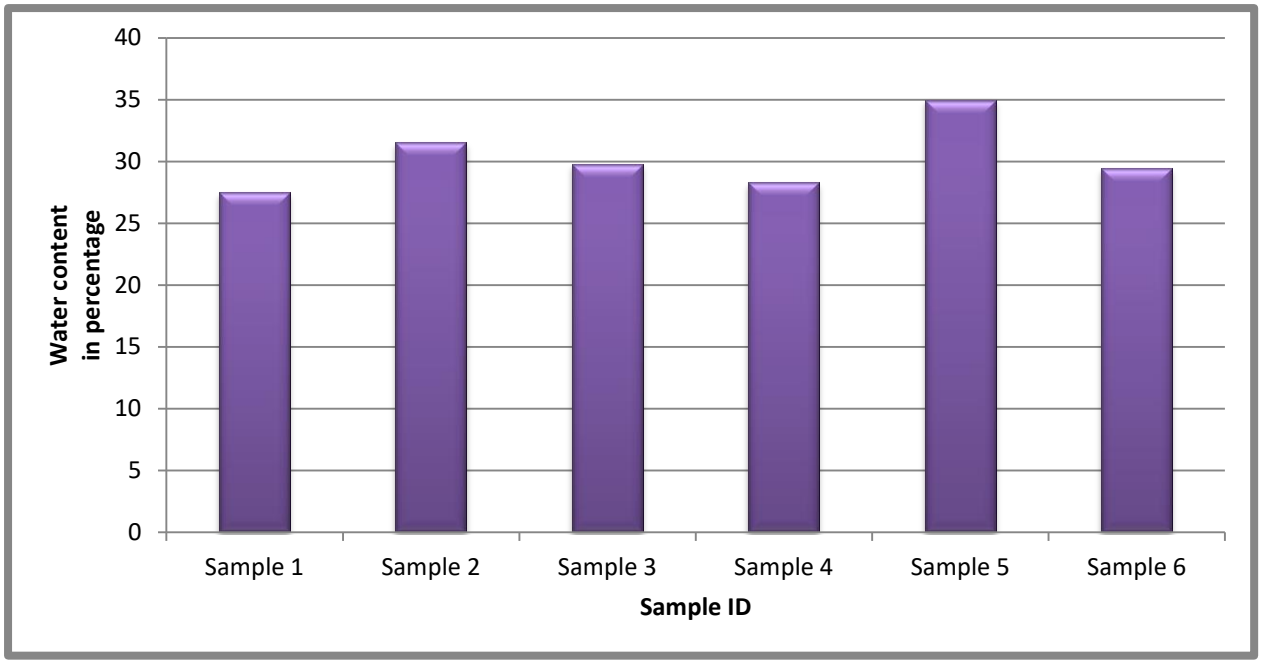

Figure 4.36: Liquid limit values within the waste rock.

4.11.2 Plastic limit

The plastic limit ranges from 22.84% to 27.71% and the average plastic limit was calculated to be 26.08% (Table 4.10). The lowest and highest plastic limit was found on sample 3 and sample 4 respectively and this was found to be 22.84% and 27.31% (Fig. 4.37).

Table 4.10: Average values of plastic limit test results

	Sample 1	Sample 2	Sample 3	Sample 4	Sample 5	Sample 6	Average
Mass of can (g)	18.06	19.03	17.96	18.07	17.99	18.01	18.19
Mass of can + moist soil (g)	25.09	22.42	24.58	24.75	25.52	24.21	24.43
Mass of can + dry soil (g)	23.67	21.58	23.36	23.32	23.95	22.95	23.14
Mass of soil solids (g)	5.62	2.89	5.40	5.31	5.96	4.94	5.02
Mass of pore water (g)	1.42	0.84	1.22	1.43	1.57	1.26	1.29
Moisture content (%)	25.21	26.91	22.84	27.71	26.47	27.31	26.08

© University of Venda

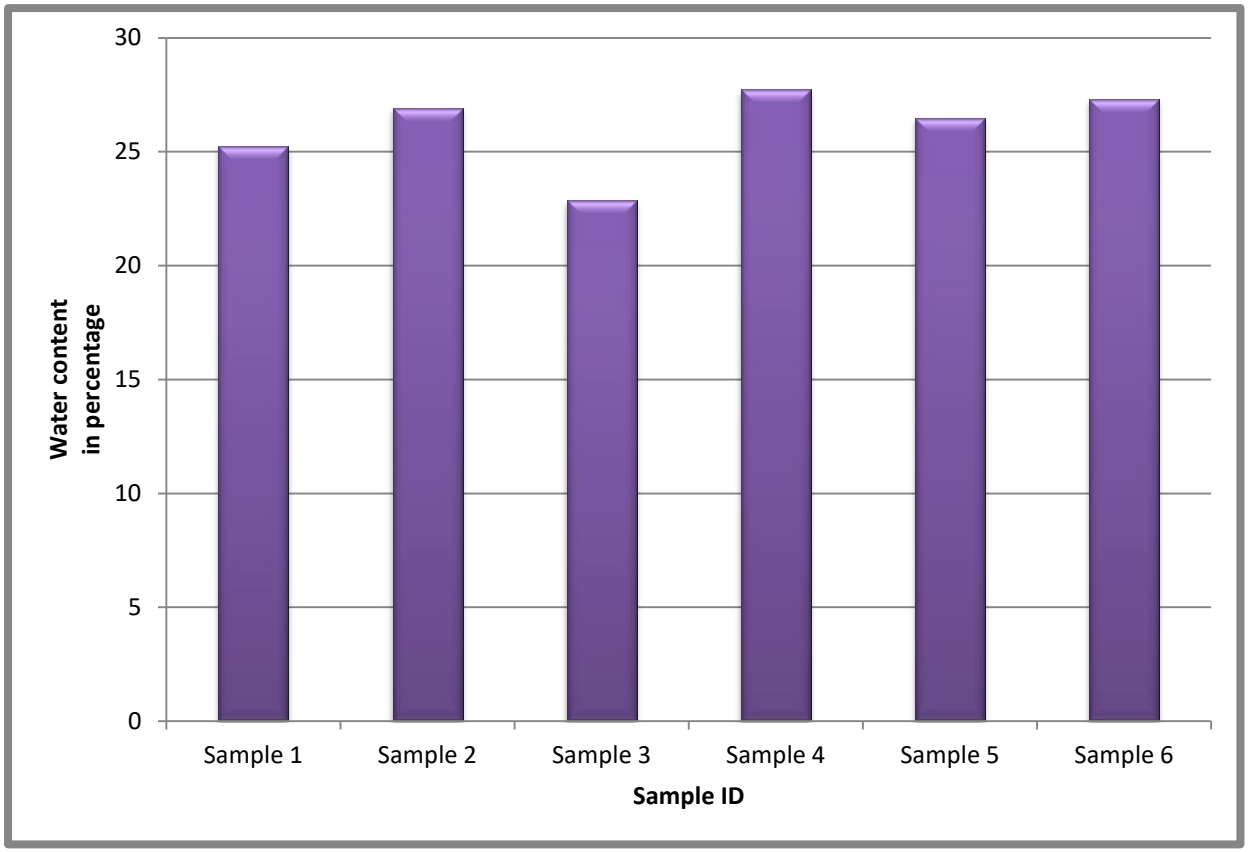

Figure 4.37: Plastic limit values within the waste rock.

4.11.3 Plasticity index

Plasticity index is the difference between the liquid limit and the plastic limit and it was calculated using the following equation (Das, 2010);

$$PI = LL - PL$$

Where: PI = Plasticity index

LL = Liquid limit

PL = Plastic limit

Das (2010) classified plasticity index into six classes; non-plastic, slightly plastic, low plasticity, medium plasticity, high plasticity and very high plasticity (Table 4.12). According to Table 4.11, plasticity index of the waste material within the waste rock dump of Consolidated Murchison mine ranged from 0.58% to 8.49% with the lowest and highest values found on sample 4 and sample 5 respectively. These values are well presented in Figure 4.38. The average plasticity index of the waste material was calculated to be 4.17%. From Table 4.11, it can be deduced that the waste

© University of Venda

material from this waste rock dump is classified as slightly plastic. The relatively low plasticity index shows that the waste rock material at Consolidated Murchison can change from solid to a liquid with little change in moisture.

