

Faculty of Engineering and Technology
Department of Mining and Geological Engineering

**GEOTECHNICAL AND GEOCHEMICAL IMPACT ASSESSMENT OF
BAMANGWATO CONCESSIONS LIMITED (BCL) MINE WASTE DUMPS ON
SELEBI-PHIKWE ENVIRONMENT, BOTSWANA**

by

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A thesis submitted to the Faculty of Engineering and Technology in fulfilment of the
requirements for the award of the Degree of Master of Engineering in Geological
Engineering of BIUST.

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ABSTRACT

Mining activities produce significant amounts of wastes (tailings, slag and waste rock) which, with loss in material strength, could trigger slope instability while the wastes minerals' chemical reactions also have the potential to produce acid mine drainage (AMD) contaminations. Plausible environmental impacts that could be caused by Bamangwato Concessions Limited (BCL) mine waste dumps was investigated through geotechnical, geochemical, and mineralogical studies. The mine wastes geotechnical properties were determined through laboratory testing and used to establish safe slope angles of the waste dumps based on the Simplified Bishop method using Rocscience Slide2 software Version 9.007. The geochemical and mineralogical studies to determine the influence of the mine wastes on the environment were through X-ray diffraction (XRD) and Inductively Coupled Plasma Optical Emission Spectrometry (ICP-OES) on samples of tailings, slag, and soil from the mine, while Inductively Coupled Plasma Mass Spectrometry (ICP-MS) was applied to water samples. Safe slope angles up to 20° were established for the tailings dam at slope heights of 40-60 m and water levels from 30-40 m height. The slag dump safe slope angle was 30° for water levels up to 20 m at slope heights of 40-60 m. The waste rock dump slopes were stable for slope heights up to 60 m and slope angles rising to 40°. The heavy metals in the tailings in order of concentrations (in ppm) was Iron(80000)>Nickel(1805.3)>Copper(1388.4)>Manganese(373.8)>Chromium(138.7)>Cobalt(97.3)>Zinc(53)>Lead(14). Statistical analysis suggested miscellaneous sources of pollution. Continuous monitoring of heavy metals concentrations around the waste dumps has been recommended.

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LIST OF ABBREVIATIONS



AMD:	Acid Mine Drainage
ARD:	Acid Rock Drainage
BCL:	Bamangwato Concessions Limited
BOF:	Blast Oxygen Furnace
CF:	Contamination Factor
C&M:	Care and Maintenance
DC:	Degree of Contamination
EC:	Electrical Conductivity
EF:	Enrichment Factor
FS:	Factor of Safety
GDP:	Gross Domestic Products
GPS:	Global Positioning system
ICP-AES:	Inductively Coupled Plasma Atomic Emission Spectroscopy
ICP-MS:	Inductively Coupled Plasma Mass Spectrometry
ICP-OES:	Inductively Coupled Plasma Optical Emission Spectrometry
I-geo:	Geo-accumulation Index
PLI:	Pollution Load Index
PSD:	Particle Size Distribution
PTE:	Potentially Toxic Element
RTZ:	Roan Selection Trust
SEM:	Scanning Electron Microscopy
TDS:	Total Dissolved Solid
TSF:	Tailings Storage Facility
US EPA:	United System Environmental Protection Agency
XRD:	X-ray Diffraction
XRF:	X-ray Fluorescence

CHAPTER 1

GENERAL INTRODUCTION

1.1 Background

The operation of a mine usually results in generation of large quantities of materials from which the commodity of interest is extracted. The excess material generated during the extraction, beneficiation, and processing of minerals is referred to as mine waste which may be dumped as heaps in the case of the solid and in ponds if it is in semi solid state (wet slurry). Some mine waste based on mineralization, could be reactive. In addition to stability of the waste dumps, crushing of the rock to release the mineralization may result in dispersion of the reactive minerals in the air and or water, which could be detrimental to the mining and surrounding environment. To manage the situation, it becomes necessary for mining companies to put in place measures that would contain or minimize the dispersion through effective management of the overburden materials, waste rock piles and tailings.

This research was carried out on Bamangwato Concessions Limited (BCL) mine, located 5 km NE of Selebi-Phikwe, a town in the Central District of Botswana (Figure 1.1) with 49,411 inhabitants based on Statistics Botswana (2015) data. The Selebi-Phikwe deposits, which consist of three orebodies, are located in the northern margin of the Central Zone of the Limpopo Belt in Southern Africa. They are hosted by amphibolite, principally composed of hornblende, feldspar, gedrite, and mica (Maier *et al.*, 2007) occurring within a sequence of grey quartzo-feldspathic and hornblende gneisses. The three orebodies are discriminated based on the sulfide contents in the host rock. The types of mineralization include massive, semi-massive, and disseminated mineralization (Nkoma and Ekosse, 2009).

BCL mining operations started in 1973 with an open pit followed by underground excavations composed of different shafts. With a concentrator and a flash furnace smelter, BCL mine excavated and smelted mixed copper-nickel ore with the final product - a matte containing Nickel, Copper and Cobalt which was sent to two refineries namely, Falconbridge in Norway and RTZ in Zimbabwe (Mikesell, 1984; Gwisai, 2009) due to the lack of a refinery at BCL mine. The waste products from the excavated, crushed and smelted

ore from the mine include the waste rock, tailings and slag which form the BCL mine waste dumps.

The climate in Selebi-Phikwe is semi-arid with savanna-type vegetation. There are two types of soils in the area described as eutric regosols and haplic luvisols with main minerals including quartz, mica, feldspars, and secondary minerals such as kaolinite, illite, and smectite (Ekosse, 2005).

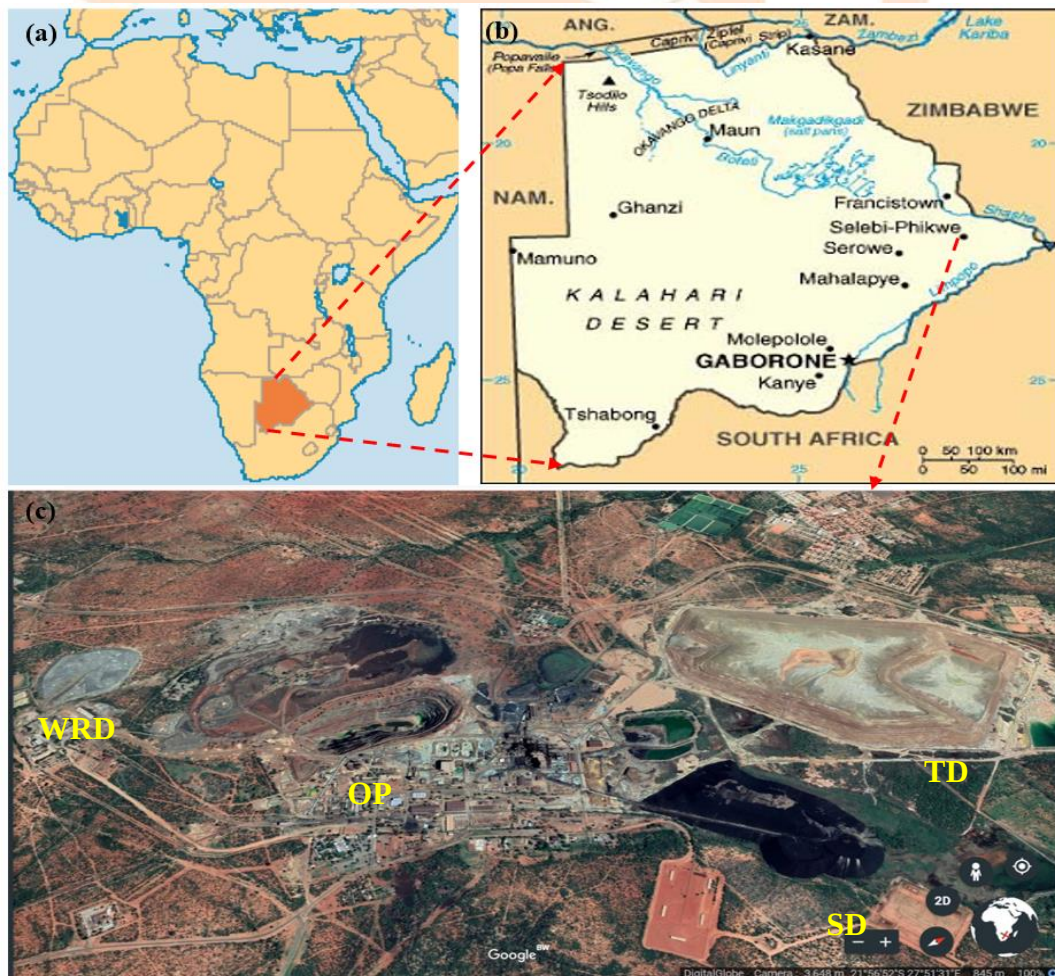


Figure 1.1 Geographical Location of BCL Mine (a) Map showing Botswana within Africa, (b) Map showing the Location of Selebi-Phikwe within Botswana (c) Map of BCL Mine within Selebi-Phikwe from Google Earth with WRD: Waste Rock Dump; TD: Tailings Dam; OP: Open Pit; and SD: Slag Dump

1.2 Problem Statement

BCL mine is currently in liquidation since 2016 and at the date of closure, BCL had a workforce of about 5,486 workers (BW government, 2017). The closure has not only

brought about a huge social impact on the inhabitants of the Selebi-Phikwe area but also, has triggered environmental concerns that need consideration. There is the possibility of environmental devastation together with huge property damages as a consequence of inadequate maintenance of unsecured waste dumps. The closure of the mine would likely impact the ability of the mine to perform some secured obligations due to a lack of funds. Some of such obligations include ensuring that the waste dumps are inert, or at least stable and contained to minimize their surface footprints. Any potential slope failures and heavy metals mobility can severely contaminate soils, surface, and groundwater, and endanger both local and downstream ecosystems with the alteration of the surface footprints of the waste dumps in the Selebi-Phikwe area. Hence the actual situation of BCL mine waste dumps requires geotechnical and geochemical investigations. This research was therefore necessitated to examine the stability and the environmental impacts of the waste dumps on the Selebi-Phikwe area.

1.3 Aim and Objectives of Research

The research was aimed at assessing the plausible effects of the mine waste dumps on the Selebi-Phikwe environment.

The objectives were to:

- obtain the geotechnical properties and assess the slope stability of the mine waste dumps;
- determine the geochemical and mineralogical compositions of the mine wastes, including soils within the mine vicinity, tailings seepage water, and treated water within the mine;
- assess the plausible pollution in the vicinity of the mine.

1.4 Importance of the Study

The results of this investigation were to bridge the critical gap on the stability of the mine waste dumps of BCL mine and provide records on the level of contamination of the water flowing within the mine, surrounding soil, tailings, and slag. The uniqueness of the research is the combination of geotechnical and geochemical studies. The findings are markers for future research in the study area.

1.5 Thesis Outline

This thesis contains 7 chapters including:

- Chapter 1 which is an introductory chapter contains the background, statement of the problem; the aim, and the objectives of the research with the outcomes of the research;
- Chapter 2 describes the various mine wastes of interest in this research and a résumé of previous investigations on methods used in assessing slope stability from the literature, then a review of geotechnical and mineralogical characterization of the dump material, and finally the effect of heavy metals in the mine waste;
- Chapter 3 deals with the methodology; the chapter provides the approach applied for the geotechnical parameters determination, the modeling for conducting the slope stability analyses, and the methods used for pollution indices determination;
- Chapters 4 and 5 provide the results obtained from the petrographic studies, geotechnical tests, slope stability analysis, mineralogical and geochemical laboratory tests;
- Chapter 6 discusses the results obtained;
- Chapter 7 presents a summary of the main findings and possible directions for further work.

CHAPTER 2

LITERATURE REVIEW

2 INTRODUCTION

This chapter summarizes the key aspects of the research which include the types of mine wastes of interest, the shear strength testing and parameters, and the influence of the mineralogy and heavy metals on the geochemical behavior of mine wastes.

2.1 Definition of Mine Wastes

Mine wastes are the useless materials obtained from the exploration stage, the mine facilities installation, the excavation, and the beneficiation process whether it is an underground mine or an open pit mine interested in metalliferous and non-metalliferous minerals (Twardowska and Szczepańska, 2004; Monjezi, 2008). Mine wastes have also been defined by Hudson-Edwards *et al.* (2011) as heterogeneous and undesirable substances which can be the ore, gangue, loose sediment, mill tailings, metallurgical slag, and wastewater. Figure 2.1 shows the common mine wastes (waste rock, tailings, and slag), which are discussed in the present study, and the stages at which they are produced during any mining process.

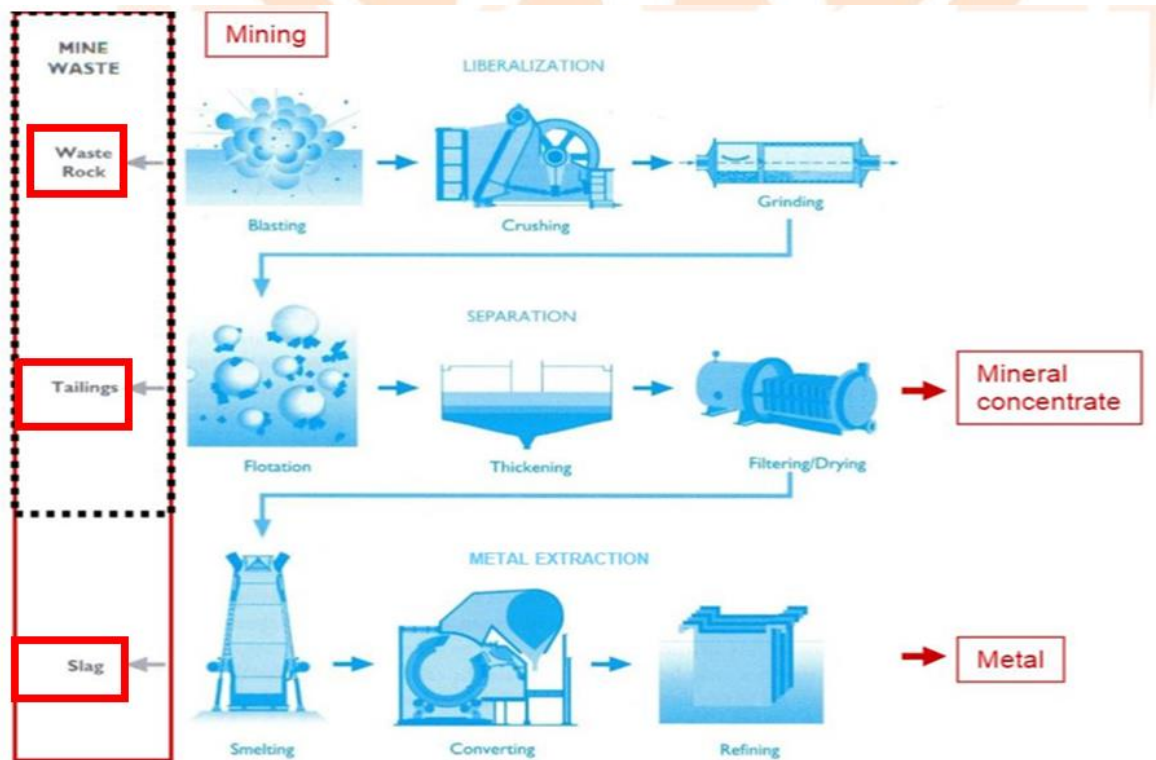


Figure 2.1 Type of mine wastes (Spitz and Trudinger, 2009)

The quantity of mine wastes remains unknown worldwide (Blight, 2011) and becomes larger due to the increasing demand for essential metals and the low concentrations of the mineral of interest in the economic ore deposit (van Zyl *et al.*, 2002). For example, in 1995, mine wastes produced from only iron, gold, lead, and bauxite (aluminum) mining amounted to 35×10^9 t globally (van Zyl *et al.*, 2002). With regards to the increasing quantity of mine wastes, it is important to have a clear definition and plausible disposal sites for proper management.

2.1.1 Mine Waste Rock Dump

Mine waste rock is defined as the less mineralized rock that is removed to reach the ore, it is obtained after the application of blasting and drilling methods and usually deposited in piles or heaps at sites called dumps (Hartman and Mutmanský, 2002; McLemore *et al.*, 2009; Blight, 2010; Solimano, 2015). Its quantity depends on the geometry and the location of the orebody and is mostly located near to the mine site to lessen its transporting cost. Waste rock is usually used for mine backfill.

The dumping site is chosen in a way that it can receive a substantial amount of waste material and its shape depends on the topography of the host site (Zahl *et al.*, 1992). There are different types of pile configuration as listed by Zahl *et al.* (1992); Aubertin *et al.* (2002); Fala (2002); Tachie-Menson (2006); Fala (2008); and these include: Valley fill pile (Figure 2.2a); Cross-valley fill pile (Figure 2.2b); Sidehill fill pile (Figure 2.2c); Ridge crest pile (Figure 2.2d), and Heaped pile (Figure 2.2e). Rock piles can be dumped by four different categories, namely end dumping, push dumping, free dumping, and the drag line spoiling (Nichols, 1987; Morin *et al.*, 1991; Quine, 1993; Herasymuik, 1996; Shum, 1999; and Tran *et al.*, 2003).

Waste rock pile failures have already been observed for instance in Gold Quarry Mine, Nevada (Sheets and Bates, 2008), and plausible unreported cases forcing geological engineers to understand the behavior of the rocks.

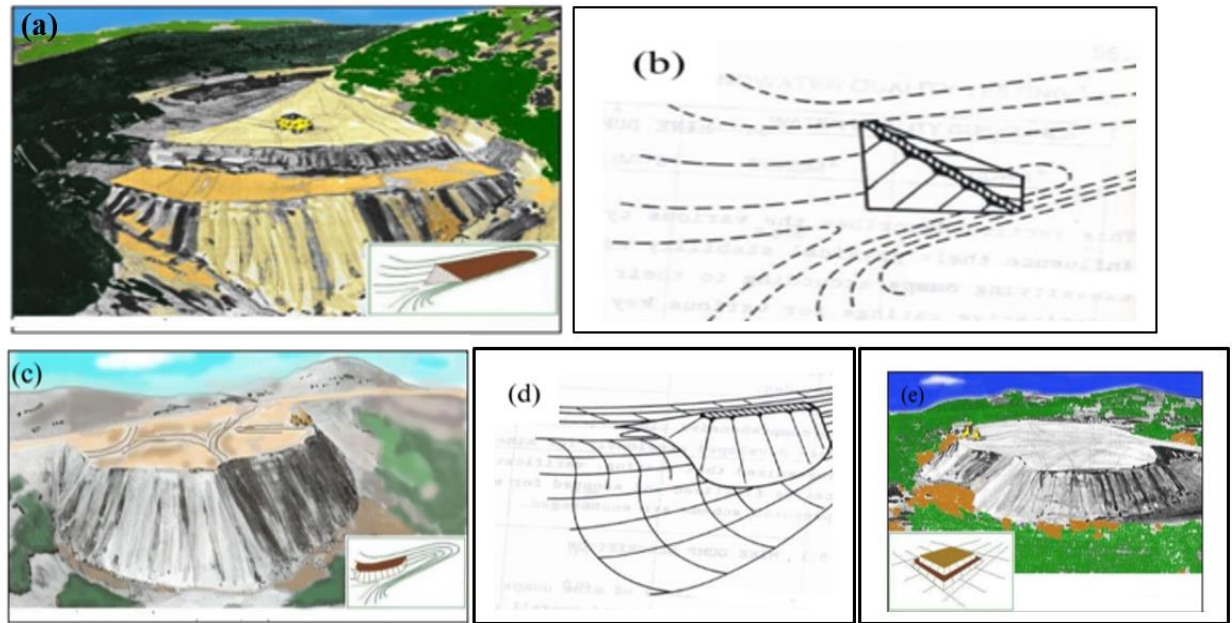


Figure 2.2 Different Rock Piles Configuration Types, (a) Valley Fill Pile; (b) Cross-valley Fill; (c) Sidehill Fill Pile; (d) Ridge Crest Pile; (e) Heaped Pile. (a), (c) and (e) adapted from McCarter (1990), (b) taken from Wahler (1979), (d) taken from Aubertin *et al.* (2002)

2.1.2 Tailings dam

The less mineralized waste collected during the concentration process while extracting the valuable product from the crushed and ground ore after the mining and milling operations is called tailings (Kossof *et al.*, 2014). Tailings, therefore, consists of the unrecoverable and uneconomic of all metals, chemicals, and minerals obtained as slurry forwarded through pipelines from the plant to the final storage site called Tailings Storage Facility (TSF) or dam (Blight, 2010) or engineered structure. The TSF is made by the tailings itself (US EPA, 1994; Yin *et al.*, 2011) or using borrowed materials, and harvested tailings (Tailings Management, 2016). The tailings percentage from the ore mined is about 60% for iron, 99% for gold, and up to 99% for copper (Vogt, 2013; Ramirez-Llodra *et al.*, 2015). Blight (2011) reported that from 1997 to 2006, the mining industry in South Africa produced 7.5×10^5 t of gold tailings.

There are two main sites selected for tailings deposition; being whether subaqueous (sea deposition) which is less used nowadays or subaerial (onshore) (Dold, 2006). The material while pouring it out can be semi-dry, thickened (Ritcey, 1989), or wet (Dold, 2010), the latter being the most common. During the tailings deposition, segregation occurs naturally

allowing the coarser-grained to settle first at the discharging point and then the finer-grained quite away following the water flow.

The methods of dumping include the downstream, the centerline, and the upstream methods, largely described by US EPA (1994) and summarized in Table 2.1. The downstream method is more stable with the accountability of the phreatic surface measurement and also expensive due to the important volume of fill material used to raise the dam. The upstream method, on the other hand, remains more economic where the beach can be controlled but has a poor seismic resistance with poor water-storage suitability. The centerline method reduces the weak points of the downstream and the upstream. The upstream remains the most used and dangerous method mentioned in the literature. For example, USCOLD (1994) and Yin *et al.* (2011) reported that the upstream method is linked to so many failures. Yin *et al.* (2011) made a summary (as shown in Table 2.2) of the consequences in China of the upstream method failures. Davies *et al.* (2000) noted that from more than 3,500 tailings dams in the world, 50% have been created using the upstream method and most of them have failed.

Table 2.1 Tailings Dumping Methods and Characteristics

Characteristics	Upstream	Downstream	Centerline
Mill tailings requirements	At least 60% sand in whole tailings. Low pulp density for grain-size segregation.	Suitable for any type of tailings	Sands or low plasticity slimes
Discharge requirements	Peripheral discharge, well-controlled beach necessary	Varies according to design detail	Peripheral discharge of at least nominal beach necessary
Water-storage suitability	Not suitable for significant water storage	Good	Not recommended for permanent storage. Temporary flood storage can be designed.
Seismic resistance	Poor in high seismic areas	Good	Acceptable
Raising rate restrictions	Less than 15-30 ft./yr most desirable. Over 50 ft./yr can be hazardous	None	Height restrictions for individuals raises may apply
Embankment fill requirements	Native soil, sand tailings, waste rock	Sand tailings, waste rock, native soils	Sand tailings, waste rock, native soil
Relative embankment cost	Low	High	Moderate
Use of low permeability cores	Not possible	Possible (inclined cone)	Possible (Central cone)

(Source: EPA, 1994)

Table 2.2 Examples of Upstream Method Failures

Name of dam	Type of Tailings	Year of Failure	Consequences due to Failure	References
Huogudu, Yunnan Tin Group Co., Yunnan	Tin	1962	171 fatalities	Yin <i>et al.</i> (2011)
Niujiaolong, Shizhuyuan Non-ferrous Metals Co., Hunan	Copper	1985	49 fatalities	Yin <i>et al.</i> (2011)
Longjiaoshan, Daye Iron Ore mine, Hubei	Iron	1994	31 fatalities	Yin <i>et al.</i> (2011)
Dachang, Nandan Tin mine, Guangxi	Tin	2000	28 fatalities	Yin <i>et al.</i> (2011)
Zhenan Gold mine, Shanxi	Gold	2006	17 fatalities	Yin <i>et al.</i> (2011)
Brazil	Iron	2019	233 fatalities	Palú and Julien (2019)

2.1.3 Slag Dump

Slag is considered as a waste collected during the pyro-metallurgical process (Ettler *et al.*, 2009; Muhlare and Groot, 2015; Xian-Li *et al.*, 2015) that can cool slowly via air-cooling technique by direct deposal in a molten form into the dump (Muhlare and Groot, 2015). It can also be cooled fast using the water granulation technique (Feng *et al.*, 2019; Kosir *et al.*, 2021) before being dumped. Slag can be ferrous or non-ferrous depending on the type of metal of interest. Slag from metals like copper, nickel, lead is considered non-ferrous and those formed from steel, titanium are classified as ferrous slag (Criado *et al.*, 2017). The slag quantity presents a land issue (Gorai *et al.*, 2003), for instance, copper slag production is more than twice the quantity of the copper metal recovered (Chew and Bharati, 2009, Fan *et al.*, 2014, Xian-Lin *et al.*, 2015). In Africa, copper slag production is estimated to be 1.23 million tons per annum (Gorai *et al.*, 2003). To reduce the mine waste dumps slope failure cases and anticipate them, qualitative knowledge of the intrinsic properties of the material is mandatory.

2.2 Geotechnical Characterization of Mine Wastes

The determination of the engineering properties of mine wastes must be ascertained for they are inherent when evaluating their disposal technology. These properties differ from site to site and according to the type of the orebody and the processing method used to remove the minerals of interest.

2.2.1 Waste Rock Characteristics

Particle size distribution (PSD)

Waste rock piles are more composed of large boulders, cobbles, and fine-grained materials with the heap height going up to 300 m (McCarter, 1990; Fala, 2002; Martin *et al.*, 2017). Due to its heterogeneity in particle size, it is mostly classified as well graded. When dumping the waste rock, there is segregation due to the gravity, with the bigger size of the material accumulating at the toe of the slope in particular during end dumping of the material (McLemore *et al.*, 2009).

The gradation can be assessed through laboratory analyses or image analysis photographs (McLemore *et al.*, 2009). Figure 2.3 shows the PSD of rocks from waste rock piles from two different mines. Unlike Figure 2.3a where the particles sizes range from gravel to sand for Quine 1 Mine and cobbles to sand for Quine 2 Mine, the particles in Figure 2.3b vary from cobbles to clay size.

Shear strength

The ability for rocks to remain stable from external loading is their strength. The Mohr-Coulomb criterion is mostly used for the failure analysis. The stress and strain relationships are mainly determined through laboratory tests such as the triaxial test and the direct shear test where the inherent parameters, the cohesion, and the internal friction angle, can be determined or through *in-situ* testing. Barton (1973) and Jaeger and Cook (1976) showed that the coarser the rock grains, the higher is the friction angle, with the friction angles for gneiss and granite ranging between 27° - 34° and 34° - 40°, respectively (see Table 2.3).

Table 2.3 Friction Angles for some Typical Rocks

Friction Angle Range (degree)	Rock Types
Low friction (0 - 27)	Schists (high mica content), shale, marl
Medium friction (27 - 34)	Sandstone, siltstone, chalk, gneiss, slate
High friction (34 - 40)	Basalt, granite, limestone, conglomerate

(Sources: Barton, 1973; Jaeger and Cook, 1976)

A fractured rock subjected to weak normal stress gives a low cohesion and a high friction angle, whereas, for high normal stress, a reduction of the friction angle is observed. According to Barton and Choubey (1977), artificial fracture surfaces can also be made on

rock samples to run a direct shear test on smooth diamond sawed surfaces. Such experiments can be done on dry and wet samples from various rocks. They found friction angle ranges between 26° - 29°; 31° - 35° and 32° for dry gneiss, dry coarse-grained granite, and dry amphibolite respectively.

The *in-situ* waste rock density can range between 1,600 kg/m³ and 2,200 kg/m³ (Williams, 2000), and the grains specific gravity ranges between 2.6 and 4.8 and upward (Kesimal *et al.*, 2004; Bussi re, 2007). For a high relative density rock, the shear strength is also higher (Boughton, 1970; Leps, 1970; Marsal, 1973; Tosun *et al.*, 1999).

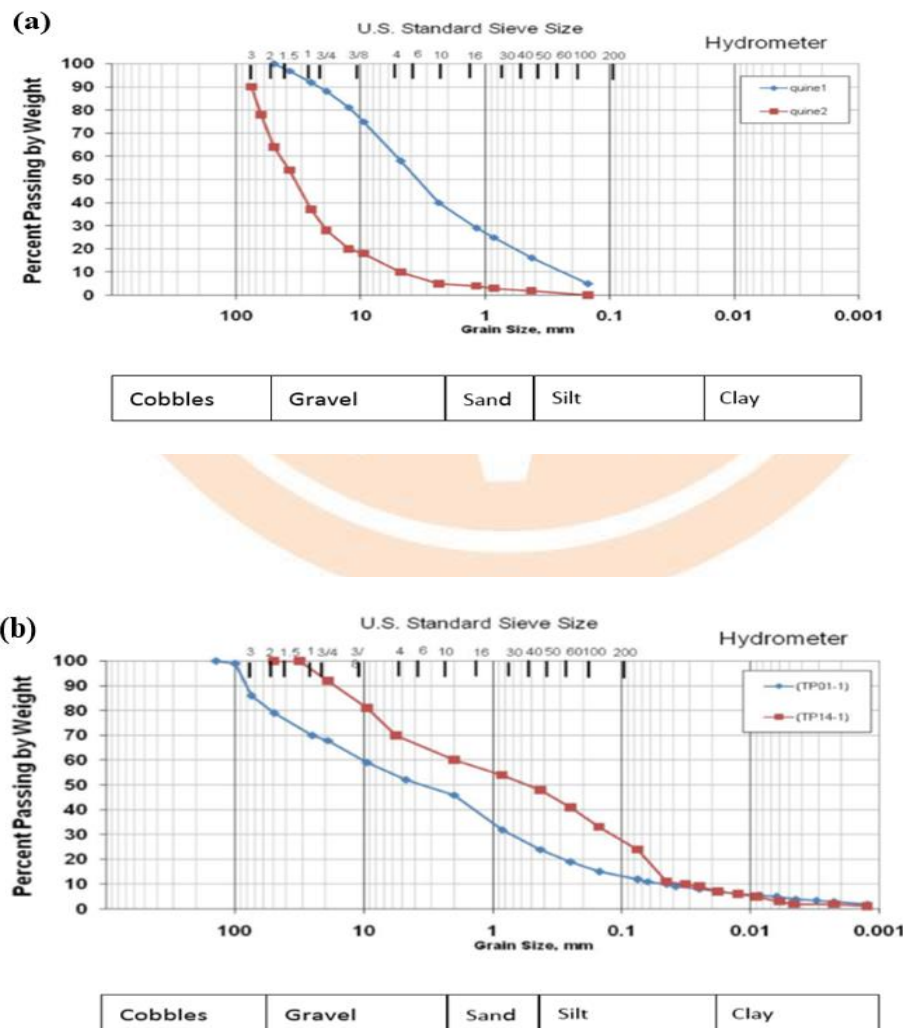


Figure 2.3 Particle Size Distribution of Waste Rock (a) Rock Piles in Canada (Ayres *et al.*, 2005) (b) Waste Rocks in Nevada (Quine, 1993) as cited in McLemore *et al.* (2009)

2.2.2 Tailings Characteristics

Particle size distribution (PSD)

The particle size distribution of tailings is determined by the sieve analysis and sedimentation technique. The particles size of tailings is intimately related to its permeability. The smaller the particle size, the lesser the permeability (Vick, 1990). Hazen (1892) provided a relationship between the effective size of particles, D_{10} , and the coefficient of the permeability (k) as shown in equation 2.1, which according to Mittal and Morgenstern (1975), Mabes *et al.* (1977), Volpe (1979), and Fell *et al.* (1993) was first used for loose sand but now is widely used.

$$K = (D_{10})^2 \quad (2.1)$$

where D_{10} is the particle diameter for which 10% of the sample is smaller.

Robertson (1994) showed that tailings are mainly composed of silt and sand. Following the same line, Aubertin (2013) indicated that tailings material from hard rock mines is mainly silty, initially loose, saturated with low hydraulic conductivity and contains 25% to 45% of solids, which have also been observed by Vick (1990) and James (2009). Thomas *et al.* (2013) and Nitish *et al.* (2013) found that copper tailing is silty in India, whereas copper tailings in a mine in China has been found sandy with fine particles (Yin *et al.*, 2011). Figure 2.4 shows the PSD curves of tailings from two different mines exploiting copper and iron (Hu *et al.* 2017). The authors found that the coarse iron and copper tailings were silty sand and the fine iron and copper tailings were sandy lean clay. More clay size particles in tailings help in the prediction of landslides (Knapen *et al.*, 2006; Wati *et al.*, 2010).

Shear strength

In the construction of a tailings dam, instability can occur due to high pore pressure (Zardari, 2011) resulting in a decrease of its strength. Like the rock strength, the strength of the tailings material is restricted to its cohesion and friction angle. In undrained conditions, the Mohr-Coulomb failure criterion (Equation 2.2) is applied.

$$S = C + \sigma \tan \phi \quad (2.2)$$

where S = shear stress, kPa; C = cohesion, kPa

σ = normal stress, kPa and ϕ = the friction angle, °.

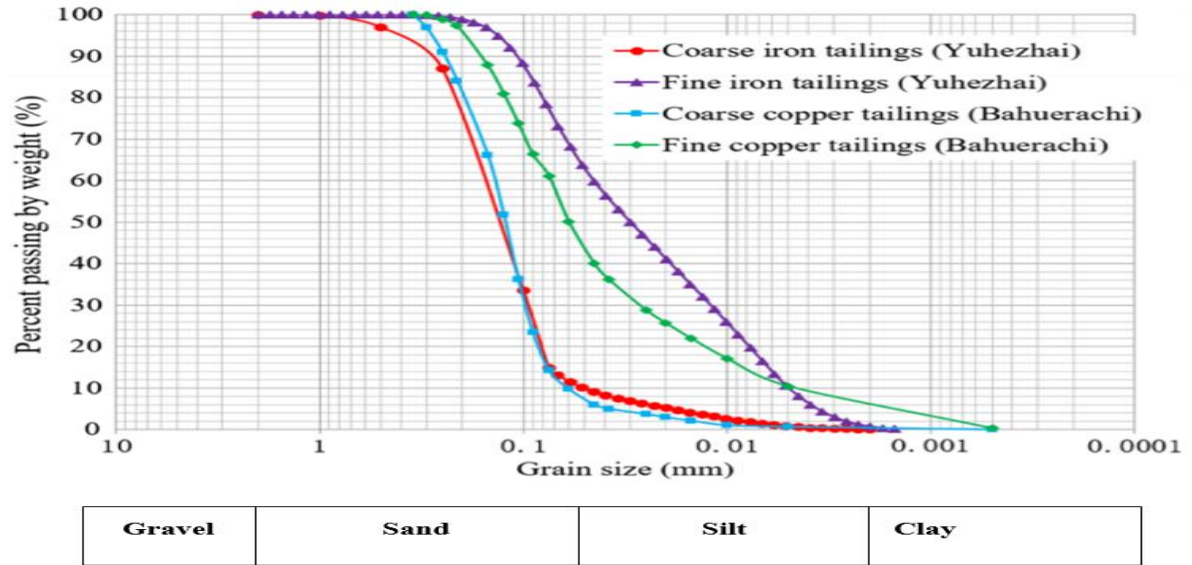


Figure 2.4 Particle Size Distribution of Tailings (Hu *et al.*, 2017)

These parameters are determined by either the direct shear test or the triaxial test. Mittal and Morgenstern (1975) and Vick (1990) described tailings as cohesionless and found friction angle ranges between 30° to 50°. Under drained conditions with a confining pressure of 40 to 300 kPa, the friction angle values found by Bhanbhro (2014) range from 37° to 42°. Under undrained conditions, Vick (1990) noted that tailings friction angle varies from 14° to 24°, whereas Bhanbhro (2014) obtained values between 40° to 43°. Table 2.4 shows a variety of values obtained from different methods that were used to quantify the friction angle and the cohesion of tailings.

Table 2.4 Friction Angle and Cohesion of Tailings from Different Methods

Cohesion (kPa)	Friction Angle (°)	Applied Methods	Sources
5.48 - 6.84	27.54 - 28.70	Triaxial test, UU	(Yin <i>et al.</i> , 2011)
6.46 - 11.09	28.05 - 31.16	Triaxial test, CD	(Yin <i>et al.</i> , 2011)
2.15 - 3.75	24.50 - 28.0	Direct shear test, saturated condition	(Yin <i>et al.</i> , 2011)
1.11	8 - 37	Direct shear test	(Igwe and Chukwu, 2018)
9.61	13 - 18	Consolidated undrained test	(Volpe, 1979)

Notes: UU: Unconsolidated-undrained test; CD: Consolidated-drained test.

The dry density of Swedish tailings of different mines varied between 1270 and 2110 kg/m³ and the bulk density from 1710 to 2420 kg/m³ with a percentage of water content ranging from 12.1 to 39.3% (Bjelkevik and Knutsson, 2005). Tailings dry densities at Jwaneng and Orapa mines in Botswana are respectively 900 and 1000 kg/m³ (Vietti *et al.*, 2008).

2.2.3 Slag Characteristics

Particle size distribution (PSD)

The slag appearance depends on the cooling procedure where an amorphous form is due to rapid cooling, whereas a slow cooling gives a dense crystal-clear shape (Dhir *et al.*, 2017). Yildirim and Prezi (2015) in their study in Indiana for example, showed that steel slag material from basic oxygen furnace (BOF) is well-graded with the particle size ranging from gravel to silt (see Figure 2.5).

Slag aggregates are very angular in shape, black, glassy, and shiny (Zain *et al.*, 2004) with rough surface texture. Some basic properties of copper slag presented in Table 2.5 have been enumerated by Madegowda (2020).

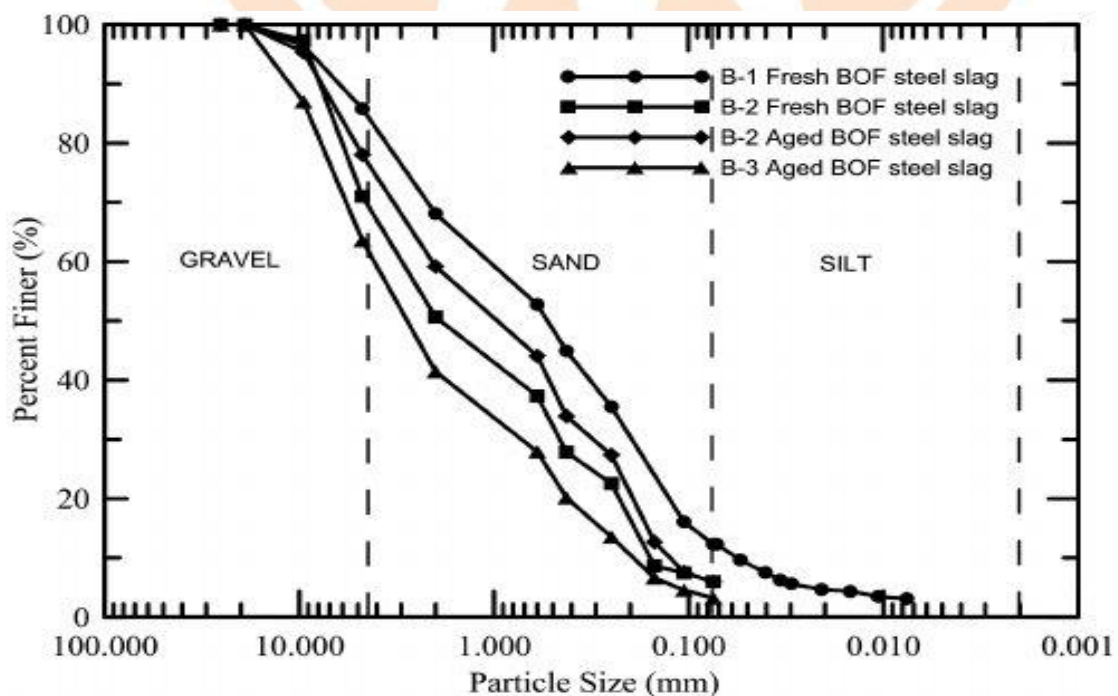


Figure 2.5 Particle Size Distribution of Steel Slag from Indiana Harbor Works West Plant (Yildirim and Prezi, 2015)

Table 2.5 Basic Properties of Copper Slag

Properties	Characteristics
Appearance	Black, glassy, angular, multifaceted
Grain size (mm)	0.2 - 2.4
Specific gravity at 25 °C	3.5
Bulk density (kg/m ³)	2009
Hardness (Moh)	7
pH	7.0
Moisture content (%)	< 0.1
Fineness modulus	2.6
Weight raise on ignition (%)	4.0

(Source: Madegowda, 2020)

2.3 Slope Stability

There are two types of slopes as shown in Figure 2.6, namely, natural slopes like hillsides and valleys (Salunkhe *et al.*, 2017) and man-made slopes like cuttings and embankments (Ishak and Zolkepli, 2016) subjected to many factors leading to instability.

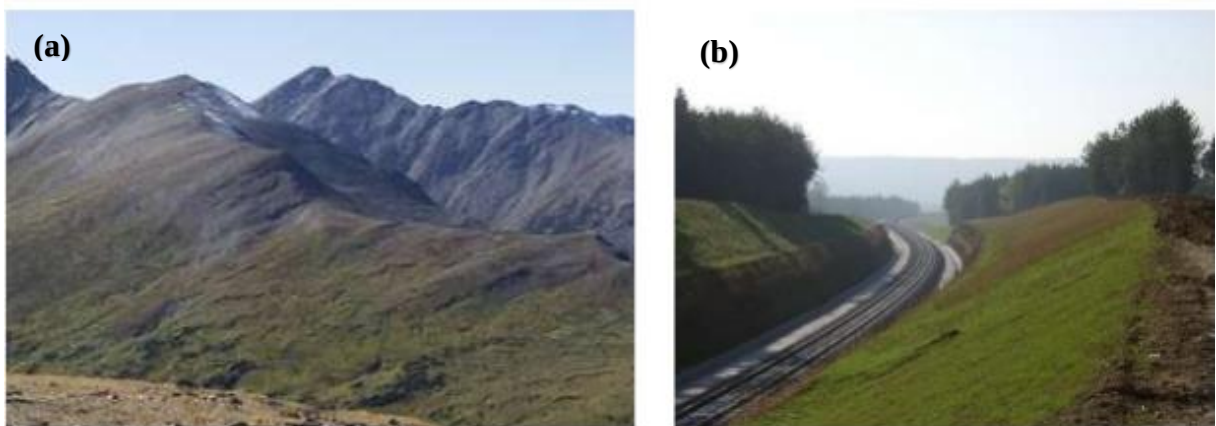


Figure 2.6 Type of Slopes, (a) Natural Slope, (b) Man-Made Slope (Irfan, 2015)

2.3.1 Factors Affecting Slope Stability

The principal factors of slope instability are described to be internal and external (Terzaghi, 1950; Selby, 1982; Varghese, 2012) with the main consequence being the movement of material downward. Gravitational action and seepage (Salunkhe *et al.*, 2017), as well as earthquake and rainfall (especially continuous heavy rains), subject the material to failure by weakening it. Due to erosion, a certain weight of soil is removed from the slope through rain and wind. This increases the height and the slope angle, therefore, resulting in the

decrease of slope stability. All these factors work on either decreasing or increasing respectively the shear strength and/or the shear stress (Abramson *et al.*, 2002).

