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Groundwater Contamination through Cyanide and Metal Migration from Tailings Dam Operation

By

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Declaration

I hereby declare that this submission is my own work towards the PhD and that, to the best of my knowledge, it contains no material previously published by another person or material which has been accepted for the award of any other degree of the University, except where due acknowledgement has been made in the text.

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Prof J.S.Y Kuma (Supervisor)	.... Signature	.... Date
Dr. Gnida Sossou (Head of Department)	.... Signature	.... Date

Dedication

To The Ancient of Days, Generations Unborn, My Family; Edward, Joseph, Michaela, Rosemary and Stephanie

And

The Lost Symbol

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Abstract

In the gold industry, only a few hundredths of an ounce of gold may be produced for every tonne of tailings generated. Tailings need to be properly managed because they constitute a major source of release of many metals and trace elements into the environment which is a potential source of groundwater contamination. The impact of Gold Fields Ghana Limited (GFGL) tailings dams were evaluated by chemical characterisation of metals and trace element concentrations in the decant pond, tailings slurry and underdrainage sump. The attenuation properties of the substrata were also evaluated with emphasis on hydraulic conductivity, permeability characteristics and geochemical properties (pH, Organic Matter Content (OMC), Cation Exchange Capacity (CEC), iron oxide (Fe_2O_3) and Anion Exchange Capacity (AEC). Finally, four years groundwater monitoring data was used to evaluate the impact of possible seepage from the dam. Results of the studies indicate that concentrations of Weak Acid Dissociable (WAD) cyanide and Total cyanide increased significantly with depth in the decant pond. pH of the pond is alkaline with mean values ranging from 9.3 to 10.6 corresponding to mean pH values in the tailings slurry at spigotting points and underdrainage sump. Total cyanide concentrations ranged from 0.36 mg/l to 1.11 mg/l over five meters depth and 0.10 mg/l to 0.29 mg/l over ten meters depth in the decant ponds of TSF 1 and TSF 2 . WAD cyanide concentrations ranged from 0.26 mg/l to 0.93 mg/l and from 0.01 mg/l to 0.16 mg/l over five and ten meters of decant depth respectively. Significant metals identified in the decant pond, underdrainage sump and tailings slurry were Sodium (Na) and Magnesium (Mg) with trace elements being Iron (Fe), Copper (Cu), Manganese (Mn), Arsenic (As), Nickel (Ni), Zinc (Zn) and Chromium (Cr), whiles alkali metals identified were Calcium (Ca) and Potassium (K). These metals and trace element were consistently present in the tailings and underdrainage. Low hydraulic conductivity and a combination of low acidic pH, relatively high Anion Exchange Capacity (AEC) and low Cation Exchange Capacity (CEC) were identified to be the major driving attenuation properties that favour cyanide attenuation in the Tailings Storage Facility (TSF) basin. The average concentration of Total cyanide detected within a radius of 3 m at the field leaching stations is 0.02 mg/l, constituting 99.9% attenuation of cyanide introduced. The hydraulic conductivity and permeability values computed from grain size analyses of the substrata material are 4.3 x 10^{-6} to 9.7 x 10^{-6} m/min and 6.1 x 10^{-12} to 13.9 x 10^{-11} m/min respectively. The TSF sub-strata material has a pH range of 4.3 to 4.8, a CEC range of 1.92 to 2.89 meq/100g and an AEC range of 881.25 to 1278.25 meq/l. The attenuation rate of cyanide at the five (5) stations remained constant at 0.021 mg/l/hr as time increased. Based on significant increase in levels of EC and pH levels as well as concentrations of Na, Ca, and Cl, it could be established that monitoring boreholes along the toe of TSF1 and TSF2 have received seepage from the tailings facilities. Similar concentrations were observed in boreholes at the South heap leach facility, specifically near the pregnant pond, monitoring boreholes located west of the North heap leach Facility and specifically around the South and North containment ponds.

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CHAPTER 1: INTRODUCTION

1.1 Background of the study

Cyanide is a useful industrial chemical and its key role in the mining industry is to extract gold and silver from crushed rock. Worldwide, mining uses about 13% of the total production of manufactured hydrogen cyanide while the remaining 87% is used in many other industrial processes “apart from mining”. During mining, a fine grind of the ore is often necessary to release metals and minerals, so the mining industry produces enormous quantities of fine rock particles, in sizes ranging from sand-sized down to as low as a few microns (USEPA, 1994a). These fine-grained wastes are known as "tailings". By far, the larger proportion of ore mined in most industry sectors ultimately becomes tailings that must be disposed off. In the gold industry, only a few hundredths of an ounce of gold may be produced for every tonne of dry tailings generated (USEPA, 1994a). Tailings need to be properly managed because they constitute a major source of release of many metals and trace elements into the environment which is a potential source of groundwater contamination.

Several gold mining companies the world over including Gold Fields Ghana Limited (GFGL) Tarkwa Mine rely on the traditional method of cyanide decay (biodegradation and volatilisation) at their Tailings Storage Facilities. According to Anon (2008), Free, WAD and Total cyanide decay in tailings dams occur by natural degradation aided by sunlight. This method of cyanide decay is affected by several factors like rainfall, temperature and depth of burial. Due to the presence of metals and other trace elements in the slurry, the tendency of residual cyanide forming complexes and salt exists (Mudder *et al.,* 1991). It takes several years for cyanide complexes, metals and trace elements to migrate into the groundwater system. The time component of the pathway of seepages may also be affected by the underlain material of a dam. According to Filipek (1999) there are three possible channels for seepages from a tailings dam into the groundwater system. These are;

- Excess water in the supernatant pond (tailings pond) of a TSF if there is not enough aeration during spigoting and less sunlight
- Failure of the under drainage system from the facility
- Seepage through the facility substrata if not constructed and compacted to required standards
- Seepage through the constructed embankment

These seepages are possible channels through which an aquifer can be contaminated through the migration of metals, trace elements and the formation of cyanide complexes and Weak Acid Dissociable (WAD) cyanide (Dzombak, 2006).

1.2 Statement of the problem

Managing a cyanide tailings dam to minimise risks to human and environmental health represents one of the key challenges that continues to face the mining industry. GFGL currently employs the Carbon-In-Leach (CIL)/Heap leach (HL) processing techniques to extract gold from the ores at Tarkwa mining leases in the Western Region of Ghana. The CIL Mill Project was commissioned in October 2004 and expected to be decommissioned in 2017 if a steady throughput of 900,000 tonnes/month is maintained. The mill was upgraded and expanded to nearly double its capacity in 2008 this necessitated the construction of an additional TSF in 2008. Two new TSFs were constructed in the following two years (TSF 2 and TSF 3) but

TSF 1 and TSF 2 were the operational ones at the time of the study. Even though cyanide is toxic to both humans and other living organisms, it remains the main lixiviate used in the extraction of gold in the CIL circuit.

Cyanide consumption is a major component of the total operating cost of a typical gold-producing plant. Only 0.3 - 0.4 g/tonne of cyanide per tonne of typical ore should be required to dissolve and extract the gold. However in practice, consumption ranges from 300 g/tonne to more than 2000 g/tonne for efficient gold extraction (Anon, 2008). The 'excess' cyanide consumed is partly accounted for by oxidation to cyanate and loss through volatilisation as HCN gas. Species of cyanide can be consumed by complexation with copper, iron and zinc, or through reaction with sulphur species to form thiocyanate. Cyanide metal complexes in particular eventually migrate into the wider environment (Anon, 2008). TSF's are designed to provide secure, long-term storage of materials containing such complexes and to avoid potential releases through seepage, overtopping, breaching, and pipe/channel failure. The more toxic forms of cyanide in TSF's are measured as WAD cyanide, free and total cyanide.

Depending on its physical, chemical, and biological properties, a contaminant that has been released into the environment may move within an aquifer in the same manner that groundwater moves (USEPA, 2015).

The time component of the pathway of seepages may also be affected by the attenuation potential of the underlain material of the dam. Historical monitoring data inferred from pH measurements and molar ratios of major cations indicates elevated parameters in boreholes around the tailings dams and cyanide leaching ponds at GFGL. The natural groundwater in the study area has been established slightly acidic with pH values ranging between 4.5 and 5.5 (Kortatsi, 2004), Samples from boreholes in the catchment returned an alkaline pH medium which may be an indication of interaction between the tailings dam and the groundwater system. Similarly, boreholes associated with catchment area have significantly higher elevated electrical conductivity, sodium, magnesium and calcium.

Cyanide and metal levels in the tailings supernatant and decant ponds need to be monitored throughout the life of the mine to predict the impact of seepages on the groundwater system. There is the need to establish the effect of cyanide leaching in the TSF's and their impact on groundwater in their respective basins. The effect of seepages from the TSF's over a period of time may contaminate the groundwater basins and affect community water supply.

1.3 Significance of the study

Metalliferous mining has played an important role in the development of industrialised nations for centuries. According to Anon (2010), the negative effect of over 100 years of mining activities is being experienced in countries like Canada, South Africa and United States of America. Gold export is Ghana's second foreign exchange earner (32%) after Cocoa. Over 30 million ounces of gold has been produced from 2004 to 2014. GFGL has been the highest gold producer over the last decade with annual production of 600,000 ounces, and directly employs over 5,000 Ghanaians with over 10,000 in the supply chain and added value. Gold production is associated with waste production that needs to be managed in order to mitigate natural resource like groundwater. Groundwater is a key natural resource which human, animals and plant species rely on in order to survive.

It takes a considerable amount of time to realise the impact of cyanide leaching on the groundwater system and environment as a whole. During the extraction of valuable metals and industrial minerals from their ores, large volumes of waste products or tailings are normally generated (Ozkan and Ipekoglu, 2002; Álvarez *et al.*, 2005). Since the amount of waste products generated increases with increase in mining activities, the storage of these (tailings) has become a matter of environmental concern for the mining industry due to its harmful effects on the environment and life forms that depend on it.

Ore processing at GFGL Tarkwa Mine has a significant risk on the groundwater system because of the following reasons:

- Chemicals used in the leaching process in association with trace metals and minor elements with the gold ore are potential sources of surface and groundwater pollution
- Pollution of water bodies will affect ecosystems leading to a change in the health status of fauna, flora and communities
- The source of drinking water for the communities in the catchment communities can be polluted from seepage of metals and chemicals into groundwater systems

1.4 Objectives of the research

The main objective of this research is to assess the source and level of groundwater contamination due to the presence of cyanide and metals in the three operational tailings dams at GFGL Tarkwa Mine**.**

To achieve the set main objective, the following specific objectives were met:

- Investigate the cyanide attenuation potential of the sub-strata of the TSF basin
- Investigate the metals and trace elements speciation and cyanide variation at depth in the decant and supernatant pond of the TSF's
- Assess the level of contamination from primary and secondary constituents in the TSF basin and their impact on water supply system

1.5 Significance of the study

Mining, in common with most other industrial activities, has often led to environmental pollution (Bernoth *et al.*, 2000). Soils, groundwater and surface waters have become contaminated, and there is increasing pressure on miners to reduce the risks associated with this contamination. The operations at GFGL Tarkwa Mine are no exception and current deposition of tailings ranges between 450,000 and 500,000 tonne/month on the average. According to Mudder (2001), the tailings slurry in the dams depend on natural degradation of cyanide process solutions, which in turn depends on several factors that most often are affected by the geochemistry of the solid phase (ore). The gold bearing ore at Tarkwa, consists of several metallic species which has the tendency to react with cyanide to form metallo-cyanide complexes. GFGL Tarkwa Mine recovers gold from cyanidation using adsorption on activated carbon. After the gold extraction the slurry which contains residual cyanide, and possible metallo-cyanide complexes and cyanates are stored in the tailings facility. These can migrate into both ground and surface water through seepage and dam failure.

In an attempt to analyse three years of groundwater monitoring, five physicochemical parameters were chosen due to their elevated concentrations in comparison to the baseline data of boreholes around the TSF's. These parameters are sodium, calcium, manganese, pH and electrical conductivity (EC) which suggest a possible physico-chemical reaction in the underlying aquifers. Seepage from the dam may take years depending on the attenuation potential of the clay substrata. According to Anon (2011), all the three TSF's have the same clay liner and similar geochemical parameters. Cyanide concentrations vary with depth, season and temperature which imply the possibility of various levels with respect to depth of burial and distance from the discharge point.

The probability that cyanide salt occur at depth is high and under anaerobic conditions in non-sterilized soil mediums, it is converted to nitrates with the potential to migrate into the groundwater system. Free cyanide concentrations decay with increased depth of burial and this makes migration into the groundwater system difficult (Mudder, 2001). Metals and other trace element concentrations may vary with depth due to association with cyanide. Groundwater in the study area is moderately acidic (4-5), occurs at a shallow depth of up to 20 meters, and has elevated EC and sodium which may suggest metal migration as well as the formation of formic acid or ammonium formate through cyanide hydrolysis or seepage of tailings into the groundwater system (Mudder, 2001). The mitigation of mining's environmental impacts is, therefore, a major challenge for the modern mining industry.

1.6 Scope of work

The study covered the existing and operational TSFs as well as boreholes around the TSF 3 which was under construction at the time the data collection. Soil samples were taken in the TSF 3 area in order to assess the attenuation potential of the substrata. Cyanide decay and metal concentrations in the supernatant pond were measured with respect to depth, whiles the attenuation potential of the substrata was assessed through both geotechnical and geochemical investigations to determine the effectiveness of the natural ground in mitigating seepages into the groundwater system. Physicochemical parameters of the monitoring boreholes around the TSFs were analysed and compared to other cyanide heap leaching facilities in the North and South of the operations.

1.7 Organisation of the study

Chapter one is the general outline and introduction of the study, whiles chapter two focuses on the description of the study area and characteristics of the TSF. Chapter three is the literature review. Chapter four is the research methodology and data collection whiles Chapter five focuses on data analysis and discussion. Chapter six presents conclusions and recommendations drawn from the study.

CHAPTER 2: PROJECT DESCRIPTION

2.1 Location and environment

Tarkwa is a mining town located in south-western Ghana in the Western Region approximately 300 km by road west of Accra, at a latitude 5°15’N and longitude 2°00’W. The GFGL Tarkwa Mine is located 4 km west of Tarkwa (Figure 2.1). The two mining leases of GFGL comprise land areas totalling over 208 km^2 and contain mining concessions for two operating mines (Tarkwa and Damang).

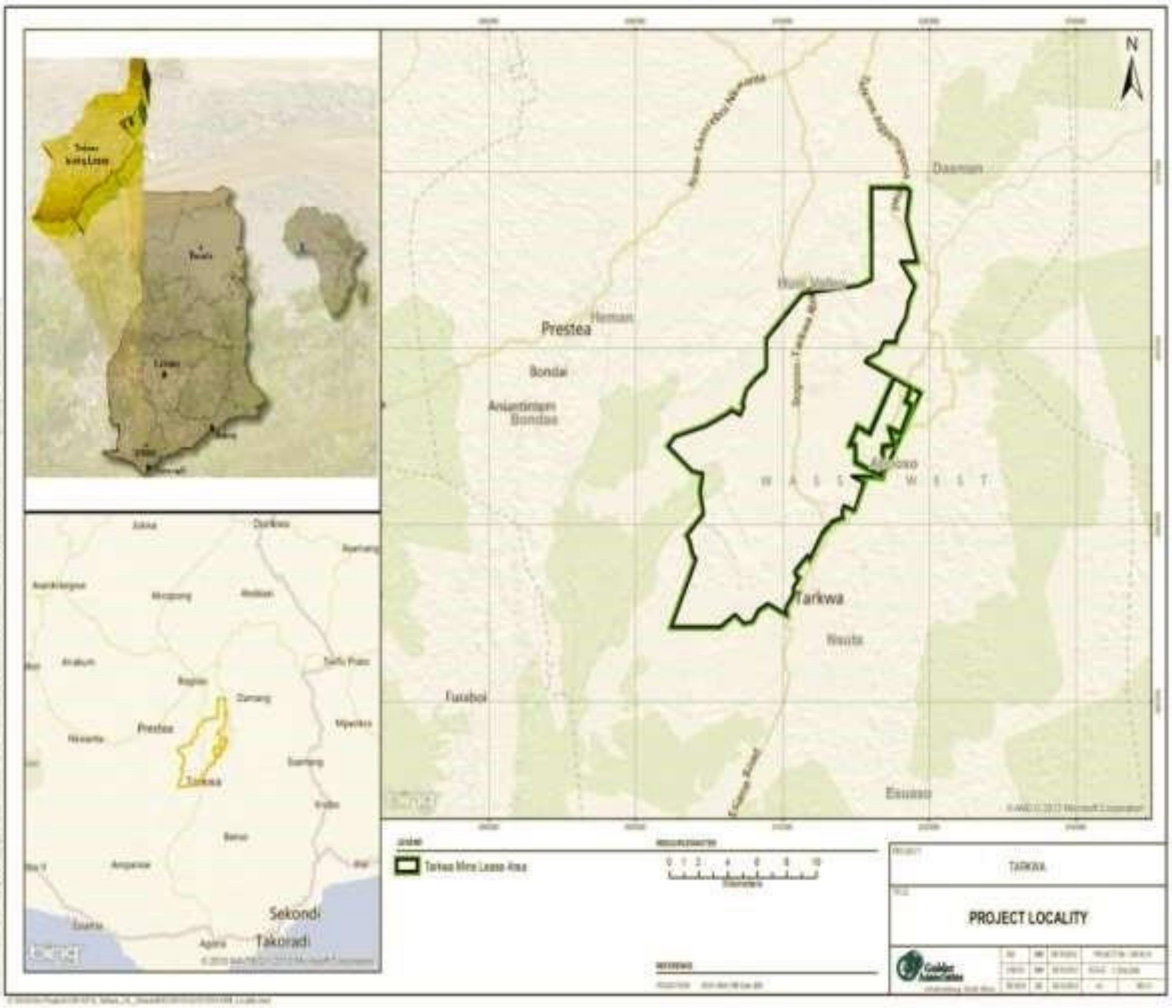


Figure 2.1. General location map of GFGL Tarkwa Mining Lease

The mining leases are located within the Huni River sub-basin of the Ankobra River. Several small villages are situated within the Tarkwa Mine (TGM) concession area.

Villages currently occupied include:

- Samahu
- Abekoase
- Huniso
- Borborbo and
- Essaman
- Tebe

The Tarkwa mine operational area is drained by a number of streams, the largest being the Suman and Abonko streams in the west, Bediabewu stream in the east and several small tributaries of the Bonsa River in the south. The Huni River forms the largest river in the Tarkwa Lease flowing from Bompieso to the north east of the lease. Three expansions of the Tarkwa mine infrastructure have been carried out so far and these are;

- Phase IV leach pad in the northern project area in 2005
- The CIL and South heaps in 2007
- Tailings expansion

2.2 Baseline information

2.2.1 Air quality and climate

The regional climate of the Tarkwa area is typical of areas situated at the same latitudes (approximate 2° North) with a wet tropical climate. The Inter Tropical Convergence Zone (ITCZ) moves over the Tarkwa area twice a year in the periods March – July and September – October, which results in a bimodal rainfall pattern, with peaks during these months. During the dry period (November – February) the climate is dominated by the Harmattan, which is a dry trade wind from the north, which is laden with Saharan dust.

Tarkwa Gold Mine is located within the south western equatorial climatic zone. This zone is described as a moist tropical climate characterised by high year round temperatures and large volumes of rainfall throughout the year. Air quality in the operational area is influenced by the seasonal variation in trade winds and intensity of diurnal rainfall events. Rainfall patterns exert an overriding influence on levels of airborne particulates, notably dust from vehicular movement, exhaust emissions and smoke from "slash and burn" agriculture (Anon, 2010). The meteorological station at Tarkwa, located south of the site provides a long-term database for climate data. In addition, air quality and weather data has been collected at the site since monitoring records began in 1999 (Anon, 2008). Average rainfall over the last 50 years at Tarkwa is 1928 mm although considerable variation exists between years. At the mine site, average rainfall between 2003 and 2010 was 12% less than the 50 year average at 1600 mm but rainfall during 2007 was exceptionally high (2517 mm) and the second highest on record over the last 50 years (Figure 2.2).

Monthly averaged dust measurements expressed as µg/m^3 show a generally poor correlation with monthly rainfall as expected given the strong inverse correlation relationship between daily rainfall intensity and dust generation over subsequent 72 hour periods.

Rainfall on site generally occurs at night or during late afternoons whilst dust measurements taken over short durations are generally undertaken during the day.

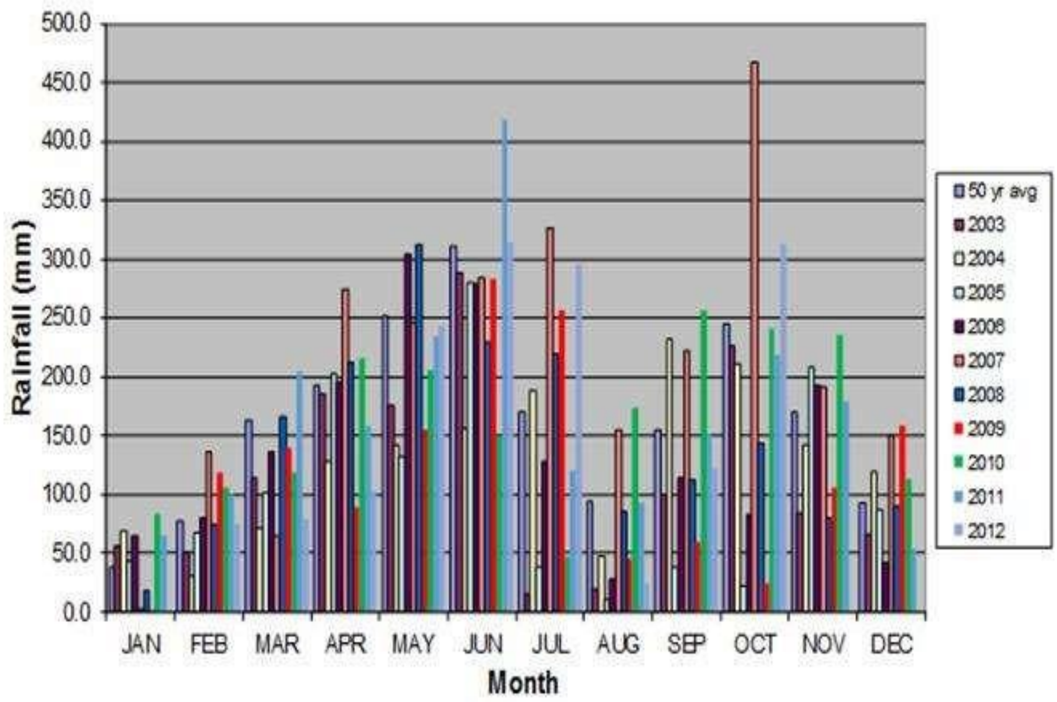


Figure 2.2. Rainfall pattern in the Tarkwa area

Monthly averaged dust concentrations during 2007 peaked during August 2007, representing the influence of major earthwork programs at TSF 2, Phase V leach pad and development of the Akontasi Haul Road system. However, in general terms (and in the absence of time specific major earthworks programs) dust generation during the drier Harmattan months of December-March is significantly greater than all other months. Average daily temperatures vary little throughout the year (24.5°C to 27.8°C) with peak readings in the months between February and April. Table 2.1 shows the monthly rainfall, evaporation and temperature data.

Table 2.1. Monthly rainfall, evaporation and temperature data (1996 – 2011)

Month	MMR 1 mm	MMR 2 mm	MMR3 mm	MMPE	PLEF (n/a)	MMLE (mm)	Net Gain or Loss	Ave Temp (^{o}c)
Jan	40	162	0	132	0.75	99	-59	26.8
Feb	79	191	0	138	0.75	104	-25	27.1
Mar	159	364	27	154	0.75	116	43	27.5
Apr	190	485	61	137	0.75	103	87	26.4
May	247	569	114	137	0.75	103	144	26.6
Jun	297	808	33	109	0.75	82	215	25.6
Jul	169	535	2	111	0.75	83	86	24.1
Aug	88	321	0	120	0.75	90	-2	24.6
Sep	150	493	6.2	121	0.75	91	60	24.9
Oct	241	537	22.1	127	0.75	95	146	25.6
Nov	178	334	49	127	0.75	95	82	26.6
Dec	92	227	0	120	0.75	90	2	26.5

Annual	1,928	2,978	1,137	1,533	n/a	1,150	779	26.0

NB; MMR 1=Mean Monthly Rainfall, MMR2=Maximum Monthly Rainfall, Mean Monthly Pan Evaporation = MMPE, Pan/Lake Evaporation Factor = PLEF, Mean Month Lake Evaporation = MMLE

2.2.2 Surface and groundwater

The general flow of the rivers in the project area is from the north to south towards the Bonsa River. The active mining areas on the Concession occur on hilly ground shedding water in varying directions. As part of the mine development, some rivers crossing the mine site in the vicinity of Phase V heap leach pad and TSFs were rerouted via stream diversions to the west of the project area. Within the mine site catchment area there tends to be a rapid response to rainfall as the terrain is hilly. The removal of the tropical vegetation for mine development as well as farms and artisanal mining activity has decreased the response time and increased the volume of run-off reaching the local rivers (Anon, 2007).

Fracture zones within major rock types (Tarkwa phyilite, Huni Sandstone and Kawere formation) control the groundwater regime in the area. There are little to no primary aquifer characteristics in the meta-sedimentary rocks underlying the Tarkwa area. Rather, aquifer potential is associated with fracture and weathering induced by secondary permeability and porosity, and hydraulic connection between individual fracture sets and weathered zones. Aquifer potential is critically dependent on active recharge, as the groundwater in static storage at any one time, in fractured rock aquifers, is rarely sufficient to sustain long term pumping (Anon, 2010).

The upper weathered zone is a transition zone providing a store of water that may enter the lower fresh rock. The water enters the fresh rock along fractured, leached and weathered areas. However, the amount of groundwater encountered in the Tarkwa area to date is not substantial. The linked fractures with near surface saprolites provide a route for the regional movement of groundwater and several villages in the area draw water from this resource. Medium to high transmissivity and a low to medium storage capacity characterize the local aquifers (Anon, 2010).

2.2.3 Water use

Water sources for domestic use in the area of the mine fall into the following 5 main categories:

- Wells approximately 10 m deep with hand pumps
- Wells approximately 5 to 10 m deep without hand pumps
- Open pits recharged from groundwater (with or without) surface water inflows
- Direct use of surface water
- Groundwater through boreholes

According to Anon (2008), the availability of water in the area tends to be seasonal. During the dry season, many of the shallow wells and some surface streams dry out and the productivity of the various wells and bores decreases. Typical water uses are for drinking, cooking, washing and bathing with other general uses including swimming and fishing. The degradation of the local water supplies is caused by a variety of factors including poor waste disposal (solid and liquid) practices, presence of faecal and coliform bacteria from domestic animals in the villages, erosion/siltation, oxidation reactions with exposed soils as water levels drop and the activities of the artisanal miners (galamsey).

2.2.4 Flora

GFGL Tarkwa mine lies between what has been classified as the Wet Evergreen and a transition zone between the Wet Evergreen and Moist Tropical Forest. The primary forest in the area has been impacted negatively by mining and agricultural activity and has been replaced by secondary vegetation growth. Mining, timber exploitation, firewood collection, charcoal burning, and farming are the major human activities. At the time of mine development the flora assemblage within the area had been extensively disturbed due to land clearing for subsistence agriculture, timber harvesting, artisanal mining and township development. As the land opened up, and access to the interior of the original forest increased, so did the exploitation of the natural resources. Much of the area around the mine site is now comprised of farm lands and some secondary forest at various levels of development although pockets of primary forest remain west of Kotraverchy Pit (Anon, 2011).

- Five distinct categories of vegetation have been observed over the Tarkwa concession area
- Primary forest which occurs in small isolated areas on steep ridges
- Swamp forest which occurs in low lying areas and along river valleys. Approximately 40% of the concession area is comprised of low lying river bottom land areas
- Secondary forests at various stages of succession covering the majority of the concession area
- Rapid vegetation growth on recently abandoned areas
- Grasslands and farming plots

2.2.5 Fauna

As with the flora in the area, the fauna have been extensively disturbed. The removal of habitat, combined with extensive hunting, has reduced the numbers of larger mammals to very low levels. There is a significant decline in biodiversity of species in the study area as result of the mining activities. The destruction of virgin rain forest habitat and the concurrent development of commercial and subsistence agriculture have triggered a significant decline in biodiversity and corporate policies aimed at preservation of wildlife have successfully ensured the survival of a very large colony of fruit bats in active mining areas (Anon, 2011).

Less common animals that have been recorded in the area include monkeys, squirrels, duiker, antelope, grass cutters and giant rats which are often hunted or trapped. Several animals occurring in the concession area, namely pangolins, giant forest squirrels, black and white columbus monkey and bird species.

2.2.6 Topography

Topographical relief, climate, vegetation and geology affect soil development in the area. Within the area, there is a tendency for the ridges of hills to be orientated in the southwest to northeast direction. The soils in the area are affected by rocks of the Tarkwaian system consisting of the Kawere Group, Banket Series, Tarkwa Phyllite and the Huni Sandstone. The rocks weather under the high rainfall regime to mainly medium to fine textured soils. In addition to the two broad categories of soils (the forest oxysols in the south and the forest ochrosol-oxysol), relief affects the development and depth of the soils. On the summits and upland sites, soils tend to be shallow (<30 cm) and well drained. On the mid-slope areas, soils are deeper (<200 cm) and imperfectly drained with the soils in the valleys being deep (>200 cm), grey and mottled grey, and imperfectly drained. In general, the soils are highly leached of bases and are slightly acidic with poor levels of nutrients. The principal land uses in the Concession are mining, agriculture, aquaculture, and forestry (Anon, 2011).

Agricultural activities include:

- Cash crop plantations of oil palm, maize, bananas and coconut palms

- Arable crops of plantain, cocoyam, and cassava and cereals
- Horticultural crops for domestic use, including citrus, corn, banana and coconut palms
- Home gardens of other fruits and vegetables including tomatoes, cabbage, and garden eggs

2.2.7 Regional geology

Ghana is underlain partly by what is known as the Basement Complex. It comprises a wide variety of Precambrian igneous and metamorphic rocks which cover about 54% Ghana, mainly in the southern and western parts. It generally consists of gneiss, phyllites, schists, migmatites, granite-gneiss and quartzites. The Basement complex is further divided into different sub provinces including the metamorphosed and folded rocks of the Birimian and Tarkwaian systems (Gyau-Boakye *et al.*, 2000). In several places these systems are intruded by sills and dykes of igneous rocks ranging from felsite and quartz porphyry to metadolerite, gabbro and norite (Kortatsi, 2004).

The geomorphology of the Tarkwa-Prestea area consists of a series of ridges and valleys parallel to each other and to the strike of the rocks. The strike of the rocks is generally in the NE-SW direction (Kesse, 1985). Both the Tarkwaian and Birimian systems are folded along axes that trend northeast (Kesse, 1985). The general type of topography reflects underlying geology (Kesse, 1985). The Tarkwaian system is an elongated and narrow syncline about 250 km long and 16 km wide (Kesse, 1985). The system consists of slightly metamorphosed, shallow-water, sedimentary strata. It is chiefly sandstone, quartzite, shale and conglomerate resting on and derived from the Birimian System (Kesse, 1985). Intrusive igneous rocks contribute to about 20% of the total thickness of the Tarkwaian System in the Tarkwa area. These range from hypabyssal felsic to basic igneous rocks (Kuma, 2004). Granitoids of the Dixcove Granitoids systems have also intruded the Tarkwaian System in many places (Kortatsi, 2004). A detailed geological map is presented in Figure 2.3.

The TSF site is generally underlain by Huni Sandstone with the south western boundary of the site formed by a ridge of Tarkwa Phyllite. In addition more recent Quaternary deposits formed by the fluvial system in the area overlie these (Anon, 2011).

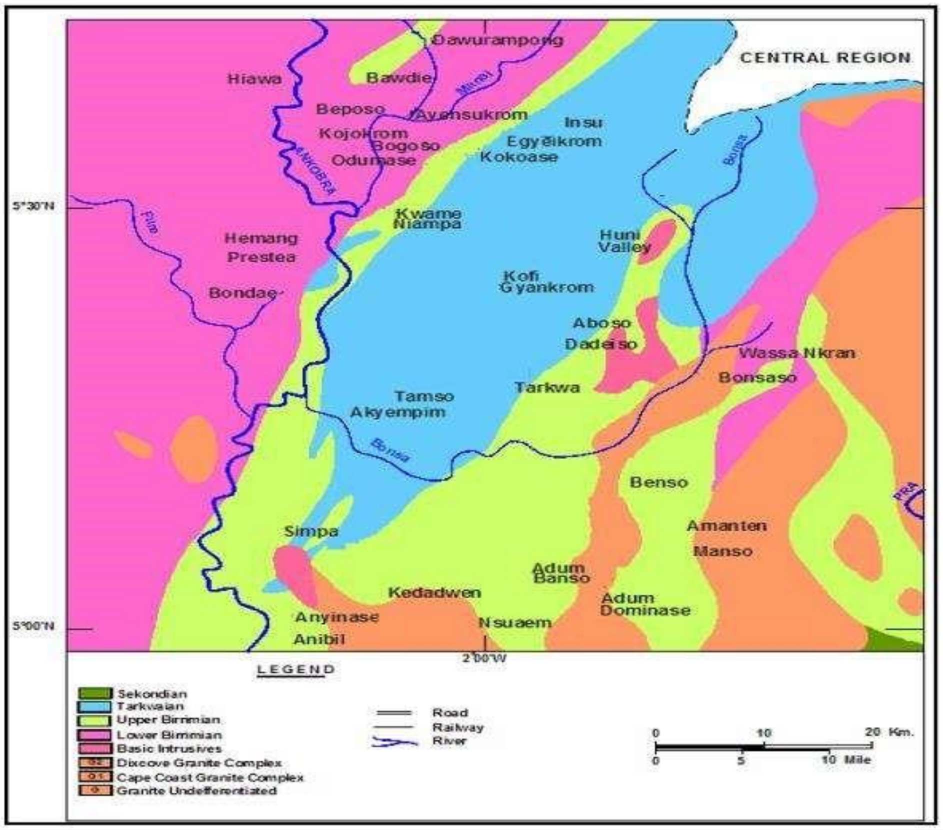


Figure 2.3. Geological map of the Tarkwa-Prestea area (Kortatsi, 2004)

The geology has led to three distinct topographical areas within the site as follows (see Table 2.2):

- Relatively steep slopes dipping to the east and north east along the ridge of Tarkwa Phyllite

- Undulating knolls generally comprising laterites underlain by Tarkwa Phyllite closer to the ridge, and Huni Sandstone on the remainder of the site
- Areas dominated by the fluvial system (water courses and adjacent flood deposits) which are generally soft and wet swampy areas

Table 2.2. Division of the Tarkwaian System

Series	Thickness (m)	Composite Lithology
Kawere Group	250-700	Quartzites, grits, phyllites and conglomerates
Banket Series	120-160	Tarkwa phyllite transitional beds and sandstones, quartzites, grits, breccias and conglomerates
Tarkwa Phyllites	120-400	Huni sandstone transitional beds, greenishgrey phyllites and schists
Huni Sandstone	1370	Sandstones, grit and quartzites with bands of phyllite

Source; Kesse, 1985

2.3 Processing methods used by the GFGL

Two main methods are used by GFGL Tarkwa Mine to recover gold from the ore. The method used in the extraction of the gold from the ore depends on the geological formation and the type of ore. The CIL and the heap leach (HL) methods are being employed to recover the gold from the ore (Anon, 2007). The cyanide solution strength is important in leaching the gold with a typical concentration range of 0.020.05 % NaCN commonly used as the complexing agent and also the alkalinity of the solution with the optimum pH being 10.3 (Barsky *et al.*, 1962). To facilitate the leaching of gold by cyanide, there should be enough oxygen supply throughout the reaction period. The decomposition of cyanide by carbon dioxide and ground acids resulting in the production of hydrogen cyanide gas is minimised by using sufficient alkali such as lime (CaO) or caustic soda (NaOH) in the leach solution to maintaining the acidity in the range of 9 to 11. The gold in the pregnant cyanide solution is recovered by adsorption on activated carbon. Although activated carbon has been used in gold-silver recovery from cyanide solutions for several decades, the mechanism of gold adsorption on activated carbon is still not fully understood

(Barsky *et al.,* 1962). The generally accepted chemical reaction for several decades known as Elsner's equation representing the dissolution of gold by cyanide is represented as follows:

$$4Au+8(NaCN)+O_2+2H_2O \rightarrow 4NaAu(CN)_2+4NaOH \quad (2.1)$$

The choice of any of the processing methods depends on the following:

- Porosity of the ore
- Dissolution rate of the ore and
- The ore grade

2.3.1 The CIL process description

The ore from the mine pits is transported to the crusher pad and tipped into the crusher which is then passed into a bin and fed into an apron feeder. The apron feeds the ore via a 3 tier conveyor system onto a stockpile. Underneath the stockpile is a reclaim tunnel which houses 3 apron feeders that feed a conveyor belt which in turn feeds a semi autogenous (SAG) mill. The slurry discharged from the mill has 60% passing through the 75 microns which implies a 40% solid ratio. Particles with sizes greater than 12 mm, known as scats are deposited onto the stockpile and

intermittently returned to the milling circuit. The slurry passing through the screen is then pumped to the cyclone classification circuit where coarse particles are separated from the fine products.

The coarse fraction is returned to the mill whilst the fine particles gravitate to a thickener via trash removal screens. In the thickener, the slurry is thickened from 22% solids to 50% solids and pumped out of the thickener to the carbon in leach tanks. The CIL circuit consist of 8 tanks in series of which seven contain activated carbon. Cyanide (NaCN) is added to the circuit to ensure dissolution of the gold and get adsorbed on the carbon. The slurry overflowing the last CIL tank forms the tailing which is pumped to the tailings storage facility. The carbon loaded with gold is recovered and cleaned of slurry. The carbon loaded with gold is then passed into an acid wash column where it is washed with hydrochloric acid (HCl) solution to remove calcium.

The carbon is then pumped into the elution column where acoustic solution at 120 ^{o}C is used to remove the gold from the carbon solution. The eluted carbon is passed through the regeneration Kiln which operates at a temperature of 700 ^{o}C to remove any organic fouling off the carbon. The gold bearing solution is pumped through the electro winning cells where the gold plates onto stainless steel cathodes as gold particles. The gold is then removed from the cathodes with high pressure sprays and dried before being smelted into an induction furnace at 1400 ^{o}C. Fluxes are added before smelting to remove all impurities from the gold. The molten gold is then poured into molds and allowed to cool down into bars. The CIL operation is represented by a flow chart in Figure 2.4.

2.3.2 The HL process description

i. Crushing and screen

The crushing plant is designed to crush the ore and reduce it to between 12.5 -19.0 mm product. The ore is directly tipped into a primary gyratory crusher to reduce the product to sizes 150-250 mm. The crushed product is fed into an apron feeder onto a conveyor to be transported to a secondary crusher. The secondary crushers are fed via screens of aperture of sizes of 50 mm and 19 mm from the upper and lower deck respectively. The secondary crushers which are gyratory in type crushes to product of sizes 37.5 mm. The under size from the screens (19 mm) joins the final product stream via another conveyor belt.

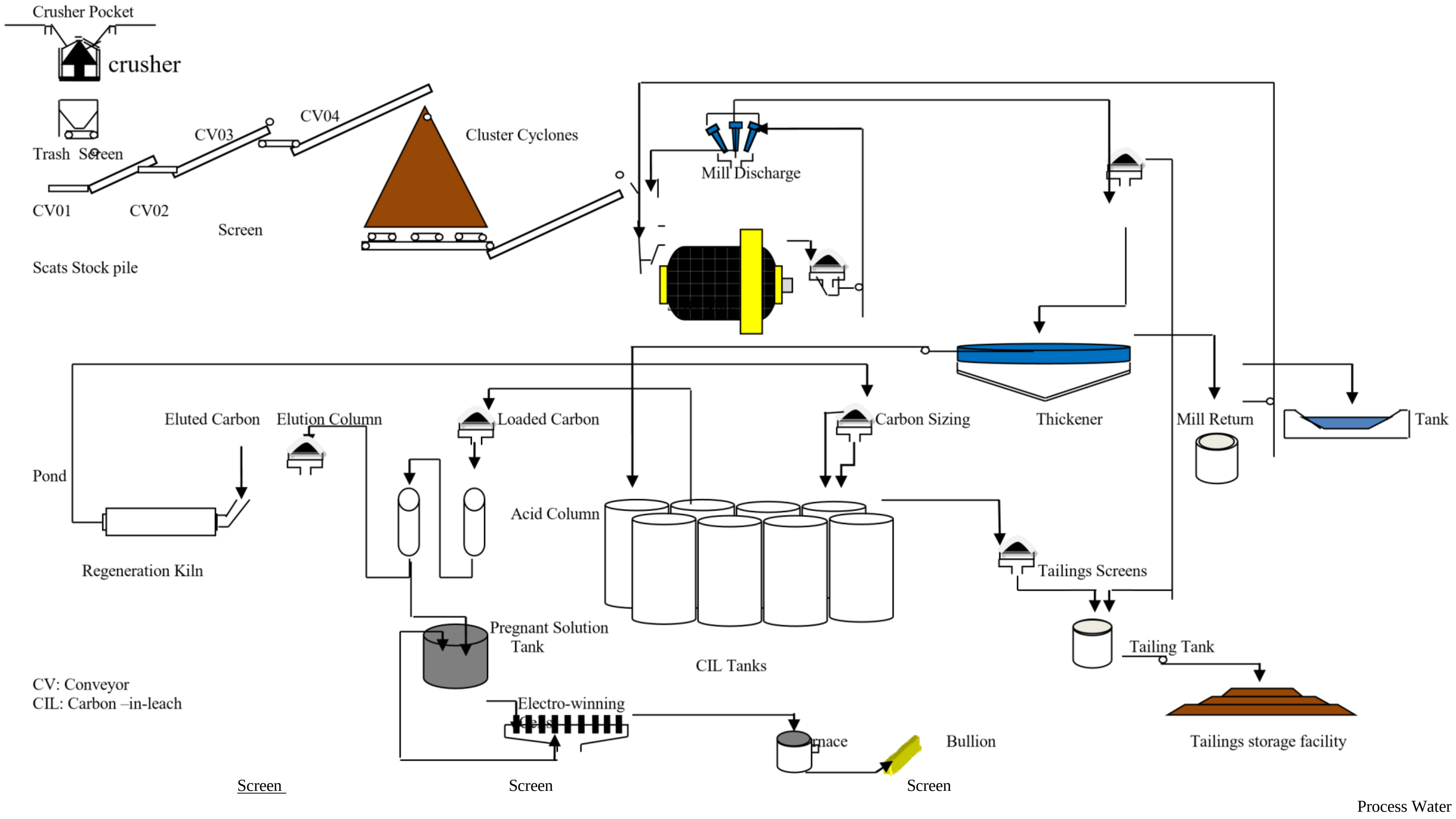

Crusher Pocket
crusher
Trash Screen
CV03
CV04
Cluster Cyclones
Mill Discharge
CV01
CV02
Screen
Scats Stock pile
Eluted Carbon
Elution Column
Loaded Carbon
Carbon Sizing
Thickener
Mill Return
Tank
Pond
Acid Column
Regeneration Kiln
Tailings Screens
Pregnant Solution Tank
Tailing Tank
CIL Tanks
CV: Conveyor
CIL: Carbon –in-leach
Electro-winning
Cells
rnace
Bullion
Tailings storage facility
Screen
Screen
Screen
Process Water

Figure 2.4. CIL process plant flow diagram (After Anon 2007)

The over size is fed back into a tertiary cone crusher and the product fed into tertiary screens of sizes 19 mm. The final crushed product is transferred to agglomeration via conveyor belts which are fitted with belt scales. The crushed ore stream is then discharged to a covered stockpile at the agglomeration plant via conveyor belt (refer to Figure 2.5).

ii. Agglomeration stacking and leaching

The ore that has been crushed to the designed product size of 12.5 mm (80 %) is placed on heaps in a conical pile. The agglomeration stockpile is designed for a live capacity of 1,350 tonnes.

The ore passes over belt scales and under cement silos that add cement at a rate of 4.0 kg/tonne of ore to bind the agglomerates. After being discharged into rotary agglomeration drums, barren process solution made of cyanide concentration of 1000 ppm is sprayed onto the ore to provide a moisture content of 5-10 % for agglomerate formation and leaching process initiation solution. The crushed and agglomerated ore is then conveyed to the leach pad and stacked using Ramp and Grasshopper conveyors, a Loading and Horizontal feed and a Radial Stacker. Leach solution is then applied to the heap after the heap is left to cure for about 3-4 days at a solution application rate of 10 litres /hr/m^2 conveyor at a pumping rate of 365-440 m^3/hr and NaCN consumption rate of 0.2 kg/tonne on the average.

The leach solution is collected after it has passed through the heap and directed into ponds based on a properly coordinated solution management practice. The ponds are constructed with 2 mm HDPE membranes liner over HDPE geogrid at the pond bottom over a 1.5 mm HDPE membrane liner over geotextile at the pond bottom and crest with a leak detection system at low point in the ponds.

The ponds include; pregnant solution ponds, intermediate solution pond, barren solution pond, excess solution pond, treatment solution and containment ponds designed to contain

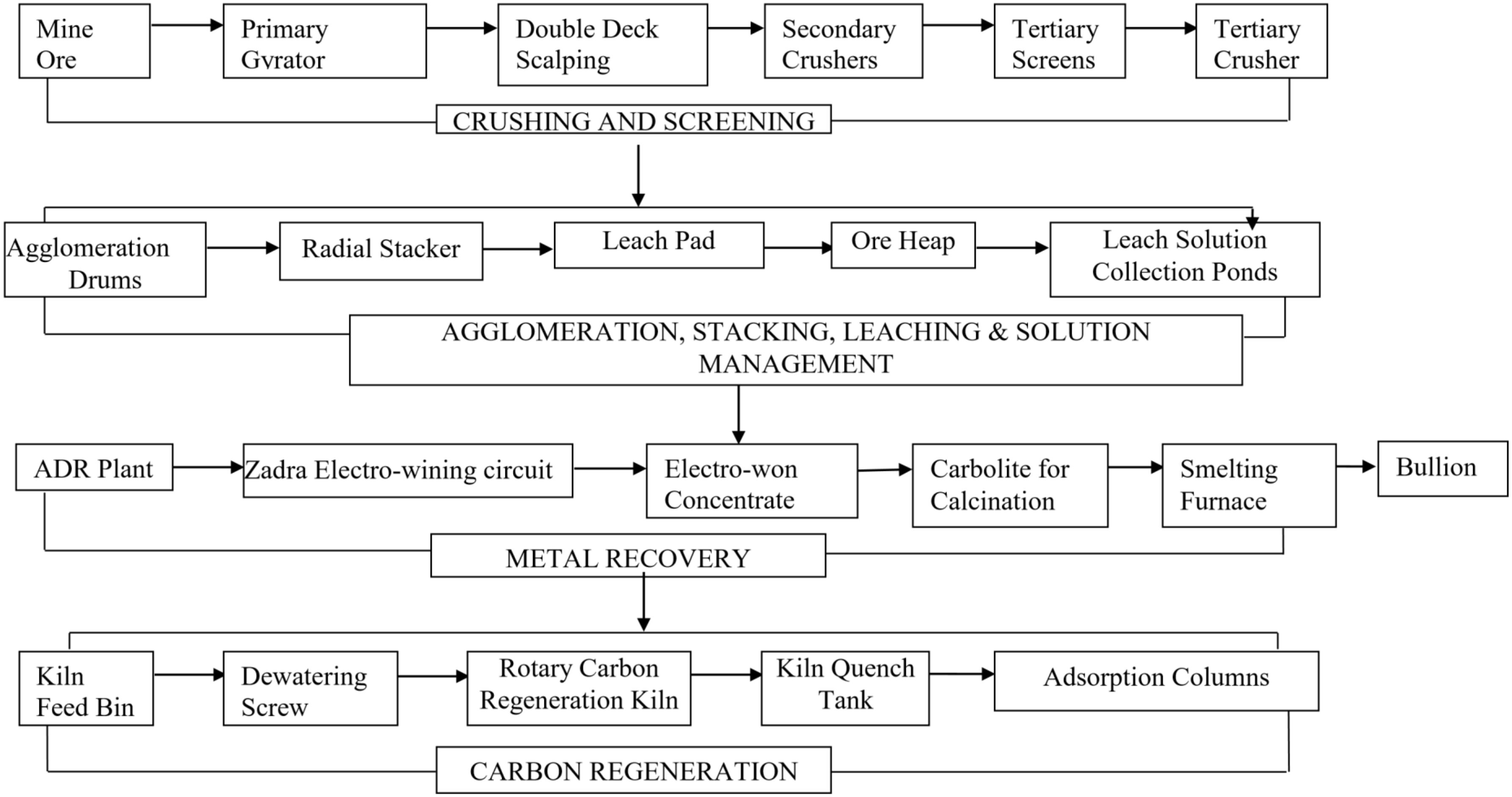

Mine Ore
Primary Gvrator
Double Deck Scalping
Secondary Crushers
Tertiary Screens
Tertiary Crusher
CRUSHING AND SCREENING
Agglomeration Drums
Radial Stacker
Leach Pad
Ore Heap
Leach Solution Collection Ponds
AGGLOMERATION, STACKING, LEACHING & SOLUTION MANAGEMENT
ADR Plant
Zadra Electro-wining circuit
Electro-won Concentrate
Carbolite for Calcination
Smelting Furnace
Bullion
METAL RECOVERY
Kiln Feed Bin
Dewatering Screw
Rotary Carbon Regeneration Kiln
Kiln Quench Tank
Adsorption Columns
CARBON REGENERATION

Figure 2.5. Heap leach process flowchart (2007)

solution based on accepted solution management practices and to contain excesses in case of heavy rainfall. Centrifugal pumps are used in pumping the solution. The pumps are fitted with floating intake lines so that the solution is withdrawn from the upper surface of the pond.

ii. Metal recovery

Gold is recovered from the pregnant leach solutions through activated carbon Adsorption/Desorption/ Recovery (ADR) plant. The pregnant solution flows in a sequence of 5 up flow closed-top carbon columns arranged in series. Each carbon column contains on the average of 2 tonnes of activated carbon.

Desorption utilizes a pressurised elution process and its sized for 4 tonnes carbon, after acid washing using 3 % hydrochloric acid (HCl) for a period of 4-6 hours to remove any carbon foulants. Hot caustic pressure at a temperature of 135 ^{o}C and a pressure of 350 kPa with an elution flow rate of 1.5 bed volumes/hour and bed volume of 2.3 m for about 8 hours.

Metal recovery is done using the Merrill Crowe Zinc precipitation system at a maximum precipitation temperature of 90 ^{o}C and zinc consumption rate of 2.5 g. The precipitate filter is the recessed plate type and with a press feed rate of 25 m^3/hour gold recovery is greater than 95 %. The filtered cake is washed of the press cloth and calcined at a temperature of about 500 ^{0}C for smelting. The calcine is fluxed using Silica, Borax, sodium nitrate, soda ash and fluorspar based on acceptable solution composition standards. The smelting of the calcine is done using a 660 kg red brass working capacity diesel fired tilting crucible with wet scrubber furnace. The bullion produced is sampled and weighed and the samples sent to the laboratory for assaying to recover the gold. The chemical reaction describing how the gold is recovered from the solution is given by equation 2.2 where zinc reacts with cyanide to form a complex.

$$2Au(CN) + Zn \square 2Au+Zn(CN)^{-2}4 \qquad (2.2)$$

2.4 Tailings storage facility description

The typical particle size distribution and Atterberg Limits of the CIL tailings from the feasibility report by Metago (Anon, 2008) is shown in Table 2.3 and 2.4.

Table 2.3. Particle size distribution of tailings

Particle size	Value % by mass
0.250 mm	100
0.075 mm	60
0.010 mm	30
0.002 mm	13

Table 2.4. Atterberg limits of tailings

Atterberg limits	Value (% by mass)
Liquid Limit	24
Plastic Limit	19 5
Plasticity Index	
USCS Classificstion	ML-CL (clayey silt)

Table 2.3 indicates that 60% of the product passes the 75 micron sieve size which means there are some 40% fine sand fractions. GFGL Tarkwa Mine previously advised that the tailings grading (particle size distribution) will always be coarser than 70% passing 75 micron i.e. there will always be a 30% or more sand fraction in the tailings reporting to the tailings facilities. It is important to note that if the tailings particle size distribution gets finer than that shown in Table 2.3, the limiting rates of

rise will decrease (i.e. a finer grading will lower the recommended maximum rate of rise from 2.0 m/year to typically 1.8 m/year which will render the TSF 1, TSF 2 and TSF 3 facilities inadequate (both in the short and long term) as illustrated in Figure 2.6. The Particle Specific Gravity (Gs) and slurry Relative Density (RD) were provided by GFGL as 2.72 and 1.45 t/m^3 respectively.

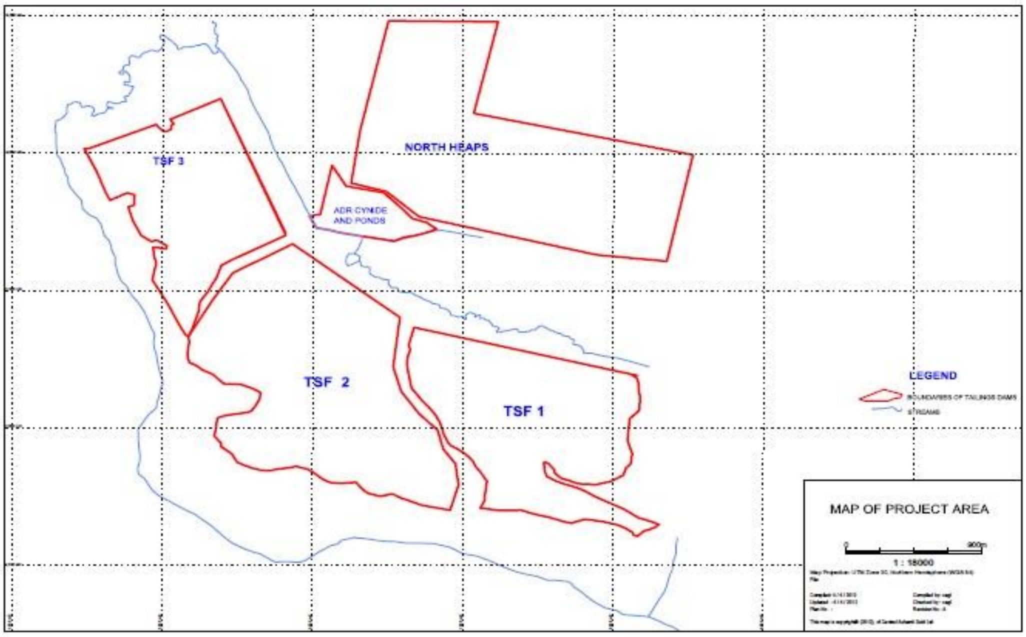


Figure 2.6. Schematic drawing of TSF (TSF 1, 2, and 3)

The assumed in-situ dry density of the deposited tailings was taken from Metago (2009b) and is as follows:

- TSF 1, TSF 2 and TSF 3 – average dry density of1.2 t/m^3
- TSF 4 (co-disposal) – average dry density of 1.15 t/m^3

Various geotechnical properties of the tailings slurry (as pumped to the TSFs) and consolidated tailings within the TSFs are provided in Table 2.5 and Table 2.6 respectively which have been calculated from the above values. The particle size distribution for the tailings produced by the existing CIL plant is presented in Figure

2.7. The information is sourced from sieve and hydrometer tests completed on a sample of tailings taken from the plant feed. The tailings has a larger sand content than the typical gold tailings (Anon, 2006) at about 50%, with the silt (37%) and clay (13%) contents being comparable to the typical tailings.

Table 2.5. Geotechnical properties of tailings slurry

Description	Unit	Low RD	High RD
Slurry RD	t/m^3	1.40	1.45
Void Ratio (e)	n/a	3.30	2.82
Moisture content	%	121	104
% of Solid Mass (w)	%	45	49
% Water by Mass	%	55	51
Volumetric Concentration (C_v)	%	23	26
Dry Density	t/m^3	0.63	0.71

Source Metago 2011

The particle specific gravity is 2.72, which is close to typical values of gold tailings associated with conglomerates.

Table 2.6. Geotechnical properties of consolidated tailings

Description	Unit	TSF
Average Dry Density	t/m^3	1.30
Average Bulk Density	t/m^3	1.76
Moisture Content at Saturation	%	50
Average Void Ratio (e)	n/a	1.31

Source Metago 2011

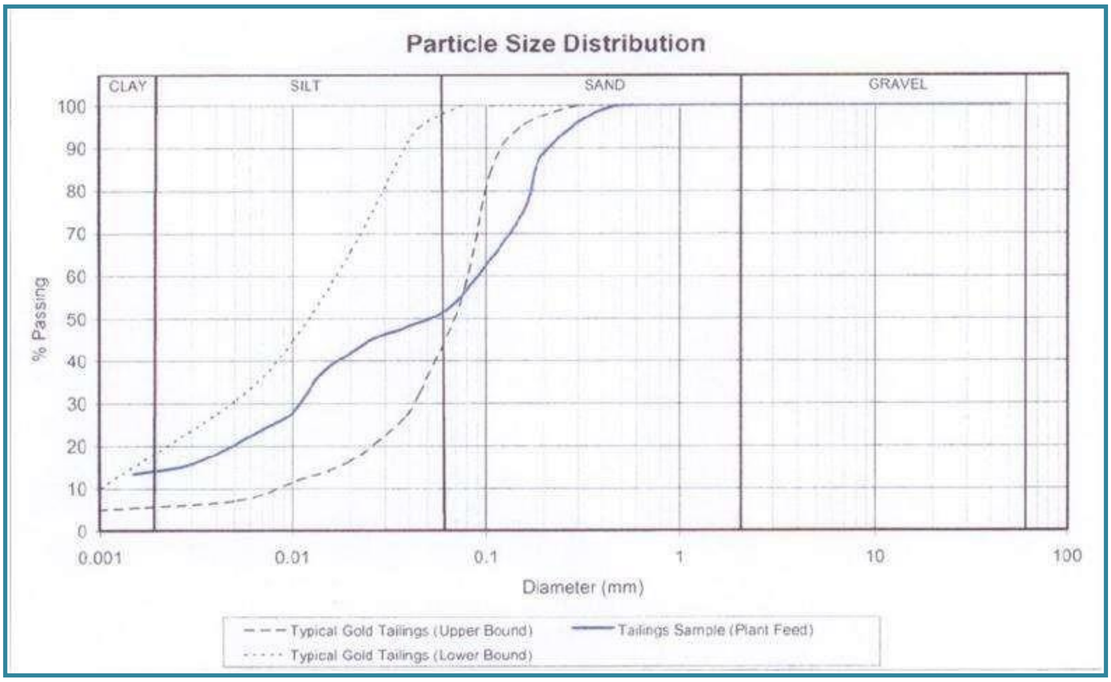


Figure 2.7. Particle size distribution of plant feed sample (Metago 2011)

Although the sand is higher than the typical tailings enveloped, the silt and the clay fractions are considered to control the behaviour, which is thought to be similar to most other gold tailing encountered.

The Atterberg Limits indicates that the tailings have a low plasticity and classify as an ML – CL material (clay silt with a low plasticity) using the Unified Soil Classification System (USCS), compared to an ML (low plasticity silt) type tailings typically found on most gold mines (Table 2.7). The CL material (clay) arises from the weathered nature of ore in the open pits.

Table 2.7. Geotechnical analysis

Grading (% passing)	
Clay < 0.002 mm	13
Sand 0.002 – 0.06 mm	39
Sand 0.06 – 2.0 mm	48
Gravel 2.0 – 60 mm	0
Boulder > 60 mm	0
Atterberg Limit	
Liquid Limit	24
Plastic Limit	19
Plasticity Index	5
USCS	**ML - CL**

2.4.1 Rate of rise limits

According to Anon (2011), under continuous saturated conditions (e.g. under a pool) the following rates of rise for the TGM tailing dams (TSF1, TSF2 and TSF3) were determined such that the settled material gains sufficient dry density to develop reasonable shear strength:

- 0.075 m/year to 0.25 m/year for an impervious foundation (i.e. clay foundation with no under drainage system, where consolidation is relatively slow)
- 0.35 m/year to 1.65 m/year for a fully drained base (i.e. under drainage to the base of the placed tailings which is fully operational at all times and does not clog or become redundant)

The determinations of the range of acceptable rates of rise were based on the following:

- The coefficient of consolidation (C_v) for gold tailings will vary from 20 to 60 m^2/year. These values were obtained from test work undertaken on the

  Tarkwa gold tailings by Metago (20011) and Anon (2002). The assumption that continuous saturated conditions exists e.g. under submerged conditions where there is no influence of rainfall, evaporation and evapotranspiration

The above values are very low due to the effect of saturated conditions and lack of moisture reduction in the tailings from sun drying. The effect of sun drying (or desiccation drying) dramatically improves tailings consolidation due to the forced removal of pore water by evaporation, and then evapotranspiration once the material becomes partially saturated, thereby increasing the allowable rate of rise.

A comparison between these rate of rise values and the current design rate of rise (2 to 2.2 m/yr) indicates that tailings material in and under the supernatant pond of the storage facilities will never consolidate properly and hence will always have low dry density values (of the order of 1 tonne/m^3) for the life of mine (Anon, 2011). This also indicates that materials in the pool area are likely to be susceptible to the liquefaction. Apart from reducing the risk of overtopping, the low dry density under the pool and associated liquefaction potential are strong motivators to maintain as small pool as possible at all times.

2.4.2 Rate of rise limits for desiccation drying

The basic assessment of the effect of sun drying due to evaporation and evapotranspiration, without resorting to direct field measurement and geotechnical modelling, is presented in the following:

- According to Anon (2011), the main gold mining areas of South Africa and Australia have evaporation (S Pan) rates in the order of 1750 mm/yr or more, and net loss (evaporation exceeds rainfall) of 500 mm/yr or more. The maximum safe rate of rise for gold tailings is normally set at 2.5 m/yr. This figure is not only adopted for slope stability reasons but also for operational purpose to allow personnel and equipment on the surface of the tailings dam to successfully construct so called perimeter "day walls" using the tailings materials itself for upstream development

- At Tarkwa the evaporation is 1530 mm/yr and has net gain (rainfall exceeds evaporation by 420 mm/yr). A direct comparison between the two climates in terms of rainfall and evaporation indicates that the gold tailings at Tarkwa should be deposited at a rate of rise below 2.5 m/yr
- Blight (1998) used solar radiation and energy balance methods between Tarkwa and the gold mining area of the Witwatersrand to make a rough comparison between the evapotranspiration rates for gold tailings in the two areas over a 6 month period. The results show that the evapotranspiration rate from the tailings in Tarkwa is within 25% less compared to that of the Witwatersrand area. Hence the maximum safe rate of rise at Tarkwa, by mere proportion, should be about 1.8 m/yr

Based on these assumption, an ideal average rate of rise of the TSF should be in the order of 1.8 to 2.0 m/yr, increasing to a maximum permissible rise of 2.2 m/yr provided intensive monitoring and instrumentation is employed and a strict management system is imposed (refer to Table2.8). The choice of an acceptable rate of rise for gold tailings in upstream development should take into account the risk associated with a TSF failure and possible flow slide should the failure extend back into liquefiable material (Anon, 2011).

Table 2.8. Summary of rate of rise criteria for TGM (After Metago, 2011)

Rate of Rise Criteria	Rate of Rise Limitation
☐ Maximum Rate of Rise at which construction of waste rock sidewalls is considered feasible	2 m/yr

Permissible Rate of Rise During Operation of the Facility	1.8 – 2.2 m/yr
Maximum Permissible Rate of Rise at the Closure of the Facility	2.2 m/yr

2.4.3 Tailings discharge rates

Tailings production rates are split between the three facilities in the following way:

- Prior to the CIL expansion and the commissioning of TSF 2 in July 2008, all production (450 000 t/month) was deposited in TSF 1
- After the CIL expansion was commissioned and operating at the design tonnage of 1000 000 t/month (Anon, 2008), the deposition in TSF 1 was reduced to its design tonnage of 350 000 t/month, and the remaining production (650 000 t/month) deposited in TSF 2
- TSF 3 was commissioned in June 2011, after which the stream of tailings being deposited in TSF 2 was split pro-rata between TSF 2 and TSF 3. The pro-rata split was based on the ratio of the remaining volumes of TSF 2 and TSF 3 respectively. The deposition rate for the three TSFs are as follows:
    - ➢ TSF 1: 350 000 t/month
    - ➢ TSF 2: 377 000 t/month
    - ➢ TSF 3: 273 000 t/month
- TSF 1 comes to the end of its present life in December 2017, if the above production rates are followed. The entire tailings stream will then be deposited in TSF 2 and TSF 3, again being split pro-rata based on the remaining design volumes. Numerically, the two streams will be approximately:
    - ➢ TSF 2: 618 000 t/month and
    - ➢ TSF 3: 382 000 t/month

The results of the rate of rise and the stage capacity calculations are summarised for each tailings facility in Table 2.9 and locations in Figure 2.8.

Table 2.9. Tailings production distribution (After Metago, 2011)

Date	CIL Total Expansion (tonnes)	Deposition on TSF 1 (tonnes)	Deposition on TSF 2 (tonnes)	Deposition on TSF 3 (tonnes)
From 2003-May 2008	450,000	450,000	-	-
June 2008 – May 2011	1,000,000	350,000	650,000	
From June 2011	1,000,000	350,000	377,000	273,000
From December 2017	1,000,000	0	618,000	382,000

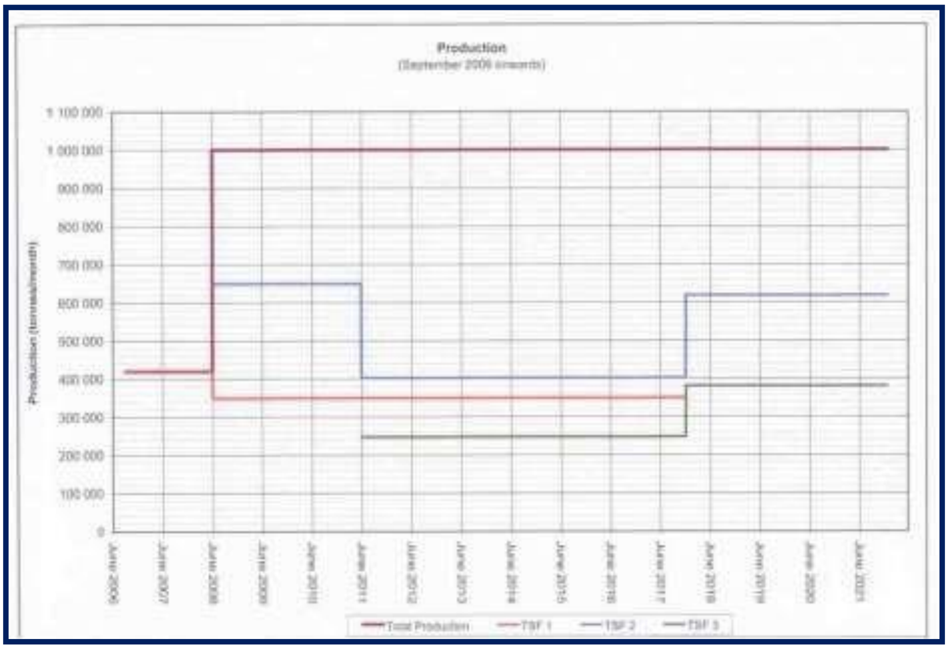


Figure 2.8. Summary of tailings production (Metago, 2011)

2.5 TSF underdrainage system

The main function of an underdrainage system is to:

- maintain or improve the stability of the TSF slopes by drawing down the phreatic surface within the TSF.
- to prevent seepage through the toe or outside slope of the TSF which would result in damage to the rehabilitated side slope and contamination of surface run-off; and

- to reduce seepage to the underlying foundation which reduces the extent of contamination of the local groundwater.

The underdrainage system for TSF 3 is based on both TSF 1 and TSF 2 designs and utilises finger drains (herringbone drains) to drain the low-lying areas within the TSF basin and a toe drain along the upstream toe of the containment wall.

The herringbone and toe drains ultimately report to an underdrainage tower (via the collector pipe system) with a sump from where drained water is pumped to the TSF beach via a submersible pump from where it will flow towards the supernatant pool.

2.5.1 Finger drains

Each finger drain consists of a 100mm diameter slotted corrugated HDPE pipe surrounded by 6mm stone wrapped in a geotextile and covered with a 150mm layer of filter sand or coarse tailings (borrowed from the existing TSF1 or TSF2) within a 500mm deep v-cut trench. Generally finger drains are spaced at 25m and are connected to a collector pipe system which conveys seepage water to the under drainage tower. Each finger drain is generally less than 100m in length and has a minimum fall of 1V:200H (0.5%).

2.5.2 Toe drains

The subsurface portion of the toe drain consists of one to three160mm diameter slotted corrugated HDPE pipes surrounded by 19mm stone in a 750 mm minimum depth trench. At the surface the toe drain is covered with a 300 mm thick layer of 6 mm stone and a 500 mm thick layer of filter sand or coarse tailings between two 800 mm high earth bunds to form an 8m wide drain. A geotextile is placed around the outside of the filter material as shown on the typical toe drain section. Each toe drain trench invert generally has a minimum fall of 1V:200H (0.5%). The sections of toe drains are either connected to the collector pipe system or directly to the underdrainage tower.

2.5.3 Pipe collection system

The collector pipe system collects seepage water from the toe and finger drains and conveys water to the underdrainage tower. The collector pipe system consists of smaller collector pipes which collect seepage from the finger drains, and larger main collector pipes which collect water from the smaller collector pipes, some finger drains and some toe drain sections. Each 700 mm deep collector pipe trench consists of one100 mm diameter slotted corrugated HDPE pipes surrounded by 19 mm stone 350 mm thick and wrapped in a geotextile (Table 2.10).

Table 2.10 Underdrainage construction (After Metago, 2008)

Description	Value	Unit
Max Seepage to Drainage Pipes from Seepage Analysis		
Toe Drain	14.1	l/hr/m
Finger Drain	2.9	l/hr/m
Design Assumptions/Parameters Minimum		
Slope	0.005	m/m
Surface Roughness (k_s)	0.03	mm
Kinetic Viscosity	1.13E-06	m^2/s
Corresponding Full Flow Capacity of 100mm Pipe	5.3	l/s
Corresponding Full Flow Capacity of 160mm Pipe	15.8	l/s
Factor of Safety for Pipe	10	n/a

Number of Drain Pipes Required Determined Considering

Underdrain	**Pipe Size (ID – mm)**	**Pipe Length (m)**
Toe Drain (max allowable length of single pipe)	160	402
Finger Drain (max allowable length of single pipe)	100	670
Collector Pipe (max allowable length of single pipe)	100	670
Main Collector Pipe (max allowable length of single pipe)	160	1,976
Main Collector Pipe (max allowable length of single pipe)	160	402

The remainder of the trench is backfilled with un-compacted earth. Each main collector pipe trench consists of one to four 160 mm diameter pipe. Slotted corrugated HDPE pipes are surrounded by 19 mm stone 350 mm thick and wrapped in geotextile. This is then covered with a varying thickness of compacted spent ore and a 350 mm thick layer of un-compacted earth at surface and the particle size distribution is illustrated in Figure 2.9. The main collector pipe trench varies in depth but has a minimum depth of 1 m. The collector pipe and main collector pipe trench invert generally has a minimum fall of 1V:200H (0.5%).

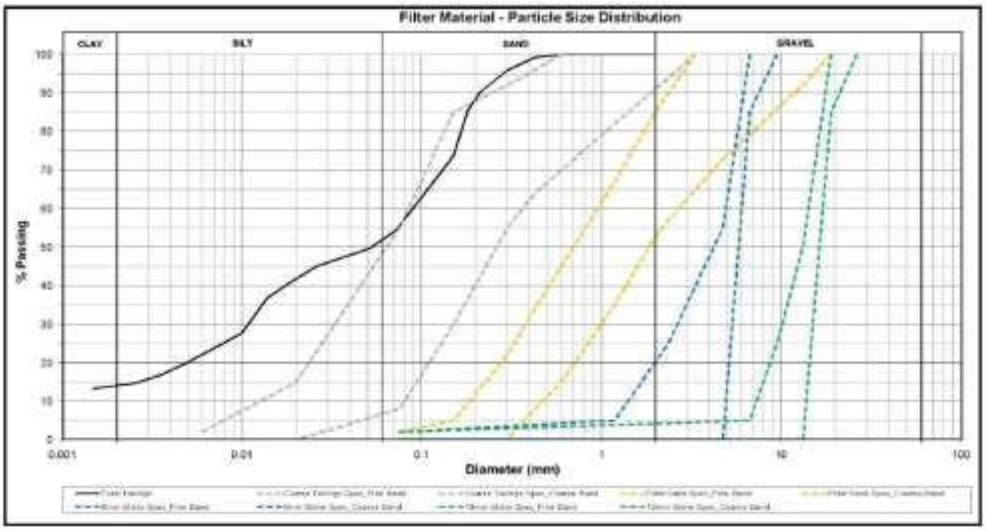


Figure 2.9. Particle Size Distribution of the Filter Material (Metago, 2011)

The drainage pipes from the toe drains and main collector pipe trenches are connected to the 2 m by 2 m (internal diameter) underdrainage tower which provides a sump from where seepage water will be pumped via a submersible pump to the
TSF beach. The tower is to be founded on competent hard rock.