Table 4.11: Values of plasticity index

	Sample 1	Sample 2	Sample 3	Sample 4	Sample 5	Sample 6	Average
Liquid Limit (LL)	27.49	31.51	29.79	28.29	34.96	29.45	30.25
Plastic Limit (PL)	25.21	26.91	22.84	27.71	26.47	27.31	26.08
Plasticity Index = LL – PL	2.28	4.6	6.95	0.58	8.49	2.14	4.17

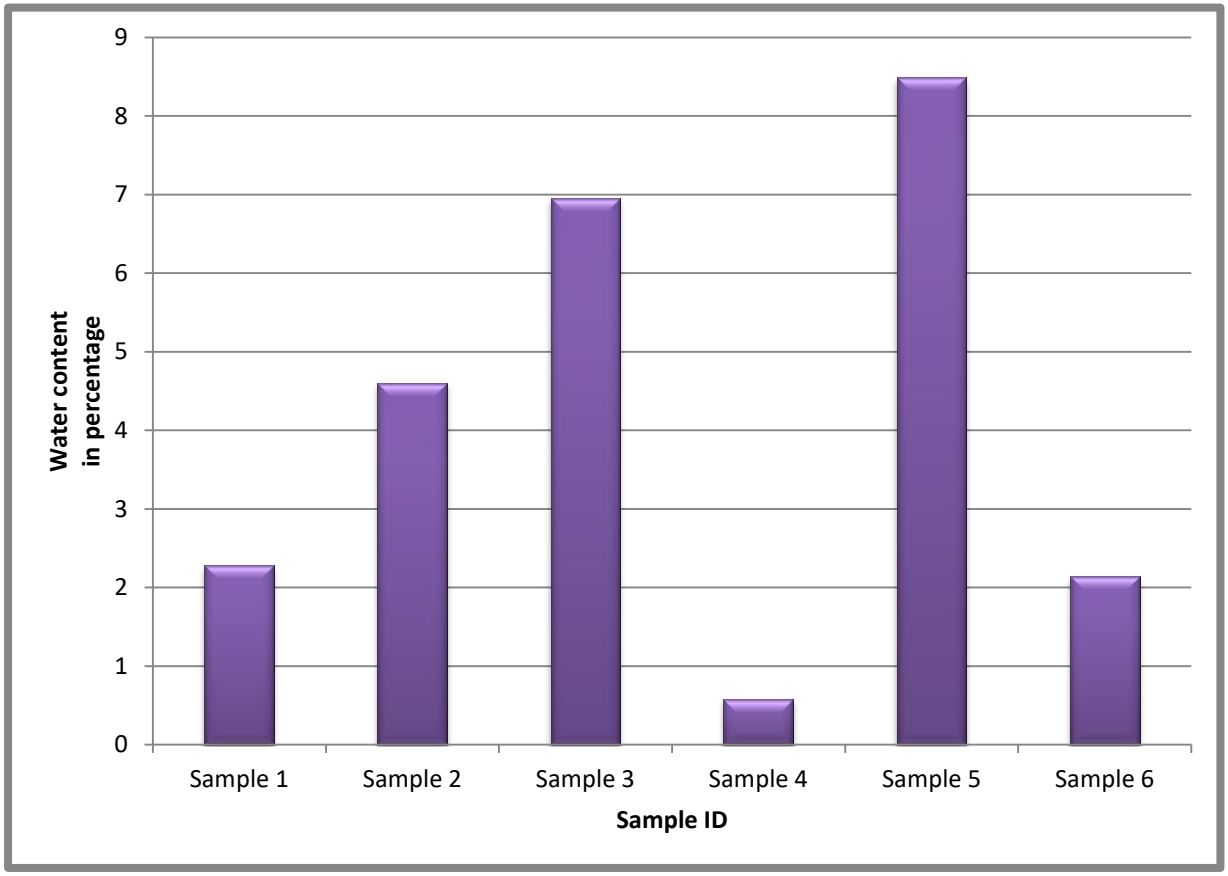

Figure 4.38: Plasticity index values within the waste rock.

© University of Venda

Table 4.12: Qualitative classification of plasticity index (Das, 2010)

Plasticity index (%)	**Description**
0	Non plastic
1-5	Slightly plastic
5-10	Low plasticity
10-20	Medium plasticity
20-40	High plasticity
>40	Very high plasticity

4.12 Compaction test

The compaction test was applied on each sample for three trials and results are presented in Appendix G. The average results of trials in each sample were noted as indicated on Table 4.13 below. Sample 1 had the lowest moisture content of 2.41% and sample 2 had the highest moisture content of 3.91% (Table 4.13). The average was calculated to be 2.88%. Sample 3 had the lowest dry density of 10.62 g/cm^3 and sample 5 had the highest dry density of 11.20 g/cm^3. The average dry density of the waste material was calculated to be 10.86 g/cm^3.

© University of Venda

Table 4.13: Average values of compaction test results

	Sample 1	Sample 2	Sample 3	Sample 4	Sample 5	Sample 6	Average
Mass of bag (g)	10.57	10.42	10.43	10.51	10.31	10.49	10.46
Dry density (g/cm^3)	10.86	10.67	10.62	10.76	11.20	11.06	10.86
Mass of wet soil (g)	5416.11	5389.97	5303.42	5377.37	5587.76	5523.13	5432.96
Mass of dry soil (g)	5289.00	5213.97	5173.52	5240.58	5454.56	5388.47	5293.35
Mass of pore water (g)	127.11	201.57	140.33	147.31	143.51	150.15	151.66
Moisture content (%)	2.41	3.91	2.71	2.81	2.63	2.79	2.88
Actual mass of dry sample (g)	5278.43	5188.40	5163.09	5230.06	5444.26	5377.98	5280.37

4.13 Classification of waste rock

AASHTO classification was used to classify the waste rock. According to the AASHTO classification, the material is classified into seven major groups: A-1 through A-7. Granular materials of which 35% or less of the material passing through the No. 200 sieve are classified under A-1, A-2 and A-3. Soils classified under groups A-4, A-5, A-6 and A-7 are mostly silt and clay-type materials of which more than 35% passes through the No. 200 sieve. The waste rock material of Consolidated Murchison has material with 29.6% passing through the 2 mm sieve, 20.6% passing through the 0.425 sieve and 7.1% passing the 0.075 mm sieve.

It was determined that the waste rock dump had an average liquid limit of 30.25%, an average plastic limit of 26.0% and an average plasticity index of 417%. These calculations where used to classify the waste rock material. Based on these calculations, it was concluded that the waste rock material falls under Group A-1-a where less than 15% of the waste material passed the No. 200 sieve. This means

© University of Venda

that the material consists largely of rock fragments (42.7%) with gravel (29.6%) and sand (20.6%) and minor silt/clay (7.1%) material. Hence, the material is excellent to good according to the AASHTO classification (Table 4.14).

Table 4.14 Revised AASHTO system of soil classification (Amadi *et al., 2015*)

General Classification	General Materials (35% or less passing 0.075 mm)							Silt-clay materials (more than 35% passing 0.075 mm)			
Group Classification	A-1		A-3	A-2				A-4	A-5	A-6	A-7
	A-1-a	A-1-b		A-2-4	A-2-5	A-2-6	A-2-7				A-7-5 A-7-6
Sieve Analysis % passing 2.00 mm (No10) 0.425 mm (No40) 0.725 mm (No200)	50max 30max 15max	50max 25max	51min 10max	35max	35max	35max	35max	36min	36min	36min	36min
Characteristics of fraction passing Liquid limit Plastic Index	6max		N.P	40max 10max	41min 10max	40max 11min	41min 11min	40max 10max	41min 10max	40max 11min	40min 11min
Usual types of significant Constituent material	Stone fragment Gravel and sand		Fine Sand	Silty or clayey Gravel and sand				Silty soils		Clayey soils	
General rating	Excellent to Good							Fair to poor			

From the sieve and Atterberg limit data, it was deduced that less than 15% of the materials passed the 0.075 mm diameter sieve, thus the material is granular. Furthermore, less than 30% passed the 0.425 mm diameter sieve and less than 50% of the material passed the 2 mm diameter sieve. Atterberg limit tests indicated that the mine waste rock had liquid limit and plastic limit of 30.25% and 26.08% respectively with a plasticity index of 4.17%. This means that the mine waste rock is suitable for construction purposes as a sub-base layer since the material fulfil the given standards. The waste rock was classified under Group A-1-a which has a general sub-grade rating of “Excellent to Good” in the AASHTO classification.

Under the AASHTO system, granular soils fall into classes A-1 to A-3. A-1 materials consist of well-graded granular materials (Das, 2010). Material classified under group A-1-a can be used satisfactorily as subgrade or sub-base material if properly drained (Das, 2010). In addition, material must be properly compacted and covered with an adequate thickness of pavement (base and/or surface cover) for the surface load to be carried. The results from compaction tests showed that the waste rock

© University of Venda

material had a high dry density which was determined to be 11.06 g/cm³ obtained at a slightly low moisture content of 2.88%. This indicates that the waste rock is competent and has an acceptable dry density.

© University of Venda

CHAPTER FIVE: CONCLUSONS AND RECOMMENDATIONS

5.1 Conclusions

- The Consolidated Murchison tailings dam was found to have distinct oxidation zone extending from the surface up to a depth of 3 m. The transitional zone was not established. The unoxidized zone was quite extensive, starting from a depth of 3 m up to the bottom of the dam.
- This study established that gold was erratically distributed within the tailings dam of Consolidated Murchison mine with the lowest and highest values of 200 mg/kg and 1880 mg/kg respectively averaging at 670 mg/kg. The value of gold within the tailings dam was found to be 8 897. 81 kg which at the current gold market price is equal to US$ 306 932 396.00 (R 4 281 706 924.20).
- This study shows that Pb, Ni and Cr had the highest average values of (ppm); 5631.5, 2062.6 and 1345 respectively, with the lowest values obtained in Cd, Co and Cu averaging (ppm); 0.01, 19.8 and 42.1 respectively.
- The values of heavy metal were found to be decreasing away from the tailings dam along the north, west, east and south. Pollution index values at 5 km away from the tailings dam (along north and west) were less than 0.7. However, pollution index values of 4.8 and 7.1 were recorded in Cr and Ni respectively at a distance of 5 km away from the tailings dam along the eastern direction. This implies that this area (eastern direction) was highly polluted by Cr and Ni.
- The waste rock was classified under Group A-1-a using the AASHTO classification system. The waste rock consisted mainly of rock fragments of 42.7% with gravel material of 29.6%, sand material of 20.6%, and limited clay/silt material of 7.1%. The waste rock material was found to be having relatively low water holding capacity

5.2 Recommendations

- The current study of the Consolidated Murchison tailings dam was limited to the first 8 m due to technical limitations. It is therefore recommended that future assessment of heavy metals within the tailings dam should cover the entire thickness of the dam which is 30 m.