2.3.2 Slope stability analysis

The assessment of slope stability is done through the determination of the factor of safety using different methods. The factor of safety (FS) is the ratio of the maximum load or stress that a soil can sustain to the actual load or stress that is applied as expressed by equation 2.3. It is simply the ratio of resisting forces to driving forces. FS less than 1 means failure or instability of the structure and FS from 1.25 and more is found safe (Bowles, 1984; Ray and de Smedt, 2009; Cheng and Lau, 2014). The distinction between infinite and finite slope is also critical while determining the stability of slopes.

$$FS = \frac{\text{Resisting Forces}}{\text{Driving Forces}} \quad (2.3)$$

Infinite slope

The sliding mass of an infinite slope is parallel to the slope, occurs on cohesionless soil, and moves on an infinite distance. The factor of safety, in this case, can be determined using the infinite slope stability analysis (Cornforth, 2005; McCarthy, 2007), which has been simplified over the years. Table 2.6 displays the most used equations for the factor of safety depending on the type of soil for man-made slopes (Das, 2013).

Table 2.6 Simplified Factors of Safety for Infinite Slope

Soil Type	Mode	Factor of Safety	Parameters
Cohesionless Soil	Dry Condition	$FS = \frac{\tan\phi'}{\tan\beta}$	ϕ' : drained angle of friction β : actual slope angle
	Presence of Seepage Water	$FS = \frac{\gamma'\tan\phi'}{\gamma_{sat}\tan\beta}$	ϕ' : drained angle of friction β : actual slope angle H: depth of slope below the surface γ_{sat} : saturated unit weight γ' : effective unit weight
Cohesive Soil	Dry Condition	$FS = \frac{c'}{\gamma H \cos^2\beta \tan\beta} + \frac{\tan\phi'}{\tan\beta}$	c' : cohesion H: depth of slope below the surface β : actual slope angle ϕ' : drained angle of friction γ : unit weight
	Presence of Seepage Water	$FS = \frac{c'}{\gamma_{sat} H \cos^2\beta \tan\beta} + \frac{\gamma'\tan\phi'}{\gamma_{sat}\tan\beta}$	c' : cohesion β : actual slope angle ϕ' : drained angle of friction H: depth of slope below the surface γ_{sat} : saturated unit weight γ' : effective unit weight

(Source: Das, 2013)

Finite slope

The finite slope critical slip surface is curved, has a limited extent, and is likely to occur on embankments, dumps or cuttings, and natural slopes (McCarthy, 2007). Numerous methods for analyzing slope stabilities have been developed over the years including numerical methods of modeling, finite element, and the limit equilibrium (Salunkhe *et al.*, 2017). The latter is widely applied in practical engineering projects while analyzing the finite slope. As a two-dimensional simulation and firstly applied by Coulomb, the limit equilibrium method can easily be used for diverse soil conditions (circular and non-circular surface), any complexity in slope shapes and gives accurate results (Hoek *et al.*, 2000; Abramson *et al.*, 2002; Matthews *et al.*, 2014) hence its simplicity and long history of use (Abramson *et al.*, 2002).

There is a great variety of analysis methods based on the limit equilibrium principle. Most of them are rooted in the method of slices, such as Morgenstern and Price (1965); Spencer (1967); Janbu (1973), and the simplified Bishop method (1955), which is the simplest and most popularly used for assessing slope stability. These methods vary from each other due to assumptions made. Table 2.7 presents the aforementioned methods with some of their characteristics. The simplified Bishop method is well known for its accuracy in slope stability analysis (Duncan, 1996) which has been theoretically proven by Zhu (2008) to be as rigorous as the so-named rigorous methods.

The method of slices is based on dividing the mass above an assumed slip surface into several vertical slices as shown in Figure 2.7. Then an equilibrium is applied on each slice to detect the critical slip surface (Behera *et al.*, 2016) and the factor of safety can be determined using the Mohr-Coulomb failure criterion equation restricted to the strength parameters, the cohesion C , and the internal friction angle ϕ . Some methods only take into account the equilibrium forces, whereas others consider the moment equilibrium or both (Nash, 1987; Duncan and Wright, 2005).

Table 2.7 Some Stability Analysis Methods for Finite Slopes

Failure Analysis Methods	Particularities and Conditions	References
Bishop method	• FS > FS of ordinary method of slices • high FS when $\alpha \geq 30^\circ$ • different soils and on circular surface • applies the force equilibrium	Murthy (2001); McCarthy (2007).
Simplified Bishop	• FS more accurate • considers the force equilibrium in one direction • the slice forces are horizontal and there is no shear force between them • different soils and on circular surface	Murthy (2001); McCarthy (2007)
Janbu	• only uses the force equilibrium in both directions • no shear between the slices	McCarthy (2007)
Spencer	• the slices side forces are parallel • satisfies force and moment equilibrium	McCarthy (2007)
Morgenstern-Price	• can be applied on different soil types using both forces and moments equilibrium whether on circular or non-circular slip surface	McCarthy (2007)

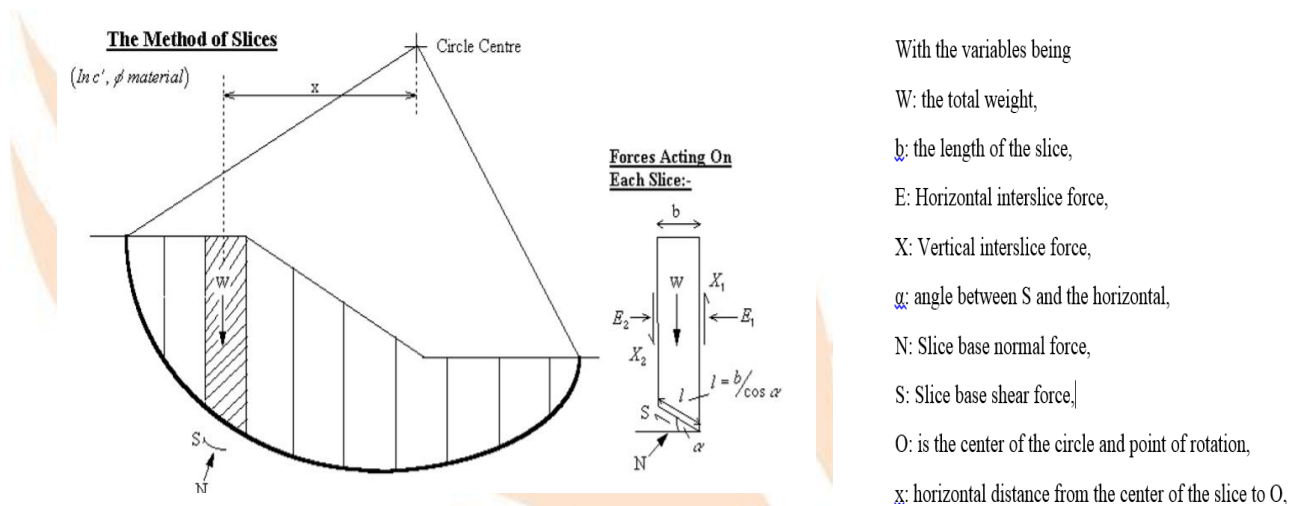


Figure 2.7 Description of Acting Forces on an Assumed Slip Failure Mass (Chowdhury *et al.* 2010)

2.4 Mineralogical and Chemical Composition of Mine Wastes

The possibility of mine wastes causing acid or neutral drainage highly depends on its chemical and mineralogical compositions, which in turn depend on the orebody (Jamieson, 2011). The mastery of the relations of water-minerals and temperature-minerals in the presence of water and other elements is only possible while characterizing mine wastes.

There are several methods for mineralogical and geochemical characterization of mine wastes listed in Tables 2.8 and 2.9, respectively but the most used are scanning electron microscopy (SEM) and X-ray diffraction (XRD) for mineralogy qualification and inductively coupled plasma mass spectrometry (ICP-MS), inductively coupled plasma optical emission spectrometry (ICP-OES) and inductively coupled plasma atomic emission spectroscopy (ICP-AES) for geochemical determination.

Table 2.8 Selected Analytical Techniques Applied for Mine Wastes Mineralogy

Analytical technique	Application to mine waste mineralogy	Example studies from the literature
Scanning electron microscopy (SEM)	Imaging, element mapping, and qualitative spot chemical analysis	Hudson-Edwards <i>et al.</i> (1999); Mendoza <i>et al.</i> (2005); Walker <i>et al.</i> (2005); Marescoti <i>et al.</i> (2009); Xian-Lin <i>et al.</i> (2015)
X-ray diffraction (XRD), including rietveld analysis	Mineral identification based on crystal structure; Rietveld analysis provides relative amounts of crystalline phases in mixtures	Wilson <i>et al.</i> (2009); Marescoti <i>et al.</i> (2009); Xian-Lin <i>et al.</i> (2015)

Table 2.9 Selected Analytical Techniques Applied for Chemical Composition of Mine Wastes

Analytical technique	Application to mine waste mineralogy	Example studies
Inductively Coupled Plasma Atomic Emission Spectroscopy (ICP-AES)	Total metal content	Gjerde <i>et al.</i> (1993); Mendoza <i>et al.</i> (2005); Rashed (2010)
ICP-OES (Inductively coupled plasma - optical emission spectrometry)	Heavy metals determination	Uysal <i>et al.</i> (2008); Chand and Prasad (2013); Ilieva <i>et al.</i> (2018)
ICP-MS (Inductively coupled plasma mass spectrometry)	Heavy metals determination	Moor <i>et al.</i> (2001)
FAAS (Flame Atomic Absorption Spectrophotometer)	Heavy metals determination	Dhanasekaran and Biju (2011); Ipeaiyeda and Ayoade (2017)
Synchrotron-based X-ray fluorescence (micro-XRF)	Element mapping and semi-quantitative spot analysis	Hayes <i>et al.</i> (2009); Walker <i>et al.</i> (2009)

2.4.1 Waste Rock

Waste rock piles are chemically and mineralogically heterogeneous, composed of non-mineralized and/or less mineralized rocks, and contain a considerable amount of metals and metalloids (Solimano, 2015). Tran *et al.* (2003) proved the heterogeneity of the waste rock mineralogy from studies on two different mines, one extracting nickel and the other gold.

The gold mine waste rock was mainly composed of pyrite, chalcopyrite, clay, mica but more of quartz and fewer iron hydroxides and oxides, whereas the nickel mine waste rock showed the presence of pyrrhotite, less chalcopyrite, limonite, goethite, more quartz, clay, and mica.

2.4.2 Tailings

The mineralogy of tailings, which may be composed of mica, talc, chlorite, arsenopyrite, pyrite, etc. (Hamel and Gunderson, 1973) and more quartz (Mlynarek *et al.*, 1995), remains the main aim of many studies as this may severely affect the environment. Jambor (1994) classified sulfide minerals in tailings from the most easily weathered to the least weathered as follows: pyrrhotite-galena-sphalerite-pyrite-arsenopyrite-chalcopyrite-magnetite, whereas for Ritcey (1989), pyrite is the weakest sulfide mineral. The chemical compositions of tailings from different types of ore are summarized in Table 2.10 while Table 2.11 shows the oxide content of tailings from a gold and copper mine.

Table 2.10 Chemical Composition of Tailings

Chemical Element	Tongchang Copper Mine, (Yin <i>et al.</i> , 2011)	Silver mine, (Mendoza <i>et al.</i> , 2005)	Fe-Ore Mine, (Parbhakar-Fox <i>et al.</i> , 2018)	Gold Mine, (Gitari <i>et al.</i> , 2018)	Copper Mine, (Gitari <i>et al.</i> , 2018)
Copper (mg/kg)	829.00	72 – 1 320	98 – 2 120	49.9	669
Manganese (mg/kg)	23.41	19 - 13 800	-	-	-
Sulfur (mg/kg)	321.00	-	-	11 526.5	-
Chromium (mg/kg)	259.00	-	-	533.3	-
Arsenic (mg/kg)	556.00	19 - 11 800	-	39.9	70
Lead (mg/kg)	288.75	780 - 43 700	15-35	15.0	88
Zinc (mg/kg)	595.12	380 - 10 000	32-102	54.4	37
Cadmium (mg/kg)	0.06	1 - 780	-	-	-
Nickel (mg/kg)	-	-	75 - 595	144.0	29

Table 2.11 Oxide Composition of Tailings

Major Elements (%)	Gold Mine Tailings (Gitari <i>et al.</i> , 2018)	Copper Mine Tailings (Gitari <i>et al.</i> , 2018)
SiO ₂	63.8	65.48
CaO	3.1	5.91
Al ₂ O ₃	35.5	12.06
Fe ₂ O ₃	16.3	8.07
MnO	0.1	0.07
MgO	4.3	2.83

2.4.3 Slag

Slag is a hot and molten liquid containing silicates and oxides with other elements depending on the type of mined ore, the added chemical products, and the cooling method. Copper slag is mainly composed of Fe: 30%–40%, SiO₂: 35%–40%, Al₂O₃: ≤10%, CaO: ≤10%, Cu: 0.5% - 2.1%, which can vary according to the type of furnace used during the smelting process (Gorai *et al.*, 2003).

Xian-Lin *et al.* (2015) found in their study, using X-ray diffraction, that copper slag is mainly composed of fayalite and magnetite. Piatak *et al.* (2015) explained this mineralogical composition by the fact that copper slag is a non-ferrous slag. The same results have been established by Chun *et al.* (2016). Table 2.12 summarizes some oxide contents of slag from different authors.

Table 2.12 Oxide Composition of Slag from Different Mines

Mines	Chemical Composition (w, %)				
	SiO ₂	CaO	Al ₂ O ₃	Fe ₂ O ₃	MgO
Iranian national copper industries & Co	41	5.2	3.8	-	1.16
Copper slag (Zain <i>et al.</i> 2004)	31 ± 2	4 ± 0.5	6 ± 0.5	36 ± 1	–
Copper Queen, Prince, USA (Mobasher <i>et al.</i> 1996)	25	10.9	15.6	45	1.70
Copper Queen, Prince, USA (Mobasher <i>et al.</i> 1996)	17	17.4	14.7	35	3.51
Etibank Ergani Copper Plant, Elazig-Turkey (Kiyak <i>et al.</i> 1999)	32	3.9	2.4	-	2.82

2.5 Heavy metals in mine wastes

2.5.1 Heavy metals

Chemical elements with densities higher than 5 g/cm³ are named heavy metals, for example, cadmium, chromium, copper, and lead (Ali and Khan, 2018). These metals are naturally present in the environment but due to anthropogenic activities, especially in the mining industry, higher contents are observed in the vicinity of the mine sites. The release of metals does not automatically occur because of their high concentration in the mine wastes but mostly depends on the condition in which they are (Lapakko, 2002). According to the climatic conditions, leaching, erosion, and the weathering through waste rock, slag dump, and tailings dam release toxic elements (Ettler *et al.*, 2009) producing metallic contaminations.

The pollution caused by mine waste's heavy metals in sediments and water has been widely studied. Malan *et al.* (2014), Rajmohan *et al.* (2014), Kowalska *et al.* (2018), Sey and Belford (2019) largely focused their work on evaluating the contamination status of heavy metals through the determination of pollution indices where the heavy metals concentrations are compared either to the average crustal abundance, the background soils or the country-rock of the studied area. The results obtained in most cases showed that the heavy metals were in excess.

2.5.2 Pollution indices

Enrichment Factor (EF): The formula was firstly introduced by Buat-Menard and Chesselet (1979) and allows to compare a heavy metal concentration in a polluted area with its natural occurrence (background or reference values) in an uncontaminated area. It represents a crucial tool for quantifying contaminants in each environment. Equation 2.4 was used to calculate the EF values.

$$EF = (C_n / C_{ref}) / (B_n / B_{ref}) \quad (2.4)$$

With C_n and C_{ref} respectively being the concentration of the examined heavy metal in the studied area and the concentration of the heavy metal used as a reference in the study area. The heavy metal of reference is generally an element appearing naturally in abundance and can either be Al, Mg, or Fe. B_n and B_{ref} are respectively the concentration of the examined

heavy metal in the reference environment and the concentration of the heavy metal used as a reference in the reference environment. Any significant difference between C_n and C_{ref} accounts for industrial activity.

The classification categories based on EF values proposed by Sutherland *et al.* (2000) are as follows:

- EF < 2, minimal anthropogenic enrichment
- $2 \leq EF < 5$, moderate anthropogenic enrichment
- $5 \leq EF < 20$, significant anthropogenic enrichment
- $20 \leq EF < 40$, remarkably high enrichment
- EF ≥ 40 , extreme enrichment

Contamination factor (CF): the factor is defined according to Tomlinson *et al.* (1980) as the ratio of the concentration of a given heavy metal in the study area (C_n) to its background value (B_n) through equation 2.5. It is an important tool that allows the examination of heavy metal contamination in each environment.

$$CF = C_n / B_n \quad (2.5)$$

The classification categories are:

- CF < 1, low contamination factor
- $1 \leq CF < 3$, moderate contamination factors
- $3 \leq CF < 6$, considerable contamination factors
- CF ≥ 6 , remarkably high contamination factor.

Degree of contamination (DC): Hakanson (1980) defines the degree of contamination as shown in equation 2.6 with CF previously defined and n, the number of heavy metals to determine. The factor represents the level of pollution of toxic elements in an environment.

$$DC = \sum_{i=1}^n CF \quad (2.6)$$

The classification of DC values is as shown below:

- DC < n, would indicate a low degree of contamination.
- $n \leq DC < 2n$, a moderate degree of contamination.

$2n \leq DC < 4n$, a considerable degree of contamination.

$DC > 4n$, remarkably high degree of contamination.

Geo accumulation Index (Igeo): the index has been calculated using the approach proposed by Müller (1969) following equation 2.7, with C_n and B_n as aforementioned, and 1.5 being a correction factor for the plausible variations for the background values. The Igeo helps in evaluating the contamination of the soil considering the current (C_n) and preindustrial (B_n) concentration of a given metal.

$$I_{geo} = \log_2 (C_n / 1.5 \times B_n) \quad (2.7)$$

The classification of I-geo adopted by Buccolieri *et al.* (2006) is presented as:

$I_{geo} \leq 0$, class 0 (unpolluted).

$0 < I_{geo} \leq 1$, class 1 (unpolluted to moderately polluted).

$1 < I_{geo} \leq 2$, class 2 (moderately polluted).

$2 < I_{geo} \leq 3$, class 3 (moderately to strongly polluted).

$3 < I_{geo} \leq 4$, class 4 (strongly polluted).

$4 < I_{geo} \leq 5$, class 5 (strongly to extremely polluted).

$I_{geo} > 5$, class 6 (extremely polluted).

Pollution load index (PLI): The index was determined using equation 2.8 proposed by Thomlinson *et al.* (1980). It helps in evaluating the retrogression of an environment due to the abundance of heavy metals.

$$PLI = [\prod_{i=1}^n CFi]^{1/n} \quad (2.8)$$

The classification of PLI values is shown below:

$PLI = 0$, denotes perfection.

$0 < PLI < 1$, normal presence of heavy metals

$PLI > 1$, indicates deterioration of site quality.

Contamination of soils by heavy metals, done through the wind or flowing water (Su *et al.*, 2014), is determined by the variation of pH whether it is acidic or alkaline, and by the

important amount of trace elements particularly heavy metals (Rybicka, 1996) with the most contaminated zone being the surroundings of the mine (Huang *et al.*, 2017). Heavy metals, contaminating soils, are known to be very harmful (Khalil *et al.*, 2008) by impacting negatively on the microorganisms and enzymatic activity, the plants and human beings (Su *et al.*, 2014), and for some at low concentrations. Although some heavy metals are necessary for the human body, the high amount of some of them can be harmful.

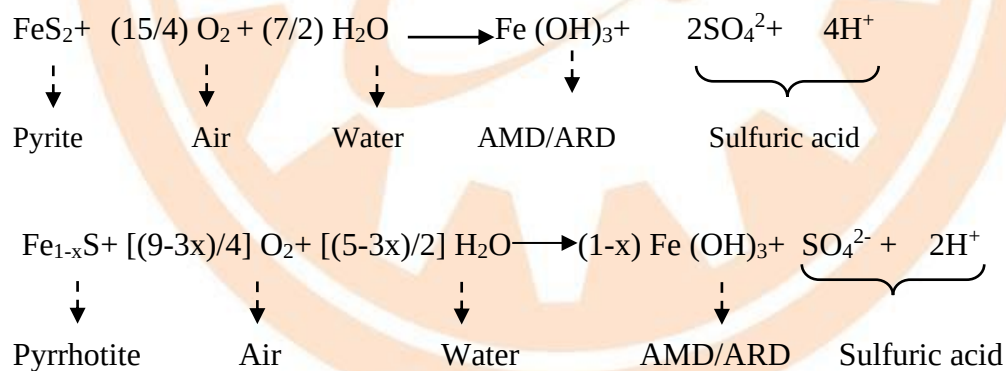
Saha and Paul (2016) widely explained the danger caused by heavy metals in the human body. For example, lead (Pb) is known to be a source of genetic mutation, and arsenic (As) source of lung cancer, urinary bladder with kidney damage. Su *et al.* (2014) reported that cadmium reduces the amount of calcium in humans and is listed 6th harmful toxic element. The International Agency for Research on Cancer (IARC, 1993) reported that cadmium (Cd), nickel (Ni), and copper (Cu) are carcinogenic to humans while lead (Pb) is a body organs killer.

2.5.3 Statistic methods in heavy metals assessment

The assessment of heavy metals in a given environment may result in large data which is difficult to analyze or interpret graphically entailing the growing need to use multivariate statistical methods. Multiple authors like Huang *et al.* (1994), Rencher (2002), Dragović and Mihailović (2009), Hani and Pazira (2011), Qiao *et al.* (2011) applied statistical approaches to geochemical assessment to analyze the anthropogenic or natural association of heavy metals through their concentrations. The customary statistical tools are the correlation matrix either Pearson or Spearman, the principal component analysis (PCA), and the hierarchical clustering analysis (HCA). The correlation matrix is used to identify heavy metals which are associated together. The PCA is applied to the geochemical analysis to help track the contamination cause (Dragović and Mihailović, 2009; Qiao *et al.*, 2011), whereas the HCA spots and groups variables that tend to behave the same way. The PCA and the HCA are sometimes adopted together to facilitate data analysis. The numerous variables are condensed into more comprehensible components (principal components), which are strong enough to exemplify the entire dataset. The analysis uses statistical parameters like the variance and covariance to identify how significant a variable is to be taken into account and drawn in the most representative first two components.

2.6 Acid Mine Drainage

Rybicka (1996) reported that the type of ores susceptible to creating metallic contamination is iron ore, non-iron ore (copper, zinc, lead, etc.), and hard coal. The refining of the commodity from the ore leads to the contamination of surface water creating Acid Mine Drainage (AMD) and the infiltration downward of the acidic water pollutes the underground water. Many authors like Blowes *et al.* (1995); Hitch *et al.* (2010) and Thomas *et al.* (2013) agreed on the possibility of the mine tailing to create acidic components, containing heavy metals and sulfide minerals, through oxidation. Tran *et al.* (2003) mentioned that leaching from sulfide minerals contained in mine waste easily produces Acid Rock Drainage (ARD) or Acid Mine Drainage (AMD). The presence of AMD is noticeable by low to very low pH, the high existence of heavy metals and sulphate. The oxidation of sulfide minerals like pyrite and pyrrhotite are most likely to generate acid (Nelson, 1978; Stumm and Morgan, 1981) through the following chemical reactions (Stumm and Morgan, 1981; Blowes *et al.*, 1994):



Blowes *et al.* (1994) added that the oxidation and hydrolysis of Fe^{2+} and Fe^{3+} respectively with the precipitation of metal hydroxide and hydroxysulfate can also generate acid. Dynamic weathering of the sulfide minerals which can lead to complete reactions occurs in a zone called vadose zone, where secondary minerals like goethite are produced.

CHAPTER 3

METHODOLOGY

3 INTRODUCTION

This chapter presents the nature of the investigated samples and the adopted methodology to carry out the current investigation. The investigation involved fieldwork and laboratory testing of samples that include density determination, gradation determination of tailings, soil, and slag through sieve analysis and sedimentation testing, direct shear test on rock and tailings samples, and *in-situ* vane shear test on tailings. Mineralogical investigations were also performed using X-ray diffraction (XRD) and assessment of the geochemical composition of investigated samples was done through Inductively Coupled Plasma Optical Emission Spectrometry (ICP-OES) and Inductively Coupled Plasma Mass Spectrometry (ICP-MS).

3.1 Fieldwork

Five types of samples namely, tailings, rock, slag, soil, and water were collected from the mine site. The tailings, soil, and slag samples collected were kept in well-sealed polyethylene bags at the time of sampling to avoid any contamination and moisture loss. A global positioning system (GPS) was used to determine the sampling locations (see Figure 3.1). The collected samples were then transported to the Mining and Geotechnical laboratory of Botswana International University of Science and Technology (BIUST) for laboratory analysis. Undisturbed samples of the tailings were taken by driving 100 mm sampling tubes into the tailings and retrieving the filled tubes by carefully digging them out of the tailings and placing them in plastic sample bags to prevent moisture losses. The tailings were deposited using the upstream method.

A vane shear test was performed at selected sampling points on a grid of 200×200 m, with the grid lines trending in a general southeast-northwest direction at the top of the tailings dam. This *in situ* test was performed with the view that the tailings material might have significant clay size proportions after crushing and beneficiation and thus behave as a soft clay material in the slurry state. The test was conducted using a Matest field vane test equipment to determine the undrained shear strength (C_u) of the tailings. At each test point,

the vane was pushed into the tailings after scraping off the top layer to a depth of about 10 cm. A torque was then applied through clockwise rotation of the vane shear equipment until the lower part of the shear vane followed the upper part thus, demonstrating the failure of the tailings. Upon failure, the vernier value T , obtained on the nonius was then recorded for the determination of the undrained shear strength (C_u) of the tailings through equation 3.1. Samples of the tailings were then taken at each test point after the completion of the vane test with the aid of a hand trowel. In total, twelve (12) tailings samples were collected.

$$C_u = 6T/7\pi d^3 \quad (3.1)$$

where: C_u = shear strength in kPa,

T = the torque reading on the nonius in N.mm,

d = blade diameter in mm.

Ten slag samples were taken from the slag dump on a square grid of approximately 40 m with the aid of a hand trowel. In addition, 16 representative soil samples were randomly collected by means of an auger from three different sites around the tailings repository. The first soil sampling site was located about 200 m north-west of the tailings repository and referred to as NW site, the second was within the mine (between the tailings dam and slag dump), and the third site on the south-east side of the mine (SE site) between the mine hospital and the mine. Twenty-five rock samples were also taken from the waste rock dumps based on the different facies available in the waste rock dumps where 4 rock types were distinguished.

Water samples (WS1a, WS1b, WS2, WS3, WS4, WS5a, WS5b, WS6a, WS6b, WS7a, and WS7b) were collected from different sites comprising seepage water from the tailings dam and the shafts going to the treatment plant to be treated with lime, water from the treatment plant through pipes into the irrigation dam, water from the irrigation dam, and overflow of the irrigation dam going outside the mine and used by farmers. The underflow goes back to the treatment plant to be treated again. During the water sampling exercise, the potential hydrogen (pH), the electrical conductivity (EC), and the total dissolved solids (TDS) were determined using a Hanna HI9811-5 portable pH meter in the field at the sampling points. The water samples were kept in sterilized polyethylene containers and transported to Standard Global Services (SGS) in South Africa for testing. It is worth noting that the water

samples have been grouped into two. Samples WS1a, WS1b, WS2, WS3, and WS4, representing Group 1 were obtained from tailings seepage water and water from the shaft, while samples WS5a, WS5b, WS6a, WS6b, WS7a, and WS7b forming Group 2 were treated water samples from the treatment plant.

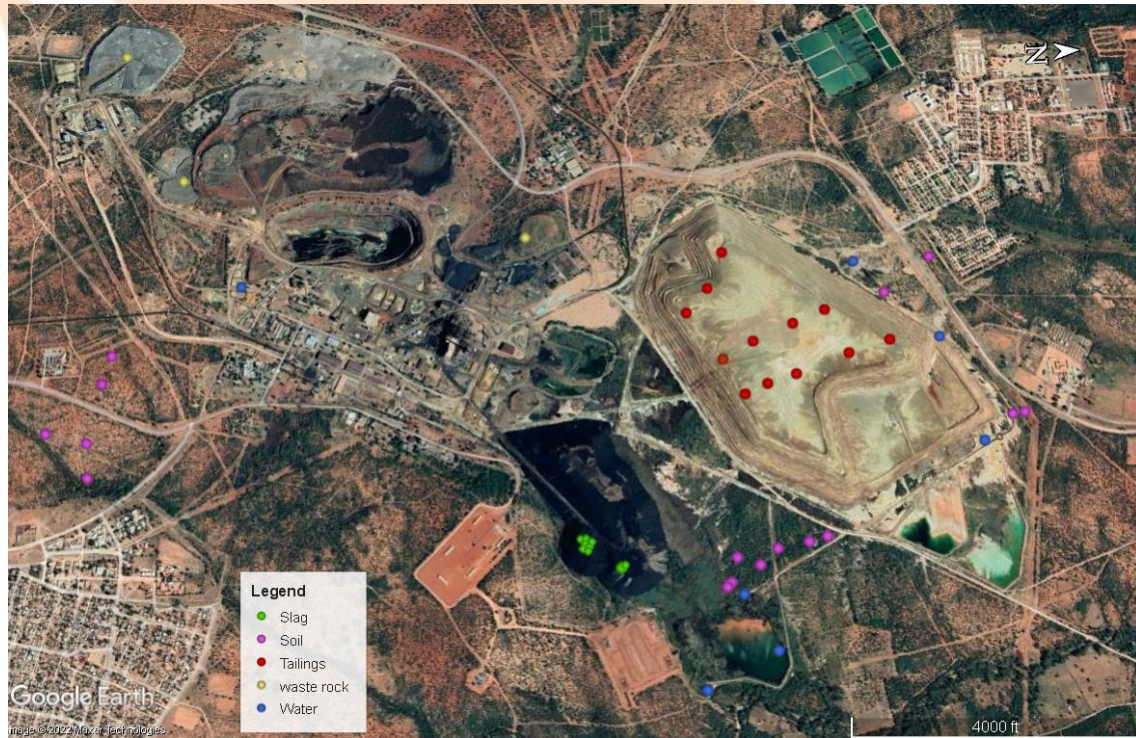


Figure 3.1 Map showing Location of Tailings, Rocks, Slag, Soil and Water Samples Collected within BCL Mine

3.2 Laboratory Testing

3.2.1 Physico-chemical parameters determination

Sixteen (16) oven-dried soil samples were used for organic matter determination through the loss of ignition method, using equation 3.2 below following AASHTO T267-86 specifications with slight modification on the temperature and the time. The test specimens were placed in clean containers and transferred into a furnace set at 400 °C for 24 hours.

$$M (\%) = \frac{m_1 - m_2}{m_1 - m_c} \times 100 \quad (3.2)$$

where: OM = organic matter, %;

m_1 = mass of container with the dry sample, g;

m_2 = mass of crucible with sample after ignition, g;

m_c = mass of crucible, g.

The pH, TDS, and EC of oven-dried tailings, soil, and slag samples were determined by mixing 20 g of the samples with 40 ml of distilled water and stirring for 30 min. The pH measurements were taken using the portable Hanna HI9811-5 pH meter.

3.2.2 Density and Unit Weight Determination

The densities of rocks, tailings, and slag lumps were obtained through the water displacement method in accordance with ISRM (1981) Suggested Methods for Rock Characterization, Testing and Monitoring for the rock and slag samples and ASTM D7263 for the tailings samples. The bulk densities of unevenly shaped lump samples were determined by coating the test specimens with melted wax and completely immersing the wax-coated test specimens in water. The test, for each sample, was done in triplicate. The density of the specimens was calculated using equation 3.3. The unit weight of the test specimens was obtained by multiplying the density determined by gravity 9.81 N/kg.

$$\rho = \frac{X}{\left[(Y-Z) - \left(\frac{Y-X}{\rho_w}\right)\right]} \quad (3.3)$$

where: ρ = density of the sample (g/cm^3)

ρ_w = density of wax (0.914 g/cm^3)

X = mass of sample in air, (g)

Y = mass of sample, (g)

Z = mass of coated sample upon immersion in water, (g)

3.2.3 Particle Size Distribution

The particle size distribution of the tailings and slag were determined through sieve analysis for the coarse-grained material ($> 75 \mu\text{m}$) and the sedimentation method through the hydrometer analysis for the fine-grained materials ($< 75 \mu\text{m}$) on 1 kg of oven-dried material. All the slag and soil samples were however subjected to dry sieving only due to their coarse nature. The 52H hydrometer was used for the hydrometer analysis with sodium hexametaphosphate (NaPO_3)₆ as the dispersing agent. The test was conducted in accordance with ASTM D422-63 specifications.

3.2.4 Direct Shear Test

The direct shear test was conducted on both tailings and rock samples. In the case of the tailings samples, a 60 × 60 × 25 mm shear box apparatus was used to determine the strength properties in accordance with ASTM D3080-72 specifications. Undisturbed test specimens were extruded from 100 mm core cutters into the 60 × 60 × 25 mm cutting ring and trimmed flush with the ring edges. The prepared test specimens were placed in the shear box equipment and subjected to constant normal stresses of 75 kPa, 150 kPa, and 300 kPa while applying an increasing horizontal stress at the Geotechnical Laboratory of the Earth and Environmental Science Department of BIUST. The normal and shear stresses were computed using equations 3.4 and 3.5 respectively. Graphs of the shear stress versus the relative shear displacement were then plotted and the peak shear stress corresponding to each applied normal stress were extracted and plotted against the applied normal stresses to determine the cohesion and friction angle graphically based on the Mohr-Coulomb failure criterion.

$$\sigma_n = \frac{9.81 \times W}{S} \times 1000 \quad (3.4)$$

where: σ_n = normal stress, kPa;

W = total mass supported by the specimen, kg;

S = cross-sectional area, mm².

$$\tau = \frac{T}{S} \quad (3.5)$$

where: τ = shear stress, kPa

T = shear load, kN

S = cross-section area, mm²

The direct shear test on rock samples was conducted in accordance with ISRM (1981) specifications in a rock shear box apparatus. Though the test is designed for determination of the shear strength of discontinuities, it was adopted for the determination of the shear strength of the contact surfaces of the rock boulders in the waste rock dumps. Because of the mixed composition of the waste rock dumps, thirty test specimens were prepared from

the mixed rock samples obtained by saw cutting and subjecting them to the direct shear test in the BIUST Mining and Geotechnical laboratory. Test specimens of rocks with rough contact surfaces, and rocks with smooth cut contact surfaces were prepared for the test based on the surface appearances of the rock samples in addition to those with pre-existing discontinuities. The test specimens were encapsulated in mortar in the shear mould former ensuring that the contact surface to be sheared was parallel to a horizontal slot of the mould and allowed to set. Upon setting, each test specimen was then loaded into the rock shear box apparatus and subjected to shear force at constant normal loads of 1 kN, 2 kN, and 3 kN during which the shear forces and displacements were recorded. The normal and shear stresses were computed using Eqs. 3.6 and aforementioned equation 3.4 (to determine the shear stress τ , respectively. Graphs of the shear stress versus the shear displacement were then plotted and from these, the peak shear stresses were extracted and plotted against the applied normal stress to determine the cohesion and the friction angle graphically.

$$\sigma_n = \frac{N}{S} \quad (3.6)$$

where: σ_n = normal stress, kPa

N = normal load, kN

S = cross-section area, mm²

The direct shear test could not be performed on the slag samples because of the coarse nature of slag and the small size of the soil shear box (60 × 60 × 25 mm) equipment available. In place of this test, the angle of repose was determined to approximate the friction angle of the slag to determine the stability of the uncompacted slag slopes.

3.2.5 Petrographic Studies

Two thin sections of each rock type were prepared for petrographic analysis to identify the mineral assemblage of the rocks, therefore, enabling their identification. The rocks were first cut into thin rectangular slices which were then pasted on a glass slide using epoxy resin at the Mineral Dressing laboratory in BIUST. The combination was ground to 0.03 mm and polished with diamond paste to have a clear view of the minerals under the microscope. The polished thin sections were then investigated at BIUST using the transmitted light microscope.

3.2.6 X-ray Diffraction Analysis

X-ray diffraction analysis on the tailings, soil and slag samples was conducted using a Bruker D8 advance diffractometer housed at BIUST. The tailings, soil, and slag samples passing sieve No. 200 were used for the test. The diffractometer was run at 40 kV and 40 mA. All the samples were scanned at different diffraction angles for values of 2 theta, with step time of 0.35 s ranging from 5 to 100° and from 8 to 100°. The software HighscorePlus was used for the identification of the peaks on the diffractograms through a database also available on the software. The diffractograms obtained showed the relative intensity versus the 2 theta values.

3.3 Slope Stability Analysis of Waste Dumps

Stability analysis of the different mine waste dumps namely tailings, slag, and waste rock with their specific properties, was performed using Rocscience Slide2 software program Version 9.007. The Slide program is based on limit equilibrium analysis and is a key tool widely used in geotechnical slope stability analysis. The main input parameters for the analysis were the material properties viz the cohesion, unit weight, and internal friction angle of the wastes obtained from laboratory testing. The output parameter was the factor of safety (FS) achieved from the Slide software analysis. The mine waste dumps were considered stable for $FS > 1.25$, (Zou *et al.*, 2018), critical for $FS = 1.25$ and unstable for $FS < 1.25$ to consider the uncertainties of the parameters.

The circular potential failure mechanism was adopted with the Simplified Bishop Method. A grid search was selected to determine the critical slip surface with the lowest FS based on a 20×20 m grid interval spacing. This grid spacing generated a large number of slip centers (441), to represent the center of the slip circles. Figure 3.2 shows the tailings dam design. As for the waste rock and slag dumps, their design was only composed of the foundation and the dumped material.

The computer simulations were performed using different scenarios comprising variations of the dump height, slope angle, and water level (taken as constant and variant depending on the scenario). The mine waste dumps have heights that are generally, lower than 60 m, therefore, 60 m was used as the maximum height of the waste dumps. The limitation in height of dumps is based on the fact that mining operations had been stalled. The landscape

of the area is flat and the only treatment to the foundations of the waste dumps was clearing of vegetation cover and the top soil.

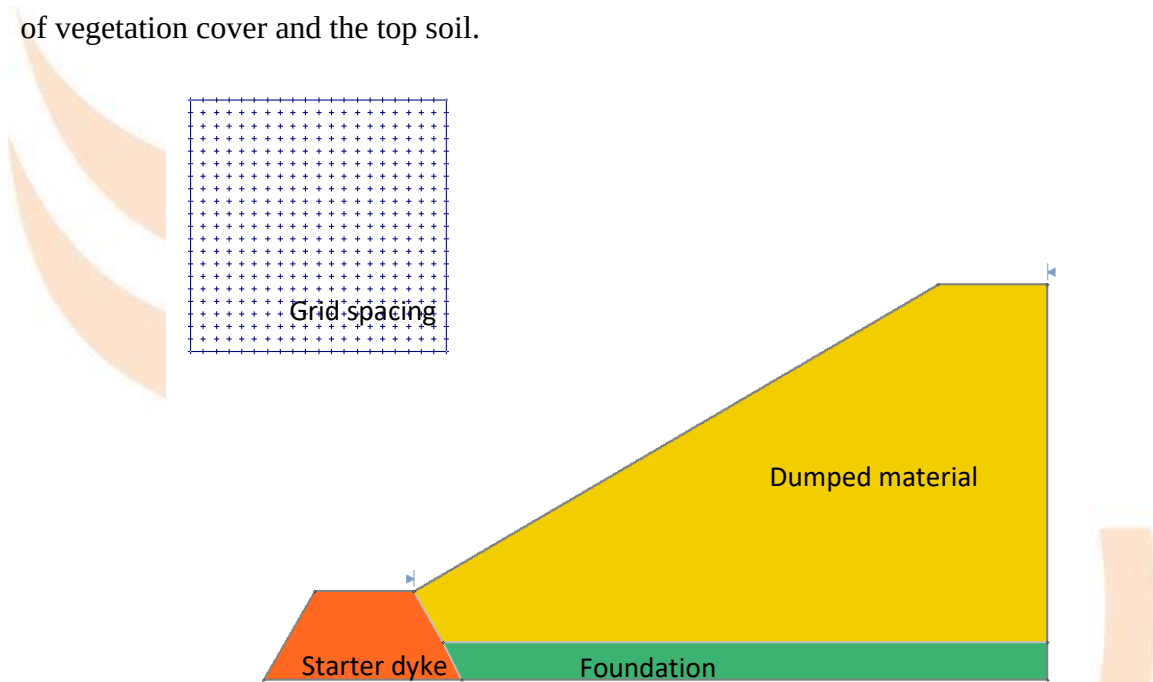


Figure 3.2 An illustration of Factor of Safety Computation using Rocscience Software Slide with the Starter Dyke only Used for Tailings Dam

The following scenarios were used for the analysis:

- Slope angles of the waste rock dump used were 30°, 35° and 40° with varying heights of 40 m, 50 m, and 60 m. It was considered that the waste rock dump is less affected by water based on its coarse-grained and porous nature due to the packing of the rock fragments, which allows fast drainage of water through it with very low to no water retention,
- The slag dump slope angles used were 30°, 35° and 40° with varied heights of 40 m, 50 m, and 60 m similar to the conditions applied in the waste rock dump slope analysis. Varied water level increment of 10 m (depth of phreatic surface) was also applied to each of the 10 m increase in height of the slag dump. The introduction of phreatic surface to the slag dump was to account for possible presence of fine particles in the slag dump which makes it possible to retain water,
- The tailings dam height was varied between 40 m, 50 m and 60 m with different slope angles of 30°, 35°, 40°, 45°, and 50° while varying the water level from 10 m at the base and at 10 m incrementally to the top of the dam. The additional slope angles of 45° and 55° were to cater for the steep slope observed during the site visit.

Variation of the water level was based on piezometric records in 2018 (Preliminary Inspection Report BCL mine, 2018) indicating the presence of water in the tailings dam.

3.4 Geochemical Analyses

The pulverized tailings, slag, and soil samples with water samples were sent to SGS, South Africa for geochemical analysis. In total, 12 tailings samples, 10 slag samples, 16 soil samples, and 11 water samples were carefully prepared. 150 g of each solid material were subjected to the inductively coupled plasma optical emission spectrometry (ICP-OES) and 100 ml of each water sample were subjected to the inductively coupled plasma mass spectrometry (ICP-MS). The elements of interest were Fe, Ni, Co, Cu, Zn, Pb, Cr, Mn, As, and Cd due to their serious impact on the environment.