2.6 Hydrogeology

Groundwater is the main source of water supply in the area. The catchment communities (Samanhu, Abekoase, Huniso and Tebe rely solely on groundwater for domestic use (Asklund *et al.,* 2005). To match the demand for potable water the number of boreholes and hand dug wells are increasing rapidly (Kortatsi, 2004). Yield varies from 0.4 to 18 m^3h^{-1} with an average of 2.4 m^3h^{-1}. The depth varies between 18 to 75 m with an average of 35 m but has little or no effect on borehole yields (Kortatsi, 2004).

Surface water taken from the River Bonsa at Bonsaso is treated and distributed to Tarkwa town. Some villages between Bonsaso and Tarkwa are also connected to the pipe (Asklund *et al.,* 2005).

In the Tarkwa-Prestea area groundwater occurrence is associated with the development of secondary porosity through fissuring and weathering. The rocks underlying the area lack primary porosity since they are consolidated. In the Tarkwaian System, and especially in the Banket Series quartzites, grits, conglomerates and Tarkwa Phyllite, the weathering depth rarely exceed 20m. Clay, silts, sandy clays and clayey sands are mostly the result of the weathering. In this area two types of aquifers occur. The weathered aquifer occurs mainly above the transition zone between fresh and weathered rock. Due to the clay and silt content of the soils, these aquifers have high porosity and storage but low permeability. The aquifer in the fractured/fissured zone occurs below the transition zone. They have relatively high transmissivity but low storage (Kortatsi, 2004). Boreholes drilled into the Huni Sandstone yielded low volumes of water (1.5 m^3/l), which is an indication of poor groundwater yield in this lithology (Anon, 1997).

A hydrogeological study conducted by Metago (2011) as part of the pre-feasibility study of the then proposed CIL project at Tarkwa Mine noted that there are little to no primary aquifer characteristics in the meta-sedimentary rocks underlying the Tarkwa area. Rather, aquifer potential is associated with fracture and weathering induced secondary permeability and porosity, and hydraulic connection between individual fracture sets and weathered zones. As such, the resulting aquifer potential will be dependent on the physical and mineralogical nature of the original rock fabric.

Aquifer potential is also critically dependent on active recharge, as the groundwater in static storage at any one time, in fractured rock aquifers, is rarely sufficient to sustain long term pumping. Recharge will occur as direct infiltration of rainfall (into weathered horizons and outcropping/subcropping fractures), the infiltration of runoff in stream courses and/or beneath swamps/ponds, and through flow from adjacent aquifers. However, the infiltration of surface water runoff (or ponding) is by far the most important recharge mechanism (Metago, 2011). Metago (2011) also states that the recharge of groundwater in the area occurs mainly by direct seepage or infiltration. In some places groundwater is in hydraulic contact with rivers and recharge from them can also take place (Kortatsi, 2004). Discharge from the aquifers will be by base flow to local and regional drainage courses (rivers and streams). Groundwater circulation in the Tarkwa-Prestea area is mainly localised due to the numerous low hills that act as groundwater divides. Groundwater circulation is mainly restricted to quartz veins, fissures and fault-brecciated zones. Groundwater velocities are not known (Kortatsi, 2004).

The TSF area is mostly underlain by Huni Sandstone and the Tarkwa Phyllite ridge which forms the south western boundary of the site. A large area of the TSF footprint is underlain by the Huni Sandstone. Available information on the Huni Sandstone appears to indicate that it does not form an aquifer and has the following shallow subsurface characteristics (Metago, 2011):

- A shallow horizon of sandy soils and extremely weathered rock;
- Hard, generally unfractured, partially weathered to fresh sandstone/quartzite below about 5m depth.

Groundwater movement will be locally controlled by fracture/joint directions, but overall flow will be towards local and regional drainage courses. Any seepage from the TSF will make its way vertically to the water table, although there may be some lateral flow components via sandier shallow saprolite horizons. After seepage has reached the water table, the plume will move down gradient at rates and velocities determined by the hydraulic gradient, permeability and specific yield (drainable porosity) of the aquitard /aquiclude (Metago, 2011).

The ultimate fate of any seepage would be as base flow to local and/or regional rivers, although by this stage the seepage would have mixed with local groundwater and subjected to hydrogeochemical attenuation processes as numerated as follows (Metago, 2011):

- Reactions between the TSF3 seepage and the groundwater: dilution, dispersion, precipitation and possibly volatilization (of cyanide as a result of mixing with low pH groundwater)
- Reactions between the soil/rock matrix, the TSF seepage: ion exchange and adsorption

According to Anon (2011), assuming that the permeability is low (which appears likely), the rate of movement away from beneath TSF will be slow. If seepage rates exceed groundwater outflow, a definite water table mound will develop. The natural water table is shallow (less than 10 m), and so the development of a water table mound could also result in potential water logging of the surface close to the facilities. The height of the water table mound could be controlled by the operation of seepage interception trenches.

The Mineral and Mining (Health, Safety and Technical) Regulations, L. I. 2182 require that a TSF must be lined with a 300 mm thick clay liner with permeability less than 1×10^{-8} m/s (Anon, 2012). Alternatively, it would be acceptable if it is demonstrated that the underlying geology provides an equivalent degree of containment. None of the three TSFs were clay lined and compacted as required by the Mining and Mineral Law (L. I. 2182, Section 265) and considering that the tailings solution contains cyanide, Anon (2012) recommended that a detailed hydrogeological investigation programme should be undertaken prior to commissioning which should, as a minimum, involve the following:

- Drill boreholes at six to eight sites and install two discrete interval piezometers (using slotted casing, gravel pack and annular seals) at, and down hydraulic gradient of each site. One piezometer should be installed over the saprolite (extremely weathered rock) profile to around 3 to 5 m depth. The deeper piezometer should be installed over the top section of competent rock to depths of between 15 and 20 m. It may be possible to install both piezometers in one borehole. If not, a second hole would be drilled at each site for the shallow piezometer
- Falling and/or rising head permeability tests should be conducted in all completed piezometers

- Conduct geochemical testing and analysis of tailings material should testing not have taken place previously
- Groundwater flow modelling undertaken to assess and predict flow rates, directions and velocities of seepage within the local groundwater system and seepage arrival times at groundwater discharge zones (i.e. Rivers)
- Water samples should be collected from all piezometers and analysed to provide initial baseline water quality data

2.7 Soils

There are mainly two types of soil in the project area, the forest oxysols in the south and the forest ochrosol-oxysol integrates in the north (Kortatsi, 2004). The forest oxysols are porous, well drained, and generally loamy brown to orange. Due to the heavy and plentiful rainfall in the south, a high degree of leaching and reduction of calcium, magnesium and other nutrients have occurred in the soil. This has made the soil acidic. The forest ochrosol-oxysol integrate into an intermediate between the forest oxysols and the forest orhrosol. The forest ochrosol-oxysol integrate is highly coloured as it is less leached, as a result of reduced rainfall in the north. It contains more of its nutrients and is therefore more alkaline than the forest oxysols in the south (Kortatsi, 2004).

The soil in the Tarkwa area consists mostly of silty-sands with minor patches of laterite, mainly on hilly areas (Kuma *et al.,* 2001). A geotechnical site investigation of the TSF site was conducted Metago (2011)as part of the design.

2.8 Mine water balance

A monthly climatic water balance was completed by GFGL Tarkwa Mine for the tailings complex, the purpose of which was to determine the following:

- The volume of excess water emanating from each TSF required to be detoxified at the Northern Heap leach Pads detoxification facility
- The volumes of water required as and when insufficient is returned from each of the TSFs into the process circuit
- The range of supernatant pool volumes that is likely to occur during the life of the facilities

2.8.1 Methodology

The water balance model as depicted in Figure 2.10 shows the water sources, contributing to the volume of the stored water on the left with the losses and uses of the stored water on the right. There are two main water sources:

- Water contained in the tailings slurry that is pumped to the facility
- Rainfall, falling onto the four different types of catchment that feeds the supernatant pond
- The supernatant (or TSF) pond, which has a run off coefficient of 1 (15% of the total tailing area)
- The dry tailings beaches, which have a run-off coefficient of 0.3 (55% of the total tailings area
- The wet tailings beaches, which have a run-off coefficient of 0.9 (30% of the total tailings area)
- The natural catchment external to the tailings facility (run-off coefficient: 0.26 which is based on the run off coefficient calculations from rational method)

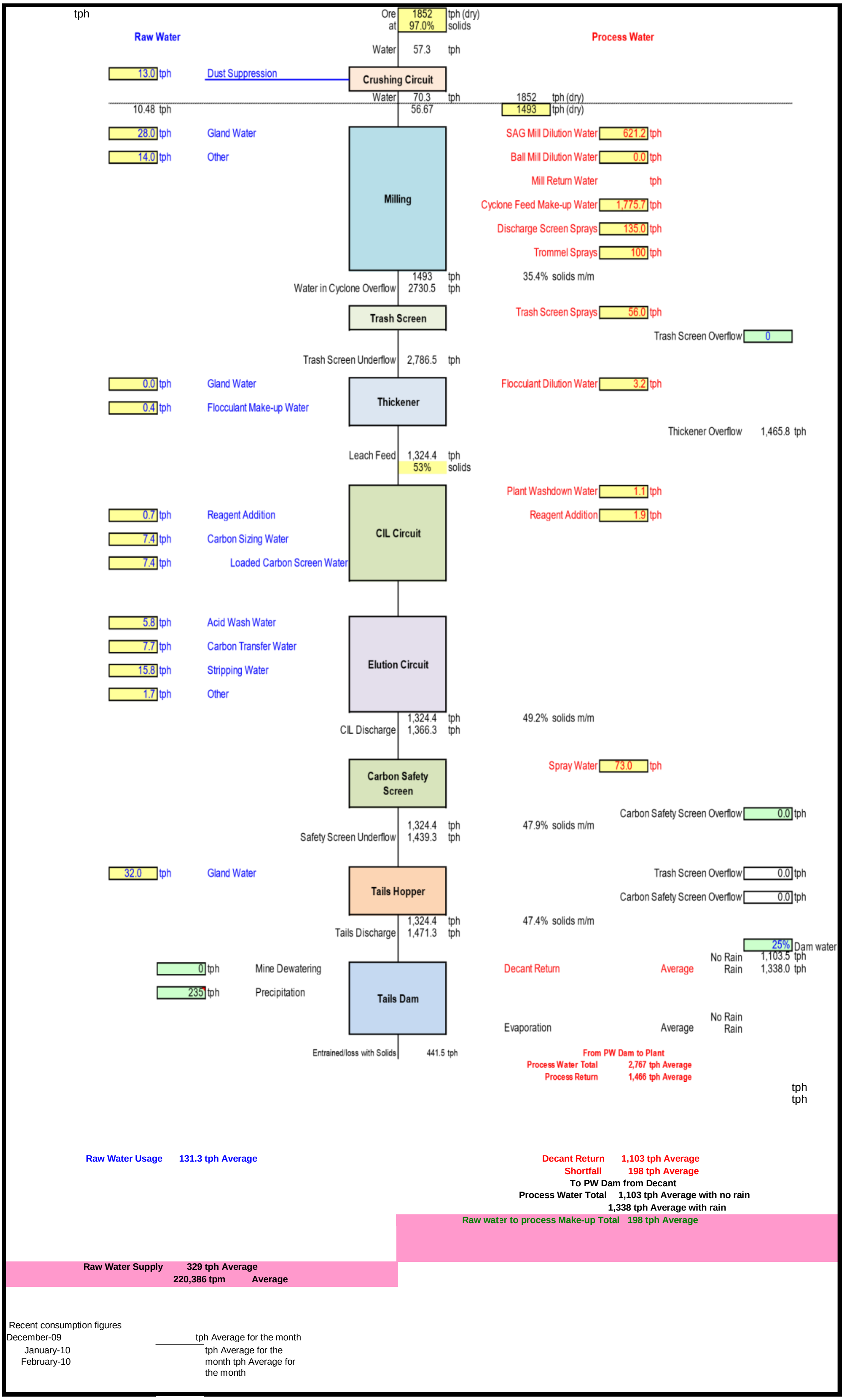

tph
Ore 1852 tph (dry)
at 97.0% solids
Raw Water
Process Water
Water 57.3 tph
13.0 tph
Dust Suppression
Crushing Circuit
Water 70.3 tph
10.48 tph
56.67
1852 tph (dry)
1493 tph (dry)
28.0 tph
Gland Water
14.0 tph
Other
Milling
SAG Mill Dilution Water 621.2 tph
Ball Mill Dilution Water 0.0 tph
Mill Return Water tph
Cyclone Feed Make-up Water 1,775.7 tph
Discharge Screen Sprays 135.0 tph
Trommel Sprays 100 tph
1493 tph
Water in Cyclone Overflow 2730.5 tph
35.4% solids m/m
Trash Screen
Trash Screen Sprays 56.0 tph
Trash Screen Overflow 0
Trash Screen Underflow 2,786.5 tph
0.0 tph
Gland Water
0.4 tph
Flocculant Make-up Water
Thickener
Flocculant Dilution Water 3.2 tph
Thickener Overflow 1,465.8 tph
Leach Feed 1,324.4 tph
53% solids
Plant Washdown Water 1.1 tph
Reagent Addition 1.9 tph
0.7 tph
Reagent Addition
7.4 tph
Carbon Sizing Water
7.4 tph
Loaded Carbon Screen Water
CIL Circuit
5.8 tph
Acid Wash Water
7.7 tph
Carbon Transfer Water
15.8 tph
Stripping Water
1.7 tph
Other
Elution Circuit
1,324.4 tph
CIL Discharge 1,366.3 tph
49.2% solids m/m
Spray Water 73.0 tph
Carbon Safety Screen
Carbon Safety Screen Overflow 0.0 tph
1,324.4 tph
Safety Screen Underflow 1,439.3 tph
47.9% solids m/m
32.0 tph
Gland Water
Tails Hopper
Trash Screen Overflow 0.0 tph
Carbon Safety Screen Overflow 0.0 tph
1,324.4 tph
Tails Discharge 1,471.3 tph
47.4% solids m/m
25% Dam water
No Rain 1,103.5 tph
Rain 1,338.0 tph
0 tph
Mine Dewatering
235 tph
Precipitation
Tails Dam
Decant Return
Average
Evaporation
Average
No Rain
Rain
Entrained/loss with Solids 441.5 tph
From PW Dam to Plant
Process Water Total 2,767 tph Average
Process Return 1,466 tph Average
tph
tph
Raw Water Usage 131.3 tph Average
Decant Return 1,103 tph Average
Shortfall 198 tph Average
To PW Dam from Decant
Process Water Total 1,103 tph Average with no rain
1,338 tph Average with rain
Raw water to process Make-up Total 198 tph Average
Raw Water Supply 329 tph Average
220,386 tpm Average
Recent consumption figures
December-09 tph Average for the month
January-10 tph Average for the
February-10 month tph Average for
the month

Figure 2.10. Water balance model (Anon, 2010)

The volume of water flowing into each TSF's pool was calculated on a monthly basis as follows:

- The slurry water volume was calculated using the production rates and the ratio of solids to water
- The rainfall volume was calculated from:
    - A "best fit" statistical distribution for each month of the monthly rainfall depths sourced from historical records
    - The external catchment area, the TSF beach area (both wet and dry), and an estimated area of the pool

The losses from each TSF were calculated (on monthly basis) and applied to the store volume in the following sequence, after the rainfall and slurry water volumes had been added to the starting volume:

- The evaporation volumes were calculated from the monthly averages of evaporation data collected by the mine and the area of the pool (15% of the total tailings area) as well as the wet beach area (30% of the total tailings area)
- The interstitial lock up volume was calculated from the void ratio of the deposited tailings. Any seepage was assumed to emanate from the interstitial water and was therefore not a direct loss in the model
- Once the two volumes above have been subtracted from the stored volume the model determines if there is enough water to supply the CIL process plant (a volume equal to the slurry water inflow for the given month). If there is sufficient water, the plant draft volume is subtracted from the stored volume, but if however it is not sufficient, the stored volume will be pumped to the plant and

the volume of the makeup water calculated. The makeup water volume is the difference between the required plant draft and the stored volume

In the event that there is still water stored in the facility (i.e. after the full plant draft has been subtracted from the stored volume), the model determines if the remaining volume is greater than the maximum permissible pool volume (750 000 m^3 for each facility). If it is greater, the difference between the stored volume and the maximum permissible volume is deemed to be excess water that will need to be detoxified at the heap leach north installation, leaving the final month end volume at 750 000 m^3. If the remaining volume is less than the maximum permissible volume, then the remaining volume is used as the month end volume. The volume 750 000 m^3 has been estimated from a safe operating level for TSF 1 decant pool, which yields an area of approximately 500 000 m^2 and an average of 1.5 m deep.

2.8.2 Results

The water balance results for each of the three facilities are presented below under the following two sections;

a) The expected following volume calculated for the median (average) monthly rainfall (statistically, the true expected value) for:
  - ➢ The decant pool volume
  - ➢ The monthly volume that will require detoxification
  - ➢ The monthly make up volume required for the process plants

b) Statistical results based on the “best fit” statistical distribution for each month of the monthly rainfall depths sourced from historical records on a monthly basis:
  - ➢ The mean volumes

- The minimum volume calculated by the model
- The maximum volume calculated by the model
- The 5th percentile (5% of the values calculated by the model are less than the 95th percentile)
- The 95th percentile (95% of the value calculated by the model are less than the 95th percentile)

The statistical results are based on specific years in the model as follows:

c) The pool volume and volumes requiring detoxification are taken from a random year (Jan – Dec 2016), by which time, under a constant production rate, the volumes would have reached a steady state seasonal cycle; and

d) The monthly make volumes are based on values from each facility's 3rd year of operation, which represents a period approximately half way in the time taken to reach the steady state seasonal cycle.

CHAPTER 3: LITERATURE REVIEW

This chapter provides a review of the literature relevant to the study. It covers and explains three main categories which are the concepts of cyanide attenuation in the tailings substrata, seasonal and depth effect of cyanide decay in the decant pond and migration of cyanide species and metals into the groundwater system. Additionally, the chapter describes the fate of cyanide and metal migration from the TSF's into the groundwater system.

3.1 Cyanide chemistry

According to Mudder (2001), there are several forms of cyanide potentially present in process solutions associated with the extraction of gold and silver. Related to cyanide are additional compounds formed through interactions with the ore, water treatment, and natural attenuation. Historically, the forms of cyanide most frequently discussed included free, WAD, total, and amenable to chlorination. Over the years, the WAD analytical procedure has been adopted by industry and the regulatory authorities for measurement of the "toxicologically significant" or "ecologically sensitive" forms of cyanide. The WAD analytical procedure measures free and other forms of cyanide. Subtraction of the WAD cyanide value from the total cyanide value provides a measure of the essentially non-toxic and stable iron cyanide level present. Applied and conducted properly, the total and WAD cyanide procedures produce reliable and meaningful results that can be used for compliance and monitoring purposes. The sensitivity of these methods is sufficient to quantify the forms and levels of cyanide that could be harmful to humans and the environment. The margin of error increases when these analytical methods are extended beyond their capabilities in an attempt to quantify cyanide at levels approaching and below detection limits at which environmental impacts become significant (Dzombak *et al.*, 2006). According to Dzombak *et al*. (2006) unwarranted concern and emphasis is placed upon the significance of unreliably low values of cyanide. In order to protect humans and the environment, reasonable levels of protection can be provided through promulgation of standards.

3.1.1 Cyanide species

According to Dzombak *et al* (2006), cyanide refers to a singularly charged anion consisting of one carbon atom and one nitrogen atom joined with a triple bond, CN^-. The most toxic form of cyanide is free cyanide, which includes the cyanide anion itself and hydrogen cyanide, HCN, either in a gaseous or aqueous state. At a pH of 9.3 - 9.5, CN^- and HCN are in equilibrium, with equal amounts of each present. At a pH of 11, over 99% of the cyanide remains in solution as CN^-, while at pH 7, over 99% of the cyanide will exist as HCN. Although HCN is highly soluble in water, its solubility decreases with increased temperature and under highly saline conditions (Anon, 2002). Both gaseous and liquid HCN are colourless and have the odour of bitter almonds, although not all individuals can detect the odour.

Cyanide is very reactive, forming simple salts with alkali earth cations and ionic complexes of varying strengths with numerous metal cations; the stability of these salts is dependent on the cation and on pH (Anon, 2002). The salts of sodium, potassium and calcium cyanide are quite toxic, as they are highly soluble in water, and thus readily dissolve to form free cyanide. Mining operations typically receive cyanide as solid or dissolved NaCN or $Ca(CN)^2$. Weak or moderately stable complexes such as those of cadmium, copper and zinc are classified as WAD (Anon, 2002). Although metal-cyanide complexes by themselves are much less toxic than free cyanide, their dissociation releases free cyanide as well as the metal cation which can also be toxic. Even in the neutral pH range of most surface water, WAD metal-cyanide complexes can dissociate sufficiently to be environmentally harmful if in high concentrations (Anon, 2002).

Cyanide forms complexes with gold, mercury, cobalt and iron that are very stable even under mildly acidic conditions. However, both ferro and ferricyanides decompose to release free cyanide when exposed to direct ultraviolet light in aqueous solutions. This decomposition process is reversed in the dark. According to Anon (2002), the stability of cyanide salts and complexes is pH dependent, and therefore, their potential environmental impacts and interactions (i.e. their acute or chronic effects, attenuation and re-release) can vary.

Metal-cyanide complexes also form salt - type compounds with alkali or heavy metal cations, such as potassium ferrocyanide ($K_4Fe(CN)_6$) or copper ferrocyanide ($Cu_2[Fe(CN)_6]$), the solubility of which varies with the metal cyanide and the cation. Nearly all alkali salts of iron cyanides are very soluble, upon dissolution these double salts dissociate and the liberated metal cyanide complex can produce free cyanide. Metal salts of iron cyanides form insoluble precipitates at certain pH levels. The cyanide ion also combines with sulphur to form thiocyanate, SCN^-. Thiocyanate dissociates under weak acidic conditions, but is typically not considered to be a WAD species because it has similar complex properties to cyanide (Anon, 2012). Thiocyanate is approximately 7 times less toxic than hydrogen cyanide but is very irritating to the lungs, as SCN^- chemically and biologically oxidizes into carbonate, sulphate and ammonia. The oxidation of cyanide, either by natural processes or from the treatment of effluents containing cyanide, can produce cyanate, OCN^{-}. Cyanate is less toxic than HCN, and readily hydrolyses to ammonia and carbon dioxide (Anon, 2002).

3.1.2 Cyanidation

The process of extracting gold from ore with cyanide is called cyanidation. The reaction, known as Elsner's Equation (Equation 3.1), is:

$$4Au+8CN^{-}+O_2+2H_2O \rightarrow 4Au(CN)_2^{-}+4OH^{-} \quad (3.1)$$

The formation of strong bond complexes such as those with iron and copper will tie up cyanide that would otherwise be available to dissolve gold. Copper cyanides are moderately stable but their formation can cause both operational and environmental concerns. Wastewater or tailings from such operations may have significantly higher cyanide concentrations than would otherwise be present in the absence of copper (Anon, 2008). High copper concentrations in the ore increase costs and lower recovery efficiencies by requiring higher cyanide application rates to compensate for reagents that complex with copper rather than gold. According to Anon (2008), cyanidation is also adversely affected by the presence of free sulphur or sulphide minerals in the ore. Cyanide will preferentially leach sulphide minerals and will react with sulphur to produce thiocyanate. These reactions will also enhance the oxidation of reduced sulphur species, increasing the requirement for lime addition to control the pH at a sufficient level to avoid the volatilisation of hydrogen cyanide (HCN).

3.2 Cyanide degradation mechanism

According to Anon (2008), 'excess' cyanide consumption is partly accounted for by oxidation to cyanate and loss through volatilisation as HCN gas (Figure 3.1).

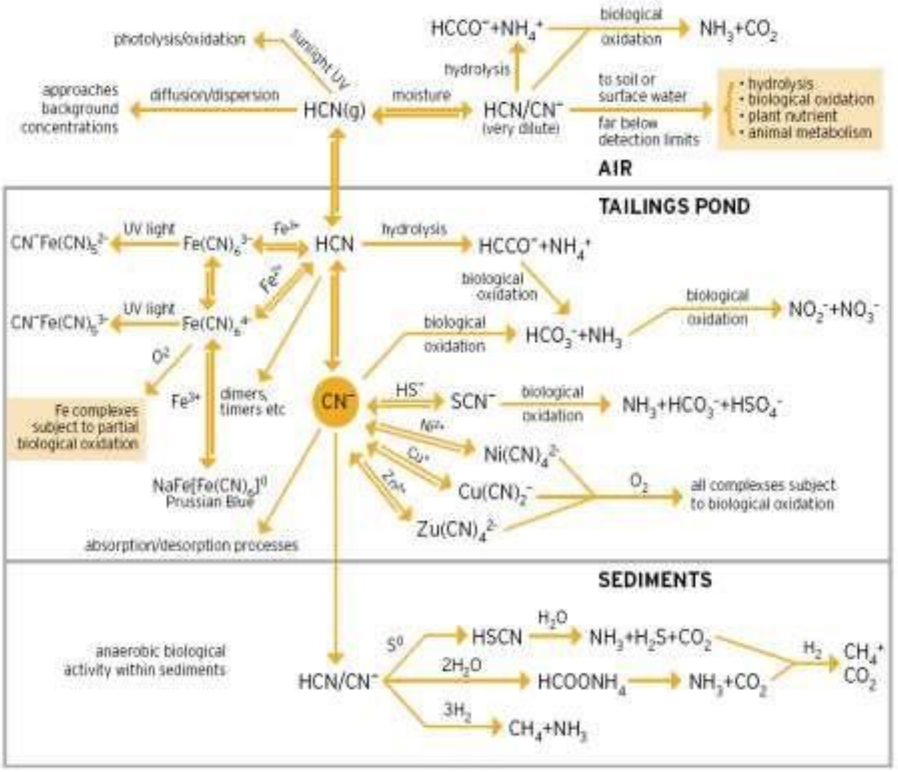


Figure 3.1. Cyanide chemical loss pathways in the environment ((Mudder et al., 1991)

Cyanide metal complexes in particular eventually find their way to tailings dams and then, potentially, into groundwater. Cyanide will be lost in the tailings dams and the wider environment through natural degradation reactions so that, in the long term, only the less toxic and strongly complexed forms remain. TSF's are designed to provide secure, long-term storage of materials containing such complexes and to avoid potential losses via seepage, overtopping, breaching, and pipe/channel failure (Anon, 2008). The more toxic forms of cyanide in TSFs are measured as WAD cyanide, free cyanide and complex forms.

3.2.1 Cyanide decay

Cyanide decay is a complex sum of many processes, each process dependent on many environmental factors such as pH, turbidity, aeration, lake mixing, mean depth and biological activity, as well as the forms of cyanide present (Mudder *et al.,*1999; Botz 1999). Experience suggests that at low concentrations an individual cyanide species will often decay exponentially, (the rate of decay will be proportional to the amount present). If a tailings dam is well mixed or has a consistent distribution of individual species or if groups of species decay independently and exponentially, then cyanide levels in the dam through time can be defined by a simple, first order differential equation (Mudder *et al.,* 1999).

According to Schmidt *et al.,* (1981), the volume of water contained in an impoundment increases with time as the tailings solids and their entrained liquid accumulate in the bottom of the impoundment. However, in almost all cases there is a net loss of cyanide in an impoundment as a result of natural cyanide degradation. In summary, natural cyanide degradation is a complex process which includes the following processes (Smith and Mudder, 1991);

- Volatilization
- Chemical oxidation
- Biological oxidation
- Hydrolysis
- Precipitation
- Complexation
- Sorption

These reactions may take place in the bulk solution or in the tailings sediment; however the predominant route of cyanide loss is typically ascribed to hydrogen cyanide volatilization from the impoundment water surface. Schmidt *et.al.,* (1981) found from tests carried out in Canada that volatilization of hydrogen cyanide accounted for 90% of cyanide removed from solution in a tailings impoundment while a combination of other reactions account for the remaining 10%. It has been found that the loss of hydrogen cyanide occurs according to a first order rate equation which means that the rate of loss of hydrogen cyanide from the impoundment is directly proportional to the hydrogen cyanide concentration in solution (Simovic, 1984). Mathematically this is represented by Equation 3.2:

$$\frac{d[HCN]}{dt} = -k[HCN] \qquad (3.2)$$

Where [HCN] represents the molar concentration of aqueous hydrogen cyanide in the impoundment solution, where *t* is time and *k* is the first order rate constant. For a tailings impoundment with a water body which is assumed to be completely mixed, the rate constant *k* is given by (Equation 3.3):

$$k = (A/v)kv \qquad (3.3)$$

Where *A* is the water surface area of the impoundment exposed to the atmosphere, *V* is the volume of solution in the completely mixed portion of the impoundment and k_v is the volatilization rate constant for hydrogen cyanide. Substituting this into Equation 3.3 yields:

$$d[HCN]/dt = -(A/V)kv[HCN] \qquad (3.4)$$

According to Simovic (1989), the significance of the parameters in Equation 3.4are:

- The term A/V is the inverse of the average depth of the impoundment, commonly referenced as $1/z$. As the surface area of the impoundment increases, the hydrogen cyanide volatilization rate increases, and similarly as the volume of the impoundment water body increases, the hydrogen cyanide volatilization rate decreases. The term A/V is a physical property of a tailings impoundment and not a function of weather conditions or solution chemistry
- The volatilization rate constant k_v is influenced by factors which accelerate or decelerate the rate of hydrogen cyanide volatilization, including temperature, wind speed, turbulence on the water surface and the degree of mixing in the impoundment water body. This rate constant is dependent upon local weather conditions but is not normally a function of the impoundment geometry or chemistry
- The product of k_v and A/V is the specific hydrogen cyanide volatilization rate constant and is a physical property of an impoundment. This constant changes with time as weather conditions vary or the impoundment geometry changes, but does not change with changes in solution chemistry (They are a function of time according to changes in weather conditions and impoundment geometry variations)

3.3 Tailings hydrogeology and geochemistry

3.3.1 Chemistry of Mill-Tailings impoundments

The primary objective of TSF design, according to Anon (1998), is to provide an appropriate level of containment for tailings, as well as security against escape of toxic forms through seepage, overtopping, breaching and pipe/channel failure.

According to Anon (2008), TSF's storing contaminated tailings should be at least equivalent to a constructed liner of 0.3 m thickness of clay, with a permeability of

10^{-8} m/s. With reference to Figure 3.2, the fate of cyanide compounds in TSF's is complex and a series of reactions takes place resulting in degradation, transformation and the occurrence of atmospheric emission (NICNAS, 2010).

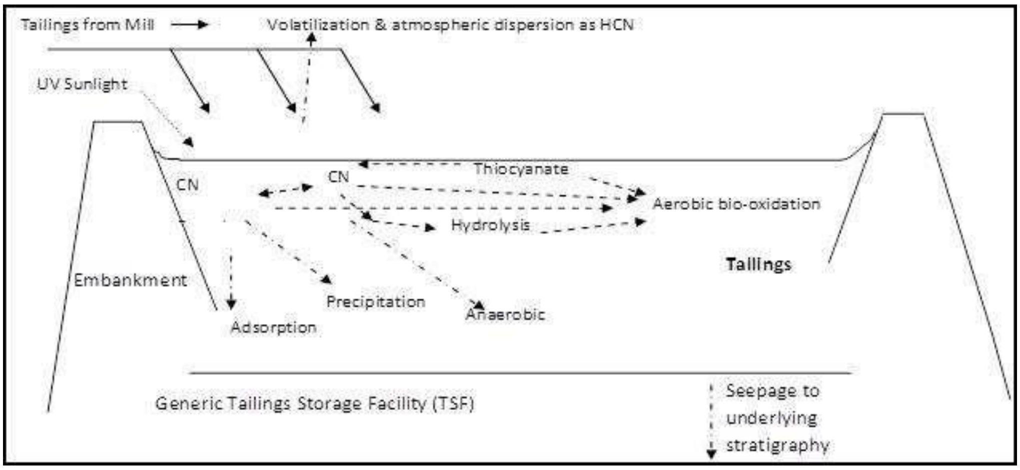


Figure 3.2. General fate of cyanide in a typical TSF (Source: NICNAS, 2010)

Generally, a fraction of cyanide will stay within a TSF for an extended period before undergoing degradation or stabilization (NICNAS, 2010), but volatilization of HCN also accounts for only a fraction of the cyanide added to a TSF (Smith *et al.,* 1993).

Ghosh *et al.,* (2006a), further explains that the fate and transport of aqueous cyanide generated by the mining industry in a TSF, involves a complex set of chemical reactions consisting of free cyanide radicals and various metal-cyanide complexes.

Part of free cyanide in the water column volatilize as gaseous HCN into the atmosphere where it can be absorbed into moisture droplets and re-enter the TSF via precipitation. The tailings sediment on the other hand, can be biologically degraded to ammonia, carbon dioxide, and nitrate (Ghosh *et al.,* 2006a).

According to Logsdon *et al.,* (1999), cyanide can also be fixed within the TSF by precipitation and adsorption of metal-cyanides and may potentially migrate in seepage to underlying strata and water resources. The natural potential for migration of contaminants, in seepage of process water to underlying formations at some TSFs, is dependent on factors such as discharge volume, concentration, climate and structural integrity (NICNAS, 2010). Iron cyanides react with other chemicals in tailings slurry in TSFs, to produce solid precipitates, which may contain a dozen insoluble cyanide salts, thereby removing cyanide from solution.

Murray *et al.,* (2000), demonstrated that any seepage of cyanide from a TSF into groundwater likely occurs in the form of stable iron complexes. The extent to which seepage occurs in a TSF, however, differs from site to site. WAD cyanide concentrations in operating TSFs generally vary, due to process and tailings management. Meanwhile an emerging international consensus, based on technical data, stipulates that WAD cyanide concentrations in open ponds should be maintained at concentrations of less than 50 mg/l (Logsdon *et al.,* 1999). High metalcyanide complex concentrations usually results in concentrations of WAD- cyanide exceeding 50 mg/l . In TSFs where cyanide has mainly been destroyed by natural degradation processes and/or planned measures, the concentrations range from 1-10 mg/l. However, those that do not have any calculated measure to reduce the cyanide concentration normally have WAD cyanide values exceeding 100 mg/l (NICNAS, 2010).

3.3.2 Tailings geochemistry

According to Johnson (1986), contaminant transport of mine and mill tailings is controlled by the hydrologic flow regime and the geochemical mobility of the contaminant. Contaminant mobility is primarily determined by chemical reactions occurring within an interface between the tailings and underlying substrata. The large geochemical differences between tailings and groundwater create disequilibrium in the groundwater with respect to the soils when seepage from tailings enters the groundwater. Precipitation and dissolution reactions involving the major chemistry occur to return the perturbed system to chemical equilibrium between groundwater and minerals. Retardation of trace elements depends upon the chemical character of the groundwater and the adsorption substrates.

Conceptually, analysis of contaminant transport from mine tailings is not different from analysis of contaminant transport away from any concentrated source of elements that is chemically different from the surrounding media. Contaminant transport analysis includes the hydrologic flow component and the geochemical mobility factor. The relative significance of the geochemical and hydrologic phenomena depends on the;

- Chemical and hydrologic conditions of the tailings and environment
- Chemical constituents of interest
- Distance along the flow path away from the tailings

The large differences between geochemistry of tailings and the groundwater/soil system create a chemical disequilibrium within the environmental system. This implies that reactions will occur to readjust the perturbed system toward chemical equilibrium, which determines the mobility (i.e., retardation) of the contaminants in the groundwater. The significant reactions are a combination of precipitation/dissolution and adsorption reactions that change the chemical character of seepage from tailings to a chemical composition similar to the natural groundwater. The question for contaminant transport analysis is the distance for the seepage to travel before the chemical character is indistinguishable from the background. The constituents of mill tailings which illustrate the concepts of geochemistry can be used to predict contaminant transport (Johnson, 1986). This is dependent on:

- the pH difference between the tailings and substrata as the major control of geochemical mobility; and
- the geochemical mobility is dominated by redox potential (Eh) differences.

Each unit is modelled independently and linked together by source terms. The concentration of an element existing in a unit is the source term for the next unit. Seepage from tailings into groundwater causes a change in the chemistry of the groundwater and the consequent disequilibrium between groundwater and minerals. This results in precipitation/dissolution reactions between the groundwater and minerals to re-establish background steady-state conditions. Modelling the mobility of contaminants within a given unit must distinguish between major and trace elements. For purposes of modelling major components are those that comprise the major mineral phases and play a role in controlling the pH and Eh (redox potential) of the system. Alternatively, trace components do not play a role in defining the chemical environment, but rather their behaviour is determined by the major chemistry of the system. The aqueous concentrations of major elements are primarily determined by precipitation/dissolution reactions. Whereas, adsorption is often the dominant mechanism for controlling the concentrations of trace elements. Thermodynamic calculations of precipitation/dissolution reactions address the major element chemistry directly. The major element chemistry influences the retardation of trace components indirectly by affecting the substrates for co-precipitation and adsorption and by determining the aqueous species of the trace components (Johnson, 1986).

Freshly precipitated hydrous oxides of iron, manganese, aluminium, and silica are commonly observed substrates for adsorption (Davis *et al.,* 1986). Different species are more likely to be adsorbed at different pH conditions. At a pH lower than the isoelectric point, hydroxyl groups on the substrate surface are positively charged and are anion adsorbers. At a pH greater than the isoelectric point, the surfaces become negatively charged and are predominantly cation adsorbers. Based upon the thermodynamic approach to modelling the potential for impacts on the groundwater (Davis *et al.,* 1986), the adsorption potential is reduced to:

- Characterization of the tailings mineralogy and the chemistry of the interstitial water
- Characterization of the soil mineralogy and the chemistry of the groundwater
- Characterization of the reactions that occur between the seepage from the tailings and the groundwater
- Development of the relationship between the reactions of the major elements and the mobility of the trace elements
- Expression of the geochemistry in a form suitable for use in contaminant transport models

3.3.3 Seepage in Tailings Dams

The fact that natural earth materials have a quantifiable and predictable ability to remove contaminant trace elements from migrating seepage has been documented for more than a decade (Griffin *et al.,*1976) and (Griffin *et al.,* 1977). An innovative suggestion by Griffin *et al.,* (1977) is that, the overall pollution from contaminants could be reduced if liners of natural earth material were designed for higher hydraulic conductivities. This would allow for attenuation of the potential groundwater contaminants in the seepage and would avoid the development of a "bathtub" condition. As described by Rouse and Pyrih (1985), a number of naturallyoccurring geochemical processes are effective in removing potential ground-water contaminants from seepage solutions.

3.4 Groundwater pollution

The chemical substances present in tailings disposed into TSF could be transported into groundwater as a result of seepage through the subsurface as already dissolved in the infiltrating water and as liquid, either readily miscible with water or hydrophobic (Rico *et al.,* 2007). As the contaminants are being transported, they are subject to mineral-water interactions which could either attenuate or enhance their mobility (Asklund *et al.,* 2005; Kresic, 2007; Taylor *et al.,* 2010). The four general forms of attenuation mechanisms of cyanide in the environment are natural degradation, chemical oxidation, precipitation and biodegradation (Logsdon *et al.,* 1999). According to Ghosh *et al.,* (2006a), various factors affect the cyanide cycle in the environment and it includes but not limited to pH, temperature, sunlight intensity and the Optimal Moisture Content (OMC) of soils.

The attenuation of cyanide mobility is due to geochemical and biogeochemical processes that naturally occur in the subsurface. Several geochemical mechanisms can account for the removal of cyanide from solutions. These mechanisms include:

- Sorption on mineral surfaces or organic detritus
- precipitation of insoluble metal-cyanide complexes typically with iron

- Chemical conversion to thiocyanate and chemical oxidation to cyanate Furthermore, cyanide can be altered by biological mechanisms and metabolized by plants:
- Under aerobic conditions in near-surface soils, cyanide will be breakdown into ammonia, nitrogen or even nitrate, and to carbon dioxide
- Under anaerobic conditions, cyanide will be decomposed to ammonium ion, nitrogen, thiocyanate, and carbon dioxide (Anon, 1988; Smith, 1988; Rouse *et al.*, 1988; Chatwin *et al.,* 1987; Simovic *et al.,* 1984; Schmidt *et al.,* 1981; Knowles, 1976)

Redox reactions and sorption reactions (such as precipitation, adsorption on the surfaces of soil mineral, absorption into the matrix of soil minerals) are the dominant mechanisms responsible for the reduction of mobility, toxicity or bioavailability of inorganic contaminants (Asklund *et al.,* 2005; Kresic, 2007; Taylor *et al.,* 2010). When conditions are not suitable desorption and dissolution of the contaminants would occur (Smedley *et al.*, 2001).

3.4.1 Importance of groundwater resource

Groundwater is the most reliable source of potable water supply and it supports a wide array of economic and environmental services. It is typically of more stable quality and better microbial quality than surface water and often requires little or no treatment to be suitable for domestic purposes, whereas surface waters generally need to be treated, often extensively (Kortatsi, 2004; Chilton *et al.*, 2006; Kresic, 2007; Taylor *et al.,* 2010). In many parts of the world groundwater sources are the single most important supply for the production of drinking-water, particularly in areas with limited or polluted surface water sources and sometimes for many communities, it may be the only economically viable option (Chilton and Seiler, 2006). For instance, in Ghana, surface water (rivers and streams) used to be the main source of drinking water for the inhabitants of the Wassa West District (Kortatsi, 2004) but upon increased realisation that the surface water is somewhat polluted as a result of intense mining activities, the Government of Ghana and some major mining companies operating in the district have provided boreholes as alternative source of drinking water and now groundwater has become the principal source of drinking water for the communities within the Wassa West District (Kortatsi, 2004). Again, in Ghana, since the urban water supply system could not meet the demand of the rural communities, the Community Water and Sanitation Agency was formed in 1999 by the state to supply water to the rural communities, which forms about 70 % of the total population. This agency relies extensively on groundwater resource.

As of 2006, the coverage for community water supply was 58. 97%. Also, in arid and semi-arid regions, where water scarcity is almost endemic, groundwater is playing a major role in meeting domestic and irrigation demands (Khodapanah *et al.,* 2009; Udayalaxmi *et al.,* 2010). On a global scale, it is estimated that more than 2 billion people depend on groundwater resource for their daily supply (Chevalking *et al.,* 2008) with 700 billion m^3 drawn out of the earth’s aquifers each year. This makes groundwater by weight, the primary mineral raw material extracted from the earth (Jean-Claude, 1995).

3.4.2 Fragility of groundwater resource

From the global dependency rate on groundwater mentioned above, it might be expected that aquifer protection to prevent groundwater quality deterioration would have received due attention (Chilton, 2006). However, even in and around urban and industrial areas, where many actual or potential sources of pollution are located, aquifer protection has, until relatively recently, not been given adequate consideration (Chevalking *et al.,* 2008; Taylor *et al.,* 2010). It could be gainsaid that the resource is often seen as a convenient and relatively cheap resource to be exploited rather than a valuable but potentially fragile resource that needs to be protected (Taylor *et al.,* 2010). This has placed the quality of the resource under increasing strain (Chevalking *et al.,* 2008). Meanwhile, once groundwater is polluted (that is, if contamination loads exceed aquifer assimilation capacity), its remediation is a very long, complex and expensive task and in many cases the groundwater resource is effectively destroyed as a potable water supply (Kresic, 2007; Taylor *et al.,* 2010).

In the Wassa West District of Ghana, where there is intensive mining activities, there have been studies conducted on groundwater quality and some of the findings are as shown in Table 3.1 with their corresponding WHO guideline limits for potable water (Kortatsi, 2004; Amonoo Neizer *et al.,* 1996; Golow *et al.,* 1996; Obiri, 2007; Rossiter *et al.,* 2010).

Table 3.1. Levels of inorganic constituents found in groundwater in Wassa West District of Ghana (Kortatsi, 2004)

Chemical substances	Levels found in Tarkwa (mg/l)	WHO Guidelines, (mg/l)
Calcium	1.27 - 111.00	100.00
Magnesium	0.58 - 22.70	30.00
Chloride	3.16 - 117.00	250.00
Sodium	2.19 - 69.70	200.00
Iron	0.001- 8.30	0.300
Sulphate	0.22 - 68.50	250.00
Potassium	0.27 - 22.70	250.00
Phosphate	<0.001- 0.35	N/A
Zinc	0.006 - 0.65	5.00
Arsenic	<0.001 - 0.07	0.01
Cadmium	<0.001 - 0.003	0.003
Chromium	<0.001 - 0.07	0.05

Copper	<0.001 - 0.21	2.00
Lead	<0.001 - 0.03	0.01
Manganese	0.01 - 2.00	0.40
Mercury	<0.001 - 0.04	0.006
Nickel	<0.001- 0.08	0.07
Nitrate	<0.001 - 378.00	50.00
Cyanide (Free)	<0.001 - 0.09	0.07

3.4.3 Sources of groundwater quality deterioration

Groundwater quality in natural systems is a result of many environmental factors. Climate, geology, biochemistry, composition of atmospheric precipitation and the nature of the hydrology are among the more important factors (Hem, 1992). Also the source of most dissolved ions in natural waters is the mineral assemblages in the rocks near the land surface (Hem, 1992). Rock composition is only one of many related geologic factors; other geologic factors, such as nature of minerals, texture, porosity, and regional structure, can affect the composition of waters (Hem, 1992).

Groundwater quality changes start in the soil, where infiltrating water dissolves carbon dioxide from biological activity in the soil to produce weak carbonic acid that may assist removal of soluble minerals from the underlying rocks (Kresic, 2007). At the same time, soil organisms consume some of the oxygen that is dissolved in the infiltrating water.

Due to rock material being predominantly inorganic in nature, contact of flowing groundwater with the rock may dissolve inorganic chemical substances into that water. This occurs because groundwater movement is typically slow and residence times long (Kresic, 2007).

Although natural contaminants may have significant impacts on groundwater quality depending on geology, numerous human made sources and contaminants, including mine waste (tailings and waste rock), land fill leachates, domestic sewage, agricultural runoff from agricultural fields and industrial wastes, have disproportionately greater negative effects on the quality of groundwater resources (Prakash *et al.,* 2006; Kresic, 2007; Taylor *et al.,* 2010). To the greatest extent, mining is identified as one of the major human made sources of groundwater quality deterioration worldwide considering the enormous amount of wastes generated among which tailings often appear to be the cause of most severe problems, particularly from an environmental point of view (Kresic, 2007; Asante *et al.,* 2009).

3.4.4 Transport of anthropogenic cyanide and metals in soil and groundwater

According to Sparks (2003), there are a number of inorganic contaminants that are important in water and soil. These include plant nutrients such as nitrates and phosphate; trace metals such as cadmium, chromium and lead; oxyanions such as arsenite, arsenate and selenite; organic chemicals; inorganic acids; and radionuclide. The sources of these contaminants include fertilizers, pesticides acidic depositions, agricultural and industrial waste and radioactive fallout. Discussions on these contamination and their sources are provided in Table 3.2.

Table 3.2. Content of various elements in soils (Lindsay, 1979)

Metal	Selected Average for Soils (Mg/kg)	Common Range fo (Mg/kg)	
		Minimum	Maximum
Al	71,000	10,000	300,000
Fe	38,000	7,000	550,000
Mn	600	20	3,000
Cu	30	2	100
Cr	100	1	1000
Cd	0.06	0.01	0.70
Zn	50	10	300
As	5	1.0	50
Se	0.3	0.1	2
Ni	4.0	5	500
Ag	0.05	0.01	5
Pb	10	2	200

Hg	0.03	0.01	0.3

Nitrates (NO_3^-) are weakly held by soils which readily leach. Contamination of groundwater with nitrate is a major problem in areas that have sandy soils. Major sources of nitrogen (N) and phosphorous (P) in the environment are inorganic fertilizers, animal manure, biosolids applications, septic systems and municipal sewage system (Sparks, 2003).

3.4.5 Trace elements

Trace elements are present at the level below <0.1% in natural materials such as the lithosphere; if their concentration are high, they can be toxic to living organisms (Sparks, 2003). Trace elements include trace metals, heavy metals, metalloids (an element having both metallic and non-metallic properties) such as As and B, micronutrients (chemical elements needed in small quantities for plant growth, that is <50 mg/g in pH) and trace inorganic. Heavy metals are those metals having densities greater than 5.0 g/cm^3. Examples are Cd, Cr, Cu, Pb, Hg and Ni.

The sources of trace elements are soil parent materials (rocks), commercial fertilizers, liming materials, biosolids irrigation waters, coal combustion residues, metal-smelting industries, auto emission and others. Table 3.3 shows the concentration of trace elements in soil forming rocks and other natural materials. Potable water contaminated with As has been linked to cancer, diabetes and cardiovascular problems in human beings(Sparks, 2003). The source of the As in drinking water, particularly inorganic As is often from weathering of minerals in rocks and soil.

According to Sparks (2003), As occurs in two major oxidation states, As (III) and As

(V). As (III) is primarily present in anoxic environments whiles As (V) is found in oxic soils (Sparks, 2003). Both As species primarily occur as oxyanions in the natural environment and forms a strong complex with oxides such as Al and Fe oxides as inner- sphere products. These oxides, and particularly Mn oxides, can affect the oxidation of As (III) to As (V) which reduces the toxicity of As. Arsenic can also occur as sulphate such as arsenopyrite (FeAsS) and enargire (Cu_3AsS_4) at mining sites. According to Maclean *et al.*, (1992), all soils naturally contain trace levels of metals and the average concentration of selected metals in soils are listed in Table 3.3. The presence of metals is soil is, therefore not indicative of contamination. The concentrations of metals in uncontaminated soil are primary related to the geology of the parent material from which they were formed.

3.4.6 Chemical parameters present in groundwater

i. Physical Parameters

a) Total Dissolved Solids (TDS)

TDS comprise inorganic salts (principally calcium, magnesium, potassium, sodium, bicarbonates, chlorides and sulphates) and small amounts of organic matter that are dissolved in water. Concentrations of TDS in water vary considerably in different geological regions owing to differences in the solubility of minerals. In fresh groundwater, the level of TDS is less than 1000 mg/l. The palatability of water with a TDS level of less than 600 mg/l is generally considered to be good; drinking-water becomes significantly and increasingly unpalatable at TDS levels greater than about 1000 mg/l (WHO, 2008). No health-based guideline value for TDS has been proposed (WHO, 2008).

Table 3.3. Occurrence and significance of trace elements in natural water (Sparks, 2003)

Elements health	Sources % of samples, service	Effects and U.S public highest	Occurrence limit(mg/l)	Significant concentration (µg/l)
As	Mining by-product, pesticides, chemicals	Toxic, possibly carcinogenic waste	0.05	5.5% (above 5µg/l) 336
Be	Coal, nuclear powers space industries	Acute and chronic toxicity, carcinogenic	Not given and	Not given
B	Coal, detergent formulations, industrial	Toxic to some plants.	1.0	98% (above 1µg/l) 500
Ca	Industrial discharge, mining waste, metal plating, water pipes	Replaces zinc biochemically, causes high blood pressure and kidney damage, destroy testicular tissues and red blood cells, toxic to aquatic	0.01	2.5%, not given
Cr	Metal plating, cooling-tower water additives (chromide) normally found as Cr (VI) in polluted water.	Essential trace element, (glucose, tolerance factor), possibly carcinogenic at Cr^{+5}	0.05	24.5%, 112,
Cu	Metal plating, industrial and domestic wastes, mining, mineral leaching	Essential trace element, not very toxic to animals. Toxic to plants and algae at moderate level.	1	74.4%, 112,
F	Natural geographical sources, industrial wastes, water additive	Prevent tooth decays at above 1mg/l, causes mottled teeth and bone damage at around 5mg/l in water	08-1.7 depending on temperature	Not given

Continuation of Table 3.3

Elements	**Sources**	**Effects and Significant**	**U.S public health service limit(mg/l)**	**Occurrence % of samples, highest concentration (µg/l)**
I	Industrial waste, natural brines, sea water intrusion	Prevents goitre	Not given	Rare in fresh water
Fe	Corroded metal, industrial wastes, acidic mine drainage, low pH water in contact with iron minerals	Essential nutrites (components of haemoglobin), not very toxic, damages materials	0.05	75.6%, 4600, 52
Pb	Industrial mining, plumbing, coal gasoline	Toxicity (anemia, kidney disease, nervous system), wildlife destruction	0.05	19.3% (above 2ug/l), 140, 23
Mn	Mining industrial wastes, acid mine drainage, microbial action on manganese minerals at low pH	Relatively mono toxic to animals, toxic to plants at high levels, stains material (bathroom fixtures and clothing)	0.05	51.4% (above 0.3µg/l), 3230, 53
Hg	Industrial waste, mining pesticides, coal	Acute and chronic toxic	Not given	Not given
Mo	Industrial waste, natural	Possibly toxic to animals,	Not given	32.7 (above 2ug/l), 5400, 120
Ag	Natural geological source	Cause blue-gray discoloration	0.05	76.5 (above 2ug/l), 1180, 64

Zn	Industrial water, metal plating, plumbing	Essential elements in metalloenzyme, acid wound healing, toxic to some plants at higher levels, major components of sewage sludge,	0.05	Not given

b) *Dissolved oxygen* **(DO)**

The dissolved oxygen content of water is influenced by temperature, chemical or biological processes taking place in the groundwater. Depletion of dissolved oxygen in water can encourage the microbial reduction of nitrate to nitrite and sulphate to sulphide (Kresic, 2007). It can also cause an increase in the concentration of ferrous iron and manganese in solution, with subsequent discoloration, when the water is aerated. No health-based guideline value is recommended by WHO.

c) *Hardness*

Hardness caused by calcium and magnesium is usually indicated by precipitation of soap scum and the need for excess use of soap to achieve cleaning (WHO 2008). Depending on the interaction of other factors, such as pH and alkalinity, water with hardness above approximately 200 mg/l may cause scale deposition in the treatment works, distribution system and pipe work and tanks. On heating, hard waters form deposits of calcium carbonate scale. Also, hard waters being high in ionic concentration would contain more toxic substances but also enough benign minerals to block and overcome their toxic effects. Soft water, with a hardness of less than 100 mg/l, may, on the other hand, have a low buffering capacity and so be more corrosive for water pipes. No health-based guideline value is proposed for hardness in drinking-water (WHO, 2008).

ii. Primary inorganic constituents

Primary inorganic constituents are the inorganic constituents that always make up more than 90% of total dissolved solids in a groundwater sample. These include: pH, bicarbonate, carbonate, sulphate, chloride, nitrate (and other nitrogen forms), potassium, sodium, calcium, magnesium, iron and silica (Kresic, 2007).

a) pH pH is the measure of acidity or basicity of a solution and it has a profound effect on the mobility and solubility of many substances in water. The pH scale runs from 0 to 14 (purely acidic to purely basic) with 7 as neutral condition. Most metallic elements are soluble (cations) in acid groundwater but could precipitate as hydroxides or basic salts with an increase in pH (Kresic, 2007). Dissolved chemical compounds and the biochemical processes in the water usually control the pH. In most unpolluted water, pH is primarily controlled by the balance between free CO_2, CO_3^{2-} and HCO_3^- ions as well as natural compounds such as humic and fulvic acids (Kresic, 2007). The pH in normal groundwater ranges from 5–8 (Buamah, *et al.*, 2008). Although pH usually has no direct impact on consumers, it is one of the most important operational water quality parameters. The optimum pH required is in the range 6.5–8.5 (WHO, 2008).

b) *Bicarbonate* (HCO_3^-) *and carbonate* (CO_3^{2-})

Bicarbonate and carbonate ions in groundwater are mostly derived from the atmospheric and soil CO_2 and dissolution of carbonate rocks such as calcium carbonate. Groundwater generally contains more than 10 mg/l but less than 800 mg/l bicarbonate (Kresic, 2007). High concentrations could result in bitter taste of water and reduce corrosiveness of piping system (WHO, 2008).

c) ***Sulphate*** **(SO_4^{2-})**

Sulphate is an oxidized form of sulphur, produced primarily when sulphide minerals undergo weathering in contact with aerated water. Sulphur is widely distributed in reduced form in both magmatic and sedimentary rocks as metallic sulphides. Pyrite constitutes a major source of sulphate in groundwater. All atmospheric precipitation contains sulphate, which is one of the major dissolved constituents. The resulting reduction of sulphate ions produces hydrogen sulphides as a by-product, most of which will remain in the subsurface as gases. If iron is present in the water under moderately reducing conditions, iron sulphide may be precipitated, thus removing both sulphide and iron from the water.

The sulphate content of normal groundwater in magmatic and sedimentary rocks is usually less than 30 mg/l but in gypsum and anhydrite deposits, sulphate concentration may reach the saturation point of gypsum, corresponding to about 1360 mg/l. Very low or even zero sulphate concentrations are typical for groundwater in which bacterial reduction has been taking place (Kresic, 2007). No health-based guideline value for sulphate has been proposed but the presence of sulphate in drinking-water may also cause noticeable taste at concentrations above 250 mg/l and may contribute to the corrosion of distribution systems. WHO has set the guideline value of sulphate at 250 mg/l for the reason of taste and odour (WHO, 2008).

d) ***Chloride*** **(Cl^-)**

Chloride is present in all rock types in concentrations lower than any of the primary constituents of groundwater. Values of chloride in normal groundwater are less than 30 mg/l, with higher values commonly indicating the admixture of mineralized water or anthropogenic pollution. Chloride ions do not significantly enter into oxidation or reduction reactions, form no important solute complexes with other ions unless the chloride concentrations are extremely high, do not form salts of low solubility, are not significantly adsorbed on mineral surfaces and play few vital biochemical roles (Kresic, 2007). No health-based guideline value is proposed for chloride in drinkingwater. However, chloride concentrations in excess of about 250 mg/l can give rise to detectable taste in water (WHO, 2008).

e) *Cyanide* (CN^-)

Cyanide is a form of nitrogen and may occur in groundwater affected by waste disposal (Dzombark *et al.,* 2006). It is extensively used in the extraction of low concentration of gold and silver from ore (Arthur *et al.*, 2004). They occur in mill tailings solution as free cyanide (HCN and CN^-), simple cyanide compounds and complex alkali-metal cyanides. Each of these complex ions has a different relative stability, ranging from weak, through moderately strong, to strong. The mobility of cyanide in groundwater is dependent on soil and water pH and on the types of minerals present in aquifer material. Acidic soils with high clay, iron and aluminium oxide, and organic matter contents will contribute to significant retardation of dissolved cyanide in groundwater. Cyanide degrades naturally by bacteria mediation to carbon dioxide and ammonia.

The toxicological significance of each individual metal–cyanide complex is determined by its ability to release free cyanide (CN^- or HCN) and the target species of concern under pertinent exposure conditions. The brain and the heart are important sites of action of cyanide in humans. Cyanide interferes with energy production and causes oxidative stress by acting on mitochondria in brain cells. Probably related to neurotransmitter release, cyanide can produce irregular heartbeats, slow heart rate, and cause heart failure. Proteins in cell mitochondria may mediate some toxic actions of cyanide (Dzombak *et al.,* 2006). It is in view of this that cyanide level in drinking water has been set at 0.07 mg/l respectively and it is considered to be protective for both acute and long-term exposure (WHO, 2008).

f) ***Potassium* (K) *and Sodium* (Na)**

Potassium is slightly less common in igneous rock but more abundant in all sedimentary rocks. In most fresh water aquifer, if the sodium concentration substantially exceeds 10 mg/l, the potassium concentration is commonly half or a tenth that of sodium (Hem, 1989; Kresic, 2007). In some dilute natural water in which the sum of sodium and potassium is less than 10 mg/l, the potassium concentration may be equal or even exceeding that of sodium (Kresic, 2007). Sodium tends to remain in solution rather persistently once liberated from silicate– mineral structures. There are no important precipitation reactions that can maintain low sodium concentrations in water. Sodium may be retained on mineral surfaces, especially by clays, which have high cation-exchange capacity. However, the interactions between surface sites and sodium are weak. Potassium is also soluble but it seldom occurs in concentrations equal to sodium in natural water and could be attributed to biological factors playing important role in controlling the availability of potassium for solution in groundwater, the high stability of potassium-bearing alumino-silicate minerals and potassium from solution being incorporated strongly into some clay-mineral structures. There is no health based guideline value proposed for sodium and potassium, however, concentrations in excess of 200 mg/l and 250 mg/l respectively may give rise to unacceptable taste (WHO, 2008).

g) ***Calcium* (Ca)**

Calcium is the most abundant of the alkali-earth metals and is a major constituent of many common rock minerals and solutes in natural waters. Its presence in aqueous system is due mainly to more soluble solids containing calcium and is governed to a great extent by the equilibria between the solute and gaseous phases of carbon dioxide species. The concentration of Ca may be elevated due to the presence of soilregolith, where the content of CO_2 in the ground is consistently 10–100 times that of the atmosphere (Kresic, 2007). CO_2 forms carbonic acid in reaction with water and the acid accelerates dissolution of calcium-bearing mineral. Concentrations of calcium in normal potable groundwater generally range between 10 and 100 mg/l (Kresic, 2007). The most commonly noticed effect of calcium in groundwater is hardness and it is due to this effect that the WHO has set the guideline value to 100 mg/l (WHO, 2008).

h) ***Magnesium*** (Mg)

Magnesium is a common alkaline-earth element essential in plant and animal nutrition. Although, magnesium and calcium behave similarly in water solutions to some extent (such as creating hardness in water), the geochemical characteristics of magnesium arequite different because its ions are smaller than calcium ions (Hem, 1989; Kresic, 2007). Sedimentary rocks containing magnesium carbonate (magnesite), $MgCO_3$, and double magnesium carbonate dolomite ($CaCO_3$ x $MgCO_3$) are soluble under the influence of CO_2 gas dissolved in the groundwater. The magnesium ion, Mg^{2+} will normally be the predominant form of magnesium in solution, while the complex $MgOH^+$ will not be significant below about pH 10 (Hem, 1989; Kresic, 2007). Magnesium occurs in significant amount in dolomites and its dissolution brings magnesium into groundwater but the chemical reaction is not readily reversible. In spite of its high solubility, its content in groundwater is generally below calcium, most probably because of the lower general abundance of magnesium, however occasional exceptions occur in magnesium-rich aquifers such as dolomite rocks (Kresic, 2007).

Due to magnesium contribution to hardness, WHO has set its guideline value to 30 mg/l (WHO, 2008).

i) *Iron* (Fe)

Iron is one of the most abundant elements in magmatic rocks particularly as part of dark coloured minerals such as pyroxenes, amphiboles, biotite, magnetite and especially the olivine. Iron in these minerals is in the ferrous form (Fe^{2+}). When these minerals are exposed to water, the iron that is dissolved generally precipitates in the vicinity as sedimentary species. In reducing conditions, when sulphur is available, the precipitate is in the form of ferrous polysulphides or siderite ($FeCO_3$) when sulphur is abundant. In oxidizing conditions, the precipitate will contain ferric oxides or oxyhydroxides such as hematite (Fe_2O_3) or goethite (FeOOH). Ferric hydroxide ($FeOH_3$) is freshly precipitated material with poorly developed crystal structure. The availability of iron for aqueous solution is strongly affected by pH and redox conditions, where catalytic action of microorganism plays the key role in oxidation to ferric (Fe^{3+}) iron under aerobic conditions or the reduction to divalent, ferrous iron (Fe^{2+}) under anaerobic conditions. The most common form of iron in solution is the ferrous ion. Ferrous ion concentrations in reducing conditions are commonly between 1 mg/l and 10 mg/l. Groundwater with pH between 6 and 8 can be sufficiently reducing to retain as much as 50 mg/l of ferrous iron at equilibrium, when bicarbonate activity does not exceed 61 mg/l. However, concentration of iron in normal groundwater rarely exceeds 1 mg/l (Kresic, 2007). Iron stains laundry and plumbing fixtures at levels above 0.3 mg/l. There is usually no noticeable taste at iron concentrations below 0.3 mg/l, and concentrations of 1 mg/l –3 mg/l can be acceptable for people drinking anaerobic well water (WHO, 2008).

iii. *Secondary Inorganic Constituents*

Secondary inorganic constituents have received particular concern within the wide diversity of contaminants affecting water resources due to their strong toxicity at low concentrations (Momodu *et al.*, 2010). Most secondary groundwater constituents occur in potable, non-polluted groundwater at very low concentrations (in trace amounts) but their natural concentrations have wide ranges depending on the local geochemical conditions in the aquifer. Some of these constituents include: manganese, arsenic, chromium, lead and mercury.

***a) Manganese* (Mn)**

Manganese is very common in soils and sediment. It is commonly found with iron as mineral oxide coatings on the surface of soil and rock grains. When groundwater contacts these coatings, the oxides are dissolved and may be transported to the groundwater. The trace metal manganese (Mn) is ubiquitous in the environment; it is a minor component of most rock types and is also present in soils from weathering and aerial deposition. Mn can exist in different oxidation states, but the most widely occurring forms in the environment are soluble Mn^{2+} when reduced and insoluble Mn^{4+} when oxidized (Borah *et al.*, 2009). Concentration ranges of manganese in groundwater span several orders of magnitude, from <0.001 mg/l to values occasionally in excess of 1 mg/l. The principal controls on manganese concentration in groundwater are pH and redox condition. Manganese is mobilised under acidic conditions. As dissolved oxygen concentrations in groundwater tend to decrease with well depth, anaerobic conditions and hence high manganese concentrations tend to occur more commonly in deep wells. Under strongly reducing conditions in the presence of dissolved sulphide, manganese can be immobilised by the formation of insoluble manganese sulphide (MnS), although this is usually only important at high pH (>8) (Hem, 1992).

Mn concentrations above 0.4 mg/l can result in metallic tasting water, staining of clothes, dishes, and products such as paper or plastics, and reduced water pressure and flow in pipes from accumulation of Mn oxides. Exposure to concentrations of manganese over the course of years has been associated with toxicity to the nervous system, producing a syndrome that resembles Parkinsonism. This type of effect may be more likely to occur in the elderly (Santos-Burgoa *et al.,* 2001; Takeda, 2003; Borah *et al.,* 2009). WHO has set a guideline value of 0.4 mg/l due to both health and aesthetic effects (WHO, 2008).

b) **Arsenic (As)**

As is the twentieth most common element in the earth’s crust, and is associated with igneous and sedimentary rocks, particularly sulphide ores (As_2S_3, As4S4 and FeAsS). Arsenic is particularly mobile at pH values typically found in groundwater (pH 6.5–8.5), that is under both oxidizing and reducing conditions (Smedley *et al.,* 2001). Dissolved arsenic in water exists primarily as oxy-anions with formal oxidation states as As^{3+} (arsenite) and As^{5+} (arsenate). Both anions are capable of adsorbing to various subsurface materials such as ferric oxides in acidic to neutral waters and clay particles. An increase in the pH to an alkaline condition may cause both arsenite and arsenate to desorb. In oxic water, dissolution of sulphide minerals, most notably pyrites and arsenopyrite, release arsenic to groundwater. The level of arsenic in natural waters generally varies between 1 and 2µg/l with concentrations being elevated in areas containing natural sources (Kresic, 2007).

Inorganic arsenic compounds are classified as carcinogenic to humans. The practical quantification limit for arsenic is in the region of 1–10 μg/l and removal of arsenic to concentrations below 10 μg/l is difficult in many circumstances. In view of the significant uncertainties surrounding the risk assessment for arsenic carcinogenicity and the practical difficulties in removing arsenic from drinking water, a guideline value of10 μg/l is retained as a goal. In view of the scientific uncertainties, the guideline value is designated as provisional (WHO, 2008).

c) Chromium **(Cr)**

Chromium is one of the less common elements in the transition group found in rocks, soils and living organism, it occurs in combination with other elements such as chromium salts, some of which are soluble in water. The pure metallic form does not exist naturally in elemental form, but only in compounds (Evanko *et al.,* 1997; Kresic, 2007). Cr^{6+} is the form of chromium commonly found at contaminated sites. Chromium can also occur in the +3 oxidation state, depending on pH and redox conditions. Cr^{6+} can be reduced to Cr^{3+} by soil organic matter, S^{2-} and Fe^{2+} ions under anaerobic conditions often encountered in deeper groundwater (Evanko *et al.,* 1997; Kresic, 2007). The leachability of Cr^{6+} increases as pH increases (Palmer *et al.,* 1994). Cr^{3+} is the dominant form of chromium at low pH of about 3.5. Cr^{3+} mobility is also decreased by adsorption to clays and oxide minerals below pH 5 (Evanko *et al.,* 1997; Palmer *et al.,* 1994).

The toxic action of chromium is confined to the hexavalent compound (Cr^{6+}), which is relatively mobile in the environment and is acutely toxic, mutagenic, teratogenic and carcinogenic and may cause death to humans and animals if ingested in large doses. In contrast, Cr^{3+} has relatively low toxicity and is immobile under slightly acidic to moderately alkaline conditions (Palmer *et al.,* 1994; Evanko *et al.,* 1997; Kresic, 2007). Due to the toxicity of Cr^{6+}, the WHO has set Cr in drinking water to 0.05 mg/l (WHO, 2008).

***d) Lead* (Pb)**

Lead is used principally in the production of lead-acid batteries, solder and alloys. The organolead compounds, tetraethyl and tetramethyl lead have also been used extensively as antiknock and lubricating agents in petrol, although their use for these purposes in many countries is being phased out. The amount of lead dissolved in water depends on several factors, including pH, temperature, water hardness and standing time of the water, with soft, acidic water being the most effective factors. Lead in humans with long term exposure can occur in acute or chronic damage to the nervous system of humans. In view of lead toxicity, WHO has set a health-based guideline limit of 0.01 mg/l.

***e) Mercury* (Hg)**

The primary source of mercury is the sulphide ore cinnabar. Mercury (Hg) is usually recovered as a by-product of ore processing. After release to the environment, mercury usually exists in mercuric (Hg^{2+}), mercurous (Hg^{2+}), elemental (Hg), or alkyllated form (methyl/ethyl mercury) (Hammer and Hammer, 2004).

Mercurous and mercuric mercury are more stable under oxidizing conditions. When mildly reducing conditions exist, organic or inorganic mercury may be reduced to elemental mercury, which may then be converted to alkyllated forms by biotic or abiotic processes. Mercury is most toxic in its alkyllated forms which are soluble in water and volatile in air. Hg^{2+} forms strong complexes with a variety of both inorganic and organic ligands, making it very soluble in oxidized aquatic systems (Hammer *et* *al.,* 2004). Sorption to soils, sediments, and humic materials is an important mechanism for removal of mercury from solution. Mercury may also be removed from solution by co-precipitation with sulphides (Hammer *et al.,* 2004).

Mercury toxicity results in mental disturbance and impairment of speech, hearing, vision and movement (Hammer and Hammer, 2004). In addition, mercury may cause the development of autoimmunity in which a person's immune system attacks its own cells. This can lead to joint diseases and ailment of the kidneys, circulatory system and neurons. At higher concentrations, mercury can cause irreversible brain damage (Momodu *et al.,* 2010). It is due to this effect from mercury that the WHO guideline is set at 0.006 mg/l (WHO, 2008).

3.4.7 Hydrochemical facies of groundwater

Chemical properties of water are presented by various methods, the most common of which are the hydrochemical facies (Al-Agha *et al.,* 2004), which are distinct zones that possess cation and anion concentration categories. The most common means to depict these zones are the use of the Collins bar diagram, the Stiff pattern diagram, the pie diagram and the Piper diagram (Kresic, 2007). The Piper diagram is the most convenient for plotting the results for multiple analyses of the major ions on the same graph (Kresic, 2007) and is useful in bringing out chemical relationships among groundwater samples in more definite terms as shown in Figure 3.3 and 3.4.