© University of Venda

- The study recommends reprocessing of gold within this tailings dam. Reprocessing of gold from gold tailings dam in South Africa is currently increasing. It has been proven that it is possible to extract gold from tailings up to 87.79% (Dehghani *et al*., 2009). Tailings residues (waste after reprocessing) should be kept in a more convenient area where no pollution will result from such tailings.
- It is recommended that waste rock be used as material for road construction since it is of excellent to good quality for road construction as indicated on the AASHTO classification system.

© University of Venda

REFERENCES

Abdul-Wahab, S. and Marikar, F. (2013). The Environmental Impact of Gold Mines: Pollution and Plants. Elsevier; Sandiego, CA, USA: 525 pp.

Ahmari, S. and Zhang, L. Y. (2012). Production of Eco-Friendly Bricks from Copper Mine Tailings Through Geopolymerization. Construction and Building Materials, Vol. 29, pp. 323-331.

Amadi, W. G., Akande, Okunlola, I. A. and Jimoh, M. O. (2015). Assessment of the Geotechnical Properties of Lateritic Soils in Minna, North Central Nigeria for Road Design and Construction. American Journal of Mining and Metallurgy, Vol. 3, No. 1, pp. 15-20.

Annicaert, B. (2013). Treatment of Tailings from Artisanal Gold Mining in Nicaragua. Master's dissertation, Ghent University. 77 pp.

Atkinson, J. (1993). An Introduction to the Mechanics of Soil and Foundations. McGraw-Hill Book Company (London), pp. 52-133.

Baker, B. J. and Banfield, J. F. (2013). Microbial Communities in Acid Mine Drainage. FEMS Microbial. Ecol, pp. 44:139-152.

Benckert, A. and Eurenius, J. (2001). Tailings Dam Constructions - Seminar on Safe Tailings Dam Constructions. Gallivare, Swedish Mining Association, Natur Yards Verket, European Commission, pp. 30-36.

Bland, J. M. and Altman, D. G. (1996). Statistics Notes: Measurement Error Proportional to the Mean. Bmj, Vol. 313, No. 7049. 106 pp.

Bitala, M. F., Kweyunga, C. and Manoko, M. L. (2009). Levels of Heavy Metals and Cyanide in Soil, Sediment and Water from the Vicinity of North Mara Gold Mine in Tarime District, Tanzania. A Report Presented to the Christian Council of Tanzania, Dodoma. 20 pp.

Blight, G. E. (1998). Construction of Tailings Dams. Case Studies on Tailings Management. International Council on Metals and the Environment (ICME), United Nations Environment Programme (UNEP), pp. 9-10.

Bosch, D. W. (1987). Retreatment of Residues and Waste Rock. Extr. Metall. Gold South Africa, Vol. 2, pp. 707-743.

© University of Venda

Bronshtein, I. N., Semendyayev, K. A., Musiol, G. and Mühlig, H. (2007). Handbook of Mathematics. Springer, Berlin, Heidelberg, New York. 1159 pp.

Collins, R. J. and Miller, R. H. (1979). Utilization of Mining and Mineral Processing Wastes in the United States. Environmental Geochemistry and Health, Vol. 1, No. 1, pp. 8-19.

Consolidated Murchison Limited Information Brochure (1991). Unpublished Mine Report. 89 pp.

Daniel, A. and Downing, B. W. (2011). Investigating the Reprocessing of Carolin Mine Tailings. Tailings and Mine Waste 2011, Vancouver, Canada, pp. 53-65.

Das, B. M. (2006). Principles of Geotechnical Engineering. Stamford. CT: Thomson Learning College. 686 pp.

Das, B. M. (2008). Advanced Soil Mechanics. Taylor and Francis Group, New York. 594 pp.

Das B. M. (2010). Principles of Geotechnical Engineering. Connecticut : Cengage Learning, pp. 73-104.

Das, B. M. (2013). Advanced Soil Mechanics. Press, pp 29-39.

Davies, M. P. (2002). Tailings Impoundment Failures: Are Geotechnical Engineers Listening. Geotechnical News, pp. 31-36.

Davis, D. R., Paterson, D. B. and Griffiths, D. H. C. (1988). Antimony in South Africa. Journal of the South African Institute of Mining and Metallurgy, Vol. 86, pp. 173-193.

Dehghani, A., Ostad-rahimi, M., Mojtahedzadeh, S. H. and Gharibi, K. K. (2009). Recovery of Gold from the Mouteh Gold Mine Tailings Dam. The Journal of the Southern Africa Institute of Mining and Metallurgy, Vol. 109, pp. 417-421.

DME: Department of Mineral and Energy (1999). Guidelines on the Safe Design and Operating Standards for Tailings Storage. Department of Minerals and Energy, Western Australia. ISBN: 0730978087. 54 pp.

Engels, J. (2006). An Expert Management System for Surface Tailings Storage. Doctoral Dissertation, University of Leeds, 285 pp.

© University of Venda

Environment and Climate Change Canada (2017). Environmental Code of Practice for Metal Mines. https://www.canada.ca/en/environment-climate-change.atom.xml. [Accessed: 22 June 2017].

Environment Canada (2012). National Pollutant Release; Available from: http://www.ec.gc.ca/inrp-npri/default.asp?lang=Enandn=4A577BB9-1. [Accessed: 24 April 2017].

Esri. (2008). ArcGIS Desktop n10.2.2: Release 10. Redlands, CA: Environmental Systems Research Institution.

Ferguson, J. E. (1990). The Heavy Elements: Chemistry, Environmental Impact and Health Effects, No. 628. 53 pp.

Fuggle, R. F. and Rabie, M. A. (1992). Environmental Management in South Africa. Juta and Company Limited, ISBN 10: 0702128473, 1142 pp.

Government Gazette. (2012). National Environmental Health Norms and Standards for Premises and Acceptable Monitoring Standards for Environmental Health Practitioners, pp. 41-42.

Hansbo, S. (1957). A New Approach to the Determination of the Shear Strength of Clay by the Fall-Cone Test. Royal Swedish Geotechnical Institute. Vol. 14, pp. 7–47.

Harada, M. (1964). Neuropsychiatric Disturbances due to Organic Mercury Poisoning During the Prenatal Period. Vol. 66. 429 pp.

Harada, M., Jajiri, M., Inoue, Y., Hotta, N., Fujino, T. and Ueda, K. (2011). Mercury Pollution in First Nations Groups in Ontario, Canada: 35 Years of Canadian Minamata Disease. Journal of Minamata Studies, Vol. 3, pp. 3-30.

Holden, W. N. (2012). Ecclesial Opposition to Large-Scale Mining on Samar: Neoliberalism meets the Church of the Poor in a Wounded Land. University of Calgary. Canada, Vol. 3, No. 3, pp. 833-861.

Holts, R. D. and Kovacs, W. D. (1981). Introduction to Geotechnical Engineering. Prentice Hall Inc. N.J., U.S.A. 717 pp.

© University of Venda

Hong-gui, D., Teng-feng, G., Ming-hui, L. and Xu, D. (2012). Comprehensive Assessment Model on Heavy Metal Pollution in Soil. Int. J. Electrochem Sci, Vol. 7, No. 6, pp. 5286-5296.

Jain, S. K., Garg, S. P. and Rai, M. (1983). Autoclaved Calcium Silicate Bricks from Gold Mine Tailings. Research and Industry, Vol. 28, pp. 170–172.

Jakubick, A., McKenna, G. and Robertson, A. (2003). Stabilisation of Tailings Deposits: International Experience. Mining and the Environment III, Sudbury, Ontario, Canada, pp. 1-9.

Karlsson, R. (1977). Consistency Limits: A Manual for the Performance and Interpretation of Laboratory Investigations. Part 6, Swedish Council for Building Research. 40 pp.

Krishna, R. (2002). Engineering Properties of Soils Based on Laboratory Testing. UIC, Vol. 44, pp. 44-59.

Laxen, P. (1984) March. The Carbon-in-Pulp Plant at Rand Mines Milling and Mining Company Limited: Problems and Developments (Retroactive Coverage). In MINTEK 50: International Conference on Mineral Science and Technology, Vol. 2, pp. 695-706.

Lehohla, P. (2013). South African Statistics. [ebook] Statistics SA. Available at: http://www.statssa.gov.za/publications/SAStatistics/SAStatistics2013.pdf [Accessed 20 February 2019].

Lottermoser, B. G. (2010). Tailings. In B. Lottermoser G. (Ed.): Mine Wastes: Characterization, Treatment and Environmental Impacts, Springer, Berlin, pp. 205-241.

Malatse, M. and Ndlovu, S. (2015). The Viability of Using the Witwatersrand Gold Mine Tailings for Brick Making. Journal of the Southern African Institute of Mining and Metallurgy, Vol. 115, No. 4, pp. 321-327.

Martin, T. E. (1999). Characterization of Pore Pressure Conditions in Upstream Tailings Dams. Tailings and Mine Waste '99, Fort Collins, Colourado, A. A. Balkema, Rotterdam, pp. 303-313.