The digestion method adopted is the well-known and reliable aqua regia with hydrochloric acid (HCl) and nitric acid (HNO₃) used to extract the metals. The dissolution procedure involved a ratio of 3:1 mixture of HCl acid and HNO₃ acid to obtain an acid-soluble digest. The samples dissolved in aqua regia were then heated to a low boil and any acid-soluble elements were oxidized. Once the reaction was complete, the samples were diluted to volume, allowed to settle, and analyzed for the elements of interest listed above.

Pollution assessment indices: Pollution indices are inherent when evaluating the soil quality and predicting the sustainability of the ecosystem. The concentrations of the chemical elements obtained from the investigated soil and water samples were used to determine the geochemical pollution indices that included the enrichment factor (EF), contamination factor (CF), degree of contamination (DC), pollution load index (PLI), and geo-accumulation index (I_{geo}).

Fe, Al, and Mn were the most used elements as metals of reference to determine EF. The chosen metal of reference in this investigation was Fe as it is the most used in the literature in respect of its natural occurrence and its wide usage in the literature in the same study area (Likuku *et al.* 2013; Motswaiso *et al.* 2019).

To determine the CF of soils, the concentrations of the elements of interest from the average continental crust as reported by Wedepohl (1995) widely used (Vineethkumar *et al.*, 2020)

were used as the background values. The background values of heavy metals in the water were from the Botswana maximum permissible limits for irrigation water for crop production (BOBS, 2011) as background values of the study area were not available. These values were adopted to compare the irrigation BOBS values with the heavy metal content in treated water as the latter is used for crop irrigation.

Statistical analysis: statistical analysis was established with the purpose of understanding the affiliation and determining the possible sources of heavy metals in the soil samples. Three statistical methods were implemented in the present study namely, the Pearson correlation, the principal component analysis (PCA), and the hierarchical clustering analysis (HCA) via Xlstat 2020. The heavy metals concentrations were computed to get the Pearson matrix where the level of association among the heavy metals was obtained. The PCA and the AHC were both used to determine the similarities and to group samples with the same pattern. Practically, the program performs an eigen analysis from the heavy metals concentration to extract the eigenvalues and eigenvectors of the variance-covariance matrix. A graph of the eigenvalues (scree plot) helps to choose the most meaningful principal components (PCs) to take into consideration. In the study, two PCs (PC1 and PC2) were considered to describe the variable groups as the remaining components merely affect the model.

CHAPTER 4

PETROGRAPHIC STUDIES AND GEOTECHNICAL RESULTS

4 INTRODUCTION

In this chapter, the results from the petrographic, geotechnical studies, and slope stability analyses were presented. The results cover the rock thin sections studied and the geotechnical characteristics of the tailings, rock, slag, and soil samples from the BCL environment.

4.1 Results of Petrographic Studies on Rock Samples

Four rock types were identified through hand specimen and thin section studies of fresh rock samples from the mine waste dumps. The rock identification was based on color, mineral composition, texture as seen in hand specimens and under the microscope. Thin section images of the samples under cross-polarized light (XPL) are presented in Figure 4.1. The four rock types identified from the studies include metaquartz-monzonite, metagranite, biotite schist, and hornblende-biotite gneiss as summarized in Table 4.1.

Table 4.1 Petrographic Characteristics of Waste Rock Samples

Characteristics	Metaquartz-monzonite	Metagranite	Biotite schist	Hornblende - biotite gneiss
Color	leucocratic	leucocratic	mesocratic	mesocratic
Grain size	medium	medium	fine-to medium	medium
Texture	granoblastic	granoblastic	porphyroblastic	granoblastic
Mineral composition	quartz 38%, plagioclase 59%, biotite 2%, chlorite 1%.	biotite 1%, quartz 12%, plagioclase 40%, microcline 47%.	biotite 80%, chlorite 3%, quartz 17%.	hornblende 30%, quartz 40%, biotite 25%, plagioclase 5%.

The metaquartz-monzonite which was leucocratic (pink), displayed a slight alternation of light and dark coarse-grained minerals with granoblastic texture. The sample was essentially built up of fresh plagioclase (59%) and quartz (38%) with biotite (2%) which is slightly chlorite-altered (1%). The quartz occurred commonly as strained anhedral free grains, with

size ranging from 0.4 to 1 mm. Plagioclase, with sizes varying from 0.8 to 1.3 mm appeared as euhedral grains with tabular and locally rounded-like shapes. The biotite, measuring approximately 0.4 mm, was enclosed in plagioclase and unevenly distributed. Both biotite and chlorite occurred as minor minerals.

The metagranite was a leucocratic rock slightly foliated with a granoblastic texture and composed of medium to coarse-grained feldspars, quartz, and biotite. The feldspar comprising microcline and plagioclase, represented the largest mineral group in the rock. Plagioclase made up 40% and was observed in places to be enclosed in quartz or as free grains. Its size ranged from 0.2 to 0.4 mm. Quartz (12%) and microcline (47%) were evenly distributed in the rock and were observed as free grains. The quartz crystals were 0.2 to 1 mm in size, while microcline measured 0.4 to 0.8 mm. Biotite (1%) on the other hand appeared enclosed in plagioclase and quartz minerals with a size of about 0.1 mm. It was tabular in shape and slightly fresh.

The biotite schist was mainly composed of biotite (70%) and quartz (27%) and well strained giving it a clear schistosity to the rock sample. The biotite minerals which were generally 0.8 to 1.2 mm in size, were tabular and elongated in shape. Chlorites (3%) are in places enclosed in biotite. Quartz crystals are anhedral varying from 0.4 to 1.2 mm in size. Opaque minerals are also visible in the sections but were not identified.

The hornblende-biotite gneiss was mesocratic with medium-to fine-grained hornblende, biotite and quartz. The foliation was highly visible with the light minerals being plagioclase and quartz and the dark minerals comprising biotite and hornblende. Biotite (25%) and hornblende (30%) were at near equal proportions and occurred evenly in the sample. Biotite was 0.4 to 1.2 mm in size. The quartz crystals (40%) were 0.8 to 1 mm, anhedral, locally contained inclusions of biotite and hornblende, and were strained with undulatory extinction. The hornblende was highly strained hence elongated and its size ranged between 0.1 to 1.2 mm. Plagioclase (5%) was completely fresh and slightly euhedral with size near 0.8 mm.

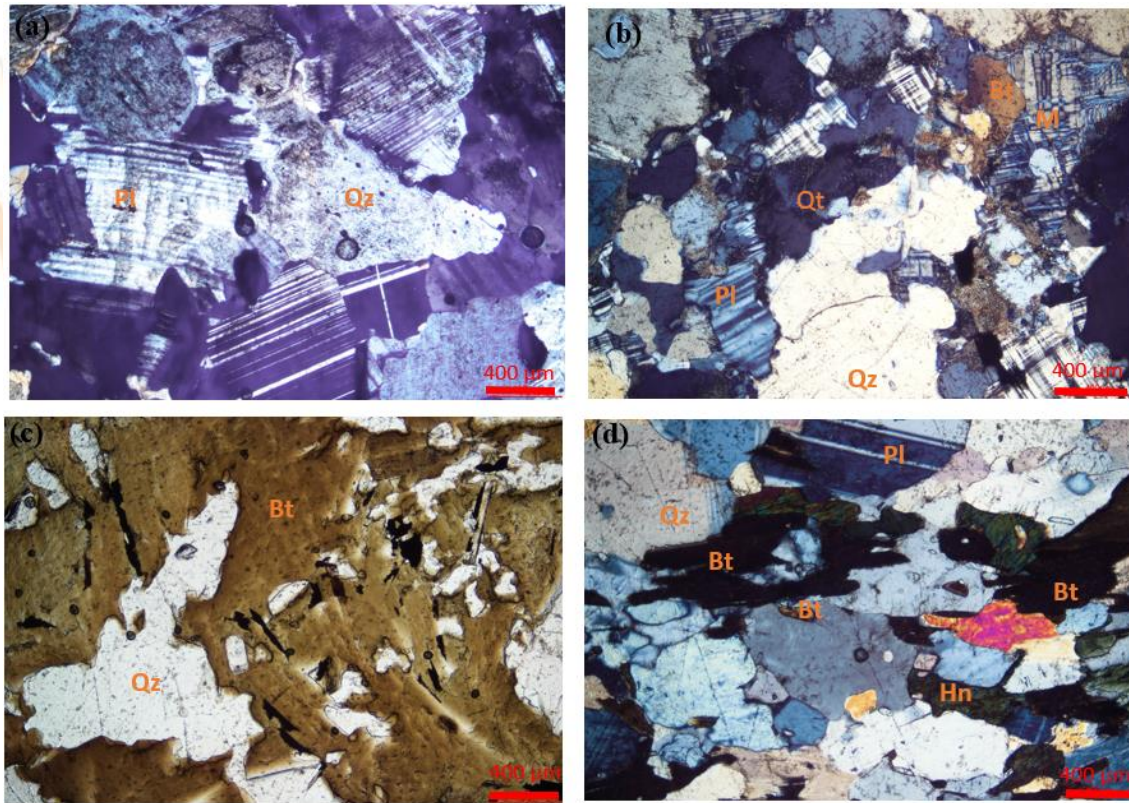


Figure 4.1 Photomicrographs in cross-polarized light, (a) Metaquartz-monzonite essentially containing quartz and plagioclase, (b) Metagranite essentially containing microcline, quartz, and plagioclase, (c) Biotite schist essentially containing quartz and biotite, (d) Hornblende-biotite gneiss essentially containing quartz, hornblende, biotite, and plagioclase. (Pl = plagioclase, Qz = quartz, Bt = biotite, M = microcline, Hn = hornblende)

4.2 Geotechnical characteristics

4.2.1 Characteristics of waste rock

Table 4.2 summarizes the densities and unit weights of the four rock-type samples described above. The mean densities of metaquartz-monzonite, metagranite, biotite schist, and hornblende-biotite gneiss were $2,903 \text{ kg/m}^3$, $2,656 \text{ kg/m}^3$, $2,573 \text{ kg/m}^3$ and $2,643 \text{ kg/m}^3$, respectively. The mean unit weights of metaquartz-monzonite, metagranite, biotite schist, and hornblende-biotite gneiss were 28.5 kN/m^3 , 26.0 kN/m^3 , 25.5 kN/m^3 , and 26.0 kN/m^3 , respectively. The average unit weight of the rocks was determined to be 26.2 kN/m^3 , which was subsequently used for slope stability analysis of the waste rock dumps.

Table 4.2 Density and Unit Weight of Waste Rock Samples

Rock Type	Mean Density (kg/m ³)	Mean Unit Weight (kN/m ³)
Metaquartz-monzonite	2,903	28.5
Metagranite	2,656	26.0
Biotite schist	2,573	25.5
Hornblende-biotite gneiss	2,643	26.0

Plots of the shear stress versus shear displacement obtained from the rock direct shear test on rough contact surfaces of metaquartz-monzonite are presented in Figure 4.2. For applied normal stresses of 551 kPa, 1102 kPa, and 1,654 kPa, the peak strengths obtained are respectively 1,158 kPa, 1,433 kPa, and 2,205 kPa (Figure 4.2 a). The corresponding residual strengths obtained are 772 kPa, 1,158 kPa, and 1,764 kPa. The peak strength values were extracted and plotted against the applied normal stresses (Figure 4.2 b). From Figure 4.2, a friction angle of 43°, and cohesion of 551 kPa were obtained for the metaquartz-monzonite.

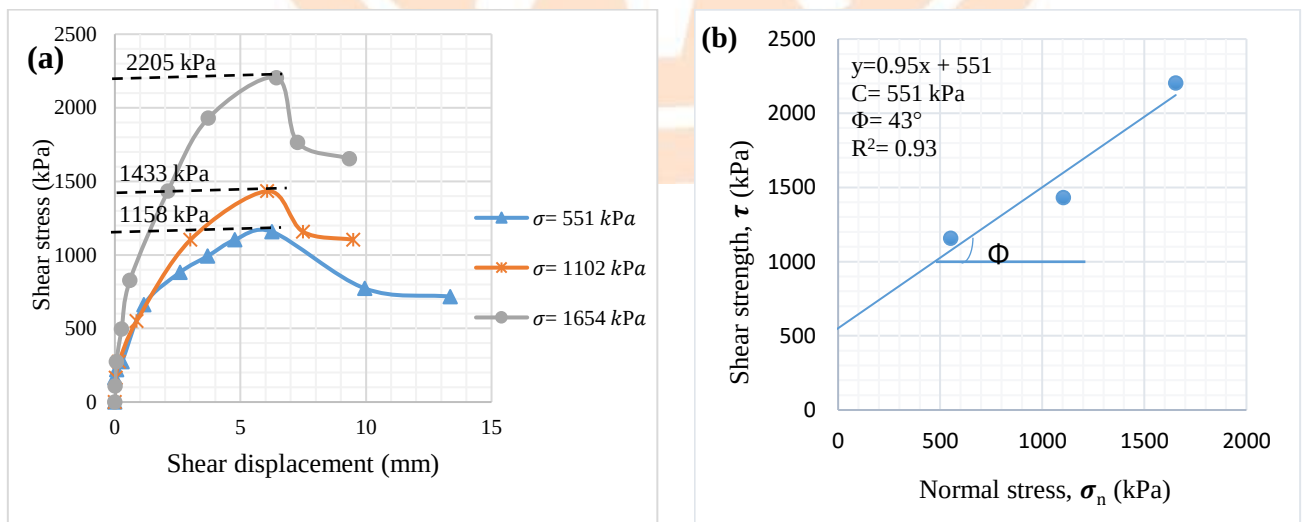


Figure 4.2 (a) Shear Stress versus Shear Displacement and (b) Mohr-Coulomb Failure Envelope for Metaquartz-monzonite

Similar plots for the direct shear test on rough contact surfaces between the four types of rocks and on smooth contact surfaces were carried out and presented in Appendix B. Summaries of the test results are submitted in Tables 4.3 and 4.4 for smooth contact surfaces and rough contact surfaces respectively.

Table 4.3 Cohesion and Friction Angle Values for Smooth Contact Surfaces of Rock Samples

Characteristics	Metaquartz-monzonite with biotite schist contact surface	Metagranite Contact surface	Metagranite with metaquartz-monzonite contact surface	Metagranite with metaquartz-monzonite contact surface	Biotite schist with hornblende-biotite gneiss contact surface
Cohesion (kPa)	0	0	183	0	0
Friction angle (°)	46	51	45	42	39

Table 4.4 Cohesion and Friction Angle Values for Rough Contact Surfaces of Rock Samples

Type of Sheared Rock Contact Surface	Rock Types Sheared	Cohesion (kPa)	Friction Angle (°)
Different Rock Types Contact Surface	Metaquartz-monzonite and biotite schist contact	435	24
	Metagranite and biotite schist contact	475	28
	Metaquartz-monzonite and hornblende-biotite gneiss contact	402	38.5
	Metagranite and biotite schist contact	479	24
	Metagranite and hornblende-biotite gneiss contact	227	34
	Metaquartz-monzonite and Metagranite contact	271	42
	Metaquartz-monzonite and Metagranite contact	137	27.5
	Biotite schist and hornblende-biotite gneiss contact	197	31.5
	Metagranite and hornblende-biotite gneiss contact	243	30
	Biotite schist and hornblende-biotite gneiss contact	308	19
	hornblende-biotite gneiss contact	323	41.5
Same Rock Type Contact Surface	Biotite schist contact	766	16
	Metagranite contact	526	32
	Metaquartz-monzonite contact	294	35

In general, the cohesion values for the smooth contact surfaces was 0 kPa and 183 kPa while for the rough contact surfaces the values obtained ranged from 137 kPa to 766 kPa. The friction angles ranged from 39° to 51° and 16° to 42° for smooth and rough contact surfaces, respectively. The average rock cohesion value obtained is 277.16 kPa (≈ 277 kPa) while an average of 34° was obtained for the friction angle. Notably, high values of cohesion resulted in lower values of friction angle.

4.2.2 Characteristics of Tailings

Table 4.5 summarizes the density and the unit weight results for the twelve tailings samples. The density of the tailings samples ranged from 1,423.2 kg/m³ to 1891.4 kg/m³ with a unit weight ranging between 14.0 kN/m³ and 18.5 kN/m³. The average density of the tailings was 1,623.0 kg/m³, while the average unit weight was 15.9 kN/m³ (≈ 16 kN/m³).

Table 4.5 Density and Unit Weight of Tailings Samples

Sample ID	Density kg/m ³	Unit Weight (kN/m ³)
TL1S1	1,869.4	18.3
TL1S2	1,603.3	15.7
TL1S3	1,517.0	14.9
TL1S4	1,519.3	14.9
TL1S5	1,539.2	15.1
TL1S6	1,466.3	14.4
TL2S1	1,856.3	18.2
TL2S2	1,603.2	15.7
TL3S1	1,891.4	18.5
TL3S2	1,619.2	16.0
TL3S3	1,567.7	15.4
TL3S4	1,423.2	14.0

Particle size distribution curves obtained from wet sieving and sedimentation test on the tailings samples are presented in Figure 4.3. The tailings material was mostly composed of sand-size particles with silt and clay. Summaries of the results for the test samples are provided in Appendix C. The grading analysis indicated a sand size range of 59% to 74%, 16% to 33% for silt, and clay size from 4% to 9%. The effective size, D_{10} , along with D_{30} and D_{60} , corresponding respectively to 10%, 30%, and 60% finer were extracted from the grading curves and used to calculate the coefficient of uniformity (C_u), the coefficient of curvature (C_c), and the coefficient of permeability (k) based on Hazen's formula. These results are summarized in Table 4.6. C_u and C_c values ranged from 4.41 to 26.16 and from 0.96 to 5.81, respectively. The C_u and C_c values of tailings material showed that the material was well graded. The coefficient of permeability of the tailings varied from 0.49×10^{-6} m/s to 11.56×10^{-6} m/s showing moderate drainage. The high sand-size proportion of the tailings inhibited the performance of the Atterberg limits tests as soil pastes could not be prepared for the liquid limit test or be rolled up into threads of 3 mm for the plastic limit test.

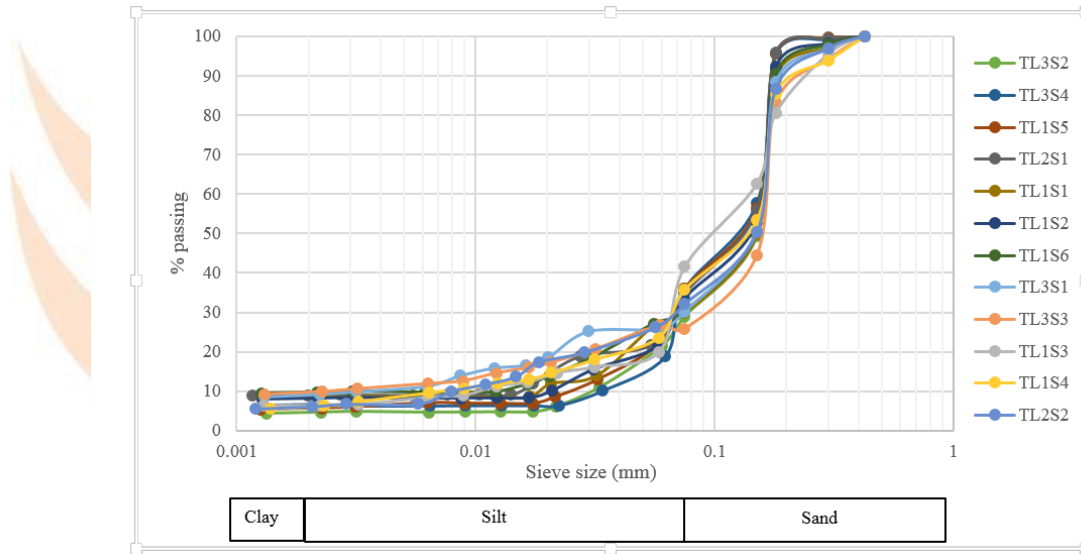


Figure 4.3 Particle Size Distribution Curves of Tailings Samples

Table 4.6 Particle Size Distribution Parameters of Tailings Samples

Sample ID	D ₁₀ (mm)	D ₃₀ (mm)	D ₆₀ (mm)	C _u	C _c	k (10 ⁻⁶ m/s)
TL1S1	0.020	0.092	0.162	8.10	2.61	4.00
TL1S2	0.020	0.070	0.150	7.50	1.63	4.00
TL1S3	0.006	0.074	0.157	26.16	5.81	3.60
TL1S4	0.015	0.070	0.150	10.00	2.18	2.25
TL1S5	0.025	0.080	0.150	6.00	1.71	6.25
TL1S6	0.012	0.070	0.160	13.33	2.55	1.44
TL2S1	0.008	0.068	0.158	19.75	3.66	0.64
TL2S2	0.018	0.075	0.160	8.89	1.95	3.24
TL3S1	0.007	0.067	0.156	22.29	4.11	0.49
TL3S2	0.010	0.070	0.140	14.00	3.50	1.00
TL3S3	0.031	0.090	0.160	5.16	1.63	9.61
TL3S4	0.034	0.070	0.150	4.41	0.96	11.56

The *in-situ* vane shear test conducted on the tailings material resulted in undrained shear strength values that varied from 13.2 kPa to 232.5 kPa as presented in Table 4.7. The average undrained strength value excluding results of samples TL1S1, TL1S3, and TL3S1 (due to their significant variance from the general trend of the results) was 65 kPa. It is envisaged that the variation in strength from one location to the other is dependent on the degree of compaction of the tailings at the top of the dam and also the proportion of sand size. Indeed,

the strength values increased from the center of the dam to the edges because of the wet nature of the center.

Table 4.7 In-situ Vane Shear Test Results

Sample ID	Torque (N.mm)	C_u (undrained shear strength), kPa
TL1S1	10.8	232.5
TL1S3	7.7	84.9
TL1S4	7.2	79.4
TL1S5	1.2	13.2
TL1S6	4.2	46.3
TL2S1	9.6	105.8
TL3S1	8.6	94.8
TL3S2	5.2	57.3
TL3S4	4.8	52.9
TL3S5	3.6	39.7

The direct shear strength test results of the tailings were presented as plots of shear stress against the relative shear displacement (Figure 4.4 a) and peak shear stress values extracted from the graphs versus the corresponding applied normal stress values of 75 kPa, 150 kPa and 300 kPa (Figure 4.4 b). Figures 4.4 (a) and (b) illustrate these plots respectively for sample TL1S1 with cohesion of 11 kPa and friction angle of 34.07°. Similar graphs constructed are presented in Appendix C. Summaries of the shear strength test results are presented in Table 4.8. Generally, the cohesion and friction angle ranged from 11 kPa to 39 kPa and 30.88° to 38.08° respectively. The average cohesion value was determined as 22.5 kPa while for the friction angle, an average of 35.4° was obtained. The average cohesion and friction angles were subsequently used as input parameters to determine the factor of safety of the tailings dam in the stability assessment of the dam.

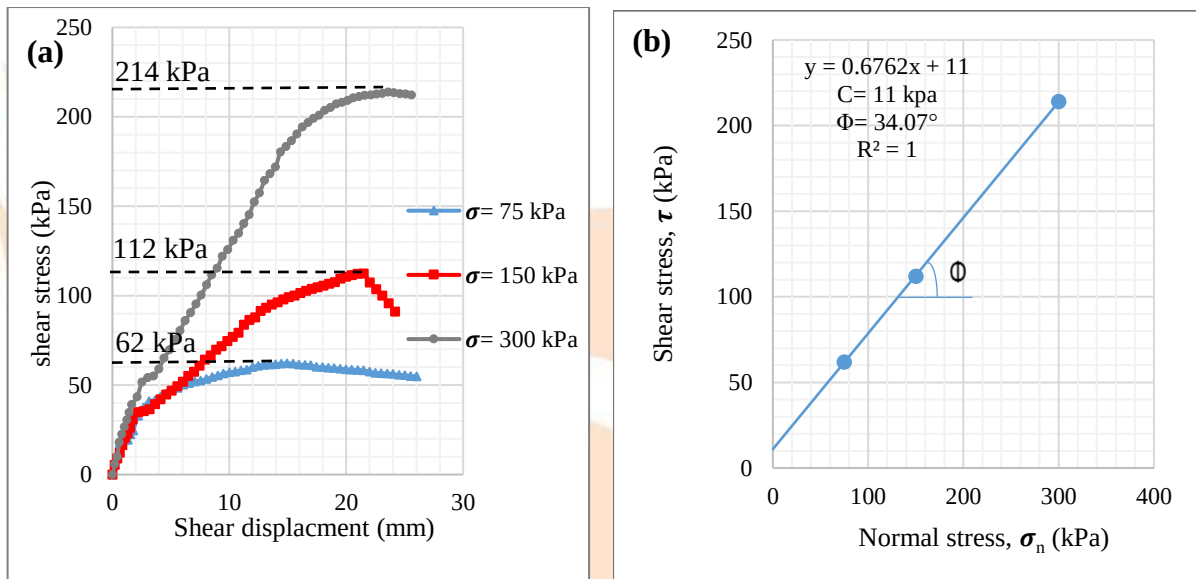


Figure 4.4 (a) Shear Stress versus Shear Displacement and (b) Mohr-Coulomb Failure Envelope for TL1S1

Table 4.8 Cohesion and Friction Angle Values for Tailings Samples

Sample ID	C (kPa)	ϕ (°)
TL1S1	11.00	34.07
TL1S3	17.00	38.30
TL1S4	39.00	35.75
TL2S1	26.50	32.74
TL3S2	17.00	38.08
TL3S4	25.00	30.88
TL3S5	21.67	37.88

4.2.3 Characteristics of slag

The density of the slag which was determined on randomly selected lumps in the laboratory ranged between 1952.42 kg/m³ and 2063.79 kg/m³ with an average density of 1997.86 kg/m³. The unit weight obtained ranged from 19.15 kN/m³ to 20.25 kN/m³ averaging to 19.6 kN/m³. The results are summarized in Table 4.9.

Table 4.9 Density and Unit Weight of Slag Samples

Sample ID	Density (kg/cm ³)	Unit Weight (kN/m ³)
SS2L2	1,952.42	19.15
	1,960.28	19.23
SS1L3	2,037.97	19.99
	2,063.79	20.25
SS2L4	2,010.92	19.73
	1,961.77	19.24
Average	1,997.86	19.60

Particle size distribution curves of the slag samples are presented in Figure 4.5. The slag material was essentially composed of sand-size particles (see Appendix D) with the proportion of gravel that varied from 0 to 13%, and sand from 86 to 98%. The proportion of fines (silt and clay) varied from 0 to 2%. The values of D_{10} , D_{30} , D_{60} allowing to calculate C_u , C_c , and k , are summarized in Table 4.10. The values of C_c and C_u suggested that the slag material is poorly graded. High coefficients of permeability were obtained varying from 1.089×10^{-3} m/s to 9.025×10^{-3} m/s which is significantly higher than the permeability of the tailings.

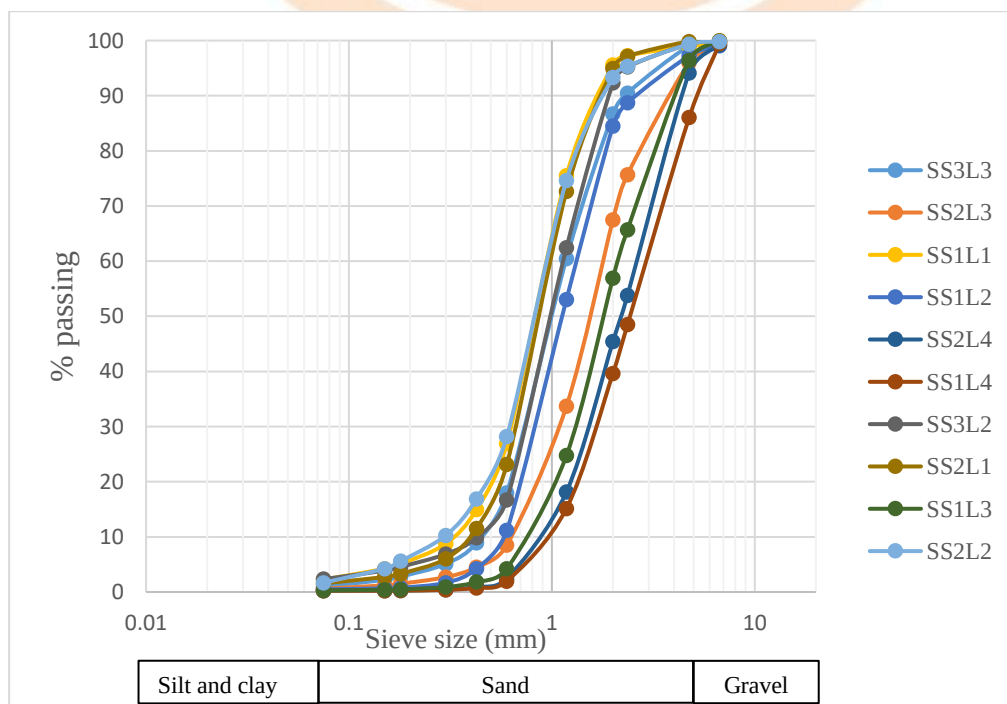


Figure 4.5 Particle Size Distribution Curves of Slag Samples

Table 4.10 Particle Size Distribution Parameters of Slag Samples

Sample ID	D₁₀ (mm)	D₃₀ (mm)	D₆₀ (mm)	C_u	C_c	k (10⁻³m/s)
SS1L1	0.45	0.76	1.17	2.60	1.10	2.025
SS2L1	0.63	1.09	1.82	2.89	1.04	3.969
SS1L2	0.49	0.75	1.08	2.20	1.06	2.401
SS2L2	0.33	0.64	0.99	3.00	1.25	1.089
SS3L2	0.57	0.86	1.36	2.39	0.95	3.249
SS1L3	0.88	1.54	2.73	3.10	0.99	7.744
SS2L3	0.95	1.68	3.09	3.25	0.96	9.025
SS3L3	0.43	0.77	1.15	2.67	1.20	1.849
SS1L4	0.39	0.68	1.03	2.64	1.15	1.521
SS2L4	0.76	1.31	2.13	2.80	1.06	5.776

As stated in Section 3.2.4, the slag samples could not be subjected to the direct shear test because of their coarse granular nature and inadequate size of the shear box equipment. Consequently, the angle of repose was adopted to approximate the friction angle based on Miura *et al.* (1997) conclusion that the angle of repose of granular material was equivalent to the angle of internal friction under extremely low confining stress at the loosest state of the material. It can also be used in determining whether or not a slope (of a stockpile, or uncompacted gravel bank, for example) will likely collapse. It is significant to state that the slag had been dumped in a loose state by tipping from dump trucks and spread over by dozers thus, receiving only minimal densification through vehicular load. Estimation of the friction angle was therefore carried out based on the determination of the angle of repose in the laboratory (see Figure 4.6). The estimated friction angle varied from 39° to 41° with an average of 40°. As a granular material, the slag shear strength was entirely due to friction and therefore, it was assumed to be cohesionless.

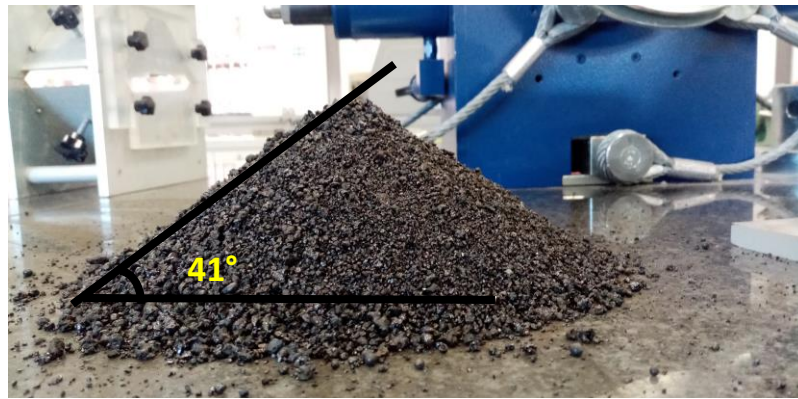


Figure 4.6 Illustration of Slag Friction Angle Determination through the Angle of Repose Method

4.2.4 Characteristics of Soil

The grading analysis for different soil sites done to determine the proportions of the particles in soil samples is summarized in Table 4.11. The particle size distribution curves of the soil samples are presented in Figure 4.7. In general, all the soil samples showed large proportions of sand size (62% to 74%) and a low proportion of gravel size (2% to 6%). Samples obtained from the NW site yielded silt proportion varying from 16% to 21% and low proportions of clay (12% to 13%). For soil samples within the mine and those from the SE, clay proportions respectively were 18% - 21% and 12% - 18%, and silt proportions 3% - 9% and 10% - 11%.

The organic matter (OM) content of samples obtained from the NW site ranged from 1.17% to 1.86%. Organic matter content of samples obtained from within the mine ranged from 1.70% to 3.77%, while samples from the SE site had organic matter content ranging from 1.65% to 2.11%. It is observed from these figures that soil samples from the NW site of the mine had lower organic contents than those from within the mine and the SE site.

Table 4.11 Grain Size Distribution of Soil Samples

Properties		NW Site	SE Site	Within the Mine
Gravel (%)	Min	5	2	2
	Max	6	2	4
Sand (%)	Min	62	69	69
	Max	65	74	74
Silt (%)	Min	16	10	3
	Max	21	11	9
Clay (%)	Min	12	12	18
	Max	13	18	21
Organic matter content (%)	Min	1.2	1.4	1.8
	Max	1.9	3.5	2.1

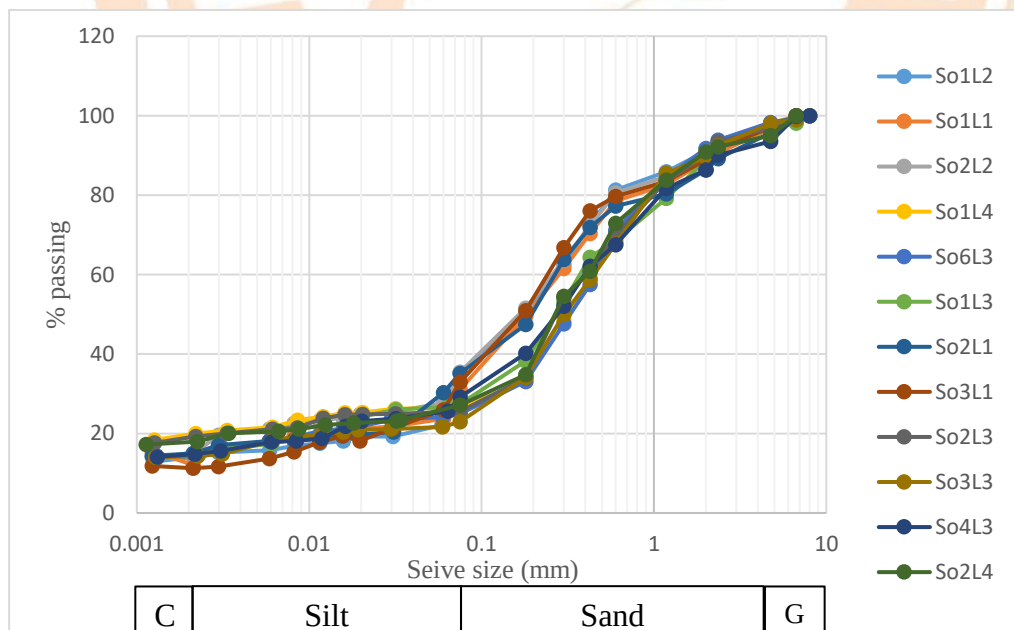


Figure 4.7 Particle Size Distribution Curves of Soil Samples from NW Site, SE Site and within the Mine with C: Clay and G: Gravel

4.3 Mine Waste Dumps Stability Analysis

Table 4.12 summarizes the material properties used in Rocscience Slide2 software for assessment of stability of the tailings dam walls, the waste rock, and slag dumps. The analysis was performed on the slope of the tailings dam wall material above the starter dyke

and slopes of the slag and waste rock dumps. Factors of safety (FS) obtained from Slide software analysis for the scenarios considered are presented in Appendix E.

Table 4.12 Summary of Material Properties Applied

Parameters	Waste Rock	Tailings	Slag
Unit weight (kN/m ³)	26	16	19.6
Friction angle (°)	34	35.4	40
Cohesion (kPa)	277	22.5	0

As mentioned in Section 3.3, the waste dumps will be considered unstable for $FS < 1.25$, critical for $FS = 1.25$ and stable for $FS > 1.25$. Based on the different scenario namely slope height, slope angle, and water level used in the analysis of the dump slopes stability, various FS were obtained for each of the scenario under consideration.

Figure 4.8 is an illustration of Slide2 program analysis output of FS for the tailings dam at slope height of 50 m, slope angle of 40°, and water level (phreatic surface) at 10 m. The value of FS (1.37) suggested that the tailings dam was stable. Figure 4.9 illustrates plots of FS against water levels at various slope heights and angles of the tailings dam from which the following observations were made:

- In Fig 4.9a, where the slope height is 40 m, a stable slope with $FS > 1.25$ was achieved with water level up to 40 m for slope angle of 30°. With increase in slope angle to 35° and 40°, $FS < 1.25$ at water levels below 35 m. The critical factor of safety ($FS = 1.25$) was achieved at water level of 35 m for these slope angles. It is also observed from the figure that a slope of 50° was unstable for all water levels. For a slope angle of 40°, stable slopes were achieved as the water level rose to 32 m.
- In Fig 4.9b where the slope height was maintained at 50 m, for slope angles of 30°, 35° and 40°, the critical factor of safety ($FS = 1.25$) was achieved at water levels of 42 m, 37 m and 29 m respectively. Slopes with angle of 45° and 50° were unstable ($FS < 1.25$) regardless of water level from 10 m to 50 m.
- In Fig 4.9c, a stable slope with $FS > 1.25$ was achieved for a constant slope height of 60 m with water level below 45 m for slope angle of 30°. Slopes with angles of 35° and 40°, resulted in $FS < 1.25$ as the water level exceeded 30 m and 32 m

respectively. An increase in slope angle to 45° and 50° resulted in a $FS < 1.25$ irrespective of the water level.

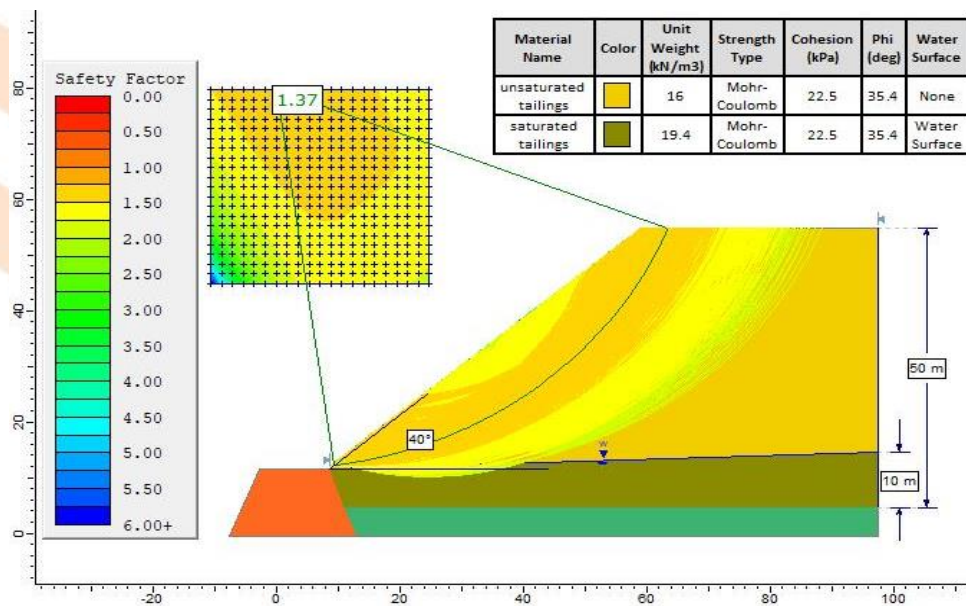
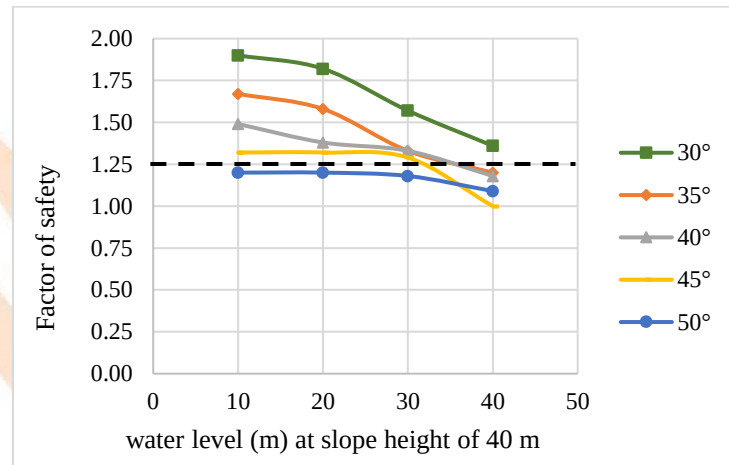
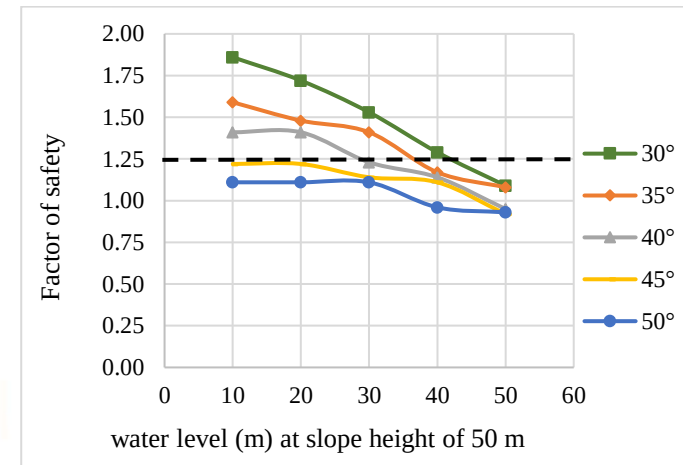


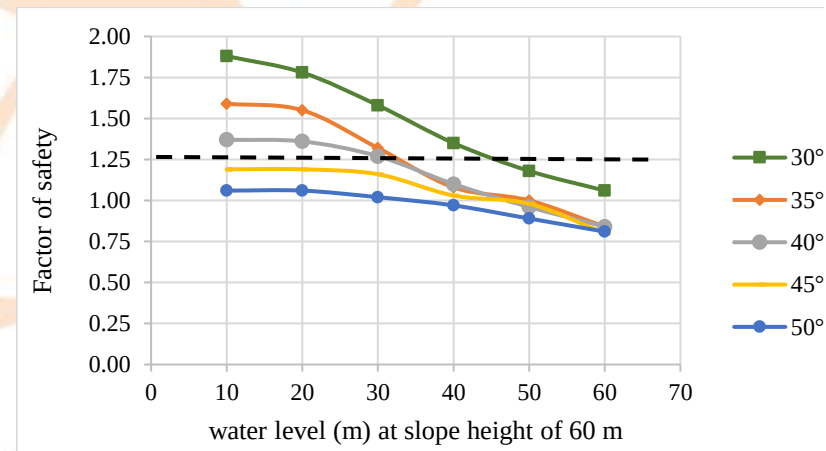
Figure 4.8 Circular Mode Analysis of Tailings Dam with Slope Height: 50 m, Slope Angle: 40° and Water Level: 10 m



(a)



(b)



(c)

Figure 4.9 Factor of Safety versus water level of Tailings Dam a) Slope Height: 40 m, b) Slope Height: 50 m, and c) Slope Height: 60 m

The angle of repose obtained (40°) was approximated to the friction angle of the stability of the slag dump. Figure 4.10 illustrates the computed FS of 1.45 for slope angle of 30°, slope height of 50 m with water level at 20 m. Generally, the slag dump was found to be stable at slope angle of 30°, and water levels only at 10 m and 20 m for slope heights 40 m, 50 m, and 60 m (see Figure 11).