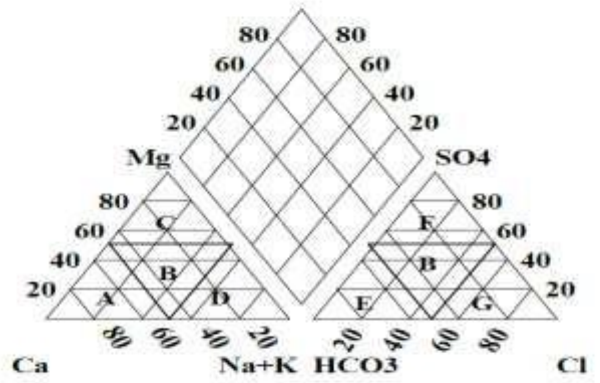


Figure 3.3. Piper diagram for groundwater classification (Kresic, 2007)

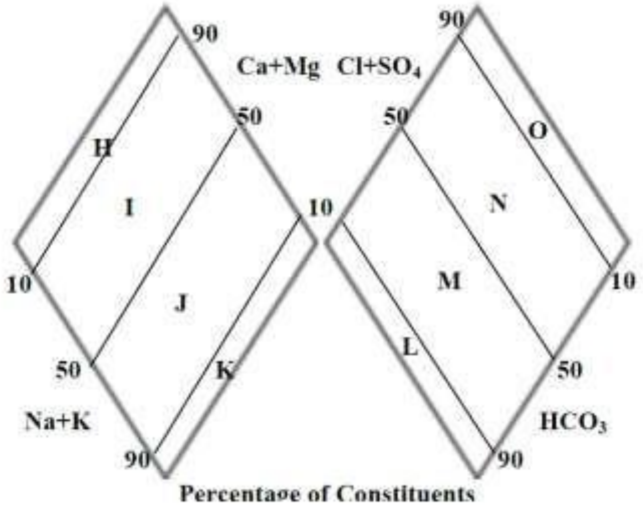


Figure 3.4. Template for classifying water into facies (Kresic, 2007)

The definition for the sections of the Piper diagram is as shown in Table 3.4 (Domenico and Schwartz, 1998) and recent classification is by the United States Geological Survey (USGS) is represented in Figure 3.5.

Table 3.4. Definition for sections on the Piper diagram

Diagram Sections	Definition for subdivision
A	Calcium dominates
B	No Dominant ion
C	Magnesium dominates
D	Sodium and potassium dominates
E	Bicarbonate dominates
F	Sulphate dominates

G	Chloride dominates
H	Calcium-magnesium facies
I	Calcium-sodium facies
J	Sodium-calcium facies
K	Sodium-potassium facies
L	Bicarbonate facies
M	Bicarbonate-chloride-sulphate facies
N	Chloride-sulphate-bicarbonate
O	Chloride-sulphate

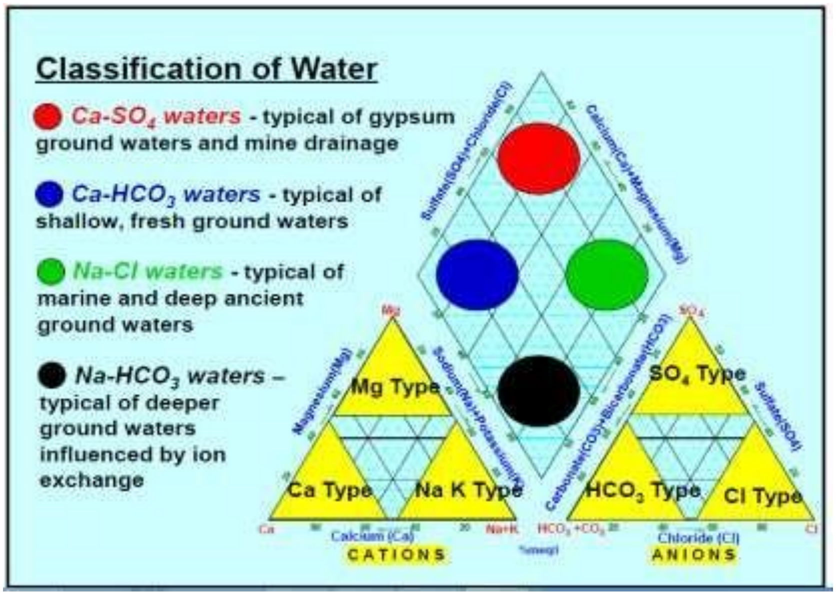


Figure 3.5. Classification of water type *(After USGS, 2011)*

3.4.8 Human health risk assessment of cyanide in water and soil

According to Dzombak *et al.*, (2006), evaluating potential human health risks of cyanide in water and soil is complicated by the chemical differences among cyanide compounds. Cyanide chemistry and environmental conditions influence potential human health risk in a number of ways potential for release from an environmental source (e.g., soil) to an exposure medium (e.g., air), and uptake into the body as a function of exposure route, exposure duration and inherent toxicity. For example, acute exposure to free cyanide (HCN, CN^-) in air and water may be an issue in occupational settings, but exposures to cyanide at inactive sites are typically subchronic (weeks to months) or chronic (at least 10% of a lifespan), and involve primarily iron–cyanide compounds in water and soil.

3.5 Cyanide attenuation in tailings sub-strata

The environmental fate of free cyanide and of metal-cyanide complexes has been under intensive investigation over the last decades and the attenuation potential of the sub-surface has been found to be helpful (Simovic *et al.,* 1984). The attenuation of cyanide mobility is due to geochemical and biogeochemical processes that naturally occur in the subsurface. Several geochemical mechanisms can account for the removal of cyanide from solutions. (Smith, 1988; Rouse *et al.,* 1988; Chatwin *et al.,* 1987; Simovic *et al.,* 1984; Schmidt *et al.,* 1981; Knowles, 1976).

According to Chatwin (1989), volatilization of free cyanide is one major physicochemical mechanism that will limit the mobility of free cyanide and weak metal-cyanide complexes in the near-surface environment. Volatilization becomes quite significant when pH is reduced below 9.4. Most soils are capable of neutralizing alkaline seepage and lowering the pH. Photodecomposition of metalcyanide complexes accelerates the volatilization process by breaking down metalcyanide complexes in the presence of sunlight. Such decomposition will take place in the tailing impoundment; however, factors such as sunlight penetration will determine the extent of photodecomposition (Smith, 1988; Rouse *et al.,* 1983; Schmidt *et al.,* 1981). Moderately strong and strong metalcyanide complexes such as ferro and ferri-cyanide complexes are exceptionally stable and have a geochemistry that is totally different from the free forms of cyanide, such as molecular hydrogen cyanide and simple ionic cyanide. Such iron-cyanide complexes are much more capable of interacting geochemically and bio-geochemically with soil and aquifer materials. These interactions lead to immobilization of the cyanide complexes and eventual decomposition by microbial activity (Smith *et al.*, 1988; Scott, 1984; Huiatt *et al.*, 1983).

Clays are classified as soils and sediments with particle size less than 2µm and contain minerals such as illite, kaolinite, smectite or montmorillonite (Craft, 2005).

According to Rouse *et al.*, (1993), to investigate the attenuation potential of any geological material, the bulk mineralogy, mineralogy of clay-sized fractions (or clay type) and some geochemical parameters should be determined.

The list of geochemical parameters that can affect the degree of attenuation taking place in a natural earth material, as indicated by Rouse *et al.*, (1993) are the following:

- Clay content.
- Cation Exchange Capacity (CEC) and exchangeable cations.
- Iron and Manganese hydrous-oxide content.
- Anion Exchange Capacity (AEC).
- Organic Carbon Content.
- Soil pH.

In addition, clay minerals have low permeability when compacted (Craft, 2005). For the purposes of this study, however, the following attenuation properties were considered; clay content, CEC, AEC, iron oxide content, optimal moisture content (OMC), soil pH, hydraulic conductivity and permeability characteristics because of their key role in the attenuation process.

CEC is the degree to which a soil can adsorb and exchange cations, on their negatively charged surface. McBride (1994) refers to it as the quantity of cations reversibly adsorbed per unit weight of mineral. It is expressed as mili-equivalent per 100 g of soil. According to Ma *et al.*, (1999), clay minerals have the property of adsorbing certain ions and retaining them in an exchangeable state, where the exchangeable ions are held on external surfaces of the mineral. CEC is used to estimate the capacity of a soil to protect water resources from metal contamination and it increases with increasing pH. It is expensive to directly measure CEC, hence soil-testing laboratories estimate, rather than measure CEC.

Estimates are made by determining the extractable cations (K^+, Ca^{2+}, Mg^{2+}, and Na^+) and estimating H^+ and Al^{3+} from the soil and the buffer pH measurements. According to Worral (1986), the CEC is determined by leaching the clay with a chosen electrolyte, so as to replace all existing cations by one particular cation; the clay is then filtered, washed free of excess electrolyte (often with alcohol rather than water to avoid hydrolysis) and an amount of chosen cation determined. In contrast to CEC, AEC is the degree to which a soil can adsorb and exchange anions and it increases as soil pH decreases.

3.5.1 Cyanide attenuation potential of clay

Fuller *et al.*, (1980), stated that attenuation tends to follow certain rules. An example is that given by Filipek (1999), which states that at a given solution pH and constant concentration of all other ions, the mass-action law requires that the attenuation capacity of a soil for a given ionic constituent will increase as the concentration of that constituent in solution increases. At a given concentration of a constituent in solution and a constant pH, the total attenuation capacity of a soil for that constituent will increase with decreasing concentrations of competing ions (ions whose charge is of the same sign as the constituent of interest and competing with each other (Filipek, 1999).The effectiveness of any of these mechanisms depends on the geochemical characteristics of both the soil and the fluid moving through it, as well as on physical factors such as climate and degree of soil saturation (Filipek, 1999).

Investigations carried out by Chatwin *et al.,* (1987), into the cyanide attenuation potential of several soils using soil column tests, revealed that specific soils can have significant cyanide attenuation abilities.

Alesii *et al.* (1976), performed column experiments with five different soil types; (sand, silty clay, and clay) and three different influent solutions of cyanide; KCN in de-ionized water, $K_3Fe(CN)_6$ in de-ionized water, and KCN in natural landfill leachate, to investigate the mobility of cyanide in the respective soils. The results of their study showed that factors such as high iron content, acidic pH, and the presence of organic matter and other metal oxides increases the attenuation of cyanide in soils. However, according to Higgins *et al.,* (2006), the soils used in Alesii *et al.,* (1976) column experiments were biologically active, and biological degradation may have influenced the fate of free cyanide in their experiment.

Generally as the pH increases, the attenuation capacity for cations increases whiles anions decreases. Furthermore, as the pH of a solution changes, the attenuation capacity of a particular soil for a given constituent changes (Filipek, 1999). Meeussen *et al.,* (1994), confirmed this assertion from results obtained from their experiments with soils from 12 manufactured gas plant sites in the Netherlands. Their experiments revealed that even clay soils with relatively high pH levels ($5.5<pH<7.2$) drastically immobilized iron cyanide transport than sandy soils with low pH levels ($3.5 <pH <5.5$). Conversely, calcareous soils with low clay content and high pH, sediments, aquifer materials dominated by sand and clays with high CECs (e.g. montmorillonite clay) are less effective at scavenging cyanides and thus, enhance the mobility of cyanide (Ghosh *et al.,* 2006c; Alesii *et al.,* 1976).

Several studies have been carried out on the adsorption of metal-cyanide complexes and free cyanide species on natural soil adsorbents as indicated by Ghosh *et al.,* (2006a). Higgins and Dzombak (2006) established that under close-to-neutral pH conditions, free cyanide adsorption on soils is linked with organic carbon content. According to Dzombak *et al.,* (2006), free cyanide does adsorb significantly on soils with substantial organic carbon content and also on soils such as clays with high AEC. Rennert *et al.,* (2002) confirmed that soil organic matter can have an important role in increasing the adsorption of both iron–cyanide complexes. However, in general, soil properties such as low pH (pH <5), presence of iron, manganese and aluminium oxides and hydroxides, clay material (e.g. kaolin, chlorite and gibbsite) with high AEC, tend to increase adsorption of anionic metal–cyanide complexes in soil and groundwater environment (Ghosh *et al.,* 2006a).

3.5.2 Methods of investigating attenuation potential of clay

Column Leaching Test (CLT) and Sequential Batching Test (SBT) can be used to investigate the attenuation potential of clay in the laboratory. As established by Fuller, (1982), CLT generally involves the addition of a solution, to the top (or bottom) of a tube or column filled with soil or rock pieces and collecting the solution that exits at the other end of the column. The column test is typically run until several "pore volumes" of solution have been flushed through the column.

For the purposes of CLTs, a pore volume is defined as the amount of solution necessary to fill all the interstitial void spaces of a given volume of soil or rock while porosity is defined as the ratio of the void volume to the total volume of a soil or rock in-situ. For that reason, the amount of solution in a pore volume for a high porosity soil or rock is expected to be greater than the amount of solution in a pore volume for a low porosity rock. The permeability of the soil or rock however, affects the time it takes to run a CLT. Thus, if the permeability is low, one pore volume of solution may require several weeks to traverse the column. The test, however, can take several months to more than a year to be completed (Filipek, 1999).

The SBT procedure on the other hand, simulates seepage solution percolating through columns of soil material when successive solutions come into contact continuously with fresh material. This testing procedure can simulate years of potential field seepage or groundwater flow in a few days of laboratory testing (Rouse *et al.,* 1993). According to Rouse *et al* (1993), SBT can be completed relatively quicker and is ideally suited for determining the attenuation characteristics on a number of samples at one time. However, the results from CLT generally give a true description of the attenuation processes. The limitation in using the CLT is that it may require several months to generate useful data for materials with low permeability.

3.5.3 Estimation of geotechnical parameters of clay

The relationship between hydraulic conductivity and grain-size distribution of granular porous media has been recognized a long time ago (Freeze *et al.,* 1979). The introduction of grain-size distribution analysis by Krumbein (1934) has been useful to Geologists and Geotechnical Engineers in distinguishing between different depositional environments (Alyamani and Sen 1993). Determination of the hydraulic conductivity and permeability characteristics of the clay materials being used as the TSF liner is essential in evaluating its effectiveness in preventing the seepage of contaminants in the TSF.

Hydraulic conductivity is a measure of the ease with which fluid flows through permeable materials, while permeability characterizes the ability of a porous medium to transmit fluids (Alyamani *et al.*, 1993). Furthermore, permeability is dependent on the physical properties (such as grain size, grain shape and arrangement and pore interconnections) of the porous medium while hydraulic conductivity depends on the properties of both the porous medium and the fluid flowing through it (Kresic, 2007). It is therefore expected for certain relationships to exist between hydraulic conductivity and statistical parameters that describes the grain-size distribution of depositional medium. Since the work by Hazen in 1893, these relationships have been studied by several investigators such as Kozeny (1927), Carman (1937, 1956), Terzaghi *et al.,* (1964), Shepherd (1989), Vukovic *et al*. (1992), and Alyamani *et al.,* (1993)

Grain-size distribution is mostly used to estimate the hydraulic conductivity of soils such as clay, due to the fact that the textural properties of clay can be obtained easily. According to Odong (2007), methods of estimating hydraulic conductivity from empirical formulae based on grain-size distribution characteristics have been developed. Although hydromechanics concepts make it more useful to characterize the diameters of soil pores rather than those of the grains, it is very difficult to determine the pore size distribution. Hence, an approximation of soil hydraulic properties based on the easy-to-measure grain size distribution, are often used as a substitute (Cirpka, 2003).

CHAPTER 4: RESEARCH METHODOLOGY

4.1 Introduction

This chapter deals with the materials and methods that were used in this study. It includes sections on desk studies, sources of data and field work to collect data for laboratory analyses. Sub-surface migration of contaminants from cyanide gold tailing impoundments is a major environmental impact that has to be incorporated into legislations in Ghana.

4.2 Sources of Information

Relevant secondary data in connection with the study were obtained from GFGL's, various departments and agencies and are shown in Table 4.1.

Table 4.1. Types and sources of data

Type of Data	Source of Data	Year of Data
GFGL Tarkwa Mine Tailings Storage Facility No 3 Detailed Designed Report and Operating Requirements	Metago, Designed report for GFGL	2010
Environmental Management Plan	GFGL Tarkwa Mine	2008-2011
Environmental Impact Statement for the Expansion of Facilities at the GFGL Tarkwa Mine	Metago,	2007
Environmental Annual Reports	GFGL Tarkwa Mine	2007-2011
Detailed monitoring map, Mine	GFGL, Tarkwa Mine	2012
Infrastructure Plan of study area		
Environmental Laboratory SOPs	GFGL, Tarkwa Mine	2011

4.3 Data Collection

In this section, sampling, sample preparation and analysis methods are described. Mathematical formulae used for the calculation of hydraulic conductivity and permeability of the clay/soil liner of the TSF are described in detail. Determination of trace metals, anions and physical parameters in water are also discussed.

Sampling was carried out in three different phases to determine plume dispersion of contaminants from the cyanide gold leaching operation at GFGL Tarkwa Mine. Field and laboratory investigations of the movement and attenuation of tailings/cyanide derived contaminants in groundwater at cyanide leaching facilities are:

- Profile sampling of the decant pond of the TSF, underdrainage sumps and spigotting points
- Groundwater sampling from boreholes around the tailings facility, cyanide ponds of the heap leach facilities and hand dug wells in the catchment communities
- Field leaching experiment to determine the attenuation potential of the clay liner and effect on monitoring boreholes at the toe of the TSF

4.3.1 Tailings sampling

Sampling of the TSF was carried out in three areas involving the supernatant pond (decant), three underdrainage sumps which collects seepages from the network of underdrainage pipes and slurry from spigotting points along the discharge pipeline. Profile depth sampling was carried out in the decant ponds on the TSF 1 and 2 respectively. Sampling regimes were undertaken from July 2010 to July 2011.

Sampling was carried out in the mornings and evening in order to identify weather-driven variations in physico-chemical parameters. Sampling locations were picked at random and accessed by a canoe and the use of a depth sampler (1.5 liters) at a meter interval and the contents emptied into a dark (500 ml) and 1000 ml sterilized sample bottles. A total of eighty-nine samples were taken at thirteen locations in both TSF 1 and 2 which were operational during the data collection. Sampling locations in the decant ponds are shown in Figure 4.1. Table 4.1 summarizes the sampling regime.

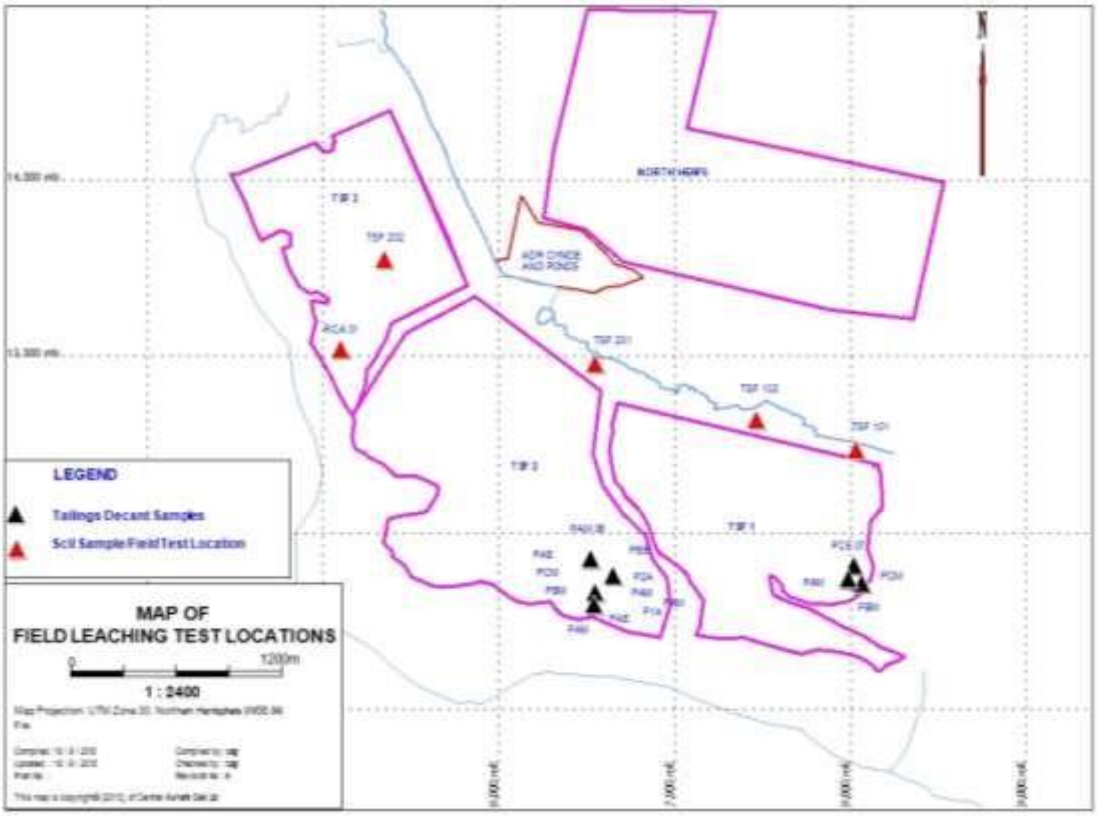


Figure 4.1. Decant sampling location map

Table 4.1. Decant sampling regime

Date	Location	Sample ID	Maximum Depth (m)
12/06/10	TSF 2	PBE06	6
	TSF 2	PAM06	7
	TSF 2	PAE06	8
09/07/10	TSF 1	PCE	5
	TSF 2	PAE07	8
07/08/10	TSF 2	PAM08	8
	TSF 2	PMB08	7
	TSF 2	PCM08	6
13/12/10	TSF 1	PAM12	8

	TSF 1	PBM10	6
23/03/11	TSF 2	PAM23	6
	TSF 1	PBM23	7
28/07/11	TSF 2	PCB	2
	TSF2	PAM	3

4.3.2 Field leaching test

The TSF 1 and 2 were fully operational at the time of the study and samples of the liner material could not be obtained. There was no evidence of excavation of unsuitable material in the basin or borrowed material as lining material which suggests that the in-situ material was used as liner. The field leaching test was undertaken in order to assess the suitability of the in-situ material of the TSF basin.A reconnaissance survey was conducted at the TSFs of GFGL to identify sites for the field leaching tests. Five sites around TSF 1, TSF 2, River at Containment Area (RCA) and the TSF 3 area were selected and their co-ordinates taken with a Brunton MNS Multi-Navigator GPS. The locations of the sites are shown in Figure 4.2
(Station 1, 2, 3, 4 and 5), with further details attached in Appendix 1.

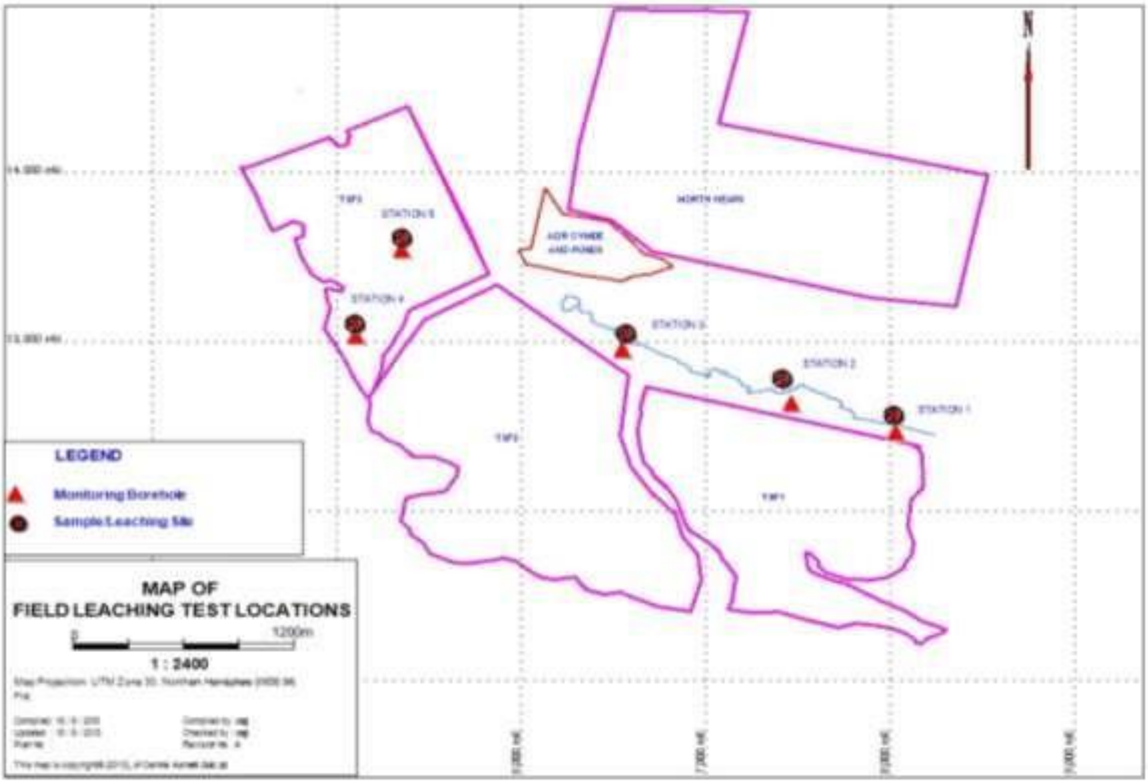


Figure 4.2. Field leaching test locations

Follow-up visits were made to the selected sites to collect samples to the laboratory for grain size analyses (sieve analysis and hydrometer tests), and geochemical analyses.

4.3.3 Field leaching test methodology

i. Soil sampling and leaching test

Ten (10) soil samples were collected from the drilled, leaching points of the five sites in July, 2010 prior to the leaching test. The same number of samples was taken in June 2011 to analyse for geochemical parameters after the leaching test.

These disturbed samples of varied texture were taken at depths ranging from 1 to1.5 m based on sample classification and the entire pipe length of 2 m. The samples were coned and quartered on a clean polyethylene sample sheet and representative samples taken and transferred into a new, clean polyethylene sample bag. Samples earmarked for dry sieve analysis weighed 7 kg each, whilst that earmarked for hydrometer tests and geochemical tests weighed 1.5 kg each. The samples were then transported to the Geotechnical Engineering Laboratory of the KNUST Civil Engineering Department and Council for Scientific and Industrial Research (CSIR) – Soil Research Institute (SRI), where geotechnical and geochemical tests were carried out. The attenuation properties of the clay samples considered are the hydraulic conductivity, permeability characteristics (porosity amd pore shapes) and geochemical properties. Grain size analyses were conducted on the clay samples to determine the hydraulic conductivity and the permeability characteristics of the clay samples. Geochemical analyses were also conducted to determine the geochemical properties of the clay material and exchangeable cations present which assist in the cyanide attenuation process.

Four monitoring points/holes of 100mm diameter was drilled at distance of 3 m from the leaching point/central hole were also similarly constructed as shown in Figure 4.3. A depth of 1.5 m was chosen to conform to the thickness of the clay layer in the TSFs base lining. A 50 mg/l NaCN solution was poured into the leaching points at the respective leaching sites, up to the 1.5 m level and leached over respective periods. The leachate in the monitoring points at the respective leaching points were sampled and anlysed for Total and WAD cyanide. The PVC pipes at each site were covered with water proof polyethylene sampling bags and sealed with masking tapes to prevent contamination, before and after sampling.

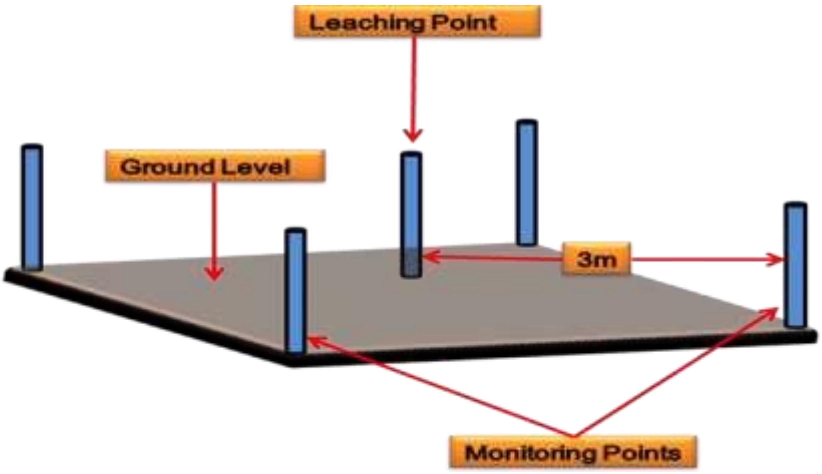


Figure 4.3. Schematic drawing of sampling/leaching test site

The NaCN concentration used conforms to GFGL mine limits for WAD cyanide within wastewaters. Furthermore, the interval of 3.0 m between the leaching and monitoring points was chosen to assess the concentration levels of detected cyanide at a distance of 3 m. Moreover, the time frame for the leaching test was short; hence if the distance was increased beyond 3m, a longer time travel would be required for the cyanide leachate to get to the monitoring points. Cyanide leachate in the monitoring points surrounding the leaching points were sampled monthly and analysed for WAD and Total cyanide. The leaching points were refilled with the 50 mg/l NaCN up to the 1.5 m mark and the monitoring points emptied of its contents after each period of sampling. The objectives of the field leaching test are:

- Assess clay geochemical parameters after subjecting it to the same cyanide concentration in the tailings slurry
- Analyse concentration of leachates through the clay substrata in order to determine impact of Total and WAD cyanide on groundwater

4.3.4 Groundwater monitoring

The physico-chemical parameters of groundwater in the TSF basin, heap leach facilities (North and South) and catchment communities were determined to establish the extent of impact of tailings in TSF on groundwater quality. This was done in view of the fact that groundwater in the TSF basin is susceptible to pollution due to its shallowness (that is about 15 m deep), which would in turn affect groundwater resources downstream through plume dispersion. The levels of the chemical parameters were ascertained by using both primary (2010 - 2011) and secondary data (from 2004 when monitoring of groundwater quality in the TSF basin started to 2009). Seasonal monitoring variations were used to ascertain the impact of tailings in TSF on groundwater especially in the dry season.

Water samples were collected from monitoring boreholes around the tailings storage facility (MBH 13, 14, 15, 16, 17, 18, 30, 31, 32, 35, 36, 37, 38, 40, 42, 43 and 44) as illustrated in Figure 4.4 and 4.5 respectively. Boreholes around the South Heap leach facility (SBH 01, 02, 03, 04, 05 and 06), boreholes around the North heap leach facility(NBH 04, 09, 19, 20, 21, 22, 23, 24 and 26) and hand dug wells in the five catchment communities. These sampling points were selected in order to assess the impact of tailings dams on groundwater and contaminant plume dispersion. Comparism was made between monitoring data from 2004 to 2008 (historic data) the study regime (2009 to 2012). The sampling was done with strict adherence to sampling protocols (Refer to Appendix 2) adopted by GFGL Environmental laboratory for Water Quality Monitoring and Quality Assurance Program. At each sampling point, three samples were collected for anion, metal and cyanide (free, total and WAD) analyses.

One litre acid-washed HDPE sampling bottles were used for all samples except those taken for cyanide, which were 500 ml acid-washed HDPE dark bottles. The sampling bottles were acid washed in order to hinder all microbial activities in them, which otherwise might cause changes to the concentrations of the inorganic constituents dissolved in the water sampled. The water samples were collected with a bailer after pumping the monitoring boreholes for five minutes in order to purge the well of stagnant water to reflect current state of water with respect to previous sampling.

Sample locations are represented in Figures 4.4 and 4.5.

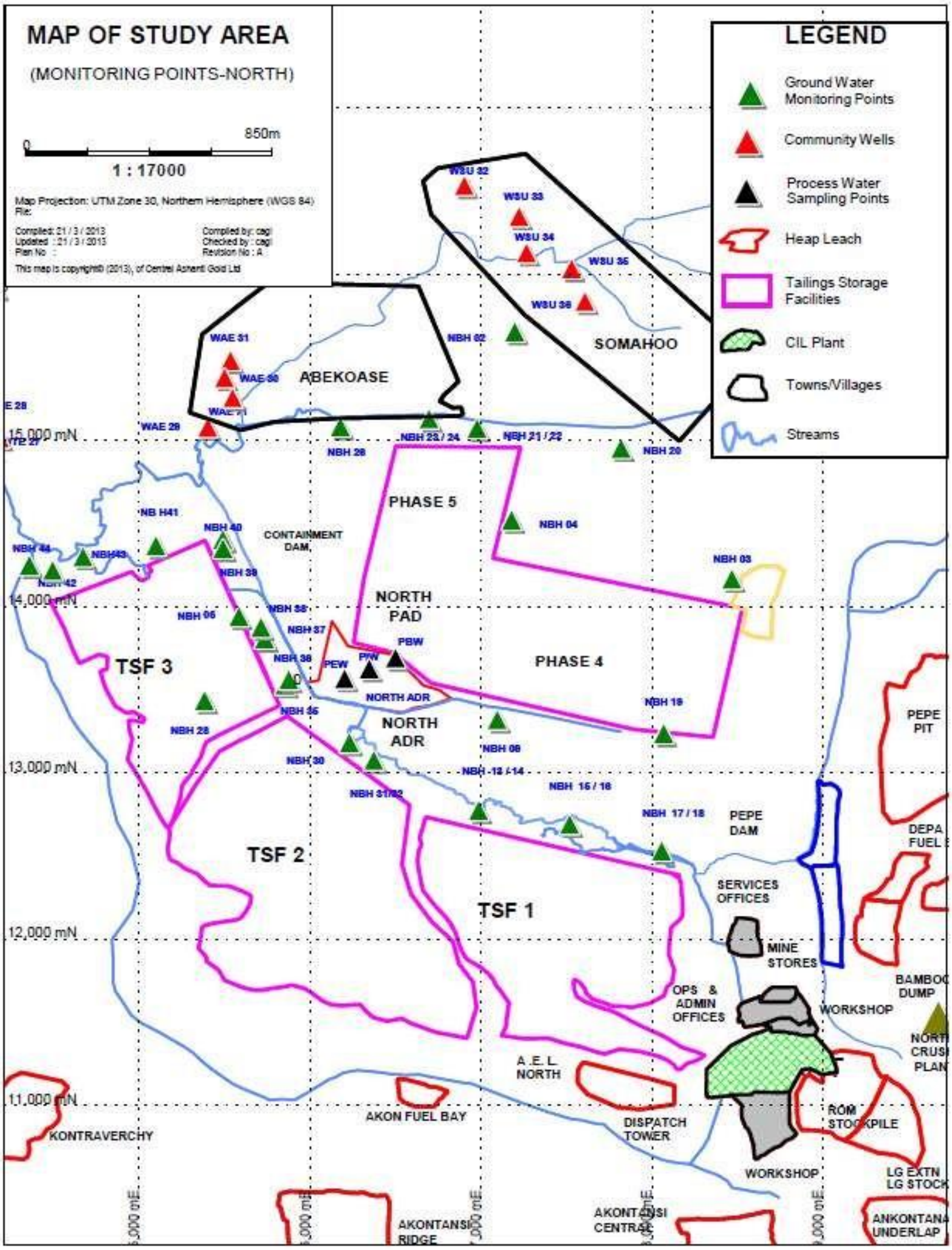


Figure 4.4. Monitoring map of the North facilities

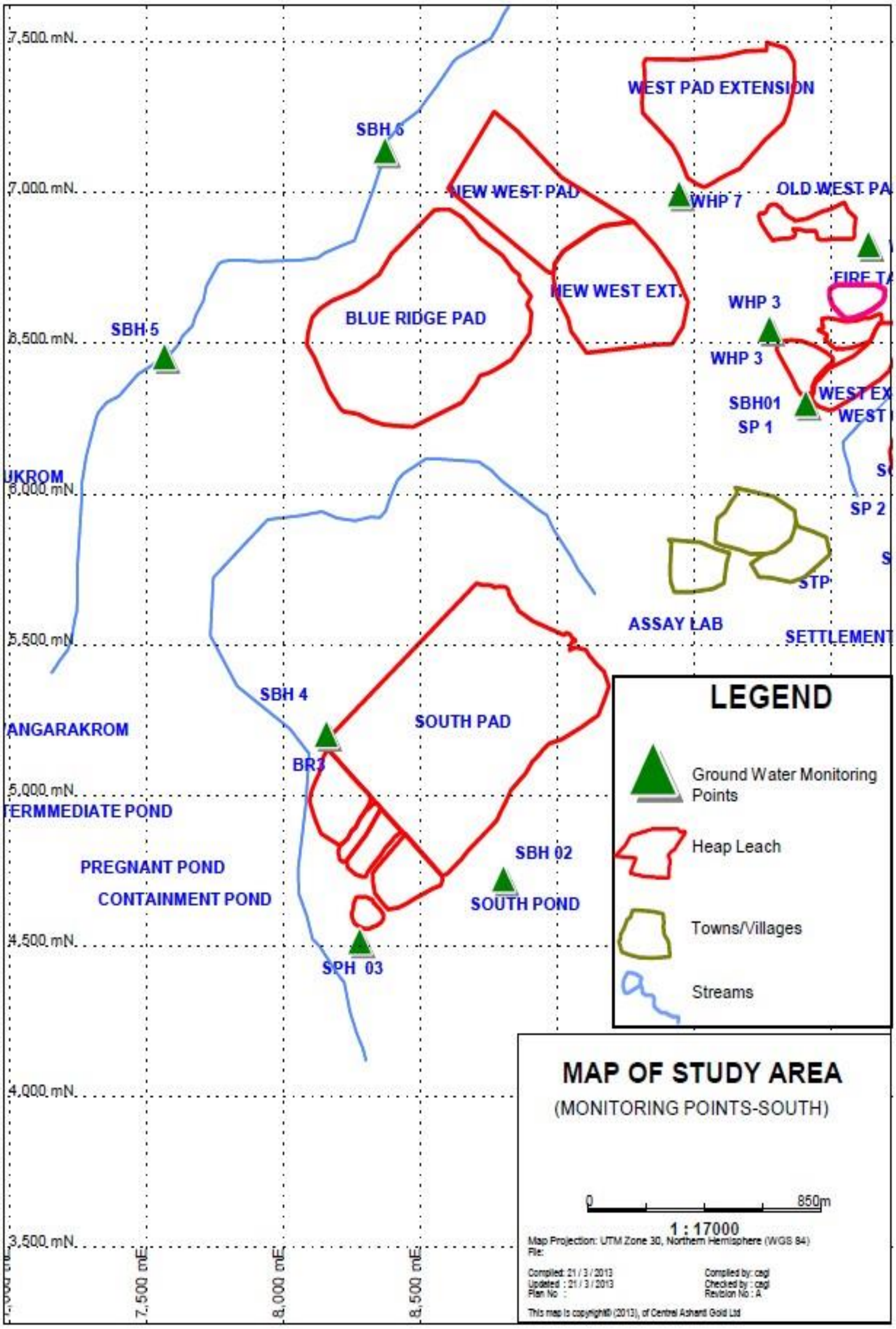


Figure 4.5. Monitoring map of the South facilities

Particulate matter was removed from samples by filtering with a 0.45-μm cellulose filter membrane as drinking water standards for most substances are based on dissolved concentrations, after which the samples were preserved. Samples for cyanide (free) were preserved through addition of 10 ml of 1.0 M sodium hydroxide solution to sample volume of 0.5 litre in order to keep the pH above 10 to stabilize cyanide before its analyses. Samples for metal analyses were preserved through acidification with 10 ml of concentrated nitric acid to sample volume of 1 litre to avoid their being precipitated. Samples for anions were without chemical preservation. The following quality control and assurance were adopted for GFGL monitoring programme:

- Quality control samples (duplicates and blanks) were taken monthly for Quality Analysis (QA)/Quality Control (QC) purposes.
- For every monthly sampling event two duplicate samples, one from South ADR Plant area and another from the North ADR Plant area were taken from monitoring bores or downstream surface water monitoring points and analysed for all of the same parameters as the primary or original sample. Duplicate samples were taken at the same time as primary samples
- Two blank samples of de-ionized water were prepared and submitted monthly to the external laboratory. One blank sample was analysed for physical parameters, whiles the second for free and total cyanide

Labelling of the duplicate and blank samples was decoded as per the sampling procedures.

4.4 Trace Metals Determination

The trace metals in the water samples were determined using Atomic Absorption Spectrophotometer (AAS). The analysis is based on the principle that the atom in the ground state absorbs the light of wavelengths that are characteristic to each element when light is passed through the atoms in the vapour state. Appendix 3 describes the operation of the AAS.

4.5 Methods of determining soil parameters

The various methods that were used to determine the geochemical parameters, grain size parameters and cyanide forms are shown in Tables 4.2, 4.3 and 4.4.

Table 4.2. Method(s) used to determine geochemical parameters

Geochemical Parameter	Method
Clay Content	X-Ray Diffractometry
CEC & Exchangeable Cations	Colorimetry (Ca^{2+}& Mg^{2+}), Flame photometry (Na^{+}& K^{+})
AEC	Colorimetry
Fe, Mn & other trace elements	Spectrophotometry (AAS)
Soil pH	pH meter
OMC	Walkley-Black titrimetry method

Table 4.3. Method(s) used to determine grain size parameters

Parameter	Method
Hydraulic Conductivity and Permeability	Dry sieve analyses and Hydrometer tests

Table 4.4. Method(s) of Cyanide determination in leachate samples

Form of Cyanide	Method
WAD CN	Midi Crest Distillation followed by Spectrophotometry
Total CN	Midi Crest Distillation followed by Spectrophotometry

4.5.1 Hydraulic conductivity and permeability characteristics

In determining the hydraulic conductivity and permeability characteristics, the soil samples were tested for grain size distribution in accordance with standard procedures as stipulated by the British Standards (BS) Institution (1990), Methods of Test for Soils for Civil Engineering Purposes (BS1377). Sieve analyses were conducted using a series of BS sieves, while the hydrometer tests were also performed on the very fine particles of the soil samples using sodium hexa-metaphosphate as the deflocculating agent.

Odong (2007) reported that particle size analysis of sediment of interest can be used to estimate hydraulic conductivity (***K***), using empirical equations relating either *K* to some size property of the sediment. Vukovic and Soro (1992) reviewed several empirical methods from previous studies and presented a dimensionally correct general formula of hydraulic conductivity:

— g 2

4.1□

$K= Cf(n)d_e$ □ v

Where *K* = hydraulic conductivity, *g* = acceleration due to gravity, *v*= kinematic viscosity, *C* = sorting coefficient (which is dependent on porous medium parameters such as grain shape, structure and heterogeneity), f*(n)* = function of porosity (*n*), and d_e = effective grain size. The kinematic viscosity (*v*) is related to dynamic viscosity (μ) and the fluid density (*ρ*) as follows:

$$V \;□\; \frac{□}{□}. \qquad (4.2)$$

The values of *C, f(n)* and d_e are dependent on the different methods used in the grain-size analyses. Vukovic and Soro (1992) reported that porosity *(n)* can be derived from the empirical relationship with the coefficient of grain uniformity *(U)* as follows:

$$n=0.255(1+0.83^{u}□ \qquad (4.3)$$

Where *U* is the uniformity coefficient of the grain size and is given by:

$$U= d^{d}1060 \qquad (4.4)$$

d_{60} and d_{10} in the above formula represents the grain size diameter in (mm) for which, 60% and 10% of the sample respectively, are finer than.

Previous studies have presented the following formulae which take the general form presented in equation (4.1) but with varying *C, f(n)* and d_e values and their fields of applicability. Hazen (1993) originally developed an equation for the determination of hydraulic conductivity of uniformly graded sand which is useful for fine sand to gravel range, provided the sediment has a uniformity coefficient less than 5 and effective grain size between 0.1 mm and 3 mm.

Hazen: $$K=\frac{g}{v}\times6\times10^{-10}\left[1+10(n-0.26)\right]d_{10}^{2} \qquad (4.5)$$

Where the symbols have their original meaning.

The Kozeny-Carman equation, as noted by Odong (2007), is one of the most widely accepted and used derivations of permeability as a function of the properties of the soil medium. This equation was originally proposed by Kozeny (1927) and was then modified by Carman (1937, 1956) to become the Kozeny-Carman (2009) equation. However, it is not appropriate for either soil with effective size above 3 mm or for clayey soils (Carrier, 2003).

Kozeny-Carman:

$$K=\frac{g}{v}\times8.3\times10^{-3}\left[\frac{n^{3}}{(1-n)^{2}}\right]d_{10}^{2} \qquad (4.6)$$

Where the symbols have their original meaning.

Alyamani and Sen (1993) further developed on the Kozeny-Carman (2009) equation in order to improve on errors by incorporating units for consistency, therefore, it is remarkably different from those that take the general form of equation 4.1. Nevertheless, equation 4.7 is the well-known equations that also depend on grain-size analysis. The formula considers both sediment grain sizes d_{10} and d_{50} as well as the sorting characteristics. However, it is not appropriate for either soil with effective size above 3 mm or for clayey soils (Carrier, 2003).

Alyamani and Sen;

2

K=1300 l 0.025(d□□ o 50-d10)□□ (4.7)

Where, K = hydraulic conductivity (m/day), I_o = intercept (mm) of line formed by d_{50} and d_{10} with the grain-size axis, d_{10} = effective grain diameter (mm), and d_{50} = median grain diameter (mm).

4.5.1.1 Hydraulic Conductivity (K) determination

The grain-size distribution curves in conjunction with the Alyamani and Sen (1993), empirical formula (equation 4.7) were used to determine the hydraulic conductivity of the soil samples. The limitations of the other discussed empirical formulae prevented their use with respect to the samples considered in this study, hence the Alyamani and Sen (1993) empirical formula was chosen. Moreover, compared to the other empirical formulae, the Alyamani and Sen (1993) formula uses the combined effects of effective grain-size diameters, average grain-size diameters, sorting characteristics and relates K to the initial slope and the intercept of grain-size distribution curves, whiles the others often use only the effective and the average grain-size diameters.

To determine K, the percentiles corresponding to increments of 5% starting from 5% were computed from the grading analysis. The result is a sequence of grainsize readings as d_5, d_{10}, d_{15}……d_{90}, d_{95} (refer to Appendix 4 for the particle grading analysis). The grain size readings were plotted against corresponding percentiles on an ordinary sheet**.** A visual inspection of these plots revealed that the initial portions of *K* invariably appeared as straight lines (Alyamani and Sen, 1993). A straight line of fit was therefore drawn through these points as shown in Appendix 5**,** from which both the intercept of the line (I_o) and the corresponding grain-size values for 50% (d_{50}) and 10% (d_{10}) finer than, were determined for each sample.

Limitations of Alyamani and Sen Formula

As reported by Odong (2007), the limitations of the Alyamani and Sen (1993) formula are as follows:

- It is very sensitive to the shape of the grading curve;
- More accurate for well-graded soil samples; and
- Generally, hydraulic conductivity estimation using grading characteristics can relatively lead to under-estimation or over-estimation. This method was adopted due to well graded of the samples.

4.5.1.2 Determination of permeability

According to Kresic (2007), permeability (K_i) and hydraulic conductivity (K) are related by the following formula (Equation 4.8):

$$K_i = K\frac{\mu}{\rho g} \tag{4.8}$$

Where μ, ρ and g have their usual meanings. However, μ and ρ are related as shown in equation 4.2. Substituting equation 4.2 into equation 4.8 gives the following formula:

$$K = K_i \frac{v}{g} \tag{4.9}$$

Where K_i = permeability (m^2), K = hydraulic conductivity (m/s), ν = kinematic viscosity of fluid (m^2/s) and g = acceleration due to gravity.

4.5.1.3 Determination of porosity

The empirical relation derived by Vukovic and Soro (1992), (equation 4.3), and parameters deduced from the grain size analysis were used in determining the porosity of the clay samples.

4.6.2 Determination of geochemical parameters

The methodology of the determination of the geochemical parameters from the clay samples are discussed in this section. Clay samples were air dried at normal room temperatures for seventy two hours and pulverized using miniature mortar and pestle. The clay-samples were then sieved through a 1 mm sieve.

4.6.2.1 Clay content

The samples were tested at the Council for Scientific and Industrial research (CSIR) Soil Research Institute (SRI) laboratory using X-Ray Diffraction equipment and were found to be kaolinitic.

4.6.2.2 CEC and Exchangeable cations

The CEC was obtained from the sum of exchangeable cations (Ca^+, Mg^{2+}, Na^+, K^+) and exchangeable acidity (Al^{3+}, H^+) (Anderson and Ingram, 1993). 2.5 g of each sample was leached with a 50 ml ammonium acetate (NH_4OAc) buffer solution with a pH of 7 (Black, 1986). Aliquots of each leachate were used to determine the exchangeable cations and the exchangeable acidity.

Total Ca^{2+} and Mg^{2+} determination

5 ml buffer solution of ammonium (NH_4^+) and ammonium chloride (NH_4Cl), with a pH of 12, was added to 15 ml aliquot of each leachate, to change leachate solution to the basic medium. 1 ml of 2% KCN and 1 ml of 10% hydroxyl amine

hydrochloride was further added to the solution, to reduce interference.

The resulting solution was titrated with 0.02 molar ethylene diamine tetra-acetic acid (EDTA) to a blue endpoint with Erichrome Black Tea (EBT) as indicator as described by Rowell (1994). The titre values were recorded as that for Ca^{2+} and Mg^{2+}.

Ca^{2+} determination

5 ml buffer solution of potassium hydroxide, with a pH of 10 and 1 ml of 2% KCN solution was added to 15 ml aliquot of each leachate, and titrated with 0.02 M EDTA to a mauve/pink endpoint with murixide as indicator as described by Rowell (1994). The titre value obtained was recorded as that for Ca^{2+}. The difference between the total Ca^{2+} and Mg^{2+} titre values and Ca^{2+} titre values gives that of Mg^{2+}.

Exchangeable acidity determination

50 ml of 1 mole potassium chloride (KCl) solution was added to 10 g of each sample and shaken for an hour. An aliquot of 25 ml was taken from each sample filtrate and titrated with 0.025 M NaOH to a pink endpoint using 50 drops of phenolphthalein as indicator as described by Page *et al.,* (1982). The titre values were recorded as that for H^+ and Al^{3+}.

The respective titre values were used to determine the concentrations of exchangeable calcium, magnesium and exchangeable acidity in me/100g, using methods employed by CSIR-SRI as adapted from Anderson and Ingram (1993); Black (1986); Page *et al.*, (1982); Motsara *et al.*, (2008) and Rowell (1994).

Sodium and Potassium determination

15 ml aliquot of each sample extracts (leachate) were aspirated into a flame photometer and their readings (emissions) recorded. Working standards were similarly aspirated into the flame photometer and their emissions recorded. The concentrations of Na^+ and K^+ are obtained from a plotted calibration curve using the emissions of the working standards.

Anion Exchange Capacity

5 ml of 0.1 molar KCl was added to 4 g of each clay sample in an Erlenmeyer flask to saturate the clay. An additional 40 ml of 0.1 M KCl was added to each mixture and pH checked and recorded. If the pH falls outside the range of 2 and 8, it is adjusted with potassium hydroxide (KOH) or hydrogen chloride (HCl) to give pH values between 2 and 8. The resulting solutions were leached with 100 ml of 0.01 M KCl to replace all the anions in the clay with chloride ions (Cl^-). The leachate obtained was discarded and the same samples in the leaching tubes were leached again with 100 ml each of 0.5 M sodium nitrate ($NaNO_3$) to displace the chloride ions in the clay with nitrate ions.

The displaced Cl^- in the new leachate is then determined by titrating an aliquot of each leachate with 0.014 N $AgNO_3$ to brick red colour using 6 drops of 5% $K_2Cr_2O_7$ as an indicator. The amount of Cl^- for each clay sample represents its AEC.

Soil pH

25 ml of distilled water was added to 10 g each of air dried clay samples and shaken for 15 minutes. A calibrated HI 9017 microprocessor pH meter was used to measure the pH of the resulting suspension.

Dissolved Iron (Fe), Manganese (Mn), Copper (Cu) and Zinc (Zn)

25 ml of combined solution of 0.05 M EDTA and 1.0 M NH_4OAc extraction solution was added to 5 g of each clay sample and shaken for 2 hours. The mixture was filtered using a 125 mm diameter filter paper and the filtrate was then tested for the presence of Fe, Mn, Cu and Zn using the Buck Scientific Model 210 VGP Atomic Absorption Spectrophotometer. 25 ml of the combined EDTA and NH_4OAc solution was used as a blank sample. The method employed in the determination of these dissolved metals was adapted from Motsara and Roy (2008).

Organic Matter Content

The OMC of the clay samples were determined by a modified Walkley-Black method adapted from Okalebo *et al.*, (1993) and Nelson and Sommers (1982). 10 ml of 1 N potassium dichromate ($K_2Cr_2O_7$) solution was added to 1 g of each clay sample in an Erlenmeyer flask and swirled gently. 20 ml of concentrated sulphuric acid (H_2SO_4) was further added to the contents of each flask and swirled gently for a minute. The addition of the concentrated H_2SO_4 causes heat to be evolved and to drive the reaction to completion. The flask was allowed to stand for 30 minutes for the mixture to cool.

The contents of the flask was diluted with 100 ml of distilled water and swirled again to ensure thorough mixing. 10 ml of concentrated orthophosphoric acid (H_3PO_4) was added to the mixture and back titrated with 1 N ferrous sulphate ($FeSO_4$) solution to a green endpoint. A blank titration was carried out in an identical way using the same reagents, but omitting the clay samples. The % organic carbon and % organic matter were calculated from the titre values using CSIR-SRI methods as adapted from Okalebo *et al.*, (1993) and Nelson and Sommers (1982).

WAD and Total Cyanide determination

Leachate samples were collected from 20 monitoring holes at the field leaching site, starting from September, 2010, for WAD and Total cyanide analyses at the

SGS Laboratory Services in Tarkwa, Ghana. Sampling protocols according to Barcelona *et al* (1985) and GFGL (2006) were followed.

Samples earmarked for Total cyanide analyses were filtered through 0.45µm filters and preserved by the addition of 2 ml of 10 molar NaOH per litre of sample. Samples were then kept at a temperature of 4^{o}C until the time of analysis according to the SOPs of the Environmental Laboratory of the Tarkwa Gold Mine (Anon, 2008b).

The Total cyanide analyses were carried out by releasing the cyanide as HCN, from cyanide complexes by means of reflux distillation and absorbed in a NaOH scrubbing solution. The CN^- in the absorbing solution was then determined colorimetrically by reacting it with chloramine-T at a pH less than 8 to convert the CN^- to cyanogen chloride. Pyridine-barbituric acid reagent was added to the resulting solution after the reaction was completed to form a blue-coloured complex. The solution was then analyzed for Total cyanide at 578nm using a UVSpectrophotometer. In the above described method as adapted from the Standards Methods for the Examination of Water and Wastewater, Method 4500-CN C and 4500-CN E (APHA, 2005), by SGS Laboratory Services, strongly bound metal cyanide complexes (including iron complexes) weak cyanide complexes and free forms of cyanide are determined as the Total cyanide.

Samples earmarked for WAD cyanide analyses were filtered through 0.45µm filters and kept at a temperature of 4°C and analysed according to the SOPs of the Environmental Laboratory of the Tarkwa Gold Mine (Anon, 2008c). Each sample was slightly acidified to pH 4.5 with lead acetate buffer and refluxed for 1 hour. The HCN gas released from the free and weakly complexed cyanide was distilled and absorbed into NaOH solution.

The CN^- in the absorbing solution was then determined colorimetrically by reacting it with chloramine-T at a pH less than 8 without hydrolysing to the cyanates. Pyridine-barbituric acid reagent was added to the resulting solution after the reaction was completed to form a blue-coloured complex. The solution was then analysed for WAD cyanide at 578nm using a UV-Spectrophotometer. The method, as enumerated above, was adapted from the Standards Methods for the Examination of Water and Wastewater, Method 4500-CN-I (APHA, 2005), by SGS Laboratory Services.

CHAPTER 5: RESULTS AND DISCUSSION

Results and discussions regarding cyanide attenuation potential of the sub-strata material of the TSFs are presented in this chapter. The results presented include geochemical parameters, estimated hydraulic conductivity, and permeability, mean porosity of the clay samples, Total cyanide and WAD cyanide. Furthermore, the relationship between the geochemical parameters and measured cyanide concentrations are discussed. The linkage existing between hydraulic conductivity of the sub-strata material and its ability to attenuate cyanide is also established.

5.1 Cyanide decay and tailings geochemistry

5.1.1 Cyanide decay

According to Botz and Mudder (1999), cyanide tailings disposed in a surface impoundment experiences a loss of cyanide due to natural degradation, frequently reducing the cyanide concentration to very low levels. Quantifying cyanide losses in terms of pond geometry, local weather conditions and feed solution chemistry has been largely empirical though in many instances, mining operations rely on surface impoundments to reduce cyanide concentration to below an internally regulated concentration or an effluent limitation. Cyanide, depending on its form and concentration tends to react readily with many other chemical elements and molecules to form, as a minimum, hundreds of different compounds (Flynn *et al.,* 1995).

Many of these broken-down compounds, while generally less toxic than the original cyanide, are known to be toxic to aquatic organisms, and persist in the environment for significant periods of time. In addition, there is evidence that some forms of these compounds can be accumulated in plant tissues (Eisler, 1991) and may be chronically toxic to aquatic life (Heming, 1985).

WAD and total cyanide levels were monitored at locations in the decant ponds of TSF1 and TSF 2. Results in Table 5.1 and Figure 5.1 and 5.2 shows that both WAD and total cyanide concentrations increased significantly with respect to depth of pond water in TSF 1 and TSF 2. Figure 5.3 and 5.4 shows dissolve oxygen and temperature variations in TSF 1 and TSF 2.

WAD and total cyanides concentrations of the two monitoring location increased with increase in depth of decant.

The pH of samples at PBM and PAM increased with increase in depth of the decant pond (see Figure 5.1 and 5.2). According to Lighthall *et al.*, (1987) the rate of natural degradation is a function of cyanide type and concentration, pH, temperature, bacteria, sunlight, aeration and pond conditions, such as area, depth, turbidity, turbulence and retention time. Total cyanide concentration at monitoring point in TSF 1 from 0.36 mg/l from the surface to 1.11 mg/l at the depth of 5 meters with WAD cyanide concentrations increased steadily from 0.26 mg/l to 0.93 mg/l over the same monitoring depth in the decant pond. A second monitoring point in the decant pond of TSF 2 also returned an increased in Total cyanide concentration from 0.10 mg/l at surface to 0.28 mg/l over 10 meters of decant water and 0.26 mg/l over 5 meters depth at monitoring point PBM in TSF 1. WAD concentrations also increased from 0.01 mg/l to 0.16 mg/l over the same depth of decant water.

Table 5.1. Total and WAD cyanide of decant pond

Station ID	Sample ID	Depth (m)	pH	Average Temperature 0^c	Dissolve Oxygen	Total Cyanide (mg/l)	WAD Cyanide (mg/l)	Toxic Cyanide
PBM (TSF1) 02011-1	PBM	1	9.8	23.6	4.4	0.36	0.26	0.10
PBM (TSF1) 02011-2	PBM	2	9.8	23.4	2.8	0.43	0.26	0.16
PBM (TSF1) 02011-3	PBM	3	9.8	23.5	3.3	0.55	0.39	0.16
PBM (TSF1) 02011-4	PBM	4	9.9	23.5	3.0	0.61	0.44	0.17
PBM (TSF1) 02011-5	PBM	5	9.9	23.2	3.1	1.11	0.93	0.18
PAM (TSF2) 02011-1	PAM	1	9.4	21.6	4.1	0.10	0.01	0.09
PAM (TSF2) 02011-2	PAM	2	9.5	21.5	4.3	0.10	0.01	0.09
PAM (TSF2) 02011-3	PAM	3	9.5	21.1	4.0	0.11	0.03	0.10
PAM (TSF2) 02011-4	PAM	4	9.9	21.0	4.4	0.13	0.09	0.10
PAM (TSF2) 02011-5	PAM	5	10.0	20.9	4.1	0.15	0.15	0.06
PAM (TSF2) 02011-6	PAM	6	10.0	20.8	4.6	0.19	0.15	0.04
PAM (TSF2) 02011-7	PAM	7	9.7	20.7	3.9	0.20	0.15	0.05
PAM (TSF2) 02011-8	PAM	8	9.9	20.6	3.4	0.25	0.16	0.09
PAM (TSF2) 02011-9	PAM	9	9.8	20.8	3.1	0.29	0.17	0.12
PAM (TSF2) 02011-10	PAM	10	9.9	20.8	4.6	0.28	0.16	0.12

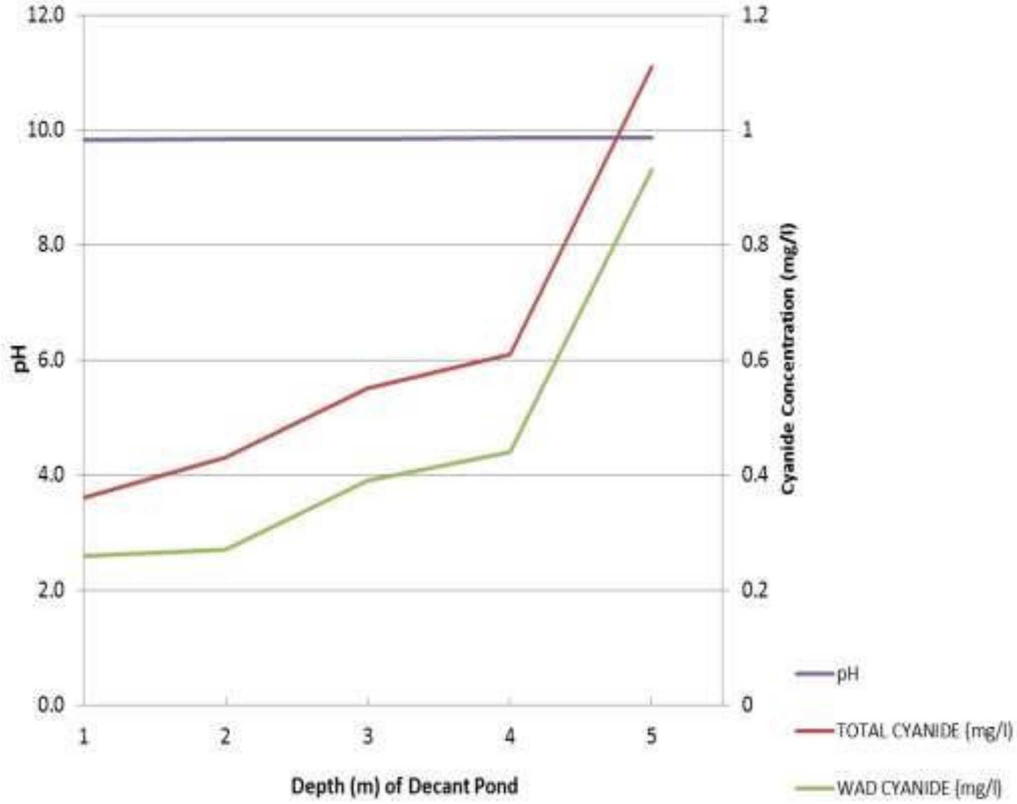


Figure 5.1. Total and WAD Cyanide levels at depth and pH levels (TSF 1)

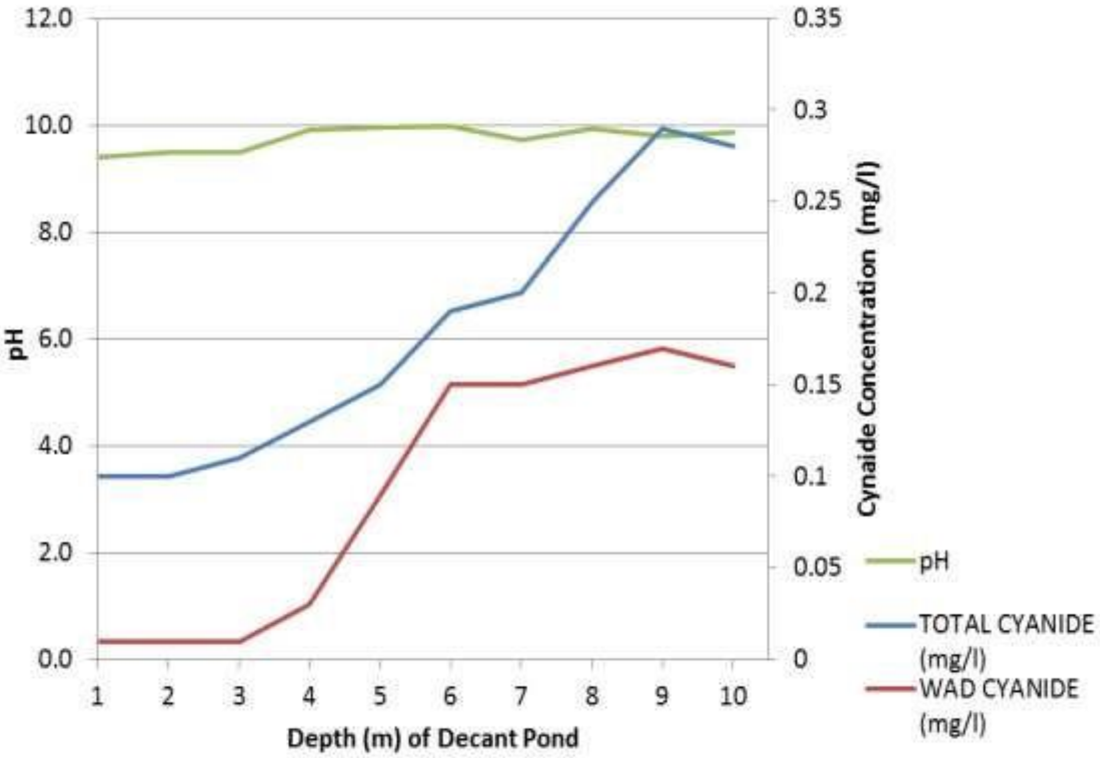


Figure 5.2. Total and WAD Cyanide levels at depth and pH Levels (TSF 2)

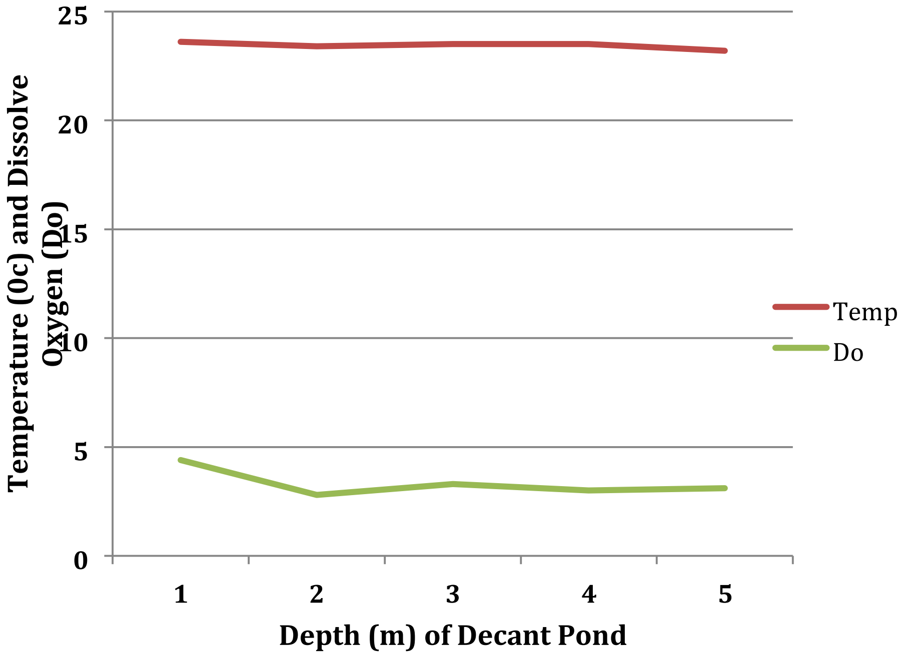


Figure 5.3. Temperature and Dissolve Oxygen variations (TSF 1)

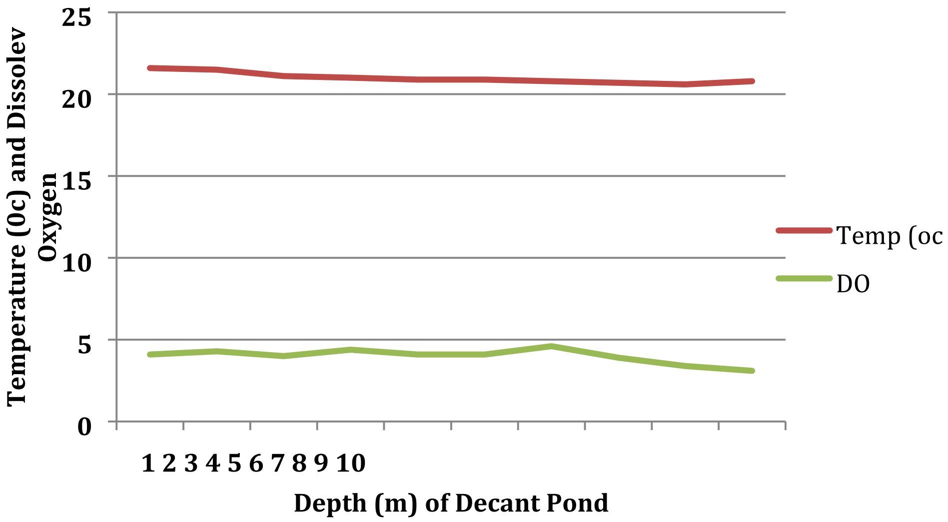


Figure 5.4. Temperature and Dissolve Oxygen variations (TSF 1)

It was observed from the cyanide analysis that the oxidation zone in the decant pond was 3 meters. Many aspects of the geochemical behaviour and toxicity of complex mixtures such as tailings are poorly known. For example, mining literature frequently state that cyanide naturally breaks down quickly, in the presence of sunlight, into relatively harmless, non-toxic substances (Moran, 2002). However, this breakdown was not observed in this study as shown in Figure 5.1 and Table 5.1, both total and WAD cyanide concentrations increased after 3 meters at PAM (TSF 1) and PBM (TSF 2) monitoring locations.

The pH concentration of the samples from the two monitoring points ranged from 9.5 to 10.0 in both TSF's. At monitoring location PBM in TSF 1, pH increased from 9.8 to 9.9 over a depth of 5 meters of decant pond and referring to Table 5.1, the second monitoring point in TSF 2 with a total depth of 10 meters of decant pond was also observed with increasing pH from 9.4 at a meter to 10.0 at seven meters and declined to 9.9 at 10 meters.

The concentration of WAD, Total cyanide and pH were dependent of two parameters which are temperature and Dissolve Oxygen (DO). The temperature and DO decreased with increase in depth of decant pond in both TSF 1 and 2. This could be attributed to non-oxidizing characteristics of the tailings material.

5.1.2 Metals, trace elements, physicals and anions concentration

According to Clarke *et al.,* (1987), cyanide species may be considered to act chronically on the environment in three distinct ways. The first is to produce a direct and ongoing minor acute poisoning, producing chronic symptoms, including anomalies in the nervous systems of many of the higher organisms. The second way is by the formation and/or the action of its compound thiocyanate in interfering with certain common metabolic pathways. Lastly, because cyanide and thiocyanate are capable of complexing with metals contained in natural water supplies, organisms using this water could either be subjected to toxic doses of certain metals or deficiencies of others.

Concentrations of trace elements, cations and anions were expressed as mean ± SEM (Standard Error of the Mean) by using the One-way ANOVA tool. This analyses and compare mean between analytes and differences resulting in probability value of $p<0.05$ were considered statistically significant and this analysis is presented in Appendix 6. The null hypothesis for the statistical analysis for the sample concentrations in the decant pond was defined as no significant relationship whiles the alternative hypothesis is defined as correlation among sample concentrations. From the statistical analysis carried out, the correlation coefficient (p) was positive and closer to 1, i.e $p<0.5$, thus there is a positive and strong correlation among the analytes and therefore the alternative hypothesis of existence of correlation among the analyte concentrations was accepted.

Anomalous trace elements concentrations in groundwater could arise from metals being retained in solution in assimilable complexes (e.g. Cu), while deficiencies in naturally occurring trace metals could result from the retention of these in nonassimilable complexes (e.g. Fe, Cu, Mn, Mg). Since there is a strong and positive correlation among the individual concentrations, the mean value for the various depths and parameters were used in the analysis of the data from the sampling regime. The pH values in the decant pond ranged between 9.5 and 10.8 with mean a concentration of 10.0 for shallow pond depths (1 to 4 meter) and 10.1 for depths ranging from 5 meters to 7 meters as represented in Table 5.2. However, lower mean pH was recorded at the depth of 8 meters in the decant pond.