© University of Venda

Matthew, B. J., Sudhakar, M. and Natarajan, C. (2013): Development of Coal Ash-GGBS Based Geopolymer Bricks. Eur Int J Sci Technol, Vol. 2, pp. 133-139.

Matshusa, K., Ogola, J. S. and Maas, K. (2012). Dispersion of Metals at Louis Moore Gold Tailings Dam, Limpopo Province, South Africa. Proccedings of the International Mine Water Association Symposium; Bunbury, Australia. 30 September 2012, pp. 334A-334.

McGlinchey, D. (2005). Characterisation of bulk solids. (Ed.). UK: Blackwell. 266 pp.

Minnitt, R. C. A. and Anhaeusser, C. R. (1992). Gravitational and DiaPiric Structural History of the Eastern Portion of the Archean Murchison Greenstone Belt, South Africa. Journal of African Earth Sciences and the Middle East, Vol. 15, No. 3-4, pp. 429-440.

Mohammed, M. M. (2015). Re-processing of Gold Tailings at Ariab Area North-East of Sudan. Doctoral Dissertation, UOFK. 83 pp.

Muir, A., Mitchell, J., Flatman, S. R. and Sabbagha, C. (2005). A Practical Guide to Re-treatment of Gold Processing Residues. Minerals Engineering, pp. 811-824.

Murthy, D. S. R., Kumar, V. and Rao, K. V. (2003). Extraction of Gold from an Indian Low-Grade Refractory Gold Ore through Physical Beneficiation and Thiourea Leaching. Hydrometallurgy, Vol, 68, No. 1, pp. 125-130.

Nel, J. H. (2008). An investigation of Mine Closure: Gold Mine Case Studies on the East Rand in South Africa. 72 pp.

Norman, D. (1998). Innovations and Trends in Reclamation of Metal Mine Tailings in Washington. Washington Geology, pp. 29-42.

Ogola, J. S. (2002). Dispersion of Heavy Metals and their Potential Impacts on the Environment: A case study of Gold Tailings Dams in Giyani Belt, Limpopo Province, South Africa. In Wolkersdorfer, Ch. and Frend, A. “Mine Water and Innovative Thinking” – Proceedings of the IMWA 2010 Symposium, Sydney, NS, Cape Breton University Press, pp. 591-594.

Plumlee, G. S. and Nash, J. T. (1995). Geoenvironmental Models of Mineral Deposits–Fundamentals and Applications. US Geological Survey Open-file Report 95-831, pp. 1-9.

© University of Venda

Rajeswari, T. R. and Sailaja, N. (2014). Impact of Heavy Metals on Environmental Pollution. Journal of Chemical and Pharmaceutical Sciences, Vol. 3, pp. 175-181.

Ramuada, T. E. (2014). Investigation of the Occurrence and Distribution of Gold and Heavy Metals in the Giyani Tailings Dams: A case Study of Klein Letaba Tailings Dam, Limpopo, South Africa, Unpublished Dissertation, University of Venda, pp. 1-60.

Rankin, W. J. (2011). Minerals, Metals and Sustainability: Meeting Future Material Needs. CSIRO publishing, pp 1-135.

Ritcey, G. M. (1989). Tailings Management: Problems and Solutions in the Mining Industry. Amsterdam, Elsevier, Vol. 6, 970 pp.

Ritcey, G. M. (2005). Tailings Management in Gold Plants. Hydrometallurgy, 320 pp.

Robertson, A. and Wels, C. (1999). Cover Placement on Extremely Weak Compressible Tailings. Robertson GeoConsultants Inc., Vancouver, Canada. 16 pp.

Robinsky, E. I. (2000). Sustainable Development in Disposal of Tailings. Tailings and Mine Waste. Colourado, USA, A. A. Balkema, Rotterdam, pp. 39-48.

Roy, S., Adhikari, G. R. and Gupta, R. N. (2007). Use of Gold Mill Tailings in Making Bricks: a Feasibility Study. Waste Management and Research, Vol. 25, No. 5, pp. 475-482.

SANS (1998). Code of Practice for Mine Residue Deposits. South African Bureau of Standards. 129 pp.

Seal II, R. R., Foley, N. K. and Wanty, R. B. (2002). Introduction to Geoenvironmental Models of Mineral Deposits. Progress on Geoenvironmental Models for Selected Mineral Deposit Types, Vol. 2, 213 pp.

Seed, H. B. and Idriss, I. M. (1967). Analysis of Soil Liquefaction: Niigata Earthquake. Journal of the Soil Mechanics and Foundations Division, Vol. 93, No. 3, pp. 83-108.

Schwarz-Schampera, U., Terblanche, H. and Oberthür, T. (2010). Volcanic-Hosted Massive Sulfide Deposits in the Murchison Greenstone Belt, South Africa. Mineralium Deposita, Vol. 45, No. 2, pp. 113-145.

© University of Venda

Sharp, B. E. (2012). Blocks from Mine Waste and Industrial Waste. http://besharp.achider.ord/IMG/pdf/blocks_from_industrial_waste.pdf. [Accessed: 13 February 2017].

Sullivan, P. J., Yelton, J. L. and Reddy, K. J. (1988). Iron Sulfide Oxidation and the Chemistry of Acid Generation. Environmental Geology and Water Sciences, Vol. 11, No. 3, pp. 289-295.

Standard Methods of Testing Road Construction Material (1986); http://asphalt.csir.co.za/tmh/. [Accessed: 19 November 2018]

Tan, K. H. (2005). Soil Sampling, Preparation, and Analysis. CRC press. 680 pp.

TriAusMin. (2010). Woodlawn Retreatment Project. Retrieved, 2017, from http://www.trioriginminerals.com.au/projects.asp?id=4. [Accessed: 25 July 2017].

Ulrich, B. F. and Fourie, A. B. (2003). Assessment of the Potential for Static Liquefaction of a Tailings Dam using Laboratory and Field Testing. In Proceedings of the 10th International Conference on Tailings and Mine Waste, Vail, Colo, pp. 12-15.

Utochkin, I. S. (2015). Ensemble Summary Statistics as a Basis for Rapid Visual Categorization. Journal of Vision, Vol 15, No. 4. 8 pp.

Viljoen, M. (2009). The Life, Death and Revival of the Central Rand Goldfied. In World Gold Conference, Vol. 2009, pp.131-138.

Vick, S. G. (1990). Planning, Design, and Analysis of Tailings Dams. Vancouver, BiTech.lSBN: 0921095120.xi. 369 pp.

Wedepohl, K. (1978). Handbook of Geochemistry. Berlin: Springer. 442 pp.

Willson, C. and Viljoen, M.J. (1986). The Athens Antimony Ore Body, Murchison Greenstone Belt. Mineral Deposits of Southern Africa: Johannesburg, South Africa, Geological Society of South Africa, pp. 333-338.

Wise-Uranium (2011). Chronology of Major Tailings Dam Failures. Retrieved, 2017, from http://www.wise-uranium.org/mdaf.html [Accessed: 16 April 2017].

Xinyi, W. (2012). Generic Decision Making on the Retreatment of Copper Tailings Dams, Vancouver, University of British Columbia, pp. 1-28.

© University of Venda

Zhang, L. (2013). Production of Bricks from Waste Materials–A Review. Construction and Building Materials, Vol. 47, pp. 643-655.

Zheng, G., Yue, L., Li, Z. and Chen, C. (2006). Assessment on Heavy Metals Pollution of Agricultural Soil in Guanzhong District. Journal of Geographical Sciences, Vol. 16, No. 1, pp.105-113.

www.miningreview.com/news/ergo-to-be-reborn/. [Accessed: 12 February 2017].

www.tailings.info/casestudies/marrespruit.html. [Accessed: 24 March 2017].

© University of Venda

University of Venda
Creating Future Leaders

APPENDICES

© University of Venda

Appendix A

Description of Tailings Sampling Holes

Tailings descrition along hole P1H1

From (m)	To (m)	Colour	.Description
0	1.6	Dark brownish	Fine dry tailings and easy to auger
1.6	8	Light brownish	Fine moist tailings and competent to auger

Tailings descrition along hole P1H2

From (m)	To (m)	Colour	Description
0	2	Dark brownish	Fine dry tailings and easy to auger
2	8	Light brownish	Fine moist tailings and competent to auger

Tailings descrition along hole P2H1

From (m)	To (m)	Colour	Description
0	1.7	Dark brownish	Fine dry tailings and easy to auger
1.7	8	Light brownish	Fine moist tailings and competent to auger

Tailings descrition along hole P2H2

From (m)	To (m)	Colour	Description
0	2	Dark brownish	Fine dry tailings and easy to auger
2	8	Light brownish	Fine moist tailings and competent to auger

© University of Venda

Tailings descrition along hole P2H3

From (m)	To (m)	Colour	Description
0	1.8	Dark brownish	Fine dry tailings and easy to auger
1.8	8	Light brownish	Fine moist tailings and competent to auger

Tailings descrition along hole P3H1

From (m)	To (m)	Colour	Description
0	2.7	Dark brownish	Fine dry tailings and easy to auger
2.7	8	Light brownish	Fine moist tailings and competent to auger

Tailings descrition along hole P3H2

From (m)	To (m)	Colour	Description
0	1.7	Dark brownish	Fine dry tailings and easy to auger
1.7	3.6	Light brownish	Fine moist tailings and competent to auger
3.6	3.9	Greyish	Sticky wet mud and very competent to auger
3.9	8	Light brownish	Fine moist tailings and competent to auger