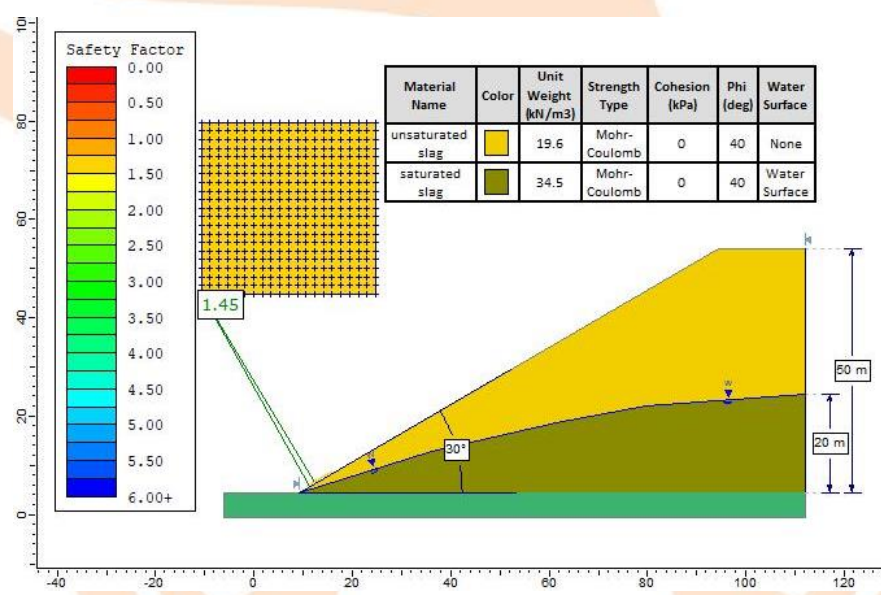


Figure 4.10 Circular Mode Analysis of Slag Dump with Slope Height: 50 m, Slope Angle: 30°, Water Level: 20 m

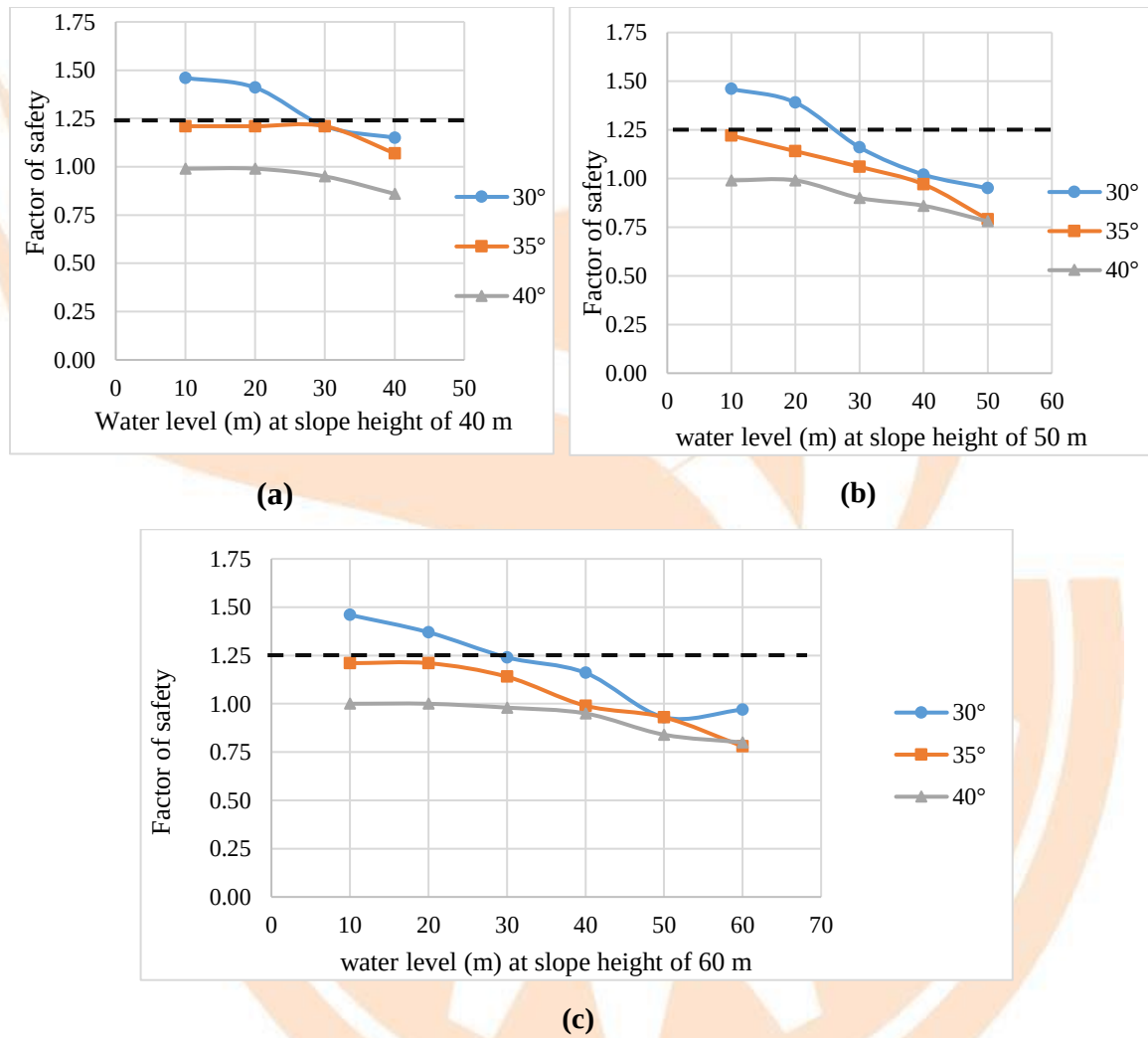


Figure 4.11 Factor of Safety versus Water Level of Slag Dump with a) Slope Height: 40 m, b) Slope Height: 50 m, and c) Slope Height: 60 m

Figure 4.12 displays an example of the determined FS for the waste rock dump. Contrary to the tailings dam and slag dump where the water level was included in the scenarios, no water levels were introduced in the analysis for the stability of the waste rock slopes because of the free draining status as indicated in Section 3.3. The FS obtained, was generally greater than 2.5 (Figure 4.13), thus, suggesting that the waste rock dump was stable.

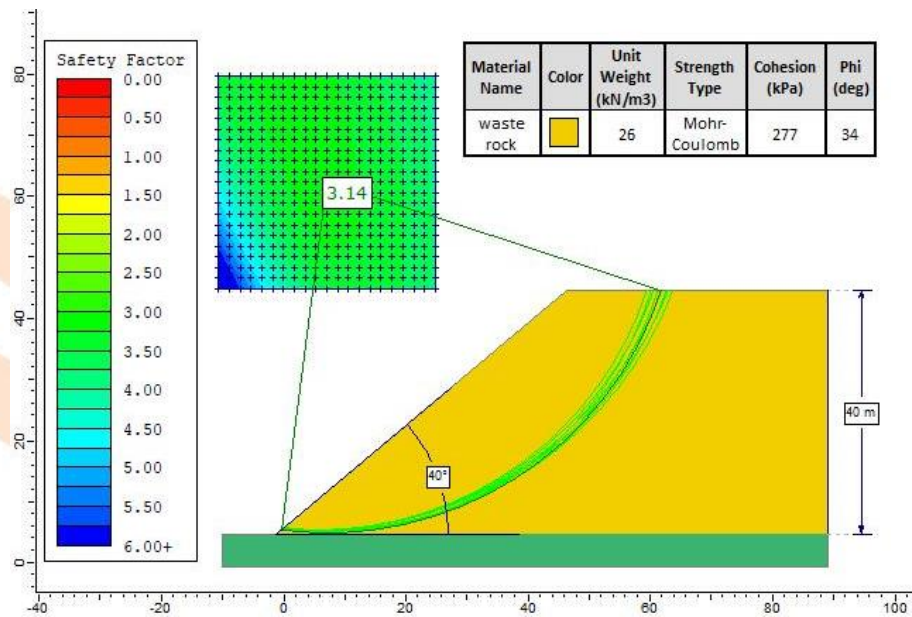


Figure 4.12 Circular Mode Analysis of Waste Rock Dump with Height: 40 m, Slope Angle: 40°

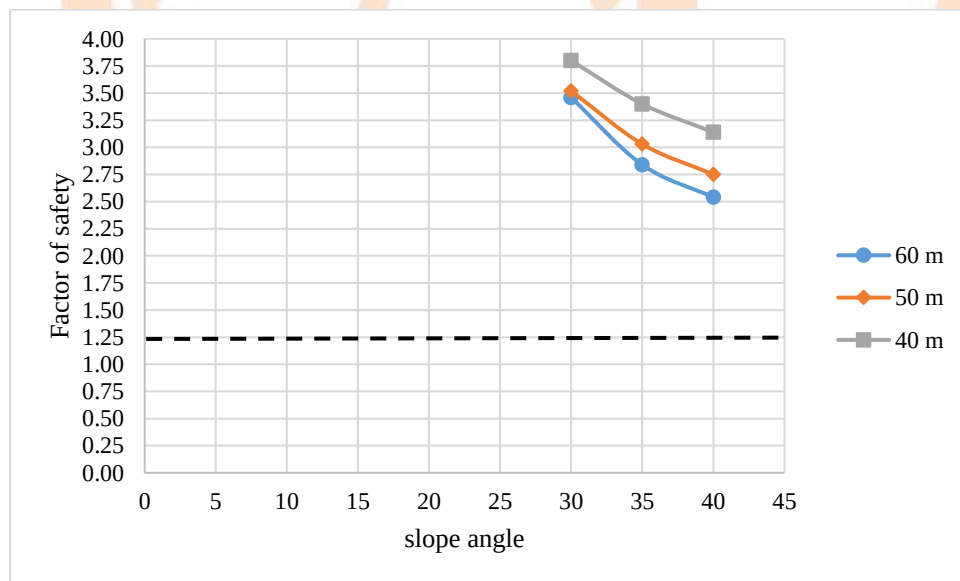


Figure 4.13 Factor of Safety versus Slope Angle of Waste Rock Dump with Slope Height: 40 m, 50 m, and 60 m

CHAPTER 5

MINERALOGICAL AND GEOCHEMICAL STUDIES RESULTS

5 INTRODUCTION

The mineralogical composition of tailings, slag, and the soil samples obtained from the X-ray diffraction are here presented. The chapter also includes the geochemical analyses conducted to determine the heavy metals distribution in the tailings, slag as well as water samples.

5.1 Mineralogical Compositions

5.1.1 X-ray Diffraction Pattern of Tailings Samples

X-ray diffraction analysis on the tailing samples indicated the mineralogical compositions are shown in Figure 5.1. The tailings samples were mainly composed of quartz with the highest peaks visible at 24° and 26° , followed by minor peaks. Sulfide minerals, namely bornite and pyrrhotite, were depicted at 33° and 49° , respectively as their highest peaks. Hematite, mica, actinolite, kaolinite, and calcite appeared as minor minerals.

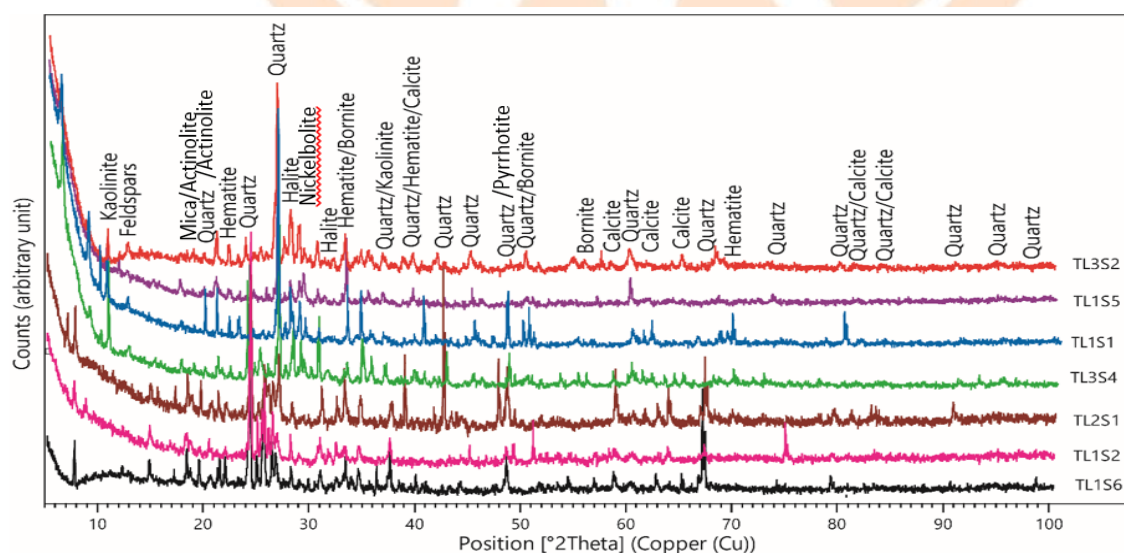


Figure 5.1 X-ray Diffractogram of Tailings Samples

5.1.2 X-Ray Diffraction Pattern of Slag Samples

The different minerals present in the slag samples are shown in Figure 5.2. Quartz was the mineral with the highest peak at 27° . Fayalite and augite were observed at 51° . Halite and

calcite were the next minerals with high peaks at 31°. The minor peaks showed hematite, magnetite, and kaolinite.

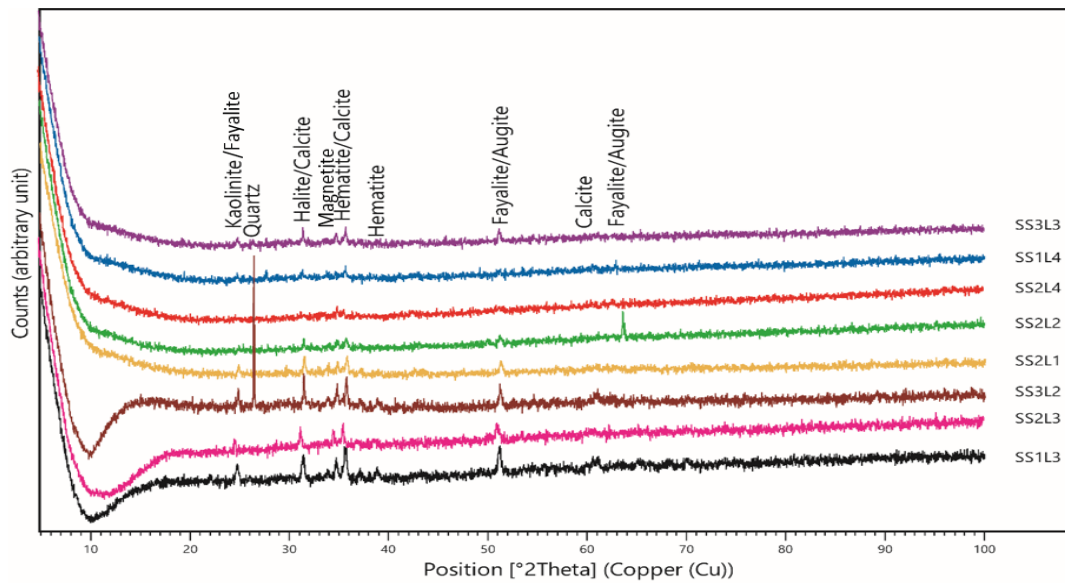


Figure 5.2 X-ray Diffractogram of Slag Samples

5.1.3 X-Ray Diffraction of Soil Samples

The diffractogram of soil samples at the NW site revealed high peaks of quartz, halite, and fluorite appearing at 23° and 27° as indicated in Figure 5.3. Sample So2L2 showed important peaks of hematite at 23° and 37°. However, hematite appeared as a minor peak in samples So1L1 and So2L1. Other minor peaks in these samples were calcite and bornite.

The mineralogical composition of soil samples collected within the mine are shown in Figure 5.4. Quartz and sulfide minerals like pyrrhotite, chalcocite, and chalcocite appeared as major peaks at 21° in So1L3, 29° in So3L3, and So5L3, and 27° in So1L3, respectively. Bornite appeared with a major peak in So1L4. Pentlandite was also observed in So3L4 at 51°. Minor peaks of chalcocite and calcite were visible in So2L4, while hematite is visible in So1L3 and So3L3.

Figure 5.5 displays the minerals present in soil samples obtained from the SE site. Quartz remained the major mineral appearing at 20°, 26°, and 42° with the second-highest peak being kaolinite at 10°. Bornite appeared in all samples at 29° with minor peaks. Hematite, calcite, and halite appeared with minor peaks at 24°, 31°, 56°, 64°, 67°, and 80°.

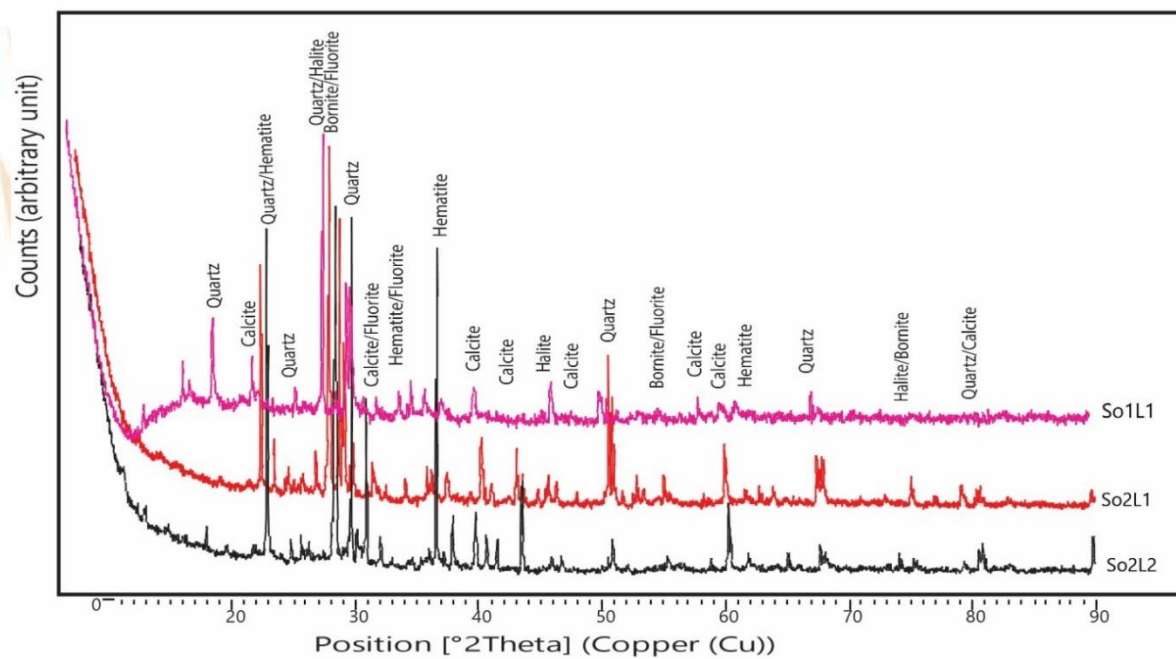


Figure 5.3 X-ray Diffractogram of Soil Samples from NW Site

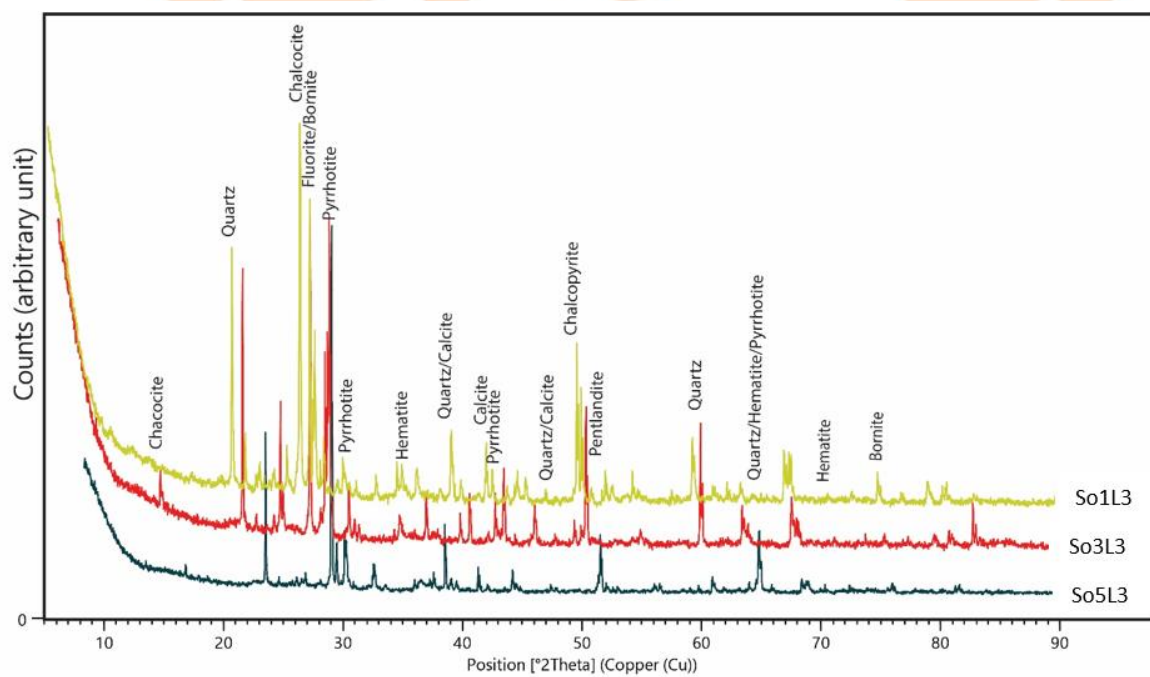


Figure 5.4 X-ray Diffractogram of Soil Samples within the Mine

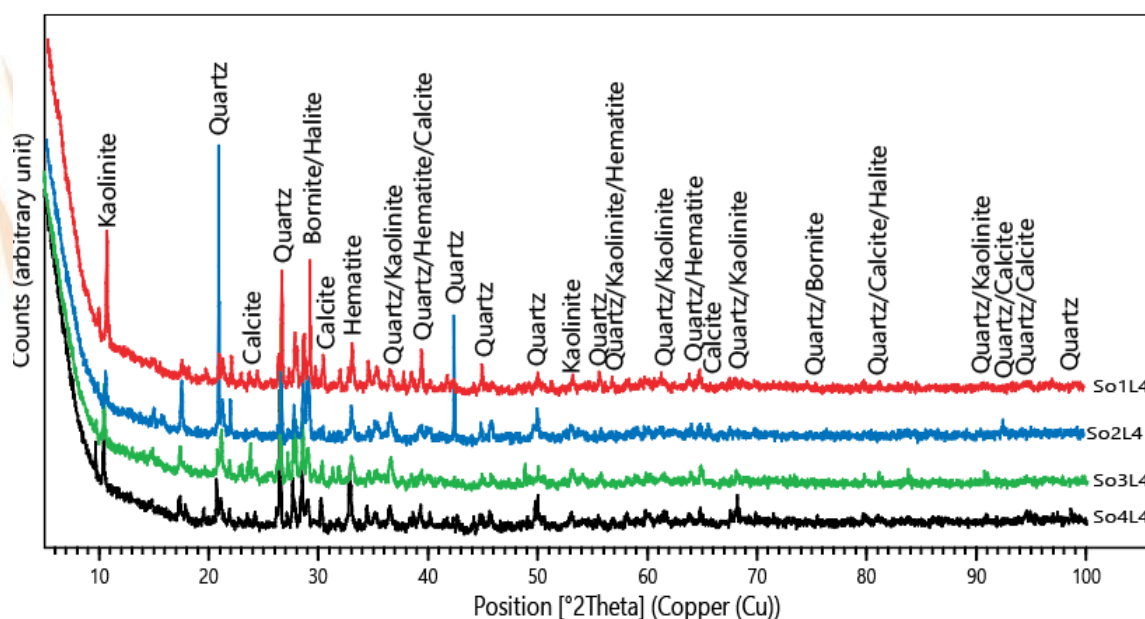


Figure 5.5 X-ray Diffractogram of Soil Samples from SE Site

5.2 Geochemical characterization

5.2.1 Water samples

Physico-chemical parameters

The total dissolved solids (TDS), the electrical conductivity (EC), and the potential hydrogen (pH) obtained from the water samples are shown in Table 5.1. The pH of the Group 1 samples (untreated water samples: WS1a to WS4) varied from very acid to acid with values ranging between 2.8 and 5.3. The conductivity values varied from 3,350 $\mu\text{S}/\text{cm}$ to 6,620 $\mu\text{S}/\text{cm}$ with TDS values varying from 1,640 mg/L to 3,310 mg/L.

The pH of Group 2 samples (treated water samples: WS5a to WS7b) varied between 6.9 and 9.3, indicating neutral to basic conditions. The TDS and the EC in this group of samples, which ranged from 1,890 mg/L to 1,970 mg/L and from 3,790 $\mu\text{S}/\text{cm}$ to 3,960 $\mu\text{S}/\text{cm}$, respectively are low compared to those of the Group 1 samples.

The Botswana Bureau of Standard (BOBS) maximum permissible limits of pH, TDS, and EC are presented in Table 5.1. The pH and TDS values of Group 2 water samples (WS5a to WS7b) fell into the range of BOBS limits, while this was not the case for the Group 1 untreated water samples (WS1a to WS4), where the values were higher. The EC values of the treated water samples were, however, slightly higher than the limits of BOBS with the maximum value of 3,960 $\mu\text{S}/\text{cm}$. For Group 1 water samples, EC values of samples WS1a

and WS1b (samples from the shafts) were slightly higher than the EC limit from BOBS but WS2, WS3, and WS4 (samples from tailings seepage) EC values were twice as high, with the maximum EC being 6,620 $\mu\text{S}/\text{cm}$. Also, the TDS values of tailings seepage water were higher than the BOBS TDS limit.

Table 5.1 Physico-chemical Parameters of Water Samples

Samples ID		pH	EC ($\mu\text{S}/\text{cm}$)	TDS (mg/L)	Location of Water Samples
Group 1 (untreated water samples)	WS1a	5.7	3310	1640	Water from shafts
	WS1b	5.3	3550	1770	
	WS2	2.8	6230	3110	Tailings Seepage water
	WS3	2.8	6460	3240	
	WS4	3.2	6620	3310	
Group 2 (treated water samples)	WS5a	6.9	3870	1920	Treated water from pipes going into the irrigation dam
	WS5b	8.0	3790	1890	
	WS6a	9.3	3950	1970	Treated water within the irrigation dam
	WS6b	9.3	3960	1970	
	WS7a	8.1	3900	1940	Treated water in irrigation outflow going outside the mine
	WS7b	8.0	3890	1940	
Acceptable limits for physical determinants for irrigation water (BOBS, 2011)		6.5-8.4	3000	2000	N/A

N/A: not applicable

Heavy metals distribution

The concentrations (in ppb) of the selected heavy metals (As, Cd, Co, Ni, Pb, Mo) obtained through the ICP-MS are presented in Table 5.2, as well as the maximum permissible limits for irrigation water from the Botswana Bureau of Standards (BOBS, 2011) used as background values.

Table 5.2 Selected Heavy Metal Concentrations (in ppb) of Water Samples with BOBS (2011)**Limit Values**

Samples		As	Cd	Co	Ni	Pb	Mo
Group 1 (untreated water samples)	WS1a	7	1	82	6201	23	< 1
	WS1b	5	1	122	8175	14	< 1
	WS2	3	1	49	928	18	< 1
	WS3	4	N/A	38	814	18	< 1
	WS4	6	1	101	1778	18	< 1
	Average untreated water	5	1	78	3579	18	< 1
Group 2 (treated water samples)	WS5a	5	1	10	707	11	8
	WS5b	6	1	11	735	10	8
	WS6a	6	1	3	154	17	10
	WS6b	7	1	4	157	19	10
	WS7a	7	1	7	624	17	10
	WS7b	8	1	6	591	10	10
	Average treated water	6	1	7	495	14	9
Acceptable limits for heavy metals concentration for irrigation water (BOBS, 2011)		100	10	50	200	2000	10

N/A: not applicable

The heavy metal concentrations (in ppb) of Group 1 water samples (WS1a to WS4) varied as follows: As: 3 to 7, Cd: < 0.5 to 1, Co: 38 to 122, Ni: 814 to 8175, Pb: 14 to 23 and Mo: < 1. For Group 2 water samples (treated samples WS5a to WS7b), the concentrations in ppb were: As: 5 to 8, Cd: 1, Co: 3 to 11, Ni: 154 to 735, Pb: 10 to 19, and Mo: 8 to 10. The heavy metal permissible limits in ppb for irrigation water from BOBS were: As: 100, Cd: 10, Co: 50, Ni: 200, Pb: 2000, and Mo: 10.

The order of abundance of the metals in untreated water samples (Group 1: WS1a to WS4) was: Ni > Co > Pb > As > Cd > Mo, whereas for treated water samples (Group 2: WS5a to WS7b), the order of metal abundance was: Ni > Pb > Mo > Co > As > Cd. Ni remained the

most abundant element in all investigated water samples with the lowest value being 154 ppb observed in WS6a and the highest 8175 ppb in WS4. However, the concentration of Mo was higher in the treated water samples than in the untreated water samples where its concentration was typically below the detection limit.

In comparison with BOBS prescribed values, the average concentrations of the heavy metals for both types of water samples were lower than the permissible limits for As, Cd, Pb. The average concentration of Co for treated water samples was lower (7 ppb) than the permissible limit (50 ppb) but the average concentration of Co for untreated water samples was higher (78 ppb). On the contrary, the average concentration of Ni for all water samples was higher than the permissible limit (200 ppb). The average concentration of Mo was under the detection limit (1 ppb) for untreated water samples, while for treated water samples, the average concentration was almost the same as the permissible limit (10 ppb).

Pollution indices

The computed contamination factor (CF), the degree of contamination (DC), and the pollution load index (PLI) are presented in Table 5.3, Figures 5.6 and 5.7. The average CF results increased in the following order: Pb (0.01) < As (0.05) < Cd (0.07) < Co (1.57) < Ni (17.9) in untreated water samples and Pb (0.01) < As (0.06) < Cd (0.06) < Co (0.14) < Mo (0.93) < Ni (2.47) in treated water samples. The DC values varied from 4.88 to 43.47 in the untreated water samples, with the average being 19.58. On the other hand, the variation in the treated water samples was from 1.94 to 4.81 with an average of 3.68. All PLI values calculated were less than 1.

The Geo-accumulation (Igeo) values were presented in Table 5.4 and Figure 5.8. The Igeo showed negative values for As, Cd, Pb, and Mo in all water samples. Only two treated water samples (WS6a and WS6b) presented negative values for Ni. Treated water samples showed I-geo values varying from 1 to 1.2, while the untreated water samples showed values from 1.4 to 4.8. For Co, most of the samples presented negative values, except a few untreated water samples (WS1a, WS1b, and WS4) with values between 0.1 and 0.7.

Table 5.3 Contamination factor (CF), degree of contamination (DC), and pollution load index (PLI) of selected heavy metals in water samples

Samples		Contamination Factor (CF)						Degree of contamination (DC)	Pollution load index (PLI)
		As	Cd	Co	Ni	Pb	Mo		
Group 1 (untreated water samples)	WS1a	0.07	0.06	1.64	31.01	0.01	-	32.78	0.30
	WS1b	0.05	0.09	2.44	40.88	0.01	-	43.47	0.32
	WS2	0.03	0.06	0.99	4.64	0.01	-	5.73	0.15
	WS3	0.04	-	0.76	4.07	0.01	-	4.88	0.25
	WS4	0.06	0.07	2.02	8.89	0.01	-	11.05	0.23
	Average	0.05	0.07	1.57	17.9	0.01	-	19.58	0.25
Group 2 (treated water samples)	WS5a	0.05	0.06	0.2	3.54	0.01	0.8	4.65	0.14
	WS5b	0.06	0.06	0.21	3.68	0.01	0.8	4.81	0.15
	WS6a	0.06	0.05	0.06	0.77	0.01	1.0	1.94	0.10
	WS6b	0.07	0.07	0.08	0.79	0.01	1.0	2.02	0.12
	WS7a	0.07	0.07	0.15	3.12	0.01	1.0	4.41	0.16
	WS7b	0.08	0.07	0.13	2.96	0.01	1.0	4.24	0.15
	Average	0.06	0.06	0.14	2.47	0.01	0.93	3.68	0.14

-: not calculated (Mo concentration value lower than the detection limit)

Table 5.4 Geo-accumulation Index (Igeo) of Selected Heavy Metals in Water Samples

Samples		As	Cd	Co	Ni	Pb	Mo
Group 1 (untreated water samples)	WS1a	-4.5	-4.6	0.1	4.4	-7	N/A
	WS1b	-4.8	-4.1	0.7	4.8	-7.7	N/A
	WS2	-5.7	-4.6	-0.6	1.6	-7.4	N/A
	WS3	-5.3	Na	-1	1.4	-7.4	N/A
	WS4	-4.7	-4.4	0.4	2.6	-7.4	N/A
Group 2 (treated water samples)	WS5a	-5	-4.6	-2.9	1.2	-8.1	-0.9
	WS5b	-4.7	-4.6	-2.8	1.3	-8.2	-0.9
	WS6a	-4.7	-4.9	-4.7	-1	-7.5	-0.6
	WS6b	-4.4	-4.4	-4.2	-0.9	-7.3	-0.6
	WS7a	-4.5	-4.4	-3.4	1.1	-7.5	-0.6
	WS7b	-4.2	-4.4	-3.6	1	-8.2	-0.6

N/A: not calculated (Mo concentration value lower than the detection limit)

5.2.2 Tailings Samples

Physico-chemical parameters

The pH of the tailings samples indicated very acid to acid values ranging from 3.9 to 5.1. Such values of pH provide a potential for Acid Rock Drainage. The TDS and EC vary from 3,150 to 6,800 mg/l and from 1,560 to 3,370 $\mu\text{S}/\text{cm}$, respectively.

Heavy metals distribution

The results of the concentrations of As, Cd, Co, Cr, Cu, Fe, Mn, Ni, Pb, Zn, and Mo in the tailings samples including the descriptive statistics with the Botswana Bureau of Standards (BOBS, 2011) are presented in Table 5.5. Fe was the metal with the highest concentration. The average concentrations of the heavy metals in descending order were Fe (80000 ppm) > Ni (1,805.3 ppm) > Cu (1,388.4 ppm) > Mn (373.8 ppm) > Cr (138.7 ppm) > Co (97.3 ppm) > Zn (53 ppm) > Pb (14 ppm). Cd, As, and Mo concentrations were below the lower detection limits in most samples. The heavy metals concentration in ppm of the average continental crust according to Wedephol (1995) were: As (1.7), Cd (0.1), Co (24), Cr (126), Cu (25), Fe (43200), Mn (716), Ni (55), Pb (14.8), Zn (65) and Mo (1.1). Concentrations of Mn (373.8 ppm) and Zn (53 ppm) remained below the average continental crust values (716 ppm and 65 ppm, respectively), while Pb content (14 ppm) is similar to the continental crust

value (14.8 ppm). The concentrations of the remaining heavy metals are higher as compared to the average continental crust values.

5.2.3 Slag Samples

Physico-chemical parameters

The pH of slag samples varied between 6.1 and 6.3, with low TDS and EC values ranged from 30 to 50 mg/l and from 80 to 130 $\mu\text{S}/\text{cm}$, respectively.

Heavy metals distribution

Table 5.6 presents the concentrations of the selected heavy metals in the slag samples, together with the average continental crust values and the Botswana Bureau of Standards (BOBS, 2011). The concentration of Cd and Pb was below the lower detection limit (1 ppm) and Fe concentration was higher than the upper detection limit (150,000 ppm). Also, the concentration of Pb in most samples (80% of total samples) was < 2 . Lead concentration was below the lower detection limit in most slag samples. The order of decreasing average concentration values was Fe ($> 150,000$ ppm) $>$ Cu (3,835ppm) $>$ Ni (2,032 ppm) $>$ Co (1,281.7 ppm) $>$ Cr (617.3 ppm) $>$ Mn (361.7 ppm) $>$ Zn (207.4 ppm) $>$ Mo (17.95 ppm) $>$ As (10.8 ppm) $>$ Cd (< 1). The concentrations of all heavy metals were higher than the average continental crust values, except Mn (361.7 ppm).

Figure 5.6 shows the average of Co, Cr, Cu, Mn, Ni, and Zn concentrations of tailings and slag. It clearly shows that Co, Cr, Cu, and Zn concentrations were higher in the slag than in the tailings probably due to the pyro-metallurgical process. Only Mn is slightly higher in the tailings than the slag.

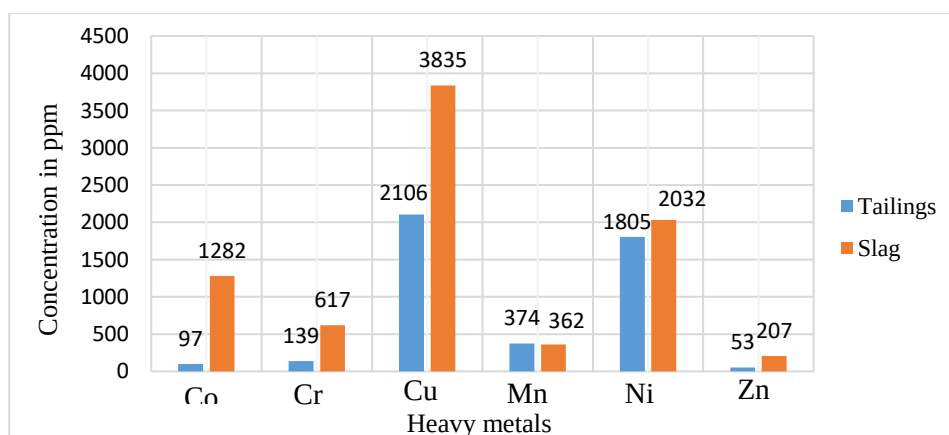


Figure 5.6 Histogram showing Concentrations of Selected Heavy Metals in Tailings and Slag Samples

Table 5.5 Selected Heavy Metal Concentrations of Tailings Samples with the Average Continental Crust and BOBS Values in ppm

Samples	As	Cd	Co	Cr	Cu	Fe	Mn	Ni	Pb	Zn	Mo
TL1S1	8	<1	61.3	106.0	1800	59300	337	1070	10.4	48.9	1.3
TL1S2	<3	<1	18.6	85.7	483	79800	294	222	6.3	29.7	<1
TL1S3	47	2.4	189.0	260.0	6200	102000	325	3650	33.9	79.7	7.2
TL1S4	<3	<1	41.5	76.5	637	60700	1030	886	5.6	37.4	1.3
TL1S5	111	6.3	437.0	466.0	>10000	135000	381	8230	69.0	152.0	16.2
TL1S6	3	<1	56.2	86.5	815	95000	313	857	7.9	36.9	<1
TL2S1	<3	<1	20.7	68.9	434	76200	287	221	7.3	34.9	<1
TL2S2	<3	<1	95.8	131.0	1410	56900	328	1960	6.7	47.6	<1
TL3S1	<3	<1	38.4	54.4	668	100000	136	598	2.6	25.4	<1
TL3S2	<3	<1	109.0	151.0	1640	56400	368	2350	8.7	62.8	<1
TL3S4	<3	<1	79.4	126.0	710	70900	507	1330	4.6	48.3	<1
TL3S5	<3	<1	21.0	52.2	475	54900	179	290	5.6	30.6	<1
Average	-	-	97.3	138.7	1388.4	78925	373.8	1805	14	53	-
Std. Dev.	-	-	117.4	117.6	1640.1	24752	227	2262	19	35	-
Median	-	-	58.8	96.3	710	73550	326.5	978	7	42.5	-
ACCV	1.7	0.1	24	126	25	43200	716	55	14.8	65	1.1
ALIW	100	10	50	100	200	5000	200	200	2000	2000	10

Table 5.6 Selected Heavy Metal Concentrations of Slag Samples with Average Continental Crust and BOBS Values in ppm

Samples	As	Cd	Co	Cr	Cu	Fe	Mn	Ni	Pb	Zn	Mo
SS1L1	10.0	< 1	1130	664	3310	> 150000	454	2510	16.2	403	42.9
SS2L1	7.8	< 1	1390	501	5730	> 150000	354	3340	< 2	166	13.6
SS1L2	14.6	< 1	1390	863	3800	> 150000	378	1810	< 2	184	12.2
SS2L2	10.6	< 1	1400	577	3860	> 150000	331	2090	< 2	152	14.7
SS3L2	6.4	< 1	1350	667	3790	> 150000	332	1860	< 2	165	13.8
SS1L3	31.7	< 1	1680	833	3470	> 150000	356	1740	< 2	171	12.8
SS2L3	9.4	< 1	1250	606	3680	> 150000	324	2060	< 2	159	13.3
SS3L3	9.8	< 1	1350	663	3770	> 150000	326	1950	< 2	161	14.5
SS1L4	4.2	< 1	944	399	3450	> 150000	400	1630	2.5	287	24.3
SS2L4	3.1	< 1	933	400	3490	> 150000	362	1330	< 2	226	17.4
Average	10.8	< 1	1282	617	3835	> 150000	362	2032	-	207.4	17.95
Std. Dev.	8.0	-	227	158	691	-	41	555	-	80	9
Median	9.6	-	1350	634	3725	-	355	1905	-	168.5	14.15
ACCV	1.7	0.1	24	126	25	43200	716	55	14.8	65	1.1
ALIW	100	10	50	100	200	5000	200	200	2000	2000	10

ACCV = Average continental crust values (Wedephol, 1995)

ALIW = Acceptable limits for irrigation water (BOBS, 2011)

-: not calculated (Cd and Pb concentration values lower than the detection limit, and Fe concentration value higher than the detection limit)

Std. Dev.: standard deviation

5.2.4 Soil Samples

Physico-chemical parameters

The pH of the soil samples from the NW site (So1L1 to So2L2) ranged from 4.9 to 5.8. Within the mine, the pH of samples So1L3 to So7L3 was between 5.6 and 7.2, while the pH of samples from the SE site (So1L4 to So5L4) ranged between 4.5 and 5.7. The average pH values, 6.2, 5.2, and 5.1 obtained respectively for samples within the mine, samples from the NW site, and samples from the SE site generally indicated that the soils were acidic. However, samples within the mine were less acid (6.2) than samples from both the NW and the SE sites.