Table 5.2. Mean physical and anion levels in decant pond of TSF 1 and 2

Depth m	pH	EC µS/cm	TDS mg/l	NH_3 mg/l	SO_4 mg/l	PO_4 mg/l	Cl mg/l	NO_3 mg/l	NO_2 mg/l
1	10.0	1112.0	744.4	0.14	33.40	0.62	26.22	171.44	128.11
2	10.0	1112.0	743.4	0.11	33.80	0.71	25.78	169.28	131.07
3	10.1	1154.8	775.9	0.13	33.60	0.56	26.74	167.22	134.94
4	10.0	1152.8	776.3	0.07	36.60	0.58	26.32	176.22	129.49
5	10.1	1154.3	778.8	0.06	34.80	0.67	22.75	221.70	116.63
6	10.1	1167.7	786.9	0.15	34.60	0.62	25.80	160.40	133.17
7	10.1	1170.0	787.5	0.10	37.67	0.66	27.80	177.63	141.87
8	9.9	1169.0	758.7	0.16	36.50	0.77	22.00	216.65	115.08

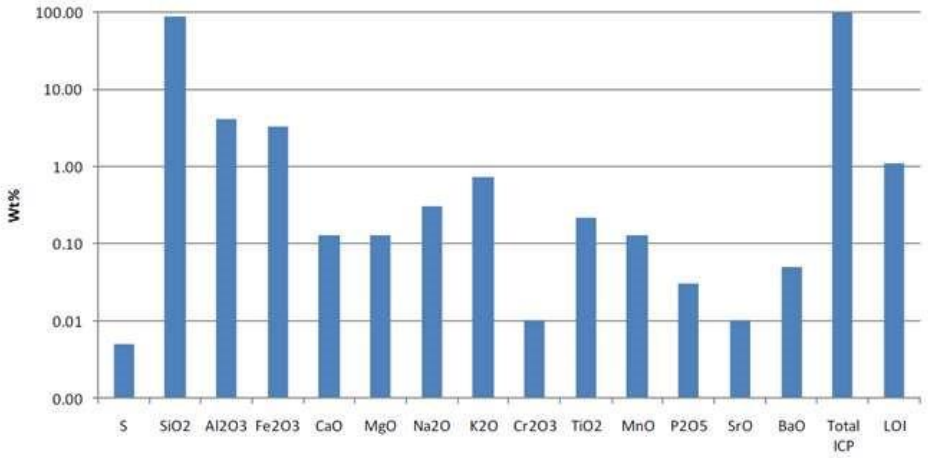
The high pH concentrations at depth imply there is a low rate of cyanide decay or conversion of cyanide into other forms such as complexes. TDS measurements denote the various types of minerals present in water in the dissolved form. In natural waters, dissolved solids include mainly carbonates, bicarbonates, chloride, sulphate, phosphate, silica, calcium, magnesium, sodium and potassium (Kresic, 2007). As expected, EC values of the decant pond exceeded that of the groundwater samples. EC concentrations increased with increase in depth which signifies that more metals are dissolved at depth of decant water. Although significant concentrations of sulphate, phosphate and ammonium were analysed, there was no clear correlation of their concentrations with depth and pH.

Table 5.3 represents the general chemical composition of tailings at TSF facilities. The analysis of the ore and tailings samples were undertaken in an attempt to establish the lithological components in order to identify major trace elements, cations and anions which have an impact on the leaching and solubility processes. This analysis will form a major component in the interpretation of elevated cations and anions in the groundwater concentrations in the TFS basin. Analysis of the mineral composition of the tailings indicated SiO_2 value of 87.96% by weight as the highest fraction and this consistent with the Tarkwaian ore.

Table 5.3. General Mineral Composition of Tailings

Mineral SQ-XRD	Chemical Formula	WT (%)
Quartz	SiO_2	87.96
Albite	$Na(AlSi_3O_8)$	4.74
Talc	$Mg_3(Si_4O_{10})O_{10}(O)_8$	2.65
Hematite	Fe_2O_3	2.57
Clinochlore/chlorite	$Mg_2Al_3(Si_3Al)O_{10}(O_8)$	1.34
Annite	$KFe_3AlSi_3O_{10}(OH)_2$	0.74

Multi-element analysis of decant pond at various depths were undertaken in order to assess the effect of impoundment depth on trace elements, cations and anions (Figure 5.5). Results of the analysis show a strong correlation between the concentration of elements and concentrations of the elements in the supernatant pond and tailings slurry at the spigoting points.



Chemical (Wt%)

Figure 5.5. Chemical composition of ore (After Anon, 2011)

The elements with the highest concentrations are Ca, Mg, Fe, Cu, Na, Mn and K. These results conform to the ore and tailings chemistry and geochemistry respectively.

Mean Na, Mn, Ca, Fe, Cu and Mg concentrations in the decant ponds of TSF 1 and TSF 2 increased with increase in depth but the same cannot be said about K which decreases with increase in depth of decant pond. K behaved differently from the other metal concentrations due to its large atomic size which replaces Na to form insoluble complexes at low pH, low DO and high nitrate concentration. Metal concentrations with depth of decant pond trends are illustrated in Figures 5.6 –5.10. Minimum, mean and maximum metal concentrations are represented in Appendix 7. Results from sample analysis from the supernatant tower (underdrainage sump which is 22 m lower in elevation compared to the level of the decant pond) returned the same elements and similar concentrations as that of the decant pond.

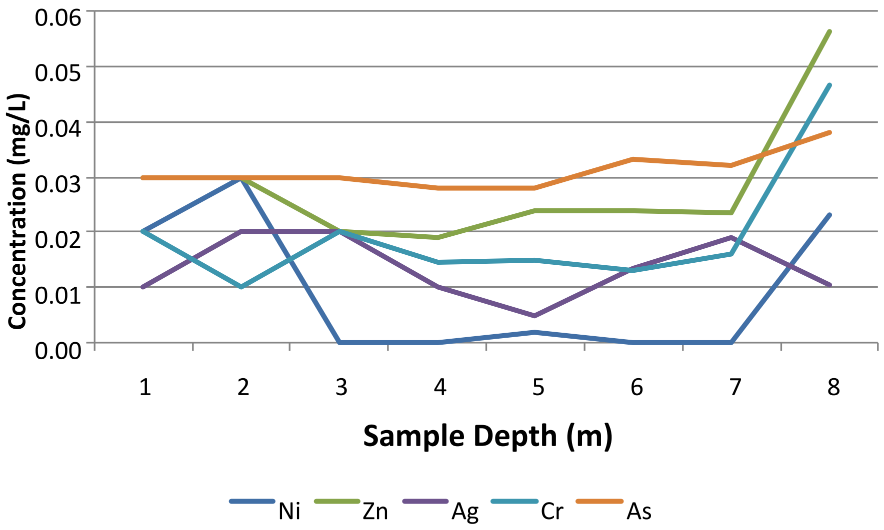


Figure 5.6. Graph of trace metals against sample depth

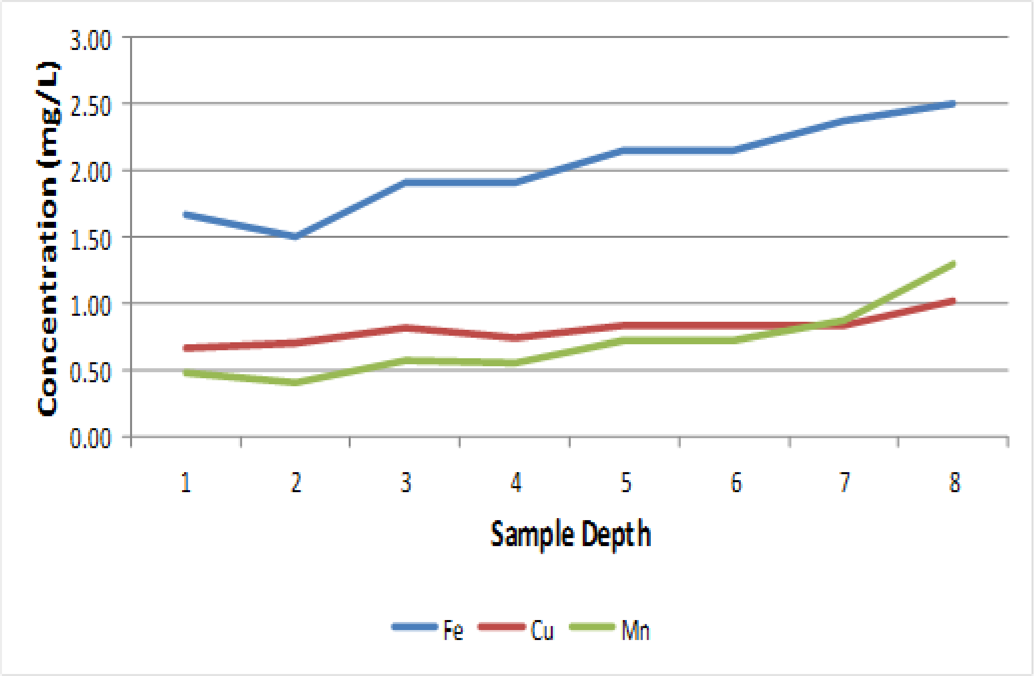


Figure 5.7. Graph of trace metals with high concentration at depth

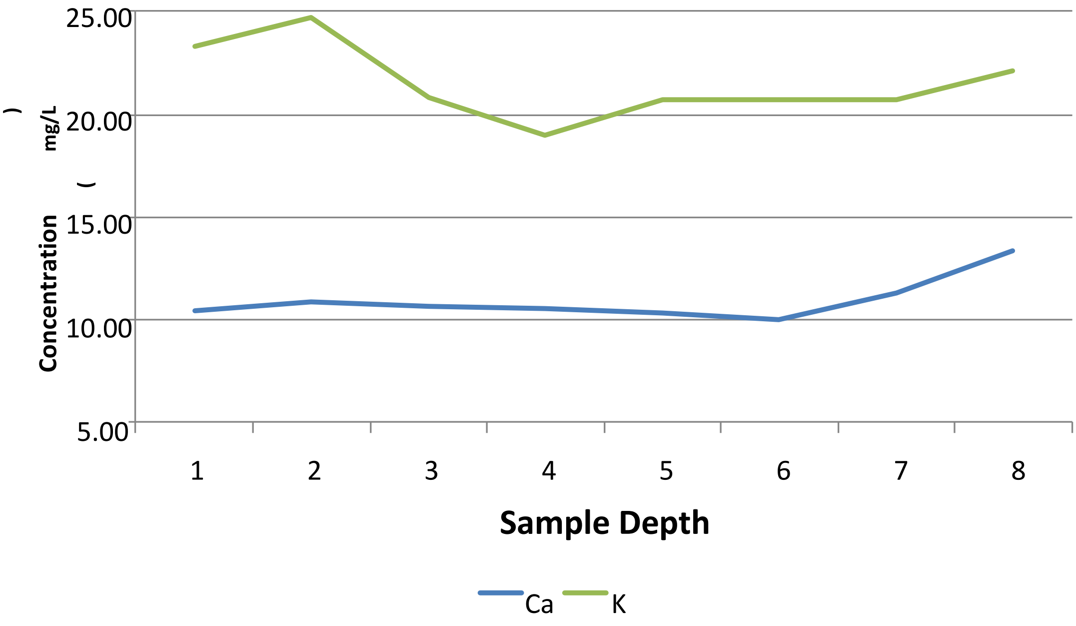


Figure 5.8. Graph of trace metals with high concentration at depth

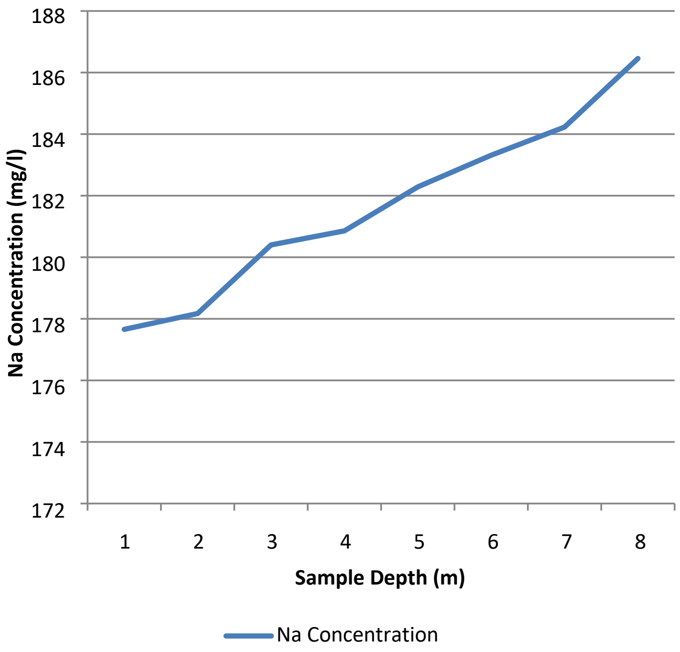


Figure 5.9. Na concentrations against sample depth

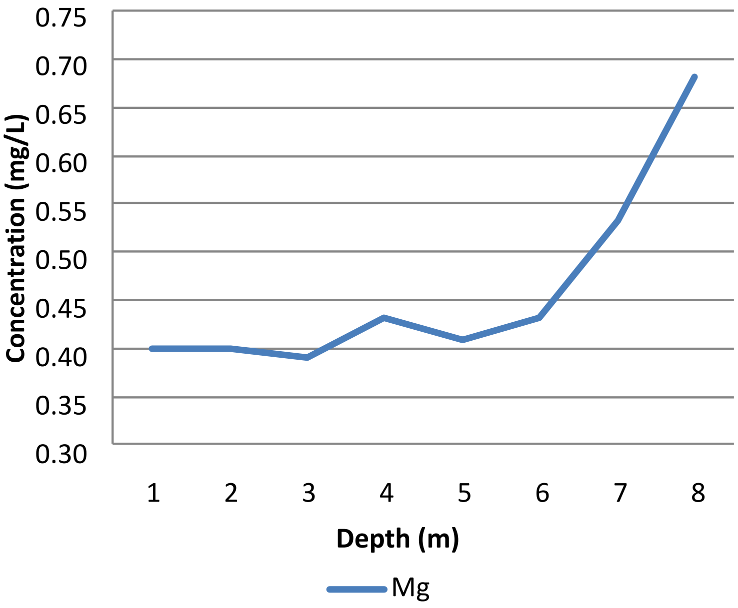


Figure 5.10. Graph of Mg concentrations against sample depth

Anions present in the decant ponds of TSF 1 and TSF 2 are Cl, SO_4, NO_2 and NO_3.

Similar cations and anions were detected in the tailings decant pond and the underdrainage system. Metal concentrations increased proportionally with increase in depth of decant pond but on the contrary, trace metal concentrations (Ni, Zn, Cr and As) increase after six meters in the decant pond. Ag concentrations decreased with increased in depth of decant pond. These metals exhibited constant concentration within the first six meters which marks the oxidation zone. Increase in Ag concentration after six meters could be attributed to formation of Ag complexes at depth.

Metals identified in the decant pond, underdrainage sump and tailings slurry were Sodium (Na) and Magnesium (Mg) with trace elements being Iron (Fe), Copper (Cu), Manganese (Mn) and Zinc (Zn), whiles alkali metals identified were Calcium (Ca) and Potassium (K) as shown in Table 5.4.

Table 5.4. Metals and trace elements in underdrainage sump

Sample Labels	Ca mg/l	Cu mg/l	Fe mg/l	Zn mg/l	Mn mg/l	Mg mg/l	K mg/l	Na mg/l
MS - 15	1.48	0.06	0.46	0.02	0.53	0.10	18.59	215.60
MS - 17	1.35	1.07	0.86	0.04	1.20	0.13	27.43	230.16
MS - 19	1.29	1.62	1.39	0.05	2.16	0.14	12.79	189.60
TSF1-SP	11.5	0.38	0.16	<0.002	0.15	0.169	26.80	173.77
TSF2-SP	9.02	0.01	1.00	0.089	0.11	0.113	17.38	247.92

The average concentrations of Na, Mg, Mn, Fe, Ca, Cu and K increase with depth whiles Zn, As, Ni and Cr increases at seven meters. Significant metals and trace elements consistent in ore, tailings, decant pond and in the underdrainage system which is an indication of non-precipitation of metals and oxidation reaction in the TSF which resulted in dissolve oxygen levels at depth.

5.2 Analyses of attenuation properties and effect on groundwater

There was the need to investigate the active clay constituents which are key components in the attenuation process of the clay liner before and after the leaching test. Analyses of results of the geochemical properties of the substrata (clay liner) were determined before and after the field leaching test and presented in this section. This was carried out to identify the geotechnical and geochemical properties of the substrata of the TSFs that are capable attenuating cyanide and to distinguish the characteristics for degrading cyanide. Clay samples from the controlled field leaching points were analysed geochemically and geotechnically prior to the introduction of 50 ppm WAD cyanide solution of 50 ppm. Results of clay properties are also discussed after the leaching test and physico-chemical parameters compared to pre-leaching properties.

5.2.1 Geochemical parameters

Geochemical investigations were carried out on clay samples collected from five sampling sites as follows and represented in Figure 5.11:

a) Along the toe of TSF 1 (Station 1 and 2)

b) TSF 2 (Station 3)

c) Basin of TSF 3 (Station 4 and 5)

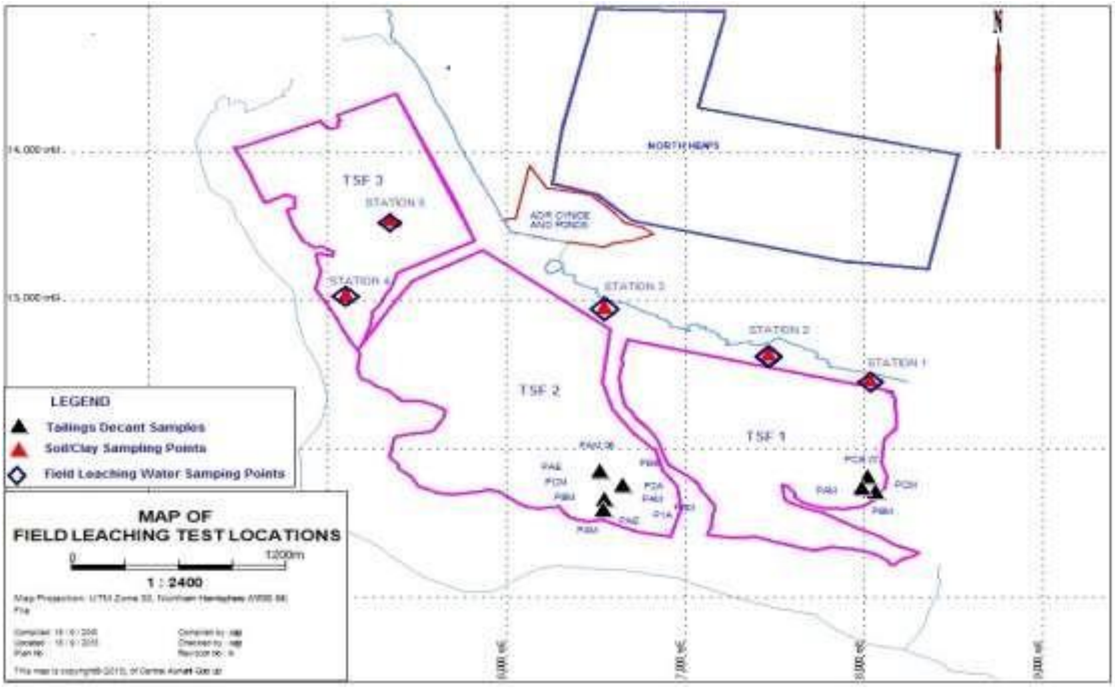


Figure 5.11. Geochemical sampling map

The pH values for the clay sampled ranged between 4.3 and 4.5 implying a moderately acidic clay medium. Samples from station 4 recorded the lowest pH of 4.3 and samples from station 3 returned the highest pH of 4.8. The oxide surfaces (notably iron, manganese, and aluminium oxides), carbonate surfaces, and insoluble organic matter can generate a significant number of positive charges as the pH decreases. The CEC results from the sampling, ranged between 1.92 meq/100g and 2.89 meq/100g and increased with increase in pH. In contrast to CEC, AEC generally decreased when pH increased and vice-versa. The AEC ranged between 882.25 meq/l and 1278.25 meq/l in reference to the results which are presented in Table 5.5. The AEC values were dependent on the OMC and this implies that anion exchange is hindered by the presence of organic and inorganic ligands in the samples resulting in the inhibition of the ion adsorption process.

Table 5.5. Geochemical parameters of sampled clays

Sample ID	**pH**	**C.E.C (meq/100g)**	**A.E.C (meq/l)**	**OMC (%)**	**Fe_2O_3 (ppm)**
Station 1	4.5	2.01	1278.75	0.36	49.57
Station 2	4.5	2.89	1057.50	0.53	63.14
Station 3	4.8	2.34	881.25	0.78	66.43
Station 4	4.3	1.92	1057.50	0.43	22.86
Station 5	4.5	2.37	881.25	0.98	29.00

The Fe_2O_3 values were also directly proportional to the OMC and affected by the pH concentrations. Geochemical parameters for CEC, AEC, OMC, pH and Fe_2O_3 of the clay samples from the substrata are effective in attenuating cyanide to tolerable and acceptable limits especially for stations 1, 2 and 3, with Station 5 being less effective due to its low Fe_2O_3, low AEC and high OMC. Graphical presentation of geochemical parameters at the leaching sites can be found in Appendix 8.

5.2.2 Geotechnical parameters of insitu material

The Alyamani and Sen (1993) empirical formula and relevant parameters derived from grain size analyses were used to determine the hydraulic conductivity of the clay samples collected from the five stations. The hydraulic conductivity values were in the range of 6.2×10^{-3} to 1.41×10^{-2} m/day with a mean of 8.3×10^{-3} m/day, and these are presented in Table 5.6. The maximum value of K was obtained from Station 3, whiles the minimum value of K was obtained from Station 5. Studies carried out by Bassington (2007) on different types of media (mixture of sand, clay and silt) indicate that the hydraulic conductivity values falls within the order of 10^{-2}

and 10^{-4} m/day. Based on this, the results of the geotechnical analysis of the TSF substrata can be said to be quite consistent with the hydraulic conductivity of clay.

Table 5.6. Summary of hydraulic conductivity of clay samples

Sample ID	**I_o(mm)**	**d_{10}(mm)**	**d_{50}(mm)**	**K(m/min)**
Station 1	1.17×10^{-3}	4.1×10^{-3}	5.34×10^{-2}	**5.2×10^{-6}**
Station 2	1.12×10^{-3}	4.1×10^{-3}	5.40×10^{-2}	**5.1×10^{-6}**
Station 3	2.17×10^{-3}	4.0×10^{-3}	4.88×10^{-2}	**9.7×10^{-6}**
Station 4	1.15×10^{-3}	4.1×10^{-3}	4.86×10^{-2}	**4.7×10^{-6}**
Station 5	1.11×10^{-3}	4.0×10^{-3}	4.68×10^{-2}	**4.3×10^{-6}**
			MEAN	**5.5×10^{-6}**

Permeability values (**k_i**) were calculated using the Kresic (2007) equation and these are presented in Table 5.7. The k_i are very low ranging from 7.72×10^{-12} to 1.39×10^{-11} m/min with a mean value of 8.42×10^{-12} m/min. Mean calculated permeability and hydraulic conductivities of the samples gave values that conform to acceptable regulatory materials for the construction of tailings substrata. Station 5 exhibited the least attenuating geotechnical permeability of $6.1 \text{ x } 10^{-12}$ m/min and hydraulic conductivity of $4.3 \text{ x } 10^{-6}$ m/min.

Table 5.7. Permeability (k_i) of clay samples

Sample ID	**K(m/min)**	**Mean Dynamic Viscosity**	**K_i(m/min)**
Station 1	5.2×10^{-6}	8.87×10^{-1}	7.72×10^{-12}
Station 2	5.1×10^{-6}	8.87×10^{-1}	7.50×10^{-12}
Station 3	9.7×10^{-7}	8.50×10^{-1}	13.9×10^{-12}
Station 4	4.7×10^{-6}	8.90×10^{-1}	6.91×10^{-12}
Station 5	4.3×10^{-6}	8.50×10^{-1}	6.10×10^{-12}
MEAN			**8.42×10^{-12}**

The estimated mean porosity of the clay liner is 26.21% (see Table 5.7) and this is shown in Table 5.8.

Table 5.8. Estimated mean porosity (n) of clay samples

No: of Samples	Mean d_{10}(mm)	Mean d_{60}(mm)	Mean $U = d_{60}/d_{10}$	Mean $n = 0.255(1 + 0.83^U)$
5	4.1×10^{-3}	7.8×10^{-2}	19.2069	0.2621 **Percentage = 26.21%**

Comparing this value with standard porosity values of 26 - 68% for clay as determined by Kresic (2007), and studies conducted by Barnie (2010), on similar clay materials in the Atankwidi sub-basin of the White Volta Basin in Ghana, the clay in the undisturbed areas of the TSF basin can be classified as clayey.

Table 5.9 gives a summary of the major attenuation properties of the TSF clay substrata material that influences cyanide attenuation (refer to Figure 5.12).

Table 5.9. Summary of attenuation potential and cyanide concentrations

Sample ID	*K* (m/day)	pH	C.E.C (meq/100g)	A.E.C (meq/l)	Total Cyanide (mg/l)	WAD Cyanide (mg/l)
Station 1	7.5×10^{-3}	4.5	2.01	1278.25	0.010	0.010
Station 2	7.3×10^{-3}	4.5	2.89	1057.50	0.020	0.011
Station 3	1.4×10^{-2}	4.8	2.34	881.25	0.030	0.010
Station 4	6.7×10^{-3}	4.3	1.92	1057.50	0.010	0.010
Station 5	6.2×10^{-3}	4.5	2.37	881.25	0.010	0.010

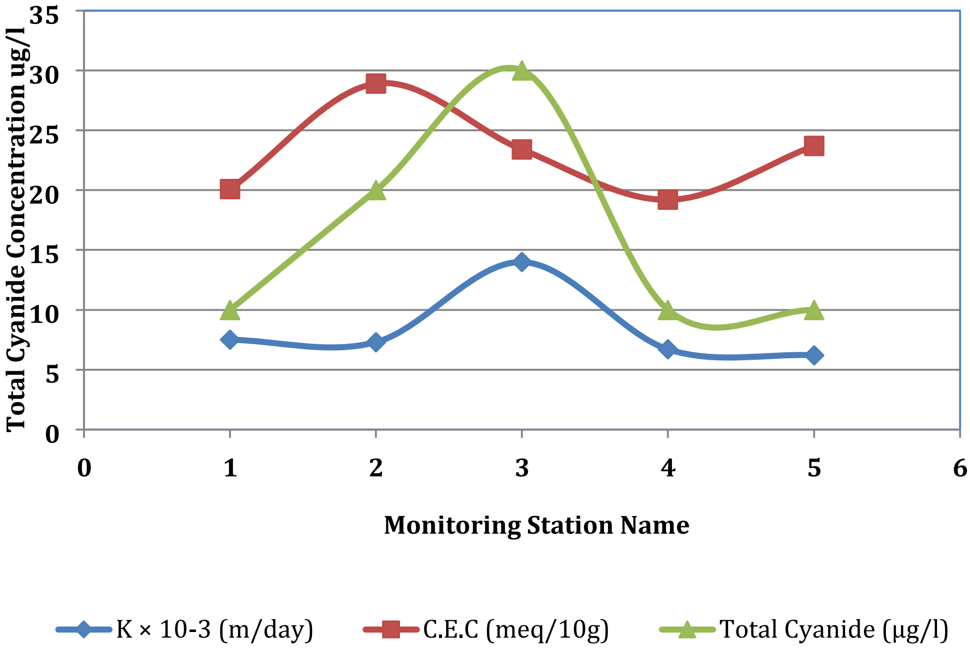


Figure 5.12. Geochemical characteristics of samples

Based on the estimated mean porosity values of the substrata material of the TSF basin, the TSF 1 and 2 areas are mainly clayey with that of TSF 3 as silty sand and is characterised by low lying and flat ground.

5.2.3 Cyanide decay analyses of the sub-strata material of the TSFs

Within a working radius of 3 m, the sub-strata material was analysed for cyanide at the respective leaching sites. Results of the average total cyanide concentrations range from 0.01 to 0.03 mg/l and WAD cyanide levels range from 0.01 to 0.011 mg/l. No concentrations of Free, WAD or Total cyanide were detected in the drilled holes prior to the leaching test. WAD cyanide results from the leaching test fall within the stipulated limits for TSFs with planned measures for cyanide degradation, where concentrations range from ~1-10 mg WAD-CN/l (NICNAS, 2010). Moreover, the Total and WAD cyanide results fall within stipulated limits of EPA-Ghana, which is 1 mg/l for total and 0.6 mg/l for WAD cyanide limit (Anon, 1997).

The analytical results of the cyanide tests of the five leaching sites are presented in Figure 5.13.

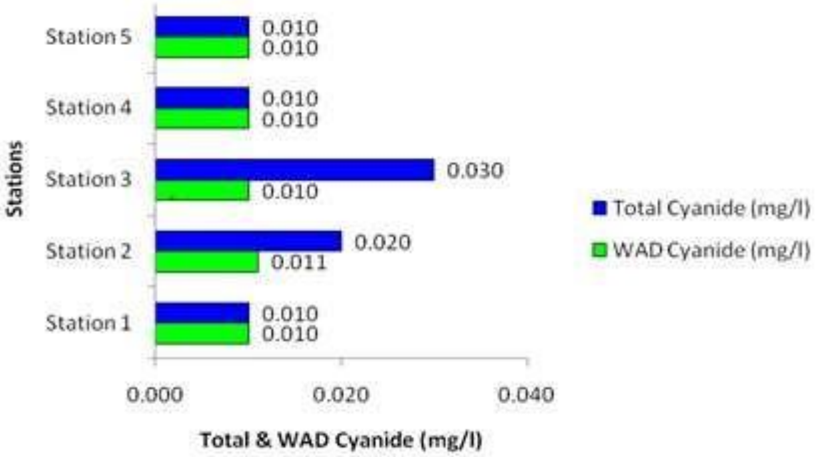


Figure 5.13. Average Total and WAD cyanide detected

However, the average cyanide detected with time at the five stations remained fairly constant over time as shown in Figure 5.14. Total cyanide decreased with time in the monitoring boreholes due to the adsorption of the solute particles by the clay particles which act as a barrier.

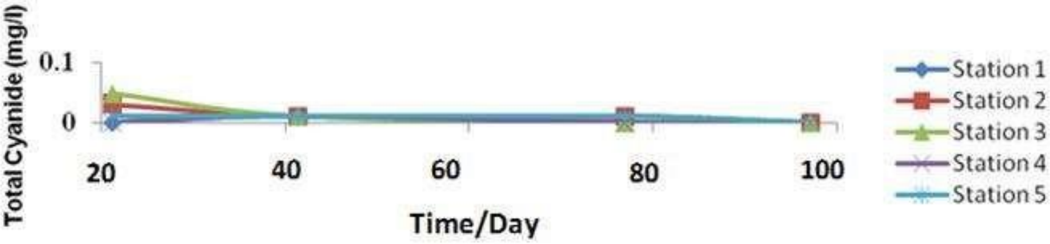


Figure 5.14. Average Total Cyanide detected with time

pH in the decant pond, tailings slurry and underdrainage supernatant tower remained alkaline with values ranging from 9.4 to 10.6. Comparing this to the natural pH of groundwater (4.5 to 5.6) in the study area, an unsaturated interface is created which could lead to water movement from the facility into aquifers.

5.2.4 Effect of hydraulic conductivity and permeability on cyanide attenuation

Total cyanide concentration from the leachate at Station 3 was highest, due to the fact that its hydraulic conductivity was the highest. Clay material from Station 5 had the least hydraulic conductivity and attenuated the introduced 50 mg/l NaCN to 0.010 mg/l of Total cyanide and WAD cyanide (Figure 5.15).

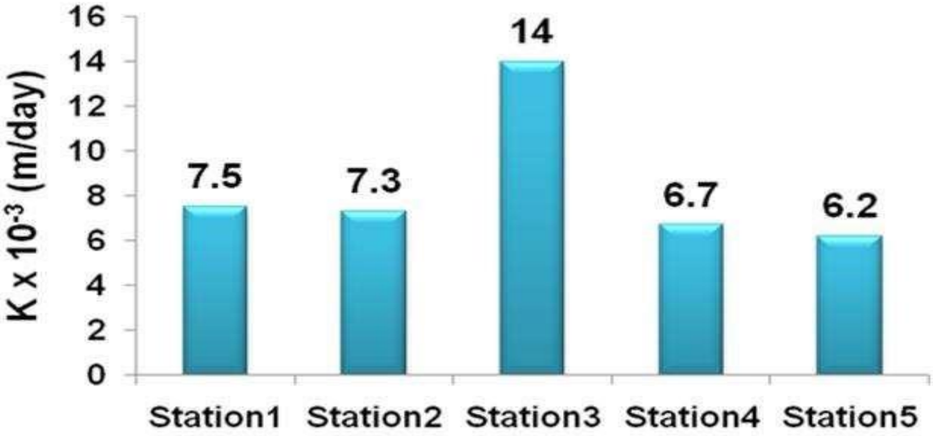


Figure 5.15. Hydraulic Conductivity results from the respective stations

5.2.5 Effect of Geochemical Parameters on Cyanide Attenuation

Based on the geochemical results from the leaching stations, it was observed that clay material from Station 5 had unfavourable geochemical parameters with respect to cyanide attenuation, such as the least AEC of 881.25 meq/100 g, a high CEC of 2.37 meq/100 g and a low iron-oxide content of 29 meq/100 g. However, it was acidic with a pH of 4.5 had the highest OMC of 0.98% and 0.010 mg/l Total and WAD cyanide concentration.

Clay material from Station 2 were able to attenuate the 50 mg/l NaCN to 0.020 mg/l of Total cyanide and 0.011 mg/l of WAD cyanide due to the fact that it was acidic having a pH of 4.5, had a high AEC of 1057.50 meq/100 g and high Feoxide content of 63.14 mg/l. The sample however, had the highest CEC of 2.89 meq/100g and low OMC (0.53%), thereby rendering it ineffective in cyanide attenuation. Despite the fact that there was 0.01 mg/l WAD cyanide in the leachate samples of Station 3, total cyanide values were the highest at 0.030 mg/l compared to the leachate from Station 2 (0.020 mg/l) and Station 5 (0.010 mg/l). Although the clay material at station 2 had the highest Fe-oxide content of 66.43 meq/100 g and a high OMC of 0.78%, the difference in cyanide attenuation was due to the fact that it had the least AEC of 881.25 meq/100 g, a high CEC of 2.34 meq/100 g and had the highest pH of 4.8. Clay material from Station 4 attenuated both Total and
WAD cyanide at 0.01 mg/l in spite of having a low OMC of 0.43% and the lowest Fe-oxide content of 22.86 mg/l. This attenuation can be attributed to the fact that clay material from Station 4 exhibited the lowest acidity with a pH of 4.3, had the lowest CEC value of 1.92 meq/100g and a high AEC of 1057.50 meq/l. Similarly, clay material from Station 1 also attenuated both Total and WAD cyanide at a value of 0.01 mg/l, although it had the lowest OMC of 0.36% and a relatively low CEC of 2.01 meq/100g.

The cyanide attenuation in Station 1 was due to the fact that Station 1 was acidic with a pH of 4.5 and had the highest AEC of 1278.25 meq/l and Fe-oxide content of 49.57 ppm. An assessment of the hydraulic conductivity, permeability and geochemical results of the clay material from the five leaching stations with respect to the Total and WAD cyanide concentrations from their leachate, revealed that low hydraulic

conductivity and a combination of low pH, relatively high AEC and low CEC were the major attenuation properties that influenced cyanide attenuation.
This agrees with studies conducted by Dzombak *et al.*, (2006a).

Significant variations were recorded in all the geochemical parameters of samples taken after the leaching test and these are the pH, AEC, CEC, Fe_2O_3 and OMC. pH decreased significantly for all the five leaching sites with values ranging from 3.0 to 3.4 and a corresponding increase in AEC ranging from a minimum value of 1650 ppm to a maximum value of 2116.67 meq/100 g. OMC for Station 5 reduced from 0.98% to 0.79% rendering it less effective in attenuating cyanide and other minerals.

Graphical representation of the geochemical parameters for the pre and post leaching test are represented in Figure 5.16 to 5.20. Fe_2O_3 levels for Stations 1(14.5%), 2 (60.9%) and 3 (49.5%) decreased after the test whiles that of Station 4 (27.3%) and 5 (8.0%) were elevated. The pH reduced after the test at all the five stations ranging from 3.0 to 3.4 compared to the pre-leaching pH range of 4.3 to 4.8. Stations 1 and 3 reduced by 33.3% whiles Stations 2, 4 and 5 reduced by 24.4%, 23.3 and 27.8% respectively. Geochemical parameters of the clay samples at various depths were analysed six months after the leaching field test. Results of the analysis are presented in Table 5.10 and 5.11.

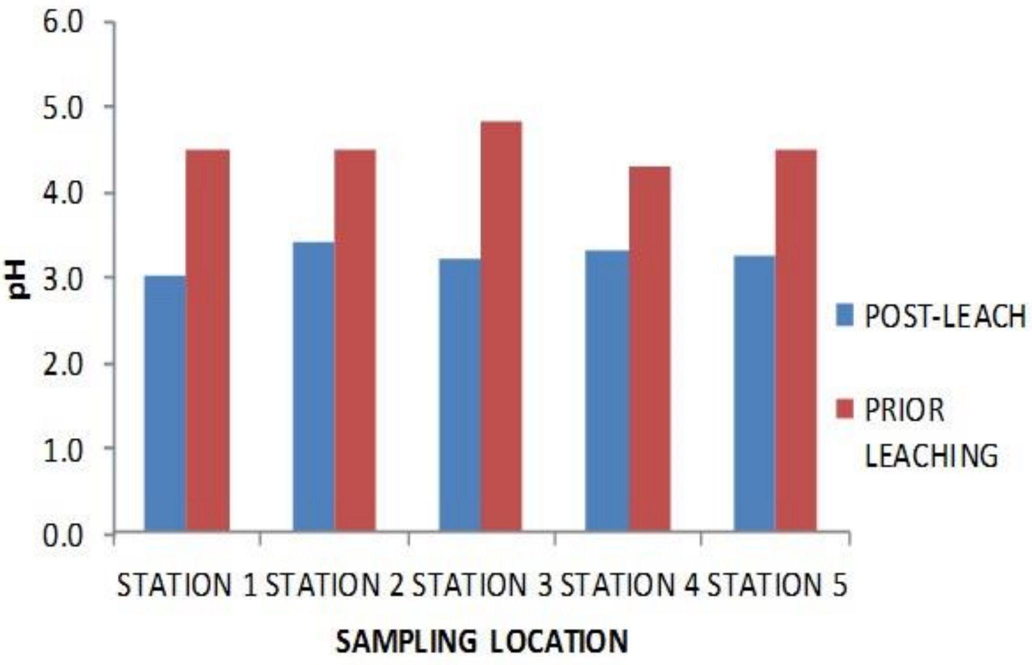


Figure 5.16. pH variation test sites

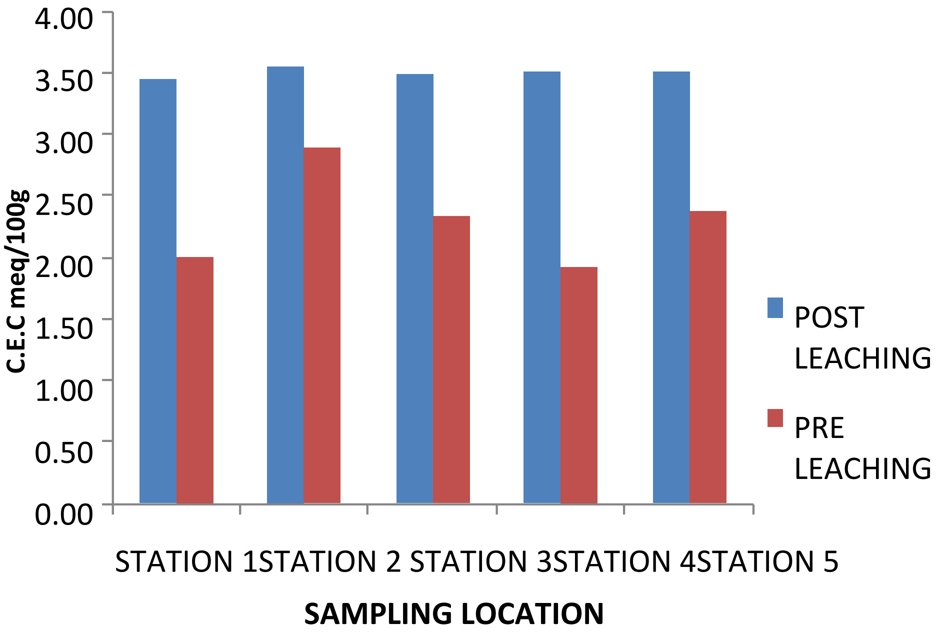


Figure 5.17. Cation variations at test sites

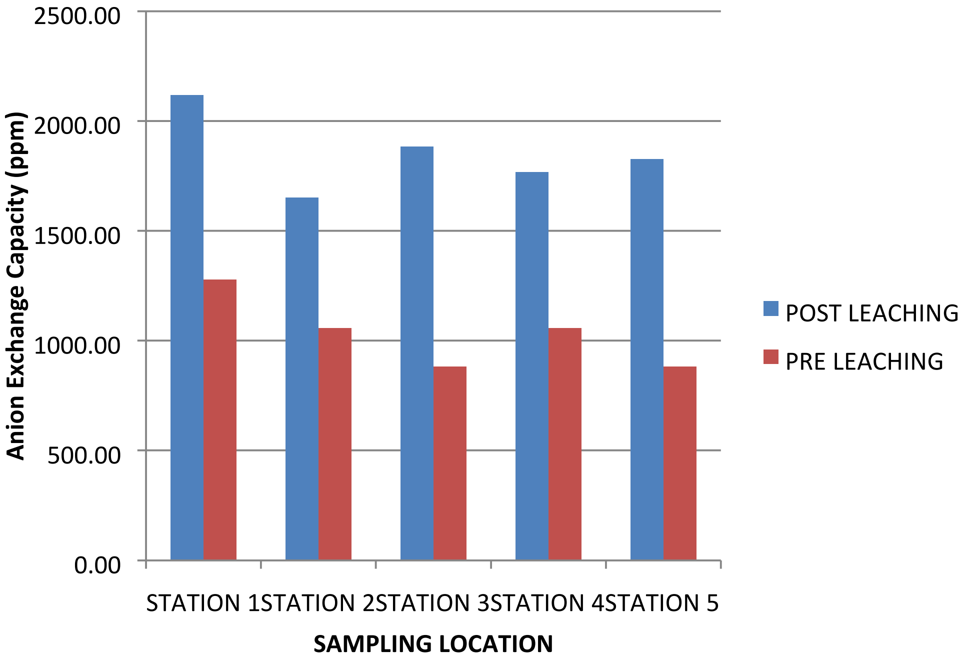


Figure 5.18. Anion exchange variations

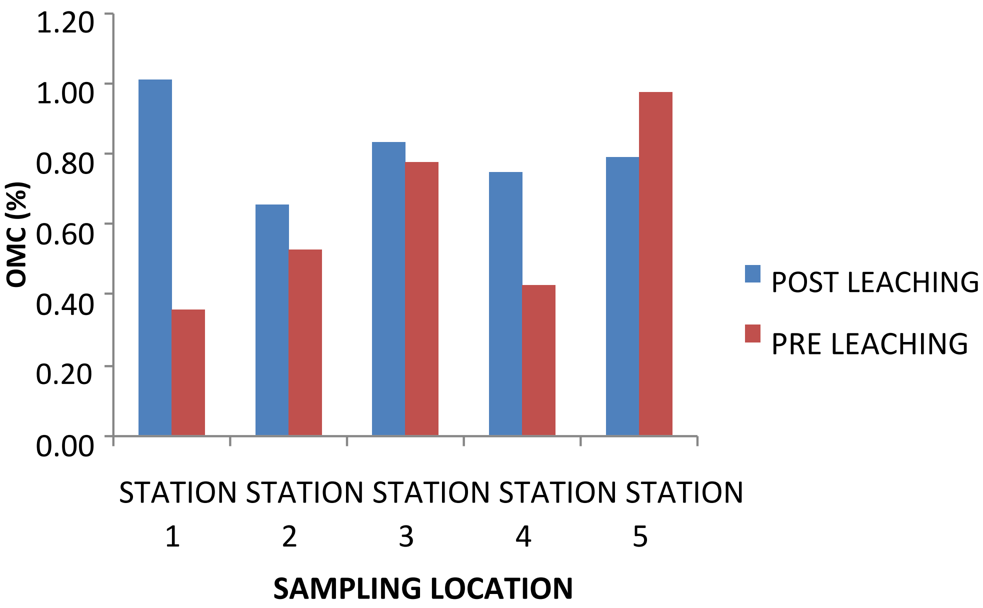


Figure 5.19. Organic matter content variation

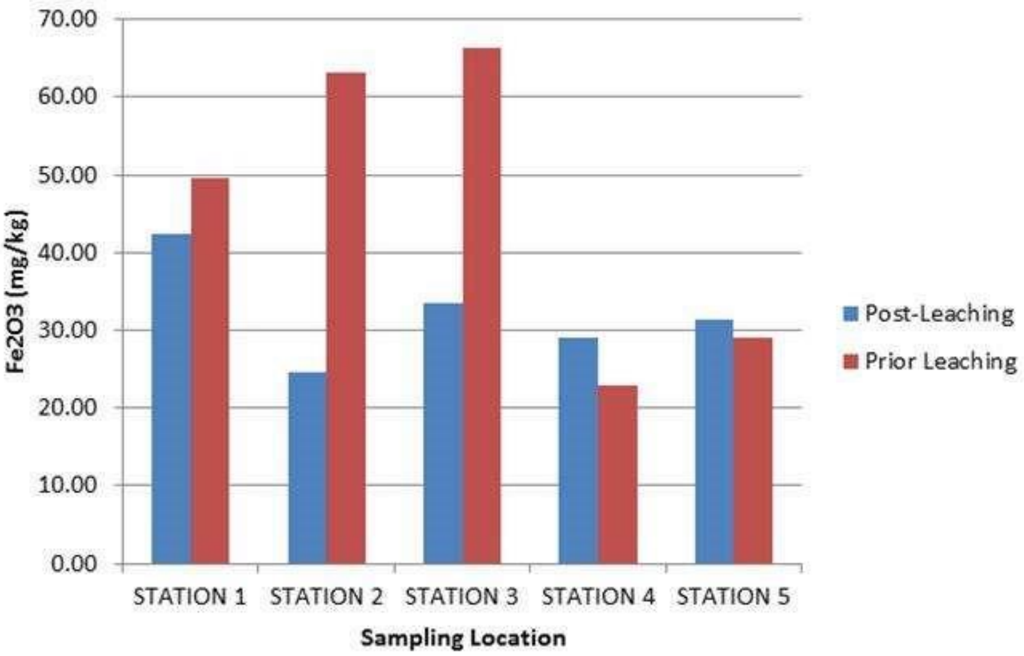


Figure 5.20. Ferrous oxide variations

The substrata material after the leaching test increased in the effectiveness of the attenuation potential of the clay material’s pH ranging between 3.0 and 3.4, AEC values up to 2117 meq/100 g. pH for all the test stations became more acidic after the leaching test. The higher CEC and OMC for Station 5 reduced from 0.98% to 0.79% rendering it less effective in attenuating cyanide and other minerals.

Table 5.10. Chemical properties after Leaching

Labels Horizon (cm)	pH 1:1 H_2O	Organic C %	Total N %	Organic Matter %	angeable meq/100g				T.E.B	Exch.A (Al+H)	E.C.E.C meq/100g	Base Sat. %
					Ca		K	Na				
Station 1	2.50	1.07	0.07	1.84	0.53	0.27	0.11	0.09	1.00	2.90	3.90	25.64
Station 1	3.20	0.37	0.04	0.64	0.53	0.53	0.24	0.13	1.43	1.80	3.23	44.27
Station 2	3.30	0.33	0.03	0.57	0.67	0.67	0.13	0.10	1.57	1.65	3.22	48.76
Station 2	3.30	0.37	0.04	0.64	0.53	0.27	1.00	0.08	1.88	1.70	3.58	52.51
Station 3	3.50	0.39	0.04	0.68	0.80	0.53	0.36	0.21	1.90	1.60	3.50	45.71
Station 3	3.60	0.35	0.04	0.61	0.80	0.53	4.01	0.29	5.63	1.65	7.28	77.34
Station 3	3.60	0.49	0.05	0.85	0.80	0.53	0.26	0.23	1.82	1.65	3.47	52.45
Station 4	3.40	0.47	0.05	0.82	0.80	0.53	1.13	0.14	2.60	1.70	4.30	60.47
Station 4	3.40	1.15	0.10	1.98	0.53	0.53	2.51	0.20	3.77	1.70	5.47	68.92
Station 5	3.60	0.41	0.04	0.71	0.53	0.27	0.13	0.15	1.08	1.65	2.73	39.56
Station 5	3.40	0.51	0.05	0.89	0.53	0.53	1.00	0.14	2.20	1.70	3.90	56.41

Table 5.11. Physical properties after leaching

Labels Horizon (cm)	Available – Bray's		nent / Dr (ppm)			
	P (ppm)	K (ppm)	Fe	MnO	Fe_2O_3	Mn
Station 1	1.04	23.42	10.35	0.38	29.6	0.30
Station 1	5.42	36.83	23.28	1.80	66.58	1.41
Station 2	1.12	23.43	10.83	0.50	30.97	0.39
Station 2	0.88	26.78	6.75	0.35	19.31	0.27
Station 3	1.20	26.78	10.50	1.15	30.03	0.90
Station 3	0.40	28.46	12.63	0.96	36.12	0.75
Station 3	2.71	33.48	16.2	0.81	46.33	0.63
Station 4	5.66	33.48	15.78	0.31	45.13	0.24
Station 4	6.94	53.57	15.87	0.31	45.39	0.24
Station 5	5.90	33.48	9.57	0.23	27.37	0.18
Station 5	15.71	50.22			71.39	

5.3 Groundwater chemistry

Generally, chemical parameters in water are essential for human health and metabolism. However, if secondary inorganic constituents occur in the water above their maximum allowable limit as indicated by the WHO guideline limit for potable water, they become detrimental to health of living organisms or impact their sensory organs and make the water objectionable to the consumer (Kortatsi, 2004; Kresic, 2007; WHO, 2008).

The first step of the water analysis was an assessment of the quality of the data (refer to Appendix 9 for the yearly geometric mean of groundwater monitoring data), which as accomplished by calculating the balance of positive and negative ions. Water fulfils the principle of electro neutrality and is therefore net charged. The level of error in the data was calculated using error of ionic balance in equation 5.1:

□cations □□anions
Error □ ________________□100 (5.1) □cations □□anions

An error of up to ±10% is considered tolerable in the study based on the shallow nature of some of the boreholes, whiles that of historical data available could not be established due to inadequate water monitoring data from 2004 to 2008. Approximately one third of the measured water samples in 2009 were within this range (refer to Appendix 10), 60% of 2010 samples, 80% in 2011 and 90% in 2012. The electroneutrality for historical data (2004 – 2008) could not be established due to inadequate anion analysis. However, the ionic balance for the boreholes in the TSF, North and South Heap leach basins improved from an initial±14% in 2009 to ±5 in 2012.

5.3.1 Major cations and anions

The main constituents of groundwater are positively and negatively charged ions, usually present at concentrations of mg/l. Major cations such as Na^+, K^+, Ca^{2+}, and Mg^{2+} and major anions such as HCO_3^-, CO_3^{2-}, SO_4^{2-} and Cl^- were plotted in a hydrochemical trilinear diagram, also known as a Piper diagram (Piper, 1944) and this is represented in Figures 5.21 to 5.24.

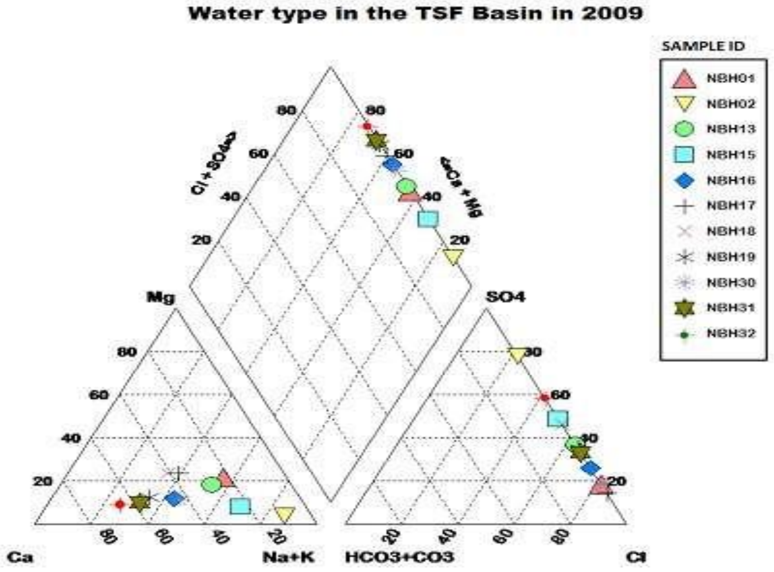


Figure 5.21. Water type in TSF basin in 2009

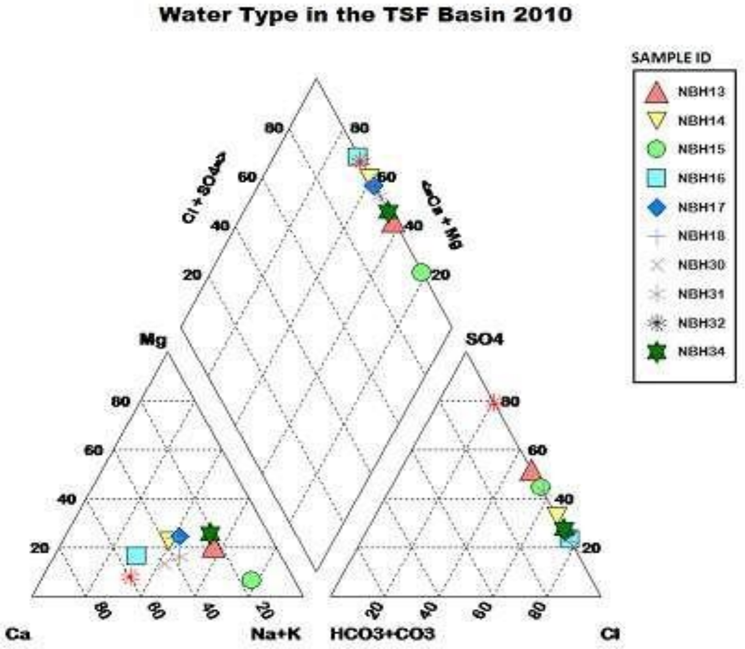


Figure 5.22. Water type in TSF basin in 2010

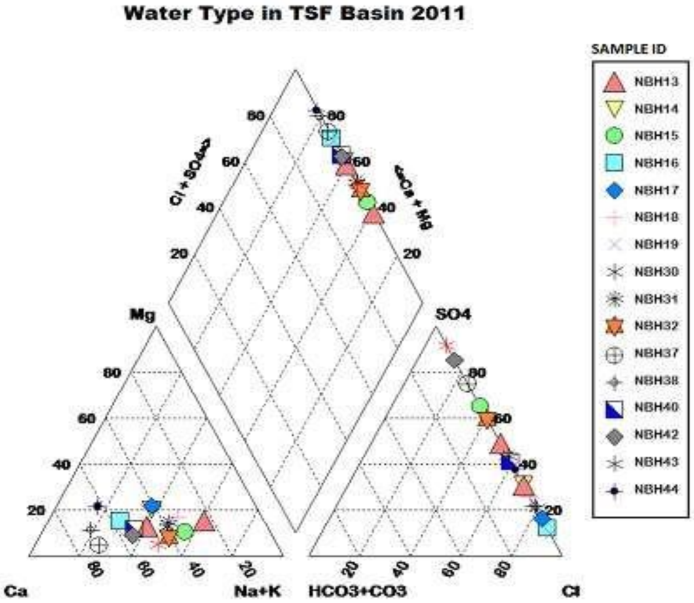


Figure 5.23. Water type in TSF basin in 2011

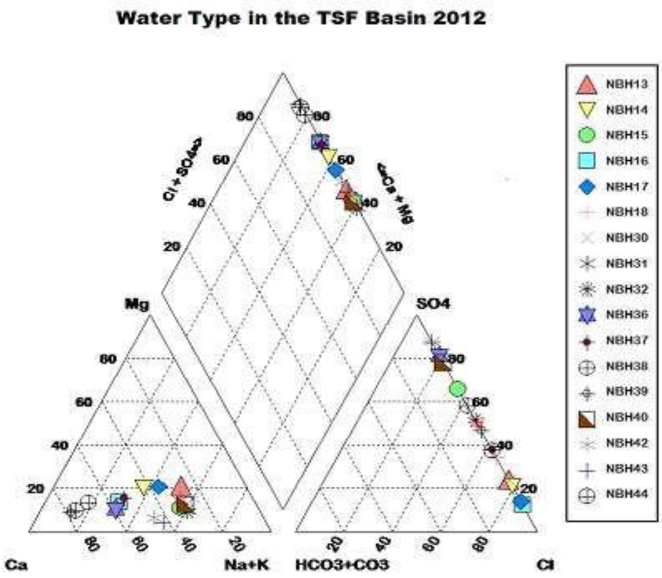


Figure 5.24. Water type in TSF basin in 2012

Note that the Piper diagram shows relative concentrations only, because the concentrations are shown on a percentage basis. The purpose of plotting the groundwater data on the piper diagram was to classify the water type, identify major cations and anions in the TSF basin and the level of contamination from 2004 to 2012. The ion compositions were plotted for all water samples in the TSF basin with an error of ion balance within the ±10 % range. Refer to Appendix 11 for explanation on the groundwater piper diagrams.

In Figure, 5.21, 5.22, 5.23, and 5.24 the triangle on the left side presents cation composition, while the triangle on the right side presents anion composition. The rectangular diagram shows both anions and cations. This plotting procedure assists in the classification of groundwater by its ion ratios.

Based on the combined chemistry of the major ions Na^+, K^+, Ca^{2+}, and Mg^2, HCO_3^-,

CO_3^{2-}, SO_4^{2-}, Cl^- and NO_3, the statistical trends for groundwater quality from January 2009 to November 2012 for boreholes within the TSF basin have been impacted greatly by the TSF operations. Hydrochemical facies in the TSF basin are Na, Na-Ca, Ca, and Ca-Na. Major anion facies include Cl and SO_4. Two controlled boreholes namely; NBH01 and NBH02 were located mid-way between the North heap leach Facilities and the two catchment communities of Abekoase and Samanhu. Na-K, Na-Ca and Na facies are the major cations whiles major anions are SO_4^{2-} and Cl. The area is a non-active mining area and the presence of high proportions of K, Ca, SO_4^{2-} and Cl can be an indication of agricultural influences on the groundwater, as waters with such characteristics are often found in agricultural lands that has experienced the usage of inorganic fertilizer. Appendix 12 is the piper plots for groundwater monitoring boreholes in the catchment communities.

Trace metals above the detection limit in the study area include Mn, Zn and Pb with maximum concentrations 0.90, 100 and 100 µg/l respectively. However, Cr, Ni, and Cd were also analysed, but their concentrations were lower than the limits of determination (0.001 mg/L). Pb and Zn anomalies could be contamination from samples preparation or at the laboratory.

5.4 Primary Inorganic Constituents

Primary inorganic constituents with significant concentrations in the TSF basin are Na, K, Ca, Mg, HCO_3, SO_4, Cl and free cyanide, as shown in Table 5.12. Concentrations are within WHO guideline limits.

Table 5.12 Statistical summary of chemical parameters

Parameter	Unit	WHO MCL	Min	Max	Mean	STD	Number of Exceedences
TDS	mg/l	1000	34	496	183	108.5	0
El. Cond.	uS/cm	1500	55	804.8	296.1	175.9	0
Temp	°C	N/A	24.2	25.2	24.7	0.27	0
DO	mg/l	N/A	4.0	5.4 **c**	4.7	0.31	0
		Primary Inorgani Constituents					
pH		6.5-8.5	5.7	8.0	6.68	0.562	0
Ca^+	mg/l	100	2.10	83.20	24.30	22.46	0
Mg^{2+}	mg/l	30	0.70	8.70	3.46	2.17	0
Na^+	mg/l	200	4.20	89.20	18.92	17.7	0
K^+	mg/l	250	0.600	20.200	2.265	3.575	0
Cl^-	mg/l	250	3.00	17.10	7.76	4.10	0
Fe^{2+}	mg/l	0.3	<0.01	0.2	0.0935	0.0512	0
HCO_3^-	mg/l	N/A	47.7	284.4	143.9	68.9	0
SO_{42-}	mg/l	N/A	1.1	197.5	19.4	39	0
NO_{2-}	mg/l	250	<0.01	0.100	0.019	0.040	0
NO_{3-}	mg/l	50	2.00	25.00	5.23	4.69	0
NH_4	mg/l	N/A	<0.01	0.7	0.1226	0.1383	0
		Secondary Inorganic Constituents					
Mn_diss	mg/l	0.40	<0.01	0.90	0.34	0.25	12
Cr_diss	ug/l	50	N/A	N/A	N/A	N/A	0
Pb_diss	ug/l	10	<0.0010	100	32.26	47.5	10
Zn	ug/l	5.0	<0.001	100	51.6	50.8	0

5.4.1 Electrical conductivity

Electrical conductivity (EC) values of the water samples from the TSF basin varied between 121 and 824 µS/cm with mean value of 359 µS/cm. These values are compared to historical values from 2004 to 2008 that ranged between 40 and 560 µS/cm with a mean of 310 µS/cm. Water samples from boreholes from which high EC values were recorded are NBH 13, 14, 15, 16, 17, 28, 35, 36, 39, 43 and 44. High EC levels were observed in the boreholes in the South Heap leach basin with particular emphasis on SBH01 located close to the pregnant pond which recorded the highest value of 398 µS/cm (mean value of 359 µS/cm) as compared to the historical mean value of 200 µS/cm. Approximately, the mean values for the North Heap leach facilities are the same as the South Heap leach and TSF basins. High concentrations of EC values were recorded at sample locations NBH06, NBH20 and NBH21. The majority of the samples had values below the WHO and Ghana Environmental Protection Agency (EPA) sector guideline value of 1500 µS/cm, but high when compared to the control point levels. The EC concentrations of various regimes were plotted on the box chart (Figure 5.25) for various regimes and facilities. The spatial distribution of the EC values is presented in Figure 5.26 and 5.27.

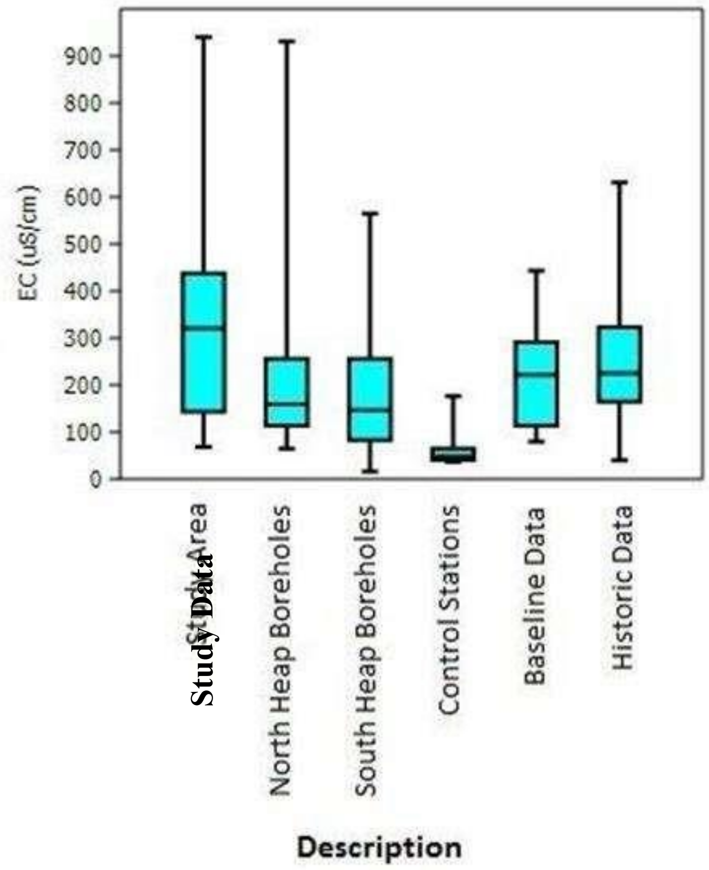


Figure 5.25. EC plot for study data

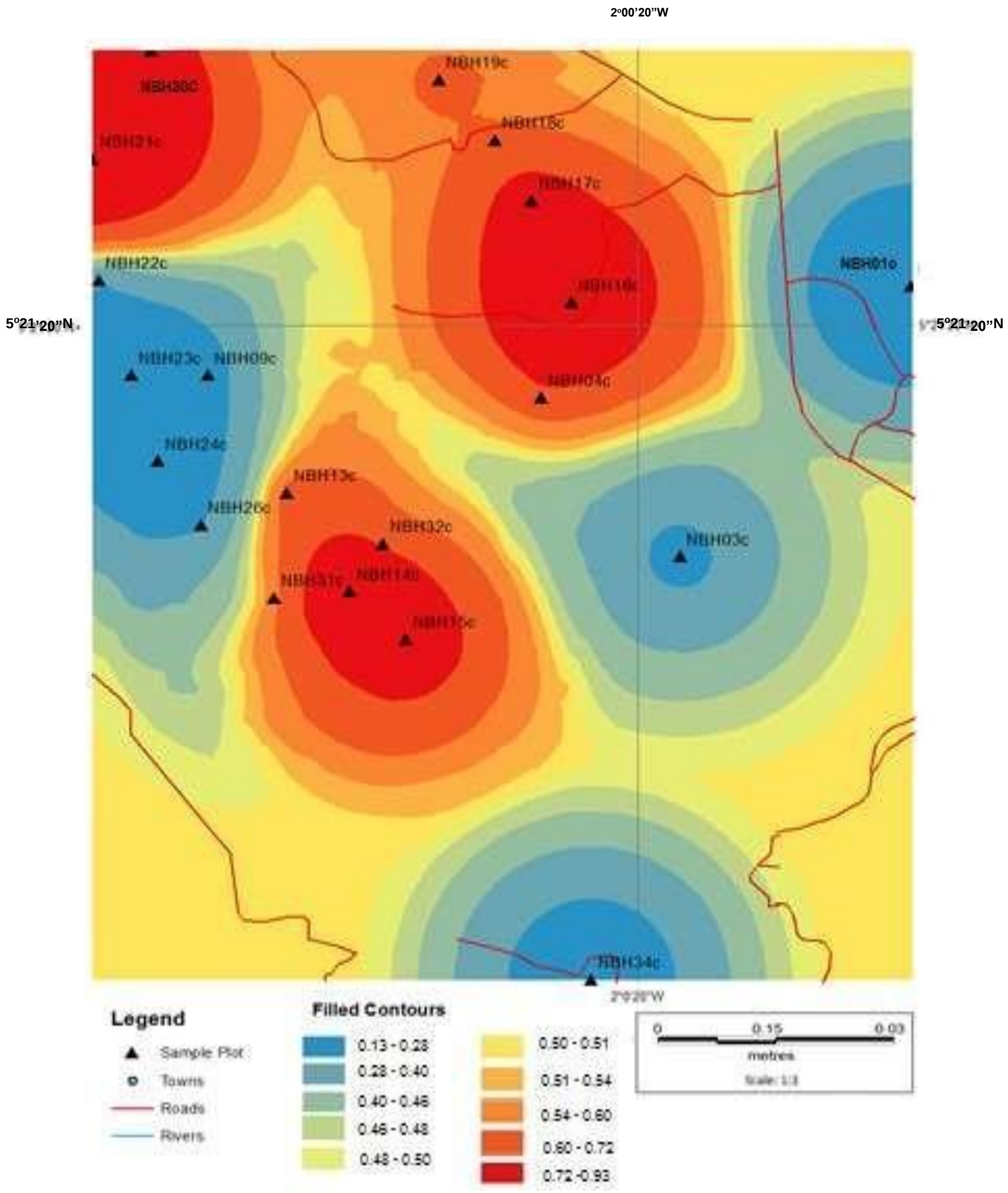


Figure 5.26. EC spatial plot in 2011

2°00'20"W

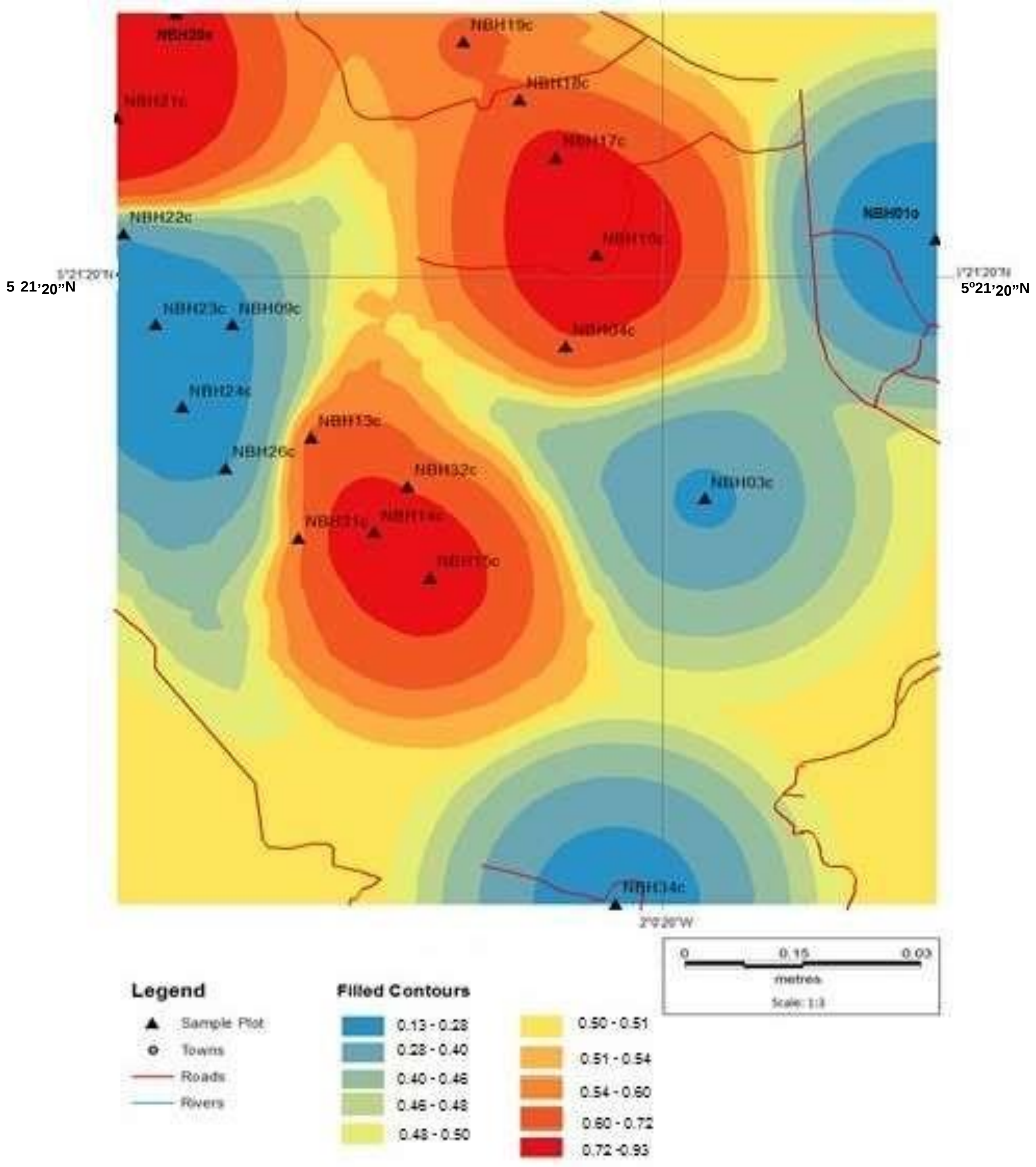


Figure 5.27. EC spatial plot in 2012

5.4.2 pH

The groundwater in the TSF basin has a pH ranging from 5.8–11.8 with a mean value of 6.5 from 2004 to 2008. On the average, this pH falls within the natural groundwater pH range of 5-8 (Buamah *et al.*, 2008) and that of potable water of 6.5-8.5 (WHO, 2008). The study period from 2009 to 2012 recorded pH values ranging from 5.9 to 8.5 and a mean value of 6.9. There is a significant pH increase in the four year average as compared to the historical data from 2004 to 2008 (Figure 5.28) and this can be attributed to seepage of tailings from the TSF into the basin. Water samples from monitoring boreholes; NBH14, NBH15, NBH16, NBH17, NBH28, NBH36, NBH38,

NBH39, NBH42, NBH43 and NBH44 returned high pH values from 2009 to 2012.