Tailings descrition along hole P3H3

From (m)	To (m)	Colour	Description
0	2	Dark brownish	Fine dry tailings and easy to auger
2	5.4	Light brownish	Fine moist tailings and competent to auger

© University of Venda

5.4	5.9	Greyish	Sticky wet mud and very competent to auger
5.9	8	Light brownish	Fine moist tailings and competent to auger

Tailings descrition along hole P3H4

From (m)	To (m)	Colour	Description
0	3	Dark brownish	Fine dry tailings and easy to auger
3	6.2	Light brownish	Fine moist tailings and competent to auger
6.2	7	Greyish	Sticky wet mud and very competent to auger
7	8	Light brownish	Fine moist tailings and competent to auger

Tailings descrition along hole P4H1

From (m)	To (m)	Colour	Description
0	1.6	Dark brownish	Fine dry tailings and easy to auger
1.6	4.2	Light brownish	Fine moist tailings and competent to auger
4.2	4.5	Greyish	Sticky wet mud and very competent to auger
4.5	8	Light brownish	Fine moist tailings and competent to auger

© University of Venda

Tailings descrition along hole P4H2

From (m)	To (m)	Colour	Description
0	2	Dark brownish	Fine dry tailings and easy to auger
2	4.4	Light brownish	Fine moist tailings and competent to auger
4.4	5.1	Greyish	Sticky wet mud and very competent to auger
5.1	8	Light brownish	Fine moist tailings and competent to auger

Tailings descrition along hole P4H3

From (m)	To (m)	Colour	Description
0	2.6	Dark brownish	Fine dry tailings and easy to auger
2.6	8	Light brownish	Fine moist tailings and competent to auger

Tailings descrition along hole P4H4

From (m)	To (m)	Colour	Description
0	2.4	Dark brownish	Fine dry tailings and easy to auger
2.4	6	Light brownish	Fine moist tailings and competent to auger
6	6.6	Greyish	Sticky wet mud and very competent to auger
6.6	8	Light brownish	Fine moist tailings and competent to auger

© University of Venda

Tailings descrition along hole P5H5

From (m)	To (m)	Colour	Description
0	3	Dark brownish	Fine dry tailings and easy to auger
3	8	Light brownish	Fine moist tailings and competent to auger

© University of Venda

Appendix B

Gold values within the tailings dam

Sample ID	Au (mg/kg)
P1H1S1	895
P1H1S2	704
P1H1S3	779
P1H1S4	514
P1H1S5	830
P1H1S6	542
P1H2S1	665
P1H2S2	351
P1H2S3	195
P1H2S4	626
P1H2S5	534
P1H2S6	702
P2H1S1	905
P2H1S2	709
P2H1S3	605
P2H1S4	671
P2H1S5	406
P2H1S6	629
P2H2S1	696
P2H2S2	619
P2H2S3	819
P2H2S4	567
P2H2S5	595
P2H2S6	484
P2H3S1	717
P2H3S2	869
P2H3S3	604
P2H3S4	734
P2H3S5	534
P2H3S6	652
P3H1S1	706
P3H1S2	643
P3H1S3	777
P3H1S4	894
P3H1S5	596
P3H1S6	745
P3H2S1	961

© University of Venda

P3H2S2	832
P3H2S3	648
P3H2S4	591
P3H2S5	499
P3H2S6	454
P3H3S1	733
P3H3S2	557
P3H3S3	907
P3H3S4	796
P3H3S5	700
P3H3S6	567
P3H4S1	694
P3H4S2	820
P3H4S3	769
P3H4S4	1880
P3H4S5	496
P3H4S6	661
P4H1S1	334
P4H1S2	276
P4H1S3	629
P4H1S4	499
P4H1S5	699
P4H1S6	829
P4H2S1	778
P4H2S2	933
P4H2S3	758
P4H2S4	441
P4H2S5	732
P4H2S6	575
P4H3S1	659
P4H3S2	625
P4H3S3	557
P4H3S4	458
P4H3S5	547
P4H3S6	614
P4H4S1	689
P4H4S2	748
P4H4S3	584
P4H4S4	864
P4H4S5	947
P4H4S6	778
P4H5S1	670

© University of Venda

P4H5S2	610
P4H5S3	590
P4H5S4	540
P4H5S5	620
P4H5S6	580

© University of Venda

Appendix C

Heavy metal values within the tailings dam

Sample ID	Cr (PPM)	Co (PPM)	Ni (PPM)	Cu (PPM)	Zn (PPM)	As (PPM)	Cd (PPM)	Pb (PPM)
P1H1S1	1203.8	21.8	2046.5	43.6	59.2	624.4	<0.1	5817.3
P1H1S2	1281.6	22.9	1930.7	8.1	63.4	647.5	0.1	6033
P1H1S3	1299.2	24.9	2209.8	6.1	65.8	565.7	<0.1	5270.6
P1H1S4	1336.3	25.1	2435.6	3.3	67	641.6	<0.1	5977.2
P1H1S5	1217.8	23	2434.9	10.2	65.4	781.9	<0.1	7284.7
P1H1S6	1132.7	21.7	1789.9	19.8	70.4	517.9	<0.1	4825.3
P1H1S7	1171.9	18.6	2238.3	76.6	52.8	524.1	<0.1	4883.3
P1H1S8	1174.1	21.1	2103.5	11.3	59.8	743.7	<0.1	6929.2
P1H2S1	1025.9	21.2	1887.8	55.1	55.4	896.2	<0.1	8349.6
P1H2S2	934.8	19.1	1955	23.1	49.6	648.4	<0.1	6040.7
P1H2S3	1160.7	22.6	2817	18.4	59.8	932.3	<0.1	8686.2
P1H2S4	1290.9	19.7	1968.6	67.2	64.7	567.4	<0.1	5286.2
P1H2S5	1303.7	22.7	2378.4	2.3	69.3	618.2	<0.1	5759.7
P1H2S6	1373.9	20.3	2040.9	12	75.2	614.3	<0.1	5723.6
P1H2S7	1125.8	20.6	1736.1	8.8	72.3	471.8	<0.1	4395.9
P1H2S8	1135.4	20.4	1841.4	15.7	85.1	558.7	<0.1	5205.7
P2H1S1	1119.8	20.1	2247.5	56.8	53.9	805.9	<0.1	7508.3
P2H1S2	1233.3	17.8	1858.8	39	60.2	510.7	<0.1	4757.7
P2H1S3	1314.6	18.6	1988.4	41.1	61.3	514.6	<0.1	4794.7
P2H1S4	1362.2	19.9	1953.5	24.8	66.8	549.4	<0.1	5118.8
P2H1S5	1630.8	21.1	2201.6	3.7	83	500.3	<0.1	4661.6
P2H1S6	1453.4	21.1	1986.7	8.9	80.7	571.2	<0.1	5321.7
P2H1S7	1355.7	19	1868.7	10.2	78.4	510.7	<0.1	4758.6
P2H1S8	1228.5	18.9	1690.9	21.4	72.4	569	<0.1	5301.2
P2H2S1	1153.1	21.3	2082.3	38.5	55.6	650.9	<0.1	6064.7
P2H2S2	1212.5	21.3	2112.3	40.3	61.7	652.8	<0.1	6081.7
P2H2S3	1173.3	19.6	1892.9	28.5	61.4	540.3	<0.1	5034
P2H2S4	1267.1	21.1	2158.4	32.6	61.8	601.7	<0.1	5606.1
P2H2S5	1497.8	21.7	2091.4	22.1	74.7	545.7	0.1	5084.5
P2H2S6	1439.3	23.1	2154	4.4	88.1	607.7	0.1	5661.5
P2H2S7	1566.3	22.3	2268.6	13.7	88.7	570.5	0.1	5314.9
P2H2S8	1278.6	21.1	2246	9	91.7	774	<0.1	7211.4
P2H3S1	1299.5	19.8	2049.3	75	65.9	669.2	<0.1	6234.5
P2H3S2	1420.5	19.3	2239.2	46.3	68.2	575.6	<0.1	5362.4