Heavy metals distribution

Results of the heavy metal concentrations in the soil samples have been presented in Table 5.7. Concentrations of Cd, As, and Mo in all samples were below the lower detection limits of the equipment used. The concentrations in ppm for the assessed heavy metals were Co (16.1), Cr (30.4), Cu (213), Fe (25,000), Mn (284), Ni (148.8), Pb (6.8), and Zn (18.8) for the NW site. The concentrations in ppm for the samples within the mine were Co (12.3), Cr (72.9), Cu (139.7), Fe (27,000), Mn (384), Ni (83.1), Pb (6.3), and Zn (43.3). The concentrations in ppm for the SE site soil samples were Co (6.3), Cr (23.8), Cu (42.2), Fe (16,000), Mn (210.2), Ni (38.4), Pb (4.4), and Zn (14.7). In general, the soil samples at the SE site showed low concentrations of Mn, Cu, Ni, Cr, Zn, Co, Pb, and Fe as compared to the samples within the mine and samples collected at the NW site. As observed from Table 5.7, the average concentrations of Cu, Ni, Zn, Co, and Pb for samples at the NW site were higher than samples within the mine, except for Cr, Fe, Mn, and Zn. Regardless of the sample collection site, the overall abundance order is $Mn > Cu > Ni > Cr > Zn > Co > Pb > Fe$.

Pollution indices

Values for the EF presented in Table 5.8 showed significant variation across the sampled areas. Based on the average values, samples at the SE site present the following order of EF: $Cu (4.3) > Ni (1.82) > Pb (0.85) > Mn (0.8) > Co (0.71) > Zn (0.59) > Cr (0.5)$. In the case of samples within the mine, the order of EF was $Cu (9.66) > Ni (2.49) > Zn (1.15) > Cr (1) > Mn (0.87) > Co (0.84) > Pb (0.71)$, while for the NW site, it was $Cu (14.83) > Ni (4.73) > Co (1.17) > Pb (0.8) > Mn (0.7) > Zn (0.5) > Cr (0.42)$.

The CF presented in Table 5.9 indicated values less than 1 for Fe, Cr, Co, Mn, Zn, and Pb across all soil sampling sites. However, the CF for Cu and Ni are 8.52 and 2.7, respectively for samples from the NW site. In the case of the SE site soil samples, only Cu presented 1.69 as a CF value. Samples within the mine yielded CF values of 5.59 and 1.51, respectively for Cu and Ni. Values of the degree of contamination (DC) presented in Table 5.9 and Figure 5.7, were less than 8 for the SE site, whereas the DC values for samples within the mine and the NW site were between 8 and 16. The values of PLI also shown in Table 5.9 and Figure 5.7 were less than 1 for all samples from the different sampling sites.

Table 5.10 and Figure 5. 8 present the calculated Igeo values. Iron, Cr, Co, Mn, Zn, and Pb yielded negative values in all samples, whereas Cu yielded values of 1.39 and 2.49, for samples within the mine and at the NW site, respectively. On the other hand, Ni showed a value of 0.77 for samples from the NW site.

Table 5.7 Selected Heavy Metal Concentrations in Soil Samples with Average Continental Crust Values in ppm

Samples		As	Cd	Co	Cr	Cu	Fe	Mn	Ni	Pb	Zn	Mo
NW site	So1L1	< 3	< 1	20	27.3	232	25,000	316	198	8	19.8	< 1
	So2L1	< 3	< 1	12.3	38.2	158	23,000	279	114	5.8	17	< 1
	So1L2	< 3	< 1	19.3	26.3	243	249,000	300	192	6.3	18.8	< 1
	So2L2	< 3	< 1	12.9	29.6	219	26,000	241	91	6.9	19.5	< 1
	Average NW site	-	-	16.1	30.4	213	25,000	284	148.8	6.8	18.8	-
Within the mine	So1L3	< 3	< 1	15.4	58.9	37.7	27,000	580	90.8	6.3	18	< 1
	So2L3	< 3	< 1	15.7	59.2	102	46,000	574	118	6.7	24.8	< 1
	So3L3	< 3	< 1	7	59.3	81.7	20,000	208	58	4.2	15.4	< 1
	So4L3	< 3	< 1	8.4	58	60	22,000	307	62.2	4.6	19.1	< 1
	So5L3	< 3	< 1	11.5	94	98.1	22,000	295	82.8	6.8	19.2	< 1
	So6L3	< 3	< 1	13.9	95.9	96.3	26,000	371	81.1	9.5	25.5	< 1
	So7L3	< 3	< 1	14.1	84.8	502	24,000	354	88.9	5.9	181	< 1
	Average within the mine	-	-	12.3	72.9	139.7	27,000	384	83.1	6.3	43.3	-
SE site	So1L4	< 3	< 1	6.6	17.1	26.2	12,000	173	24	4.1	10.6	< 1
	So2L4	< 3	< 1	5	19.9	40	14,000	179	41.2	4.2	9.8	< 1
	So3L4	< 3	< 1	9.3	34.4	95.6	22,000	247	72.9	3.1	24.8	< 1

So4L4	< 3	< 1	5.6	28.4	21.5	19,000	233	27.6	5.1	14.9	< 1
So5L4	< 3	< 1	5.2	19	27.5	14,000	219	26.5	5.7	13.2	< 1
Average SE site	-	-	6.3	23.8	42.2	16,000	210	38.4	4.4	14.7	-
ACCV	2	0.1	24	126	25	43,200	716	55	14.8	65	1.1

ACCV = Average continental crust values (Wedepohl, 1995)

∴ As, Cd, and Mo concentration values lower than the detection limit

Table 5.8 Enrichment Factor (EF) of Selected Heavy Metals in Soil Samples with Fe as Metal of Reference

Samples		Fe	Co	Cr	Cu	Mn	Ni	Zn	Pb
NW site	So1L1	1	1.44	0.37	16.04	0.76	6.22	0.53	0.93
	So2L1	1	0.96	0.57	11.87	0.73	3.89	0.49	0.74
	So1L2	1	1.4	0.36	16.86	0.73	6.06	0.5	0.74
	So2L2	1	0.89	0.39	14.56	0.56	2.75	0.5	0.77
	Average NW site	1	1.17	0.42	14.83	0.7	4.73	0.5	0.8
Within the mine	So1L3	1	1.03	0.75	2.41	1.3	2.64	0.44	0.68
	So2L3	1	0.62	0.44	3.86	0.76	2.03	0.36	0.43
	So3L3	1	0.62	1.01	6.99	0.62	2.26	0.51	0.61
	So4L3	1	0.69	0.91	4.76	0.85	2.24	0.58	0.62
	So5L3	1	0.94	1.46	7.71	0.81	2.96	0.58	0.9
	So6L3	1	0.95	1.25	6.33	0.85	2.42	0.64	1.05
	So7L3	1	1.04	1.19	35.55	0.88	2.86	4.93	0.71
	Average within the mine	1	0.84	1	9.66	0.87	2.49	1.15	0.71
SE site	So1L4	1	0.96	0.47	3.65	0.84	1.52	0.57	0.97
	So2L4	1	0.64	0.48	4.9	0.77	2.3	0.46	0.87
	So3L4	1	0.77	0.54	7.58	0.68	2.63	0.76	0.42
	So4L4	1	0.54	0.53	2.01	0.76	1.17	0.54	0.8
	So5L4	1	0.66	0.46	3.35	0.93	1.47	0.62	1.17
	Average SE site	1	0.71	0.5	4.3	0.8	1.82	0.59	0.85

Table 5.9 Contamination Factor (CF), Degree of Contamination (DC) and Pollution Load Index (PLI) of Selected Heavy Metals Soil Samples

Samples		Contamination Factor (CF)								Degree of contamination (DC)	Pollution load index (PLI)
		Fe	Cr	Cu	Co	Mn	Ni	Zn	Pb		
NW site	So1L1	0.58	0.22	9.28	0.83	0.44	3.6	0.3	0.54	15.8	0.84
	So2L1	0.53	0.3	6.32	0.51	0.39	2.07	0.26	0.39	10.78	0.68
	So1L2	0.58	0.21	9.72	0.8	0.42	3.49	0.29	0.43	15.93	0.8
	So2L2	0.6	0.23	8.76	0.54	0.34	1.65	0.3	0.47	12.89	0.69
	Average NW site	0.57	0.24	8.52	0.67	0.4	2.7	0.29	0.46	13.85	0.75
Within the mine	So1L3	0.63	0.47	1.51	0.65	0.81	1.65	0.28	0.43	6.42	0.68
	So2L3	1.06	0.47	4.08	0.29	0.8	2.15	0.38	0.45	9.68	0.8
	So3L3	0.47	0.47	3.27	0.35	0.29	1.05	0.24	0.28	6.42	0.52
	So4L3	0.5	0.46	2.4	0.48	0.43	1.13	0.29	0.31	6.01	0.57
	So5L3	0.51	0.75	3.92	0.58	0.41	1.51	0.3	0.46	8.43	0.72
	So6L3	0.61	0.76	3.85	0.59	0.52	1.47	0.39	0.64	8.84	0.82
	So7L3	0.56	0.67	20.08	0.28	0.49	1.62	2.78	0.4	26.89	1.08
	Average within the mine	0.62	0.58	5.59	0.46	0.54	1.51	0.67	0.42	10.38	0.74

SE site	So1L4	0.29	0.14	1.05	0.21	0.24	0.44	0.16	0.28	2.8	0.28
	So2L4	0.33	0.16	1.6	0.39	0.25	0.75	0.15	0.28	3.91	0.36
	So3L4	0.5	0.27	3.82	0.23	0.34	1.33	0.38	0.21	7.1	0.51
	So4L4	0.43	0.23	0.86	0.22	0.33	0.5	0.23	0.34	3.13	0.35
	So5L4	0.33	0.15	1.1	0.17	0.31	0.48	0.2	0.39	3.12	0.32
Average SE site		0.38	0.19	1.69	0.24	0.29	0.7	0.23	0.3	4.01	0.36

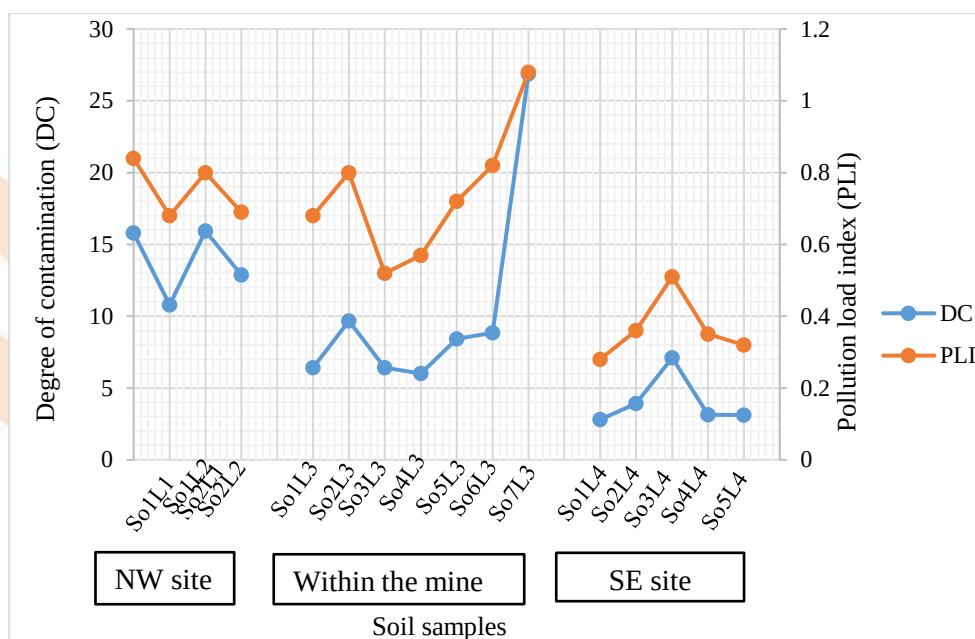


Figure 5.7 Pollution Load Index and Degree of Contamination of Selected Heavy Metals in Soil Samples

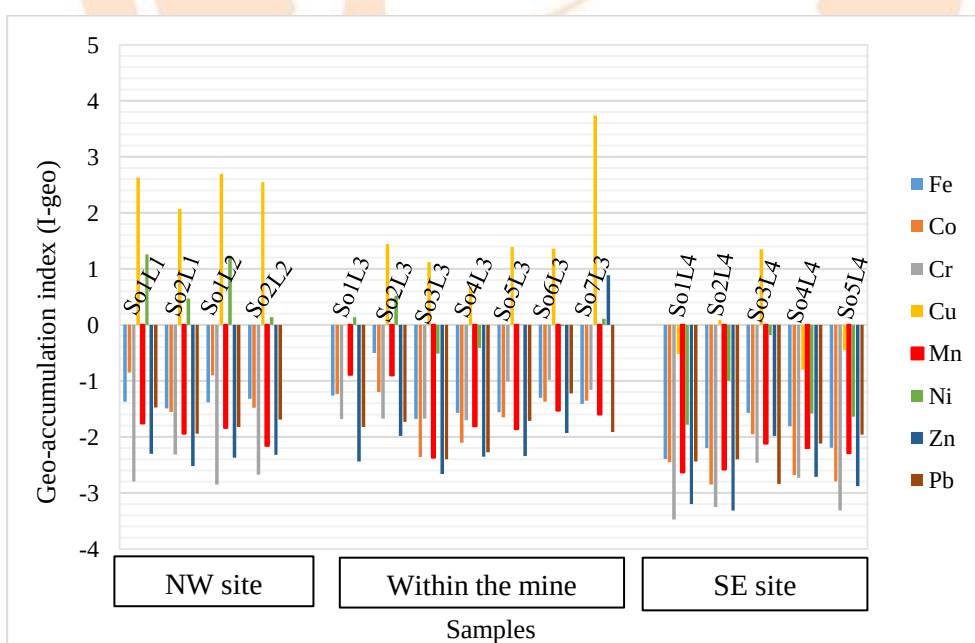


Figure 5.8 Geo-accumulation Index (I_{geo}) of Selected Heavy Metals in Soil Samples

Table 5.10 Geo-accumulation Index (Igeo) of Selected Heavy Metals in Soil Samples

Soil Samples		Fe	Co	Cr	Cu	Mn	Ni	Zn	Pb
NW site	So1L1	-1.37	-0.85	-2.79	2.63	-1.76	1.26	-2.30	-1.47
	So2L1	-1.49	-1.55	-2.31	2.07	-1.94	0.47	-2.52	-1.94
	So1L2	-1.38	-0.90	-2.85	2.70	-1.84	1.22	-2.37	-1.82
	So2L2	-1.32	-1.48	-2.67	2.55	-2.16	0.14	-2.32	-1.69
	Average NW site	-1.39	-1.19	-2.65	2.49	-1.93	0.77	-2.38	-1.73
Within the mine	So1L3	-1.26	-1.23	-1.68	0.01	-0.89	0.14	-2.44	-1.82
	So2L3	-0.50	-1.2	-1.67	1.44	-0.90	0.52	-1.98	-1.73
	So3L3	-1.68	-2.36	-1.67	1.12	-2.37	-0.51	-2.66	-2.40
	So4L3	-1.57	-2.10	-1.70	0.68	-1.81	-0.41	-2.35	-2.27
	So5L3	-1.56	-1.65	-1.01	1.39	-1.86	0.01	-2.34	-1.71
	So6L3	-1.30	-1.37	-0.98	1.36	-1.53	-0.02	-1.93	-1.22
	So7L3	-1.41	-1.35	-1.16	3.74	-1.60	0.11	0.89	-1.91
	Average within the mine	-1.33	-1.61	-1.41	1.39	-1.57	-0.02	-1.83	-1.87
SE site	So1L4	-2.39	-2.45	-3.47	-0.52	-2.63	-1.78	-3.20	-2.44
	So2L4	-2.20	-2.85	-3.25	0.09	-2.58	-1	-3.31	-2.40
	So3L4	-1.57	-1.95	-2.46	1.35	-2.12	-0.18	-1.98	-2.84
	So4L4	-1.81	-2.68	-2.73	-0.80	-2.20	-1.58	-2.71	-2.12
	So5L4	-2.19	-2.79	-3.31	-0.45	-2.29	-1.64	-2.88	-1.96
	Average SE site	-2.03	-2.54	-3.04	-0.06	-2.37	-1.24	-2.82	-2.35

CHAPTER 6

DISCUSSION OF RESULTS

6 INTRODUCTION

This chapter discusses the results of geotechnical and geochemical tests on the rock, soil, tailings, and water samples. It also discusses the stability of the mine waste dumps.

6.1 Effect of Mineralogy and Sample Roughness on Waste Rock Shear Strength Parameters

Due to the heterogeneity of the rocks in the waste rock dumps, the direct shear test was conducted on contact surfaces between different rock types and also between the same rock types. The results obtained showed variations in the cohesion and friction angle depending on the different mineral assemblages of the contact surfaces and the surface roughness of test specimens. The more irregular the shear surfaces were, the higher the frictional resistance especially for quartz-rich rocks, which had moderate to significant asperities on the rough rock specimen surfaces. Brittle minerals such as quartz and feldspar were visibly dominant on the surfaces of the metagranite and metaquartz-monzonite. The shearing of the quartz-rich rock samples with rough surfaces resulted in high friction angles indicating that the shear strength was dependent on the rock-material friction angle, whereas rocks rich in biotite tended to have lower friction angles and high cohesion because biotite has a low friction coefficient. For example, from Table 4.4, the shear strength test between same rock types like metaquartz-monzonite or hornblende-biotite gneiss (with quartz content of 38% and 40%, respectively) resulted in high friction angles of 35° and 41.5°, respectively. However, there was no specific trend with the shearing along contacts between different rock types.

Although the surfaces of saw-cut rock samples were smooth without significant asperities, the friction angles obtained were high (from 39° to 51°) with mostly no cohesion and an isolated cohesion value of 183 kPa (see Table 4.3) signifying that the shear strength along the contact surfaces was defined solely by the rock-material friction angle and the size and shape of the grains. The medium to coarse grained quartz grains might have resulted in low cohesive force of attraction hence the low cohesion.

6.2 Effect of mineralogy and grain size on tailings shear strength

The undrained shear strengths obtained from the vane shear test ranged from 39.7 kPa to 94.8 kPa. The spatial distribution of the shear strength can be explained by the non-homogeneity of the tailings material mainly due to the tailings flow deposition as slurry throughout the dam. This was also visible from the cohesion and the friction angle found from the direct shear test (respectively 11 kPa to 39 kPa and 30.88° to 38.30°). During deposition of the tailings, the solids settle fast, while the water flows to the center of the repository with the finest particles. This segregation is mainly based on the particle density with an increase of the fines fractions (silt and clay) noticeable towards the center of the dam. Thus, indicating the flow direction of the tailings deposition during the operation life of the mine. The dam edges were composed of coarse angular particles which must have inhibited the free movement of the sand-size particles, likely to be the quartz minerals as observed from the XRD. This natural gradation was likely to have reduced the water level within the dam thus resulting in higher strength on the dam edges than the center. As BCL production operations have been inactive since October 2016, the tailings deposition has ceased, thus, allowing the peripheries of the tailings embankment to dry out faster than the center of the dam which still shows presence of water. Similar findings have been reported by Witt *et al.* (2004) where the tailings segregation is related to its permeability and strength as summarized in Figure 6.1.

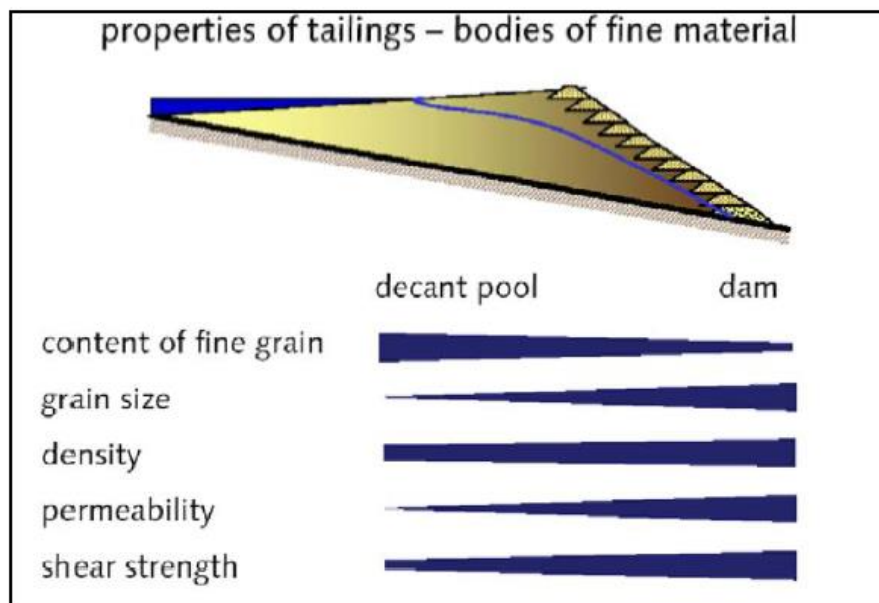


Figure 6.1 Correlation between Tailings Physical Properties within the Dam in Relation to the Deposition Point

The gradation of the tailings was dominated by the sand size with percentages ranging from 58-72%, clay forming 6-20%, and silt content of 11-30%. Coarser particles (sand size particle in this case) are known to be more permeable than fine particles. The permeability (obtained via Hazen's formula) varied from 0.49×10^{-6} m/s to 11.56×10^{-6} m/s, indicating quite medium permeability for the dam. Water infiltration rate through rainfall may increase the pore water pressure. However, the statistics of the annual rainfall of the Selebi-Phikwe area showed maximum annual precipitation of 231.7 mm from 1999 to 2019, which was quite low. This low rainfall pattern could be favorable for the stability of the dam. Therefore, the strength of tailings material may be less affected by water retention. However, continuous and excessive rainfall could trigger stress redistribution within the tailings dam. Consequently, a number of studies (Uchimura *et al.* 2015; Patil *et al.* 2021) have indicated the importance of vegetation on the slope to lower the damages, or construction of retaining walls.

6.3 Effect of Grain Size on Slag Shear Strength

As already stated in Section 4.3.3, the cohesion of the slag is zero due to its coarse sand size. The shear strength consequently was only dependent on its friction angle. The friction angle of such material tends to be close to the angle of repose (natural slope angle during the deposition of the slag). The angularity of the slag grains increases the friction among the particles, hence the 40° obtained was relatively reasonable as friction angle. The permeability which was in the range of 1.089×10^{-3} m/s to 9.025×10^{-3} m/s was quite high to enhance drainage throughout the dump, thus, resulting in a low effect on the shear strength of the slag material.

6.4 Mine Waste Dumps Stability

The different scenarios applied showed the influence of the slope parameters (slope height and slope angle) which highly affected the stability of the mine waste dumps. The tailings dam and the slag dump stability were dependent on the water level with the FS reducing as the water level increased. Since the mine is not operational, the dumping of the mine wastes has ceased and the study area falls within a low seismic activity zone (Simon *et al.* 2012), the dumps are not likely to be adversely affected by any seismic event. The tailings dam is however, likely to be subjected to shear strength reduction through hydration of minerals which results in the change of the physical properties of the material. Internal erosion and

high pore pressure resulting from occasional heavy rainfalls, and formation of discontinuities such as tension cracks that could lead to shear strength reduction would likely affect the stability of the dam. In the case of the waste rock and slag dumps, any reduction in particle interlocking is likely to influence their stability. No slope failure of any of the dumps under study has been reported up to date, however, the stability analysis of the dumps carried out showed the diverse possibilities of failure.

6.5 Degree and Sources of Pollution of Water and Soil

The anthropogenic enrichment and contamination of soil and water of heavy metals were based on the pollution indexes (EF, CF, DC, Igeo, and PLI) determined as detailed in Chapters 2 and 3. The interpretation has been done under the classification defined by Tomlinson *et al.* (1980), Hakanson (1980), Sutherland *et al.* (2000), and Buccolieri *et al.* (2006).

The contamination factor (CF) evaluates the contamination level of a given metal. The water samples showed minimal contamination ($CF < 1$) for As, Cd, Co, Pb, and Mo, except for Ni, which yielded high contamination ($CF \geq 6$) in the untreated water samples and moderate contamination ($1 < CF < 3$) for the treated water samples. The degree of contamination (DC) based on the CF values showed considerable pollution ($12 < DC < 24$) for the untreated water samples, while the treated water showed a low degree of contamination ($DC < 6$). According to Buccolieri *et al.* (2006)'s classification, the Igeo values also showed moderate contamination in the treated water samples and high contamination observable in the untreated water samples. PLI variation for all the water samples denoted low contamination. In general, the indexes showed that the untreated water is contaminated in Co and Ni, whereas the treated water exhibited low contamination in Ni. The exceeding amount of Ni in the untreated water was likely to be attributed to the fact that the composition of the flowing water from the shafts is influenced by the geological substrate of the area rich in pyrrhotite. Also, because the tailings were Ni-rich, the water in contact with tailings will be rich in Ni. Motswaiso *et al.* (2019) found a high concentration of Ni from the Mathathane river, which receives the overflow of the irrigation dam. It was observed that 66.67% of the tailings samples yielded Mo concentrations < 1 ppb while the average Mo concentration in slag was 17.95 ppb. This possibly, could imply that the slag may be the source of Mo in the treated water since the slag dump is close to the irrigation dam (which receives treated

water) than the tailings dam. Nevertheless, further studies may be required to establish its source.

The acidic pH (2.8 to 5.7) obtained from the untreated water samples resulted from the oxidation of sulfide minerals (chalcopyrite and pyrrhotite) present in the tailings causing the acidic outflow drainage. The oxidation of chalcopyrite and pyrrhotite (see chemical reactions in p. 27) as explained by Dold (2010) releases hydroxide iron ($\text{Fe}(\text{OH})_3$). The hydrolysis and oxidation of the latter complex is the main acid-generating product.

Acidic pH for the seepage water obtained in 2003 varied between 1.7 and 2.4 with an average of 2.2 (Schipper and Schwartz, 2005). The high TDS (3,110 mg/l to 3,310 mg/l) and EC values (6,230 $\mu\text{S}/\text{cm}$ to 6,620 $\mu\text{S}/\text{cm}$) from Table 5.1 could be attributed to the turbulent flow of cations present in solutions. After the treatment by adding lime, the water is sent to the irrigation dam through pipes. This possibly, was the main reason for the basic composition of the treated water. Lime is used as an inorganic amendment to immobilize heavy metals like Cd, Cu, Ni, Pb, and Zn (Guo *et al.*, 2006).

The enrichment factor (EF) assesses the anthropogenic enrichment of metal in a given area. The soil samples from the SE site showed minimal enrichment ($\text{EF} < 2$). The CF values indicated low contamination for Fe, Cu, Co, Ni, and Zn but moderate contamination for Cr and Mn. The Igeo results indicated that the site was uncontaminated with Fe, Co, Cr, Cu, Ni, Mn, and Zn ($\text{I-geo} \leq 0$). Also, DC values implied low contamination, whereas PLI values were consistent with no contamination. The overall interpretation lead from zero to minimal contamination on the SE site. The concentration obtained from this site may be from the local background values with zero to very low effect from the mining activities. The minimal enrichment and low contamination of heavy metals may be geogenic. The Pearson matrix showing the correlation between heavy metals in soils is presented in Table 6.1 for the SE site. The inter-heavy metals correlation implied a significant and high positive correlation between all heavy metals except for Pb which showed considerable to high negative correlation. The principal component analysis (PCA) performed to observe the general trend of the heavy metals was done using two principal components which accounted for 94.3% of the total variance. The biplot obtained presented in Figure 6.2a confirmed that Mn, Fe, Cr, Zn, Ni, Cu, and Co correlate due to their narrow angles, while Pb was completely unrelated with Mn due to the right angle, and negatively correlated with

the other heavy metals due to their obtuse angles. The dendrogram from the clustering analysis presented in Figure 6.2b is mainly composed of two clusters that associated only Pb and Mn together to form one cluster and the remaining heavy metals form another cluster.

Table 6.1 Pearson Matrix Correlation of Selected Heavy Metals for SE Site Soil Samples

Variables	Co	Cr	Cu	Fe	Mn	Ni	Pb	Zn
Co	1							
Cr	0.71	1						
Cu	0.87	0.73	1					
Fe	0.70	1.00	0.73	1				
Mn	0.49	0.85	0.50	0.88	1			
Ni	0.81	0.76	0.99	0.77	0.51	1		
Pb	-0.80	-0.48	-0.80	-0.44	-0.02	-0.79	1	
Zn	0.87	0.91	0.86	0.92	0.85	0.83	-0.53	1

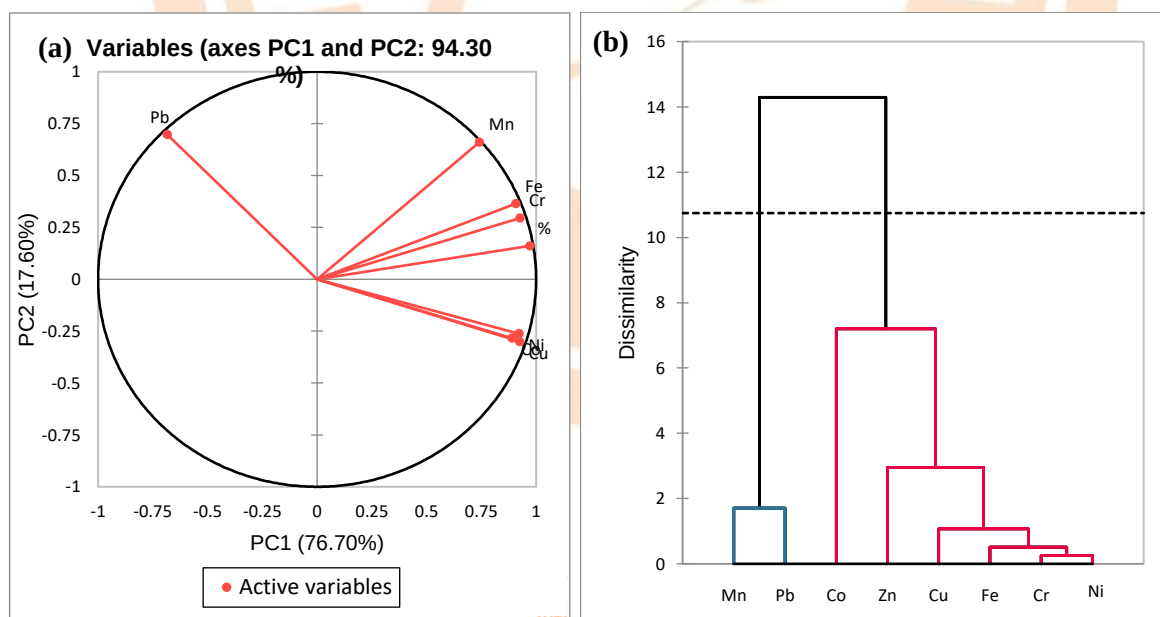


Figure 6.2 Multivariate Analysis for SE Site Soil Samples, (a) PCA variable factor map PC1 (76.7%) and PC2 (17.6%) and (b) Dendrogram with Two Main Clusters, Mn and Pb forming One Cluster and Co, Zn, Cu, Fe, Cr and Ni forming the Second Cluster

The statistical analysis suggested a different provenance (unlikely geogenic) for Mn and Pb. Vehicular exhaust gas tends to release elements like Pb into the atmosphere, even unleaded gasoline (Chrastný *et al.*, 2018), possibly explaining the source of Pb. Mn, as a transition

metal, is classified in the siderophile and lithophile groups and is highly mobile. Thus, Mn was likely leached out from elsewhere and deposited at the SE site.

The enrichment factor of the selected heavy metals for the soil within the mine showed minimal enrichment for Co, Cu, Ni, and Zn and moderate enrichment for Cr and Mn. The CF showed low contamination only for Co, whereas the site was moderately contaminated with Fe, Cu, Mn, Ni, and Zn but highly contaminated with Cr. The site was also weakly polluted with Fe and Zn and varied from unpolluted to moderately polluted with Co, Cr, Cu, Mn, and Ni based on the Igeo values. The DC and PLI values obtained signpost respectively, a considerable degree of contamination and a deterioration of the site. It can therefore be concluded that the soil within the mine was contaminated. Higher concentrations of heavy metals were recorded within the mine as a result of high proportions of clay (18 to 21%) where clay particles can fulfill their metal adsorption capacity. The statistical analysis showed a lack of affinity in Pearson correlation between heavy metals as observed from Table 6.2. The map of the variable factors as shown in Figure 6.3a showed 76.26% of the total variance. This indicated a positive correlation between Cu, Cr, Zn, Pb, Co, Ni, whereas Fe and Mn together almost formed right angles with Cr, Co, and Zn denoting non-correlation. While the clustering analysis presented in Figure 6.3b associated Ni, Fe, Co, Mn in one cluster and Cu, Zn, Cr and Pb formed another cluster. Cu, Ni, and Co have a similar behaviour in the geochemical environment, thus explaining the high correlation between these elements in the PCA variable factor map. On the other hand, in BCL mine, Cu is essentially partitioned in chalcopyrite, whereas Ni preferentially partitions in pyrrhotite and pentlandite, and that may explain the lack of clustering between Cu and Ni. Moreover, the mine was producing a concentrate composed of Cu, Ni, and Co. The main minerals pentlandite and pyrrhotite are Co-bearing minerals and Co was recovered as a by-product when recovering Cu and Ni (Davis, 2000), reason why Co was clustered with Ni. Also, emissions from the smelter and vehicles, and chemical products used by the mine could be sources of heavy metals. Artiola *et al.* (2019) mentioned that smelting and refining processes generate hazardous metals like Co, Cr, Cd, Zn, etc.

Table 6.2 Pearson Matrix Correlation of Selected Heavy Metals Soil Samples within the Mine

Variables	Co	Cr	Cu	Fe	Mn	Ni	Pb	Zn
Co	1							
Cr	0.22	1						
Cu	0.24	0.37	1					
Fe	0.65	-0.27	-0.09	1				
Mn	0.85	-0.29	-0.13	0.77	1			
Ni	0.89	0.04	0.17	0.88	0.83	1		
Pb	0.65	0.69	-0.04	0.28	0.34	0.45	1	
Zn	0.27	0.32	0.99	-0.10	-0.07	0.17	-0.05	1

Values in bold are different from 0 with a significance level $\alpha=0.05$

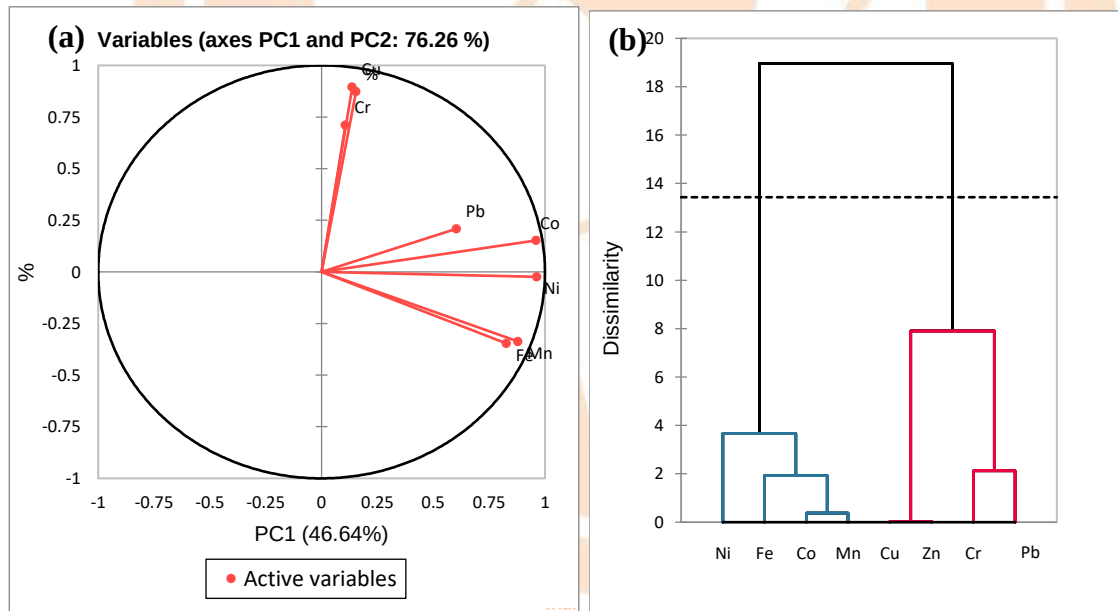


Figure 6.3 Multivariate Analysis for Soil Samples within the Mine, (a) PCA Variable Factor Map of PC1 (46.64%) and PC2 (29.62%) and (b) AHC Dendrogram with two Main Clusters, Ni, Fe Co, and Mn forming one cluster and Cu, Zn, Cr and Pb forming another Cluster

The NW site showed minimal enrichment for Co, Cr, and Zn but indicated moderate anthropogenic enrichment for Cu, Mn, and Ni. The site is weakly contaminated with Zn, moderately contaminated with Fe, Cr, Co, and Mn, but considerably contaminated with Cu and Ni based on the CF values. Also, from the Igeo calculation, the site was unpolluted with Fe, Cr, and Zn, but moderately polluted with Ni and moderately to heavily polluted with Cu, which justified the value of $PLI > 1$ indicating deterioration of the site. The NW site is in the windward direction of the tailings dam and exhibited additional Ni and Cu content which can be explained by the effect of windblown tailings, lixiviation of the sulfide

minerals, and geochemical processes. Also, previous studies (Likuku *et al.*, 2013) found the NW site to be subjected to particulate air matter (PAM) caused by the concentrator at the time the mine was operating. Thus, the presence of heavy metals may be attributed to different sources; geogenic and anthropogenic. Pearson correlation of samples from the NW site presented in Table 6.3 showed moderate to high correlation between Co with Cu, Mn, Ni, Pb, and Zn. Also, moderate to high positive correlation was noticeable between Cu with Fe, Ni, Pb, and Zn, then Pb with Fe, Co, Zn, Mn, and Ni. Chromium, on the other hand, was moderate to highly negatively correlated with the other heavy metals. It is visible in the variable factor map shown in Figure 6.4a that the PCA was concordant to the heavy metals correlation. The cluster analysis symbolized by Figure 6.4b indicated two possible sources of heavy metals as per the two main clusters from the dendrogram. On one hand, Pb, Mn, Fe, and Zn form a cluster, and Cu, Co, Ni, and Cr form another cluster on the other hand.

Table 6.3 Pearson Matrix Correlation of Selected Heavy Metals for NW Site Soil Samples

Variables	Co	Cr	Cu	Fe	Mn	Ni	Pb	Zn
Co	1							
Cr	-0.79	1						
Cu	0.77	-1.00	1					
Fe	0.52	-0.51	0.54	1				
Mn	0.84	-0.32	0.30	0.33	1			
Ni	0.97	-0.63	0.61	0.53	0.93	1		
Pb	0.57	-0.62	0.59	-0.31	0.34	0.43	1	
Zn	0.55	-0.87	0.85	0.02	0.09	0.35	0.86	1

Values in bold are different from 0 with a significance level $\alpha = 0.05$

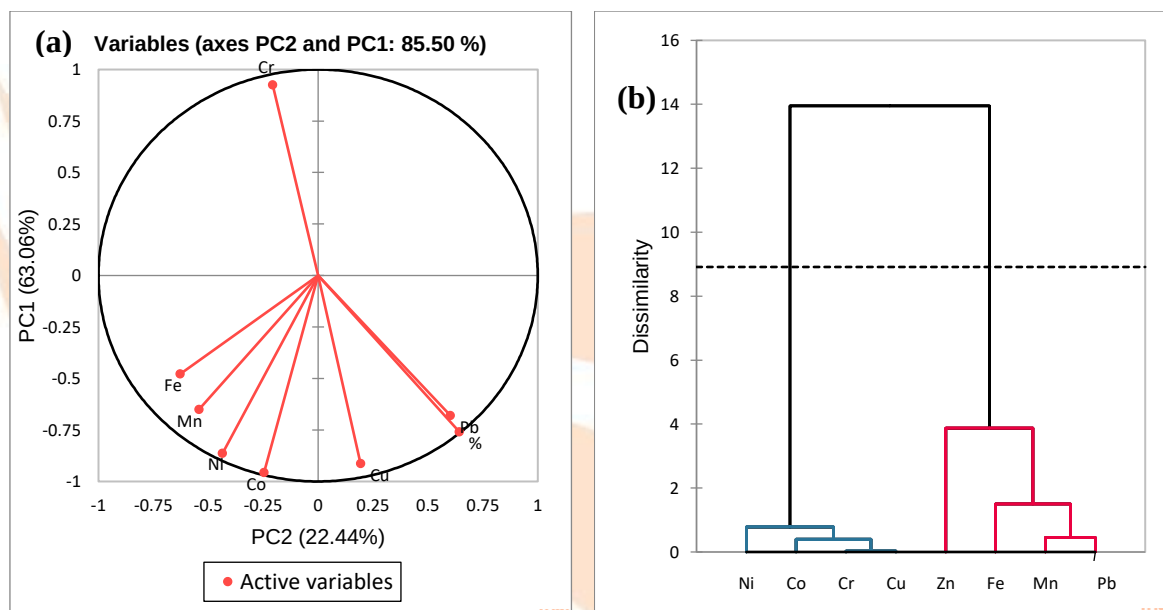


Figure 6.4 Multivariate analysis NW Site Soil Samples, (a) PCA Variable Factor Map of PC1 (63.06%) and PC2 (22.44%), (b) HCA Dendrogram with two main Clusters where Ni, Co, Cr, and Cu form one cluster and Zn, Fe, Mn, and Pb form another Cluster

6.6 Environmental Implications of the Pollution

The XRD results obtained from the tailings and the NW site soils denoted the presence of bornite and pyrrhotite, whereas the soil within the mine showed the presence of pyrrhotite, bornite, chalcocite, pentlandite, and chalcopyrite. Although pyrite was not detected by the XRD analysis, it was identified as one of the principal minerals in the study area (Sracek *et al.*, 2018). Wakefield (1976) however, estimated the sulfide minerals in the orebody was approximately 1% of pyrite, against 84% pyrrhotite, 8% pentlandite, and 8% chalcopyrite. Chalcopyrite is classified as the more resistant sulfide mineral, and hence less harmful than pyrite and pyrrhotite as it needs 5 moles of water for its fractionation. Pyrrhotite is 20 to 100 times more easily oxidizable than pyrite when the atmospheric conditions are reunited. Its reaction produces marcasite (Jambor, 1994), having the same chemical formula as pyrite, hence sharing the same chemical reaction. Also, Schippers *et al.* (2007) from a geomicrobiological investigation in the study area, reported the presence of a microorganism (*acidithiobacillus ferrooxidans*) which accelerates the hydrolysis and oxidation of sulfide minerals

From equations 6.1 and 6.2, one of the consequences of AMD from the oxidation/hydrolysis of pyrrhotite and chalcopyrite is the acidity of the environment due to the release of H^+ ions as observed from the tailings seepage water. The overall pH values indicated very acid to acid tailings, soil, and the seepage water samples with high TDS and EC and tend to be

neutral for the slag samples with low TDS and EC. At low pH, there is high solubility and mobility of some heavy metals becoming more bioavailable than in a neutral or alkaline pH, while others are apt to create insoluble sulfates. For instance, Cu, Zn, Cd, are mobile and available at low pH (Serpaud *et al.*, 1994) contrary to As, Mo, and Pb which are less mobile (Serpaud *et al.*, 1994; Perrono, 1999).