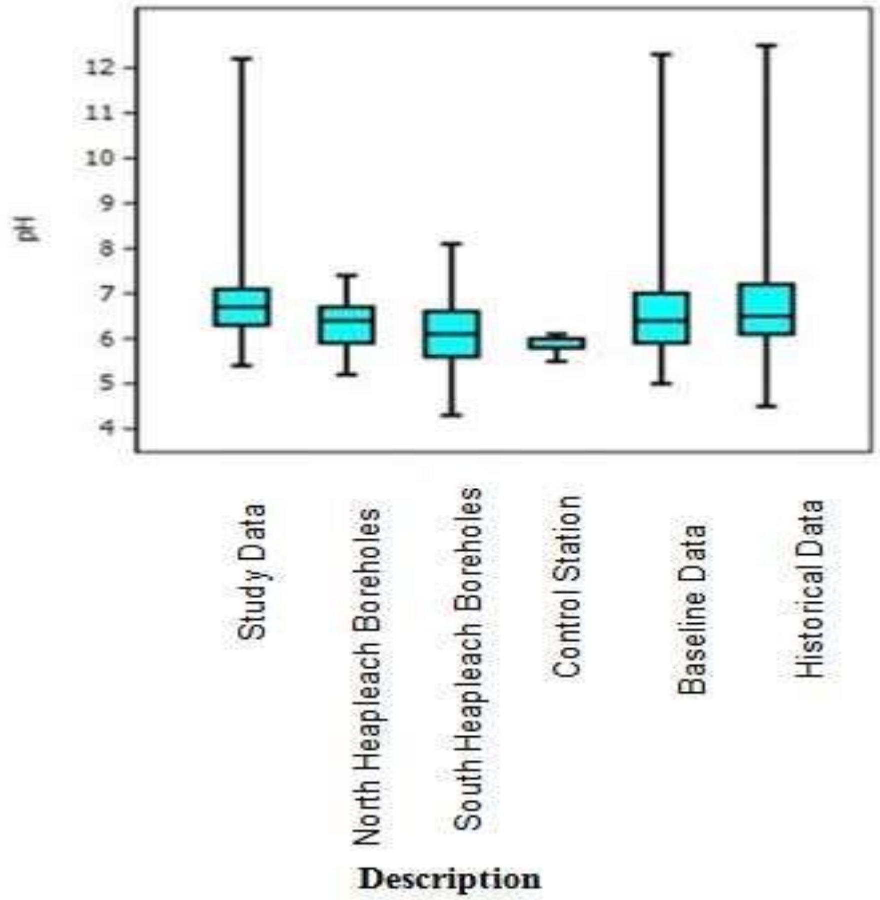


Figure 5.28. pH variations

The pH of water samples from boreholes SBH01 near the pregnant pond at the South heap leach and NBH20 near Phase 5 North heap leach and close to pregnant pond were elevated probably due to seepage from leaching processes. The HDPE liners of the South and North pregnant ponds were replaced in 2010 due to rapturing or from cracking and probably the source point impact on the pH of the surrounding groundwater. The spatial distribution map is represented in Figure 5.29 and 5.30 respectively.

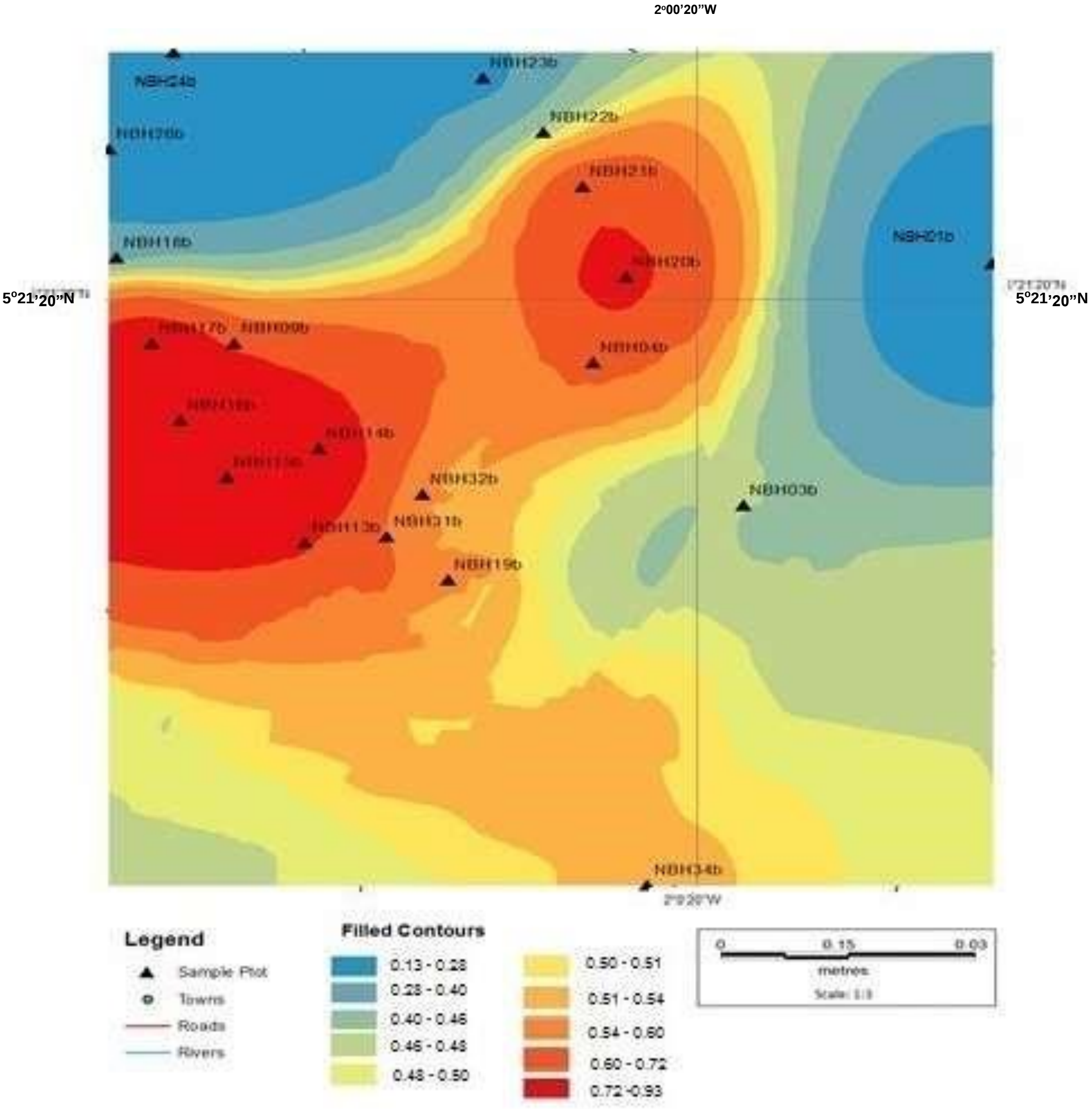


Figure 5.29. pH spatial plot in 2011

2°00'20"W

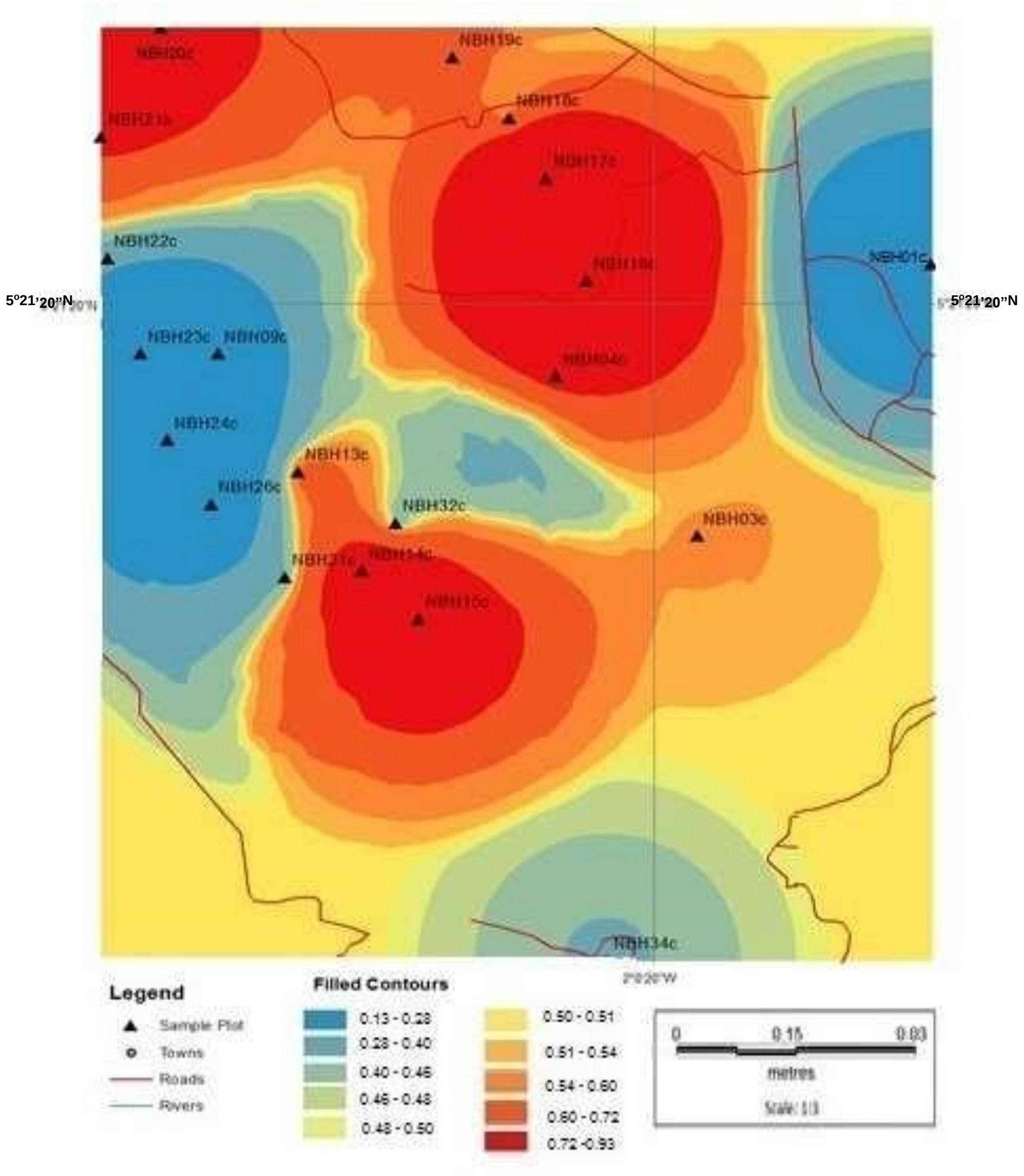


Figure 5.30. pH spatial plot in 2012

5.4.3 Sodium

Sodium in the TSF basin range between a minimum value of 5.8 mg/l and 94.7 mg/l with a mean value of 29 mg/l. Monitoring boreholes with high Na concentration are NBH44, NBH13, NBH14 and NBH28. Mean Na concentration increased from the historical value of 23 mg/l to 29 mg/l during the study period. Two boreholes around the North Heap leach basin, namely NBH06 and NBH21 that are also located near the process water containment and the pregnant pond areas respectively, recorded high Na concentration of 145 mg/l and 91 mg/l. Graphical presentation of Na concentrations in the TSF basin and other facilities for the study period (2009 – 2012) and historical data is shown in Figure 5.31.

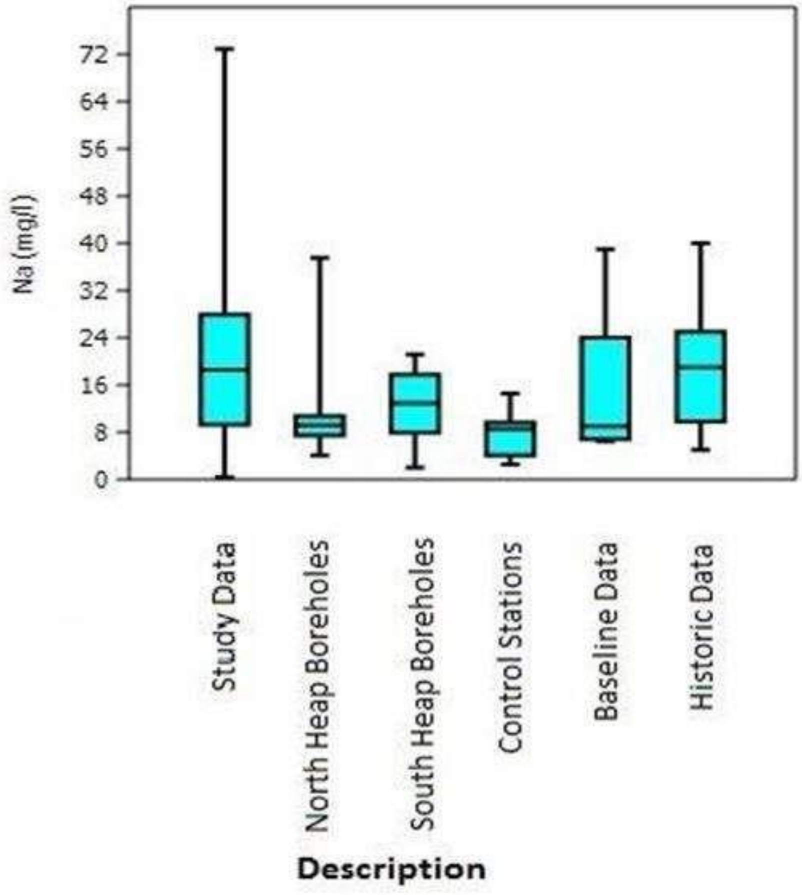


Figure 5.31. Na variations

The elevated Na concentrations could suggest seepage of cyanide solution into the groundwater basin around these facilities. Spatial concentration of sodium in the study area is shown in Figure 5.32 and 5.33.

2°00'20"W

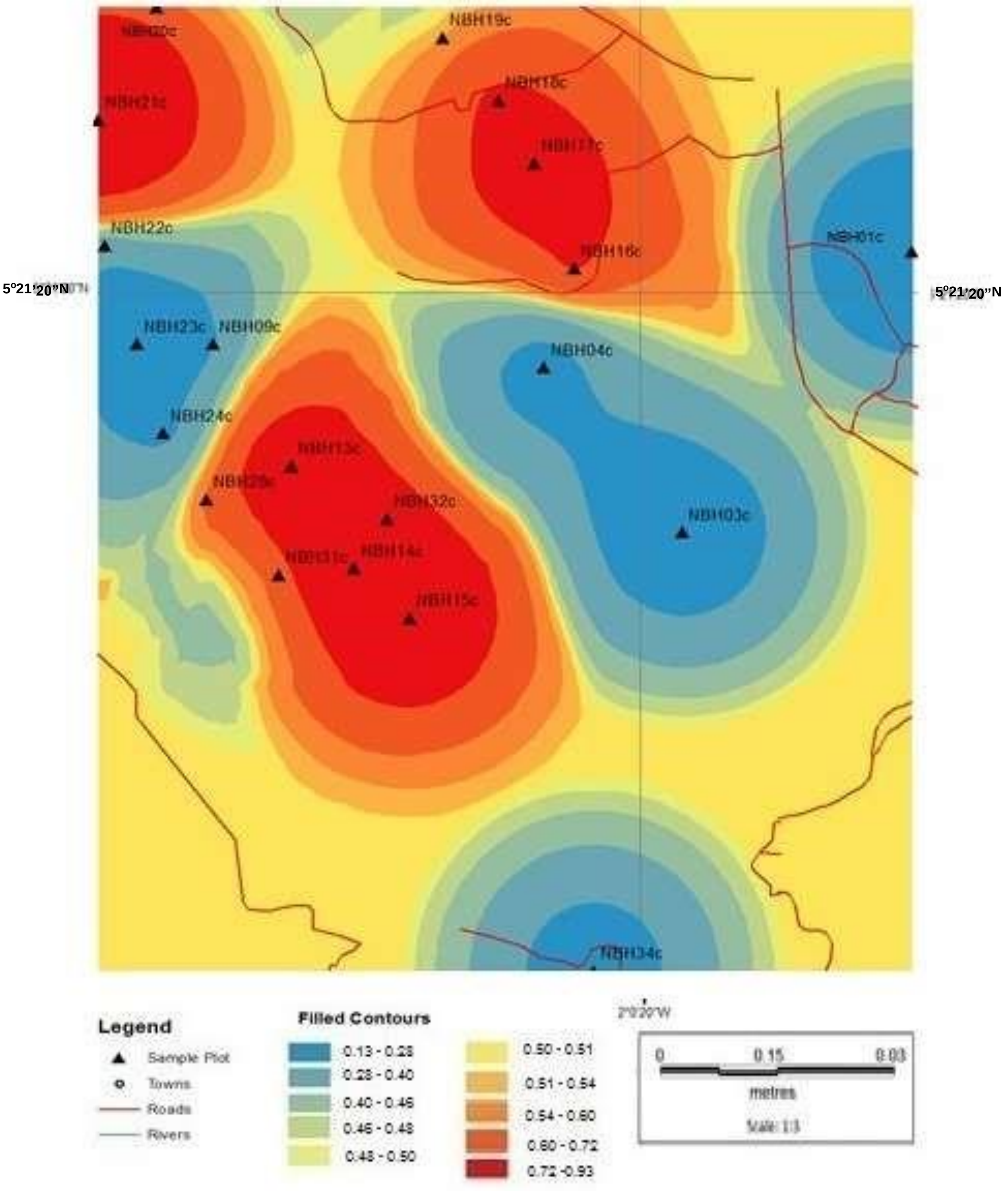


Figure 5.32. Na spatial plot in 2011

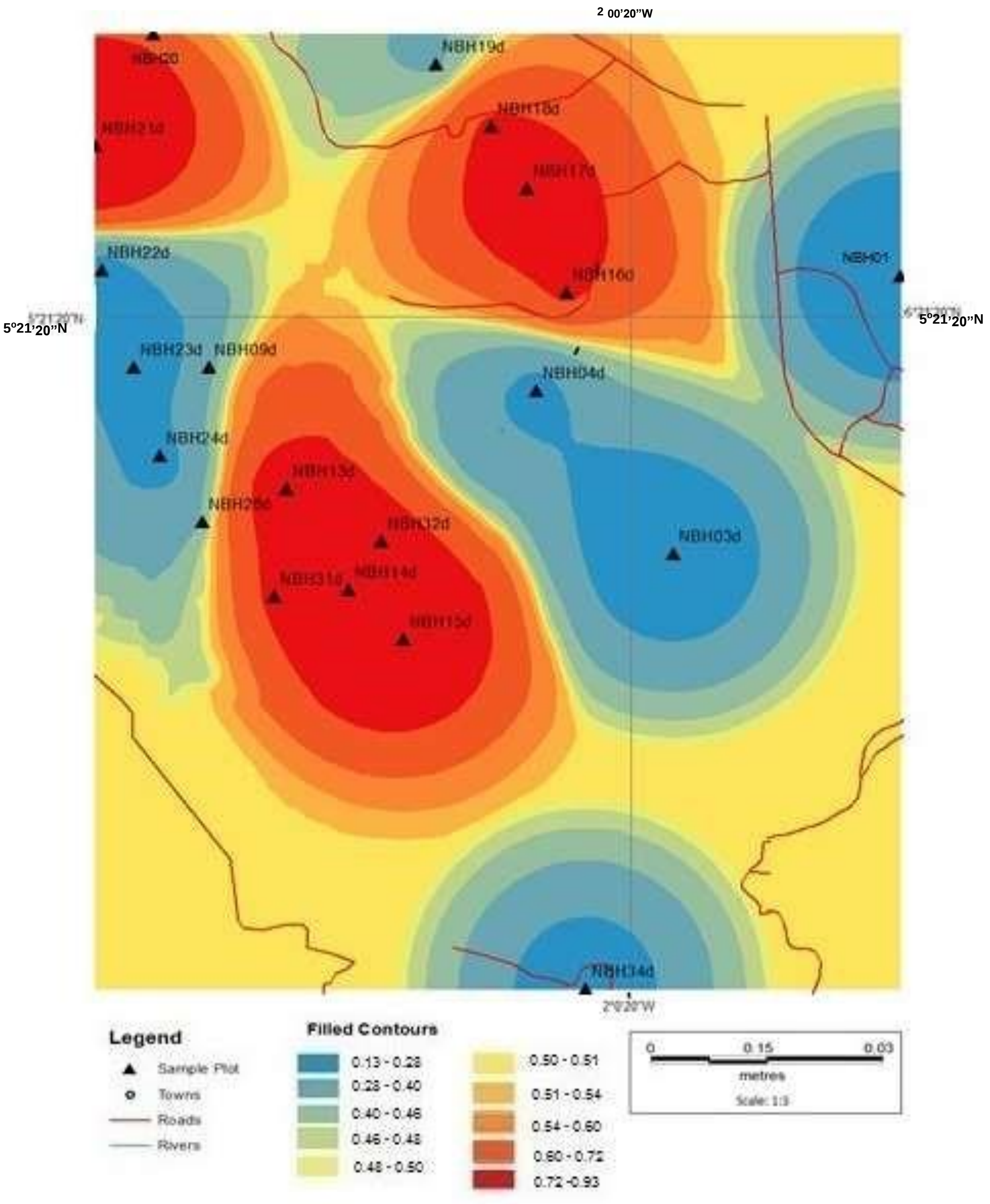


Figure 5.33. Na spatial plot in 2012

5.4.4 Calcium

The mean value of Ca for the four year period (2009 to 2012) returned a concentration of 29.7 mg/l with minimum value of 3.68 mg/l and maximum of 107.61 mg/l in the TSF basin. The highest concentrations were recorded at NBH035 located at the toe of TSF3 (north-west section of the basin).

Historical Ca concentrations in the TSF basin range between 6.3 mg/l and 28.1 mg/l with a mean value of 17.5 mg/l. Mean Ca concentration at NBH35 in 2010 returned 107.6 mg/l which exceeded the WHO guideline limit of 100 mg/l. Elevated average concentrations were also recorded at NBH14, NBH16, NBH38, NBH39, NBH42 and NBH44. Although these average values are within WHO guideline limits, there was an increase in concentration during the study and historical period. as illustrated in Figure 5.34. Spatial distribution of calcium in the study area is represented in Figure 5.35 and

5.36.

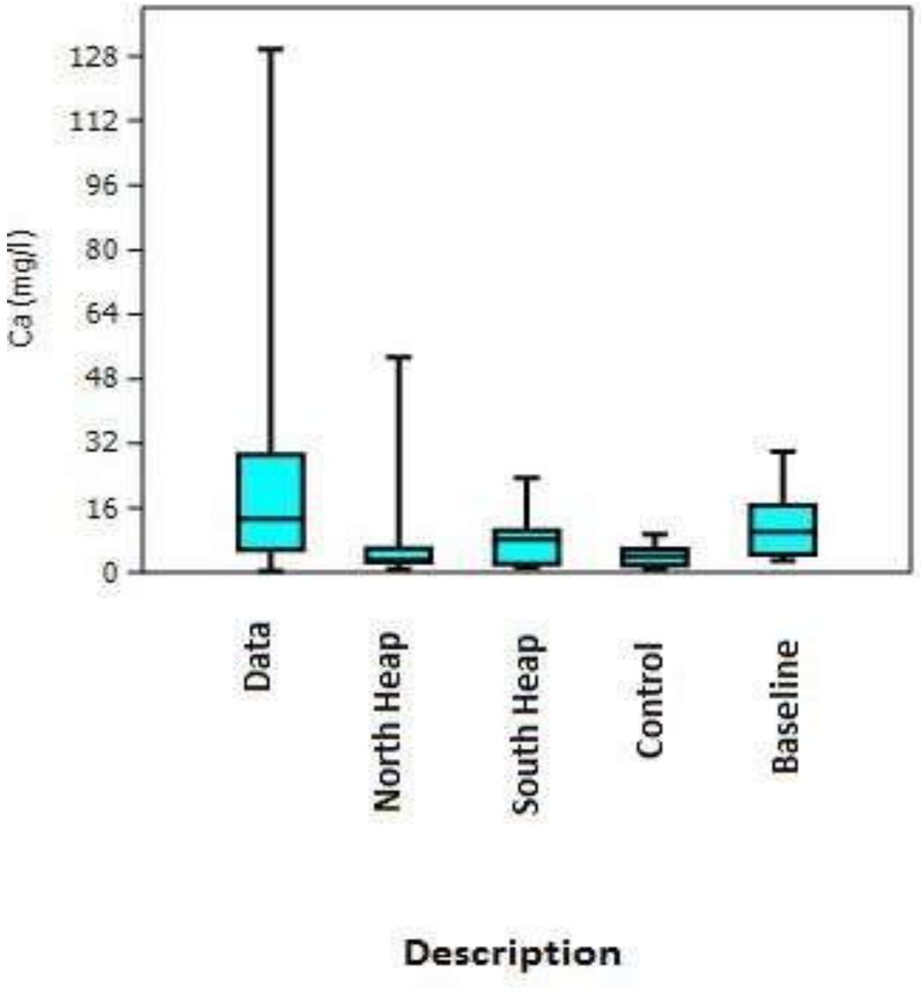


Figure 5.34. Calcium variations

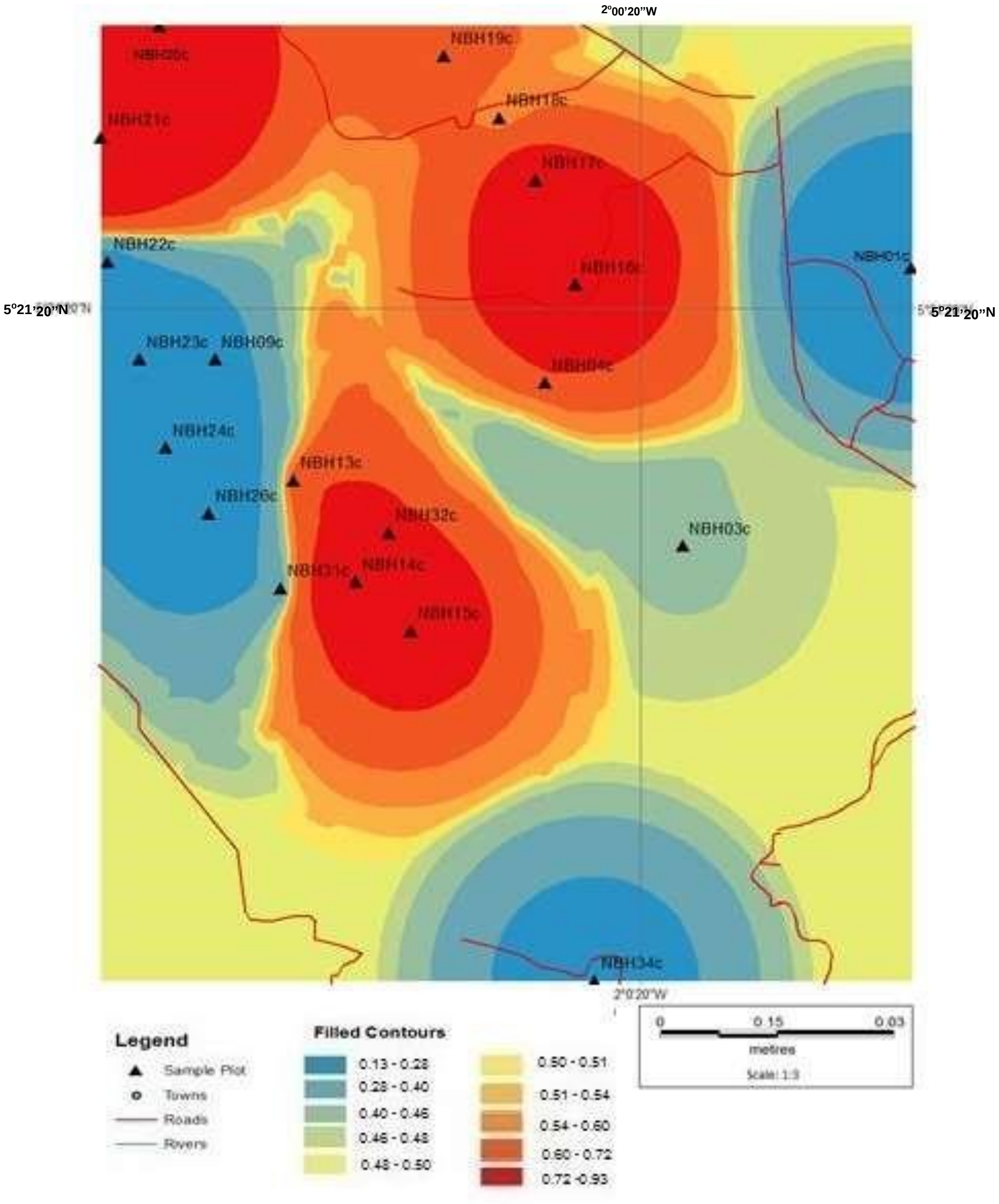


Figure 5.35. Ca spatial plot in 2011

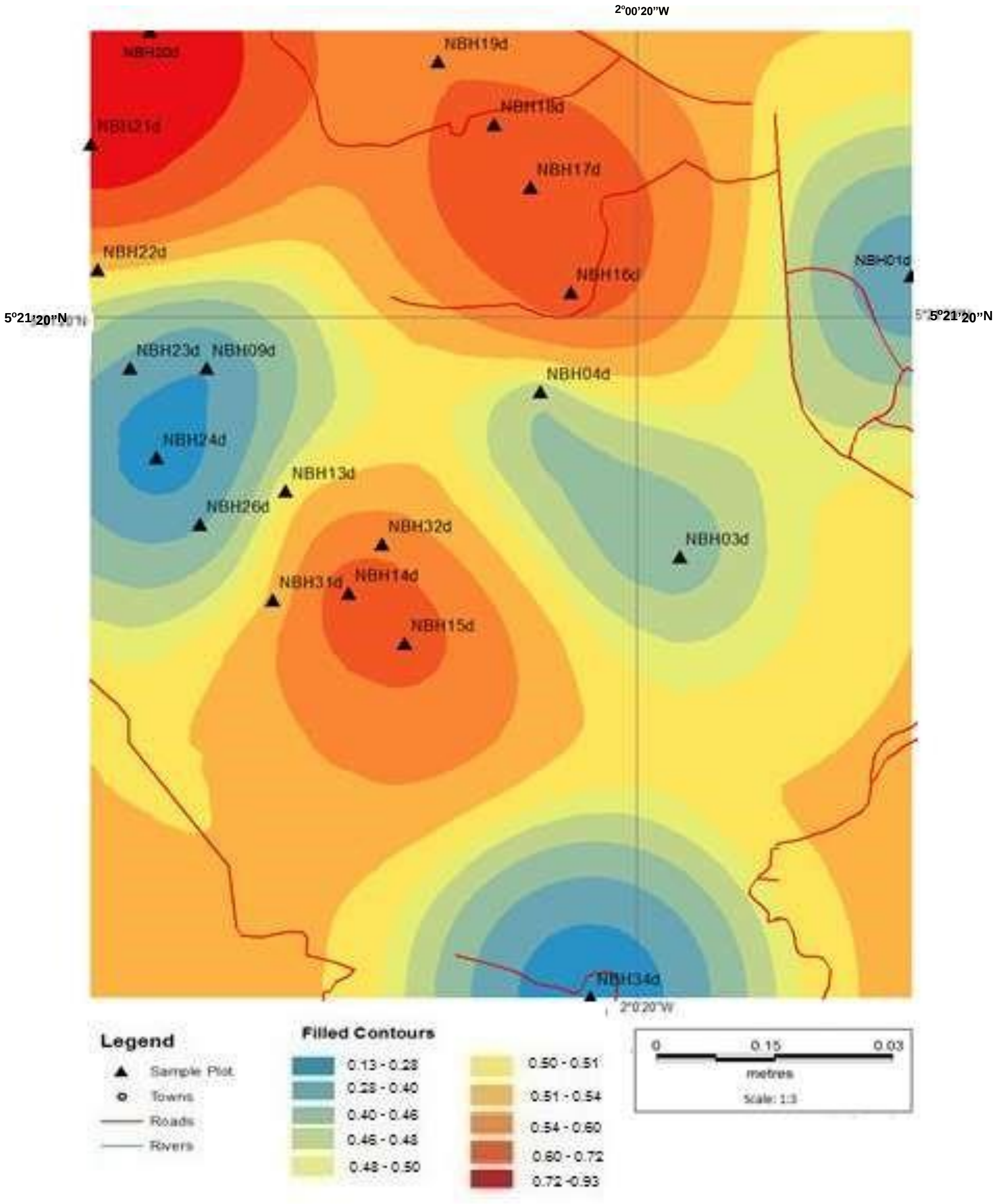


Figure 5.36. Ca spatial plot in 2012

Relatively high Ca concentrations in boreholes located at the TSF basin could be attributed to calcium bicarbonates in the geological formation and lime used in the dozing process in managing pH levels in the tailings slurry. Ca concentrations in 2004 and 2005 ranged between 41 mg/l and 205 mg/l. Monitoring boreholes NBH20 and NBH21 in the North Heap leach basin exhibited elevated Ca concentrations of up to 60.3 mg/l compared to maximum value of 19.5 mg/l in the historical regime. Elevated concentrations were recorded at SBH01 in 2011 and 2012, located in the South Heap leach basin. Boreholes with elevated calcium levels also exhibited high sulphate concentration suggesting possible seepage of the process solution.

5.4.6 Chloride

There was not enough data on Cl concentration during the period 2004 to 2008 to make any meaningful analysis. However, consistent Cl analysis commenced during the research data collection (2009 – 2012). Yearly average Cl concentrations range between 2.45 mg/l to 30 mg/l with mean value of 9.30 mg/l. Although all the samples have chloride concentrations well below the WHO guideline limit of 250 mg/l, there was significant increase in its concentration in boreholes in the TSF basin.

Monitoring boreholes with higher concentrations of Cl compared to the mean of the yearly averages are NBH13, NBH14, NBH15, NBH38, NBH42 and NBH43. Low Cl concentrations were recorded in the North and South Heap leach basins.

The mean Cl values during the study period is far lower than that of the mean for the historical, baseline, control and heap leach monitoring data (Figure 5.37). The low Cl values could be attributed to formation of Na complexes. The spatial Cl concentration is represented in Figure 5.38and 5.39.

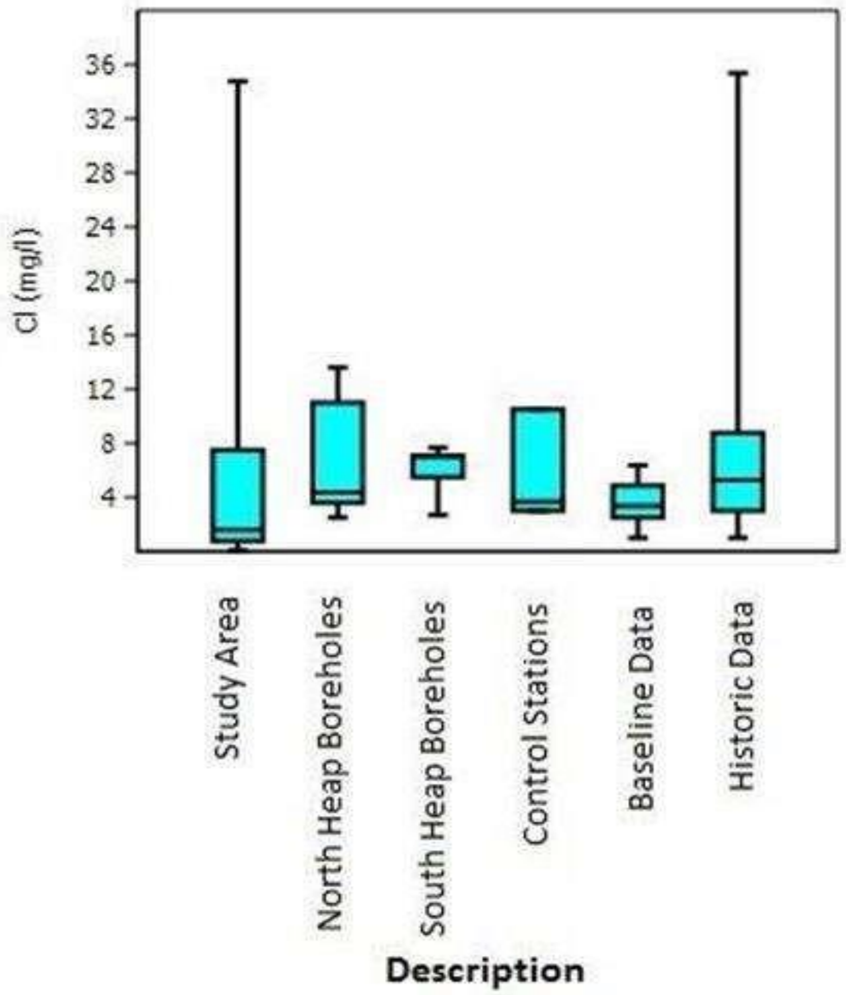


Figure 5.37. Cl variations

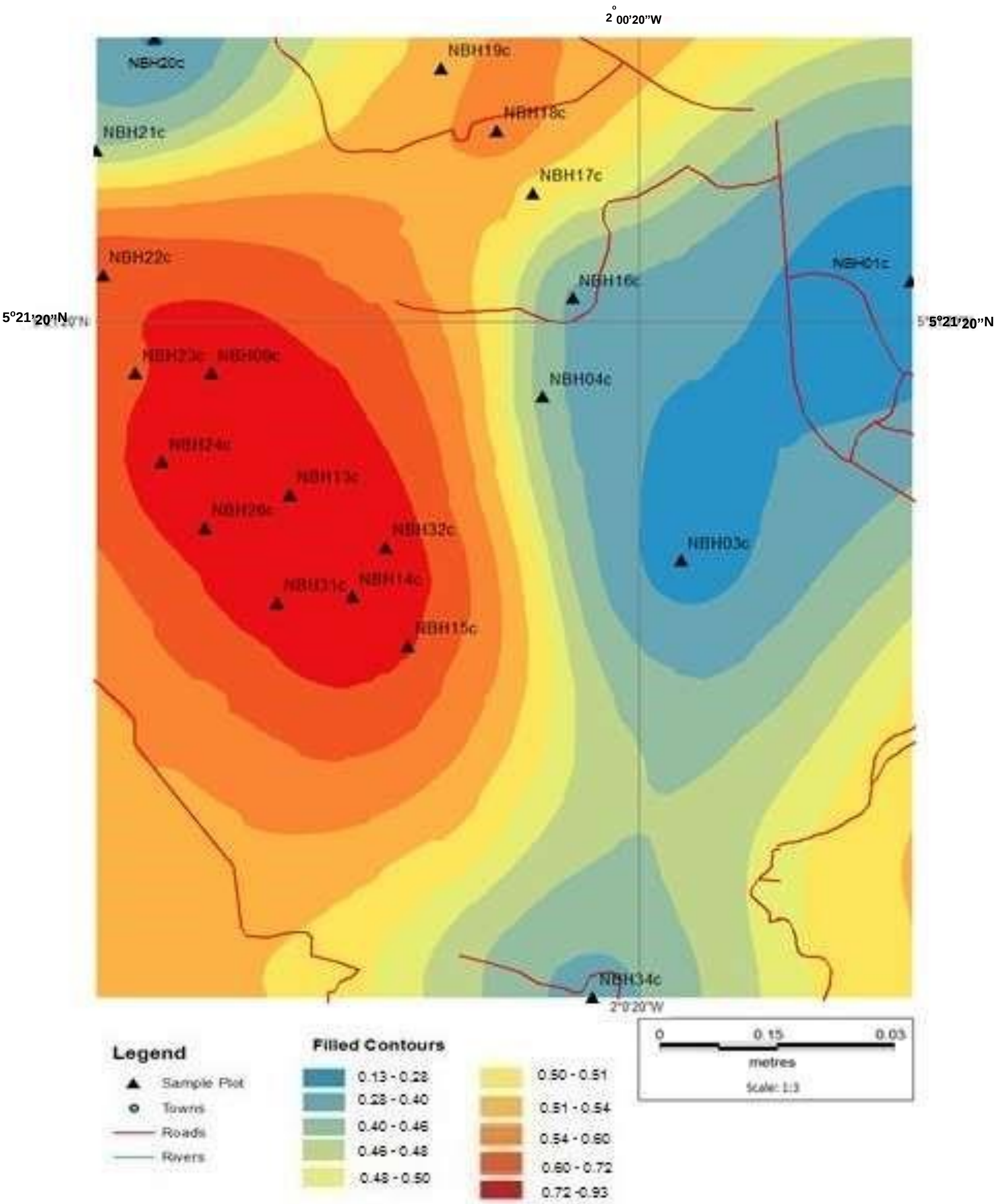


Figure 5.38. Cl spatial plot in 2011

2°00'20"W

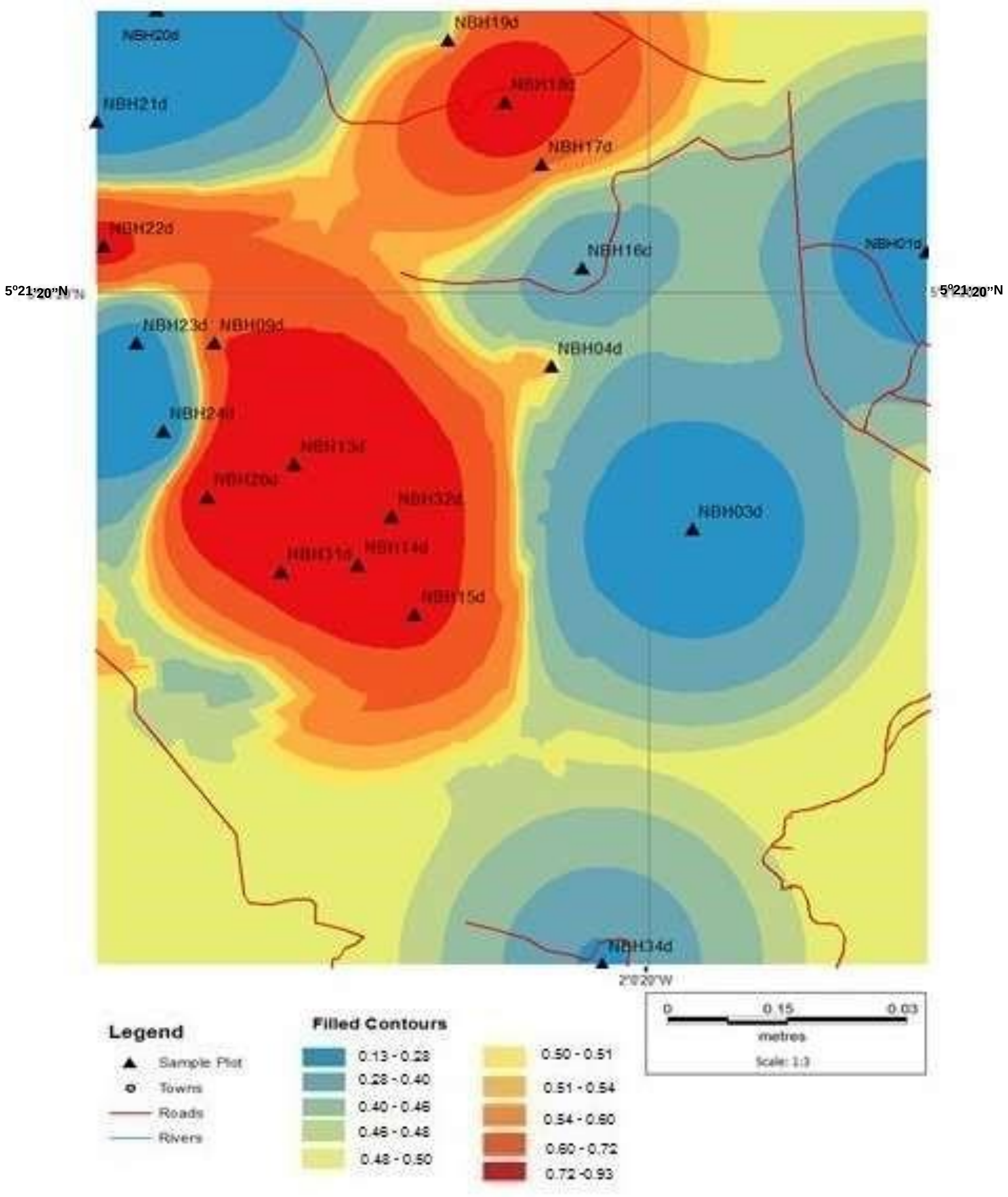


Figure 5.39. Cl spatial plot in 2012

5.4.7 Sulphate

Sulphate concentrations in the study area are low (<25 mg/l including the North and South Heap leach basins. Yearly averages range from a minimum concentration of 1.0 mg/l to a maximum value of 220.5 mg/l with yearly mean concentration of 22.30 mg/l. The highest SO_4 concentrations of 600 mg/l and 150 mg/l were recorded at boreholes NBH43 and NBH42 respectively in March 2011 and April 2012. Trends in monthly and yearly average concentrations at NBH42 are high as compared to the general trend in the TSF basin. High SO_4 concentrations could suggest possible $CaSO_4$ and H_2SO_4 used in the dosing process. Secondarily, the high pH coupled with high electrical conductivity and calcium concentrations could suggest seepage from process water (refer to Figure 5.40 and 5.41).

2°00’20”W

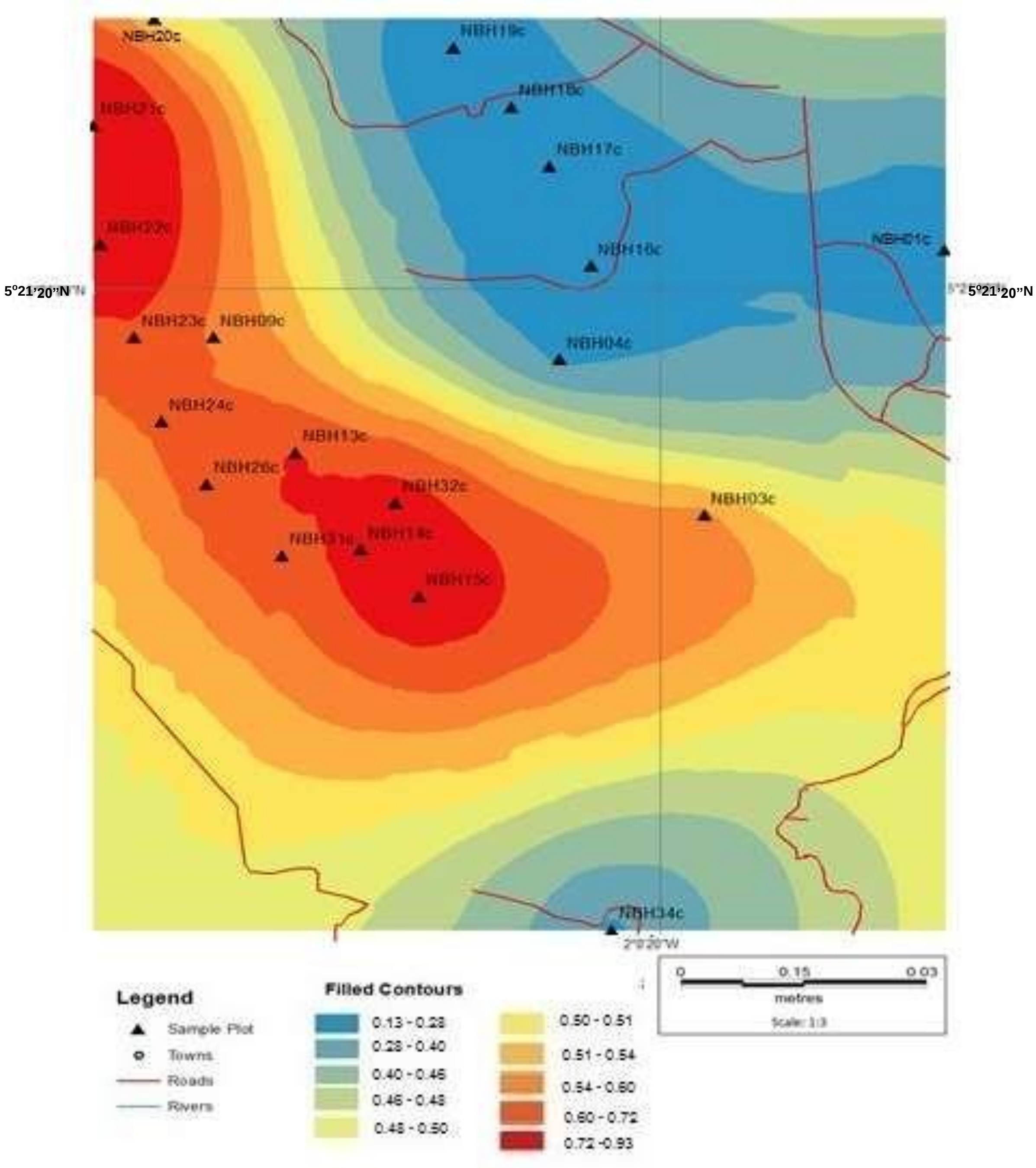


Figure 5.40. Sulphate spatial plot in 2011

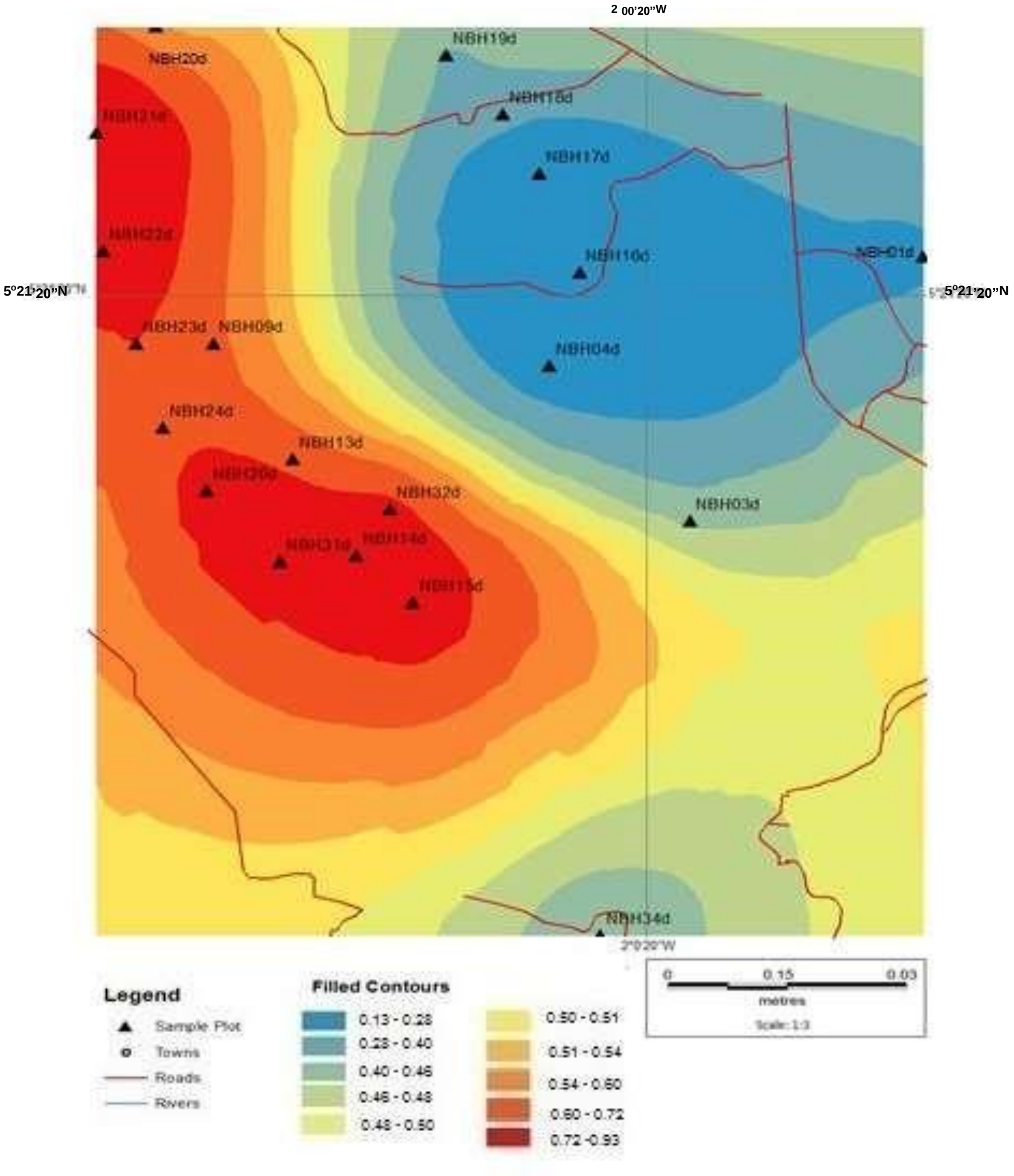


Figure 5.41. Sulphate spatial plot in 2012

5.5 Secondary inorganic constituents

The significant secondary inorganic constituents in the study area are Mn, Cd and Pb. There were a total of 10 exceedances for Pb concentration, twelve for Mn and eightyfour exceedances for Cd. The source of Pb concentration in the water samples in the study area could be established due to the absence of galena in the ore and waste geochemistry. Contamination during sample preparation and at the laboratory could account for the high Pb concentrations.

Mg concentrations ranged from <0.001 (detection limit) to a maximum value of 1.528 mg/l in the TSF basin with a mean value of 0.376 mg/l. There was no significant increase in the Mn concentration compared to its historical data available. The spatial distribution of Mn in the study area is represented in Figure 5.42 and 5.43.

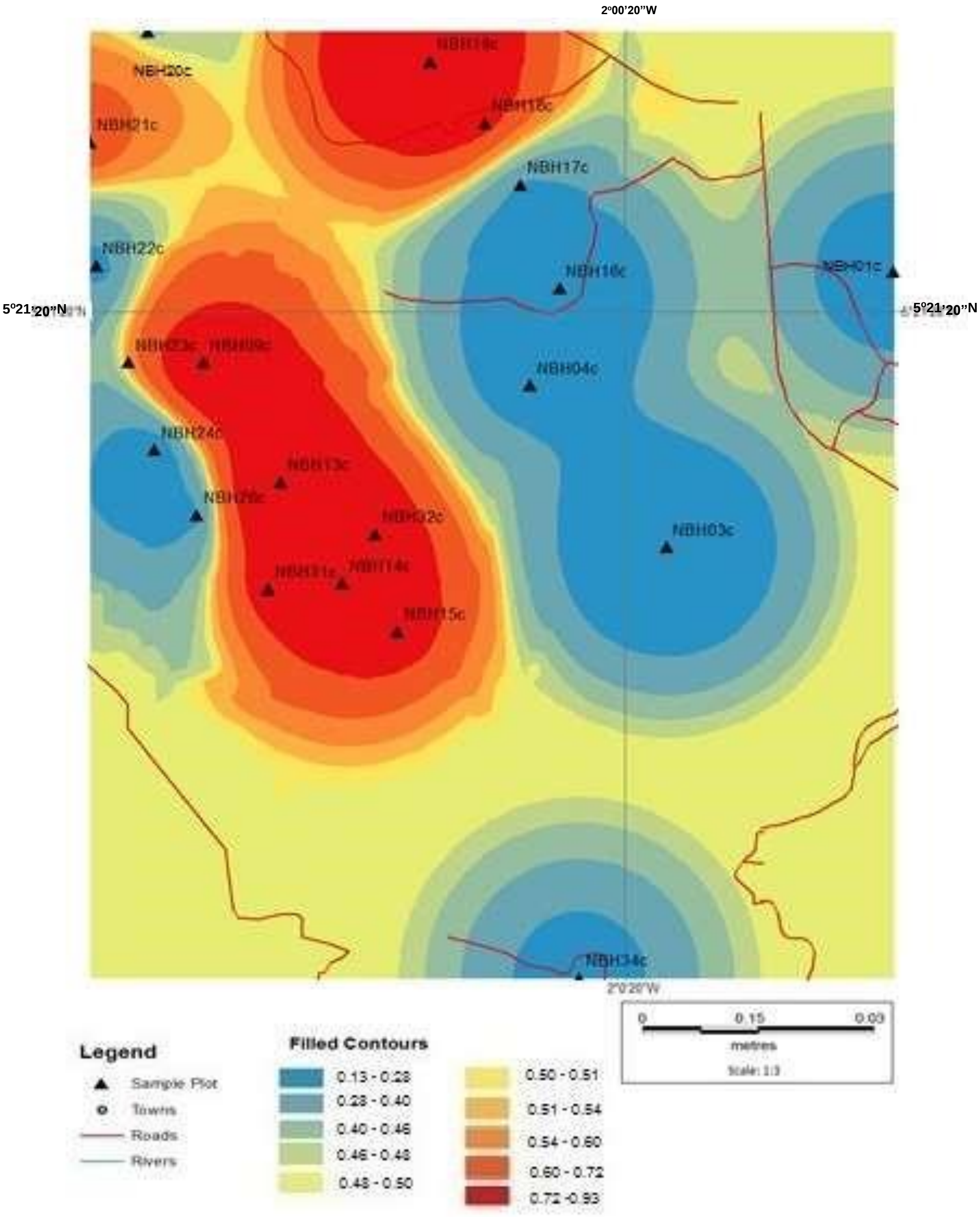


Figure 5.42. Mn spatial plot in 2011

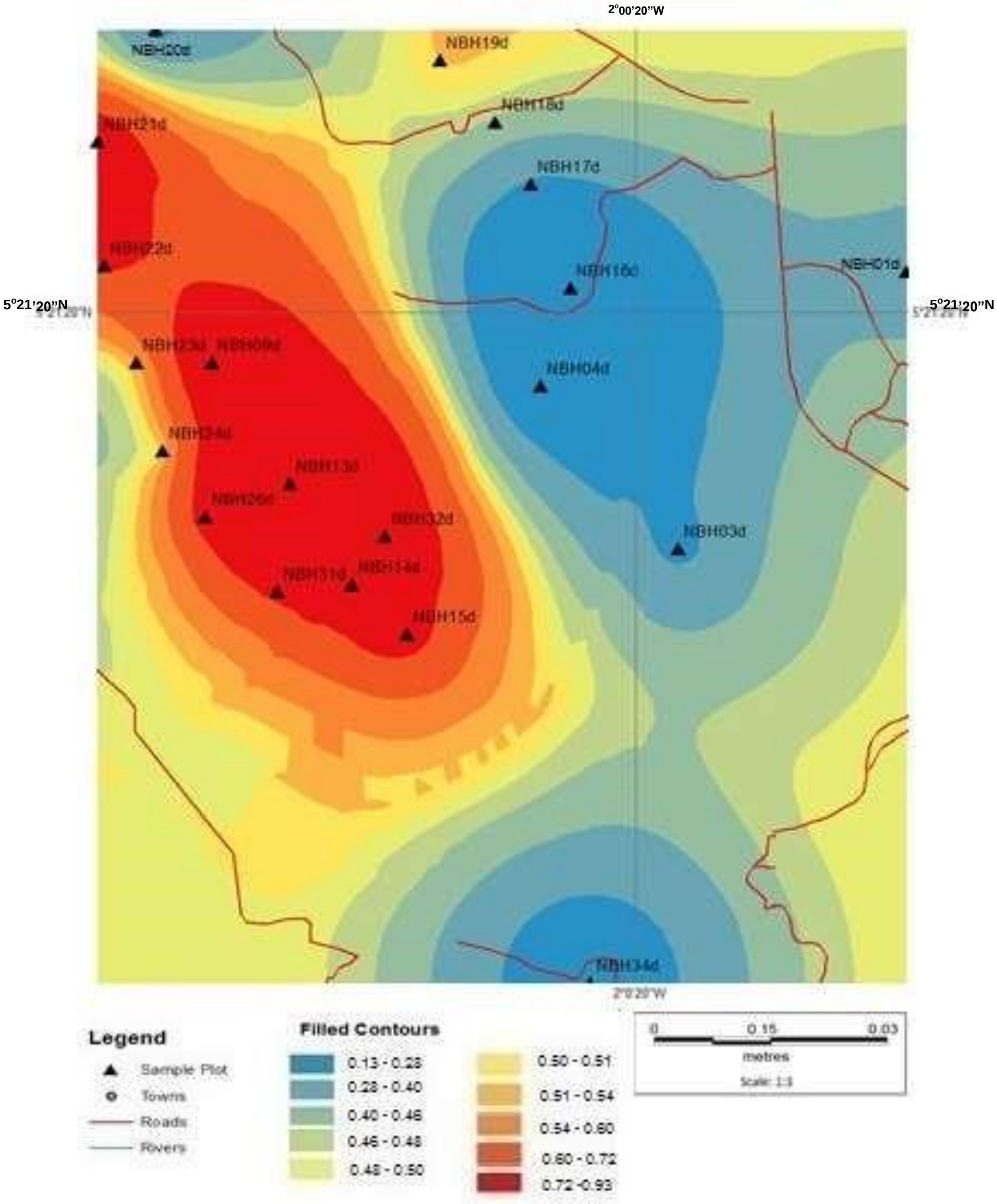


Figure 5.43. Mn Spatial Plot in 2012

CHAPTER 6: CONCLUSION AND RECOMMENDATIONS

6.1 Conclusion

Results from the studies show that high concentration of ions in the groundwater in the TSF basin is a result of contamination from surface processing activities. This is consistent with the significantly elevated values recorded in the TSF basin, cyanide ponds for both North and South HL facilities as compared to concentrations in groundwater samples from non-cyanide leaching areas.

The following conclusions were drawn from the studies:

- Mean calculated permeability and hydraulic conductivities of the samples gave values that conform to acceptable regulatory materials for the construction of tailings substrata. Station 5 exhibited the least attenuating geotechnical capability of 6.1 x 10^{-12} m^2/min and 6.2 x 10^{-3} m/day. Confirmed geochemical parameters for CEC, AEC, OMC, pH and Fe_2O_3 of the substrata are effective in attenuating cyanide to tolerable and acceptable limits. Recent (December 2012) geotechnical tests carried out on the substrata around the monitoring boreholes indicates a mean permeability value of 8.42 x 10^{-12} m/min which is lower than the stipulated guideline limit of 10^{-8} m/s
- Both WAD and total cyanide concentrations increase significantly with increase in depth in the decant pond and are consistent in both TSF 1 and 2. Total cyanide ranged from 0.1 mg/l to 1.11 mg/l whiles WAD cyanide concentrations ranged from 0.01 mg/l to 0.93 mg/l. pH in the decant pond, tailings slurry and underdrainage supernatant tower remains alkaline with values ranging from 9.4 to 10.6. Comparing this to the natural pH of groundwater (4.5 to 5.6) in the study area, an unsaturated interface is created which could lead to water movement from

the facility into aquifers. Ore and tailings geochemistry indicates that silica the dominant ion and Ca, Mg, Fe, Cu, Na, Mn and K are also present in the tailings dam. Average metal concentrations increased with depth of decant pond. Increased in average concentrations of Ni, Zn, Cr and As were recorded at depths exceeding 7 m in the decant pond was recorded whiles Ag concentrations decreased over the same depth of decant pond. It can be concluded from the study that the concentration of Ag decreased in concentration at depth of the decant pond due to metal complex formation

- Based on the concentration of Ec, pH, Na, K, Ca, as major cations and HCO_3, SO_4 and Cl as major anions, it could be established that monitoring boreholes along the toe of TSF1 and TSF3 have received seepage from the tailings facility. Similar concentrations were observed in boreholes near the South Heap leach facility, specifically near the pregnant pond also from monitoring boreholes located west of the North Heap leach Facility and specifically around the ponds and North containment pond. Monitoring boreholes with possible seepages in the TSF basin are NBH13, NBH14, NBH16, NBH17 all at the toe of TSF1; NBH28, NBH36 NBH38, NBH39, NBH42, NBH43 and NBH44. All these boreholes are located along the toe of TSF1, TSF2 and TSF3. Boreholes with possible seepages in the South Heap leach is SBH01 which is located near the pregnant pond; NBH20 and NBH21 and are located near the North Heap leach pregnant pond and process water containment area respectively

6.2 Recommendations

The following recommendations have been made from conclusions drawn from the studies;

- Review of relevant sections of existing legislation on TSFs with regards to lining monitoring regimes and sampling regime for cyanide in tailings dams should be expanded to cover other species
- Geotechnical parameters should be established across the basin floor before permitting
- TSFs should rigorously monitored to detect the migration of toxics into the groundwater system
- Sampling regime for cyanide in tailings dams should be expanded to cover other species of cyanide such as: thiocynate, cyanate, metal cyanide complexes and organic cyanide
- Aquifers with TSF area should completely defined before siting to prevent contamination

REFERENCES

Alesii, B.A. and Fuller, W.H. (1976). The mobility of three cyanide forms in soils. In: Proceedings Residual Management by Land Disposal, Hazardous Waste Research Symposium, EPA-600/9-76-015, U.S. Environmental Protection Agency, Cincinnati, OH: 213.

Al-Agha, M.R. and El-Nakhal H. A. (2004). Hydrochemical facies of groundwater in the Gaza strip, Palestine. *Hydrological Sciences–Journal–des Sciences, ISI Journals Citation Report*, 2003. Hydrologiques, 49(3): 356-371.

Álvarez, R., Ordóñez, A., Loredo, J. and Pendás, F. (2005). State of the art of Passive Treatment Technologies for the abatement of Cyanides from the Mining Industry: Experiences in Asturias (Spain). Proceedings of 9th International Mine Water Congress: 177-183.

Alyamani, M.S. and Sen, Z. (1993). Determination of Hydraulic Conductivity from Grain-Size Distribution Curves. *Groundwater*, 31(4): 551-555.

Amonoo Neizer E.H., Nyamah D., Bakiamoh S. B. (1996). Mercury and arsenic pollution in soil and biological 523 samples around the mining town of Obuasi, Ghana. Water Air Soil Pollution, 91:363-373.

Anderson, J.M. and Ingram, J.S.I. (1993). Tropical Soil Biology and Fertility, A Handbook of methods. (eds.), 2nd Edition. CAB International, U.K: 221.

Anon (2012), Mineral and Mining (Health, Safety and Technical) Law (L.I 2182): 180186.

Anon (2011). Metago Report, Tarkwa Gold Mine Tailings Storage Facility No.3. Detailed Design Report and Operating Requirements: 2–45.

Anon (2010). Department of Primary Industries, (2010). Management of Tailings Storage Facilities. Internet accessed on May 20th 2010 at www.new. dpi.vi.gov.au.earth-resources

Anon (2008). Cyanide Management, Leading Practice Sustainable Development Program for the Mining Industry, Minister for Resources and Energy, Minister for Tourism, Australia: 3–23.

Anon (2008a). Environmental Management Plan, 2008-2011. Gold Fields Ghana Limited, Tarkwa Gold Mine Report (Unpublished): 5-102.

Anon (2008b). Tarkwa Environmental Laboratory Standard Operating Procedure for Total Cyanide (Unpublished): 1–12.

Anon, (2008c). Tarkwa Gold Mine Water Quality Monitoring and Quality Assurance Program (Unpublished): 1–8.

Anon, (2007). Environmental Impact Statement for the Expansion Facilities at Tarkwa Gold Mine, Report by Messrs Metago: 1–45.

Anon, (2006). Tarkwa Gold Mine Water Quality Monitoring and Quality Assurance Program (Unpublished).

Anon, (2002). International Management Institute, *www.cyanidecode.org, Additional* Information on Cyanide: 1–2.

Anon, (1998). *Cyanide Management*, a booklet in a series on Best Practice Environmental Management in Mining, Commonwealth of Australia: 1–157.

Anon, (1997). Environmental Protection Agency General Effluent Quality Guidelines for Discharges into Natural Water Bodies, Accra, Ghana: 5–10.

Arthur, J., Holloway N, Brackley, I., Skelton, R., Philpott, K. and Barnes, L. (2004). An independent technical report on the Tarkwa Gold Mine, Ghana. Report prepared for Gold Fields Limited and IAMGold Corporation. G:\341290 Sadiola

Yatela\NI43-101 Report_Tarkwa v16.doc. SRK Consulting Engineers and Scientists: 98.

Asante, K.A. and Ntow W.J. (2009). Status of environmental contamination in Ghana, the perspective of the research scientist. Interdisciplinary Studies on Environmental Chemistry- Environmental Research in Asia, TERRAPUB: 253–260.

Asklund, R. and Eldvall, B. (2005). Contamination of water resources in Tarkwa mining area of Ghana. A minor field study for Master of Science thesis. Department of Engineering Geology Lund University, Sweden: 62.

Barsky, G., Swainson, S.J. and Hedley, N., (1962). Dissolution of Gold and Silver in cyanide solutions, Trans. Am. Inst. Min. Metal. Eng., 122: 660.

Barnie, S. (2010). Hydrogeological and Hydrochemical Framework of Groundwater for irrigation in the Atankwidi sub-basin of the White Volta Basin. MSc. Thesis (unpublished), Kwame Nkrumah University of Science and Technology, Kumasi, Ghana: 74 -85.

Black, C.A. (1986). Methods of soil analysis. Part I. Physical and mineralogical properties, including statistics of measurement and sampling. Part II. Chemical and microbiological properties. Agronomy series. ASA. Madison. Wis. USA.

Blight, G.C. (2002). "Waste Gypsum as an Embankment Material," Proceedings, 7th International Conference on Soil Mechanics and Foundation Engineering, Mexico, 1969: 39-43.

Botz, M.M and Mudder T.I., (1999). Modelling of cyanide degradation in tailings impoundments, SME Annual Meeting, March 1-3, 1999, Denver, Colorado: 1-8.

Borah, K.K., Bhuyan B. and Sarma H.P. (2009). Heavy metal contamination of groundwater in the Tea Garden Belt of Darrang District, Assam, India. CODEN ECJHAO: *E-Journal of Chemistry,* 6 (1): 501-507.

Buamah, R., Petrusevski, B. and Schippers, J.C. (2008). Presence of arsenic, iron and manganese in groundwater within the gold-belt zone of Ghana. Practical Paper. IWA Publishing. *Journal of Water Supply: Research and Technology*—AQUA, 57 (7): 519-521.

Carman, P.C. (1937). Fluid Flow through Granular Beds. Trans. Institute of Chemical Engineering: 15-150.

Carman, P.C. (1956). Flow of Gases through Porous Media. Butterworths Scientific Publications, London.

Carrier, W.D. (2003). Goodbye, Hazen; Hello, Kozeny-Carman. *Journal of Geotechnical and Geo-environmental Engineering*. 40 (120): 260 -261.

Chatwin, T.D. (1989). Cyanide Attenuation/Degradation in Soil. Resource Recovery and Conservation Company (R^2C^2), Salt Lake City, UT: 60.

Chatwin, T.D., Trepanowski, J. and Wadsworth, M.E. (1987). *Attenuation of Cyanide in Soils. Phase I report.* Resource Recovery and Conservation Company and the University of Utah: 114.

Chevalking, S., Knoop L. and Van Steenbergen F. (2008). Ideas for groundwater management.Meta and IUCN. Wageningen, The Netherlands: 142.

Cherry, J.A., Shepherd, T.A., and Morin, K.A. (1982). Chemical composition and geochemical behaviour of contaminated groundwater at uranium tailings impoundments, for presentation at the SME-AIME Annual Meeting, Dallas, Texas, February 14-18th 1982: 408-421.

Chilton, J. and Seiler K.P. (2006). Groundwater occurrence and hydrogeological environment in protecting groundwater for health: Managing the quality of drinking-water sources. World Health Organization. IWA Publishing, London, UK: 21-46.

Chilton J. (2006). Assessment of aquifer pollution, vulnerability and susceptibility to the impact of abstraction in protecting groundwater for health: Managing the quality of drinking-water sources. World Health Organization. IWA Publishing, London, UK: 199-239.

Clarke MC, Linkson PB (1987). The Toxicity and Degradation of Cyanide in Mine Tailings. Mining and Environment, A Professional Approach, National Conference, Brisbane: 87 -93.

Craft, D. (2005). *Reclamation: Managing Water in the West*. Seepage Chemistry Manual, Report DSO-05-03. Bureau of Reclamation, Technical Service Center Denver, Colorado: 2-64.

Community Water and Sanitation Agency (2006). National coverage statistics corporate brochure. CWSA, Ghana: 46.

Dzombak, D.A., Ghosh, R.S. and Young, T.C. (2006a).Physical–Chemical Properties and Reactivity of Cyanide in Water and Soil. In: *Cyanide in Water and Soil. Chemistry, Risk, and Management* CRC Press, Boca Raton, Florida, USA, 2006: 61.

Domenico, P.A., and Schwartz, F.W. (1998). Principles of Aqueous Geochemistry in Physical and Chemical Hydrogeology. 2nd Ed. John Wiley & Sons, New York, 11: 238-254.

Ebbs, S.D., Wong-Chong, G.M., Bond, B.S., Bushey, J.T. and Neuhauser, E.F. (2006a). Biological Transformation of Cyanide in Water and Soil. In: *Cyanide in Water and Soil. Chemistry, Risk, and Management* CRC Press, Boca Raton, Florida, USA, 2006: 94.

Ebbs, S.D., Bushey, J.T., Bond, B.S., Ghosh, R.S. and Dzombak, D.A. (2006b). Cyanide Phytoremediation. In: *Cyanide in Water and Soil. Chemistry, Risk, and Management.* CRC Press, Boca Raton, Florida, USA, 2006: 481.