© University of Venda

P2H3S3	1306.4	19.5	2253.7	46.4	63.8	641.2	<0.1	5974.2
P2H3S4	1514.7	19.9	2003.6	52.2	70	527.4	<0.1	4913.5
P2H3S5	1331.7	21.9	2028.7	5.9	72.4	587.8	<0.1	5476.2
P2H3S6	1666.5	21.8	2065.2	13.2	83.8	549.8	0.1	5122.1
P2H3S7	1730.6	22.1	2094.6	19.3	86.4	496.8	<0.1	4628.6
P2H3S8	1496.2	19.9	1790.9	23.7	87.1	527.9	<0.1	4918.5
P3H1S1	1180.1	18.5	1744.1	83.4	59.9	718.9	<0.1	6697.6
P3H1S2	1273.6	19.5	2085.4	42.6	65.8	800.5	<0.1	7458.1
P3H1S3	1231.1	19.8	2708.2	76	65	737	0.1	6866.8
P3H1S4	1269.1	18.5	2282.5	46.9	65.5	620.2	<0.1	5777.9
P3H1S5	1231.8	18.2	2021.3	32.6	68.3	619.5	<0.1	5771.4
P3H1S6	1292.2	18.2	1973.5	48.1	82.1	572	<0.1	5329.7
P3H1S7	1210.1	19.4	2085	22.7	72.8	631.4	<0.1	5883
P3H1S8	1250.2	18.9	1961.7	15.9	72.3	612.1	<0.1	5702.9
P3H2S1	1269.2	19.2	2216.9	49.1	64.4	700.3	<0.1	6524.4
P3H2S2	1271.8	18	2042.4	33.3	61.2	546.2	<0.1	5088.5
P3H2S3	1567.3	19.1	2190.5	48.7	70.1	547.8	<0.1	5104
P3H2S4	1358.1	19	1880.9	16.1	68.7	525.2	<0.1	4892.9
P3H2S5	1397	20.4	2016.6	7.9	81.7	576.8	0.1	5374.2
P3H2S6	1374	20.3	2012.7	6.6	82.2	545.6	<0.1	5083.1
P3H2S7	1462.9	19	1994.9	10	82.5	582.1	<0.1	5423.3
P3H2S8	1218.6	18.9	1705.8	10.7	79.1	540.3	<0.1	5033.9
P3H3S1	1258.9	22.4	2096.8	106.5	65	636.4	<0.1	5928.8
P3H3S2	1258.4	20.9	1784.7	80.7	62.3	452.5	<0.1	4215.6
P3H3S3	1369.5	20.2	1913.5	69.8	65	556.5	<0.1	5184.4
P3H3S4	1643	22.1	2164.1	148.6	79.5	698.3	0.1	6505.7
P3H3S5	1660.8	21.6	1968.1	77.4	75.5	528	<0.1	4919.4
P3H3S6	1570	22.1	1998	27.4	77.8	504.6	0.1	4701.1
P3H3S7	1229.2	20.6	2040.3	56.5	61.9	668.7	<0.1	6230
P3H3S8	1704.6	21.7	2078.4	26.4	87.1	462.3	<0.1	4306.9
P3H4S1	1509.7	19.2	1985.9	61.3	64.2	517.3	<0.1	4819.7
P3H4S2	1824.2	19.6	2049.1	261	81.7	494.8	<0.1	4609.9
P3H4S3	1687.4	19.3	2060.1	84.8	78	481.1	<0.1	4482.6
P3H4S4	1672.3	19	2106.3	139.8	76.2	523	<0.1	4873.1
P3H4S5	1632.1	19.8	1894.1	72.4	74.4	518.8	<0.1	4833.3
P3H4S6	1397.6	19.1	1862.7	22.2	69.9	501.5	<0.1	4672.6
P3H4S7	1597.3	20.7	1919.3	18.1	82.1	467.4	<0.1	4354.7
P3H4S8	1620.1	21	1939.6	26.8	89.7	543.3	<0.1	5061.8
P4H1S1	1111	18.9	2353.6	60.6	59.7	1016.6	0.1	9471.8

© University of Venda

P4H1S2	1125.2	19.6	2143	34	63	773.7	0	7208.2
P4H1S3	1282.3	20.1	2483.8	56.8	64.3	659.6	0.1	6145
P4H1S4	1395.6	20.6	2363	14.1	70.3	627.6	<0.1	5847.1
P4H1S5	1217.9	19.4	1939.2	54.2	63.7	719.6	<0.1	6704.8
P4H1S6	1270.9	18.2	1892.3	47	66.7	670.3	<0.1	6245
P4H1S7	1179	19.1	1880.3	38.4	61.8	719	<0.1	6698.5
P4H1S8	1298.3	18.5	2024.6	50.3	72.6	539.7	<0.1	5028
P4H2S1	1329.9	19	2123.4	92.5	66.3	687.9	<0.1	6409.5
P4H2S2	1462.2	18.1	1995	87.1	69.3	532.7	0.1	4963.1
P4H2S3	1459.3	18	1967.7	50.1	66.7	513	<0.1	4779.4
P4H2S4	1463.3	20.8	2190.6	12.7	74.6	572	<0.1	5329
P4H2S5	1664.1	19.2	2001	16.3	74.7	517.8	<0.1	4824.2
P4H2S6	1439	19	1920.6	17.6	78.9	490.3	<0.1	4568.3
P4H2S7	1162.9	18.1	1756.7	12.5	73.3	432.9	<0.1	4033
P4H2S8	1463.8	18.9	1781.1	25.5	81.3	549.9	<0.1	5123.4
P4H3S1	1246.3	17	2000.5	21.4	59.7	540.2	0.1	5033
P4H3S2	1429	18.7	2042.6	26.9	69.2	552.7	0.1	5149.4
P4H3S3	1488.7	19.2	2156.5	13.5	71	581.7	<0.1	5420
P4H3S4	1337	20.1	2124.5	26.9	70.7	586.8	<0.1	5467.5
P4H3S5	1348.5	18.5	1954.7	17.3	69	617.1	<0.1	5749.4
P4H3S6	1311.3	17.6	1989.9	25.2	73	596.4	<0.1	5556.4
P4H3S7	1161.3	17.9	1773.7	23.2	73.7	562.9	<0.1	5244.2
P4H3S8	1227.6	18	2160	36.1	77	583	<0.1	5432
P4H4S1	1266	20.1	2247.2	46.2	63.1	791.5	0.1	7373.9
P4H4S2	1312.9	20	2653.4	52.7	68	716.6	<0.1	6676.6
P4H4S3	1188.4	18.9	2241.9	90.1	61.6	901.3	<0.1	8396.9
P4H4S4	1270.3	19	2059.6	93.4	61.7	747.6	<0.1	6964.9
P4H4S5	1302	17.3	2166.3	110.3	65.9	521.8	<0.1	4861.1
P4H4S6	1374	18.7	2125.9	132.4	68.9	636	0.1	5925.5
P4H4S7	1289.9	17.3	2129.1	41.7	62.2	558.7	0.1	5205.2
P4H4S8	1292.6	17.5	1986	45.9	66.6	522.1	<0.1	4864.4
P4H5S1	1280.7	18.7	2123.7	53.4	63.0	673.2	0.1	6272.1
P4H5S2	1401.4	18.9	2230.3	55.6	68.8	600.7	0.1	5596.4
P4H5S3	1378.8	18.7	2122.0	51.2	66.4	665.3	<0.1	6198.8
P4H5S4	1356.9	20.0	2124.9	44.3	69.0	635.5	<0.1	5920.5
P4H5S5	1438.2	18.3	2040.7	48.0	69.9	552.2	<0.1	5144.9
P4H5S6	1374.8	18.4	2012.1	58.4	73.6	574.2	0.1	5350.1
P4H5S7	1204.7	17.8	1886.5	25.8	69.7	518.2	<0.1	4827.5
P4H5S8	1328.0	18.1	1975.7	35.8	75.0	551.7	<0.1	5139.9

© University of Venda

Appendix D

Dispersion of heavy metals along northern part of the tailings dam

Sample ID	Cr (PPM)	Co (PPM)	Ni (PPM)	Cu (PPM)	Zn (PPM)	As (PPM)	Cd (PPM)	Pb (PPM)
A1	985.3	66.4	1235.8	138.1	158.3	130.5	<0.1	1482.1
A2	754.6	58.2	1164.9	114.5	144.8	117.2	<0.1	1240.6
A3	682.4	52.3	981.5	98.2	150.9	85.1	0.1	982.4
A4	385.1	45.9	908.4	86.1	132.4	73.6	<0.1	851.7
A5	257.6	38.2	784.3	64.8	120.9	52.8	<0.1	768.1
A6	207.8	26.1	562.9	51.9	112.8	21.9	<0.1	687.1
A7	199.4	23.8	325.7	52.8	113.1	19.4	<0.1	448.6
A8	175.3	24	149.7	35.6	103.5	15.9	<0.1	253.1
A9	166.2	19.2	86.2	44.8	84.9	13.7	<0.1	116.7
A10	143.7	14.6	57.9	41.5	62.3	8.3	<0.1	84.5
A11	53.4	6.4	28.1	23.5	48.6	6.4	<0.1	56.3

Dispersion of heavy metals along eastern part of the tailings dam

Sample ID	Cr (PPM)	Co (PPM)	Ni (PPM)	Cu (PPM)	Zn (PPM)	As (PPM)	Cd (PPM)	Pb (PPM)
B1	1206.7	75.1	1136.2	87.2	128.3	98.5	0.1	834.7
B2	1006.4	62.9	1017.5	56.8	106.7	83.4	<0.1	735.1
B3	984.9	54.5	946.6	31.9	95.3	74.8	<0.1	696.6
B4	428.7	11.2	619.6	116.5	69.5	71.3	0.1	664.1
B5	819.6	20.7	1475.2	89.2	129.9	78.3	<0.1	729.9
B6	1157.8	25.3	1873.4	63.4	156.1	198.5	<0.1	1849.4
B7	1712.2	24.7	3060.6	128.9	160.5	136.2	<0.1	1269.1

© University of Venda

B8	850.1	27.4	1376.0	204.70	152	649	0.1	6046.50
B9	379.70	23.10	530.40	87.60	50.90	227.20	<0.1	2117.10
B10	553.90	23.50	710.10	128.70	86.00	47.20	<0.1	439.80
B11	359.10	17.20	285.70	115.20	78.50	21.90	<0.1	187.60

Dispersion of heavy metals along southern part of the tailings dam

Sample ID	Cr (PPM)	Co (PPM)	Ni (PPM)	Cu (PPM)	Zn (PPM)	As (PPM)	Cd (PPM)	Pb (PPM)
C1	784	24.21	1054.8	145.6	167.3	195.6	<0.1	1822.6
C2	506.7	20.4	874	132.1	121.6	72.9	<0.1	679.2
C3	497.8	22.7	727.8	123.6	111.5	62	<0.1	577.9
C4	484.7	21.6	690.8	116.7	109.7	45.9	0.1	427.7
C5	455.4	18.8	668.6	115	81.9	21.5	<0.1	200.6
C6	427.5	16.9	593.5	106.1	85.5	14.3	<0.1	133