The tailings seepage water is collected in a trench and directed through pipes to the treatment plant for treatment. The concern appears to be the downward seepage from the trench which could affect the groundwater. Though soil samples were not collected at different depths, the vertical distribution of heavy metals can be estimated through the PSD of the soil samples. Proportions of clay from soil within the mine and SE site were higher than those from the NW site. Clay particles due to their large surface area can adsorb metals, reducing in this case, its migration downward. Therefore, the soil within the mine and the SE site are likely to reduce the contamination rate with depth, unlike the NW site. Also, the different soil sites appeared not to be highly leached probably due to the low precipitation and presence of vegetation in places.

Low acidity, TDS, and EC values were obtained for the slag samples. This could be explained by the high presence of kaolinite and calcite, which are acid-neutralizing minerals, obtained from the XRD. Akoto *et al.* (2016) stated that there is sorption of heavy metals at alkaline pH making them less mobile and favoring the precipitation of hydroxides and carbonates. Also, slag was cited as an inorganic amendment that immobilizes Cd, Pb, Cr, and Zn (Guo *et al.*, 2006). Moreover, no presence of sulfide minerals was detected in the slag samples, thus the tailings were likely to be the main source of AMD and pollution of the NW site.

6.7 Health Implications and Toxicity of Heavy Metals

The fine particles of sulfide-bearing tailings as illustrated by the gradation analysis make them more readily available in the environment because the smaller the particles, the faster they react in adequate conditions and can easily be inhaled and ingested as well. Although the slag material contained more contaminants (see Figure 5.6), the coarse size of the particles makes it less harmful than the tailings as it requires more water moles to break them down. The low precipitation rate in Botswana acts favorably in this case.

As aforementioned, in an acidic environment, some heavy metals are more bioavailable and mobile impacting negatively the ecosystem and human health. Table 6.4 (from Rambabu *et al.*, 2020) summarizes the selected heavy metals that have adverse effects on human beings and animals.

Table 6.4 Heavy Metals Toxicity

Heavy Metals	Targeted Human Organs	Ecosystem Effect
Arsenic (As)	Kidney, bronchitis, bones	Growth inhibition, food chain poisoning
Cadmium (Cd)	Kidney, lungs	Lower germination
Nickel (Ni)	Nose, lungs	Prevent germination, enzyme, and protein production
Copper (Cu)	Liver, kidney	Reproduction and photosynthesis inhibition
Lead (Pb)	Mental organ	Chlorophyll production and plant growth prevention
Manganese (Mn)	Nervous system	-
Chromium (Cr)	Liver, kidney	Root damage and low plant growth
Zinc (Zn)	Nervous membrane	-

(Source: Rambabu *et al.*, 2020)

CHAPTER 7

CONCLUSIONS AND RECOMMENDATIONS

7.1 Conclusions

The present study focused on the environmental assessment of mine waste dumps of BCL copper-nickel mine. The stability analysis of the mine waste dumps and the potential contamination from the mine wastes via pollution indices calculation were assessed. The factor of safety for each mine waste dump was computed using the deterministic method with input data derived from geotechnical tests. The following conclusions are highlighted based on the results:

- The unit weight and average strength parameters of the tailings from direct shear test were respectively 16 kN/m^3 , cohesion (C) = 22.5 kPa and friction angle (Φ) = 35.4° .
- Based on the Simplified Bishop method of slope stability analysis:
 - The various scenario adopted for the analysis indicated that the most stable slopes of the tailings with $FS > 1.25$, were achieved when the slope angle was 30° and the slope height varied from 40 – 60 m with maximum water level of about 40 m.
 - At 35° slope angle, the tailings dams of height 40, 50 and 60 m were stable with $FS \geq 1.25$ when the height of water level was between 30 and 35 m.
 - The tailings with slope angle of 50° were most unstable with $FS < 1.25$ irrespective of water level and dam height from 40 - 60 m.
- The average strength parameters of the rocks obtained from the direct shear test were cohesion (C) of 289 kPa and friction angle (Φ) of 35° . Based on these average strength parameters, the waste rock dump is estimated to remain stable for height between 40 m to 60 m with slope angle from 30° to 40° .
- As a cohesionless material, the slag strength is mainly dependent on the friction angle, estimated to be 40° from the angle of repose method. The slag dump was found to be stable at slope angle of 30° , and water level at 10 m and 20 m.
- From the pollution indices, the SE site was moderately contaminated in Cr and Mn with minimal anthropogenic enrichment and the NW site with soil within the mine were contaminated by Cu and Ni, and Cr respectively with moderate anthropogenic enrichment.

- The Slag contained higher concentrations of Ni, Cu, As, Co, Cr, Fe, and Zn nonetheless based on the PSD and the mineralogy, the tailings were found to be the main source of pollution.
- The tailings mineralogy was mainly composed of sulfide minerals promoting an acidic environment where heavy metals are more bioavailable and mobile.
- Though the untreated water denoted moderate and very high contamination in Co and Ni respectively, the treatment process applied by the mine resulted in moderate contamination risk in Ni.

7.2 Recommendations

Due to the constant improvement of technology to provide more accurate geotechnical results, it is recommended that future work should include:

- Application of different laboratory testing methods such as triaxial test along with other numerical models such as finite element for a wide critical view of the stability of the dumps.
- Use of piezometers or the application of geophysical methods to localize with certitude the water level in the tailings dam for better stability analysis.

As acid mine drainage and soil contamination from mining deeds are rising, there is growing attention to its remediation worldwide. The following recommendations can be applied to mitigate its negative impacts:

- Continuous monitoring of heavy metals concentrations to elaborate on the acid generation rate.
- Construction of pollution control bund walls around the waste dumps,
- Biological treatment through the usage of bioreactors and bacteria results in sulfate reduction.
- Phytoremediation treatment using green plants, by either phytostabilization where the heavy metals are less available hence immobilized or phytoextraction where the heavy metals are used through the plant metabolism to make them part of their biomass.
- Government attention on mine regulations (229 and 231) from law Act. (1978).

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APPENDICES

APPENDIX A

ADDITIONAL THIN SECTION IMAGES IN CROSS-POLARIZED LIGHT AND PLANE POLARIZED LIGHT

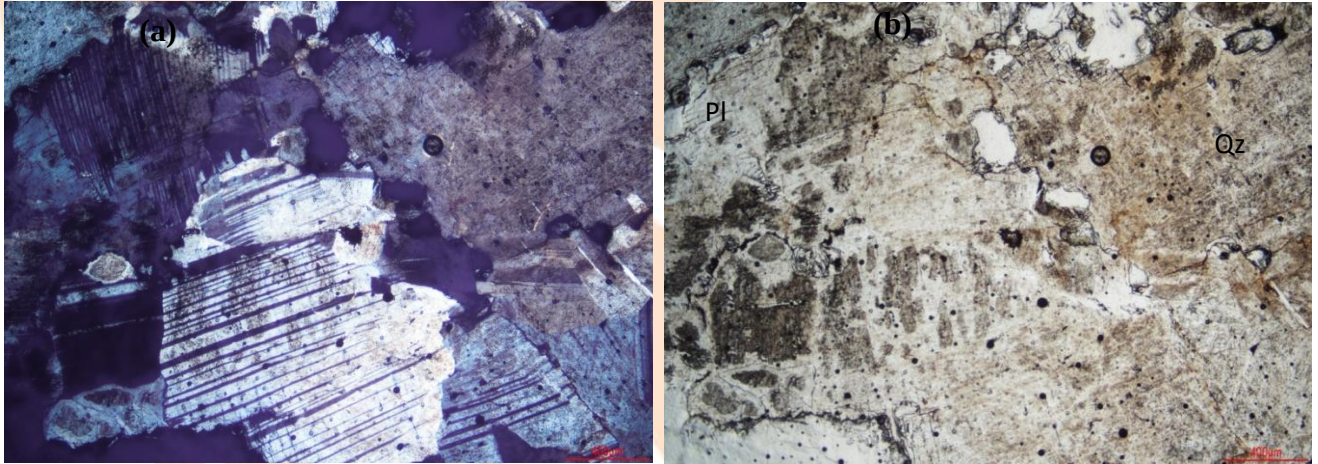


Figure A.1 Thin section image of metaquartz-monzonite with Pl= plagioclase, Qz = quartz, Bt = biotite, (a) Cross-polarized light; (b) plane polarized light.

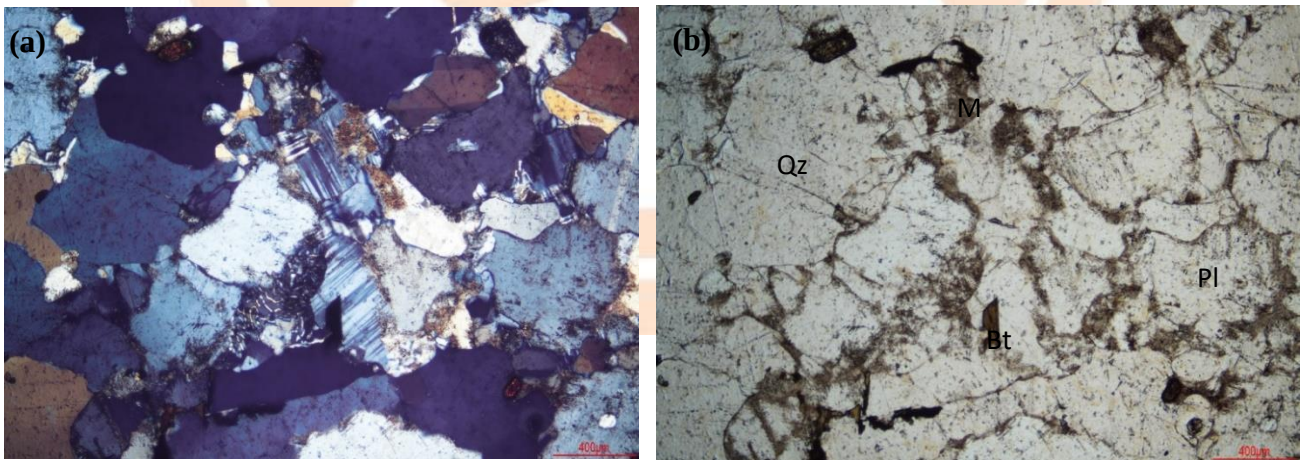


Figure A.2 Thin section image of metagranite, with Pl = plagioclase, Qz = quartz, Bt = biotite, M = microcline; (a) in cross-polarized light, (b) plane polarized light

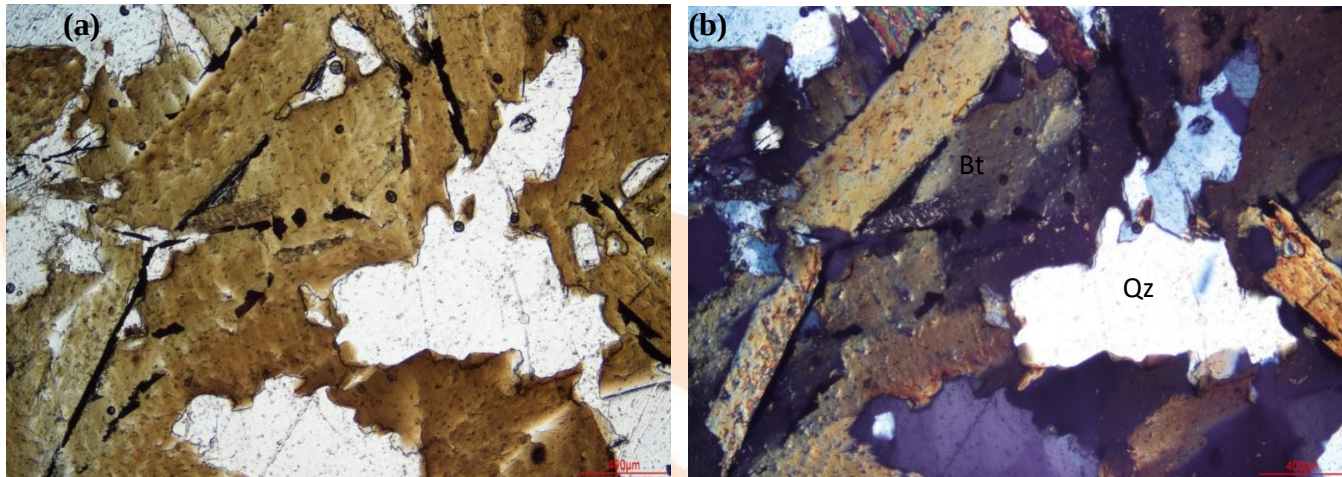


Figure A.3 Thin section image of biotite schist, with Qz = quartz, Bt = biotite, (a) cross-polarized light, (b) plane polarized light.

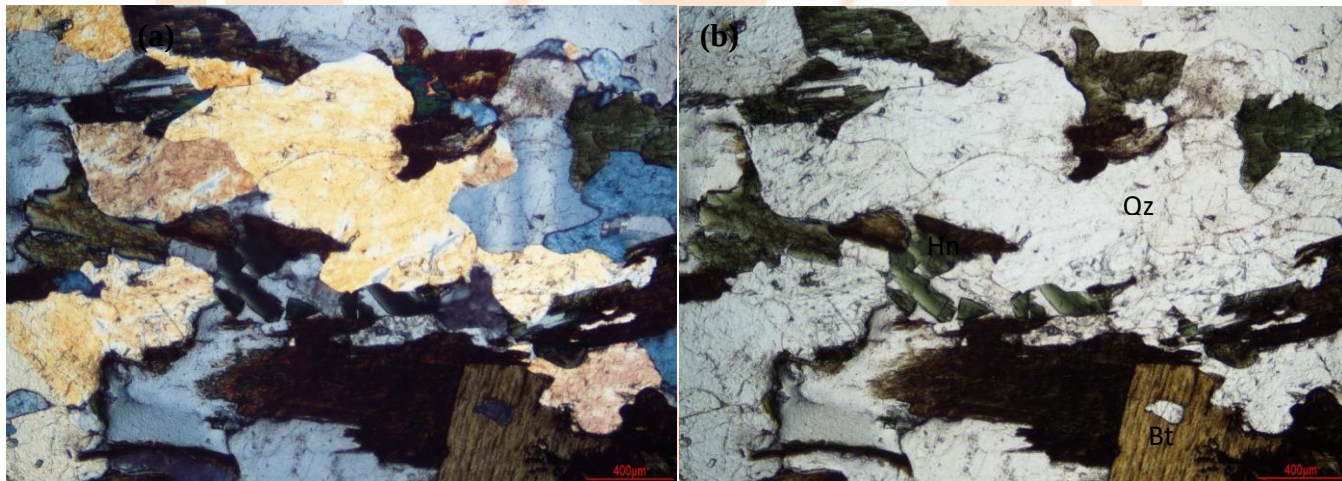


Figure A.4 Thin section image hornblende-biotite gneiss with Qz = quartz, Bt = biotite, Hn = hornblende; (a) Cross-Polarized Light, (b) Plane Polarized Light.

APPENDIX B

SHEAR STRESS VERSUS SHEAR DISPLACEMENT/MOHR-COULOMB FAILURE ENVELOPE PLOTS FOR ROCK SAMPLES

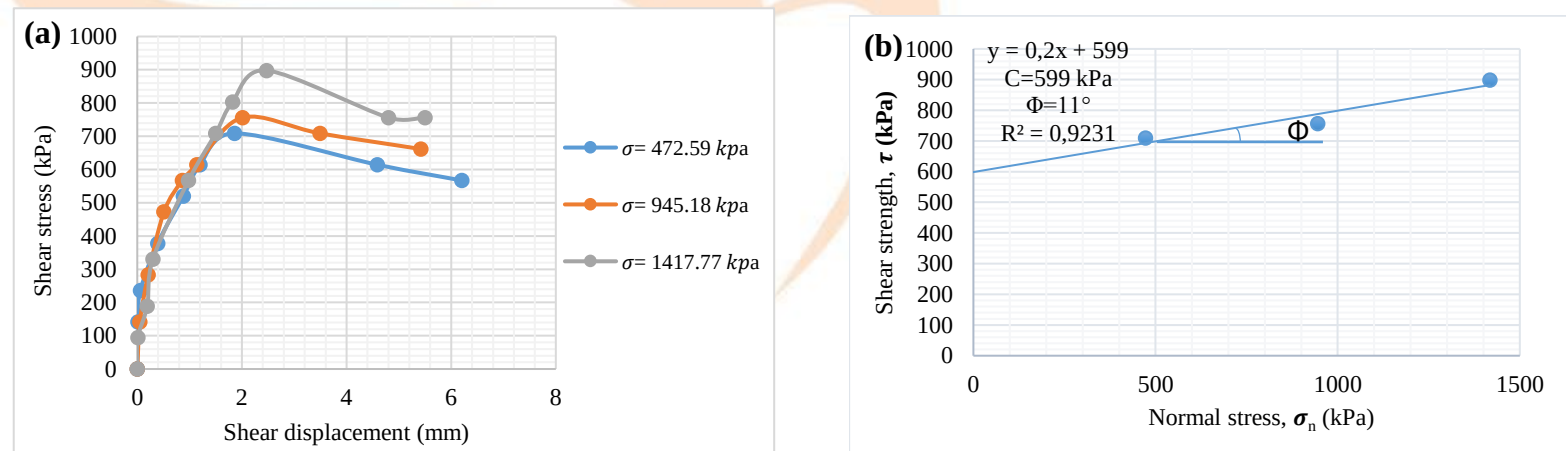
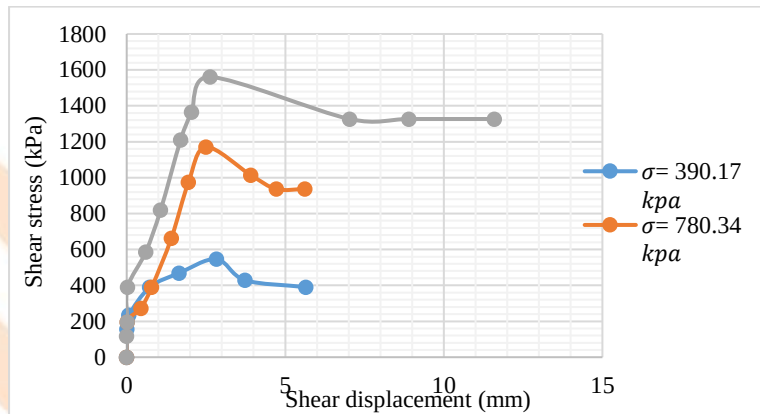
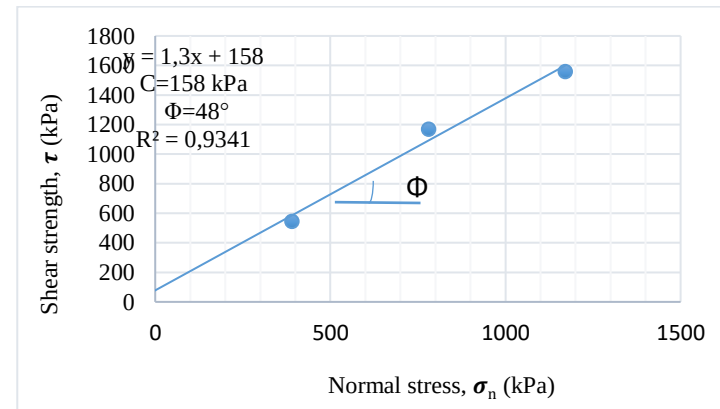


Figure B.1 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for biotite Schist

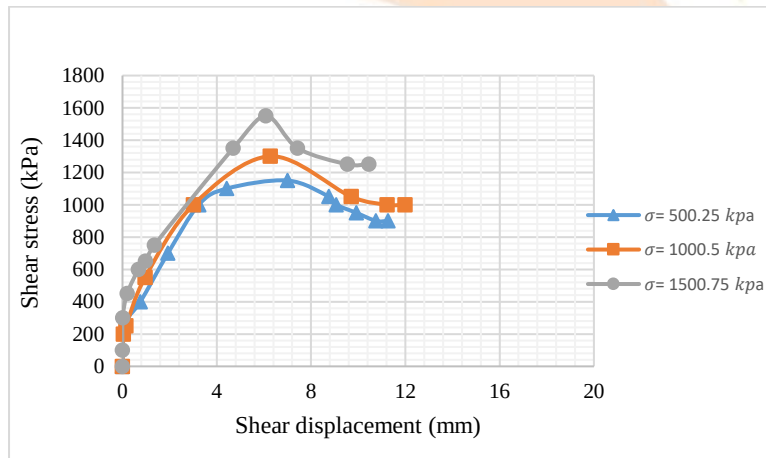


(a)



(b)

Figure B.2 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for hornblende-biotite gneiss



(a)

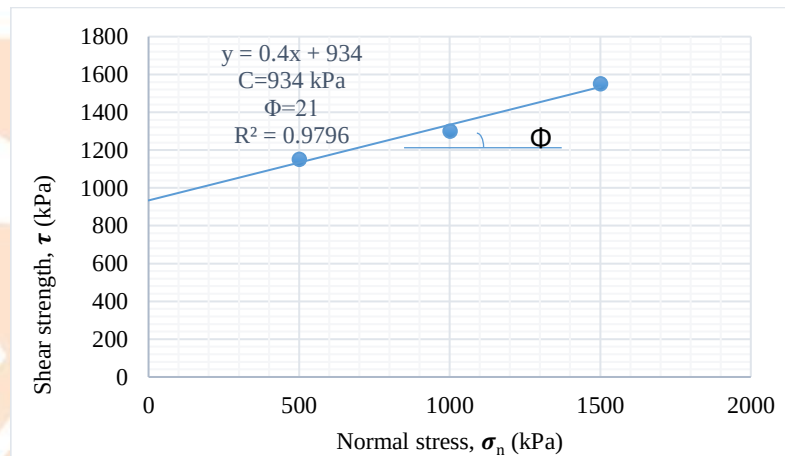
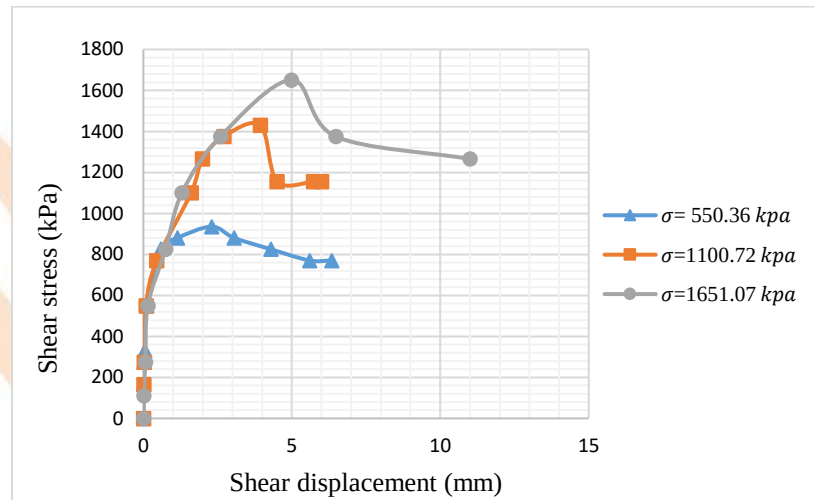
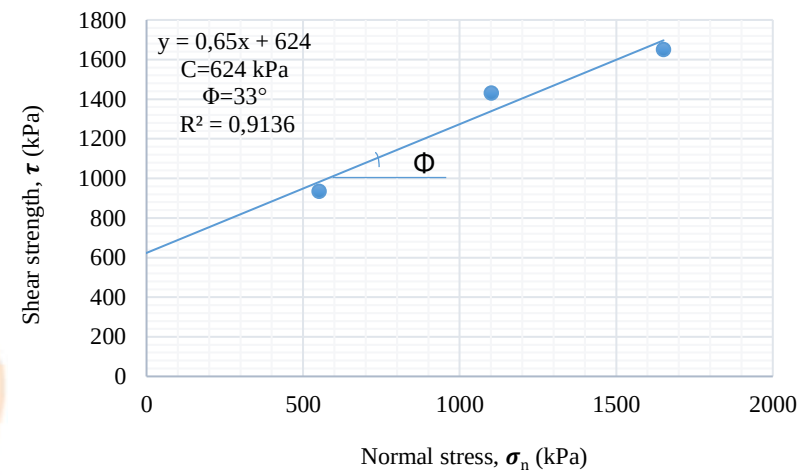


Figure B.3 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb Failure Envelope for biotite schist



(a)



(b)

Figure B.4 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for metagranite

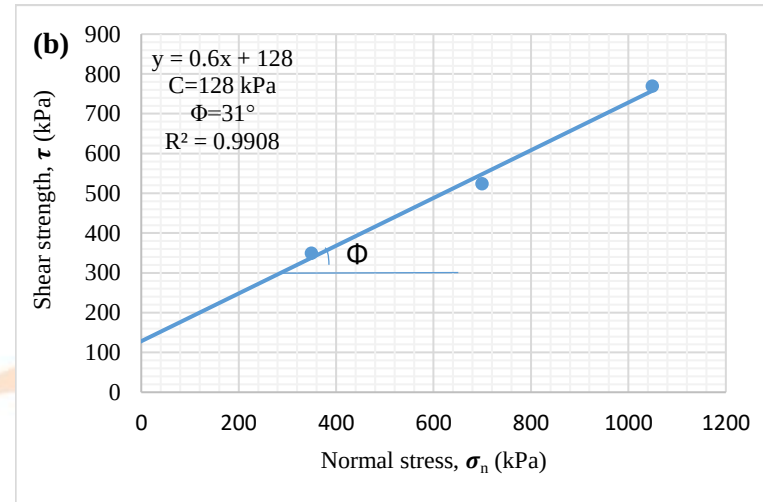
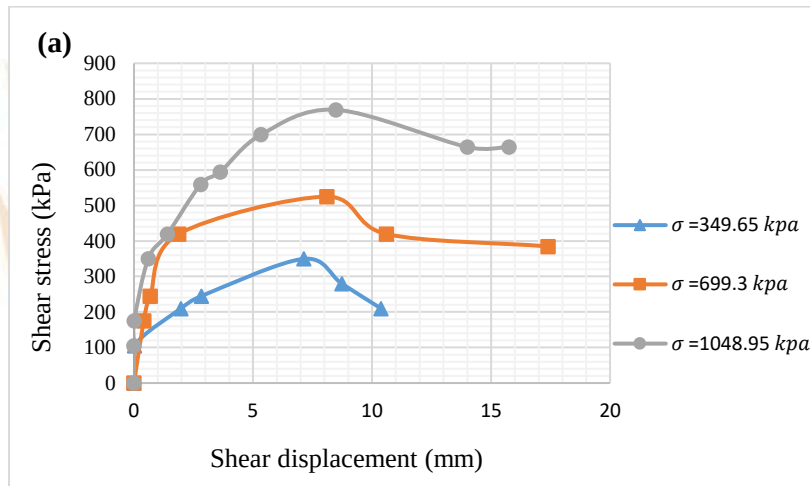


Figure B.5 (a) Shear stress versus Shear displacement, (b) Mohr-Coulomb failure envelope for metaquartz-monzonite

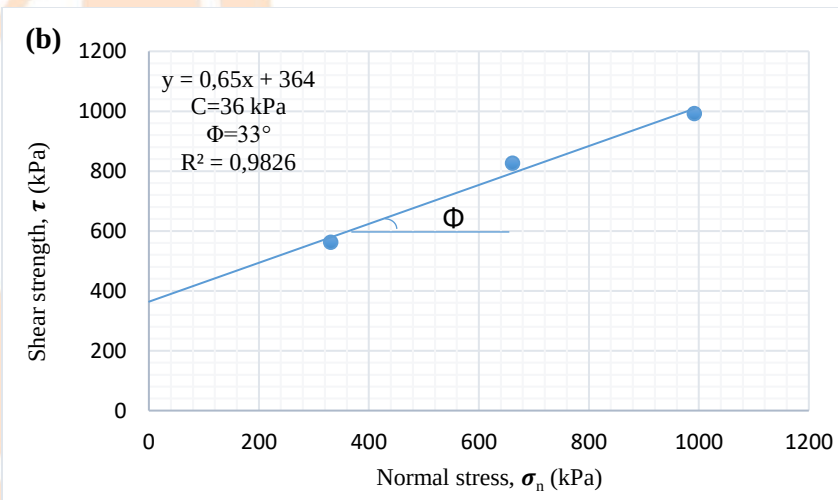
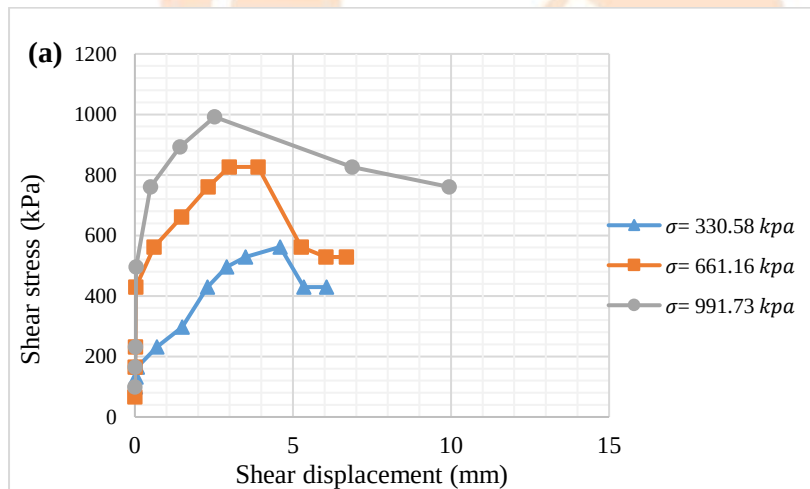


Figure B.6 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for metaquartz-monzonite

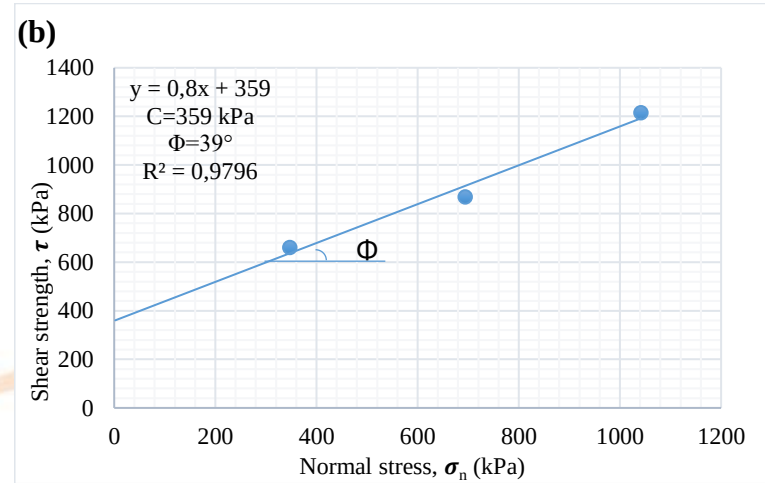
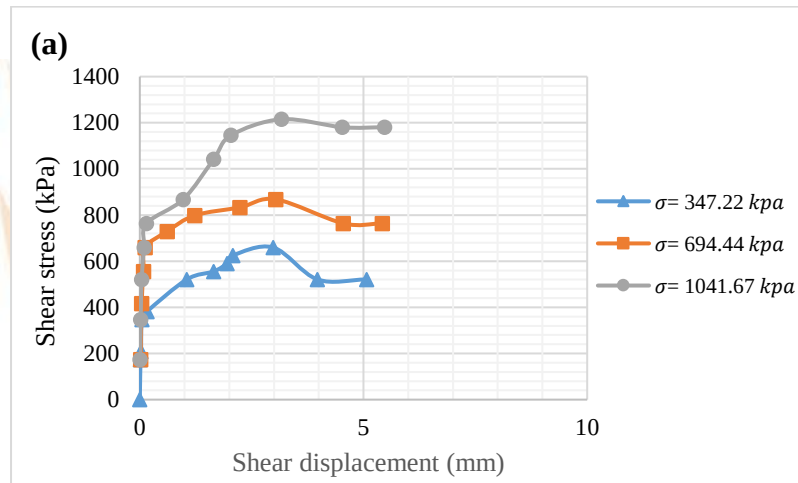


Figure B.7 (a) Shear Stress versus shear displacement, (b) Mohr-Coulomb failure envelope for metagranite with metaquartz-monzonite

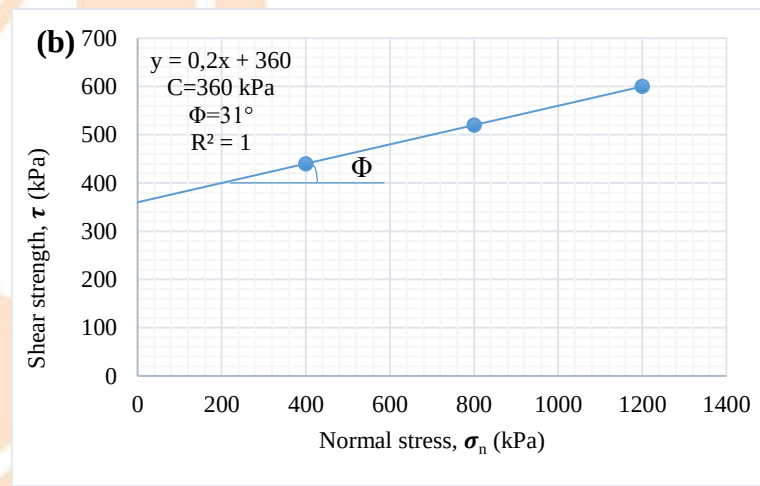
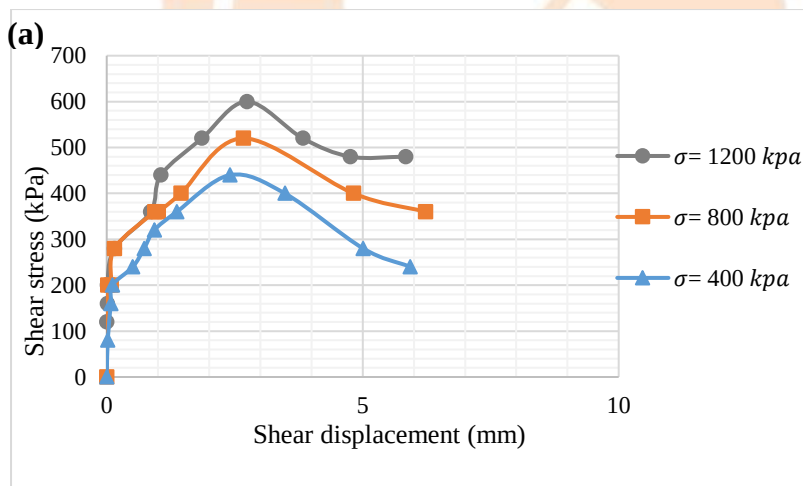


Figure B.8 (a) Shear Stress versus shear displacement, (b) Mohr-Coulomb failure envelope for metagranite with biotite schist

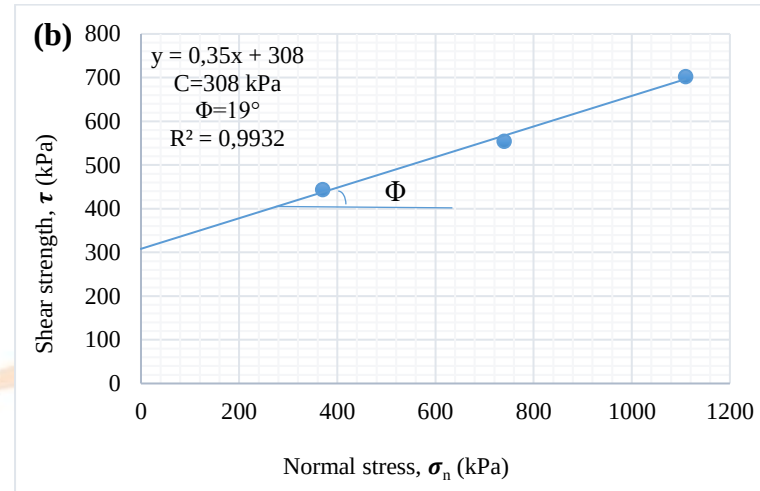
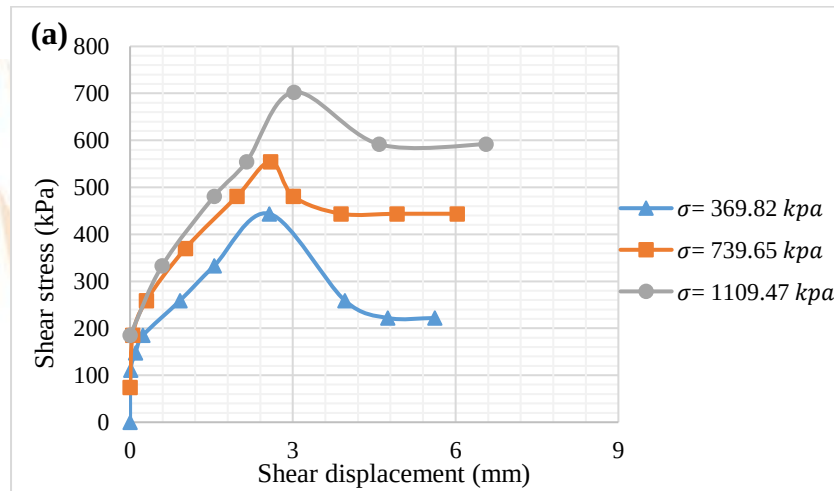


Figure B.9 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for biotite schist with hornblende-biotite gneiss

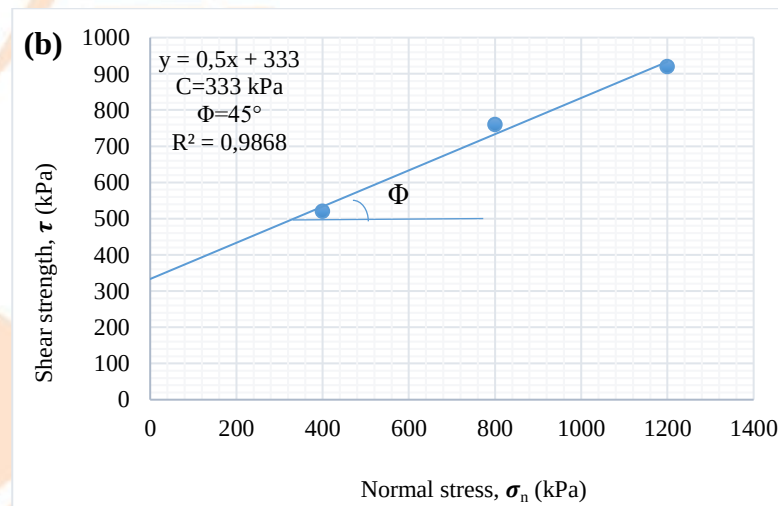
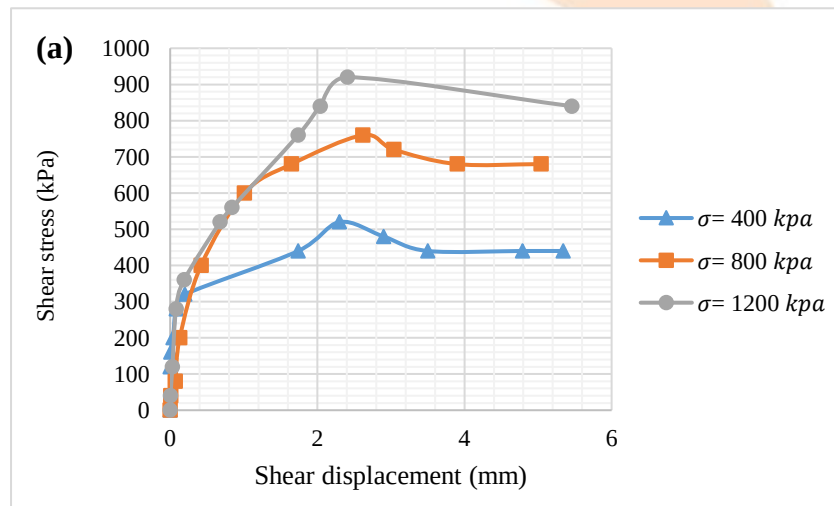


Figure B.10 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for metaquartz-monzonite with hornblende-biotite gneiss

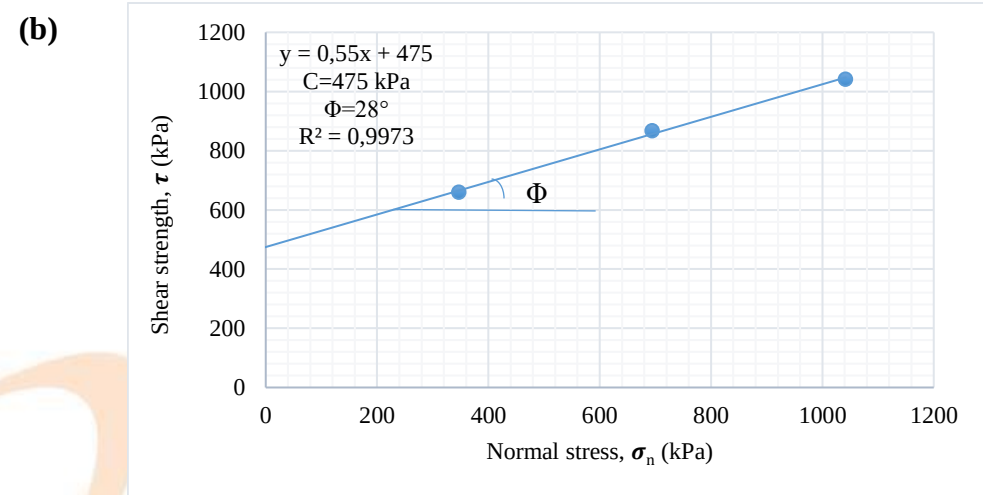
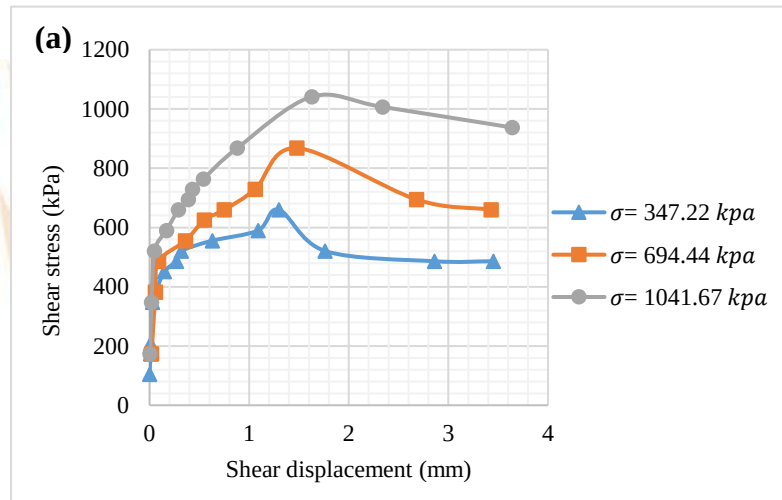