Eisler, R., 1991, Cyanide Hazards to Fish, Wildlife, and Invertebrate: A Synoptic Review: Contaminant Hazard Review report 23, U. S. Dept. Interior, Fish and Wildlife Service: 55.

Evanko C.R., Dzombak D.A., Mellon P.E.C. (1997). Remediation of metalscontaminated soils and groundwater. Technology evaluation report for GroundWater Remediation Technologies Analysis Center.E-SeriesTE-97-01, Pittsburgh: 1-44.

Filipek, L.H. (1999). Determination of the source and pathway of Cyanide-bearing Mine Water Seepage. In: The Environmental Geochemistry of Mineral Deposits. *Part B: Case Studies and Research Topics*. Filipek, L.H. and Plumlee, G.S., (eds.): Reviews in Economic Geology. Vol. 6B. Society of Economic Geologists, Inc.: 549-565.

Freeze, R.A., and Cherry, J.A. (1979). Groundwater. Prentice Hall Inc., Englewood Cliffs, New Jersey: 10 -600.

Fuller, W.H. (1982). Methods for conducting soil column tests to predict pollutant migration: Proceedings, Land Disposal of Hazardous Wastes Symposium, EPA600/9-82-002: 87-105.

Fuller, W.H., Amoozergaar-Fard, A., Niebla, E.E. and Boyle, M. (1980). Influence of leachate quality on soil attenuation of metals. In: *Proceedings, Disposal of Hazardous Waste Symposium*, EPA-600/9-80-010: 108-117.

Flynn, C. M. and S. M. Haslem, 1995, Cyanide Chemistry-Precious Metals Processing and Waste Treatment: U. S. Bureau of Mines Information Circular 9429: 85282.

Ghosh, R.S., Ebbs, S.D., Bushey, J.T., Neuhauser, E.F. and Wong-Chong, G.M. (2006a). Cyanide Cycle in Nature. In: *Cyanide in Water and Soil. Chemistry, Risk, and Management*, CRC Press, Boca Raton, Florida, USA, (2006): 225.

Ghosh, R.S., Meeussen, J.C.L., Dzombak, D.A. and Nakles, D.V. (2006c). Fate and Transport of Anthropogenic Cyanide in Soil and Groundwater. In: *Cyanide in Water and Soil. Chemistry, Risk, and Management.* CRC Press, Boca Raton, Florida, USA, 2006: 197.

Golow, A. A., Schlueter A., Amihere, M. S, Granson H.L.K, Tetteh M. S. (1996). Distribution of arsenic and sulphate in the vicinity of Ashanti Goldmine at Obuasi, Ghana. Bull Environ Contam Toxicol, 56: 703-710.

Griffin, R. A. (1976)."Attenuation of Pollutants in Municipal Landfill Leachate by Clay Minerals: Part 1-Column Leaching and Field Verification," Ill. St. Geol. Surv. Envir. Geol. Notes No. 76, Urbana, Nov.1976.

Griffin, R.A. (1977). "Attenuation of Pollutants in Municipal Landfill Leachate by Clay Minerals: Part 2-Heavy-Metal Absorption," Ill. St. Geol. Surv. Envir. Geol. Notes No. 79, Urbana, Apr. 1977.

Gyau-Boakye, P. and Dapaah-Siakwan, S. (2000). *Hydrogeologic Framework and Borehole Yields in Ghana, Hydrogeology Journal, 8: 405-416.*

Hammer, M.J. and Hammer Jr. M.J. (2004). Water Quality in Water and Waste Water Technology. 5th Edn. New Jersey: Prentice-Hall, 4(5):139-159.

Hazen, A. (1993). Some Physical Properties of Sands and Gravels, with special reference to their use in Filtration. 24th Annual Report, Massachusetts State Board of Health, Pub. Doc., 34: 539-556.

Hem, J.D. (1992). Study and Interpretation of the chemical characteristics of natural water. USGS Water-Supply Paper: 2254.

Hem J.D. (1989). Chemical factors that influence the availability of iron and manganese in aqueous systems. Geological Society, Am Bull 1972, 83: 443-450.

Heming, T., R.V. Thurston, E. L. Meyn, and R. Zajdel, 1985, Acute Toxicity of Thiocyanate to Trout: Trans. Am. Fish Soc., 114: 895-905.

Higgins, C.J. and Dzombak, D.A. (2006). Free Cyanide Sorption on Freshwater Sediment and Sediment Components, *Soil & Sediment Contamination,* 15: 497510.

Huiatt, J. L. (1983). "Cyanide from Mineral Processing," Proceedings of a Workshop, National Science Foundation, Salt Lake City, Utah, February 2-3, 1982.

Jean-Claude, R. (1995). The evolution of groundwater quality in France: perspectives for enduring use for human consumption. *The Science of the Total Environment Journal*, 1995, 171: 3-16.

Johnson, K. (1986). Geochemical Model of the Migration of Trace Metals from Uranium Mill Tailings. PhD, Thesis, South Dakota School of Mines and Technology, Rapid City, SD: 8-48.

Johnson, K. (1985). Evaluation of Aquifer Reclamation by Geochemical Modelling and Solid Analysis. Proceedings of the Seventh Symposium on Uranium Mill Tailings Management, Colorado State University, Fort Collins, CO: 421 -430.

Kesse G.O. (1985). *The Mineral and Rock Resources of Ghana,* A.A. BalkemaNetherlands.

Khodapanah, L., Sulaiman W.N.A. and Khodapanah N. (2009). Groundwater quality assessment for different purposes in Eshtehard District, Tehran, Iran. *European Journal of Scientific Research Euro Journals Publishing,* Inc, 36 (4): 543-553.

Knowles, C.J., (1976). "Microorganisms and Cyanide," Bacteriological Reviews, September, 1976: 652-781.

Kortatsi, B.K. (2004). Hydrochemistry of groundwater in the mining area of TarkwaPrestea, Ghana, PhD Thesis (unpublished), University of Ghana, Legon-Accra, Ghana: 600 -607.

Kortatsi, B. K. (2004). Groundwater quality in the Wassa West District of the Western Region of Ghana .*West African Journal of Applied Ecology,* 11:25-41.

Kozeny, J. (1927). Uber Kapillare Leitung Des Wassers in Boden. Sitzungsber Akad. Wiss. Wien Math. Naturwiss. Kl., Abt.2a, 136:271-306.

Kresic, N. (2007). Groundwater chemistry in hydrology and groundwater modelling. Second edition. CRC Press, Taylor and Francis group. Boca Raton: 345-399.

Krumbein, W. C. (1934). Size Frequency Distributions of Sediments. J. Sed. Petrology, 4: 65-77.

Kuma, J.S. (2004). Is groundwater in the Tarkwa gold mining district of Ghana potable? *Environmental Geology,* 45: 391-400.

Kuma, J.S and Younger, P. L. (2001). Pedological characteristics related to groundwater occurrence in the Tarkwa area, Ghana, *Journal of African Earth Sciences*, 33(2): 363-376.

Lighthall PC, Bruce IG, McLeod HN (1987). Tailings Management for Cyanide Leach Gold Plant. Klohn Leonoff Ltd, Richmond, British Columbia.

Logsdon, M.J., Hagelstein, K. and Mudder, T.I. (1999). *The Management of Cyanide in Gold Extraction*. International Center for Metals and the Environment, Ottawa, Canada: 1-38.

Ma, C. and Eggleton, R.A. (1999). Cation Exchange Capacity of Kaolinite. *Clays and Clay Minerals,* 47(2): 174-180.

McBride, M.B. (1994). *Environmental Chemistry of Soils*. Oxford University Press, Inc. New York. U.S.A: 20-406

Metago, (2011). Tarkwa Gold Mine Tailings Storage Facility Number 3 Detailed Design Report and Operating Requirement, Metago Project Number 109-069 Report No.2.

Momodu M.A. and Anyakora C.A. (2010). Heavy metal contamination of groundwater: The Surulere Case Study. Research Journal Environmental and Earth Sciences. Maxwell Scientific Organization; 2(1): 39-43.

Moran, R.E. (2002), De-Coding Cyanide, An Assessment of Gaps in Cyanide Regulation at Mines. A Submission to the European Union and the United Nations Environmental Programme: 1 – 25.

Motsara, M.R. and Roy, R.N. (2008). Guide to Laboratory Establishment for Plant Nutrient Analysis. FAO Fertilizer and Plant Nutrition Bulletin 19. Rome, Italy.

Murray, L., Thompson M., Voigt K. and Jeffery J. (2000). Mine Waste Management at Ok Tedi Mine, Papua New Guinea. In: *Tailings and mine waste '00*. Balkema, Rotterdam: 507–515.

Nelson, D. W. and Sommers, L.W. (1982). Total carbon, organic carbon and organic matter. In: *Methods of Soil Analysis. 2. Chemical and Microbiological Properties*. Page, A.L., Miller, R.H. and Keeney, D.R. (eds.). Agronomy 9: 301312.

NICNAS (2010). Sodium Cyanide, February 2010. *Priority Existing Chemical Assessment Report* No. 31. Sydney, Australia. Internet accessed on April 3rd 2010 at www.nicnas.gov.au

Odong, J. (2007). Evaluation of Empirical Formulae for Determination of Hydraulic Conductivity based on Grain-Size Analysis. *Journal of American Science*, 3(3): 54-59.

Okalebo, J.R., Gathua, K.W. and Woomer, P.L. (1993). Laboratory methods of soil and plant analysis: A working manual. SSEA Technical Publication No: 1. Nairobi, Kenya.

Ozkan, S. and Ipekoglu, B. (2002). Investigation of Environmental Impacts of Tailings Dams. *Journal of Environmental Management and Health*, 13(3): 242-248.

Page, A.L., Miller, R.H. and Keeney, D. R. (eds.). (1982). Methods of Soil Analysis. Part 2. Chemical and Microbiological Properties. 2nd Edition. Agronomy Series 9, ASA, SSSA, Madison, Wis. USA.

Palmer, C.D. and PuIs R.W. (1994). EPA groundwater issue: natural attenuation of hexavalent chromium in groundwater and soils. United States Environmental Protection Agency, Office of Research and Development, Office of Solid Waste and Emergency Response, 94, 5: 1501-1505.

Prakash, K.L. and Somashekar R.K. (2006). Groundwater quality - Assessment on Anekal Taluk, Bangalore Urban district, India. *Journal of Environmental Biology. Triveni Enterprises, Lucknow (India)*, 27(4): 633-637.

Rennert, T. and Mansfeldt, T. (2002). Sorption of Iron–Cyanide Complexes in Soils, *Soil Science Society of American Journal:* 66-437.

Rossiter H.M. A., Owusu P.A, Awuah E, MacDonald A.M., and Schäfer A.I. (2009). Chemical drinking water quality in Ghana: water costs and scope for advanced treatment. Science of the Total Environment; 408 (2010): 2378–2386.

Rouse, J.V. and Pyrih, R.Z. (1993). Geochemistry. In: *Geotechnical Practice for Waste Disposal*. Daniel, D.E., (ed.), Chapman& Hall, London (1993): 2:19-20.

Rouse J.V. and Pyrih, R.Z. (1985) "Natural Geochemical Attenuation of Contaminants Contained in Acidic Seepage," Proceedings, international Conference on New Frontiers for Hazardous Waste Management," September 1985, EPAl60019851025: 192-1 99.

Rouse, J.V., and Pyrih, R.Z., (1988). "Natural Geochemical Attenuation of Trace Elements in Migrating Precious-Metal Process Solutions, “Proceedings, Perth International Gold Conference, Perth, Australia, October 28 - November 1, 1988: 77-81.

Rowell, D. L. (1994). Soil Science: Methods and Application. Longman Scientific and Technical, John Wiley and Sons Inc. New York: 117-119.

Santos-Burgoa, C., Rios, C., Mercado, L.A., Arechiga-Serrano, R., Cano-Valle, F., Eden- Wynter, R.A., Texcalac-Sangrador, J.L., Villa-Barragan, J., RodriguezAgudelo, P.Y. and Montes, S. (2001). Exposure to manganese: health effects on the general population, a pilot study in Central Mexico, Environ. Res, 85: 90104.

Schmidt, J.W., Simovic L. and Shannon, E. (1981). “Development Studies for Suitable Technologies for the Removal of Cyanide and Heavy Metals from Gold Milling Effluents”, Proceedings of the 36th Industrial Waste Conference, Purdue University: 831-846

Scott, J.S. (1984). "An Overview of Cyanide Treatment Methods for Gold Mill Effluents, “Proceedings, Conference on Cyanide and the Environment, Tucson, Arizona, December, 1984: 307-330.

Shepherd, R.G. (1989). Correlations of Permeability and Grain Size. Groundwater, 1(6): 663-638.

Simovic, L. (1984).“Kinetics of Natural Degradation of Cyanide from Gold Mill Effluents”, Master of Engineering Thesis, McMaster University, Ontario.

Simovic, L. Snodgrass W.J. Murphy, K.L. and Schmidt, J.W. (1984).“Development of a Model to Describe the Natural Degradation of Cyanide in Gold Mill Effluents”, Conference on Cyanide and the Environment, Tucson, Arizona: 413-432.

Simovic, L. and Snodgrass, W.J. (1984), "Development of a Model to Describe the Natural Degradation of Cyanide in Gold Mill Effluent", *Pro*ceedings, Conference on Cyanide and the Environment, Tuscon , Arizona, Published by Geotechnical Engineering Program, Colorado State University, 2: 413-432.

Simovic, L. and Snodgrass, W.J. (1989). “Tailings Pond Design for Cyanide Control at Gold Mills Using Natural Degradation”, Proceedings from Gold Mine Effluent Treatment Seminars, Vancouver: 58-81.

Smedley, P.L. and Kinniburgh, D.G. (2001). A review of the source, behaviour and distribution of arsenic in natural waters. Appl. Geochem. British Geological Survey, Wallingford, Oxon OX10 8BB, U.K, 17: 517–568.

Smith, A. and Mudder, T. (1991). The Chemistry and Treatment of Cyanidation Wastes, Mining Journal Books Ltd., London, 3: 1-345.

Smith, A. and Mudder, T. (1993). The Environmental Geochemistry and Fate of Cyanide. *Proceedings of the Society of Economic Geologists Meeting, Denver, Colorado.,* April 1993, 14: 44.

Smith, A., and Struhsacker, D.W. (1988). "Cyanide Geochemistry and Detoxification Regulations, “Introduction to Evaluation, Design, and Operation of Precious Metal Heap leaching Projects, Society of Mining Engineers, Littleton, Colorado: 275-292.

Sparks, D.L, (2003). Environmental Soil Chemistry, Second Edition, Academic Press, San Diego California, USA, 1: 2-20

Taylor, P., Owen, R. and Tuinhof, A. (2010). Groundwater management in IWRM. Training manual. Capnet: 114.

Udayalaxmi G., Himabindu D. and Ramadass G. (2010). Geochemical evaluation of groundwater quality in selected areas of Hyderabad, A.P., *India. Indian Journal of Science and Technology Indian Society for Education and Environment, India*, 3(5): 546-553.

U.S. EPA (1994a). Technical document acid mine drainage prediction. EPA 530-R-94-036 NTIS PB94-201829 U.S. Environmental Protection Agency Office of Solid Waste Washington, DC: 52.

U.S. Environmental Protection Agency (1989), *Transport and Fate of Contaminants in the Subsurface,* Seminar Publication, EPA/625/4-89/019, Sept. 1989: 52-62.

Vukovic, M., and Soro, A. (1992). Determination of Hydraulic Conductivity of Porous Media from Grain-Size Composition. Water Resources Publications, Littleton, Colorado: 1-44.

World Health Organization (2008). Guidelines for drinking-water quality 3rd Ed incorporating the 1st and 2nd Addenda, Volume 1 recommendations. World Health Organisation, Geneva: 1-83.

Worral, W.E (1986). Clays and Ceramic Raw Materials. 2nd Edition, Elsevier Applied Science Publishers, London: 89-123.

APPENDICES

APPENDIX 1

Location of leaching/ sampling sites

Sample ID	GPS Co-ordinates	Location
Station 1	N590310 E608717	Right hand Side of NBH 18
Station 2	N590773 E608454	Right hand Side of NBH 16
Station 3	N591670 E607979	Behind NBH 31 & NBH 32
Station 4	N592598 E607634	About 200m before NBH 28
Station 5	N593461 E697778	On the right side of RCA

APPENDIX 2

Water quality monitoring and quality assurance program

TGM-ENV-ENV 01

1.0 PURPOSE

To ensure that:

- Water quality monitoring at GFGL's operating areas are undertaken in accordance with Ghanaian regulatory requirements and in response to external complaints from surrounding communities.
- Water quality samples are collected in a manner that minimizes potential for contamination and monitoring equipment is operated and handled as specified by the manufacturer.
- Monitoring data is recorded and reported to required parties.
- Monitoring boreholes are kept locked to prevent tampering.

2.0 PROCEDURE SUMMARY

a. Safety First
b. Go through the check list of items required for field sampling activities before initiating a sampling exercise.
c. Sample at specified locations and at the specified frequency for specified parameters.
d. Sample bottles are sterilized and not opened until samples are to be taken.
e. Follow sampling procedures to ensure representative samples are obtained.
f. Preserve samples as required with acid or base.
g. Keep all samples at $4^{O}C$ prior to delivering to the Lab.
h. Undertake on site analysis of some physical parameters (pH, Conductivity, TDS, DO & Temperature) following specified procedures.
h. Obtain specified quality control samples.
i. Maintain and clean monitoring equipment as specified by the manufacturer's operating manual.
j. Modify or enhance monitoring procedures/locations when a change in operations or infrastructure occurs with potential affect on water quality or when required due to regulatory requirements.

3.0 PROCEDURE DETAIL *3.1 Safety First*

Personal Protective Equipment including the following must be provided and used during sampling: rubber boots, safety goggles, rubber hand gloves (surgical type), hard hat.

3.2 Checklist

The under listed items will be checked or prepared prior to leaving the office to undertake the sampling exercise:

- Adequate quantity of washed/sterilized and properly labelled sample bottles
- Cooler box

- Adequate number of ice packs for cooler box
- Adequate quantity of preservation reagents (Nitric+Sulphuric Acid, Sodium hydroxide)
- On-site measuring instrument Quanta Multi Probe (calibrated in office)
- Flow meter (where applicable)
- Sampling data recording sheet, field note book, pen, pencil
- Appropriate PPE in place (Hand gloves, gum boots, goggles, hard hat etc.)
- Wash bottle filled with distilled water
- Tissue paper
- Cleaning/sterilizing agents (methylated spirits)
- Keys to all monitoring boreholes
- Generator with adequate fuel in fuel tank
- Submersible pump(s), 2" and/or 3", with all hose/pipes, etc.
- Regulator for submersible pumps
- Water depth indicator for sampling of boreholes
- Bucket (for ground water sampling)
- Hand bailing equipment
- Any other required items

3.3 Sampling Schedule & Parameters

Water quality sampling point locations for surface, ground and potable water are shown on Attachment 1 and marked with their descriptive codes. A key to these codes is provided in Attachment 2. The locations of these monitoring points were selected based on their ability to serve as indicators should there be any form of contamination emanating from GFGL's mining and processing operations.

The indicated sampling points will be monitored at defined frequencies (weekly, monthly or quarterly) and analysed for a defined set of parameters based upon regulatory requirements, their proximity to the operational area, the type of activity or infrastructure located upstream, and the potential influence on the environment downstream. Sampling frequencies and parameters for each water quality sampling point are indicated in Attachment 3.

Laboratory analysis of the parameters indicated will be completed using the methods indicated in Attachment 7. Field analysis of some basic physical parameters will be accomplished utilizing the Quanta Multi-Probe as described in Attachment 4.

3.4 Sample Bottle Preparation

a) All sample bottles must be washed or sterilized before use. Washing/sterilization will be done by the contract laboratory.
b) One litre sample bottles will be used for all samples except those taken for cyanide, which will be 500 ml. Dark bottles will be used for samples taken for cyanide.
c) Sample bottles must not be opened before sampling.
d) The type of bottle to be used for each sample will be in accordance with the specifications indicated in "Requirements for Analytical Parameters".

e) Sample bottles will be clearly and neatly labelled with the descriptive code of every sample to be taken before leaving the office. Labels will be created with permanent marker on white plastic bottles and by fixing a paper label and taping over it on dark coloured bottles.

3.5 Surface Water Sampling Procedure

a) Select the sample bottle(s) labelled for the specific sampling point.
b) Carry the sample bottle(s), bucket, cool box with ice packs, Quanta Multi Probe instrument, preservatives, labels (where required), and data recording materials to the sampling site.
c) Samples are taken in the middle of the creek/stream **not** at the banks.
d) Sample bottles should only be opened immediately prior to when the sample is about to be taken.
e) With the sample bottle oriented in the upstream direction (against flow), obtain a reasonable quantity of water (about $^1/_3$ full) and rinse the sample bottle three times with stream water prior to taking the sample.
f) Water quality samples should be taken with the bottle fully immersed in the stream/river to obtain a representative sample. Avoid sampling surface film as this contaminates the sample. The sample bottle should be nearly filled, leaving enough room for addition of preservative but sufficient to expel air before capping.
g) Preservative is added based on the parameter required for analysis to ensure proper storage/fixation of sample following the guidelines in Table 1 in “Sample Preservation, Storage and Submission”.
h) After sampling, the bottle should be squeezed to expel air before capping it.
i) Samples are then stored immediately in the cool box with ice packs.
j) On site analysis of pH, TDS, Ec, DO, and temperature is completed using the Quanta Multi-Probe as described in “On Site Measurement” and results recorded on an “On Site Water Sampling Analysis Sheet” as provided in Attachment 8.
k) If the bucket is used for analysis with the Quanta Multi-Probe (because of weeds, rocks, or overly shallow depth), the bucket must be rinsed three times with water from the center of the stream prior to filling for completing on-site monitoring.
l) When sampling activities are completed, samples are transported to the Environmental Department office in the cool box, removed from the cool box and placed in the refrigerator (set at 4°C) for storage until they are to be taken to the commercial laboratory and submitted for analysis. Maximum holding times for samples are indicated in Table 1 in “Sample Preservation, Storage and Submission”.

3.6 Ground Water Borehole Purging

a) Piezometer and boreholes should be purged 24 hours before samples are taken. The well should be purged until dry or a minimum of three well volumes should be purged prior to each sampling event. The well diameter and total depth will be used to determine the purging volume whenever the well cannot be pumped dry.

b) Carry the submersible pump, hoses/pipes, generator set, regulator, and water depth indicator, preservatives, labels (where required), and data recording materials to the well site.
c) Unlock the padlock of the borehole using the keys and remove the cap of the casing.
d) Inspect the well site and note the general condition of the well for leaks, evidence of tampering, condition of well caps; report any problems to the Environmental Superintendent or Environmental Manager.
e) Remove the pump, generator set, regulator and water depth indicator from their carrying cases where applicable.
f) Determine the static water level by measuring the water depth from the top of the well casing to the surface of water in the well by lowering the indicator in the well until the audible beep is heard. Record the static water level.
g) Calculate the required purge volume by subtracting the depth to water from the well total depth and multiplying the resulting height of water in the well by the area of the well as $(3\pi\times [\,{}^{D}/_{2}\,]^{2})$, where D is the diameter of the well/casing.
h) Connect the pipes/hoses to the pump, the pump to the regulator and the regulator to the generator set.
i) Lower the pump into the borehole to be purged.
j) Switch the generator set on and set the regulator frequency between 150 and 200 Hz.
k) Start the pump to pump the required volume of water out for purging by using a vessel (bucket, drum) of known volume to pump into or by pumping for a fixed time at a known pumping rate.
l) After purging is complete, switch off the pump and set the regulator to zero Hz.
m) Turn off the generator set and remove the pump from the borehole or well.
n) In order to avoid cross-contamination of wells, drain the pump and hose by inverting pump and raising the hose/pipe for all water remaining to drain out.
o) Wash the pump and tube with clear water and clean the outside with soft cloth.
p) Take all equipment utilized back to the vehicle for transportation to the next well to be purged or back to the Environmental Department offices.

3.7 Ground Water Sampling Procedure

a) Keep the order in which the wells are sampled consistent with the order of purging and from one sampling event to the next.
b) Select the sample bottle(s) labelled for the specific sampling point. Sample bottles should only be opened immediately prior to when the sample is about to be taken.
c) Carry the sample bottle(s), cool box with ice packs, bucket, hand bailer, submersible pump, hoses/pipes, generator set, regulator, Quanta Multi Probe instrument, water depth indicator to the well site, and data recording materials to the sampling point.
d) Unlock the padlock of the borehole using the keys and remove the cap of the casing.
e) Remove pump, generator set, regulator and hoses/pipes out of their carrying cases where applicable.
f) Connect the hoses/pipes to the pump, the pump to the regulator and the regulator to the generator set.
g) Lower the pump into the borehole to be sampled.
h) Switch the generator set on and set the regulator frequency between 150 and 200 Hz.
i) Start the pump and run until water comes out of the hose/pipe for three seconds.
j) Sample bottles should only be opened immediately prior to when the sample is about to be taken.

k) Rinse the sample bottle three times with water pumped from the well.
l) Fill the sample bottle with water pumped from the well. The sample bottle should be nearly filled, leaving enough room for addition of preservative but sufficient to expel air before capping.
m) Preservative is added to the sample based on the parameter required for analysis to ensure proper storage/fixation of sample following the guidelines in Table 1 in "Sample Preservation, Storage, and Submission".
n) After sampling, the bottle should be lightly squeezed to expel air before capping it.
o) Samples should be stored immediately in the cool box with ice packs.
p) Rinse the bucket three times with water pumped from the well and then fill it half full for use in on-site analysis.
q) On site analysis of pH, TDS, Ec, and temperature is completed using the Quanta Multi-Probe as described in "On Site Measurement" and Attachment 4 and record results on an "On Site Water Sampling Analysis Sheet" as provided in Attachment 8.
r) After the on-site measurement is complete, switch off the pump and set the regulator to zero Hz.
s) Turn off the generator set and remove the pump from the borehole or well.
t) In order to avoid cross-contamination of samples, drain pump and hose by inverting pump and raising the tube for all water remaining to drain out.
u) Wash the pump and tube with clear water and clean the outside with soft cloth.
v) Empty the bucket and use the hand bailer to obtain enough water from the well to rinse the bucket three times and then fill it half way.
w) Complete on-site analysis of DO using the Quanta Multi-Probe as described in "On Site Measurement" and Attachment 4 and record results on an "On Site Water Sampling Analysis Sheet" as provided in Attachment 8.
x) If external laboratory analysis of DO is requested for a particular sampling point, the hand bailing device is then used to rinse the appropriately labelled sample bottle three times prior to filling the sample bottle with water from the well, squeezing to expel air, and capping.
y) Replace the cap of the casing and lock the padlock, ensuring regular lubrication of the padlock.
z) Drain the hand bailing device and rinse with clear water.
aa) When sampling activities are completed, samples are transported to the Environmental Department office in the cool box, removed from the cool box and placed in the refrigerator (set at 4^oC) for storage until they are to be taken to the contract laboratory and submitted for analysis. Maximum holding times for samples are indicated in Table 1 "Sample Preservation, Storage and Submission".

3.8 Potable Water Sampling Procedure

a) Select the appropriately labelled sterilized bottle and do not open until immediately before the sample is to be taken.
b) Sterilise the tap with methylated spirit or by heating prior to turning tap on.
c) Open tap full for water to flow for about 1 minute to flush pipeline to tap.
d) Reduce the flow rate from the tap and rinse sample bottle thoroughly with the tap water for about 1 minute (3 times or more).
e) Fill the sample bottle with water from the tap. The sample bottle should be nearly filled, leaving enough room to expel air before capping.

f) After sampling, the bottle should be lightly squeezed to expel air before capping it (with gloves on).
g) Change gloves after each sampling event and ensure your fingers do not touch the inside of the bottle cap
h) Place the sample in cool box immediately.
i) When sampling activities are completed, samples are transported to the Environmental Department office in the cool box, removed from the cool box and placed in the refrigerator (set at 4^{o}C) for storage until they are to be taken to the contract laboratory and submitted for analysis. Maximum holding times for samples are indicated in Table 1 in “Sample Preservation, Storage and Submission”.

3.9 Sample Preservation, Storage & Submission

a) Sample preservation, holding times and bottle types for various sample types will be as specified in Table 1.
b) Trace Metals “total” analysis (incl. Calcium, Magnesium, Potassium and Sodium) – For “total” analysis of metals sample preservation will be accomplished through acidification with 4 ml of concentrated nitric acid (70%) to the sample volume of 1Litre. Samples are then kept at a temperature of about 4^{o}C until submission. For “dissolved” concentrations of metals no chemical preservation required, samples to be kept at 4^{o}C.
c) Cyanide (total and free) – Sample preservation will be accomplished through addition of 10 ml of 1.0M sodium hydroxide solution to the sample volume of 1Litre. Samples are then kept at a temperature of about 4^{o}C until submission.
d) Nitrate, Nitrite, Ammonia, Phosphorus, Oil & Grease – Sample preservation will be accomplished through addition of 4 ml of analytical grade Sulphuric Acid to the sample volume of 1 Litre. Samples are then kept at a temperature of 4^{o}C until submission.
e) Physical Parameters – No chemical sample preservation is required. Samples are kept at a temperature of about 4^{o}C until submission.
f) Other Geochemical Parameters – No chemical sample preservation is required. Samples are kept at a temperature of about 4^{o}C until submission.
g) Microbiological Parameters (Coliforms, plate count) – no chemical sample preservation is required. Samples are kept at a temperature of about 4^{o}C until submission.
h) Samples will always be submitted to the external laboratory within 24 hours after collection and will in no case be held longer than the holding time specified in Table 1.
i) A laboratory submission sheet as provided in Attachment 5 will be prepared for all samples to be submitted to an external laboratory indicating the name of the person submitting the sample, all the parameters to be analyzed, date submitted, when results are required and any other specific instructions. This sheet will accompany all samples submitted to the external laboratory. Results should be requested to be returned within ten days of submission, unless earlier returns are required. A copy will be returned to the Environmental Department file. All analyses will be

completed by an external laboratory except those completed on site as specified at other locations in this procedure.

j) Samples from boreholes will be specified to be analyzed for dissolved concentrations of constituents, samples from surface water locations will be specified to be analyzed for total concentrations of constituents where applicable.

k) Sample methods utilized by the external laboratory will be as specified in Attachment 7.

Table 1: Sample Preservation, Storage and Bottle Type Requirements

Parameter [1]	**Preservation Method**	**Maximum Holding Time**	**Container Type & Minimum Size [3] (mL)**
General Parameters			
pH (standard units)	Cool	24 hours	P – 1000
SC (µmhos/cm)	Cool	28 days	P – 1000
Total Dissolved Solids	Cool	7 days	P – 1000
Total Suspended Solids	Cool	7 days	P – 1000
Turbidity (NTU)	Cool	48 hours	P – 1000
Color (color units)	Cool	48 hours	P – 1000
Alkalinity, total	Cool	14 days	P – 1000
Hardness, as CaCO3	Calculated	Calculated	Calculated P-
Silica	Cool	28 days	1000
Cations			
Calcium	Nitric	6 months	P – 100
Magnesium	Nitric	6 months	P – 100
Potassium	Nitric	6 months	P – 100
Sodium	Nitric	6 months	P – 100
Anions			
Bicarbonate, as CaCO3	Cool	28 days	P – 1000
Carbonate, as CaCO3	Cool	28 days	P – 1000
Chloride	Cool	28 days	P – 1000
Fluoride	Cool	28 days	P – 1000
Sulfate	Cool	28 days	P – 1000
Nutrients, Cyanide & Other Chemicals			
Nitrate, as N	Cool	48 hours	P – 1000
Nitrate+Nitrite, as N	Sulfuric	28 days	P – 250
Ammonia, as N	Sulfuric	28 days	P – 250
Phosphorus, ortho	Sulfuric	28 days	P – 250
Phosphorus, total	Sulfuric	28 days	P – 250
Oil & Grease	Sulfuric	28 days	G – 250 (P
Cyanide, total	NaOH	14 days	acceptable)
Cyanide, free	NaOH	14 days	P dark – 500

			P dark – 500
Metals and Metalloids			
All metals except Mercury	Nitric	6 months	P – 100
Mercury (Hg)	Nitric	28 days	P – 100
1. For Dissolved metals, filter on site before acidification or do not acidify but ensure no air is added to the container and sample is kept refrigerated 2. Samples for microbiology must be contained in a sterilized sampling bottle and refrigerated 3. P-Plastic (polyethylene) G-Glass			

3.10 On Site Water Quality Measurements

a) The Quanta Multi Probe instrument will be used in the field to measure the pH, conductivity, TDS, temperature and dissolved oxygen of samples as required in the Water Quality Sampling program indicated in Attachment 3. Detailed operating procedures for this instrument are provided in Attachment 4 and summarized below.

b) The Quanta Multi Probe instrument will be calibrated with standard solutions in the office before the monitoring exercise is undertaken in the field in accordance with the procedures indicated in Attachment 4.

c) At the field sampling site, the equipment probes are removed from their housing case.

d) For surface water samples, the probes of the measuring instrument are immersed well below the surface of the water the middle of the stream/river. If the stream is too deep to enter or too shallow for proper immersion, a bucket will be used to obtain a sample of water from the centre of the stream. After rinsing the bucket 3 times with water from the stream at the sampling point.

e). For ground water samples from boreholes, rinse and fill a bucket using the methodology described under “Ground Water Sampling Procedure” and immerse the probes of the instrument below the surface of the water in the bucket.

f) Switch the instrument on and take readings as indicated in the operating procedures provided in Attachment 4. Do not switch the instrument on until the probes are immersed or damage to the equipment may result.

g). Record all readings obtained on the On Site Water Analysis Sheet form provided in Attachment 8, keeping the probes immersed.

h) Switch the instrument off and remove from the water being analyzed. Do not remove the probes from the water until the instrument is shut off.

i) Clean the probes carefully with distilled water from the wash bottle by squeezing distilled water out against the probes.

j). Wipe the probe with tissue paper taking care not to touch the tip of the probe as this may cause damage to the probe.

k) Place the multi probe into the housing case and transport it safely to the next sampling location or to the Environmental Department Office for storage.

3.11 Quality Control Samples

a) Quality control samples (duplicates and blanks) will be taken monthly for QA/QC purposes.
b) For every monthly sampling event two duplicate samples, one from South ADR Plant area and another from the North ADR Plant area, will be taken from monitoring bores or downstream surface water monitoring points and analyzed for all of the same parameters as the primary or original sample.
c) Sample locations will be chosen at random and may be from surface or ground water.
d) The same preservatives are used for the duplicate samples as would be utilized for the primary or original sample.
e) Duplicate samples will be taken at the same time that primary samples are taken.
f) Duplicate samples will be submitted to another external laboratory for analysis.
g) Two blank samples of de-ionized water will be prepared and submitted monthly to the external laboratory. One blank sample will be specified to be analyzed for physical parameters, the second will be specified to be analyzed for free and total cyanide. The same type and quantity of preservative required for physical parameters and cyanide will be added to the blank samples.
h) Labelling of the duplicate and blank samples will be done in a manner so that the external laboratory cannot identify the location/descriptive code of the site.

3.12 Stream Flow Measurement

a) Remove the instrument from the carrying case.
b) Calibrate the flow meter as described in Attachment 6.
c) Perform stream flow measurements at stream sampling locations following the procedures provided in Attachment 6.
d) Record all required data for stream flow measurements on the Stream Flow Data Recording Sheet provided as Attachment 9, including width of stream, depths of each segment, and average and maximum speeds at each measurement location.
e) Place the flow meter back into its carrying case after the measurement.

4.0 Record Keeping and Reporting

a) Prior to obtaining any sample from the monitoring point, it is important to record any Observations made at the monitoring location such as description of the water quality (Muddy, clear, oil at the surface etc), any fauna death, any recipient discharges, climatic condition (rainy), fauna footprint and etc in the field sampling sheet.
b) Electronic copies of analysis results from the primary external laboratory will be maintained electronically on the Environmental Manager’s computer and on a computer in the offices of the Environmental Coordinator.
c) Water quality analysis data from on site monitoring activities and from results obtained from external laboratories will be entered into an electronic database and maintained on a computer in the offices of the Environmental Coordinator. Updated copies of these records will be provided to the Environmental Manager on a quarterly basis.
d) Flow rate data from on site monitoring activities will be entered into an electronic database and maintained on a computer in the offices of the Environmental

Coordinator. Updated copies of these records will be provided to the Environmental Manager on a quarterly basis.

e) Written field data collection sheets will be stored in files located at the offices of the Environmental Coordinator.

f) Analysis results received from external laboratories will be printed out and filed in the files located at the offices of the Environmental Coordinator.

g) The Environmental Manager will maintain a copy of summarized water quality records in his EMS files and will report to the General Manager when there are exceedances of regulatory standards. A descriptive summary of water quality monitoring results will be included in the monthly environmental report to the General Manager, providing information on exceedances where applicable.

h) Water quality monitoring will be undertaken in accordance with the Water Quality Sampling Program in Attachment 3 and monitoring results will be reported to the EPA by the Environmental Manager as indicated in the Table 2 below:

Table 2: Water Quality Monitoring Reporting Requirements

Type	Location	Frequency	Reporting Requirements
All scheduled surface and ground water monitoring locations	As specified in Attachment 1	See Attachment 3	Basic physical parameters and cyanide levels (where applicable) in EPA monthly returns (locations as required by EPA) and all results in Annual Env. Report.
External Complaints	Complainants' location	When complaints is received	Upon completion to complainants and EPA and in Annual Env. Report.

5.0 Maintenance and Cleaning of Equipment

a) Regular inspection and cleaning of electronic water quality and flow monitoring equipment will be completed as detailed in the operating procedures for the equipment to ensure proper operating condition.

b) Any required maintenance will be completed as soon as possible when the need arises to avoid last-minute delays in monitoring, including regular battery replacement (or recharging). Batteries may be left in meters for monthly intervals between monitoring events unless prohibited by the manufacturer.

c) Report any faulty equipment to either the Environmental Superintendent or the Environmental Manager immediately after a problem is identified with any water monitoring equipment.

6.0 Review

a) This procedure will be modified or enhanced should additional monitoring equipment of another type be utilized for water quality or flow rate monitoring or should changes in sampling locations, parameters or frequency be necessary due to changes in operational activities at the Tarkwa Gold Mine or due to regulatory requirements.

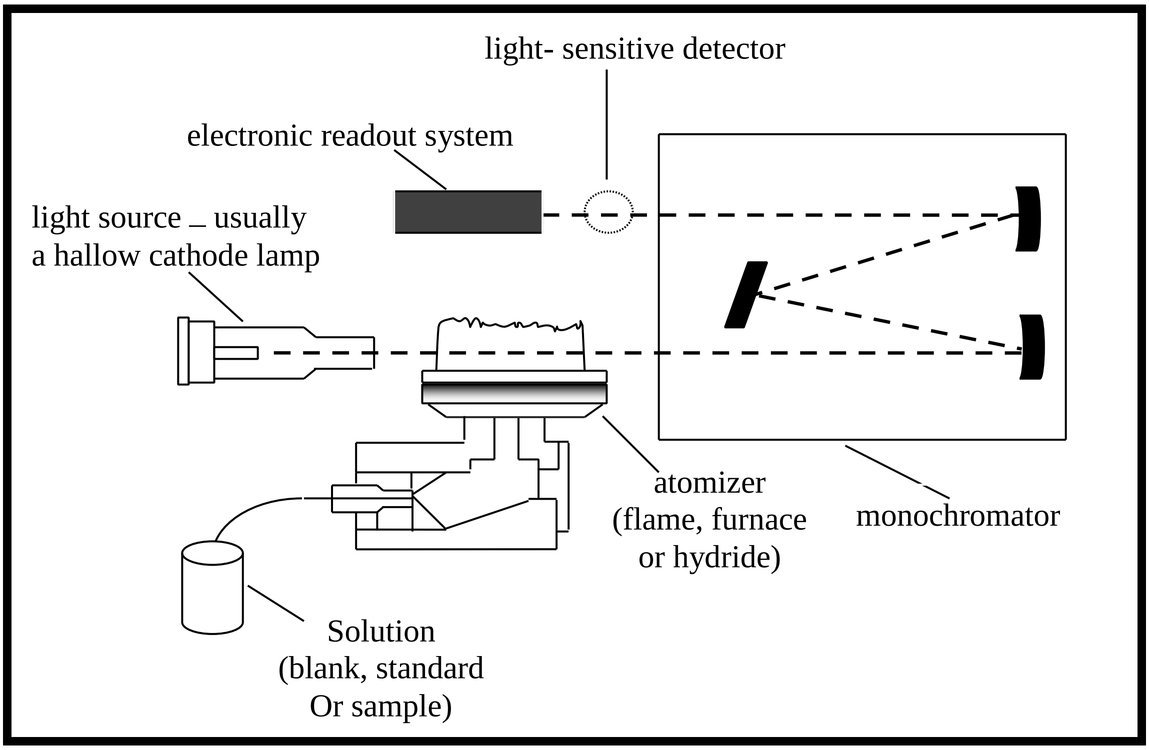
b) Any person tasked to perform water quality monitoring activities will be given a thorough description of instrument protection and operation procedures, sample collection and handling procedures, sample submission procedures, and data handling/reporting procedures, and will accompany personnel experienced in these functions to the field during an actual water quality monitoring exercises for further instruction prior to being allowed to perform any water quality monitoring activity by himself.

APPENDIX 3

Operations of the AAS

The equipment consists of the following components;

- A lamp compartment which contains the hollow cathode lamps of the analyte of interest.
- Atomising chamber which vaporises the sample in the flame transforming it into unexcited ground state atoms to absorb light at specific wavelength. The source of energy for the production of free atoms is usually heat commonly from an air/acetylene or nitrous –oxide/acetylene flame. Usually, the sample is introduced as an aerosol into the flame and the burner aligned in the optical path so that the light beam passes through the flame, where the light is absorbed.
- An optical system which directs light from the source through the atom population into the monochromator. The monochromator isolates the specific analytical wavelength of the light emitted by the hollow cathode lamp from the other non-analytical lines including those of the fill gas.
- A photomultiplier tube to measure the light accurately.
- The display of the results of the analysis.



Schematic Diagram of AAS (After Faanu)

APPENDIX 4

Grading analysis of soil samples at leaching sites

Grading Analysis

CIVIL ENGINEERING DEPARTMENT
GEOTECHNICAL ENGINEERING LABORATORY
GRADING ANALYSIS

PROJECT:-M.Sc. Thesis
SITE:-Gold Fields Ghana Limited, Tarkwa

SAMPLE No: TSF1 - 001

GRADING TEST Total Dry Weight (g): 50

Sieve size		Weight retained (g)	Percentage retained (%)	Percentage passing (%)
BS designation	Metric (mm)			
3 in	75.00			
2 1/2 in	63.00			
2 in	53.00			
1 1/2 in	37.10			
1 in	26.50			
3/4 in	19.00			
1/2 in	13.20			
3/8 in	9.50			
1/4 in	6.70			
3/16 in	4.75			

Total Dry Weight (g): 50

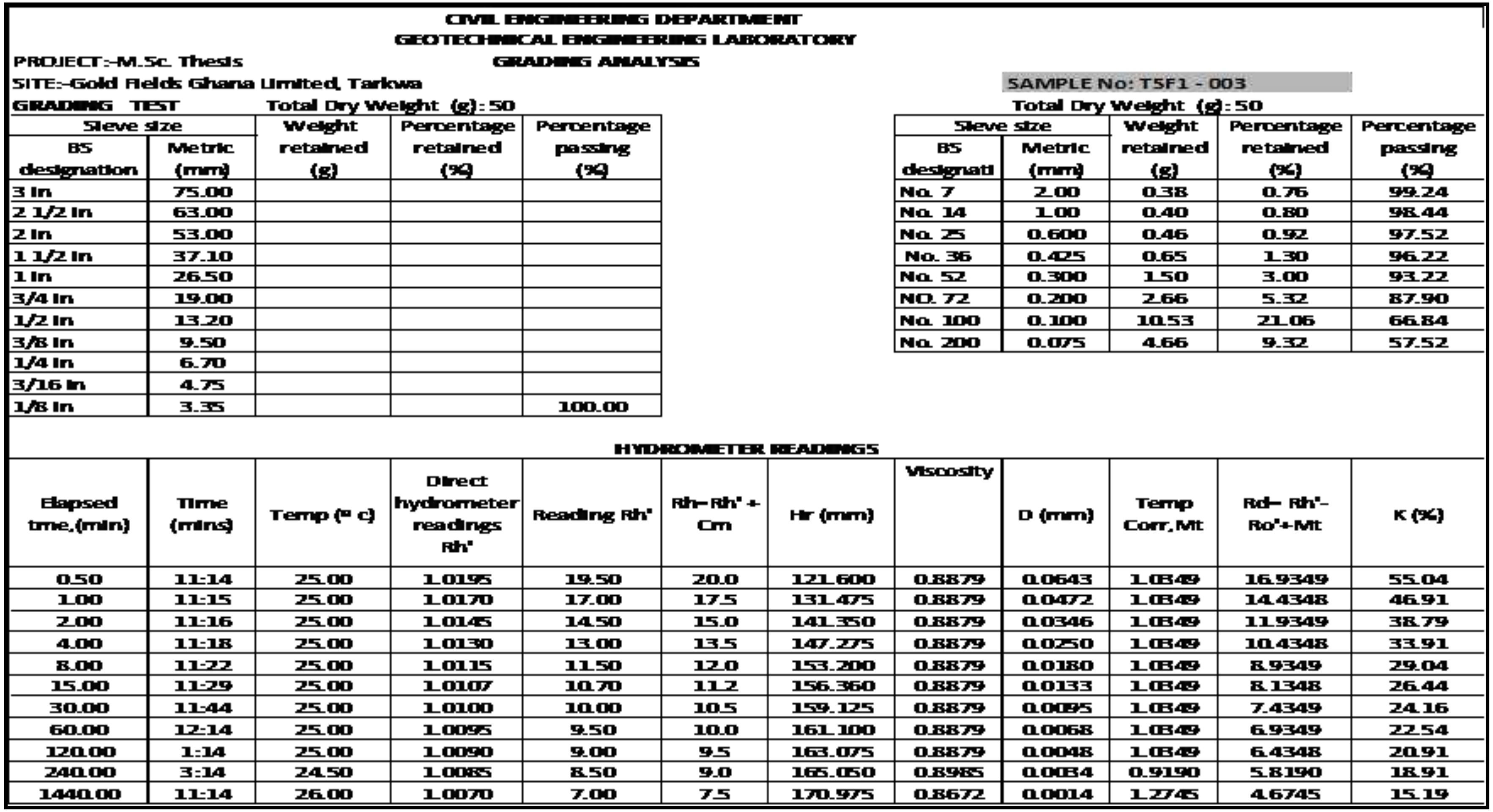
Sieve size		Weight retained (g)	Percentage retained (%)	Percentage passing (%)
BS designati	Metric (mm)			
No. 7	2.00	0.15	0.30	99.70
No. 14	1.00	0.38	0.76	98.94
No. 25	0.600	0.50	1.00	97.94
No. 36	0.425	0.58	1.16	96.78
No. 52	0.300	0.97	1.94	94.84
NO. 72	0.200	1.59	3.18	91.66
No. 100	0.100	8.59	17.18	74.48
No. 200	0.075	5.65	11.30	63.18

ROMETER READINGS

1/8 in	3.35			100.00

HYD

Elapsed tme,(min)	Time (mins)	Temp (^{O}C)	Direct hydrometer readings Rh'	Reading Rh'	Rh=Rh' + Cm	Hr (mm)	Viscosity	D (mm)	Temp Corr,Mt	Rd= Rh'Ro'+Mt	K (%)
0.50	11:14	25.00	1.0205	20.50	21.0	117.650	0.8879	0.0632	1.0349	17.9349	58.29
1.00	11:15	25.00	1.0175	17.50	18.0	129.500	0.8879	0.0469	1.0349	14.9349	48.54
2.00	11:16	25.00	1.0145	14.50	15.0	141.350	0.8879	0.0346	1.0349	11.9349	38.79
4.00	11:18	25.00	1.0125	12.50	13.0	149.250	0.8879	0.0252	1.0349	9.9348	32.29
8.00	11:22	25.00	1.0110	11.00	11.5	155.175	0.8879	0.0181	1.0349	8.4348	27.41
15.00	11:29	25.00	1.0102	10.20	10.7	158.335	0.8879	0.0134	1.0349	7.6348	24.81
30.00	11:44	25.00	1.0095	9.50	10.0	161.100	0.8879	0.0095	1.0349	6.9349	22.54
60.00	12:14	25.00	1.0090	9.00	9.5	163.075	0.8879	0.0068	1.0349	6.4348	20.91
120.00	1:14	25.00	1.0085	8.50	9.0	165.050	0.8879	0.0048	1.0349	5.9348	19.29
240.00	3:14	24.50	1.0080	8.00	8.5	167.025	0.8985	0.0035	0.9190	5.3190	17.29
1440.00	11:14	26.00	1.0070	7.00	7.5	170.975	0.8672	0.0014	1.2745	4.6745	15.19



CIVIL ENGINEERING DEPARTMENT
GEOTECHNICAL ENGINEERING LABORATORY
GRADING ANALYSIS

PROJECT:-M.Sc. Thesis
SITE:-Gold Fields Ghana Limited, Tarkwa
GRADING TEST Total Dry Weight (g): 50

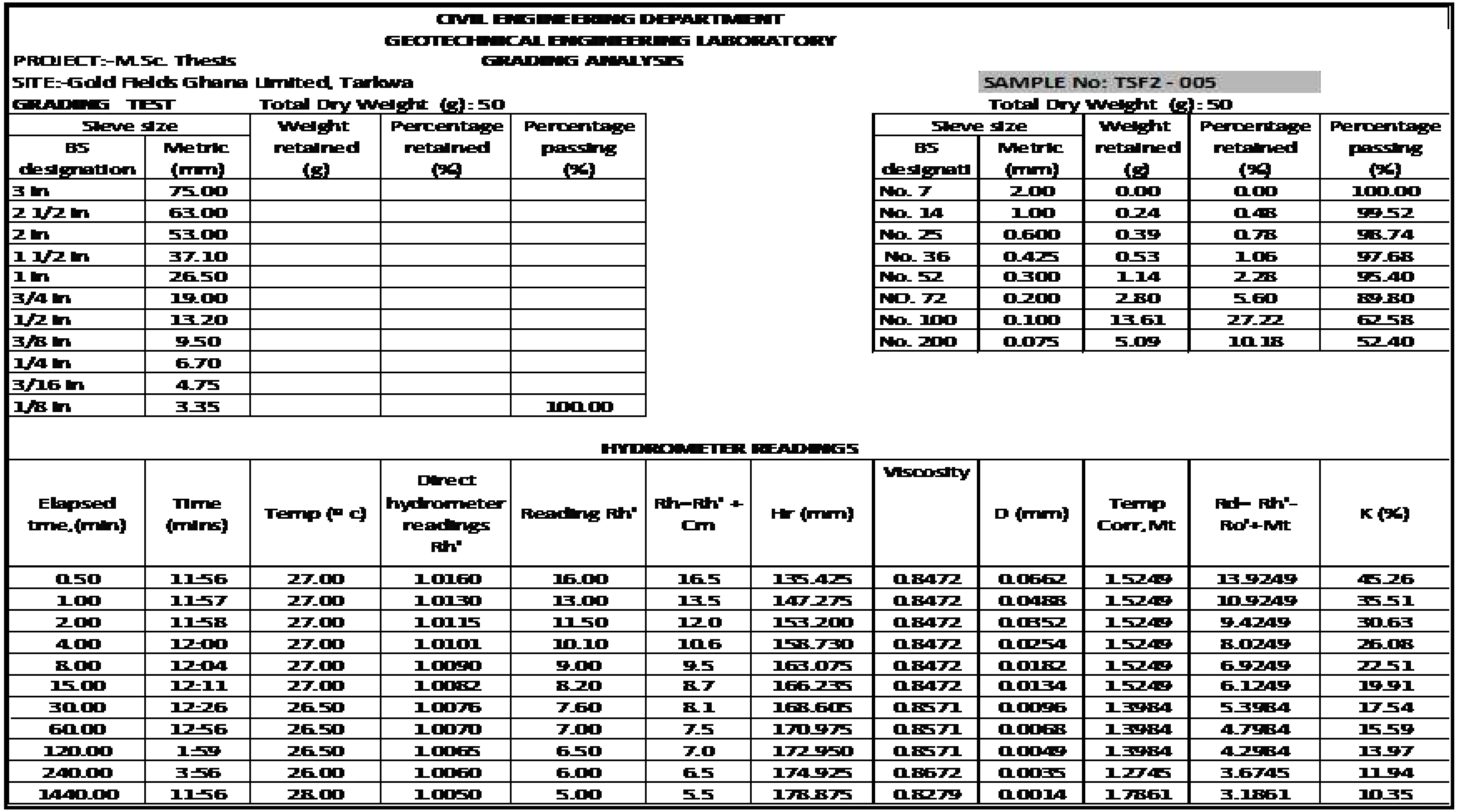
Sieve size		Weight retained (g)	Percentage retained (%)	Percentage passing (%)
BS designation	Metric (mm)			
3 in	75.00			
2 1/2 in	63.00			
2 in	53.00			
1 1/2 in	37.10			
1 in	26.50			
3/4 in	19.00			
1/2 in	13.20			
3/8 in	9.50			
1/4 in	6.70			
3/16 in	4.75			
1/8 in	3.35			100.00

SAMPLE No: TSF1 - 003
Total Dry Weight (g): 50

Sieve size		Weight retained (g)	Percentage retained (%)	Percentage passing (%)
BS designati	Metric (mm)			
No. 7	2.00	0.38	0.76	99.24
No. 14	1.00	0.40	0.80	98.44
No. 25	0.600	0.46	0.92	97.52
No. 36	0.425	0.65	1.30	96.22
No. 52	0.300	1.50	3.00	93.22
NO. 72	0.200	2.66	5.32	87.90
No. 100	0.100	10.53	21.06	66.84
No. 200	0.075	4.66	9.32	57.52

HYDROMETER READINGS

Elapsed time,(min)	Time (mins)	Temp (° c)	Direct hydrometer readings Rh'	Reading Rh'	Rh=Rh' + Cm	Hr (mm)	Viscosity	D (mm)	Temp Corr, Mt	Rd= Rh'-Ro'+Mt	K (%)
0.50	11:14	25.00	1.0195	19.50	20.0	121.600	0.8879	0.0643	1.0349	16.9349	55.04
1.00	11:15	25.00	1.0170	17.00	17.5	131.475	0.8879	0.0472	1.0349	14.4348	46.91
2.00	11:16	25.00	1.0145	14.50	15.0	141.350	0.8879	0.0346	1.0349	11.9349	38.79
4.00	11:18	25.00	1.0130	13.00	13.5	147.275	0.8879	0.0250	1.0349	10.4348	33.91
8.00	11:22	25.00	1.0115	11.50	12.0	153.200	0.8879	0.0180	1.0349	8.9349	29.04
15.00	11:29	25.00	1.0107	10.70	11.2	156.360	0.8879	0.0133	1.0349	8.1348	26.44
30.00	11:44	25.00	1.0100	10.00	10.5	159.125	0.8879	0.0095	1.0349	7.4349	24.16
60.00	12:14	25.00	1.0095	9.50	10.0	161.100	0.8879	0.0068	1.0349	6.9349	22.54
120.00	1:14	25.00	1.0090	9.00	9.5	163.075	0.8879	0.0048	1.0349	6.4348	20.91
240.00	3:14	24.50	1.0085	8.50	9.0	165.050	0.8985	0.0034	0.9190	5.8190	18.91
1440.00	11:14	26.00	1.0070	7.00	7.5	170.975	0.8672	0.0014	1.2745	4.6745	15.19



CIVIL ENGINEERING DEPARTMENT
GEOTECHNICAL ENGINEERING LABORATORY
GRADING ANALYSIS

PROJECT:-M.Sc. Thesis
SITE:-Gold Fields Ghana Limited, Tarkwa
GRADING TEST Total Dry Weight (g): 50

Sieve size		Weight retained (g)	Percentage retained (%)	Percentage passing (%)
BS designation	Metric (mm)			
3 in	75.00			
2 1/2 in	63.00			
2 in	53.00			
1 1/2 in	37.10			
1 in	26.50			
3/4 in	19.00			
1/2 in	13.20			
3/8 in	9.50			
1/4 in	6.70			
3/16 in	4.75			
1/8 in	3.35			100.00

SAMPLE No: TSF2 - 005
Total Dry Weight (g): 50

Sieve size		Weight retained (g)	Percentage retained (%)	Percentage passing (%)
BS designati	Metric (mm)			
No. 7	2.00	0.00	0.00	100.00
No. 14	1.00	0.24	0.48	99.52
No. 25	0.600	0.39	0.78	98.74
No. 36	0.425	0.53	1.06	97.68
No. 52	0.300	1.14	2.28	95.40
NO. 72	0.200	2.80	5.60	89.80
No. 100	0.100	13.61	27.22	62.58
No. 200	0.075	5.09	10.18	52.40

HYDROMETER READINGS

Elapsed time.(min)	Time (mins)	Temp (° c)	Direct hydrometer readings Rh'	Reading Rh'	Rh–Rh' + Cm	Hr (mm)	Viscosity	D (mm)	Temp Corr.Mt	Rd– Rh'-Ro'+Mt	K (%)
0.50	11:56	27.00	1.0160	16.00	16.5	135.425	0.8472	0.0662	1.5249	13.9249	45.26
1.00	11:57	27.00	1.0130	13.00	13.5	147.275	0.8472	0.0488	1.5249	10.9249	35.51
2.00	11:58	27.00	1.0115	11.50	12.0	153.200	0.8472	0.0352	1.5249	9.4249	30.63
4.00	12:00	27.00	1.0101	10.10	10.6	158.730	0.8472	0.0254	1.5249	8.0249	26.08
8.00	12:04	27.00	1.0090	9.00	9.5	163.075	0.8472	0.0182	1.5249	6.9249	22.51
15.00	12:11	27.00	1.0082	8.20	8.7	166.235	0.8472	0.0134	1.5249	6.1249	19.91
30.00	12:26	26.50	1.0076	7.60	8.1	168.605	0.8571	0.0096	1.3984	5.3984	17.54
60.00	12:56	26.50	1.0070	7.00	7.5	170.975	0.8571	0.0068	1.3984	4.7984	15.59
120.00	1:59	26.50	1.0065	6.50	7.0	172.950	0.8571	0.0049	1.3984	4.2984	13.97
240.00	3:56	26.00	1.0060	6.00	6.5	174.925	0.8672	0.0035	1.2745	3.6745	11.94
1440.00	11:56	28.00	1.0050	5.00	5.5	178.875	0.8279	0.0014	1.7861	3.1861	10.35

CIVIL ENGINEERING DEPARTMENT
GEOTECHNICAL ENGINEERING LABORATORY
GRADING ANALYSIS

PROJECT:-M.Sc. Thesis
SITE:-Gold Fields Ghana Limited, Tarkwa

SAMPLE No: TSF3 - 009

GRADING TEST Total Dry Weight (g): 50

Sieve size		Weight retained (g)	Percentage retained (%)	Percentage passing (%)
BS designation	Metric (mm)			
3 in	75.00			
2 1/2 in	63.00			
2 in	53.00			
1 1/2 in	37.10			
1 in	26.50			
3/4 in	19.00			
1/2 in	13.20			
3/8 in	9.50			
1/4 in	6.70			
3/16 in	4.75			
1/8 in	3.35			100.00

Total Dry Weight (g): 50

Sieve size		Weight retained (g)	Percentage retained (%)	Percentage passing (%)
BS designati	Metric (mm)			
No. 7	2.00	0.00	0.00	100.00
No. 14	1.00	0.17	0.34	99.66
No. 25	0.600	0.19	0.38	99.28
No. 36	0.425	0.21	0.42	98.86
No. 52	0.300	0.40	0.80	98.06
NO. 72	0.200	0.88	1.76	96.30
No. 100	0.100	12.12	24.24	72.06
No. 200	0.075	7.97	15.94	56.12

ROMETER READINGS

HYD

Elapsed tme,(min)	Time (mins)	Temp (º c)	Direct hydrometer readings Rh'	Reading Rh'	Rh=Rh' + Cm	Hr (mm)	Viscosity	D (mm)	Temp Corr,Mt	Rd= Rh'Ro'+Mt	K (%)
0.50	11:39	24.50	1.0190	19.00	19.5	123.575	0.8985	0.0652	0.9190	16.3190	53.04
1.00	11:40	24.50	1.0155	15.50	16.0	137.400	0.8985	0.0486	0.9190	12.8190	41.66
2.00	11:41	24.50	1.0124	12.40	12.9	149.645	0.8985	0.0359	0.9190	9.7190	31.59
4.00	11:43	24.50	1.0105	10.50	11.0	157.150	0.8985	0.0260	0.9190	7.8190	25.41
8.00	11:47	24.50	1.0090	9.00	9.5	163.075	0.8985	0.0187	0.9190	6.3190	20.54
15.00	11:54	24.50	1.0086	8.60	9.1	164.655	0.8985	0.0137	0.9190	5.9190	19.24
30.00	12:09	24.50	1.0079	7.90	8.4	167.420	0.8985	0.0098	0.9190	5.2190	16.96
60.00	12:39	24.50	1.0070	7.00	7.5	170.975	0.8985	0.0070	0.9190	4.3190	14.04
120.00	1:39	24.50	1.0068	6.80	7.3	171.765	0.8985	0.0050	0.9190	4.1190	13.39
240.00	3:39	24.50	1.0065	6.50	7.0	172.950	0.8985	0.0035	0.9190	3.8190	12.41
1440.00	11:39	26.50	1.0056	5.60	6.1	176.505	0.8571	0.0014	1.3984	3.3984	11.04

CIVIL ENGINEERING DEPARTMENT
GEOTECHNICAL ENGINEERING LABORATORY
GRADING ANALYSIS

PROJECT:-M.Sc. Thesis
SITE:-Gold Fields Ghana Limited, Tarkwa
GRADING TEST **Total Dry Weight (g): 50**

Sieve size		Weight retained (g)	Percentage retained (%)	Percentage passing (%)
BS designation	Metric (mm)			
3 in	75.00			
2 1/2 in	63.00			
2 in	53.00			
1 1/2 in	37.10			
1 in	26.50			
3/4 in	19.00			
1/2 in	13.20			
3/8 in	9.50			
1/4 in	6.70			
3/16 in	4.75			
1/8 in	3.35			100.00

SAMPLE No: TSF3 - 013
Total Dry Weight (g): 50

Sieve size		Weight retained (g)	Percentage retained (%)	Percentage passing (%)
BS designati	Metric (mm)			
No. 7	2.00	0.00	0.00	100.00
No. 14	1.00	0.34	0.68	99.32
No. 25	0.600	0.27	0.54	98.78
No. 36	0.425	0.21	0.42	98.36
No. 52	0.300	0.37	0.74	97.62
NO. 72	0.200	1.09	2.18	95.44
No. 100	0.100	12.05	24.10	71.34
No. 200	0.075	6.38	12.76	58.58

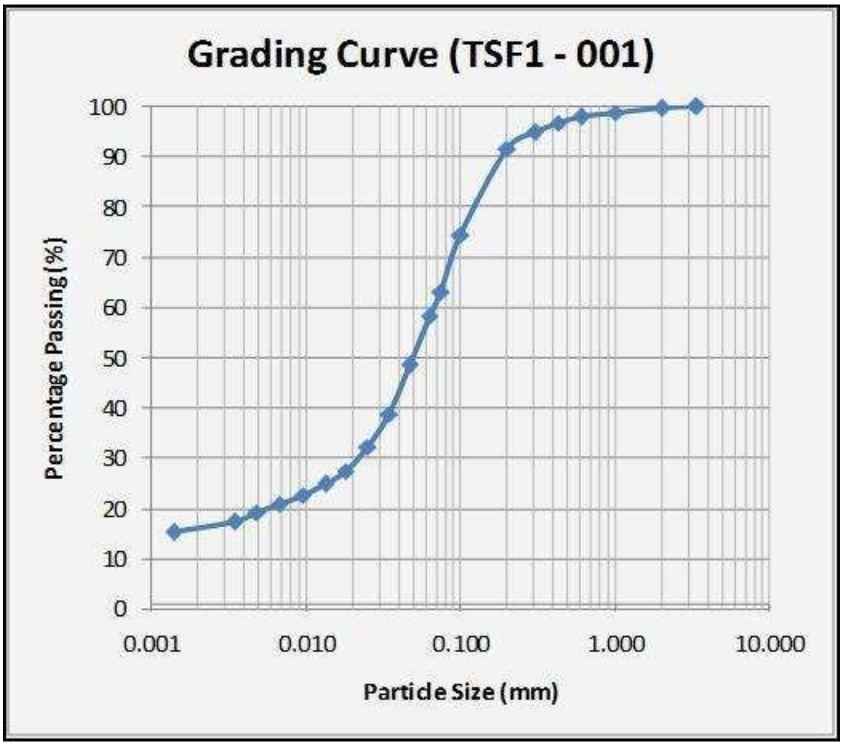
HYDROMETER READINGS

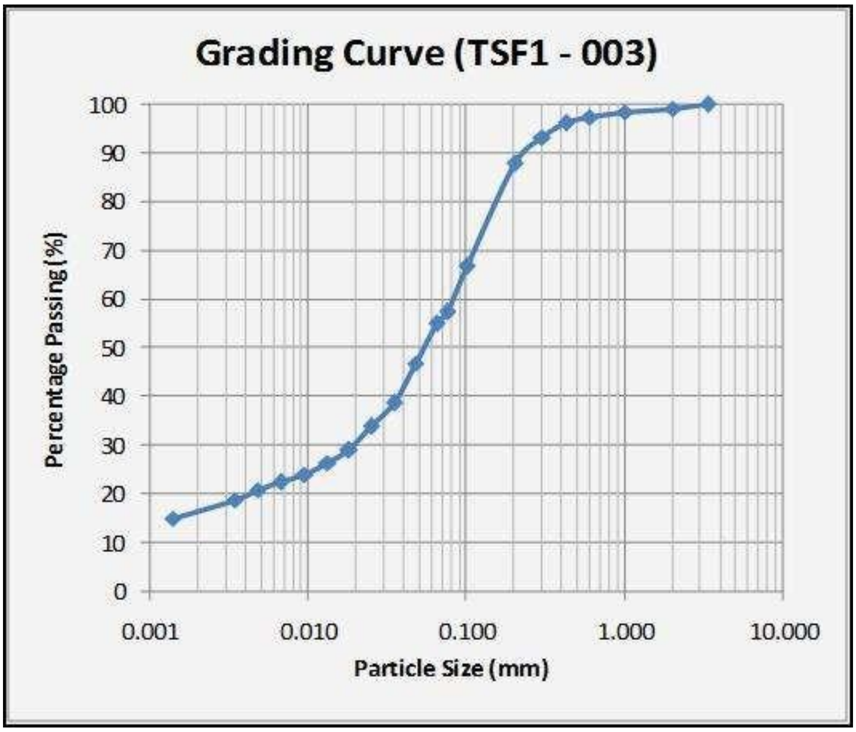
Elapsed time,(min)	Time (mins)	Temp (° c)	Direct hydrometer readings Rh'	Reading Rh'	Rh-Rh' + Cm	Hr (mm)	Viscosity	D (mm)	Temp Corr,Mt	Rd= Rh'- Ro'+Mt	K (%)
0.50	11:39	27.00	1.0190	19.00	19.5	123.575	0.8472	0.0633	1.5249	16.9249	55.01
1.00	11:40	27.00	1.0160	16.00	16.5	135.425	0.8472	0.0468	1.5249	13.9249	45.26
2.00	11:41	27.00	1.0135	13.50	14.0	145.300	0.8472	0.0343	1.5249	11.4249	37.13
4.00	11:43	27.00	1.0115	11.50	12.0	153.200	0.8472	0.0249	1.5249	9.4249	30.63
8.00	11:47	27.00	1.0105	10.50	11.0	157.150	0.8472	0.0178	1.5249	8.4249	27.38
15.00	11:54	27.00	1.0100	10.00	10.5	159.125	0.8472	0.0131	1.5249	7.9249	25.76
30.00	12:09	27.00	1.0095	9.50	10.0	161.100	0.8472	0.0093	1.5249	7.4249	24.13
60.00	12:39	26.50	1.0089	8.90	9.4	163.470	0.8571	0.0067	1.3984	6.6984	21.77
120.00	1:39	26.00	1.0084	8.40	8.9	165.445	0.8672	0.0048	1.2745	6.0745	19.74
240.00	3:39	26.00	1.0080	8.00	8.5	167.025	0.8672	0.0034	1.2745	5.6745	18.44
1440.00	11:39	27.50	1.0070	7.00	7.5	170.975	0.8375	0.0014	1.6542	5.0542	16.43

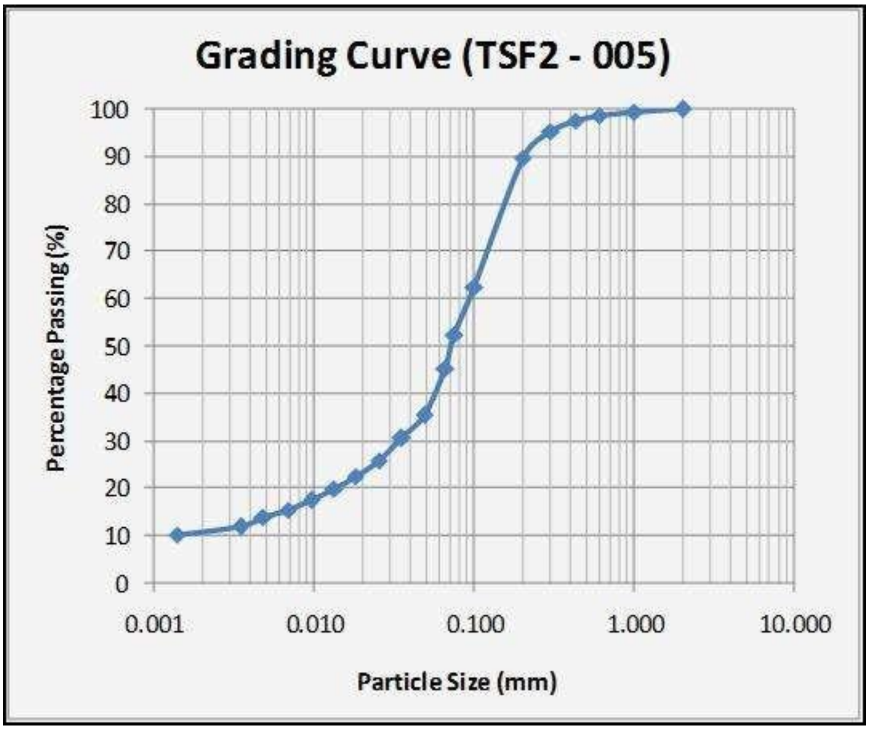
APPENDIX 5

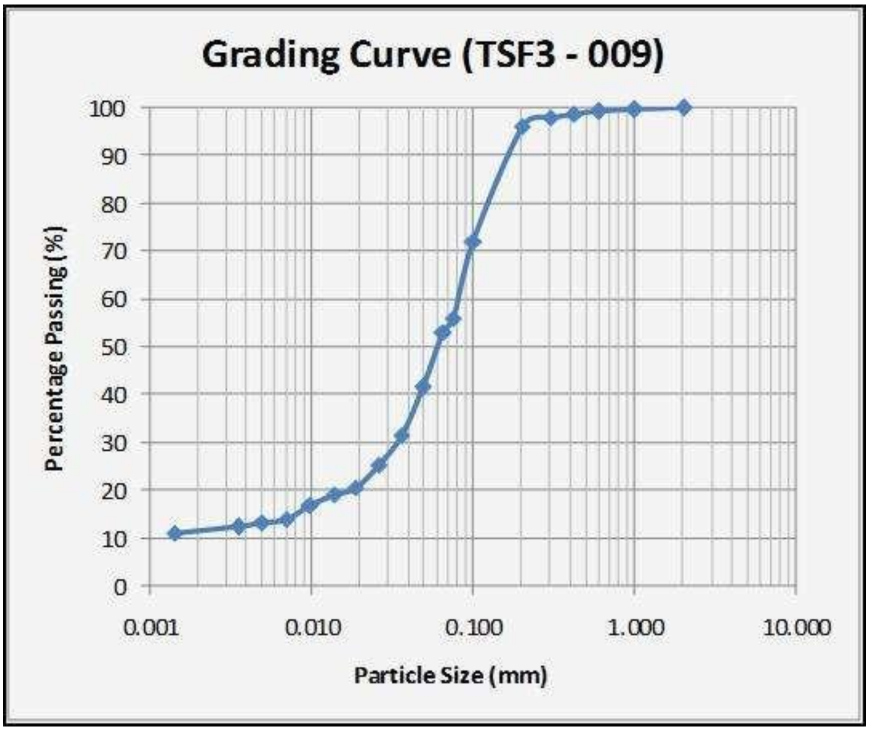
Grading Curves of Soil Samples in the TSF Basin