Dispersion of heavy metals along western part of the tailings dam

Sample ID	Cr (PPM)	Co (PPM)	Ni (PPM)	Cu (PPM)	Zn (PPM)	As (PPM)	Cd (PPM)	Pb (PPM)
D1	386.9	22.9	452.5	291.8	165.6	55.9	<0.1	520.6
D2	316.1	19.8	512.8	123.1	144.5	44.9	<0.1	418.4
D3	259.2	27.8	506.7	121.7	143.1	29.7	<0.1	308
D4	266.3	27.6	483.6	108.1	132.5	33.1	<0.1	277.1
D5	213.3	25.3	280	89.4	128.9	22.1	<0.1	180.6
D6	133.8	24.3	158	83.2	125.9	19.4	<0.1	206.2
D7	129.5	20.5	102.2	81.6	118.9	12.5	0.1	116.1
D8	105.6	18.3	62.1	57.6	117	10.6	<0.1	99.1

© University of Venda

D9	93.1	16.9	68.5	51.2	73.9	8.5	<0.1	79
D10	78	16.5	86.1	45	72.6	8.3	<0.1	77.4
D11	27	5.7	23.1	14.6	14.3	4.2	<0.1	38.4

© University of Venda

Appendix E

Sieve analysis results

Sieve analysis results of sample 1

Sieve (mm)	Mass of empty sieve (g)	Mass of sieve + Soil (g)	Mass of soil (g)	Percentage retained (%)	Cumulative retained (%)	Percentage passing (%)
40	772.35	1032.66	260.31	5.27	5.27	94.73
31.5	774.1	1231.10	487.00	9.86	15.14	84.86
20	757.97	1132.38	374.41	7.58	22.72	77.27
10	626.11	1474.18	848.07	17.18	39.91	60.09
5	582.10	1158.96	576.86	11.68	51.60	48.40
4	388.74	516.09	127.35	2.58	54.18	45.82
3.15	369.22	583.25	169.03	3.42	57.60	42.39
2	327.03	596.05	269.02	5.45	63.05	36.94
1	293.90	625.69	331.79	6.72	69.78	30.22
0.425	293.14	540.48	247.34	5.01	74.79	25.21
0.25	242.95	475.90	232.95	4.72	79.51	20.49
0.15	258.21	538.60	280.39	5.68	85.19	14.80
0.075	263.42	559.40	295.98	5.99	91.19	8.81
Pan	530.38	965.00	434.62	8.80	100	0.00

Sieve analysis results of sample 2

Sieve (mm)	Mass of empty sieve (g)	Mass of sieve + Soil (g)	Mass of soil (g)	Percentage retained (%)	Cumulative retained (%)	Percentage passing (%)
50	1059.70	1771.40	7117.70	11.70	11.70	88.30
40	772.35	2667.45	1895.10	31.15	42.85	57.14
31.5	744.10	1213.31	469.21	7.71	50.57	49.42
20	757.97	380.57	377.40	6.20	44.36	55.63
10	626.11	1353.81	727.70	11.96	56.33	43.66
5	582.1	1043.13	46.03	7.58	63.91	36.08
4	388.74	505.91	117.17	1.92	65.64	34.15
3.15	369.22	500.89	13.67	2.16	68.00	31.99
2	327.03	531.65	204.62	3.36	71.36	28.63
1	293.90	552.96	259.06	4.25	75.62	24.37
0.425	293.14	483.08	189.94	3.12	78.75	21.24
0.25	242.95	531.7.	288.78	4.74	83.49	16.50
0.15	258.21	501.48	243.27	3.99	87.49	12.50
0.075	263.42	620.26	356.84	5.86	93.36	6.63
Pan	530.38	933.87	403.49	6.63	100	0

© University of Venda

Sieve analysis results of sample 3

Sieve (mm)	Mass of empty sieve (g)	Mass of sieve + Soil (g)	Mass of soil (g)	Percentage retained (%)	Cumulative retained (%)	Percentage passing (%)
50	1059.70	1506.84	447.14	8.47	8.47	91.53
40	772.35	1188.55	416.20	7.88	16.36	83.64
31.5	744.10	1722.31	978.21	18.53	34.89	65.11
20	757.97	1374.05	616.08	11.67	46.57	53.43
10	626.11	1234.66	608.55	11.53	58.10	41.9
5	582.10	1018.12	436.02	8.26	66.36	33.64
4	388.74	496.83	108.09	2.04	68.41	31.59
3.15	369.22	482.30	113.08	2.14	70.55	29.45
2	327.03	506.72	179.69	3.40	73.96	26.04
1	293.90	519.86	225.96	4.28	78.24	21.76
0.425	239.14	471.85	178.71	3.38	81.63	18.37
0.25	242.95	417.53	174.58	3.30	84.94	15.06
0.15	258.21	458.48	200.27	3.79	88.73	11.27
0.075	263.42	487.71	224.29	4.25	92.98	7.02
Pan	530.38	900.49	370.11	7.01	100	0

Sieve analysis results of sample 4

Sieve (mm)	Mass of empty sieve (g)	Mass of sieve + Soil (g)	Mass of soil (g)	Percentage retained (%)	Cumulative retained	Percentage passing (%)
50	1059.70	1931.55	871.85	14.88	14.88	85.12
40	772.35	997.69	225.34	3.84	18.72	81.27
31.5	744.10	1173.18	429.08	7.32	26.04	73.96
20	757.97	1645.67	887.7	15.15	41.18	58.82
10	626.11	1620.54	994.43	16.97	58.15	41.85
5	582.10	1198.06	615.96	10.51	68.66	31.34
4	388.74	527.87	139.13	2.37	71.04	28.96
3.15	369.22	516.30	147.08	2.51	73.55	26.45
2	327.03	544.12	217.09	3.70	77.25	22.75
1	293.90	551.07	257.17	4.39	81.64	18.36
0.425	239.14	472.57	233.43	3.98	84.7	15.30
0.25	242.95	401.98	159.03	2.71	87.41	12.59
0.15	258.21	409.01	150.8	2.57	89.99	10.01
0.075	263.42	466.97	203.55	3.47	93.46	6.54
Pan	530.38	913.35	382.97	6.53	100	0

© University of Venda

Sieve analysis results of sample 5

Sieve (mm)	Mass of empty sieve (g)	Mass of sieve + Soil (g)	Mass of soil (g)	Percentage retained (%)	Cumulative retained (%)	Percentage passing (%)
50	1059.70	1636.48	576.78	11.77	11.76	88.24
40	772.35	978.72	206.37	4.21	15.98	84.02
31.5	744.10	1067.06	322.96	6.59	22.57	77.43
20	757.97	1380.57	622.6	12.70	35.27	64.73
10	626.11	1160.87	534.76	10.91	46.18	53.82
5	582.10	1116.66	534.56	10.91	57.09	42.91
4	388.74	527.69	138.95	2.84	59.93	40.07
3.15	369.22	514.29	145.07	2.96	62.89	37.11
2	327.03	557	229.97	4.69	67.58	32.42
1	293.90	587.52	293.62	5.99	73.57	26.43
0.425	239.14	513.55	274.41	5.60	78.07	21.93
0.25	242.95	455.23	212.28	4.33	82.4	17.6
0.15	258.21	484.21	226	4.61	87.01	12.99
0.075	263.42	598	334.58	6.83	93.84	6.16
Pan	530.38	832.1	301.72	6.16	100	0

Sieve analysis results of sample 6

Sieve (mm)	Mass of empty sieve (g)	Mass of sieve + Soil (g)	Mass os soil (g)	Percentage retained (%)	Cumulative retained	Percentage passing (%)
50	1059.70	1520.06	460.36	8.66	8.66	91.34
40	772.35	1174.42	402.07	7.56	16.23	83.77
31.5	744.10	1366.52	622.42	11.71	27.94	72.06
20	757.97	1395.52	637.55	12.00	39.94	60.06
10	626.11	1266.3	640.19	12.04	51.99	48.01
5	582.10	1073.32	491.22	9.24	61.24	38.76
4	388.74	507.14	118.4	2.22	63.47	36.53
3.15	369.22	493.1	123.88	2.33	65.8	34.2
2	327.03	513.78	186.75	3.51	69.31	30.69
1	293.90	561.63	267.73	5.03	74.35	25.65
0.425	239.14	513.19	274.05	5.15	78.5	21.5
0.25	242.95	448.28	205.33	3.86	82.36	17.64
0.15	258.21	484.96	226.75	4.26	66.63	33.37
0.075	263.42	585.57	322.15	6.06	92.69	7.31
Pan	530.38	918.37	387.99	7.30	100	0

© University of Venda

Appendix F

Atterberg limit results

F.1 Liquid limit results

Liquid limit results for Sample 1

	Trial 1	Trial 2	Trial 3	Average
Mass of the can, W1 (g)	18.04	18.07	18.01	18.04
Mass of can + moist soil, W2 (g)	28.95	26.05	30.94	28.65
Mass of can + dry soil, W3 (g)	26.33	24.37	28.32	26.34
Mass of soil solids (g)	8.29	6.3	10.31	8.30
Mass of pore water (g)	2.52	1.68	2.62	2.27
Moisture content (%)	30.4	26.67	25.41	27.49
Number of blows	14	31	70	38