Figure B.11 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for metagranite with biotite schist

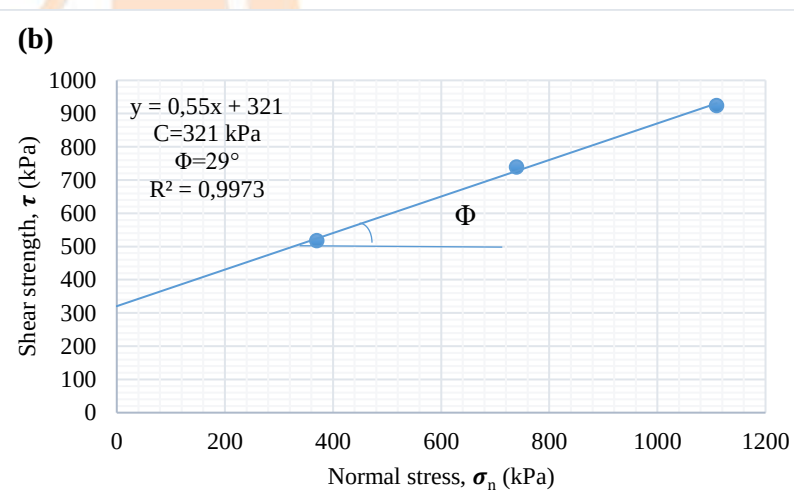
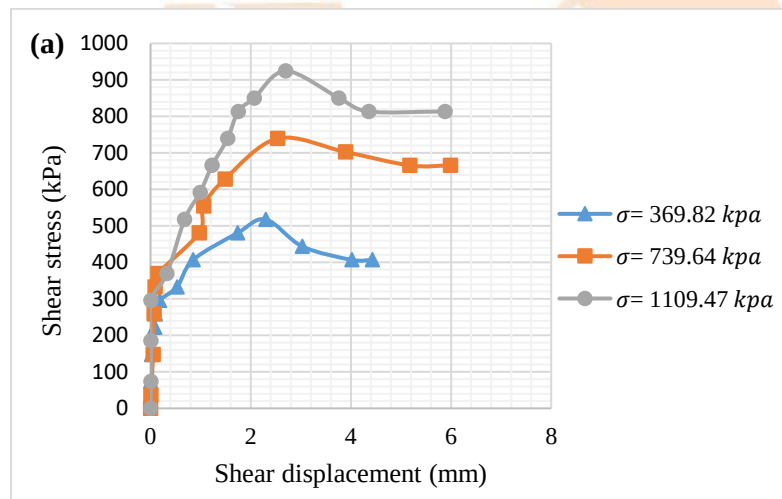


Figure B.12 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for biotite schist with hornblende-biotite gneiss

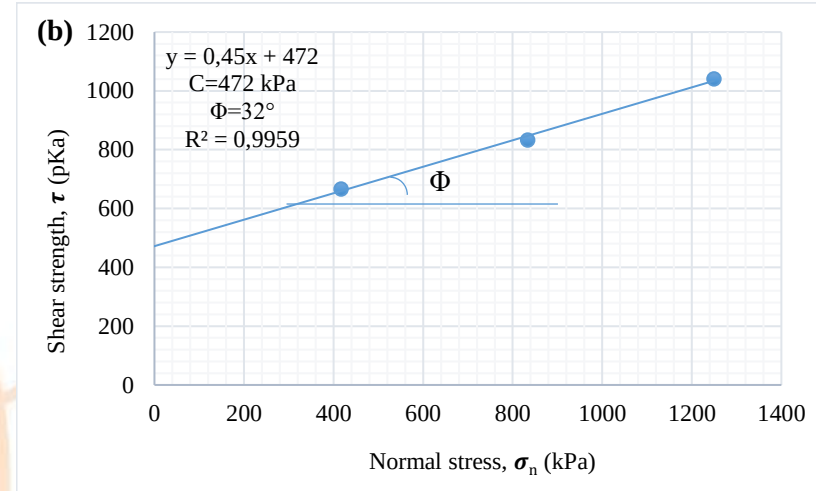
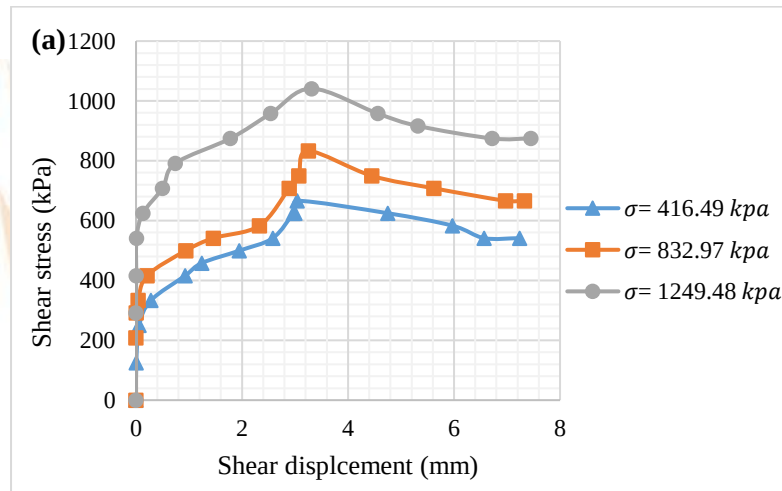


Figure B.13 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for metaquartz-monzonite with hornblende-biotite gneiss

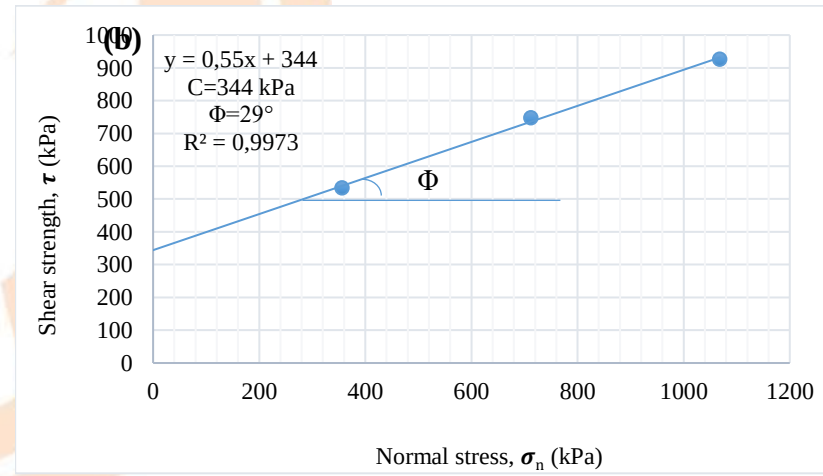
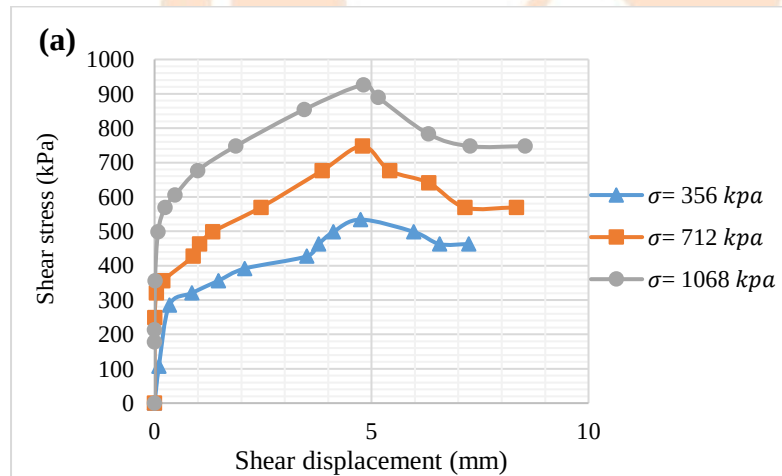


Figure B.14 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for metagranite with hornblende-biotite gneiss

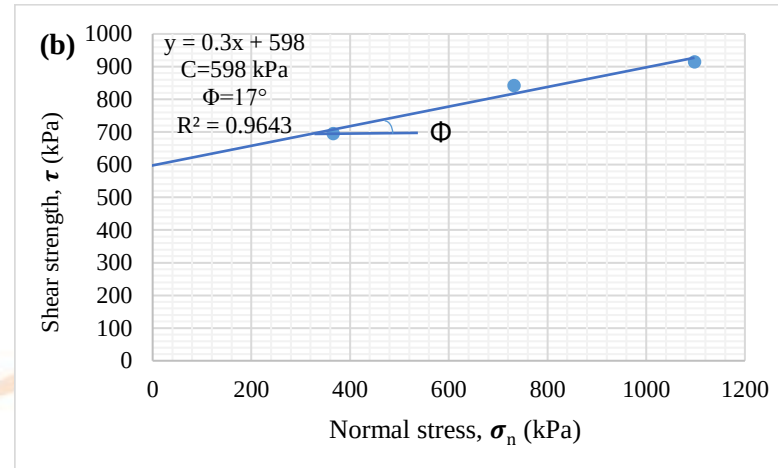
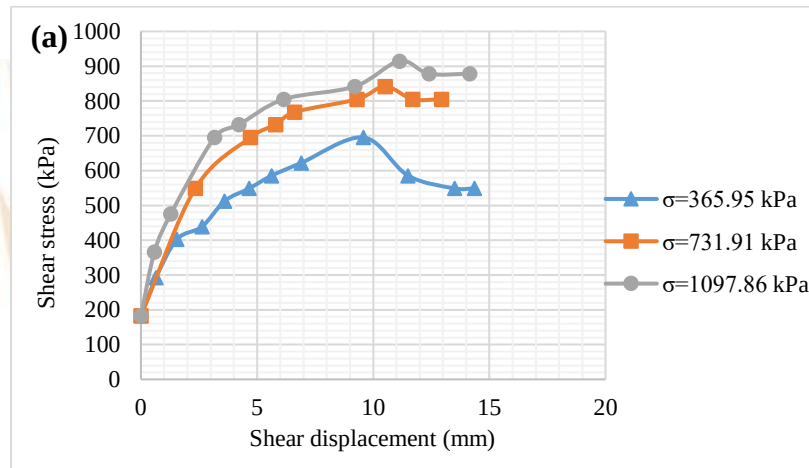


Figure B.15 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for metagranite with biotite schist

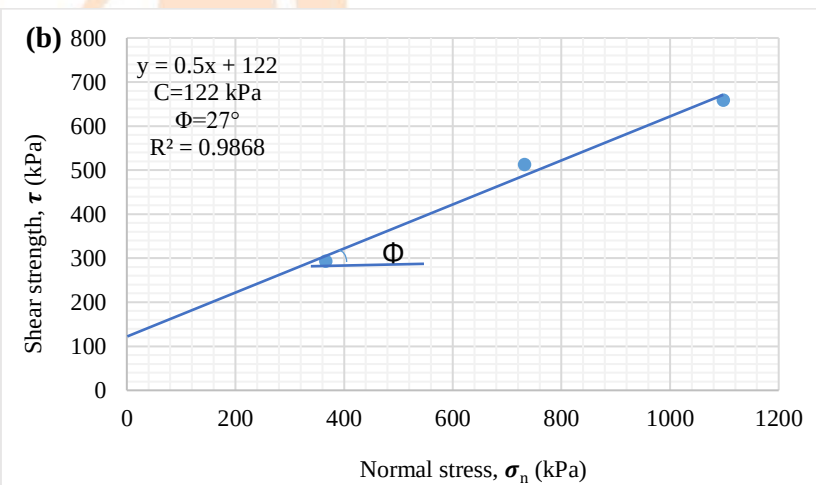
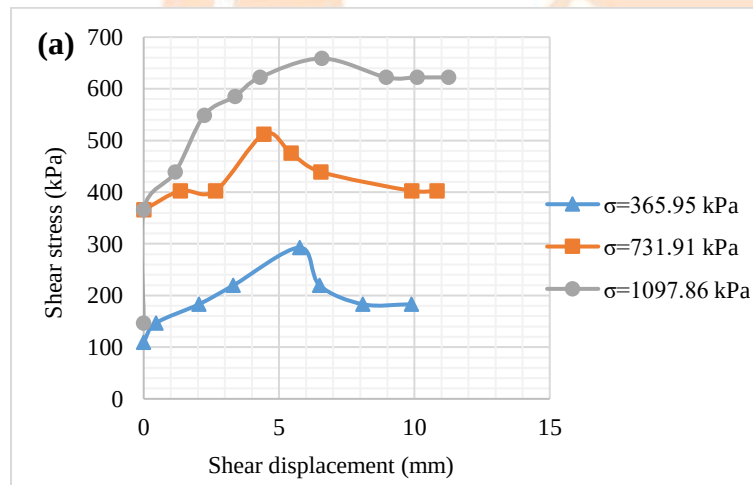


Figure B.16 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for metagranite with hornblende-biotite gneiss

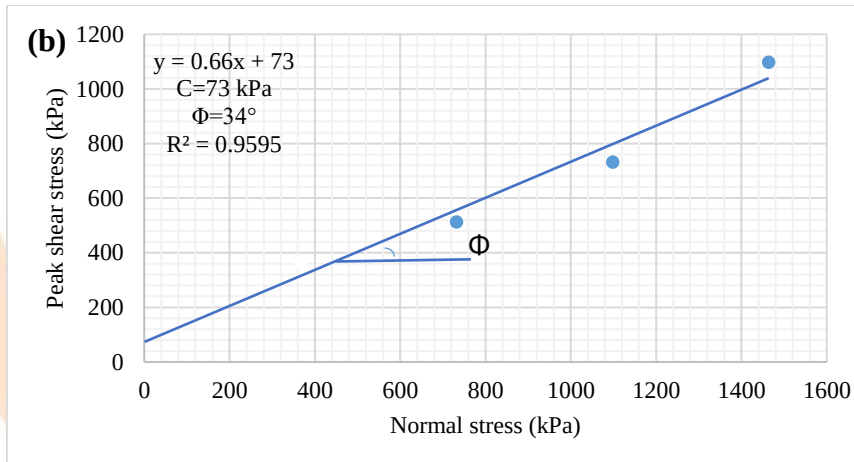
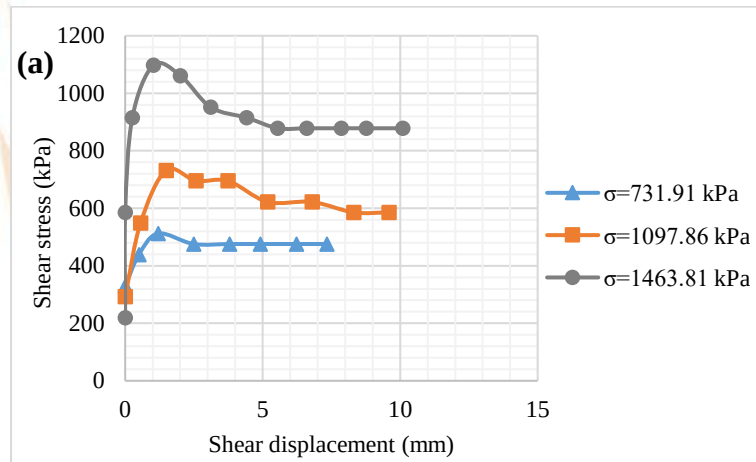


Figure B.17 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for biotite schist with hornblende-biotite gneiss

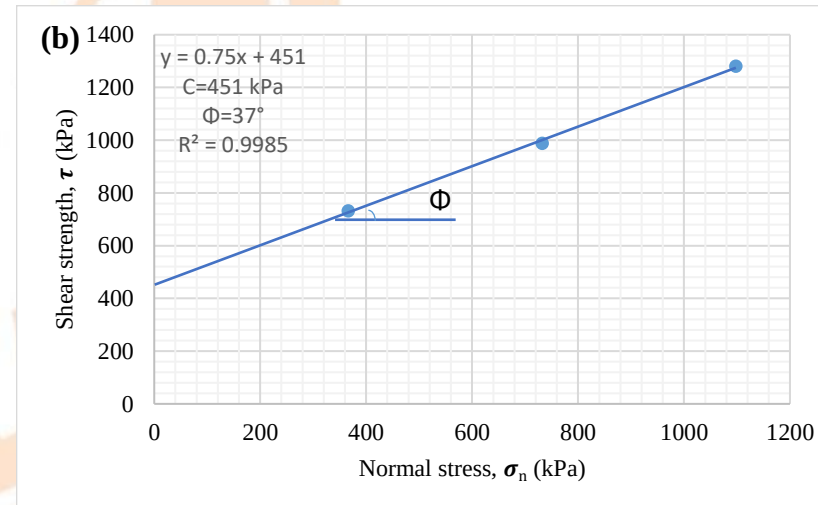
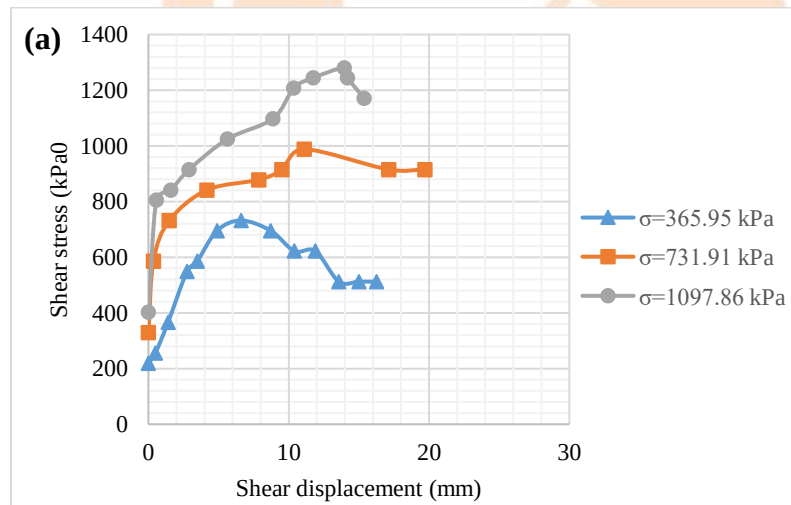


Figure B.18 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for metaquartz-monzonite with biotite schist

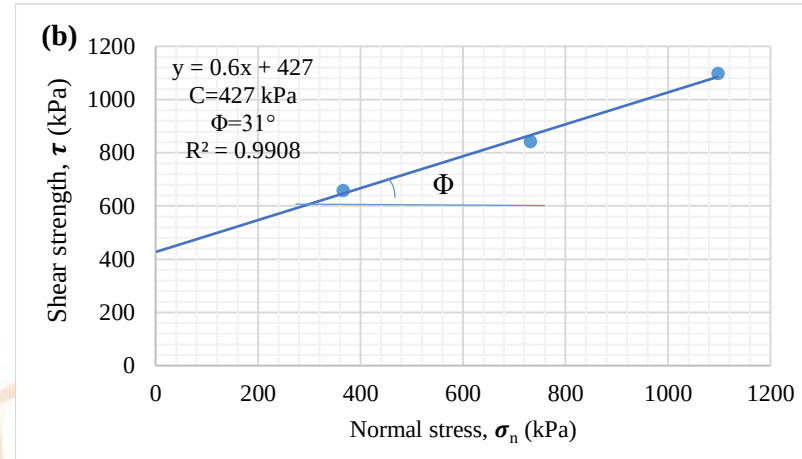
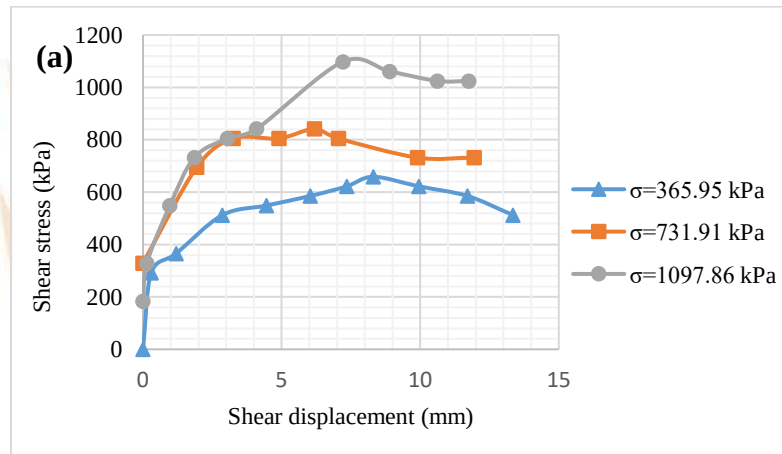


Figure B.19 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for metagranite

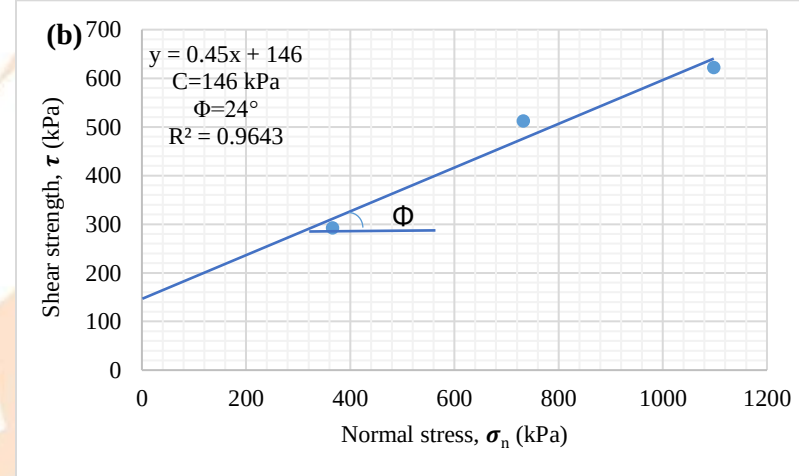
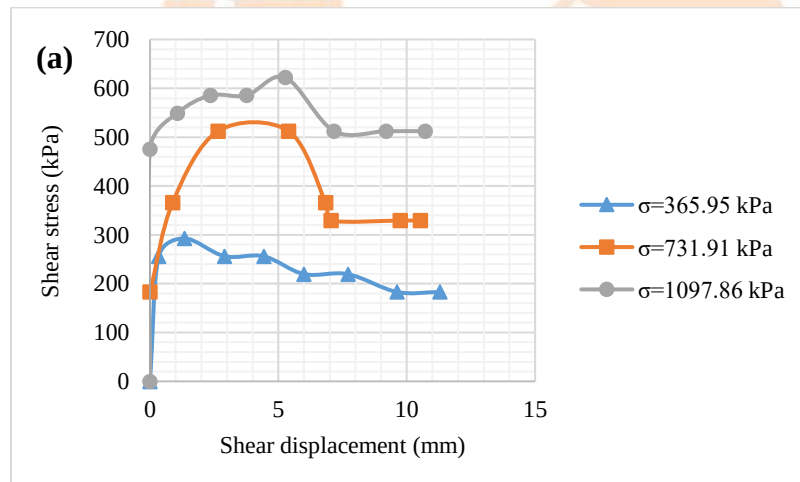


Figure B.20 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for metagranite with metaquartz-monzonite

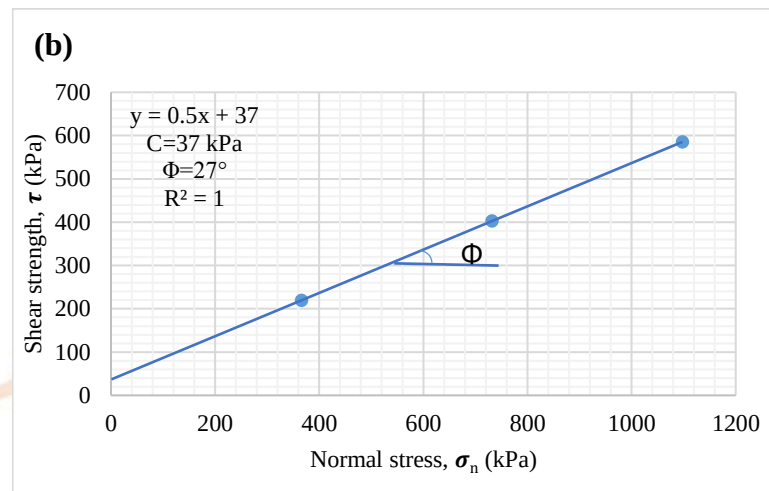
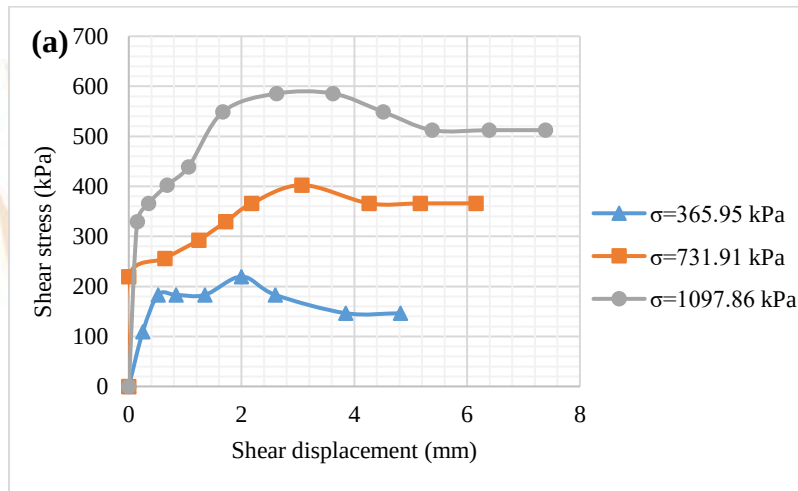


Figure B.21 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for metaquartz-monzonite

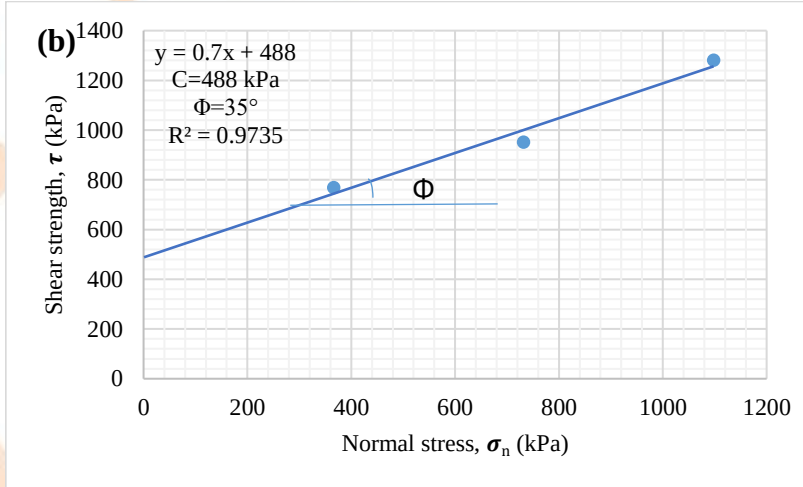
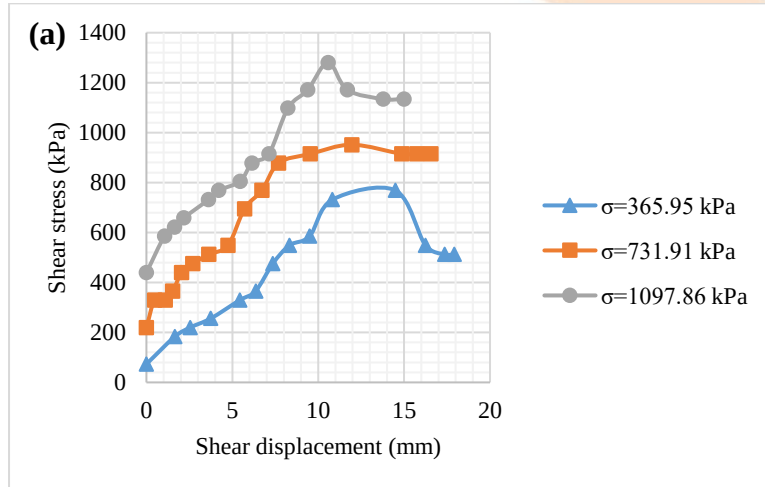


Figure B.22 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for hornblende-biotite gneiss

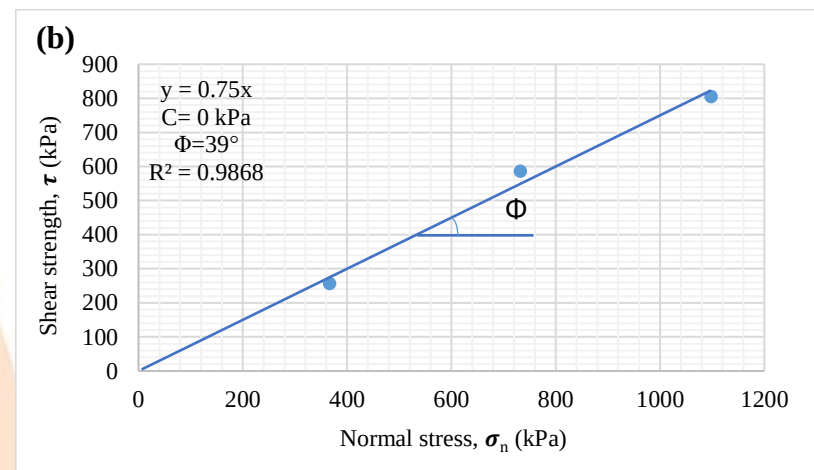
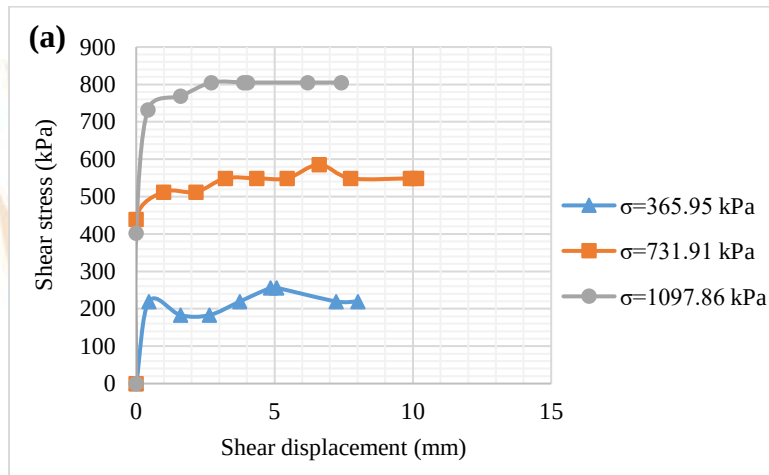


Figure B.23 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for biotite schist with hornblende-biotite gneiss

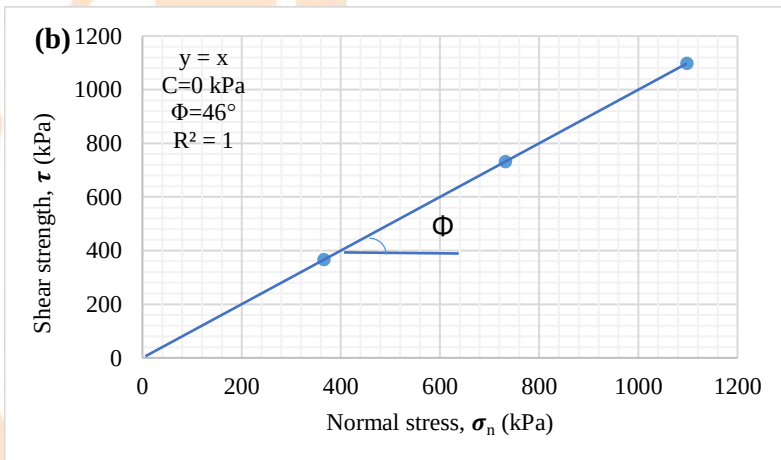
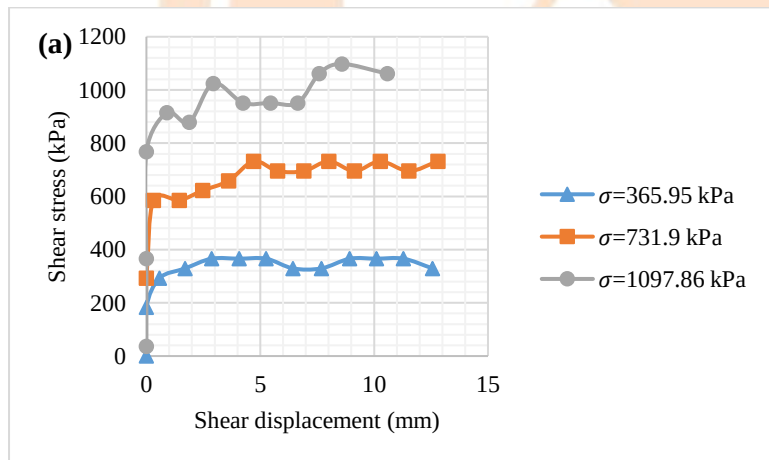


Figure B.24 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for metaquartz-monzonite with biotite schist

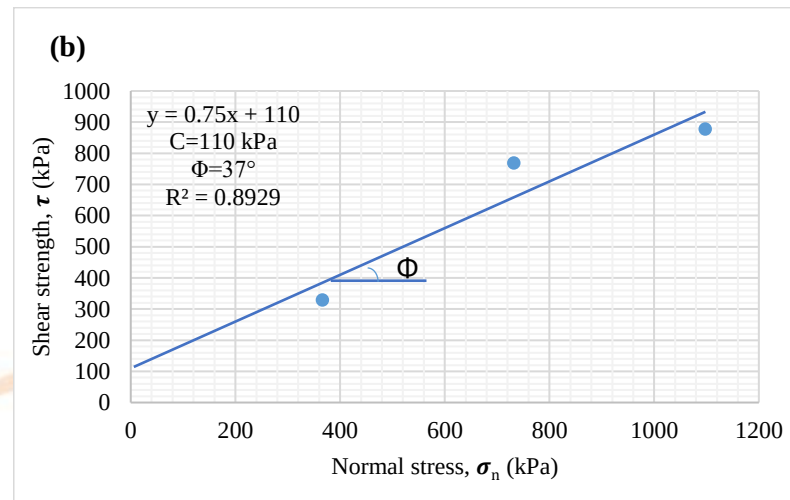
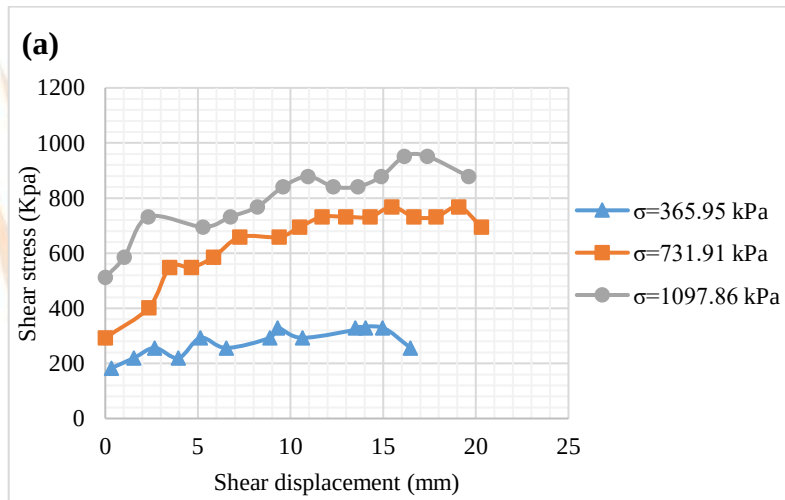


Figure B.25 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for metagranite with hornblende-biotite gneiss

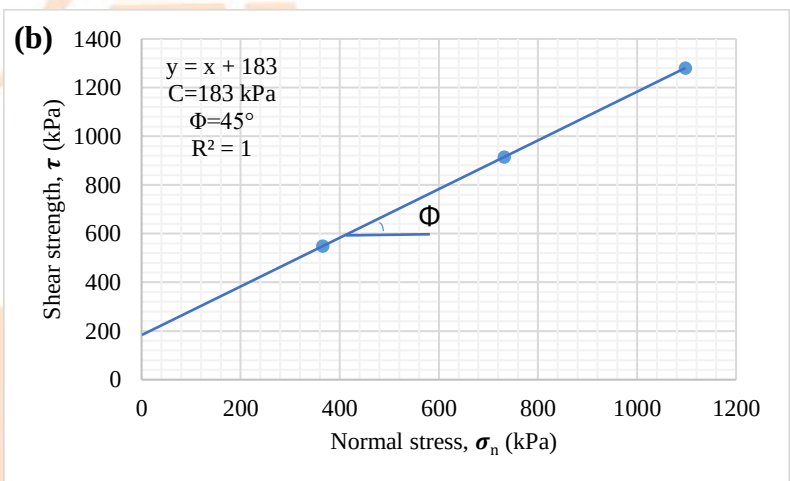
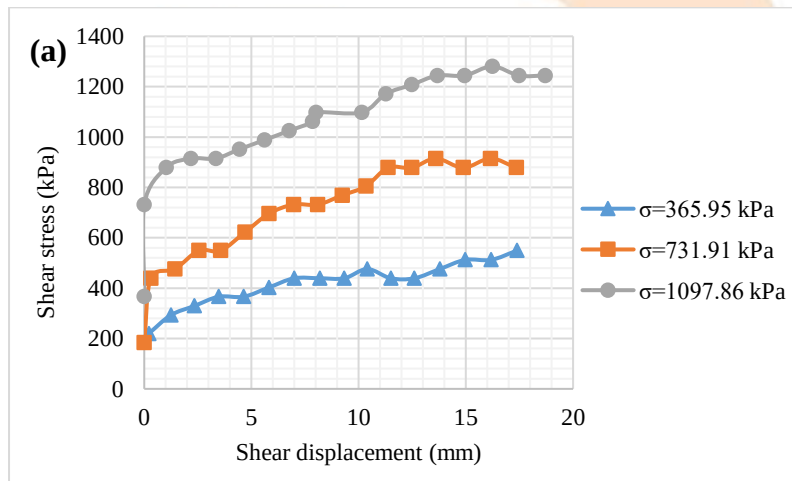


Figure B.26 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for metagranite with metagranite

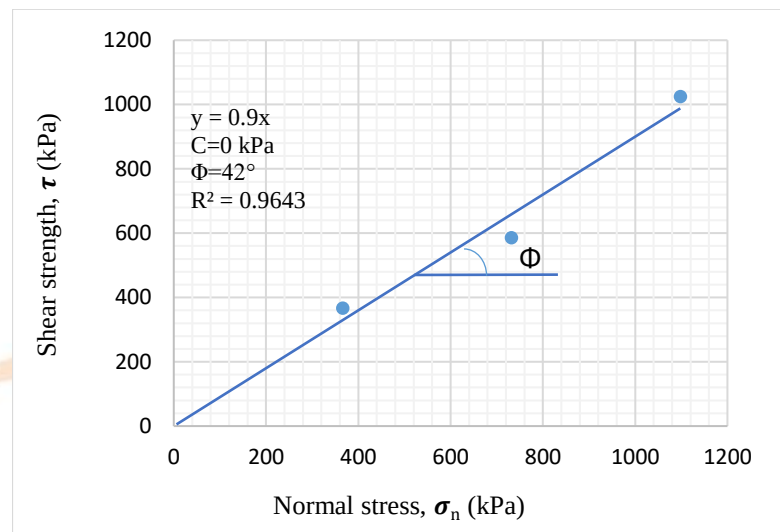
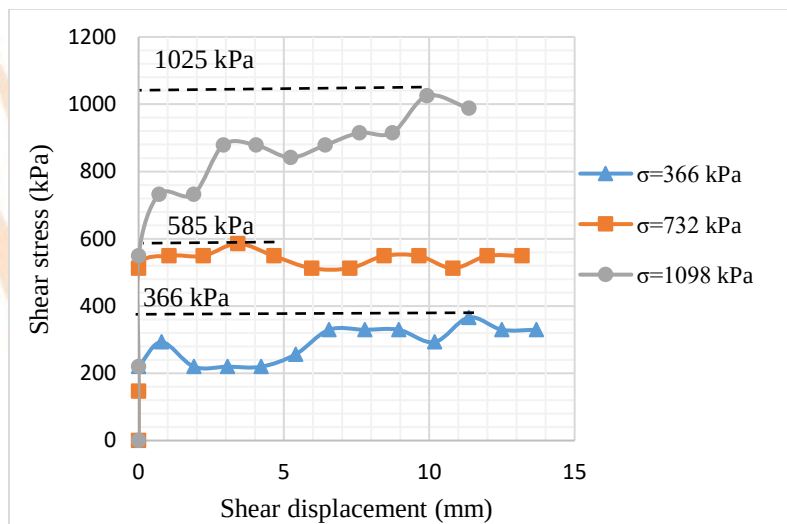


Figure B.27 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for metaquartz-monzonite and metagranite

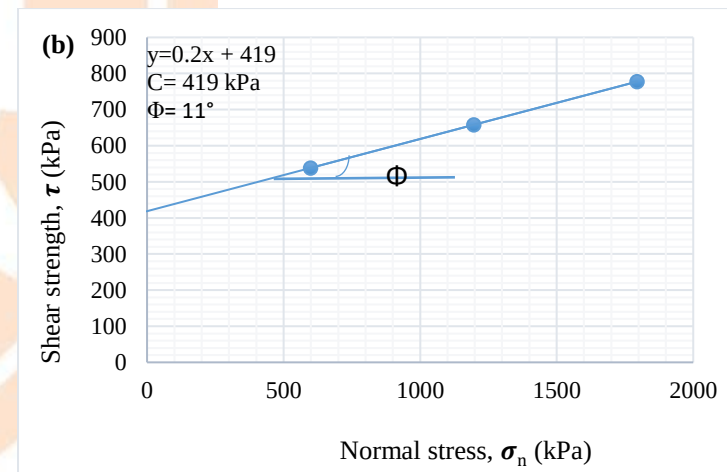
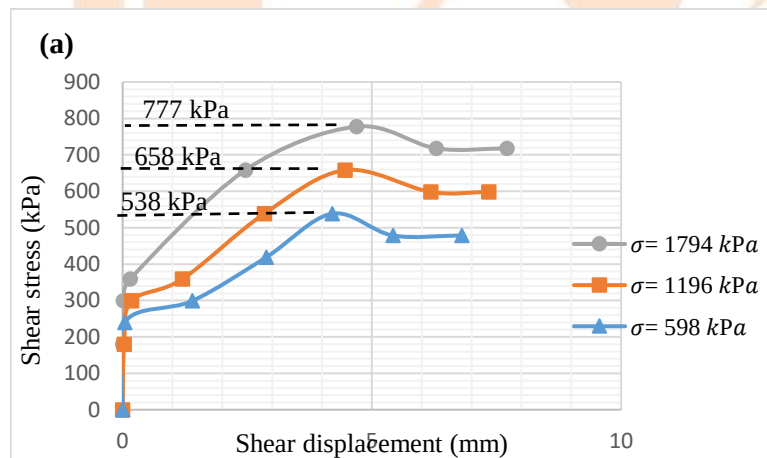