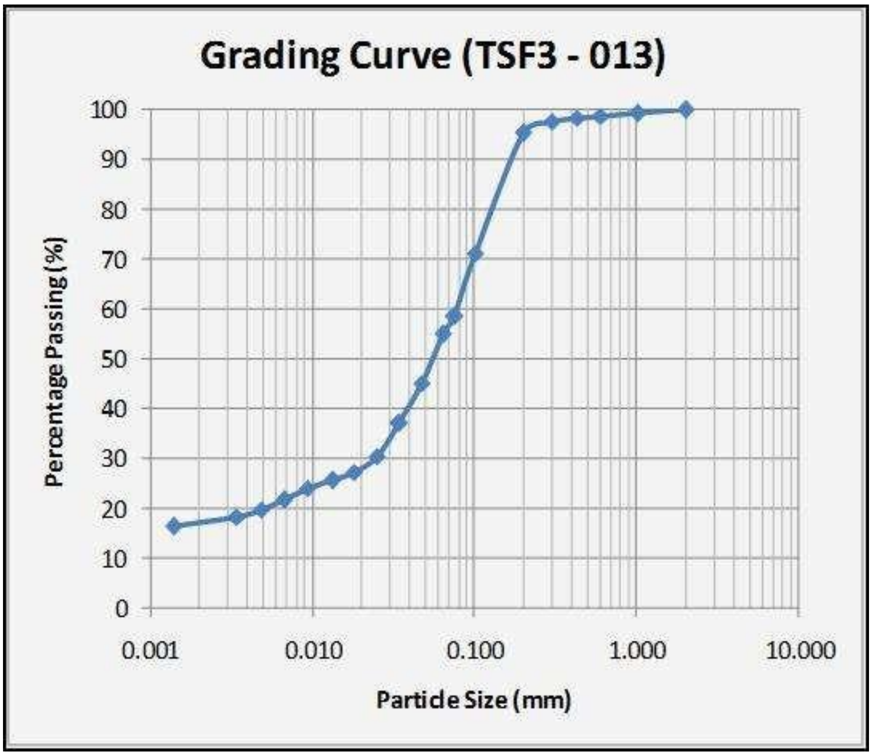
Grading Curves of Soil Samples in the TSF Basin



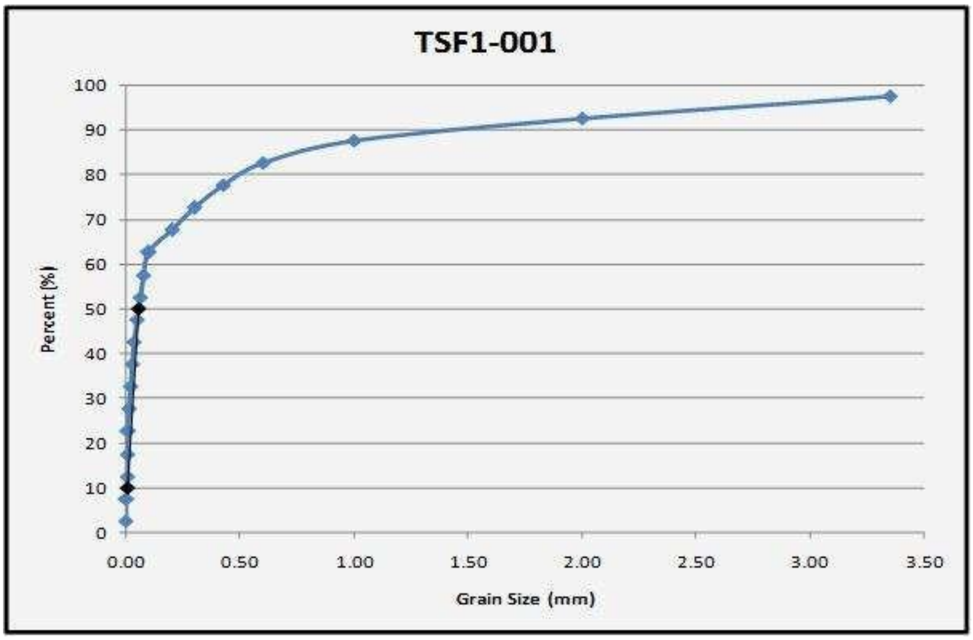


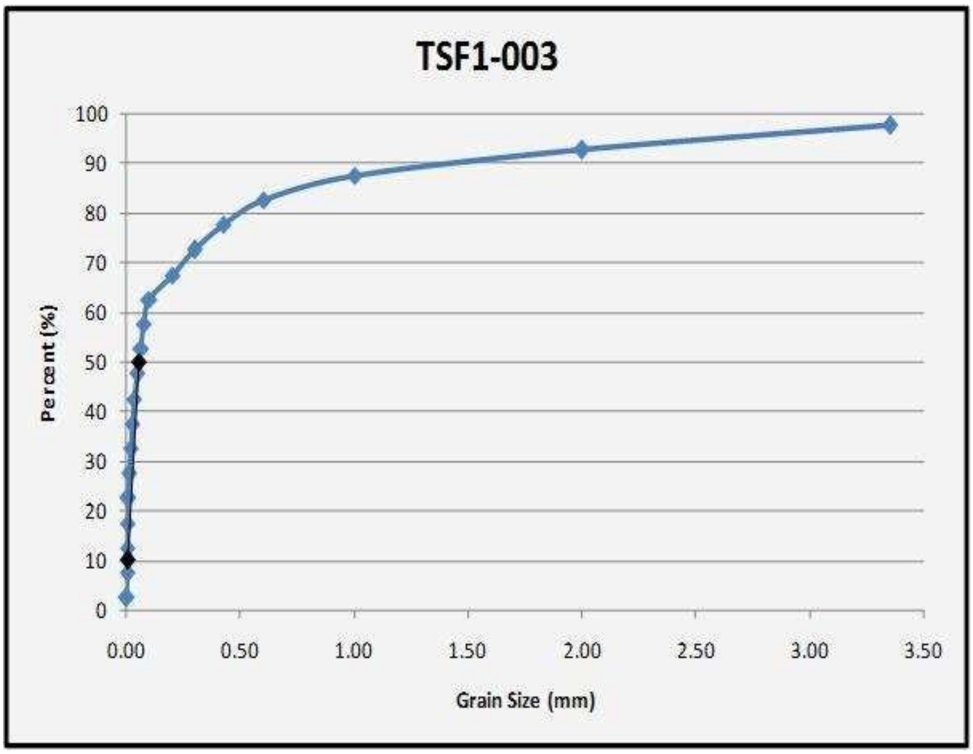

Grading Curve (TSF2 - 005)
Percentage Passing (%)
100
90
80
70
60
50
40
30
20
10
0
0.001
0.010
0.100
1.000
10.000
Particle Size (mm)

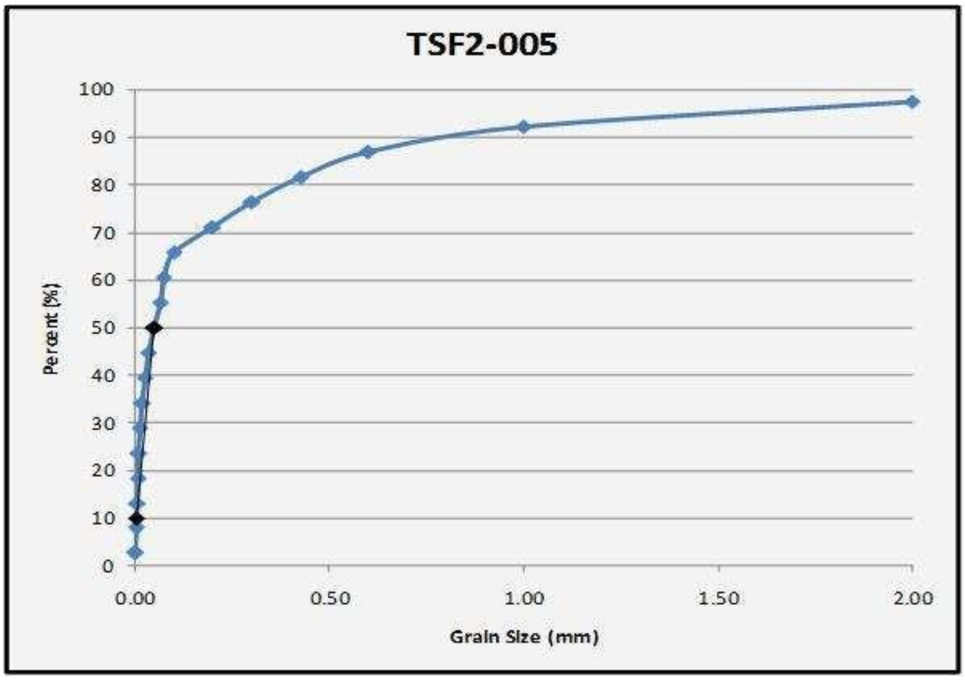

Grading Curve (TSF3 - 009)
Percentage Passing (%)
100
90
80
70
60
50
40
30
20
10
0
0.001
0.010
0.100
1.000
10.000
Particle Size (mm)

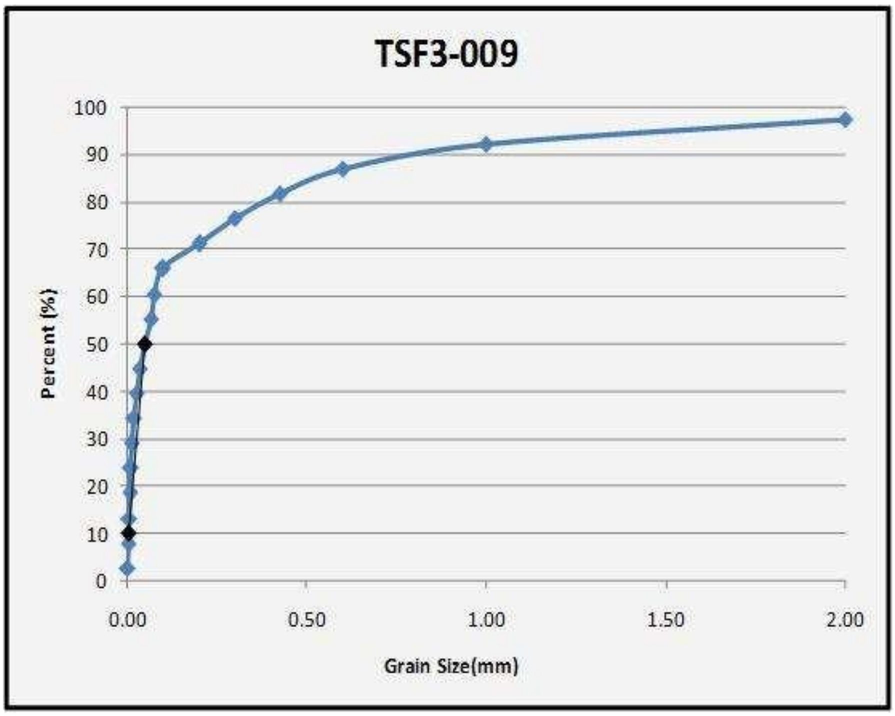


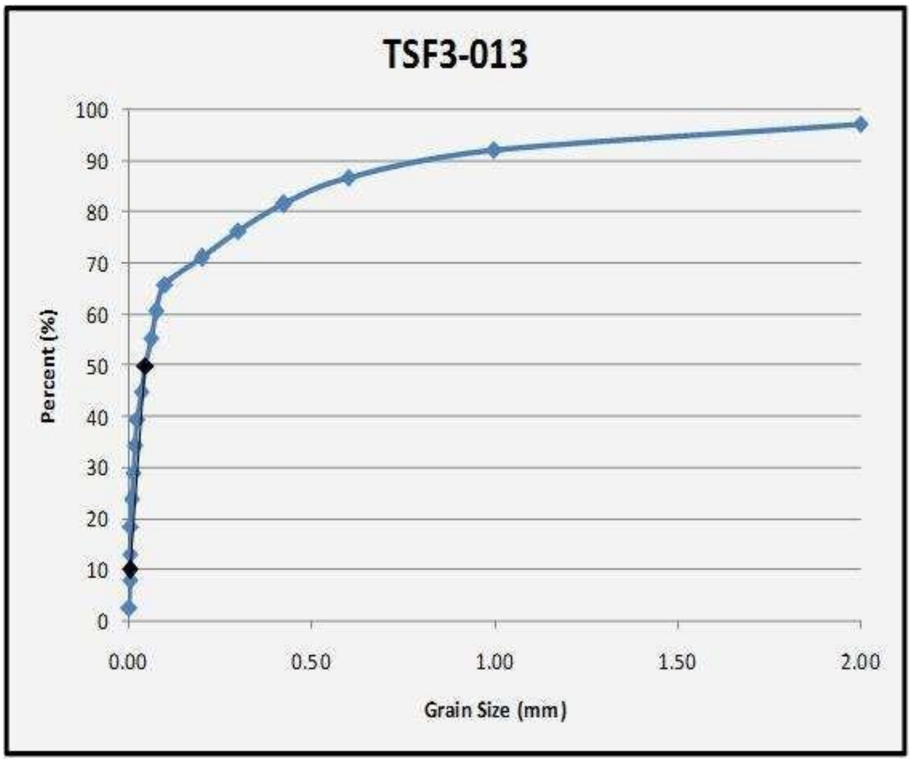
Grading curves of soil samples plotted on a normal scale




TSF1-003
Percent (%)
100
90
80
70
60
50
40
30
20
10
0
0.00
0.50
1.00
1.50
2.00
2.50
3.00
3.50
Grain Size (mm)


TSF2-005
Percent (%)
100
90
80
70
60
50
40
30
20
10
0
0.00
0.50
1.00
1.50
2.00
Grain Size (mm)


TSF3-009
Percent (%)
100
90
80
70
60
50
40
30
20
10
0
0.00
0.50
1.00
1.50
2.00
Grain Size(mm)


TSF3-013
Percent (%)
100
90
80
70
60
50
40
30
20
10
0
0.00
0.50
1.00
1.50
2.00
Grain Size (mm)

APPENDIX 6

Statistical analysis of decant pond physico-chemical parameters

Sample ID	Date	Station ID	pH	Ec	TDS	Ca	Mg	Fe	Cu	Cd	Mn	Pb	Zn	Na	Mg	K	Ag	SO4	Cl	As	NO3	NO2
				µS/cm	Mg/l	Mg/l	Mg/l	Mg/l	Mg/l	Mg/l	Mg/l	Mg/l	Mg/l	Mg/l	Mg/l	Mg/l	Mg/l	Mg/l	Mg/l	Mg/l	Mg/l	Mg/l
PAE0601	12-Jun-10	1	10.28	1091	740			2.600	0.49	<0.002	0.18	<0.01	<0.05					40	41.4	0.022	94.3	187
PAE0602	12-Jun-10	2	10.39	1095	746			2.600	0.51	<0.002	0.22	0.01	<0.05					39	40.7	0.022	97.7	191
PAE0603	12-Jun-10	3	10.51	1113	756			3.800	0.71	<0.002	0.44	<0.01	<0.05					40	42.1	0.022	102	199
PAE0604	12-Jun-10	4	10.42	1096	746			3.300	0.64	<0.002	0.32	<0.01	<0.05					39	40.9	0.022	98.3	192
PAE0605	12-Jun-10	5	10.35	1101	748			3.200	0.6	<0.002	0.27	<0.01	<0.05					38	40	0.022	91.6	193
PAE0606	12-Jun-10	6	10.40	1098	750			3.700	0.7	<0.002	0.35	<0.01	<0.05					39	39	0.034	94.6	193
PAE0607	12-Jun-10	7	10.49	1102	749			3.100	0.58	<0.002	0.25	<0.01	<0.05					38	39.8	0.032	93.7	195
PAE0608	12-Jun-10	8	10.49	1097	745			3.500	0.67	<0.002	0.38	<0.01	<0.05					39	39	0.038	94.3	192
PAM0601	12-Jun-10	1	10.63	1096	963			3.000	0.65	<0.002	0.36	<0.01	<0.05					38	38.4	0.034	76.8	184
PAM0602	12-Jun-10	2	10.50	1106	969			2.800	0.69	<0.002	0.36	<0.01	<0.05					38	39.5	0.03	77.4	189
PAM0603	12-Jun-10	3	10.78	1104	968			3.400	0.73	<0.002	0.45	<0.01	<0.05					40	41.5	0.032	70	199
PAM0604	12-Jun-10	4	10.56	1107	971			3.200	0.63	<0.002	0.35	<0.01	<0.05					40	39	0.032	81	188
PAM0605	12-Jun-10	5	10.81	1114	977			3.000	0.56	<0.002	0.34	<0.01	<0.05					40	39.8	0.032	82.2	194
PAM0606	12-Jun-10	6	10.76	1111	976			3.000	0.58	<0.002	0.29	<0.01	<0.05					39	39.7	0.034	75.7	196
PAM0607	12-Jun-10	7	10.64	1104	971			2.700	0.54	<0.002	0.22	<0.01	<0.05					39	39	0.032	75.2	189
PBE0601	12-Jun-10	1	10.42	1095	745			2.800	0.55	<0.002	0.23	<0.01	<0.05					39	39.6	0.032	81	193

PBE0602	12-Jun-10	2	10.37	1098	745			2.700	0.55	<0.002	0.21	<0.01	<0.05					38	39.8	0.032	91.8	193
PBE0603	12-Jun-10	3	10.28	1098	746			2.800	0.58	<0.002	0.24	<0.01	<0.05					39	41	0.032	97.1	190
PBE0604	12-Jun-10	4	10.43	1095	744			3.000	0.58	<0.002	0.25	<0.01	<0.05					39	41.4	0.03	96.8	192
PBE0605	12-Jun-10	5	10.47	1110	754			3.100	0.67	<0.002	0.32	<0.01	<0.05					38	39..5	0.03	NA	
PBE0606	12-Jun-10	6	10.53	1100	752			2.300	0.55	<0.002	0.2	<0.01	<0.05					39	39.7	0.032	83.7	196
PAE0701	09-Jul-10	1	9.72	1068	703	12.00	0.8	2.200	0.72	<0.002	0.55	<0.01	<0.05									
PAE0702	09-Jul-10	2	9.72	1066	703	12.00	0.8	1.500	0.54	<0.002	0.26	<0.01	<0.05									
PAE0703	09-Jul-10	3	9.74	1077	709	12.00	<0.5	3.000	1	<0.002	0.92	<0.01	<0.05									
PAE0704	09-Jul-10	4	9.72	1081	712	13.00	<0.5	2.000	0.54	<0.002	0.43	<0.01	<0.05									
PAE0705	09-Jul-10	5	9.76	1083	716	11.00	<0.5	1.500	0.5	<0.002	0.24	<0.01	<0.05									
PAE0706	09-Jul-10	6	9.79	1079	711	12.00	<0.5	2.000	0.68	<0.002	0.46	<0.01	<0.05									
PAE0707	09-Jul-10	7	9.76	1079	710	12.00	<0.5	1.900	0.66	<0.002	0.41	<0.01	<0.05									
PAE0708	09-Jul-10	8	9.76	1080	713	12.00	<0.5	2.100	0.51	<0.002	0.29	<0.01	<0.05									
PCE0701	09-Jul-10	1	9.82	1009	665	15.00	<0.5	1.600	0.66	<0.002	0.37	<0.01	<0.05									
PCE0702	09-Jul-10	2	9.86	1003	664	16.00	<0.5	1.600	0.65	<0.002	0.35	<0.01	<0.05									
PCE0703	09-Jul-10	3	9.92	1018	671	16.00	<0.5	1.400	0.59	<0.002	0.31	<0.01	<0.05									
PCE0704	09-Jul-10	4	9.96	1019	670	16.00	0.6	1.600	0.58	<0.002	0.41	<0.01	<0.05									
PCE0705	09-Jul-10	5	9.96	1016	670	16.00	0.5	1.800	0.59	<0.002	0.45	<0.01	<0.05									

PAM0801	07-Aug-10	1	9.78	1204	757	14.00	<0.5	4.000	0.79	<0.002	1.13	<0.01	<0.05	178.253								
PAM0802	07-Aug-10	2	9.85	1198	753	16.00	<0.5	2.000	0.49	<0.002	0.39	<0.01	<0.05	175.153								
PAM0803	07-Aug-10	3	9.84	1201	754	16.00	<0.5	1.900	0.47	<0.002	0.36	<0.01	<0.05	180.245								
PAM0804	07-Aug-10	4	9.83	1203	756	15.00	0.6	1.800	0.46	<0.002	0.3	<0.01	<0.05	181.215								
PAM0805	07-Aug-10	5	9.83	1203	756	16.00	0.5	2.600	0.59	<0.002	0.58	<0.01	<0.05	182.154								
PAM0806	07-Aug-10	6	9.85	1202	756	16.00	0.6	2.000	0.47	<0.002	0.33	<0.01	<0.05	183.654								
PAM0807	07-Aug-10	7	9.85	1199	755	17.00	0.8	2.300	0.53	0.007	0.5	<0.01	<0.05	183.968								
PAM0808	07-Aug-10	8	9.83	1195	751	16.00	0.7	1.900	0.45	<0.002	0.31	<0.01	<0.05	185.265								
PBM0801	07-Aug-10	1	9.83	1199	754	14.00	<0.5	1.700	0.4	<0.002	0.27	<0.01	<0.05	174.653								
PBM0802	07-Aug-10	2	9.84	1198	753	15.00	<0.5	1.900	0.47	<0.002	0.34	<0.01	<0.05	174.965								
PBM0803	07-Aug-10	3	9.84	1203	757	15.00	<0.5	1.700	0.41	<0.002	0.24	<0.01	<0.05	182.145								
PBM0804	07-Aug-10	4	9.86	1202	756	16.00	<0.5	1.800	0.45	<0.002	0.3	<0.01	<0.05	182.782								
PBM0805	07-Aug-10	5	9.85	1201	756	16.00	<0.5	2.900	0.67	<0.002	0.72	<0.01	<0.05	183.254								
PBM0806	07-Aug-10	6	9.85	1202	756	16.00	<0.5	2.300	0.54	<0.002	0.55	<0.01	<0.05	183.965								
PBM0807	07-Aug-10	7	9.85	1202	756	16.00	0.6	2.000	0.51	<0.002	0.36	<0.01	<0.05	185.263								
PCM0801	07-Aug-10	1	10.00	1192	750	9.00	<0.5	1.400	0.48	<0.002	0.25	<0.01	<0.05	183.245								
PCM0802	07-Aug-10	2	10.04	1190	748	11.00	<0.5	1.600	0.66	<0.002	0.36	<0.01	<0.05	183.786								
PCM0803	07-Aug-10	3	10.15	1198	753	11.00	0.5	1.500	0.66	<0.002	0.31	<0.01	<0.05	184.142								

PCM0804	07-Aug-10	4	10.14	1200	755	10.00	<0.5	1.800	0.7	<0.002	0.43	<0.01	<0.05	184.562								
PCM0805	07-Aug-10	5	10.23	1207	759	8.00	<0.5	2.700	0.97	<0.002	0.74	<0.01	<0.05	186.254								
PCM0806	07-Aug-10	6	10.22	1206	759	9.00	<0.5	2.000	0.56	<0.002	0.44	<0.01	<0.05	186.652								
PAM-1	13-Dec-10	1	9.98	1241	866	6.818	0.515	0.989	0.877	0.002	0.878	0.121	0.032	182.443	0.515	21.331	0.010	32	5.4		307.0	38.64
PAM-2	13-Dec-10	2	9.82	1248	860	7.021	0.501	0.968	0.958	0.002	0.855	<0.002	0.024	184.098	0.501	21.218	0.018	31	5.0		313.0	44.90
PAM-3	13-Dec-10	3	9.95	1219	848	8.002	0.528	1.066	1.313	<0.002	1.068	<0.002	0.028	181.656	0.528	21.438	0.012	37	4.4		305.5	44.10
PAM-4	13-Dec-10	4	9.48	1207	851	7.901	0.489	1.460	1.110	<0.002	1.022	<0.002	0.021	181.882	0.489	18.746	0.004	38	6.2		366.5	37.45
PAM-5	13-Dec-10	5	9.96	1204	852	7.834	0.466	1.753	1.258	<0.002	1.515	<0.002	0.027	183.287	0.466	21.744	0.006	34	5.4		381.0	39.98
PAM-6	13-Dec-10	6	10.04	1216	843	7.989	0.530	2.208	1.264	0.002	1.585	<0.002	0.031	183.544	0.530	22.002	0.007	30	4.6		328.0	36.56
PAM-7	13-Dec-10	7	10.08	1260	848	8.772	0.499	2.928	1.272	<0.002	2.066	0.088	0.023	185.656	0.499	21.313	0.016	36	4.6		364.0	41.60
PAM-8	13-Dec-10	8	9.98	1224	836	8.434	0.554	2.211	1.261	<0.002	2.428	0.100	0.044	186.858	0.554	21.770	0.004	34	5.0		339.0	38.15
PBM-1	13-Dec-10	1	9.83	1188	844	6.223	0.320	0.645	1.008	<0.002	0.438	0.006	0.030	180.944	0.320	19.884	0.004	18	6.3		298.1	37.90
PBM-2	13-Dec-10	2	9.88	1196	840	6.168	0.288	0.598	0.923	<0.002	0.313	0.002	0.036	182.668	0.288	20.123	0.006	23	3.9		266.5	37.45
PBM-3	13-Dec-10	3	9.87	1196	838	5.217	0.268	0.555	1.042	0.003	0.713	0.068	0.019	183.008	0.268	20.284	0.011	27	4.7		261.5	42.60
PBM-4	13-Dec-10	4	10.09	1201	842	5.044	0.244	0.761	1.154	<0.002	0.957	<0.002	0.023	185.228	0.244	19.541	0.002	27	4.1		238.5	38.01
PBM-5	13-Dec-10	5	10.11	1190	846	5.102	0.267	0.748	1.203	<0.002	0.978	0.008	0.018	186.844	0.267	20.112	0.004	24	5.8		332.0	39.55
PBM-6	13-Dec-10	6	10.21	1210	841	6.011	0.282	1.006	1.248	0.004	1.108	<0.002	0.017	188.244	0.282	18.998	<0.002	26	6.0		220.0	44.30
PAM-1	23-Mar-11	1	9.481	1214	757.000	7.108	0.427	0.920	0.992	<0.002	0.833	0.296	0.019	181.639	0.427	20.831	0.005					

PAM-2	23-Mar-11	2	9.917	1203	753.000	7.277	0.434	0.942	0.962	0.002	0.820	<0.002	0.018	182.871	0.434	21.011	0.022					
PAM-3	23-Mar-11	3	9.954	1210	754.000	7.892	0.458	1.269	1.296	<0.002	1.168	<0.002	0.027	185.843	0.458	21.533	0.019					
PAM-4	23-Mar-11	4	9.983	1211	756.000	7.263	0.439	1.680	1.073	<0.002	1.122	<0.002	0.019	184.736	0.439	17.574	<0.002					
PAM-5	23-Mar-11	5	9.729	1211	756.000	8.268	0.477	1.821	1.300	<0.002	1.525	<0.002	0.027	185.712	0.477	21.979	0.005					
PAM-6	23-Mar-11	6	9.923	1210	756.000	8.313	0.499	2.355	1.311	<0.002	1.708	<0.002	0.033	188.584	0.499	21.810	0.010					
PAM-7	23-Mar-11	7	9.789	1207	755.000	8.878	0.502	3.027	1.312	<0.002	2.021	0.227	0.025	188.630	0.502	21.193	0.022					
PAM-8	23-Mar-11	8	9.872	1209	751.000	8.844	0.620	2.186	1.278	<0.002	1.488	0.330	0.048	188.707	0.620	21.870	<0.002					
PBM-1	23-Mar-11	1	9.957	1210	754.000	5.099	0.216	0.596	0.946	<0.002	0.624	0.017	0.020	160.809	0.216	19.361	<0.002					
PBM-2	23-Mar-11	2	9.948	1203	753.000	5.268	0.223	0.550	0.881	<0.002	0.264	<0.002	0.029	161.025	0.223	19.307	<0.002					
PBM-3	23-Mar-11	3	9.913	1212	757.000	5.217	0.217	0.582	0.962	<0.002	0.655	0.272	0.013	165.703	0.217	20.142	0.024					
PBM-4	23-Mar-11	4	9.954	1212	756.000	4.883	0.239	0.770	1.130	<0.002	0.943	<0.002	0.023	168.409	0.239	19.169	<0.002					
PBM-5	23-Mar-11	5	9.743	1211	756.000	4.947	0.245	0.769	1.195	<0.002	0.961	0.153	0.015	168.510	0.245	19.892	0.023					
PBM-6	23-Mar-11	6	9.957	1210	756.000	5.103	0.261	0.977	1.252	0.007	1.088	<0.002	0.022	177.600	0.261	19.683	<0.002					
PBM-7	23-Mar-11	7	9.689	1209	756.000	21.321	0.852	3.091	1.940	<0.002	12.827	0.087	0.077	161.084	0.852	22.733	0.017					
PAM08-1	28/Jul/11	1	9.60	880	560	14.748	0.366	0.541	0.339	<0.002	0.374	0.009	0.037	185.193	0.366	28.231	<0.002					
PAM08-2	28/Jul/11	2	9.63	882	559	12.198	0.379	0.395	0.324	<0.002	0.347	<0.002	0.024	186.412	0.379	32.244	0.019					
PAM08-3	28/Jul/11	3	9.63	881	562	13.275	0.426	0.373	0.249	<0.002	0.298	0.025	0.041	189.485	0.426	32.646	0.002					
PCB08-1	28/Jul/11	1	9.91	881	563	10.738	0.152	0.233	0.487	<0.002	0.218	<0.002	0.062	171.601	0.152	30.408	0.012					

PCB08-2	28/Jul/11	2	9.85	882	562	11.788	0.204	0.784	1.147	<0.002	0.716	0.008	0.036	172.761	0.204	34.071	0.010					

Statistical Analysis - pH

pH	Count	Mean	SE Mean	ST Dev	Min	Median	Max	Range	S. Var	Confidence Level	Skewness	Sum
Depth 1	14	9.945	0.083463	0.312292	9.481	9.871	10.629	1.148	0.097526	0.180311904	0.887323	139.23
Depth 2	14	9.971857	0.069954	0.261743	9.626	9.8695	10.497	0.871	0.068510	0.151126039	1.052499	139.606
Depth 3	13	10.02754	0.089213	0.321661	9.626	9.916	10.775	1.149	0.103466	0.194377926	1.293656	130.358
Depth 4	12	10.03525	0.090872	0.314788	9.48	9.969	10.563	1.083	0.099092	0.200006929	0.169506	120.423
Depth 5	12	10.06708	0.097509	0.337782	9.729	9.9615	10.81	1.081	0.114097	0.214616567	1.058347	120.805
Depth 6	11	10.13918	0.095085	0.31536	9.792	10.042	10.76	0.968	0.099452	0.211861767	0.808137	111.531
Depth 7	8	10.01775	0.126951	0.359071	9.689	9.849	10.643	0.954	0.128932	0.300190472	1.128394	80.142
Depth 8	5	9.9856	0.130195	0.291125	9.759	9.872	10.487	0.728	0.084754	0.361479505	1.847383	49.928

Statistical Analysis - EC

Ec	Count	Mean	SE Mean	ST Dev	Min	Median	Max	Range	S. Var	Confidence Level	Skewness	Sum
Depth 1	14	1112	31.93176	119.4777	880	1142	1241	361	14274.92	68.98437568	-1.02462	15568
Depth 2	14	1112	31.73621	118.746	882	1148	1248	366	14100.62	68.56190614	-1.03831	15568

Depth 3	13	1133.077	27.69047	99.83942	881	1196	1219	338	9967.91	60.33235911	-1.48784	14730
Depth 4	12	1152.833	19.66455	68.11999	1019	1200.5	1212	193	4640.333	43.28137813	-0.76493	13834
Depth 5	12	1154.25	19.09749	66.15564	1016	1195.5	1211	195	4376.568	42.0332858	-0.9072	13851
Depth 6	11	1167.636	17.06792	56.6079	1079	1202	1216	137	3204.455	38.02970589	-0.70498	12844
Depth 7	8	1170.25	23.19001	65.59127	1079	1200.5	1260	181	4302.214	54.83567135	-0.32532	9362
Depth 8	5	1161	30.07158	67.2421	1080	1195	1224	144	4521.5	83.49209462	-0.53947	5805

Statistical Analysis - TDS

TDS	Count	Mean	SE Mean	ST Dev	Min	Median	Max	Range	S. Var	Confidence Level	Skewness	Sum
Depth 1	14	744.3571	28.51385	106.6891	560	752	963	403	11382.55	61.60042771	0.032967	10421
Depth 2	14	743.4286	28.61908	107.0828	559	750.5	969	410	11466.73	61.82776587	0.08432	10408
Depth 3	13	759.4615	26.20986	94.50098	562	754	968	406	8930.436	57.10637284	0.201233	9873
Depth 4	12	776.25	22.5802	78.22011	670	756	971	301	6118.386	49.6986899	1.486116	9315
Depth 5	12	778.8333	22.85887	79.18544	670	756	977	307	6270.333	50.31202595	1.502716	9346
Depth 6	11	786.9091	22.31406	74.00737	711	756	976	265	5477.091	49.71882811	1.937508	8656
Depth 7	8	787.5	29.55564	83.59597	710	755.5	971	261	6988.286	69.88797745	1.873676	6300
Depth 8	5	759.2	20.45581	45.74057	713	751	836	123	2092.2	56.7944225	1.525659	3796

Statistical Analysis - Fe

Fe	Count	Mean	SE Mean	ST Dev	Min	Median	Max	Range	S. Var	Confidence Level	Skewness	Sum
Depth 1	14	1.658857	0.298967	1.118634	0.2330	1.5000	4.0000	3.767	1.251341	0.645879792	0.670745	23.224
Depth 2	14	1.4955	0.220064	0.823402	0.3950	1.5500	2.8000	2.405	0.677992	0.475418387	0.310924	20.937
Depth 3	13	1.7958	0.31174	1.123994	0.3730	1.5000	3.8000	3.427	1.263364	0.679223049	0.536647	23.345
Depth 4	12	1.9309	0.243004	0.841792	0.7610	1.8000	3.3000	2.539	0.708614	0.5348491	0.457813	23.171
Depth 5	12	2.1576	0.25343	0.877907	0.7480	2.2105	3.2000	2.452	0.77072	0.557795404	-0.43365	25.891
Depth 6	11	2.1678	0.232849	0.772273	0.9770	2.2080	3.7000	2.723	0.596406	0.518820312	0.242383	23.846
Depth 7	8	2.6308	0.17543	0.49619	1.9000	2.8140	3.1000	1.2	0.246204	0.414824967	-0.60451	21.046
Depth 8	5	2.3794	0.285429	0.638239	1.9000	2.1860	3.5000	1.6	0.407349	0.792478173	2.024451	11.897

Statistical Analysis -Cu

Cu	Count	Mean	SE Mean	ST Dev	Min	Median	Max	Range	S. Var	Confidence Level	Skewness	Sum
Depth 1	14	0.6706	0.059908	0.224156	0.3390	0.6550	1.0080	0.669	0.050246	0.129423856	0.207043	9.389
Depth 2	14	0.6968	0.063943	0.239252	0.3240	0.6550	1.1470	0.823	0.057241	0.138139862	0.390314	9.755
Depth 3	13	0.7702	0.091711	0.33067	0.2490	0.7100	1.3130	1.064	0.109343	0.199822026	0.313295	10.012
Depth 4	12	0.7539	0.080074	0.277384	0.4500	0.6350	1.1540	0.704	0.076942	0.176241406	0.600903	9.047
Depth 5	12	0.8422	0.091094	0.31556	0.5000	0.6700	1.3000	0.8	0.099578	0.20049748	0.492504	10.106
Depth 6	11	0.8323	0.106146	0.352047	0.4700	0.6800	1.3110	0.841	0.123937	0.236508133	0.557137	9.155
Depth 7	8	0.9180	0.187356	0.529924	0.5100	0.6200	1.9400	1.43	0.280819	0.443027537	1.199154	7.344
Depth 8	5	0.8338	0.181493	0.40583	0.4500	0.6700	1.2780	0.828	0.164698	0.503904823	0.440931	4.169

Statistical Analysis - Mn

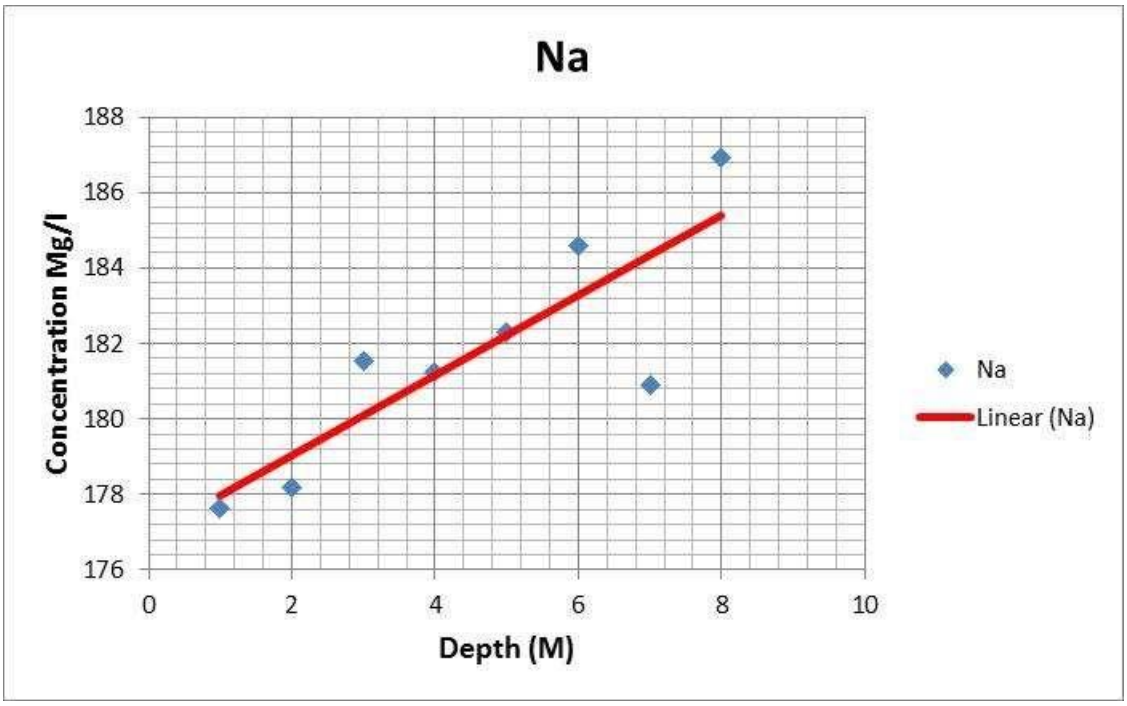
Mn	Count	Mean	SE Mean	ST Dev	Min	Median	Max	Range	S. Var	Confidence Level	Skewness	Sum
Depth 1	14	0.4789	0.077345	0.2894	0.1800	0.3720	1.1300	0.95	0.083752	0.167094297	1.118651	6.705
Depth 2	14	0.4146	0.057734	0.216022	0.2100	0.3485	0.8550	0.645	0.046666	0.12472756	1.370239	5.805
Depth 3	13	0.5517	0.089651	0.32324	0.2400	0.4400	1.1680	0.928	0.104484	0.195331946	0.910896	7.172
Depth 4	12	0.5695	0.096209	0.333276	0.2500	0.4200	1.1220	0.872	0.111073	0.211753571	0.778706	6.834
Depth 5	12	0.7199	0.130119	0.450746	0.2400	0.6500	1.5250	1.285	0.203172	0.286390563	0.849151	8.639
Depth 6	11	0.7372	0.163103	0.540952	0.2000	0.4600	1.7080	1.508	0.292629	0.363416435	0.922253	8.109
Depth 7	8	2.3318	1.523898	4.310235	0.2200	0.4550	12.8270	12.607	18.57812	3.603446377	2.656183	18.654

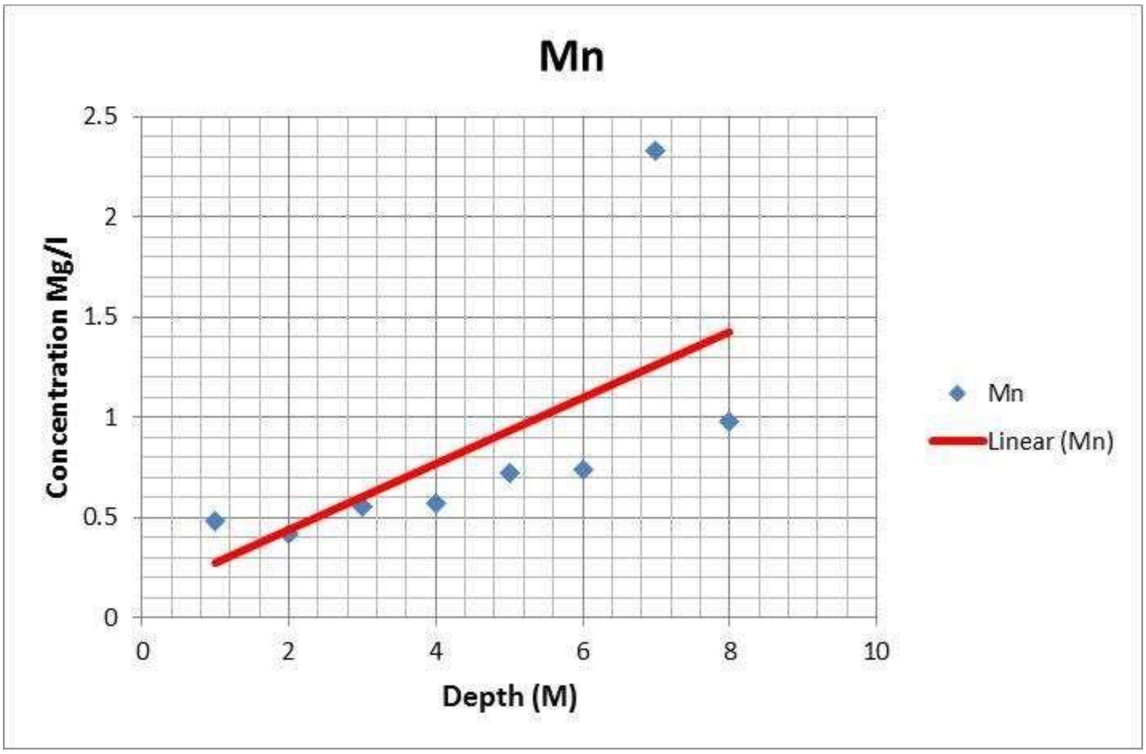
Depth 8	5	0.9792	0.426601	0.953908	0.2900	0.3800	2.4280	2.138	0.909941	1.184433695	1.118753	4.896

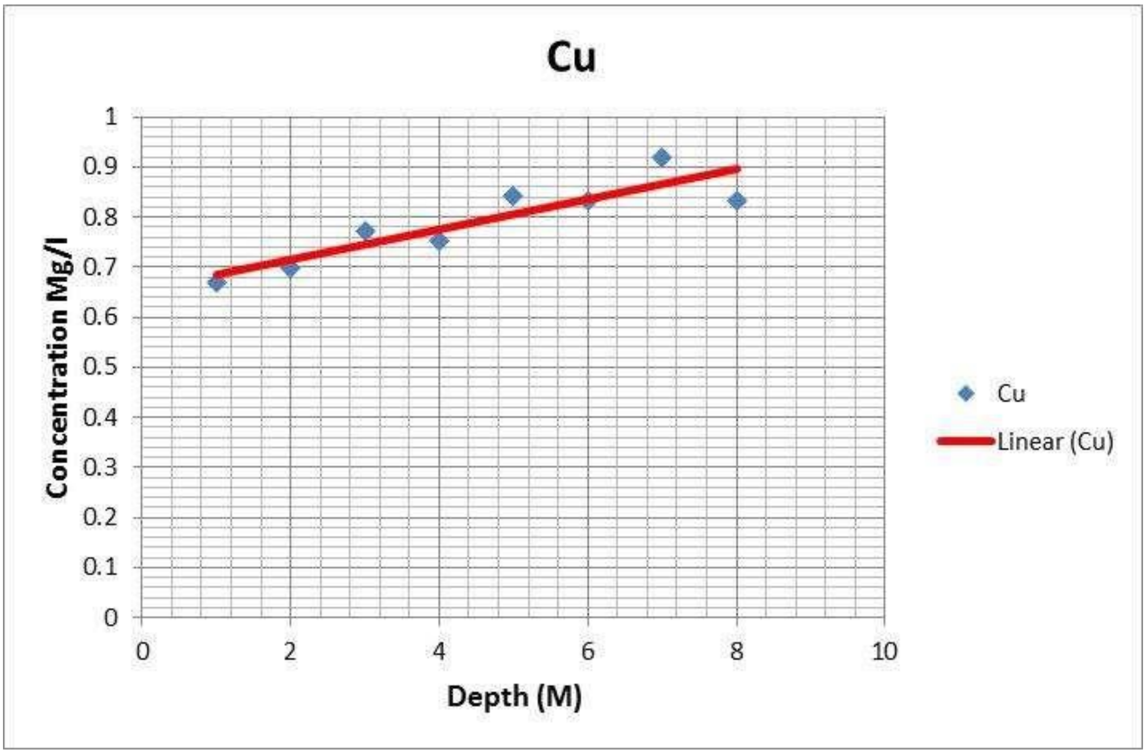
Statistical Analysis - Na

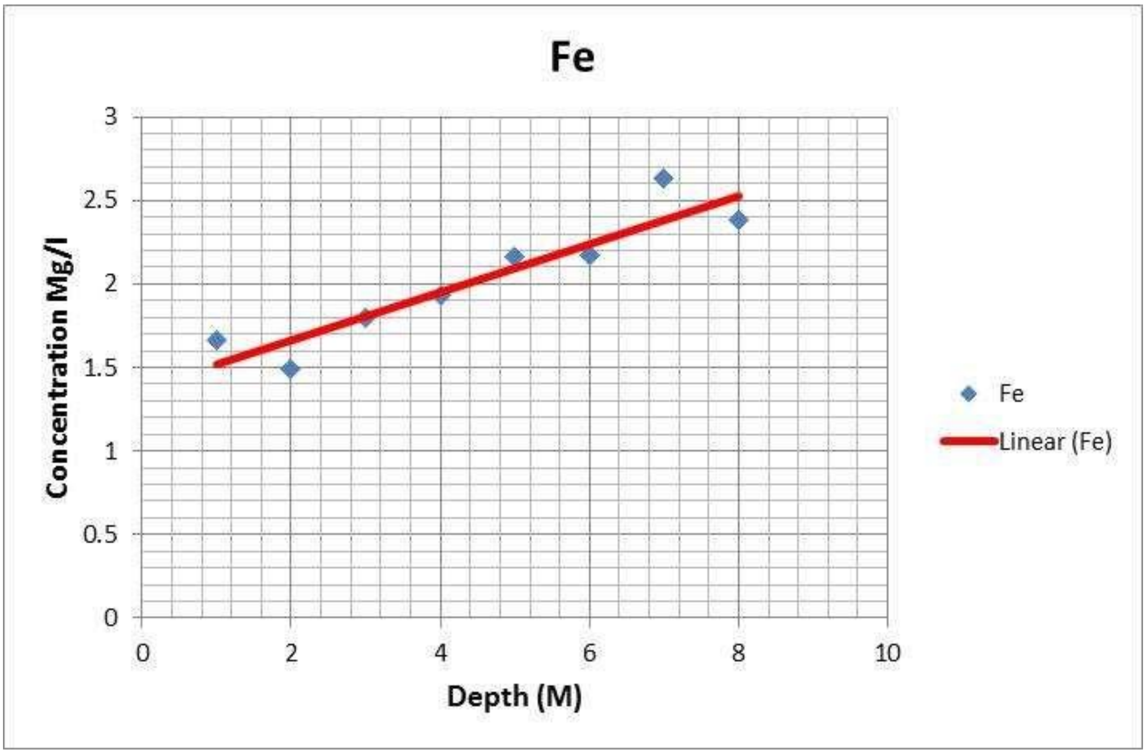
Na	Count	Mean	SE Mean	ST Dev	Min	Median	Max	Range	S. Var	Confidence Level	Skewness	Sum
Depth 1	9	177.64	2.548123	7.644369	160.81	180.94	185.19	24.384	58.43638	5.875982277	-1.51102	1598.78
Depth 2	9	178.19	2.68618	8.058539	161.03	182.67	186.41	25.387	64.94005	6.194341235	-1.27583	1603.739
Depth 3	8	181.53	2.476425	7.004389	165.70	182.58	189.49	23.782	49.06147	5.855815739	-1.86244	1452.227
Depth 4	7	181.26	2.217809	5.867771	168.41	182.78	185.23	16.819	34.43073	5.426782717	-2.28393	1268.814
Depth 5	7	182.29	2.39042	6.324456	168.51	183.29	186.84	18.334	39.99874	5.849146275	-2.22885	1276.015
Depth 6	7	184.61	1.42072	3.758872	177.60	183.97	188.58	10.984	14.12912	3.476376718	-0.99546	1292.243
Depth 7	5	180.92	5.017433	11.21932	161.08	185.26	188.63	27.546	125.8732	13.93062683	-2.10377	904.601
Depth 8	3	186.94	0.994535	1.722586	185.27	186.86	188.71	3.442	2.967302	4.279140704	0.222374	560.83

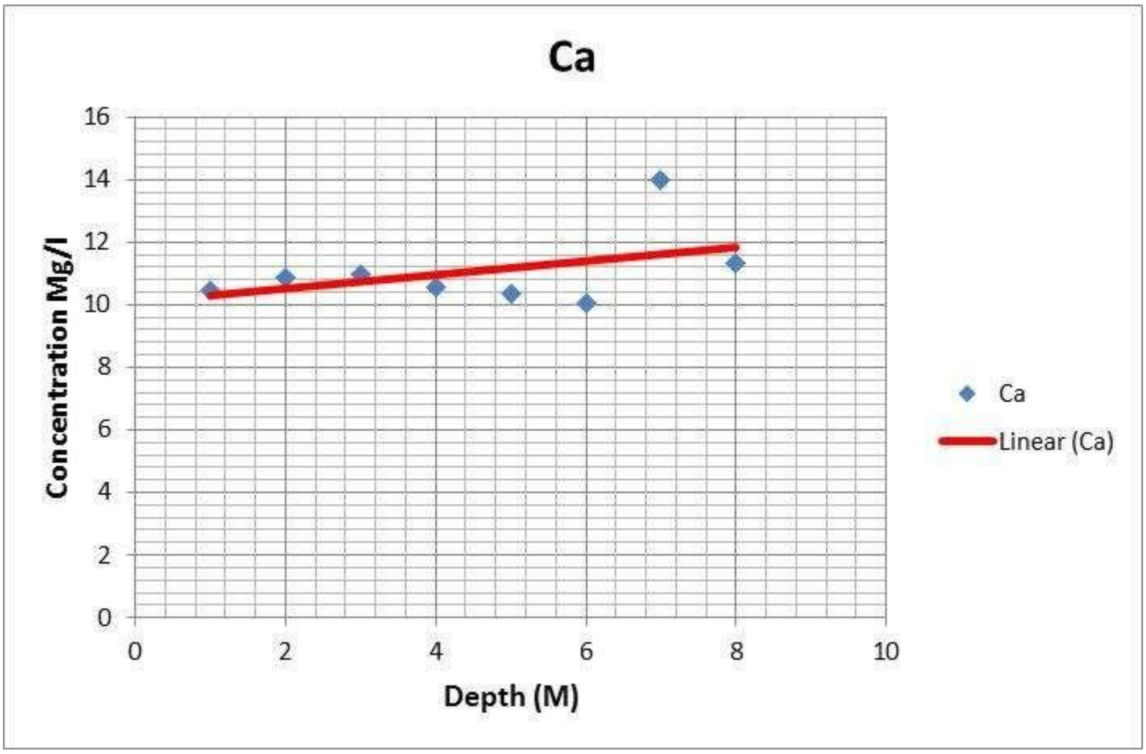
Correlation Test for Decant Pond Physico-chemical

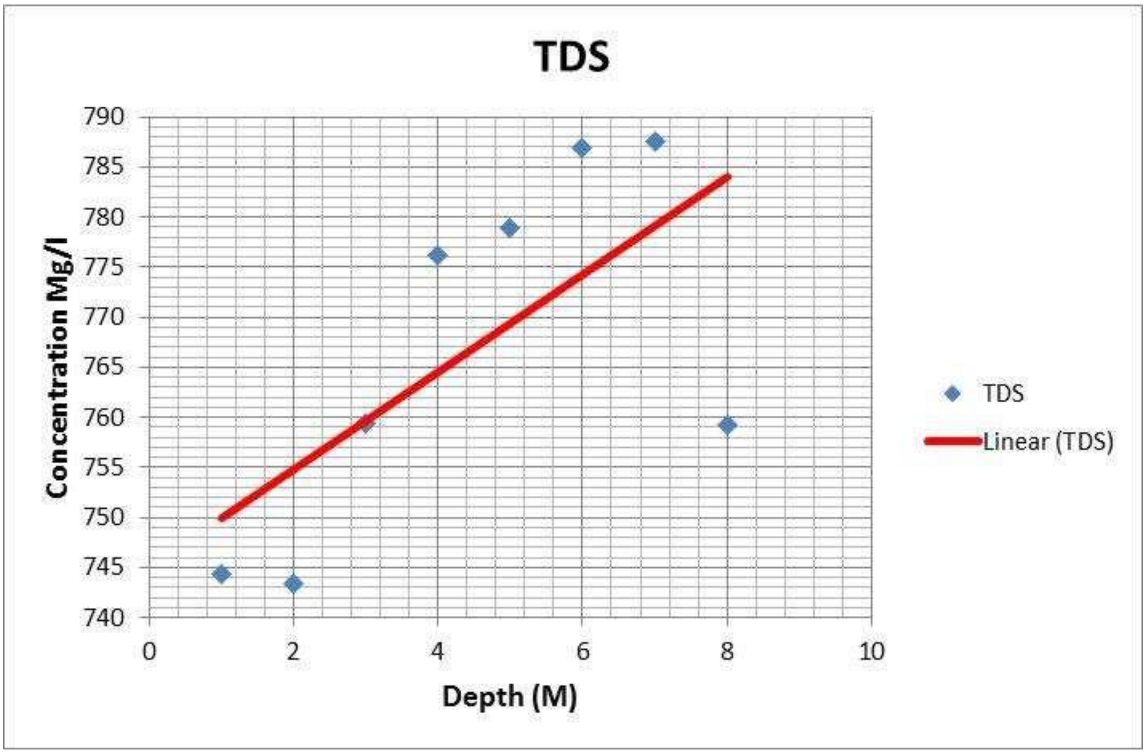


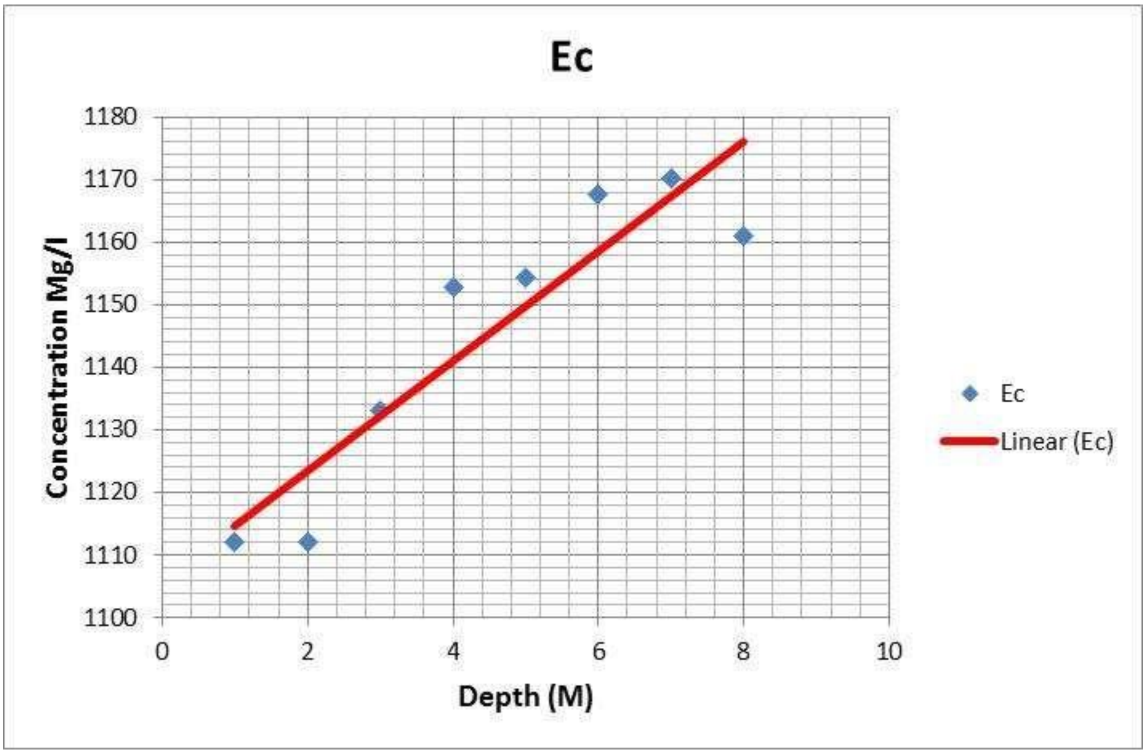



Cu
Concentration Mg/l
1
0.9
0.8
0.7
0.6
0.5
0.4
0.3
0.2
0.1
0
0
2
4
6
8
10
Depth (M)
◆ Cu
Linear (Cu)


Fe
Concentration Mg/l
3
2.5
2
1.5
1
0.5
0
0
2
4
6
8
10
Depth (M)
◆ Fe
Linear (Fe)


Ca
Concentration Mg/l
16
14
12
10
8
6
4
2
0
0
2
4
6
8
10
Depth (M)
◆ Ca
Linear (Ca)


TDS
Concentration Mg/l
790
785
780
775
770
765
760
755
750
745
740
0
2
4
6
8
10
Depth (M)
◆ TDS
Linear (TDS)


Ec
1180
1170
1160
1150
1140
1130
1120
1110
1100
Concentration Mg/l
0
2
4
6
8
10
Depth (M)
Ec
Linear (Ec)

APPENDIX 7

Mean metals and trace elements in decant pond of TSF 1 and 2

	Ca	Mg	Fe	Cu	Cd	Ni	Mn	Pb	Zn	Na	K	Ag	Cr	As
Depth	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l
1	10.43	0.40	1.66	0.670	<0.002	0.02	0.48	0.09	0.03	177.64	23.34	0.01	0.02	0.03
2	10.88	0.40	1.50	0.700	<0.002	0.03	0.41	0.01	0.03	178.19	24.66	0.02	0.01	0.03
3	10.70	0.39	1.91	0.810	<0.002	0.00	0.57	0.17	0.02	180.39	20.85	0.02	0.02	0.03

4	10.60	0.43	1.92	0.740	<0.002	0.00	0.55	0.27	0.02	180.87	19.00	0.01	0.01	0.03
5	10.34	0.41	2.16	0.837	<0.002	0.00	0.72	0.01	0.02	182.27	20.75	0.01	0.01	0.03
6	10.03	0.43	2.15	0.827	<0.002	0.00	0.73	0.15	0.02	183.31	20.68	0.01	0.01	0.03
7	11.29	0.53	2.37	0.832	<0.002	0.00	0.86	0.16	0.02	184.22	20.73	0.02	0.02	0.03
8	13.32	0.68	2.50	1.018	<0.002	0.02	1.29	0.17	0.06	185.48	22.12	0.01	0.05	0.04
Supernatant	10.26	0.14	0.58	0.193	<0.002	0.03	0.13	0.02	0.09	169.02	22.09	0.02	<0.002	0.04
Tailings 1	12.50	0.15	0.75	0.962	<0.002	0.03	1.25	0.02	0.01	210.80	15.07	0.02	<0.002	0.04
Tailings 2	11.37	0.12	0.90	0.920	<0.002	0.03	1.30	0.02	0.03	211.79	19.61	0.02	<0.003	0.03

Mean Na Concentrations in Decant pond

Depth, m value	Minimum	Parameter mg/l Maximum value	Mean value	Standard Deviation
1	160.81	185.19	177.64	7.644
2	161.03	186.41	178.19	8.059
3	165.70	185.84	180.39	6.722
4	165.70	185.23	180.87	6.861
5	168.41	186.84	182.27	6.361
6	168.51	188.58	183.31	6.867
7	177.60	188.63	184.22	4.077
8	161.08	188.71	180.48	13.006

Mean Mg Concentrations in Decant pond

		Parameter Mg, mg/l	Mean	
Depth, m	**Minimum value**	**Maximum value**	**value**	**Standard Deviation**
1	0.15	0.80	0.40	0.215
2	0.22	0.53	0.39	0.142
3	0.22	0.53	0.39	0.142
4	0.22	0.60	0.43	0.168
5	0.24	0.50	0.41	0.121
6	0.25	0.60	0.43	0.158
7	0.26	0.80	0.53	0.195
8	0.55	0.85	0.68	0.128

Mean Cu Concentrations in Decant pond

		Parameter C, mg/l	Mean	
Depth, m	**Minimum value**	**Maximum value**	**value**	**Standard Deviation**
1	0.34	1.01	0.67	0.224
2	0.32	1.15	0.70	0.239
3	0.41	1.31	0.81	0.304
4	0.45	1.15	0.74	0.260
5	0.50	1.30	0.84	0.309
6	0.47	1.31	0.83	0.346
7	0.51	1.31	0.83	0.373
8	0.45	1.94	1.02	0.579

Mean Ca Concentrations in Decant pond

		Parameter C, mg/l	Mean	
Depth m	**Minimum value**	**Maximum value**	**value**	**Standard Deviation**
1	5.10	15.00	10.43	3.739
2	5.27	16.00	10.88	3.935
3	5.22	16.00	10.70	4.355
4	5.04	16.00	10.60	4.501
5	4.88	16.00	10.34	4.607
6	4.95	16.00	10.03	4.231

7	5.10	17.00	11.29	4.598
8	8.43	21.32	13.32	5.405

Mean Fe Concentrations in Decant pond

Depth m	Parameter Fe, mg/l Minimum value	Maximum value	Mean value	Standard Deviation
1	0.23	4.00	1.66	1.119
2	0.40	2.80	1.50	0.823
3	0.56	3.80	1.91	1.086
4	0.58	3.30	1.92	0.867
5	0.75	3.20	2.16	0.878
6	0.77	3.70	2.15	0.806
7	0.98	3.10	1.37	0.726
8	1.90	3.30	2.50	0.641

Mean Mn Concentrations in Decant pond

		Parameter Mn, mg/l		
Depth m	Minimum value	Maximum value	Mean value	Standard Deviation
1	0.18	1.13	0.48	0.289
2	0.21	0.86	0.41	0.216
3	0.24	1.17	0.57	0.328
4	0.25	1.12	0.55	0.314
5	0.24	1.53	0.72	0.450
6	0.20	1.71	0.73	0.534
7	0.22	2.07	0.86	0.776
8	0.29	2.83	1.29	1.139

Mean K Concentrations in Decant pond

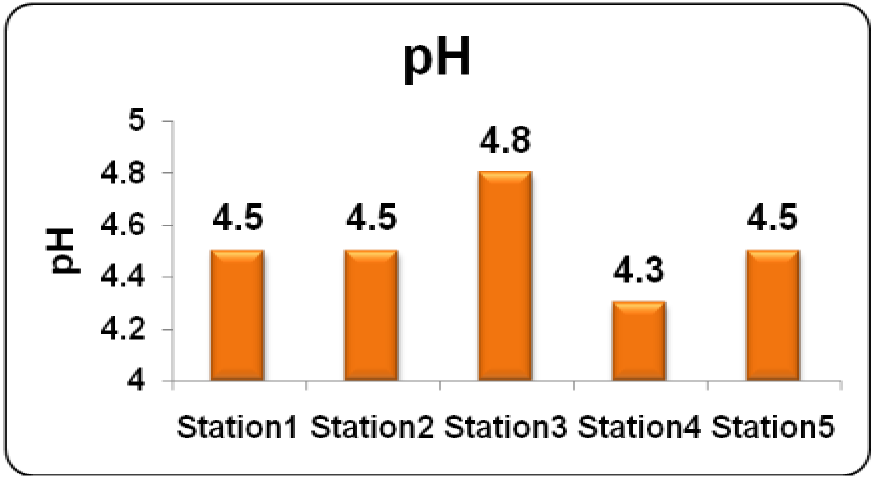
		Parameter K, mg/l		
Depth m	Minimum value	Maximum value	Mean value	Standard Deviation
1	19.36	30.41	23.34	4.733
2	19.31	34.07	24.66	6.641
3	20.14	21.53	20.85	0.738
4	17.57	20.14	19.00	1.110

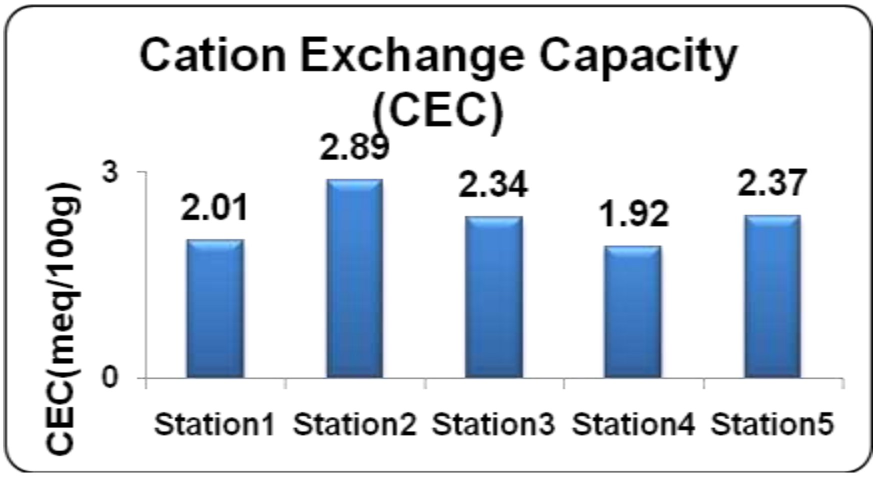
5	19.17	21.98	20.75	1.342
6	19.00	22.00	20.68	1.469
7	19.68	21.31	20.73	0.908
8	21.77	22.73	22.12	0.529

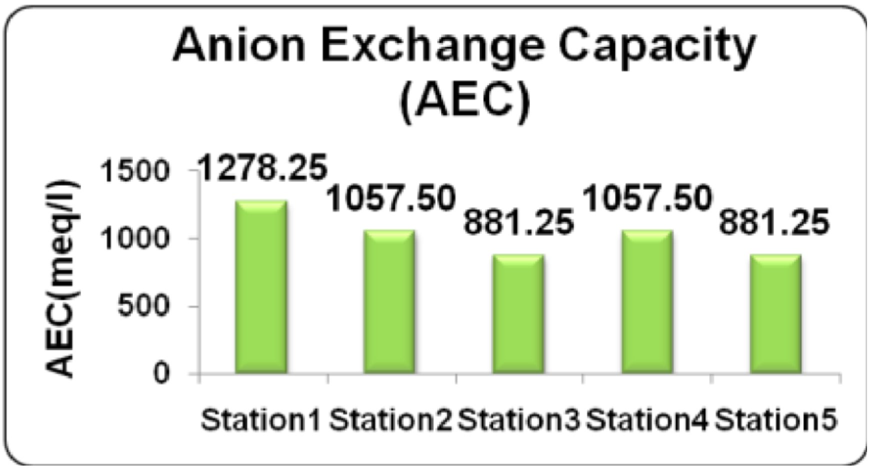
APPENDIX 8

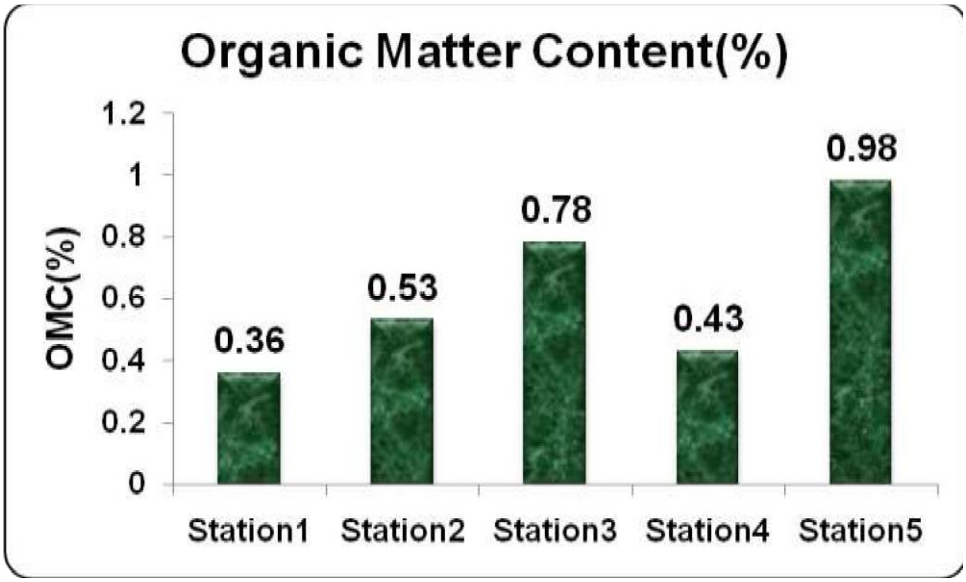
Graphs of geochemical parameters

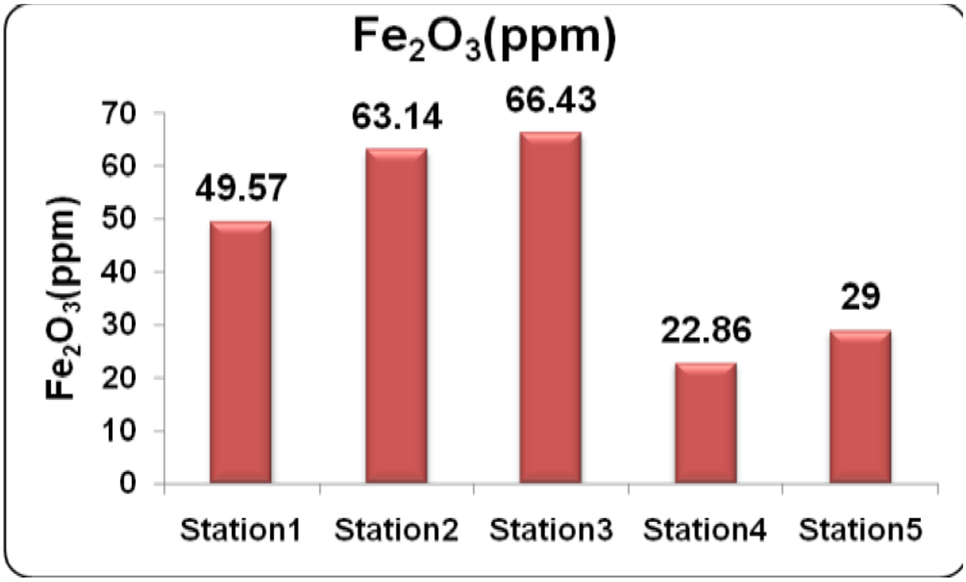
Graphs of Geochemical Parameters








Organic Matter Content(%)
OMC(%)
1.2
1
0.8
0.6
0.4
0.2
0
0.36
0.53
0.78
0.43
0.98
Station1
Station2
Station3
Station4
Station5


Fe2O3(ppm)
Fe2O3(ppm)
70
60
50
40
30
20
10
0
49.57
63.14
66.43
22.86
29
Station1
Station2
Station3
Station4
Station5

APPENDIX 9

Mean yearly groundwater monitoring data (2009-2012)