Liquid limit results for Sample 2

	Trial 1	Trial 2	Trial 3	Average
Mass of the can, W1 (g)	18	18.04	17.98	18.01
Mass of can + moist soil, W2 (g)	32.23	36.74	28.41	32.46
Mass of can + dry soil, W3 (g)	29.06	33.01	26.51	29.53
Mass of soil solids (g)	11.06	14.97	8.53	11.52
Mass of pore water (g)	3.17	3.73	1.9	2.93
Moisture content (%)	38.78	32.35	23.39	31.51
Number of blows	11	26	54	30

© University of Venda

Liquid limit results for Sample 3

	Trial 1	Trial 2	Trial 3	Average
Mass of the can, W1 (g)	18.07	18.02	17.98	18.02
Mass of can + moist soil, W2 (g)	35.48	31.35	31.76	32.86
Mass of can + dry soil, W3 (g)	30.82	28.37	29.09	29.43
Mass of soil solids (g)	12.75	10.35	11.11	11.40
Mass of pore water (g)	4.68	2.98	2.67	3.44
Moisture content (%)	36.55	28.79	24.03	29.79
Number of blows	12	26	63	34

Liquid limit results for Sample 4

	Trial 1	Trial 2	Trial 3	Average
Mass of the can, W1 (g)	17.96	19.09	17.94	18.33
Mass of can + moist soil, W2 (g)	31.08	29.44	31.26	30.59
Mass of can + dry soil, W3 (g)	27.94	26.98	28.8	27.91
Mass of soil solids (g)	9.98	7.89	10.86	9.58
Mass of pore water (g)	3.14	2.46	2.46	2.69
Moisture content (%)	31.46	31.19	22.21	28.29
Number of blows	15	26	49	30

© University of Venda

Liquid limit results for Sample 5

	Trial 1	Trial 2	Trial 3	Average
Mass of the can, W1 (g)	18.1	17.94	18.06	18.03
Mass of can + moist soil, W2 (g)	28.78	28.55	31.81	29.71
Mass of can + dry soil, W3 (g)	26.25	25.25	28.98	26.83
Mass of soil solids (g)	8.15	7.26	10.9	8.77
Mass of pore water (g)	2.53	3.3	2.83	2.89
Moisture content (%)	31.04	45.45	25.96	34.15
Number of blows	16	31	64	37

Liquid limit results for Sample 6

	Trial 1	Trial 2	Trial 3	Average
Mass of the can, W1 (g)	17.99	18.97	18.08	18.35
Mass of can + moist soil, W2 (g)	31.62	29.52	34.74	31.96
Mass of can + dry soil, W3 (g)	28.24	27.13	21.27	25.55
Mass of soil solids (g)	10.25	8.16	13.09	10.50
Mass of pore water (g)	3.38	2.39	3.47	3.08
Moisture content (%)	32.56	29.29	26.5	29.45
Number of blows	12	24	44	27

© University of Venda

F.2 Plastic limit results

Plastic limit results sample 1

	Trial 1	Trial 2	Trial 3	Average
Mass of the can, W1 (g)	18.09	18.06	18.02	18.06
Mass of can + moist soil, W2 (g)	25.03	25.27	24.97	25.09
Mass of can + dry soil, W3 (g)	23.68	23.74	23.60	23.67
Mass of soil solids (g)	5.59	5.68	5.58	5.62
Mass of pore water (g)	1.35	1.53	1.37	1.42
Moisture content (%)	24.15	26.94	24.55	25.21
Number of blows	14	31	70	38

Plastic limit results sample 2

	Trial 1	Trial 2	Trial 3	Average
Mas of the can, W1 (g)	18.98	19.05	18.06	18.70
Mass of can + moist soil, W2 (g)	21.70	21.75	23.81	22.42
Mass of can + dry soil, W3 (g)	20.94	21.09	22.72	21.58
Mass of soil solids (g)	1.96	2.04	4.66	2.89
Mass of pore water (g)	0.76	0.66	1.09	0.84
Moisture content (%)	38.78	32.35	23.39	31.51
Number of blows	11	26	54	30

© University of Venda

Plastic limit results for sample 3

	Trial 1	Trial 2	Trial 3	Average
Mass of the can, W1 (g)	17.94	17.98	17.97	17.96
Mass of can + moist soil, W2 (g)	22.11	26.40	25.24	24.58
Mass of can + dry soil, W3 (g)	21.31	24.93	23.85	23.36
Mass of soil solids (g)	3.37	6.95	5.88	5.40
Mass of pore water (g)	0.8	1.47	1.39	1.22
Moisture content (%)	23.74	21.15	23.64	22.84
Number of blows	15	26	49	30

Plastic limit results for sample 4

	Trial 1	Trial 2	Trial 3	Average
Mass of the can, W1 (g)	18.26	18.05	17.91	18.07
Mass of can + moist soil, W2 (g)	23.18	23.33	27.73	24.75
Mass of can + dry soil, W3 (g)	21.95	22.25	25.75	23.32
Mass of soil solids (g)	3.69	4.4	7.84	5.31
Mass of pore water (g)	1.23	1.08	1.98	1.43
Moisture content (%)	33.33	24.54	25.26	27.71
Number of blows	12	31	64	36

© University of Venda

Plastic limit results for sample 5

	Trial 1	Trial 2	Trail 3	Average
Mass of the can, W1 (g)	18.02	17.99	17.96	17.99
Mass of can + moist soil, W2 (g)	24.20	27.44	24.91	25.52
Mass of can + dry soil, W3 (g)	22.83	25.48	23.53	23.95
Mass of soil solids (g)	4.81	7.49	5.57	5.96
Mass of pore water (g)	1.37	1.96	1.38	1.57
Moisture content (%)	28.48	26.17	24.77	26.47
Number of blows	16	31	64	37

Plastic limit results for sample 6

	Trial 1	Trial 2	Trial 3	Average
Mass of the can, W1 (g)	18.05	17.97	18.02	18.01
Mass of can + moist soil, W2 (g)	23.66	24.15	24.82	24.21
Mass of can + dry soil, W3 (g)	22.50	22.87	23.49	22.95
Mass of soil solids (g)	4.45	4.9	5.47	4.94
Mass of pore water (g)	1.16	1.28	1.33	1.26
Moisture content (%)	26.07	26.122	29.75	27.31
Number of blows	12	24	44	27

© University of Venda

Appendix G

Compaction test results

Compaction test results for sample 1

	Trial 1	Trial 2	Trial 3
Mass of bag (g)	10.23	10.58	10.90
Dry Density (g/cm^3)	10.84	11.06	10.67
Wet soil (g)	5397.88	5512.55	5337.90
Dry soil (g)	5281.82	5381.82	5199.84
Mass of pore water (g)	116.06	130.73	138.06
Moisture content (%)	2.20	2.43	2.66
Actual mass of dry sample	5271.59	5371.24	5188.94

Compaction test results for sample 2

	Trial 1	Trial 2	Trial 3
Mass of bag (g)	10.61	10.32	10.34
Dry Density (g/cm^3)	11.11	10.36	10.54
Wet soil (g)	5560.26	5275.14	5334.52
Dry soil (g)	5411.57	5095.45	5134.90
Mass of pore water (g)	148.69	179.69	199.62
Moisture content (%)	2.75	3.53	3.89
Actual mass of dry sample	5400.96	5085.13	5124.56

© University of Venda

Compaction test results for sample 3

	Trial 1	**Trial 2**	**Trial 3**
Mass of bag (g)	10.21	10.48	10.61
Dry Density (g/cm^3)	10.3	10.61	10.95
Wet soil (g)	5128.64	5301.92	5479.69
Dry soil (g)	5016.56	5170	5334.01
Mass of pore water (g)	112.08	131.92	145.68
Moisture content (%)	2.23	2.55	2.73
Actual mass of dry sample	5006.35	5159.52	5323.40

Compaction test results for sample 4

	Trial 1	**Trial 2**	**Trial 3**
Mass of bag (g)	10.61	10.92	10.01
Dry Density (g/cm^3)	10.46	10.70	11.11
Wet soil (g)	5218.64	5349.01	5564.46
Dry soil (g)	5096.82	5212.07	5412.84
Mass of pore water (g)	121.82	136.94	151.62
Moisture content (%)	2.39	2.63	2.80
Actual mass of dry sample	5086.21	5201.15	5402.83

© University of Venda

Compaction test results for sample 5

	Trial 1	**Trial 2**	**Trial 3**
Mass of bag (g)	10.24	10.36	10.32
Dry Density (g/cm^3)	10.97	11.23	11.40
Wet soil (g)	5461.62	5598.16	5703.51
Dry soil (g)	5343.53	5468.71	5551.45
Mass of pore water (g)	118.09	129.45	152.06
Moisture content (%)	2.21	2.37	2.74
Actual mass of dry sample	5333.29	5458.35	5541.13

Compaction test results for sample 6

	Trial 1	**Trial 2**	**Trial 3**
Mass of bag (g)	10.64	10.38	10.45
Dry Density (g/cm^3)	10.78	11.09	11.31
Wet soil (g)	5381.26	5541.81	5661.32
Dry soil (g)	5252.22	5403.20	5510
Mass of pore water (g)	129.04	138.61	151.32
Moisture content (%)	2.46	2.57	2.75
Actual mass of dry sample	5241.58	5392.82	5499.55

© University of Venda