Figure B.28 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for metaquartz-monzonite and biotite schist

Appendix C: Shear stress versus shear displacement/ Mohr-coulomb failure envelope plots for tailings samples

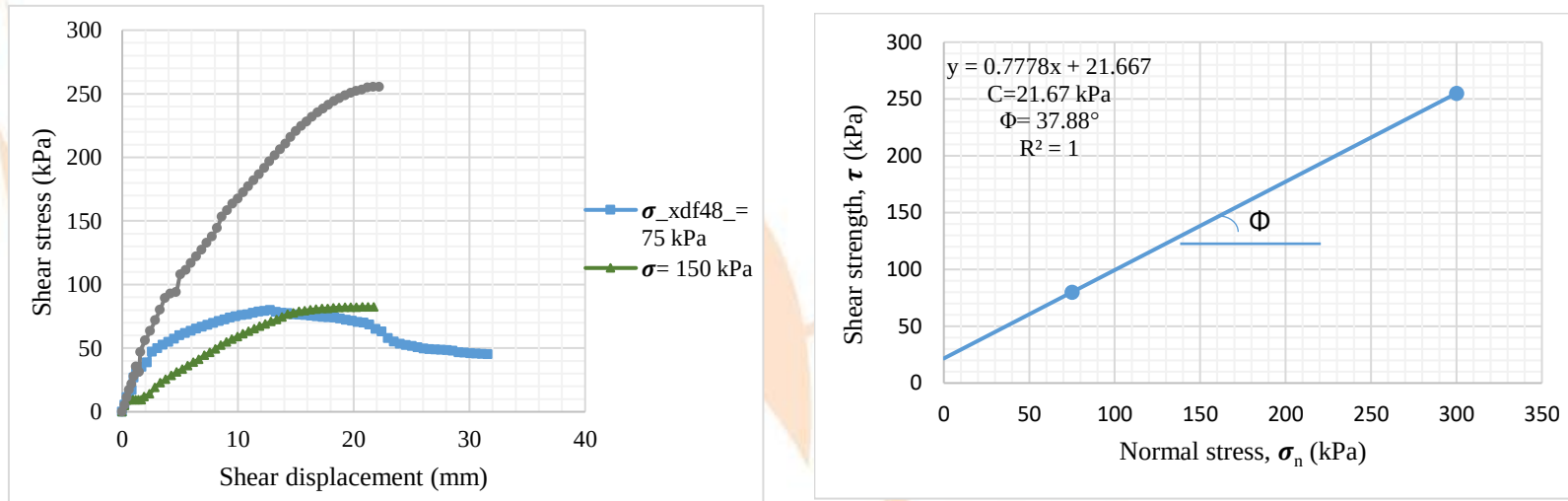


Figure C.1 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for TL3S5

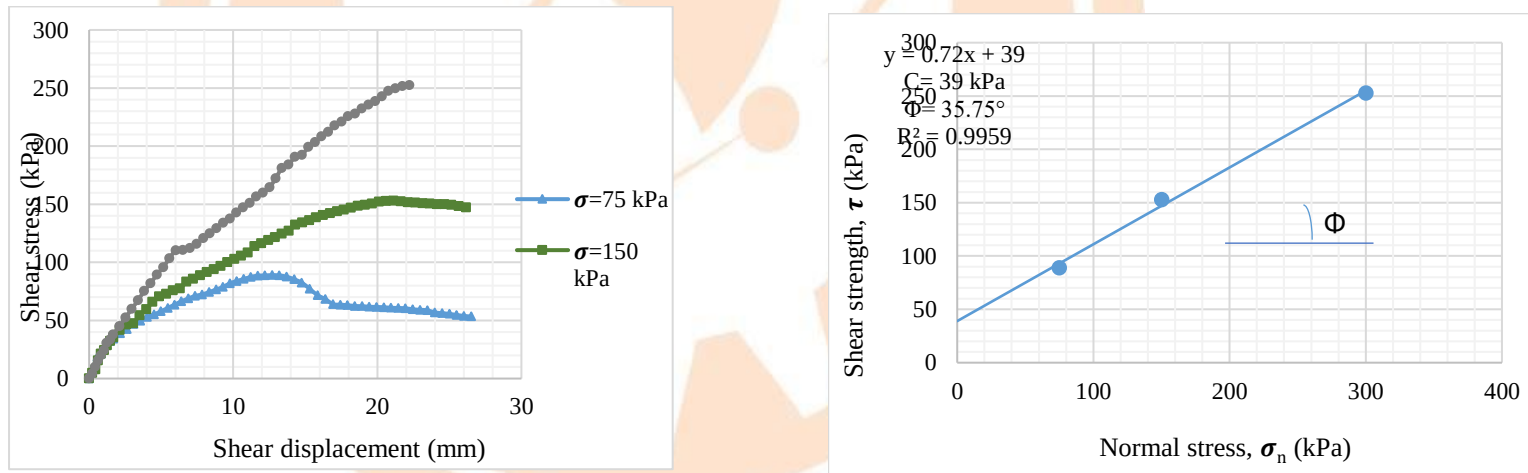


Figure C.2 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for TL1S4

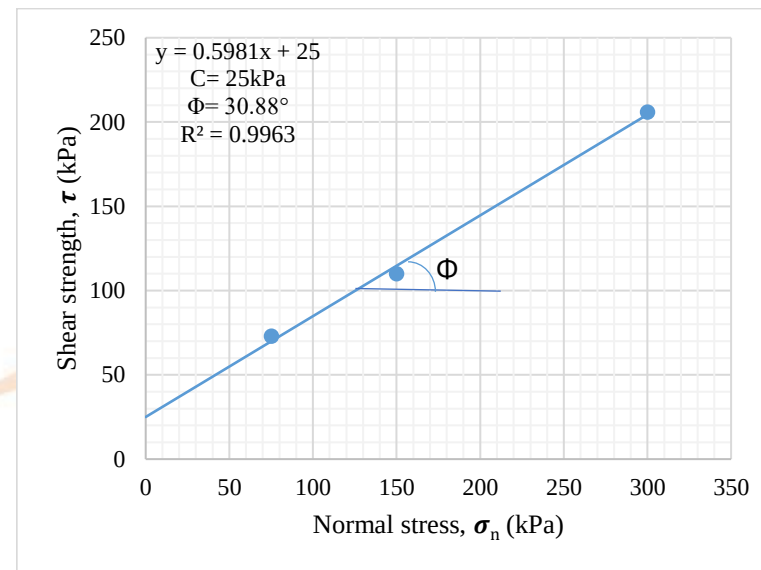
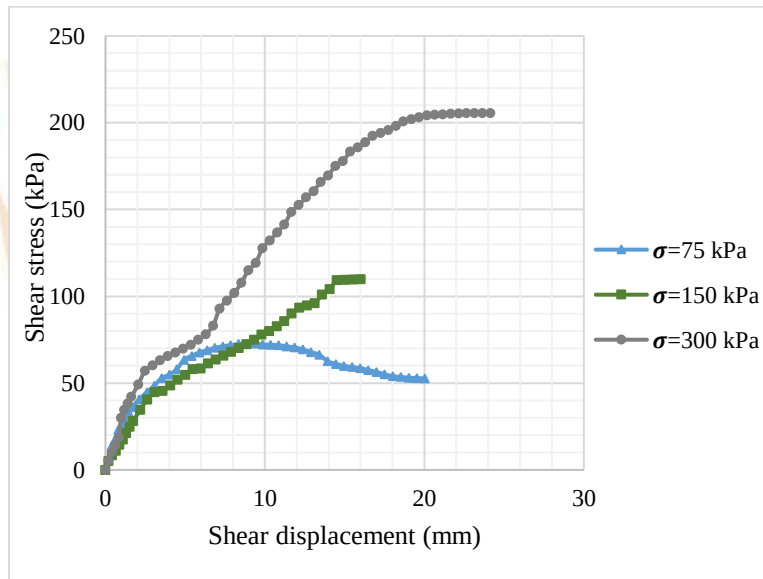


Figure C.3 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for TL3S4

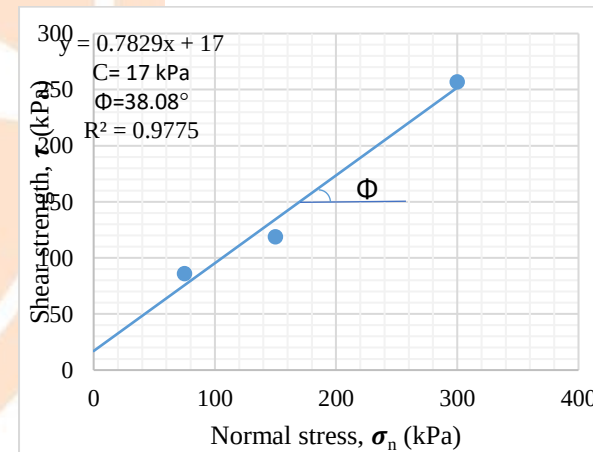
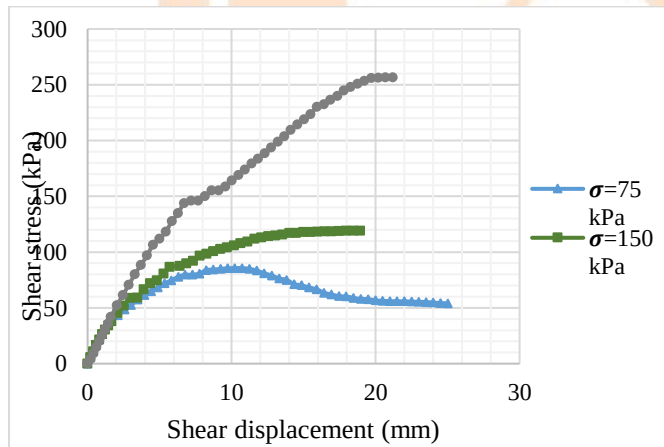


Figure C.4 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for TL3S2

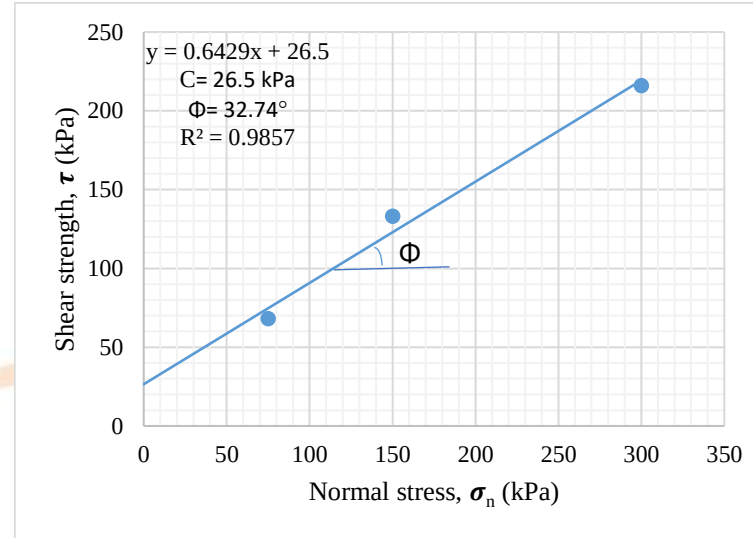
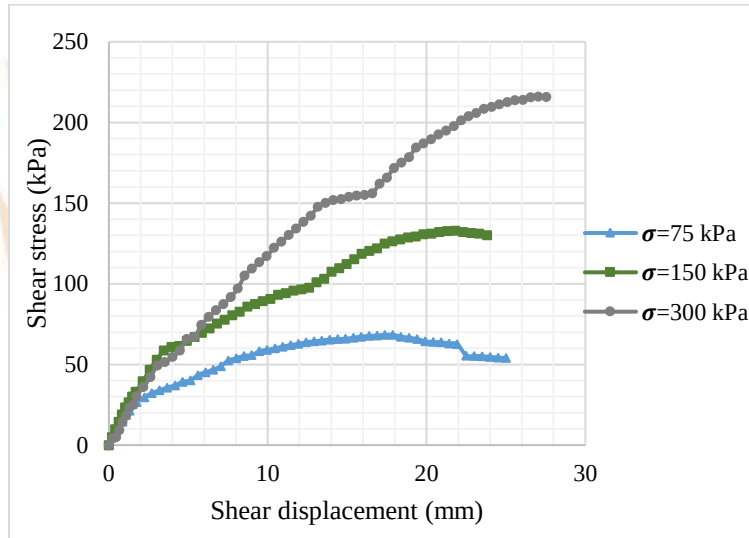


Figure C.5 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for TL2S1

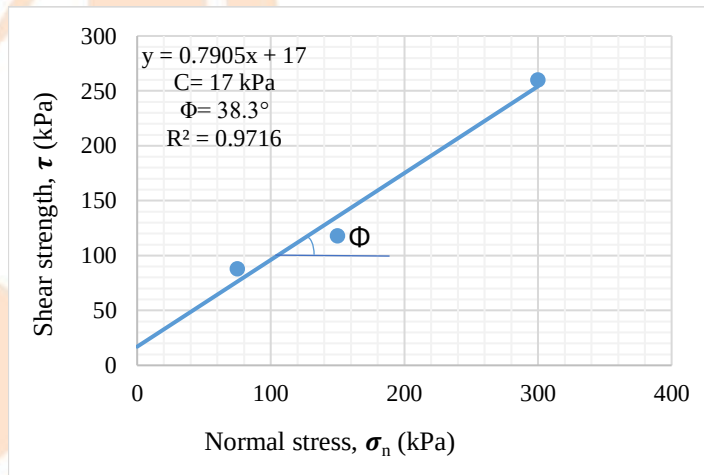
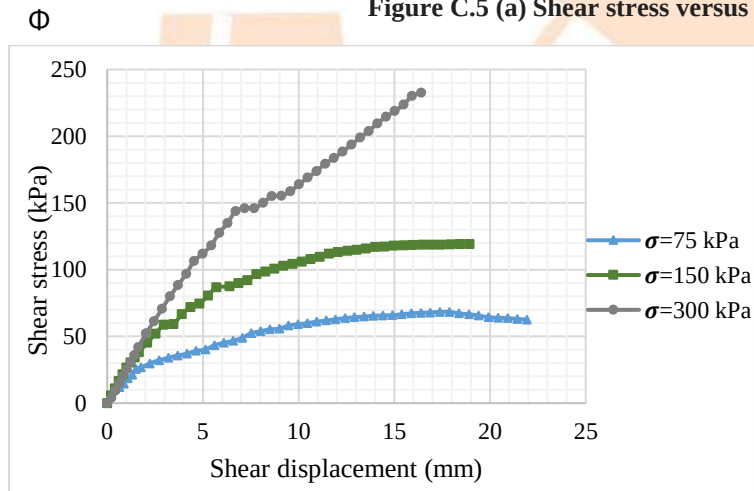


Figure C.6 (a) Shear stress versus shear displacement, (b) Mohr-Coulomb failure envelope for TL1S3

APPENDIX D

TABLES AND GRAPHS

Table D.1 Grain size distribution of tailings samples

Sample ID	Sand (%)	Silt (%)	Clay (%)
TL1S1	70	21	9
TL1S2	64	27	9
TL1S3	59	33	8
TL1S4	66	29	5
TL1S5	64	27	9
TL1S6	65	16	7
TL2S1	71	20	9
TL2S2	71	24	5
TL3S1	71	23	6
TL3S2	72	24	4
TL3S3	74	17	9
TL3S4	67	24	9

Table D.2 Grain size distribution of slag samples

Sample ID	Gravel (%)	Sand (%)	Silt and Clay (%)
SS1L1	1	97	2
SS2L1	0	98	2
SS1L2	3	97	0
SS2L2	1	98	1
SS3L2	1	97	2
SS1L3	4	96	0
SS2L3	4	95	0.
SS3L3	1	98	1
SS1L4	13	86	1
SS2L4	6	94	0

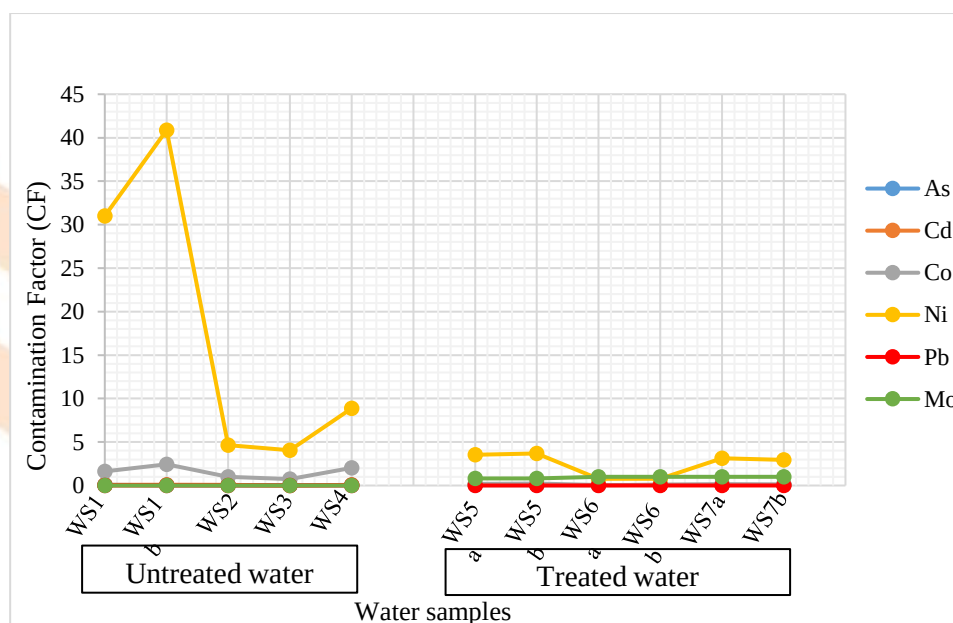


Figure D.1 Contamination factor (CF) of selected heavy metals in water samples

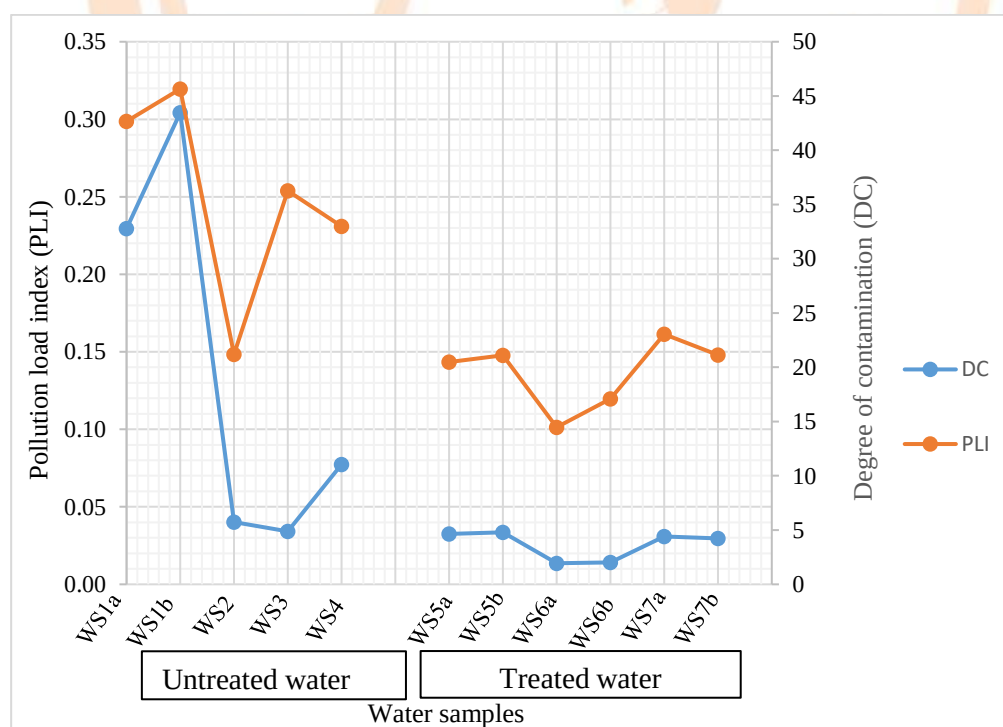


Figure D.2 Pollution load index (PLI) and degree of contamination (DC) of selected heavy metals in water samples

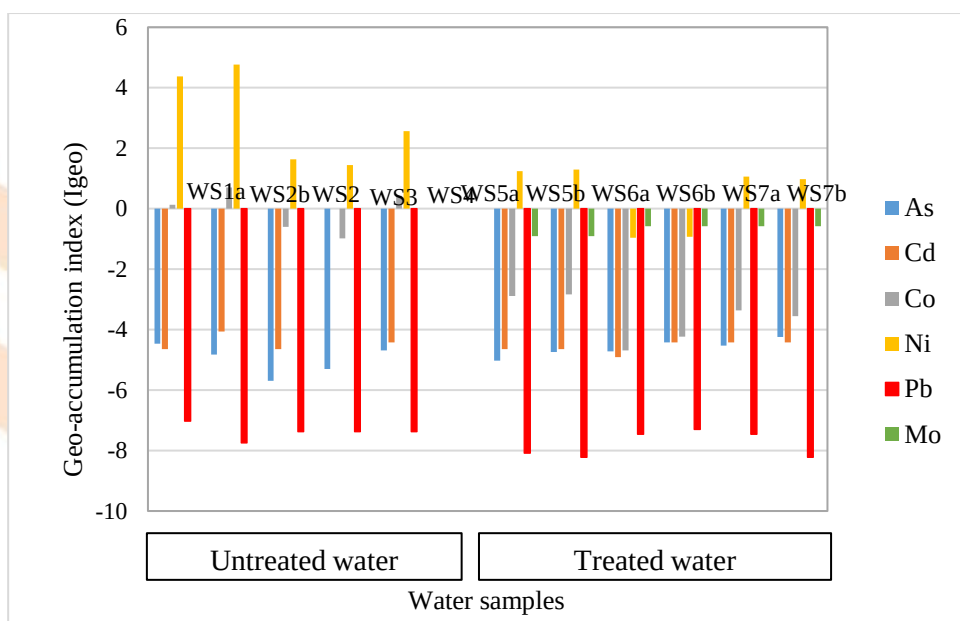


Figure D.3 Geo-accumulation index (Igeo) of selected heavy metals in water samples

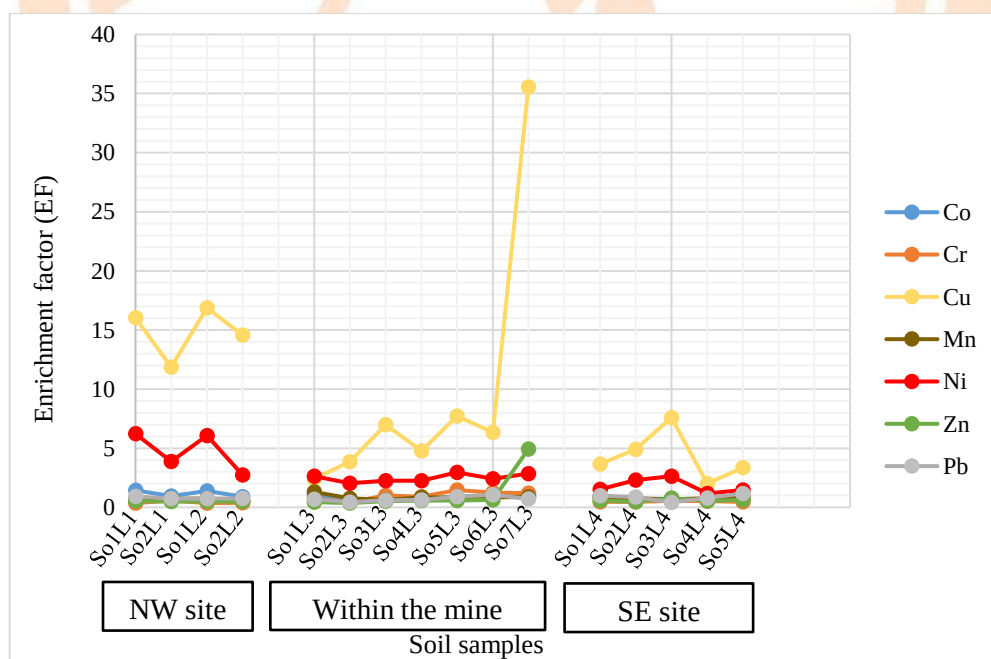


Figure D.4 Enrichment factor (EF) of selected heavy metals in soil samples using Fe as metal of reference

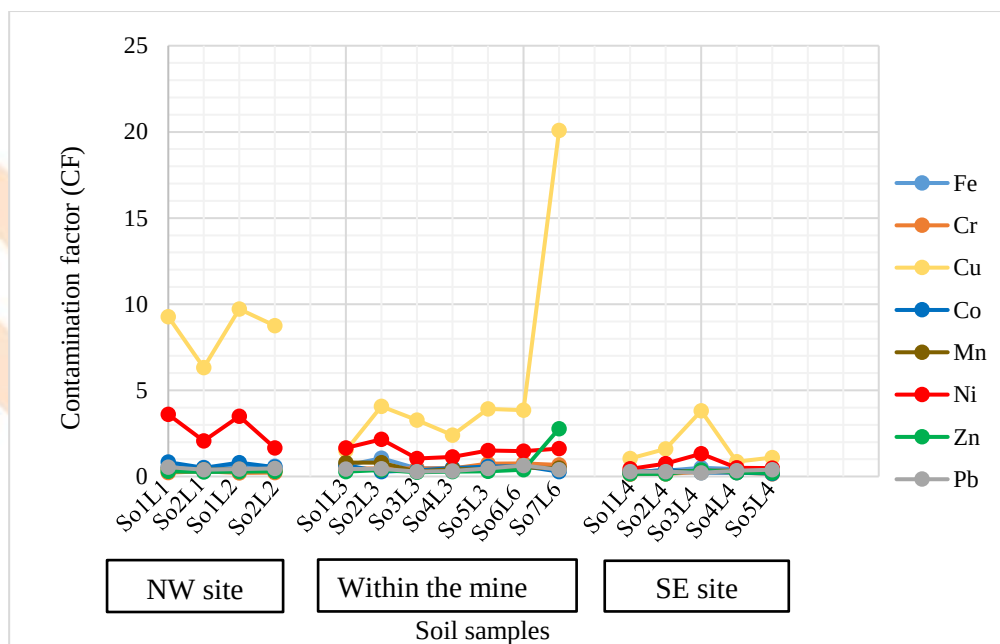


Figure D.5 Contamination factor (CF) of selected heavy metals in soil samples

Appendix E: Part results from mine dumps stability analyses

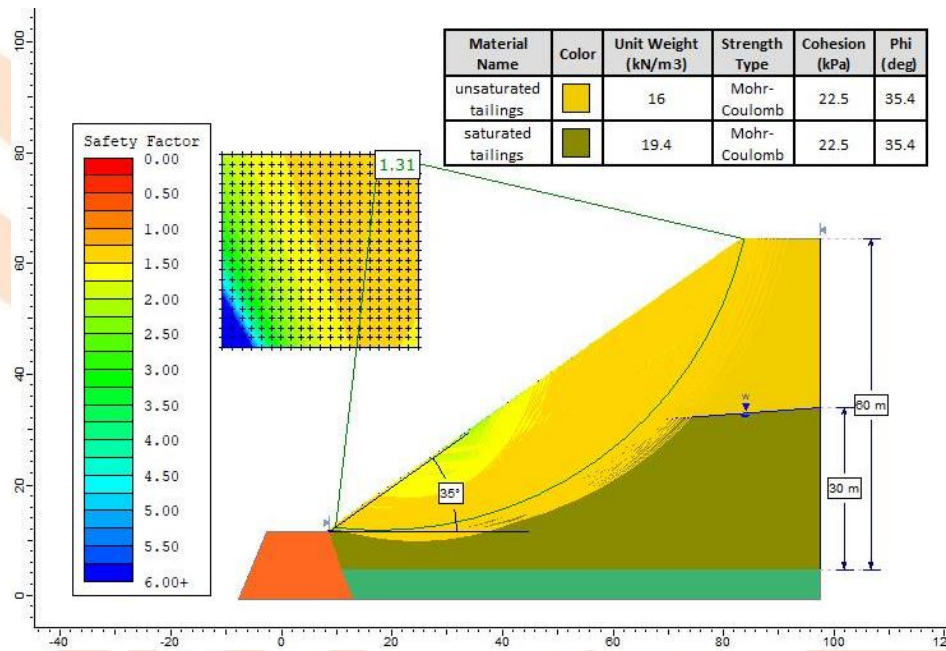


Figure E.1 Slide 2 Software analysis for Factor of safety of tailings dam with slope height: 60 m, slope angle: 35°, water level: 30 m.

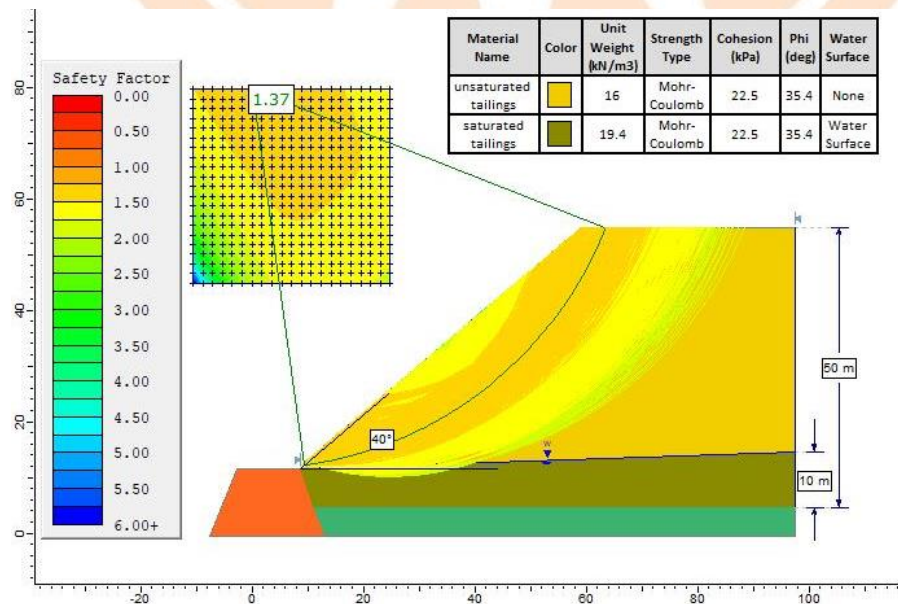


Figure E.2 Slide 2 Software analysis for Factor of safety of tailings dam with slope height: 50 m, slope angle: 40°, water level: 10 m.

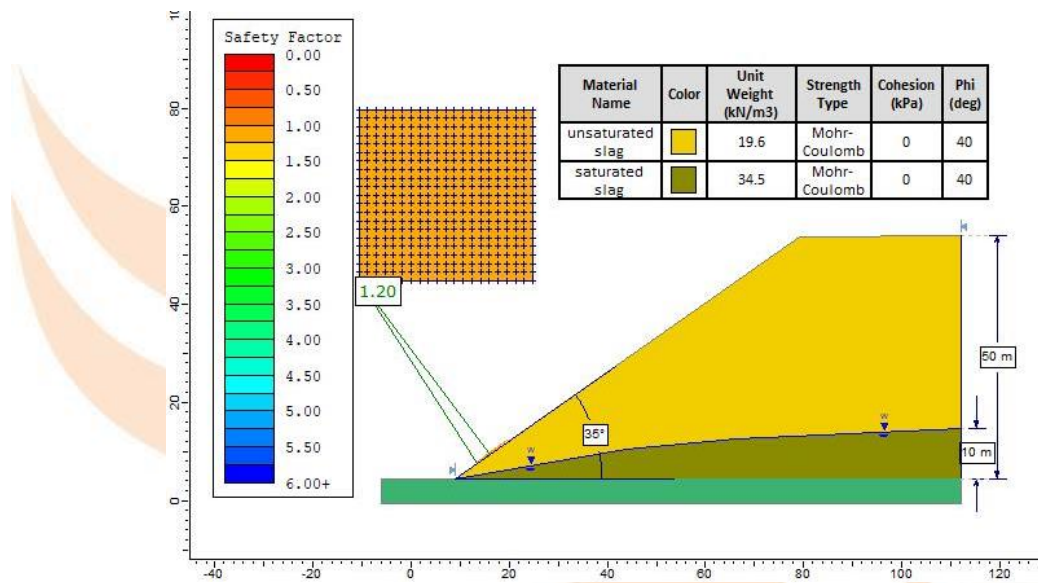


Figure E.3 Slide 2 Software analysis for Factor of safety of slag dump with slope height: 50 m, slope angle: 35°, water level: 10 m

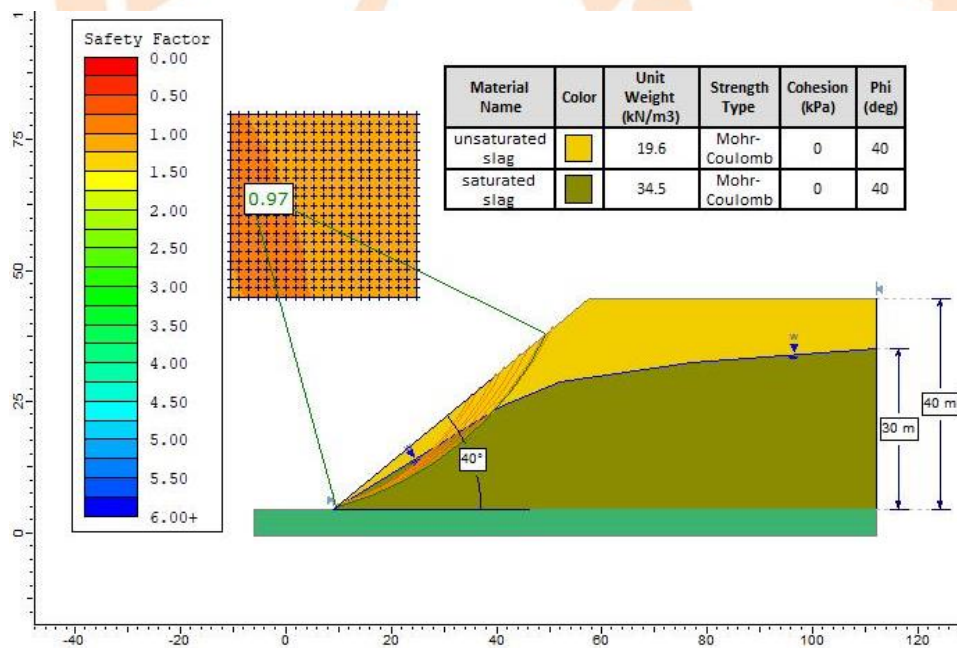


Figure E.4 Slide 2 Software analysis for Factor of safety of slag dump with slope height: 40 m, slope angle: 40°, water level: 30 m

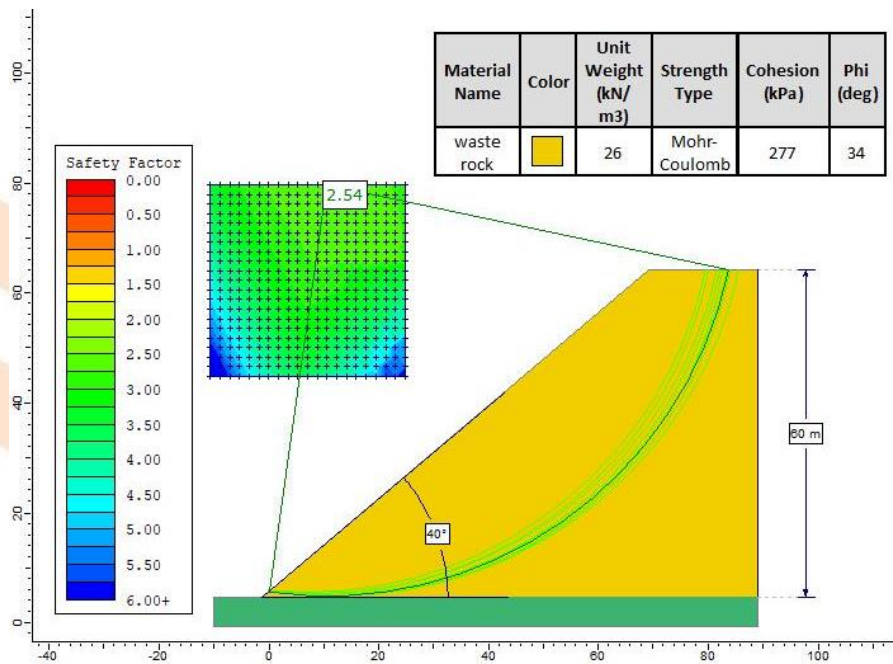


Figure E.5 Slide2 Software analysis for Factor of safety of waste rock dump with slope height: 60 m, slope angle: 40°

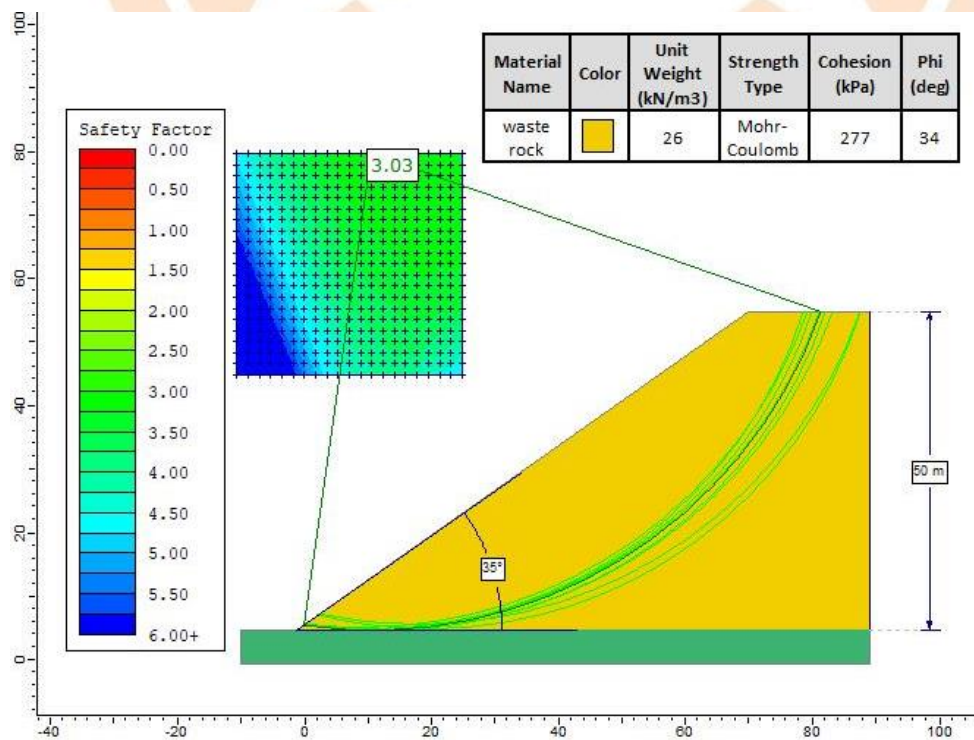


Figure E.6 Slide2 Software analysis for Factor of safety of waste rock dump with slope height: 50 m, slope angle: 35°

APPENDIX E CONTINUED

Table E1. Summary of Slide2 Analysis output of Factor of Safety for various Tailings dam slope angles and height and water levels (All scenarios)

(a)

Slope height (m)	Slope angle (°)	Water level (m)	Factor of safety
40	30	10	1.90
		20	1.82
		30	1.57
		40	1.36
	35	10	1.67
		20	1.58
		30	1.33
		40	1.20
	40	10	1.49
		20	1.38
		30	1.33
		40	1.18
	45	10	1.32
		20	1.32
		30	1.29
		40	1.00
	50	10	1.20
		20	1.20
		30	1.18
		40	1.09

(b)

Slope height (m)	Slope angle (°)	Water level (m)	Factor of safety
50	30	10	1.86
		20	1.72
		30	1.53
		40	1.29
		50	1.09
	35	10	1.59
		20	1.48
		30	1.41
		40	1.17
		50	1.08
	40	10	1.41
		20	1.41
		30	1.23
		40	1.14
		50	0.95
	45	10	1.22
		20	1.22
		30	1.14
		40	1.11
		50	0.92
	50	10	1.11
		20	1.11
		30	1.11
		40	0.96
		50	0.93

=

(c)

Slope height (m)	Slope angle (°)	Water level (m)	Factor of safety
60	30	10	1.88
		20	1.78
		30	1.58
		40	1.35
		50	1.18
		60	1.06
	35	10	1.59
		20	1.55
		30	1.32
		40	1.08
		50	1.00
		60	0.84
	40	10	1.37
		20	1.36
		30	1.27
		40	1.10
		50	0.96
		60	0.84
	45	10	1.19
		20	1.19
		30	1.16
		40	1.03
		50	0.98
		60	0.80
	50	10	1.06
		20	1.06
		30	1.02
		40	0.97
		50	0.89
		60	0.81

Table E2 Summary of Slide2 Analysis output of Factor of Safety for various slag dump slope angles and height and water levels (All scenarios)

(a)

Slope height (m)	Slope angle (°)	Water level (m)	Factor of safety
40	30	10	1.46
		20	1.41
		30	1.21
		40	1.15
	35	10	1.21
		20	1.21
		30	1.21
		40	1.07
	40	10	0.99
		20	0.99
		30	0.95
		40	0.86

(c)

Slope height (m)	Slope angle (°)	Water level (m)	Factor of safety
60	30	10	1.46
		20	1.37
		30	1.24
		40	1.16
		50	0.93
		60	0.97
	35	10	1.21
		20	1.21
		30	1.14
		40	0.99
		50	0.93
		60	0.78
	40	10	1.00
		20	1.00
		30	0.98
		40	0.95
		50	0.84
		60	0.80

(b)

Slope height (m)	Slope angle (°)	Water level (m)	Factor of safety
50	30	10	1.46
		20	1.39
		30	1.16
		40	1.02
		50	0.95
	35	10	1.22
		20	1.14
		30	1.06
		40	0.97
		50	0.79
	40	10	0.99
		20	0.99
		30	0.90
		40	0.86
		50	0.78

Table E3. Summary of Slide2 Analysis output of Factor of Safety for various waste rock slope angles and height

Slope height (m)	Slope angle (°)	Factor of safety
40	30	3.80
	35	3.40
	40	3.14
50	30	3.52
	35	3.03
	40	2.75
60	30	3.46
	35	2.84
	40	2.54