DATE	Station ID	DESCRIPTION	Temp °C	pH Unit	TDS mg/L	Conductivity uS/cm	TSS mg/L	DO mg/L	T Hardness mg/L	Alkalinity mg/L	App. Colour mg/L	True Colour mg/L	Ca++ mg/L	Mg++ mg/L	Na+ mg/L	K+ mg/L	Cl- mg/L	F- mg/L	NO3 mg/L	NO2 mg/L	NH_3- mg/L	PO_4 mg/L	SO_4 mg/L	CN(F) mg/L	CN(T) mg/L	Fe mg/L	Mn mg/L	Cu mg/L	Zn mg/L	Pb mg/L	Hg mg/L	Cr mg/L	Ni mg/L	Cd mg/L	As mg/L
2004	NBH 01			6.0	52.8	97.0		5.4	11.9	17.0														0.01	0.01			0.01						0.01	1.00
2005	NBH 01			6.2	37.8	49.0		5.1	10.5	15.1														0.01	0.01			0.01						0.01	5.50
2006	NBH 01			7.0	2240.3	307.8		3.3	9.4	825.0			2.70	1.00	4.20	0.80	1.0		1.85			0.21	3.20			0.03	0.05	0.02	0.50						1.00
2007	NBH 01			5.3	24.8	49.7		4.7	13.5	14.0																		0.01							1.00
2008	NBH 01			5.5	28.3	48.1		4.8	21.0	71.7			0.55	0.48	3.14	0.70	4.0		0.80	0.01		0.36		0.01		0.14	0.01		0.38					0.02	
2009	NBH 01			5.6	37.0	64.7		5.4	35.6	53.6			1.66	0.98	4.15	1.06	3.3		2.10	0.03		0.34	1.00	0.00		0.10	0.17	0.02	0.07	0.04	0.00	0.01	0.07	0.02	2.41
2010	NBH 01			5.7	32.9	49.7		5.1	23.8	61.3			1.05	0.53	3.42	1.13	3.3		3.20			0.48	1.00	0.01		0.01		0.02	0.05				0.04	0.01	1.65
2011	NBH 01		24.2	5.8	25.3	40.8	17.0	4.5	15.3	45.0	77.4	3.0	1.65	0.56	4.27	1.02	3.6	0.15	3.42	0.02	0.16	0.43	1.00	0.01		0.09	0.04	0.01	0.06	0.07		0.01	0.02		1.34
2012	NBH 01		24.8	5.9	42.1	68.3	17.0	4.4	23.4	43.0	61.3	1.8	4.41	0.93	4.91	0.81	3.8	0.01	5.33	0.02	0.17	0.58	1.50	0.00		0.04	0.12	0.02	0.06	0.07	0.00	0.05	0.04	0.00	1.44
2004	NBH 02			6.8	103.0	136.5		6.2	34.5	76.0														0.01	0.01			0.01						0.01	5.50
2005	NBH 02			7.0	92.0	117.5		5.9	28.5	51.6														0.01	0.01			0.01						0.01	1.00
2006	NBH 02			6.5	97.0	109.8		3.3	31.7	48.7			9.50	1.80	9.50	0.60	1.8		0.02	0.01		0.01	3.50			0.03	0.01		0.12						1.00
2007	NBH 02			6.1	61.2	106.7		4.9	30.4	40.9														0.01											1.00
2008	NBH 02			6.4	66.5	114.8		5.1	34.7	56.7			1.42	0.59	6.76	0.55	4.0			0.00		2.17		0.01			0.21	0.01	0.20	0.01					1.26
2009	NBH 02			6.5	177.0	301.0		5.0	44.1	94.9			3.19	0.73	14.67	30.99	3.3		2.34	0.01		0.96	15.50	0.00		0.21	0.81	0.01	0.03	0.04	0.00	0.01	0.04	0.01	3.33
2010	NBH 02			6.6	131.4	196.4		5.1	48.3	128.8			1.75	0.66	11.58	21.39	2.5		4.10			0.89	3.00	0.01		0.55	0.80		0.06						2.51
2011	NBH 02		24.1	6.7	82.0	134.2	8.0	5.1	28.7	88.3	72.4	7.3	3.44	0.65	12.02	12.95	3.0	0.11	3.34	0.02	0.09	0.54	1.00	0.01		0.06	0.31		0.04	0.04		0.01	0.04	0.01	1.01
2012	NBH 02		24.8	6.4	74.4	119.2	322.3	5.0	21.3	68.9	726.4	2.0	1.35	0.59	10.61	12.49	3.3	0.61	4.31	0.01	0.11	0.80	2.17	0.01		0.05	0.23	0.03	0.03	0.11	0.00	0.04	0.03	0.00	1.50

2006	NBH 03			6.7	136.5	199.8		3.4	85.1	89.1																		0.01							
2007	NBH 03			6.2	87.0	159.5		5.4	67.4	56.1														0.01											1.00
2008	NBH 03			6.2	74.0	125.2		5.6	53.3	70.0			8.17	0.99	4.42	0.94	2.9			0.02		0.22		0.01				0.01	0.19	0.03					1.16
2009	NBH 03			6.3	94.7	166.0		6.2	51.8	53.3			11.06	1.38	3.11	0.63	4.8		16.81	0.01		0.36	7.57	0.00		0.04	0.02	0.01	0.03	0.05	0.00	0.02	0.05	0.02	0.32
2010	NBH 03			6.5	79.9	122.7		5.6	54.8	67.5			7.06	1.65	3.13	0.76	2.0		5.80			0.52	8.00	0.01				0.01	0.30				0.03	0.01	1.48
2011	NBH 03		24.2	6.6	79.4	126.7	25.1	4.6	58.6	74.3	127.0	4.5	11.81	1.82	5.71	0.54	4.1	0.38	6.27	0.03	0.16	0.55	10.60	0.01		0.11	0.01	0.01	0.09	0.05		0.02	0.02		1.13
2012	NBH 03		24.7	6.6	76.7	121.6	76.3	4.5	50.8	74.4	111.9	2.5	8.15	1.62	5.58	0.64	3.7	0.11	17.21	0.02	0.07	0.77	5.83	0.00		0.10	0.11	0.04	0.07	0.07	0.01	0.07	0.03	0.00	1.28
2006	NBH 04			6.0	58.3	75.0		4.1	24.4	29.4																									1.50
2007	NBH 04			5.7	38.0	69.7		4.9	23.3	71.1																									
2008	NBH 04			6.3	156.5	238.5		4.6	103.0	20.0			12.82	3.67	7.21	0.63	5.6		1.50	0.01		0.35		0.01			0.13		0.23						1.06
2009	NBH 04			6.4	144.8	249.0		5.3	88.7	113.6			11.49	4.93	8.17	0.53	5.4		2.43	0.01		0.47	1.50	0.00		0.06	0.09	0.01	0.03	0.02	0.00	0.02	0.01	0.04	2.10
2010	NBH 04			6.5	147.2	226.2		5.1	109.0	152.5			8.76	4.72	7.17	0.84	1.4		2.90			0.80		0.01		0.17	0.43	0.01	0.44			0.02			1.73
2011	NBH 04		24.6	6.6	114.2	197.0	10.2	4.4	69.4	97.1	33.8	7.0	12.25	4.91	8.64	0.70	6.5	0.23	2.93	0.01	0.09	0.84		0.00		0.06	0.08	0.01	0.15	0.01		0.02	0.03	0.01	1.17
2012	NBH 04		24.4	6.6	117.0	186.1	9.4	4.0	81.6	123.9	31.6	2.0	7.29	4.42	8.69	0.77	6.7	0.10	4.74	0.01	0.03	0.71		0.00		0.03	0.07	0.04	0.07	0.05	0.00	0.06	0.05	0.00	1.27
2004	NBH 05			6.4	70.0	77.0		3.6	19.2	35.0														0.01	0.01			0.18						0.01	1.00
2005	NBH 05			6.4	59.3	69.0		5.5	14.6	23.5														0.01	0.01			0.01						0.01	1.00
2006	NBH 05			5.8	49.0	65.5		2.8	14.5	20.4																		0.01							1.00
2007	NBH 05			5.6	41.0	71.5		4.7	16.6	31.4																									1.00
2004	NBH 06			7.1	221.0	295.8		4.6	164.5	242.0			49.50	4.10	23.00	1.00	2.0		0.42			0.04	2.00	0.01	0.01	0.01	0.16	0.02	0.07					0.01	1.00

2005	NBH 06			7.8	280.8	366.3		2.9	180.2	223.4														0.01	0.01			0.01						0.01	1.00
2006	NBH 06			6.4	165.0	246.8		2.6	90.8	116.3																		0.07							1.00
2007	NBH 06			6.6	193.4	364.6		4.9	131.0	86.1														0.01											1.00
2008	NBH 06			5.9	176.8	291.6		4.4	42.0	83.3			9.63	2.20	18.21	46.74	9.1		3.60	0.01		0.34		0.01		0.09	0.28		0.12						1.23
2009	NBH 06			6.0	316.6	552.9		5.3	69.0	133.6			12.07	2.78	49.89	1.42	14.3		13.97	0.45		0.49	100.57	0.01	0.02	0.06	0.21	0.02	0.04	0.01	0.00	0.02	0.01	0.03	1.88
2010	NBH 06			6.0	273.0	422.0		4.8	55.8	88.8			8.44	2.21	41.71	1.49	9.8		2.20			0.64	13.00	0.01		0.08	0.31		0.05			0.02	0.05	0.01	1.62
2004	NBH 07			6.0	121.2	211.8		4.1	21.5	24.0			11.80	2.60	14.00	1.40	12.8		0.59	0.17			8.30	0.01	0.01	0.03	0.35	0.01	0.35				0.01	0.01	1.00
2005	NBH 07			6.7	89.0	140.5		4.6	34.2	26.9														0.02	0.02			0.01						0.01	1.00
2006	NBH 07			5.6	104.0	174.5		2.9	34.1	32.0																		0.02							
2007	NBH 07			5.6	130.0	244.0		2.8																											
2004	NBH 08			6.0	37.0	64.0		15.4	14.1	12.0			7.00	1.00	6.20	0.40	0.5		0.08				3.30	0.01	0.01	0.02	0.09	0.07	0.12					0.01	1.00
2005	NBH 08			6.3	36.5	64.5		4.5	9.2	11.3														0.01	0.01			0.01						0.01	1.00
2006	NBH 08			5.5	49.6	57.9		3.4	16.4	13.9																		0.01							
2007	NBH 08			5.2	46.8	90.4		5.0	19.4	62.6																		0.01							
2009	NBH 08			6.1	91.0	160.0		4.8					5.10	2.12	6.86	0.28								0.00										0.01	1.09
2006	NBH 09			8.0	123.5	234.8		3.1	69.4	103.2																								0.02	1.50
2007	NBH 09			5.8	85.8	164.5		4.9	51.9	45.3														0.01				0.01							1.00
2008	NBH 09			5.6	105.0	180.3		4.3	87.0	61.7			2.65	1.04	8.78	0.72	9.1		0.50			0.18		0.01		3.40	0.29	0.01	0.10						3.59
2009	NBH 09			5.8	129.9	228.1		4.8	64.1	113.6			7.12	2.67	12.07	1.04	8.1		1.13	0.02		0.31	5.43	0.01		0.29	0.84	0.02	0.02	0.03	0.00	0.01	0.01	0.03	2.20

2010	NBH 09			5.9	95.2	146.3		4.6	55.3	100.0			3.79	1.80	10.76	1.10	5.3					0.42	4.00	0.01			0.55		0.24			0.08			1.47
2011	NBH 09		24.3	6.0	87.9	140.0	19.0	4.7	80.4	84.3	222.2	10.0	9.98	2.05	13.23	0.94	11.7	0.17	1.28	0.01	0.16	0.50	6.20	0.01		0.19	0.48	0.01	0.09	0.03		0.02	0.03		1.29
2012	NBH 09		24.7	6.2	111.2	170.9	10.8	3.9	63.6	54.4	96.1	7.8	8.03	2.72	14.60	1.75	7.9	0.22	3.82	0.03	0.37	0.43	7.75	0.00		0.13	0.77	0.09	0.09	0.08	0.00	0.07	0.05	0.01	1.07
2004	NBH 10			6.5	208.8	389.8		5.5	42.9	130.0														0.01	0.02			0.01						0.01	1.00
2005	NBH 10			7.3	135.3	211.0		3.4	55.8	96.9														0.01	0.01			0.01						0.01	1.00
2006	NBH 10			6.0	78.5	151.3		3.3	24.4	44.1																								0.01	1.00
2007	NBH 10			6.1	131.0	261.8		3.3	61.0	60.7														0.01											
2004	NBH 11			6.5	87.4	121.8		4.4	76.2	120.0			11.40	2.77	10.65	0.95	2.1		0.82	0.01		0.02	0.96	0.01	0.01	0.49	0.44	0.01	0.09	0.02	0.00	0.01	0.01	0.01	1.44
2005	NBH 11			7.2	92.7	105.6		4.5	43.1	55.7			10.27	2.90	9.67	0.63	4.2		0.96	0.01		0.02	1.98	0.01	0.01	0.15	0.40	0.01	0.07					0.01	1.33
2006	NBH 11			9.8	119.2	171.3		4.5	36.2	51.9			12.31	1.56	20.71	4.33	3.0		1.21	0.08		0.01	23.06			0.02			0.01						4.80
2007	NBH 11			9.8	93.7	173.1		5.9	46.8	48.7																									4.50

DATE	Station ID	DESCRIPTION	Temp °C	pH Unit	TDS mg/L	Conductivity uS/cm	TSS mg/L	DO mg/L	T Hardness mg/L	Alkalinity mg/L	App. Colour mg/L	True Colour mg/L	Ca++ mg/L	Mg++ mg/L	Na+ mg/L	K+ mg/L	Cl- mg/L	F- mg/L	NO3 mg/L	NO2 mg/L	NH_3- mg/L	PO_4 mg/L	SO_4 mg/L	CN(F) mg/L	CN(T) mg/L	Fe mg/L	Mn mg/L	Cu mg/L	Zn mg/L	Pb mg/L	Hg mg/L	Cr mg/L	Ni mg/L	Cd mg/L	As mg/L
2004	NBH 12			6.7	141.1	230.7		4.1	122.7	253.5			14.58	3.70	17.48	1.72	2.4		1.56	0.01		0.02	1.05	0.01	0.01	0.81	1.74	0.01	0.09	0.02	0.00	0.01	0.01	0.01	2.00
2005	NBH 12			7.5	144.1	136.1		5.3	63.6	91.1			20.21	5.18	21.13	1.09	1.6		1.07	0.01		0.02	1.24	0.01	0.01	0.19	0.93	0.01	0.05					0.01	2.00
2006	NBH 12			5.8	48.6	64.4		4.1	17.3	20.3			3.49	1.96	6.50	0.75	4.0		0.78	0.01		0.01	3.25			0.30	0.32		0.07					0.02	
2007	NBH 12			6.0	36.6	68.6		5.4	16.1	27.3																									
2004	NBH 13			6.7	118.3	94.5		4.0	40.8	63.0			8.52	1.62	7.98	1.17	6.7		0.39	0.01		0.02	1.72	0.01	0.01	0.22	0.22	0.01	0.09	0.01	0.00	0.01	0.01	0.01	1.00

2005	NBH 13			7.2	92.9	130.0		5.9	35.6	30.0			8.73	2.20	12.79	1.70	10.3		4.56	0.03		0.02	8.54	0.01	0.01	0.17	0.37	0.01	0.08					0.01	1.00
2006	NBH 13			6.3	133.7	203.3		3.9	45.7	66.7			12.15	3.76	26.88	1.63	8.8		2.30	0.11		0.01	16.63			0.24	0.46		0.06						1.00
2007	NBH 13			6.4	124.3	237.3		5.0	68.7	55.4														0.01											1.00
2008	NBH 13			6.3	159.3	275.9		5.0	58.0	126.7			6.91	2.55	18.42	1.06	12.8			0.04		0.50		0.01			0.79	0.01	0.02						2.06
2009	NBH 13			6.4	217.6	378.3		4.2	88.3	157.1			11.55	4.50	24.69	1.13	17.7		2.95	0.12		0.36	13.80	0.01		0.14	0.85	0.01	0.01	0.03	0.00	0.01		0.01	1.96
2010	NBH 13			6.7	223.7	340.6		4.2	111.0	203.8			9.08	4.88	22.86	1.11	7.3					0.69	21.00	0.01		0.48	0.87	0.02	0.07			0.02		0.01	1.34
2011	NBH 13		24.8	6.8	228.8	368.4	35.3	5.2	100.6	174.4	1062.0	7.4	14.62	5.78	41.88	1.60	11.7	0.37	1.47	0.06	1.19	0.78	14.17	0.01		0.09	0.88	0.01	0.08	0.03		0.01	0.01		1.49
2012	NBH 13		24.3	6.8	209.8	339.0	40.5	3.7	111.7	102.8	629.8	3.5	13.36	6.38	29.41	1.52	21.6	0.43	4.93	0.04	0.34	0.61	9.00	0.01		0.11	0.97	0.04	0.03	0.10	0.00	0.07	0.03	0.00	1.32
2004	NBH 14			11.4	1263.8	833.9		4.4	41.4	193.0			119.55	0.23	26.83	15.58	3.7		0.47	0.16		0.01	15.65	0.03	0.05	0.12	0.02	0.02	0.07	0.10	0.00	0.02	0.01	0.02	1.14
2005	NBH 14			9.5	209.6	242.9		5.2	90.6	68.6			48.73	0.44	15.71	4.14	10.4		1.23	0.22		11.28	23.47	0.01	0.01	0.03	0.02	0.02	0.01					0.01	1.00
2006	NBH 14			6.7	189.5	285.3		4.3	94.0	117.5			27.99	4.69	27.50	2.08	8.8		3.18	0.69		0.02	13.95			0.80	0.50	0.01	0.10						1.00
2007	NBH 14			6.9	196.0	377.3		5.0	122.4	131.5														0.01											2.75
2008	NBH 14			6.9	251.7	434.9		4.9	163.7	218.3			35.86	4.29	19.44	1.16	10.2		0.40	0.03		0.91		0.01			0.45	0.01	0.01						3.97
2009	NBH 14			7.0	300.8	519.4		5.3	136.6	202.9			27.81	8.05	23.70	4.03	9.0		1.84	0.05		0.37	5.67	0.00		0.15	0.06	0.01	0.01	0.04	0.00	0.02		0.02	2.46
2010	NBH 14			7.1	305.0	466.0		4.5	179.3	298.8			23.29	7.95	26.43	1.28	7.1		2.80			0.77	7.00	0.01			1.03		0.04					0.01	2.55
2011	NBH 14		24.8	6.9	274.9	441.3	8.6	5.2	114.7	230.0	129.2	12.0	32.91	9.63	34.19	1.28	12.2	0.32	1.30	0.02	0.12	0.59	7.40	0.01		0.05	0.85	0.01	0.04	0.05		0.02	0.02	0.03	2.82
2012	NBH 14		24.5	7.2	270.1	438.2	5.8	4.0	121.1	180.0	91.6	3.0	32.05	8.96	31.38	1.75	19.1	0.40	2.33	0.13	0.06	0.52	7.13	0.00		0.07	0.48	0.08	0.03	0.09	0.00	0.06	0.05	0.00	2.72
2004	NBH 15			6.3	74.8	127.7		4.1	27.4	47.0			6.22	1.23	9.27	1.20	1.1		0.39	0.03		0.02	2.88	0.01	0.01	3.66	0.46	0.32	0.09	0.01	0.00	0.01	0.01	0.01	1.63
2005	NBH 15			7.4	112.2	232.4		4.9	46.8	67.5			12.06	3.23	21.75	1.09	3.1		1.10	0.09		0.02	6.71	0.01	0.01	0.21	0.44	0.01	0.07					0.01	5.50

2006	NBH 15			6.2	168.5	216.1		3.4	50.7	59.1			14.53	4.04	23.88	4.99	5.9		4.70	0.02		0.02	21.44			1.19	0.87	0.02	0.13						5.00
2007	NBH 15			6.4	103.1	195.7		5.1	55.4	103.8														0.01				0.01							2.67
2008	NBH 15			6.3	135.9	234.5		4.8	54.7	128.3			3.93	0.99	17.93	2.83	14.4		0.80	0.01		0.56		0.01		0.02	1.69	0.02	0.04					0.01	2.21
2009	NBH 15			6.4	196.1	337.5		5.5	70.3	86.4			9.32	1.92	30.06	3.17	18.6		5.15	0.02		0.30	23.86	0.00		0.12	1.53	0.01	0.03	0.01		0.02		0.03	2.28
2010	NBH 15			6.7	262.0	400.0		4.8	73.5	158.8			8.58	2.04	46.14	3.23	35.3		12.30			0.57	42.00	0.01		0.12	0.65		0.06	0.03		0.01	0.04		2.66
2011	NBH 15		24.3	7.0	270.8	433.1	12.2	4.4	132.6	188.1	108.7	13.6	26.67	5.11	50.36	2.15	8.1	0.26	6.23	0.02	0.20	0.75	21.67	0.01		0.09	0.30	0.01	0.05	0.03			0.02	0.01	3.63
2012	NBH 15		23.9	6.8	198.0	325.4	24.6	3.8	68.0	240.6	173.2	8.1	16.90	3.45	35.87	2.82	9.5	0.30	12.32	0.03	0.04	0.65	25.43	0.00		0.07	0.51	0.08	0.03	0.07		0.07	0.04	0.00	2.40
2004	NBH 16			7.2	179.0	235.8		4.0	95.6	114.0			28.82	2.83	22.32	4.17	2.2		0.78	0.10		0.05	8.55	0.01	0.01	2.61	0.52	0.02	0.42	0.01	0.01	0.01	0.01	0.01	1.89
2005	NBH 16			7.5	128.6	168.0		5.3	77.3	88.4			22.86	1.71	17.25	2.26	3.1		5.67	0.13		0.06	10.95	0.01	0.01	0.72	0.70	0.01	0.19	0.11				0.01	1.00
2006	NBH 16			8.7	203.1	263.6		4.0	73.8	116.1			23.41	2.46	30.50	4.79	3.6		4.14	0.41		0.07	11.99	25.60		0.07	0.02	0.01							8.18
2007	NBH 16			7.8	198.4	381.7		5.2	132.4	126.0														0.01											8.50
2008	NBH 16			7.8	266.4	461.0		5.1	95.3	231.7							8.5			0.04		1.74		0.01				0.01						0.01	6.73
2009	NBH 16			7.7	316.4	545.8		5.9	81.3	260.8			41.56	6.61	43.63	4.71	5.9		2.74	0.04		0.51	2.80	0.01		0.11	0.02	0.01		0.02		0.03		0.01	5.70
2010	NBH 16			7.7	320.6	491.0		5.7	72.8	333.8			36.01	6.45	21.95	1.50	1.3		2.80			0.82	1.00	0.01				0.02	0.04				0.03		6.16
2011	NBH 16		24.6	7.8	279.8	447.8	9.9	5.2	102.0	266.4	18.0	3.0	48.32	7.80	26.75	1.36	5.5	0.53	2.73	0.01	0.11	0.93	1.00	0.01		0.06	0.04	0.02	0.04	0.03	0.00	0.01	0.02		6.65
2012	NBH 16		23.7	7.7	271.4	445.9	7.3	4.1	85.2	150.6	23.3	2.0	43.46	6.18	25.66	2.65	5.3	0.40	3.29	0.04	0.02	0.63	1.00	0.00		0.07	0.02	0.03	0.03	0.06	0.00	0.10	0.02	0.00	6.51
2004	NBH 17			7.2	157.4	247.4		4.3	37.5	63.0			4.48	1.45	6.73	0.97	2.1		0.84	0.02		0.02	2.78	0.01	0.01	1.65	0.45	0.01	0.24	0.01	0.00	0.01	0.01	0.01	1.75
2005	NBH 17			7.2	55.6	88.8		5.2	16.5	21.1			5.01	1.51	7.54	1.09	3.4		0.74	0.02		0.03	4.01	0.01	0.01	0.25	0.44	0.01	0.12					0.01	1.00
2006	NBH 17			6.8	188.1	286.2		4.0	96.5	139.1			26.44	7.21	26.88	2.80	2.7		1.06	0.01		0.01	1.03			0.05	0.59	0.01	0.04						1.38

2007	NBH 17			6.9	166.2	316.5		5.1	111.6	156.0														0.01											2.00
2008	NBH 17			6.7	197.4	336.1		4.1	130.0	335.0			24.62	5.35	17.06	1.37	9.2			0.00		0.51		0.01			0.48	0.00	0.04					0.01	3.36
2009	NBH 17			6.9	249.6	432.6		5.0	129.1	210.0			20.68	7.91	24.60	1.11	6.3		3.35	0.02		0.44	1.40	0.01		0.06	0.38	0.01	0.02	0.04	0.00			0.01	3.73
2010	NBH 17			7.1	262.8	399.8		4.8	140.8	276.3			17.10	7.72	23.54	1.41	1.2		3.50			0.85		0.01			0.64	0.03	0.09				0.06	0.01	3.84
2011	NBH 17		24.4	7.1	230.1	368.3	8.4	4.5	117.4	212.9	96.0	8.3	27.04	8.74	28.01	1.43	7.1	0.63	3.85	0.02	0.09	0.72	2.00	0.01		0.07	0.25	0.02	0.03	0.06		0.01	0.04	0.01	3.64
2012	NBH 17		24.2	7.2	231.2	372.9	17.8	3.9	129.1	283.9	96.4	3.0	21.19	7.15	28.21	1.89	5.9	0.62	4.06	0.02	0.05	0.52	1.33	0.00		0.05	0.25	0.02	0.03	0.04	0.00	0.04	0.03	0.00	4.30
2004	NBH 18			10.9	676.1	1007.4		4.7	325.0	1523.0			120.45	0.13	19.47	6.82	8.3		0.17	0.02		0.02	18.88	0.03	0.04	0.08	0.01	0.03	0.07	0.05	0.01	0.02	0.01	0.01	1.00
2005	NBH 18			10.0	219.4	217.4		6.1	86.3	122.1			49.31	0.66	17.46	4.73	5.0		0.82	0.06		0.02	25.21	0.01	0.01	0.58	0.32	0.01	0.14					0.01	1.75
2006	NBH 18			6.4	104.3	161.2		4.1	50.8	67.2			13.83	3.95	17.04	1.14	4.1		1.82	0.01		0.02	3.86			0.03	0.34	0.01	0.05		0.00			0.01	1.00
2007	NBH 18			6.3	91.3	172.9		5.2	62.2	93.4														0.01				0.01							1.67
2008	NBH 18			6.2	105.4	189.6		4.6	73.3	120.0			4.03	1.39	5.11	0.30	5.5		1.70	0.00		2.18		0.01		0.33	0.16	0.01	0.06					0.01	2.73
2009	NBH 18			6.3	124.9	230.9		4.9	66.0	92.9			11.10	3.82	10.61	0.55	5.1		2.93	0.03		0.40	5.86	0.01		0.07	0.34	0.01	0.02	0.06	0.01	0.01	0.01	0.03	2.19
2010	NBH 18			6.3	110.4	173.2		4.7	54.0	70.0			7.22	2.82	9.03	0.49	1.3		11.40			0.58		0.00			0.29	0.02	0.09	0.06					1.94
2011	NBH 18		24.6	6.2	120.6	193.8	24.2	4.6	63.3	75.6	81.5	4.3	11.07	3.35	18.49	0.61	10.9	0.22	21.64	0.02	0.08	0.65	5.33	0.01		0.08	0.34	0.02	0.04	0.06			0.06	0.01	1.43
2012	NBH 18		24.6	6.5	160.2	255.8	29.1	4.1	70.3	232.2	104.0	2.7	9.98	3.78	24.10	1.10	14.0	0.22	19.54	0.02	0.02	0.27	10.33	0.00		0.06	0.31	0.06	0.05	0.09	0.00	0.03	0.03	0.00	3.50
2006	NBH 19			6.8	187.5	234.8		3.2	93.8	121.9																		0.01							2.33
2007	NBH 19			6.8	150.0	259.6		5.5	103.5	95.9														0.01											3.00
2008	NBH 19			6.6	166.0	271.2		4.8	149.0	215.0			33.55	2.13	11.85	0.77	5.1		0.80	0.01		0.32		0.01			0.23	0.01	0.07						4.75
2009	NBH 19			6.6	182.9	321.2		5.3	115.1	147.1			17.35	2.39	12.48	0.77	4.5		1.48	0.01		0.57	3.25	0.01		0.14	0.13	0.01	0.04	0.02	0.01	0.01	0.01	0.02	2.56

2010	NBH 19			6.7	165.4	253.2		4.7	110.5	175.0			12.11	3.49	11.20	0.74	1.4		2.70			0.96		0.01			0.36		0.11			0.02	0.02		1.72
2011	NBH 19		24.4	6.7	160.3	256.3	17.7	4.5	89.3	136.3	197.5	1.7	24.28	2.86	13.99	0.78	6.7	0.19	2.90	0.01	0.84	0.73	1.00	0.01		0.06	1.18	0.02	0.06	0.05		0.02	0.02		1.45
2012	NBH 19		24.4	6.9	156.7	252.9	8.0	4.1	99.1	103.9	73.6		16.26	2.25	13.08	0.96	5.2	0.39	5.91	0.02	0.05	0.40	2.00	0.00		0.05	0.34	0.06	0.09	0.04	0.00	0.04	0.05	0.00	1.78
2008	NBH 20			6.9	339.0	560.0		4.4					58.77	4.59	14.77	1.03	2.7		1.00	0.01		0.94		0.01			1.29	0.01	0.05	0.01					4.84
2009	NBH 20			6.9	314.8	539.8		5.0	100.4	243.6			51.52	5.49	13.24	2.15	6.0		3.03	0.02		0.37	2.14	0.01		0.04	0.72	0.01	0.02	0.03	0.00	0.01	0.01	0.02	4.32
2010	NBH 20			7.1	278.7	427.8		4.9	151.5	262.5			39.52	7.86	12.74	1.07	0.2		3.60			0.56	4.00	0.01			0.28	0.03	0.17	0.04			0.02		2.68
2011	NBH 20		24.7	7.0	257.0	413.4	4.3	4.6	117.4	237.1	27.8	2.3	58.35	6.74	15.98	0.81	4.0	0.13	3.65	0.01	0.10	0.77	2.75	0.01		0.07	0.28	0.02	0.05	0.04		0.01	0.06		3.48
2012	NBH 20		24.3	7.2	276.7	445.6	6.1	4.3	113.7	156.7	28.3	2.0	59.38	5.42	15.82	1.04	3.2	0.16	4.29	0.01	0.11	0.47	3.63	0.00		0.04	0.05	0.06	0.05	0.06	0.00	0.05	0.03	0.01	5.55
2008	NBH 21			6.8	221.0	365.0		5.9					30.45	2.31	9.73	1.44	5.2		1.50	0.02		1.02		0.01			0.58	0.01	0.16	0.01					2.19
2009	NBH 21			6.8	198.7	342.1		5.9	102.3	157.9			26.39	2.71	9.66	1.47	4.0		2.26	0.01		0.60	6.00	0.01		0.03	0.64	0.01	0.02	0.03	0.00	0.02	0.02	0.02	1.21
2010	NBH 21			6.8	327.9	502.9		5.1	149.8	188.8			24.75	5.35	28.30	1.63	1.2		1.90			0.56	108.00	0.01			0.50	0.01	0.13						1.46
2011	NBH 21		25.0	6.7	391.8	628.7	5.9	5.5	173.8	167.5	25.8	3.3	64.89	6.65	53.35	1.96	6.0	0.53	2.43	0.01	0.07	0.62	105.46	0.01		0.08	0.32	0.01	0.07	0.05		0.01	0.02		1.63
2012	NBH 21		24.7	6.8	282.6	450.9	4.8	4.3	118.9	263.3	15.2	3.0	49.19	7.01	38.35	1.63	4.6	0.27	2.29	0.04	0.04	0.54	51.00	0.01		0.11	0.93	0.04	0.08	0.05	0.00	0.07	0.03	0.00	2.20
2010	NBH 22			6.5	155.8	239.3		6.0	44.0	73.3			8.59	1.68	18.74	2.02	3.5		2.80			0.33	5.00	0.01		0.02	0.11		0.06					0.01	1.37
2011	NBH 22		24.9	6.4	83.9	135.3	127.6	5.3	43.0	75.6	754.2	2.5	9.29	1.20	11.11	1.20	10.4		15.10	0.04	0.13	0.43	24.17	0.01		0.10	0.20	0.01	0.05	0.05		0.01	0.01	0.01	1.51
2012	NBH 22		24.4	6.5	71.0	113.7	161.3	5.0	40.4	188.1	411.1	1.5	9.19	1.15	7.37	1.45	8.8	0.11	4.30	0.02	0.02	0.28	14.86	0.00		0.07	0.49	0.01	0.06	0.05		0.06	0.01	0.00	1.79
2008	NBH 23			6.3	154.0	252.0		3.6					15.91	1.56	6.19	0.67	9.2		107.60	0.00				0.00		9.34	3.49	0.03	0.02						4.32
2009	NBH 23			5.9	97.0	166.3		5.0	41.3	71.4			7.55	1.35	8.29	1.38	7.0		2.39	0.02		0.54	8.43	0.01		0.03	0.40	0.01	0.02	0.02	0.00	0.01	0.02	0.01	1.88
2010	NBH 23			5.9	78.7	121.0		4.2	46.8	68.8			3.87	1.41	7.85	0.53	0.4		5.20			0.77	6.00	0.01			0.26	0.02	0.11				0.05		1.76

2011	NBH 23		25.1	5.9	61.6	98.8	102.7	4.5	34.4	53.1	411.3	6.7	4.36	1.72	8.67	0.49	4.6		3.37	0.11	0.18	0.62	9.33	0.01		0.05	0.29	0.02	0.09	0.05		0.01	0.05	0.00	0.49
2012	NBH 23		24.9	6.1	84.1	142.1	112.8	3.9	41.0	64.4	300.0	4.0	13.63	1.49	9.56	0.95	5.1	0.15	3.10	0.02	0.02	0.32	9.63	0.00		0.06	0.26	0.04	0.05	0.05	0.00	0.03	0.03	0.00	1.72

DATE	Station ID	DESCRIPTION	Temp °C	pH Unit	TDS mg/L	Conductivity uS/cm	TSS mg/L	DO mg/L	T Hardness mg/L	Alkalinity mg/L	App. Colour mg/L	True Colour mg/L	Ca++ mg/L	Mg++ mg/L	Na+ mg/L	K+ mg/L	Cl- mg/L	F- mg/L	NO3 mg/L	NO2 mg/L	NH_3- mg/L	PO_4 mg/L	SO_4 mg/L	CN(F) mg/L	CN(T) mg/L	Fe mg/L	Mn mg/L	Cu mg/L	Zn mg/L	Pb mg/L	Hg mg/L	Cr mg/L	Ni mg/L	Cd mg/L	As mg/L
2008	NBH 24			6.1	85.0	138.6		5.0					5.31	0.81	5.23	1.02	3.2		2.10	0.02		0.54		0.00			0.37	0.02	0.09	0.01					1.25
2009	NBH 24			5.8	77.7	133.1		5.2	37.9	57.9			4.04	1.19	7.94	1.17	4.3		5.06	0.01		0.40	9.86	0.00		0.03	0.27	0.01	0.02	0.02	0.00	0.02	0.02	0.01	1.10
2010	NBH 24			5.9	66.8	96.9		4.7	30.5	66.3			1.78	1.11	7.28	0.41	2.1		5.90			0.73	6.00	0.00		0.04	0.24	0.02	0.30						
2011	NBH 24		25.1	5.9	55.4	88.8	5.5	4.8	26.5	72.5	30.2	1.5	3.40	1.22	8.97	0.34	10.9	0.24	4.46	0.01	0.08	0.60	10.33	0.00		0.08	0.18	0.02	0.07	0.02		0.01	0.03		1.50
2012	NBH 24		24.7	6.0	57.3	90.9	6.6	3.9	25.2	77.8	27.6	2.0	1.97	1.24	9.11	0.47	4.4	0.16	4.37	0.01	0.10	0.26	7.88	0.00		0.06	0.24	0.09	0.06	0.08	0.00	0.05	0.04	0.01	1.65
2008	NBH 26			6.1	174.0	288.0		4.0					20.72	2.75	10.31	1.94	8.7			0.02		0.38		0.01			0.37	0.02	0.04	0.01					5.52
2009	NBH 26			5.9	136.4	239.5		5.0	75.4	97.1			9.27	3.23	10.49	1.74	10.0		1.04	0.01		0.55	12.71	0.01		0.07	0.31	0.01	0.03	0.03	0.00	0.01	0.01	0.01	4.44
2010	NBH 26			5.9	98.0	149.8		4.3	45.5	88.8			3.75	2.08	10.26	0.85	2.8		5.60			0.45	37.00	0.01			0.29	0.03	0.16	0.03			0.05		1.89
2011	NBH 26		25.0	6.0	94.2	150.7	12.8	5.2	40.4	61.3	73.7	2.7	5.05	2.26	15.86	0.75	8.8	0.31	2.14	0.02	0.09	0.94	13.83	0.00		0.09	0.22	0.01	0.10	0.02		0.01	0.05	0.01	0.79
2012	NBH 26		24.5	6.2	127.4	203.6	13.6	3.9	69.7	53.3	197.3	8.3	9.18	4.09	14.01	1.14	11.2	0.19	4.17	0.01	0.22	0.30	15.38	0.00		0.07	0.47	0.03	0.06	0.07	0.00	0.11	0.02	0.01	1.45
2008	NBH 28			12.5	1439.0	2450.0		4.4					145.13	0.06	11.55	5.92	10.9		10.20			0.55		0.01			0.37	0.02	0.03	0.02			0.01		1.85
2009	NBH 28			7.7	300.3	521.8		5.2	122.4	280.0			44.56	2.62	19.42	3.96	7.4		3.96	0.19		0.56	21.29	0.00		0.06	0.12	0.01	0.01	0.02	0.00	0.01	0.03	0.01	5.02
2010	NBH 28			6.9	335.9	515.3		5.2	120.0	288.8			45.06	2.32	26.13	1.83	4.7		2.60			0.65	45.00	0.01			0.19	0.01	0.10	0.03			0.02		1.84
2011	NBH 28			6.9	378.0	560.0								2.16	40.14	1.12								0.01										0.01	2.55
2008	NBH 30			5.9	78.0	127.7		4.8					3.82	0.76	5.03	1.32	15.7			0.07		0.59		0.01			1.05	0.01	0.04						1.46

2009	NBH 30			5.9	72.9	125.3		4.9	39.0	59.3			5.29	0.87	5.80	1.20	8.3		1.08	0.02		0.32	2.57	0.00		0.06	0.30	0.01	0.03	0.04	0.00		0.01	0.02	1.74
2010	NBH 30			6.4	85.7	131.1		4.8	83.5	99.8			6.92	1.20	6.83	1.08	4.2		2.50			0.70	3.00	0.01			0.87		0.11						1.30
2011	NBH 30		25.2	6.2	75.6	121.6	15.9	5.4	36.3	76.9	203.3	4.5	7.79	1.48	10.01	0.99	10.2		1.48	0.07	0.12	0.66	3.83	0.00		0.08	0.69	0.02	0.09	0.04		0.01	0.12		0.59
2012	NBH 30		24.7	6.2	78.6	125.1	24.8	4.2	40.7	82.2	176.1	2.0	3.73	1.44	10.22	0.95	10.5	0.29	3.40	0.01	0.13	0.31	4.63	0.00		0.07	0.61	0.03	0.04	0.08	0.00	0.11	0.02	0.00	1.12
2008	NBH 31			7.3	260.0	428.0		4.2					41.33	2.34	9.29	3.56	9.1					0.47		0.01			1.02	0.01	0.00						2.59
2009	NBH 31			6.0	111.3	190.4		5.2	55.9	97.9			13.89	1.44	7.90	1.65	6.0		1.85	0.02		0.36	4.00	0.00		0.08	0.14	0.02	0.02	0.04	0.00	0.02	0.02	0.11	1.89
2010	NBH 31			6.4	105.8	161.9		4.6	82.0	145.0			6.75	1.71	9.48	0.89			4.20			0.48	7.00	0.01			0.37	0.02	0.09	0.06					1.17
2011	NBH 31		25.0	6.3	100.3	161.6	9.8	4.2	54.4	74.4	73.7	2.8	9.89	2.22	13.34	1.27	8.1	0.16	3.17	0.02	0.14	0.59	8.67	0.00		0.10	0.35	0.01	0.07	0.02		0.01	0.17	0.01	1.64
2012	NBH 31		24.6	6.5	120.6	193.5	16.3	4.1	62.8	62.8	189.8	9.0	9.08	2.56	17.37	1.18	7.0	0.19	1.93	0.02	0.06	0.32	10.25	0.00		0.07	0.47	0.06	0.07	0.05	0.00	0.11	0.04	0.01	1.74
2008	NBH 32			7.0	96.0	156.7		4.2					8.83	1.04	6.65	1.19	5.9		1.60	0.07		1.30		0.01		0.08	0.10		0.04						
2009	NBH 32			6.4	151.2	252.6		4.9	88.7	140.3			23.27	1.91	9.20	2.40	6.2		2.89	0.05		0.38	11.57	0.00		0.04	0.40	0.01	0.00	0.03		0.01	0.02	0.01	1.98
2010	NBH 32			6.3	128.0	191.5		4.8	69.8	137.5			16.09	1.37	9.63	1.65	2.8		15.70			0.59	21.00	0.01		0.01	0.74	0.01	0.14						1.12
2011	NBH 32		24.8	6.1	108.6	175.6	172.0	4.4	50.1	110.0	1781.0	4.3	11.87	1.53	16.95	1.14	9.4		2.65	0.02	0.21	0.52	17.67	0.01		0.05	0.43	0.01	0.06	0.05		0.01	0.03		1.12
2012	NBH 32		24.9	6.2	115.2	191.4	386.4	3.9	54.6	95.6	1151.2	13.2	9.71	1.79	20.78	1.60	10.7	0.12	3.82	0.02	0.06	0.31	16.86	0.01		0.08	0.62	0.02	0.05	0.06	0.00	0.09	0.03	0.00	
2008	NBH 34			6.2	137.0	225.0		5.8	83.0	110.0			12.14	3.44	6.33	0.90	5.3		3.40	0.33		0.20		0.00			0.28		0.15						
2009	NBH 34			6.2	81.3	140.6		4.9	47.9	76.4			5.13	2.01	6.15	1.73	4.0		4.37	0.03		0.45	1.25	0.00		0.06	0.11	0.01	0.09	0.05	0.00	0.02	0.02	0.01	1.61
2010	NBH 34			6.5	76.9	117.1		5.2	43.3	138.8			3.27	2.42	8.98	0.89	0.6		2.70			0.84	1.00	0.00			0.06	0.01	0.08						1.83
2011	NBH 34		23.9	6.3	54.5	87.0	5.4	5.9	32.6	52.1	26.0	4.0	3.42	1.55	7.03	0.42	4.4	0.14	4.48	0.01	0.04	0.77	1.33	0.00		0.03	0.06	0.02	0.09	0.03		0.01	0.02		1.10
2012	NBH 34		24.6	6.5	50.5	81.7	9.0	5.1	27.8	98.1	33.6		1.72	1.32	7.18	0.54	5.4	0.13	4.45	0.01		0.44	2.67	0.00		0.07	0.05	0.05	0.11	0.04	0.00	0.04	0.03	0.01	1.76

2011	NBH 35		25.7	6.7	378.0	620.2	1.9	5.2	107.7	295.0	70.0	10.0	121.25	4.53	10.93	1.80	6.5	0.53	3.13	0.01	0.12	0.73	14.75	0.01		0.04	0.37	0.01	0.20	0.01		0.01	0.08	0.01	1.00
2011	NBH 36			9.9	391.3	674.3		5.2	210.0	285.0			59.06	0.57	16.91	6.95	5.8		2.20	0.02		0.52	20.50	0.01		0.01	0.07	0.01	0.01	0.03		0.02	0.02		1.50
2012	NBH 36																																		
2011	NBH 37			6.9	122.8	201.7		5.2	57.0	177.5			16.36	2.34	13.14	2.39	6.9		9.58	0.18		0.51	6.00	0.00			0.13	0.00	0.06	0.01		0.01	0.01	0.00	2.86
2011	NBH 38			8.0	197.5	321.3		5.7	77.0	115.0			33.52	3.48	16.38	4.44	43.3		4.88	0.39		0.63	14.50	0.01		0.08	0.10	0.01	0.04			0.01		0.01	2.10
2011	NBH 39			8.8	258.0	423.0		5.3	131.5	160.0			45.39	4.15	12.10	5.43	4.2		2.73	0.04		0.87	16.75	0.01		0.05	0.02	0.01	0.03			0.04	0.02	0.00	1.99
2011	NBH 40			6.9	186.3	305.3		5.4	106.5	170.0			29.88	3.62	18.89	1.01	4.2		34.05	0.06		0.83	9.67	0.00		0.05	0.19	0.01	0.04			0.01	0.01	0.01	1.40
2011	NBH 42			7.6	409.5	669.0		4.3	178.0	220.0			69.59	6.47	46.20	4.62	10.1		2.10	0.05		0.90	85.88	0.01		0.05	0.46	0.01	0.02	0.12			0.01	0.00	8.09
2011	NBH 43			7.9	458.5	750.3		5.4	176.0	125.0			67.75	3.85	74.12	13.69	14.2		2.83	0.08		0.51	348.13	0.01		0.05	0.19	0.01	0.05	0.11		0.01	0.02		4.98
2011	NBH 44			7.3	245.0	402.0		4.5	114.0	177.5			47.31	10.74	12.10	0.90	6.5		3.08	0.01		0.63	6.67	0.01		0.15	0.33	0.01	0.02	0.08			0.01		3.18
2004	SBH 01			7.0	270.0	359.0		4.0	154.0	162.0														0.01	0.03			0.01						0.01	1.00
2005	SBH 01			7.3	215.0	321.8		4.4	138.1	162.6														0.02	0.02			0.01						0.01	1.00
2006	SBH 01			6.7	178.5	351.0		1.1	76.9	129.9																		0.01							1.00
2007	SBH 01			6.7	180.6	347.4		1.6	91.3	133.9														0.01				0.01							1.00
2008	SBH 01			6.5	205.8	356.5		2.7	94.0	203.3			22.98	2.98	7.60	6.26	14.0			0.01		0.43		0.00		1.98	0.17	0.02	0.09	0.01				0.02	2.00
2009	SBH 01			6.8	224.9	398.1		3.9	82.3	213.6			19.57	3.71	11.29	4.92	15.8		5.05	0.03		1.33	6.67	0.01		0.13	0.19	0.01	0.01	0.01		0.01	0.03	0.02	1.55
2010	SBH 01			6.8	242.3	369.6		3.9	121.8	431.3			25.05	2.59	7.88	5.65	7.7					0.52		0.01		0.46	0.12	0.05	0.11	0.04					1.07
2011	SBH 01		24.4	7.1	214.9	347.4	47.7	3.1	121.6	196.0	779.8	13.0	42.90	3.37	17.37	5.45	13.2	1.09	10.60	0.08	4.21	0.39	8.00	0.01		0.19	0.25	0.01	0.04	0.02		0.03	0.01	0.00	1.13
2012	SBH 01		24.7	7.1	233.0	374.0	84.4	3.0	82.2	222.8	440.1	6.4	46.48	3.02	12.14	5.46	9.9	0.67	11.13	1.12	4.65	0.31	3.40	0.00		0.05	0.37	0.05	0.11	0.05	0.00	0.09	0.02		1.53

DATE	Station ID	DESCRIPTION	Temp	pH	TDS	Conductivity	TSS	DO	T Hardness	Alkalinity	App. Colour	True Colour	Ca++	Mg++	Na+	K+	Cl-	F-	NO3	NO2	NH3-	PO_4	SO_4	CN(F)	CN(T)	Fe	Mn	Cu	Zn	Pb	Hg	Cr	Ni	Cd	As
2004	SBH 02			6.8	109.3	129.3		4.6	29.7	74.0														0.01	0.01			0.01						0.01	2.00
2005	SBH 02			6.9	115.3	147.0		5.0	32.9	62.9														0.01	0.01			0.01						0.01	1.00
2006	SBH 02			6.4	121.3	144.8		3.4	29.6	65.5																									
2007	SBH 02			6.3	82.2	144.0		4.9	67.4	56.7																									1.00
2008	SBH 02			5.9	95.0	163.0		5.0	40.0	106.7			1.85	0.78	18.32	0.49	8.2		2.20	0.01		0.44		0.01			1.10	0.01	0.10					0.02	
2009	SBH 02			6.0	104.0	180.2		5.4	37.3	93.1			2.15	0.67	16.22	0.45	6.6		2.80	0.01		0.54	3.33	0.01		0.07	0.54		0.04	0.03	0.00	0.03	0.02	0.03	3.34
2010	SBH 02			6.1	97.9	149.1		5.3	54.5	117.5			1.43	0.75	17.00	0.54	6.0		1.70			0.69		0.01			1.05	0.02	0.15			0.01		0.01	2.52
2011	SBH 02		23.9	6.4	88.5	142.5	9.7	4.9	33.9	78.7	27.0	6.5	3.05	1.01	19.70	0.50	5.7	0.07	2.35	0.01	0.07	0.81	1.80	0.00		0.34	0.78	0.01	0.11	0.03	0.01		0.02	0.01	1.26
2012	SBH 02		24.9	6.2	87.4	141.4	33.6	5.0	29.3	88.3	52.6	1.3	1.64	0.88	19.12	0.64	7.4	0.15	5.17	0.02	0.05	0.38	2.00	0.00		0.05	0.84	0.06	0.13	0.04	0.00	0.04	0.02		1.39
2004	SBH 03			6.5	126.5	247.0		5.8	85.1	97.0														0.01	0.01			0.01						0.01	1.00
2005	SBH 03			6.9	139.3	189.0		5.2	68.3	72.3														0.01	0.01			0.01						0.01	1.00
2006	SBH 03			6.3	112.3	195.5		3.6	48.3	74.2																									
2007	SBH 03			6.2	122.8	228.3		4.9	89.2	78.5														0.01											1.00
2008	SBH 03			5.8	128.5	218.3		4.3	64.7	98.3			22.90	2.17	6.94	1.99	9.4			0.02		0.55		0.00			0.54	0.02	0.09				0.01	0.03	
2009	SBH 03			5.9	145.3	257.5		5.0	76.7	110.0			13.33	2.50	8.12	2.58	9.2		3.50	0.01		0.43	14.43	0.01		0.15	0.65		0.03	0.01		0.01	0.04	0.05	1.86
2010	SBH 03			5.9	117.7	177.6		4.6	78.3	112.5			6.57	1.78	7.06	1.46	14.2		1.20			1.20	7.00	0.01		1.27	0.56	0.03	0.17				0.03		
2011	SBH 03		24.2	6.0	80.2	127.9	24.8	4.5	42.8	64.7	132.5	3.0	6.51	1.79	7.47	1.02	6.8	0.34	2.84	0.10	0.27	0.52	9.20	0.01		0.23	0.44	0.01	0.11		0.01	0.14	0.03	0.01	1.01
2012	SBH 03		24.6	5.9	81.9	132.2	27.1	3.6	51.7	71.1	257.2	4.5	5.38	1.85	7.38	1.62	6.3		5.70	0.01	0.22	0.37	10.38	0.01		0.03	0.49	0.07	0.13	0.09	0.00	0.14	0.02		1.10

			°C	Unit	mg/L	uS/cm	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
2006	SBH 04			6.4	156.0	195.8		3.1	73.0	81.0																									2.00
2007	SBH 04			6.1	85.4	147.4		5.2	33.4	38.9																		0.01							1.00
2008	SBH 04			5.7	106.3	176.9		4.3	47.7	83.3			17.97	1.75	12.36	0.67	7.4		0.60	0.00		0.44		0.01			0.83	0.01	0.17				0.01	0.02	2.00
2009	SBH 04			6.2	119.1	211.5		4.8	76.9	115.0			8.47	152.53	11.47	1.26	5.8		2.27	0.01		0.46	4.00	0.00		0.65	0.43		0.04	0.03		0.02	0.04	0.05	1.96
2010	SBH 04			6.2	94.7	145.0		4.8	62.0	112.5			5.07	1.03	6.73	1.53	4.2		2.20			0.61	7.00	0.01			0.47	0.04	0.13	0.01				0.01	1.28
2011	SBH 04		23.9	6.5	131.5	245.4	10.7	5.1	72.7	103.3	66.2	12.0	15.80	2.13	14.75	1.62	5.0	0.35	2.29	0.02	0.05	0.74	8.60	0.01		0.04	0.27	0.01	0.10	0.09	0.79	0.00	0.13	0.01	1.39
2012	SBH 04		24.5	6.5	124.6	200.0	11.9	4.3	76.1	105.0	58.2		11.17	1.55	12.67	0.83	4.9	0.22	2.67	0.01	0.03	0.52	7.75	0.00		0.03	0.27	0.06	0.09	0.06	0.00	0.13	0.04	0.00	1.18
2006	SBH 05			5.4	40.1	49.8		4.0	10.0	14.5																									
2007	SBH 05			5.3	28.2	51.3		5.0	11.9	22.8														0.01				0.01							
2008	SBH 05			6.0	122.3	196.7		4.5	80.3	95.0			27.63	6.53	8.74	1.04	3.9		2.90	0.05		0.63		0.01			0.73	0.01	0.35	0.03				0.03	1.33
2009	SBH 05			5.7	85.0	136.6		5.5	55.7	85.7			6.05	1.79	6.71	0.44	4.9		3.63	0.01		0.52	2.00	0.00		0.04	0.04	0.01	0.10	0.01	0.00	0.01	0.04	0.04	1.87
2010	SBH 05			5.4	33.0	49.5		5.3	36.8	51.3			1.09	0.38	5.65	0.24	4.9		2.70			1.35		0.00			0.08	0.01	0.15	0.02				0.01	1.26
2011	SBH 05		23.9	5.6	29.8	47.4	3.0	4.7	16.5	33.1	29.0	6.7	1.19	1.26	10.25	1.95	3.6	0.64	4.66	0.02	0.05	0.58	1.00	0.00		0.07	0.05	0.01	0.18	0.03	0.34		0.03	0.01	1.28
2012	SBH 05		24.6	5.6	29.1	47.0	9.1	4.1	14.2	50.0	20.4	1.0	0.57	0.43	5.77	0.28	3.1	0.25	4.37	0.01	0.04	0.41	5.00	0.00		0.04	0.07	0.09	0.15	0.07	0.00	0.08	0.03		1.28
2006	SBH 06			5.1	14.0	17.3		4.1	3.7	8.1																									
2007	SBH 06			4.8	12.2	22.4		5.4	5.8	13.3																		0.01							1.00
2008	SBH 06			4.9	15.0	25.9		4.3	12.7	20.0			0.40	0.16	1.36	0.21	1.7		2.40	0.02		0.19		0.00			0.04	0.01	0.14	0.09				0.02	
2009	SBH 06			4.9	17.0	29.0		5.3	12.3	45.0			0.71	0.15	2.17	0.19	3.5		2.20	0.01		0.72	1.00	0.00		0.08	0.06	0.01	0.03	0.02			0.05	0.03	2.58

2010	SBH 06			5.0	15.7	23.6		5.3	23.8	45.0			0.27	0.12	1.82	0.19	1.8		5.00			0.63		0.00			0.06	0.05	0.05	0.03			0.02		1.31
2011	SBH 06		24.0	5.2	13.3	21.5	10.8	4.5	15.0	32.5	26.0	4.0	0.67	0.17	2.86	0.17	1.3	0.29	3.15	0.02	0.12	0.61	1.50	0.00		0.08	0.04	0.01	0.10	0.03		0.07	0.04		0.50
2012	SBH 06		24.5	5.2	13.8	22.2	186.9	4.2	9.6	32.8	188.4	2.0	0.61	0.24	1.76	0.26	2.0	0.11	23.53	0.01	0.03	0.34	2.00	0.00		0.05	0.12	0.04	0.14	0.08		0.08	0.02	0.00	1.29

APPENDIX 10

Ionic Balance Table for samples from groundwater monitoring boreholes

Ionic Balance Table for Samples from monitoring boreholes

Station ID	Location	Sampling Date	Water Type	Temp	pH (lab)	DO	El. Cond.	TDS	C a	Mg	Na	K	C l	HCO3	SO4	NO3	CN_diss	Cd_diss	Cu_diss	Fe	Mn_diss	NH4	Ni_diss	Ni_tot	Pb_diss	Zn	Electro Neutrality %: (Cations-Anions)/(Cations +Anions)	Total dissolved solids	Sum of Anions (mg/l)	Sum of Cations (mg/l)	Sum of Ions	Calculated density based on the partial molal ionic volumes
				°C		mg/l	uS/cm	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	ug/l	ug/l	ug/l	mg/l	mg/l	mg/l	ug/l	ug/L	ug/l	ug/l	%	mg/l	mmol/l	mmol/l	mg/l	g/cm3
NBH01	Samanhu - Abekoase road	2009	Na-HCO3	23.8	5.6	5.4	65	37	1.66	0.98	4.15	1.06	3.31	35.6	1.0	2.10	4	16.000	19.500	0.104	0.174		65.000		43.000	70.000	-31	50	46.01	178.52	224.53	1.0000
NBH01	Samanhu - Abekoase road	2010	Na-HCO3-Cl	24.6	5.7	5.2	49	33	0.92	0.51	3.41	1.02	3.05	24.3	1.0	3.20	11	7.000	12.500	0.030	0.053		24.500		10.000	31.333	-35	38	42.55	81.24	123.79	1.0000
NBH01	Samanhu - Abekoase road	2011	Na-Ca-HCO3-Cl	24.3	5.8	4.3	41	25	1.64	0.57	4.20	0.95	3.48	15.1	1.0	3.33	11		5.000	0.083	0.050		15.500		74.000	61.143	-10	31	34.06	89.19	123.25	1.0000
																						0.14										
NBH01	Samanhu - Abekoase road	2012	Na-Ca-HCO3	24.9	5.9	4.4	65	40	4.06	0.90	4.85	0.78	3.95	22.3	1.4	5.08	4.0	4.5	15	0.0	0.1	0	36.8		68.667	60.100	-7	44	36.92	127.18	164.10	1.0000
NBH02	Samanhu - Abekoase road	2009	K-Na-HCO3-SO4		6.5	5.0	301	177	3.19	0.73	14.67	30.99	3.33	44.1	15.5	2.34	5.0	14.0	13	0.2	0.8		40.5		40.500	29.000	18	116	70.27	146.39	216.66	1.0001
NBH02	Samanhu - Abekoase road	2010	K-Na-HCO3		6.6	5.4	208	141 89	1.70	0.64	12.13	21.35	2.15	41.3	3.5	4.10	9.0			0.5	0.6		10.0		14.000	40.667	17	88	60.05	87.11	147.16	1.0001
NBH02	Samanhu - Abekoase road	2011	Na-K-HCO3		6.6	5.0	144		3.48	0.64	12.43	16.08	3.08	28.6	1.3	2.88	5.0			0.1	0.3		39.7		42.667	35.500	31	69	41.00	117.15	158.15	1.0001
				24.3														9.0				0										
NBH02	Samanhu - Abekoase road	2012	Na-K-HCO3	24.6	6.5	5.1	193	120	2.40	0.70	12.40	20.20	3.00	33.7	5.6	3.30	0.0	0.0	0	0.2	0.5	0	0.0		100.000	0.000	21	82	45.90	36.30	82.20	1.0001
NBH03	Drivers gallery close to North Agglo	2009	Ca-HCO3-NO3		6.3	6.2	166	95	11.06	1.38	3.11	0.63	4.82	51.8	7.6	16.81	4.0	22.0	14	0.0	0.0		54.0		50.000	29.500	-27	97	85.00	135.20	220.20	1.0001
NBH03	Drivers gallery close to North Agglo	2010	Ca-HCO3		6.5	5.6	124	82	6.80	1.61	3.18	0.77	2.60	56 56.5	5.5	5.80	5.0	8.0	10	0.1	0.0		32.0		10.000	168.000	-31	83	74.90	230.38	305.28	1.0001
NBH03	Drivers gallery close to North Agglo	2011	Ca-HCO3		6.6	4.7	123	77	11.66	1.74	5.36	0.57	3.78		10.5	6.49	6.0		16	0.1	0.0		19.0		45.667	87.250		97	83.65	142.07	225.72	1.0001
																											-17					
				24.4																		0										
NBH03	Drivers gallery close to North Agglo	2012	Ca-HCO3	24.6	6.5	5.2	133	82	9.40	1.60	4.30	0.70	3.80	53.8	7.3	11.10	0.0	0.0	0	0.1	0.0	0	0.0		0.000	100.000	-24	92	76.20	116.10	192.30	1.0001
NBH03	Drivers gallery close to North Agglo Phase 5	2012	Ca-Na-HCO3-NO3 Ca-Mg-Na-HCO3	24.8	6.6	4.4	120	75	7.99	1.65	5.60	0.63	3.85	50.9	5.4	15.34	3.0	3.0	38	0.1	0.1	0	31.7		66.667	62.800	-24	92	78.63	151.00	229.63	1.0001
NBH04	heap leach	2009			6.4	5.3	249	145	11.49	4.93	8.17	0.53	5.38	88.7	1.5	2.43	4.0	38.0	14	0.1	0.1		8.0		18.000	28.000		123	102.01	112.71	214.72	1.0001
																											-11					
NBH04	Phase 5 heap leach	2010	Ca-Mg-Na-HCO3		6.6 6.6	5.1 4.3	224	148	8.34	4.66 4.91	7.35 8.66	0.83 0.69	3.00 6.53	101.2	1.7 1.4	2.90 2.87	6.0 5.0		11 9	0.2 0.1	0.4 0.1		10.0 24.0			262.667	-22	131	114.75	304.72	419.47	1.0001 1.0001
NBH04	Phase 5 heap leach	2011	Ca-Mg-Na-HCO3				198	115	12.46					72.3												143.000	-1	110	88.33	209.89	298.22	
				24.6														7.0				0			34.000							
NBH04	Phase 5 heap leach	2012	Na-Mg-Ca-HCO3	24.4	6.6	3.9	183	115	7.25	4.46	8.61	0.76	6.58	81.5	1.6	4.39	4.0	4.5	36	0.0	0.1	0	49.8		45.600	64.700	-18	115	98.12	176.18	274.30	1.0001
NBH04	Phase 5 heap leach	2012	Ca-Mg-Na-HCO3 Na-SO4-HCO3	24.5	6.5 6.0	4.7 5.3	213	131	9.90	4.70 2.78	8.20	0.70 1.42	5.40	85.9 69	1.5	3.10	0.0 7.0	0.0	0	0.1 0.1	0.2 0.2	0	0.0		0.000	100.000	-13	120	96.10	123.80	219.90	1.0001 1.0002
NBH06	North Containment area	2009					553	317	12.07		49.89		14.30		100.6	13.97		27.0	17				12.0		10.000	43.500	-12	264	204.84	165.36	370.20	
NBH06	North Containment area	2010	Na-Ca-HCO3-SO4		6.2	5.0	446	298	11.96	2.57	38.91	1.79	7.20	113.8	34.0	2.20	7.0	7.0		0.9	0.6		54.0		20.000	55.333	-4	213	164.20	172.13	336.33	1.0002
NBH06	North Containment area	2011	Na-HCO3-SO4 Na-Ca-HCO3		6.1 5.8	5.2 4.8	500	307	12.00	2.70 2.67	44.40	1.60 1.04	10.80	91.4 64.1	67.3	8.10 1.13	0.0 5.0	0.0	0	0.5 0.3	0.4 0.8	0	0.0 5.0		0.000	0.000	-9 -10	239	177.60	61.10 99.24	238.70	1.0002 1.0001
NBH09	North Containment area	2009					228	130	7.12		12.07		8.06		5.4			30.0	17						26.000	24.000		103	83.72		182.96	

NBH09	North Heap Leach road close to ADR	2010	Na-HCO3		5.9	4.7	149	99	4.02	1.82	10.54	1.38	6.35	54.7	9.0		6.0		20	0.2	0.8		7.0		20.000	213.333	-18	89	76.05	258.85	334.90	1.0001
NBH09	North Heap Leach road close to ADR	2011	Na-Ca-HCO3-Cl	24.5	6.0	4.8	138	86	9.46	1.99	13.11	0.93	11.68	74.6	6.3	1.27	6.0		15	0.2	0.4	0	17.0		32.000	108.125	-15	120	100.05	166.21	266.26	1.0001
NBH09	North Heap Leach road close to ADR	2012	Na-Ca-HCO3	24.7	6.3	3.8	170	110	7.90	2.82	14.57	1.75	7.97	64	7.3	3.82	4.0	7.0	91	0.1	0.8	0	54.3		82.200	84.700	-5	112	87.32	264.69	352.01	1.0001
NBH09	North Heap Leach road close to ADR	2012	Na-Ca-HCO3	24.6	6.0	4.5	171	106	7.10	2.30	12.60	1.30	8.50	64.4	7.0	2.10	0.0	0.0	0	0.2	0.7	0	0.0		0.000	100.000	-12	107	82.20	124.20	206.40	1.0001
NBH13	TSF 1	2009	Na-Ca-HCO3-Cl		6.4	4.2	378	218	11.55	4.50	24.69	1.13	17.73	88.3	13.8	2.95	10.0	13.0	12	0.1	0.8				31.500	10.500	-5	166	132.78	77.71	210.49	1.0001
NBH13	TSF 1	2010	Na-HCO3		6.7 6.8	4.4 5.0	336	225	8.45	4.63 5.66	23.37	1.19 1.57	14.55 11.43	100.7	21.0 14.8		12.0 12.0	11.0	18	0.4 0.1	0.9 0.8		10.0 17.7		10.000 27.250	64.333 74.167	-14 15	175	148.25	141.54	289.79	1.0001 1.0002
NBH13	TSF 1	2011	Na-Ca-HCO3				364	226	14.40					99.3					12									193	139.40	169.11	308.51	
											41.82																					
				24.9												1.47						1										
NBH13	TSF 1	2012	Na-Ca-Mg-HCO3-Cl	24.2	6.8	3.6	333	206	13.41	6.47	29.20	1.49	20.38	111.9	8.7	4.93	7.0	3.5	44	0.1	1.0	0	25.4		102.200	31.714	-2	198	153.30	156.56	309.86	1.0002
NBH13	TSF 1	2012	Na-Ca-HCO3	24.5	6.7	4.3	353	219	12.00	5.30	29.80	1.30	16.00	100	14.6	3.10	0.0	0.0	0	0.2	0.9	1	0.0		0.000	0.000	-1	184	134.10	50.00	184.10	1.0001
NBH14	TSF 1	2009	Ca-Na-Mg-HCO3		7.0	5.3	519	301	27.81	8.05	23.70	4.03	8.97	136.6	5.7	1.84	5.0	18.0	11	0.1	0.1				38.500	7.000	9	217	158.08	99.64	257.72	1.0002
NBH14	TSF 1	2010	Ca-Na-Mg-HCO3		7.1 6.9	4.7 5.2	462	308	22.14 33.21	7.86 9.73	25.21	1.28 1.26	10.65 12.93	171.5	7.0 7.8	2.80 1.12	7.0 7.0	6.5	10 9	0.0 0.1	0.9 0.8				10.000 50.750	47.500 44.333	-6	249	198.95	121.37	320.32	1.0002 1.0002
NBH14	TSF 1	2011	Ca-Na-Mg-HCO3		7.2	3.9	442	275	32.02	9.08	34.00 31.16	1.71	19.58	125.7	7.0	2.08	4.0	25.0	76	0.1	0.5				90.000	33.333	21	227	154.90	181.48	336.38	1.0002
NBH14	TSF 1	2012	Ca-Na-Mg-HCO3		7.0	4.8	431	265	28.80	8.70		2.10	13.00	123.1	6.9	2.00	0.0	3.5	0	0.1	0.6				0.000	0.000		227	156.19	240.31	396.50	1.0002
NBH14	TSF 1	2012	Ca-Na-Mg-HCO3				464	287						139.2				0.0									15	230	161.50	68.80	230.30	
																											10					
																						0										
				24.9 24.4																		0	24.0 53.0									
				24.7							28.50											0	0.0									
NBH15	TSF 1	2009	Na-Ca-HCO3-Cl-SO4		6.4	5.5	338	196	9.32	1.92	30.06	3.17	18.63	70.3	23.9	5.15	5.0	26.5	9	0.1	1.5				12.500	26.500	-4	164	122.94	107.99	230.93	1.0001
NBH15	TSF 1	2010	Na-HCO3-Cl-SO4		6.7	5.0	384	255	7.71	1.91	41.84	3.32	30.00	66.8	32.5	12.30	6.0			0.1	1.0		38.0		31.000	43.000	-6	198	147.60	136.82	284.42	1.0002
NBH15	TSF 1	2011	Na-Ca-HCO3	24.5	7.0	4.7	434	271	27.04	5.05	50.73	2.13	9.13	131.9	22.8	5.50	5.0	8.5	12	0.1	0.4	0	21.0		27.500	42.500	15	255	174.62	169.00	343.62	1.0002
NBH15	TSF 1	2012	Na-Ca-HCO3	24.2	6.8	4.7	371	231	15.60	3.20	39.60	2.80	16.90	85	26.1	8.40	0.0	0.0	0	0.1	0.9	0	0.0		0.000	0.000	6	199	136.70	62.20	198.90	1.0002
NBH15	TSF 1	2012	Na-Ca-HCO3-SO4	24	6.9 7.7	3.7 5.9	328	199	18.23 41.56	3.75 6.61	35.71 43.63	2.73 4.71	9.70 5.93	70.9 81.3	25.1	10.55	4.0 8.0	4.0	76	0.1 0.1	0.5 0.0	0	38.4		73.600 15.000	31.429	14	178	120.61	210.31	330.92	1.0001 1.0002
NBH16	TSF 1	2009	Ca-Na-HCO3		7.7	5.7	546	316	34.31	6.35		1.56	2.45	74.5	2.8	2.74	7.0	10.0	11		0				8.000		49	189	100.77	117.53	218.30	1.0001
NBH16	TSF 1	2010	Ca-Na-Mg-HCO3				489	316							1.0	2.80			19.5				16.3			51.000	41	145	87.75	150.64	238.39	
											21.57																					
NBH16	TSF 1	2011	Ca-Na-Mg-HCO3	24.7	7.8	5.4	449	280	48.52	7.81	26.79	1.36	5.45	106.9	1.0	2.63	6.0		19.0	0.1	0	0.1	19.0		32.750	35.167	36	201	122.51	157.78	280.29	1.0002
NBH16	TSF 1	2012	Ca-Na-HCO3	23.9	7.7	4.1	438	266	43.31	6.35	25.70	2.54	5.32	80.2	1.0	2.99	4.0	3	30.0	0.1	0	0.0	22.3		48.333	29.500	43	168	93.90	162.77	256.67	1.0002
NBH16	TSF 1	2012	Ca-Na-HCO3	24.3	7.7 6.9	5.3 5.0	480	295	41.90 20.68	6.80 7.91	29.40 24.60	2.50 1.11	4.80 6.28	85.7	1.5 1.4	2.80 3.35	0.0 6.0	0	0.0 9.7	0.1 0.1	0	0.1	0.0		0.000	0.000	42	176	95.30	80.70 94.85	176.00	1.0002 1.0002
NBH17	TSF 1	2009	Na-Ca-Mg-HCO3				433	250						129.1				14			0				38.000	17.000	8	195	146.13		240.98	
NBH17	TSF 1	2010	Na-Ca-Mg-HCO3		7.1	5.0	397	266	16.41	7.32	22.99	1.46	2.70	140.8	1.3	3.50	5.0	9	29.0	0.0	1		33.0			76.000	0	197	153.30	195.97	349.27	1.0002

NBH17	TSF 1	2011	Ca-Na-Mg-HCO3	24.5	7.2	4.6	367	230	26.95	8.71	27.66	1.41	6.73	137.4	1.8	3.33	7.0	12	16.7	0.1	0	0.1	35.0		62.400	33.429	13	215	156.84	162.13	318.97	1.0002
NBH17	TSF 1	2012	Na-Ca-Mg-HCO3	24.4	7.1	4.6	391	243	21.30	7.80	25.90	1.50	5.50	134.1	1.4	3.60	0.0	0	0.0	0.0	0	0.1	0.0		0.000	0.000	8	202	145.20	57.00	202.20	1.0002
NBH17	TSF 1	2012	Na-Ca-Mg-HCO3	24.3	7.2	3.8	367	227	21.33	7.36	28.19	1.85	6.13	129	1.3	4.06	4.0	3	18.0	0.0	0	0.1	28.0		41.000	26.000	10	200	145.13	134.04	279.17	1.0002
NBH18	TSF 1	2009	Ca-Na-Mg-HCO3		6.3	4.9	231	125	11.10	3.82	10.61	0.55	5.06	66	5.9	2.93	5.0	26	11.0	0.1	0		14.0		64.000	22.500	-1	106	84.85	99.91	184.77	1.0001
NBH18	TSF 1	2010	Na-Ca-Mg-HCO3		6.3	4.9	167	109	6.65	2.71	8.54	0.49	2.55	53.8	1.0	11.40	5.0		16.0	0.0	0				55.000	81.000	-10	88	73.75	115.68	189.43	1.0001
NBH18	TSF 1	2011	Na-Ca-HCO3-NO3	24.4	6.2	4.6	188	117	10.70	3.23	18.03	0.59	10.13	59.3	4.8	21.27	6.0	6	27.5	0.1	0	0.1	49.3		45.000	45.833	-3	129	101.75	161.60	263.35	1.0001
NBH18	TSF 1	2012	Na-Ca-HCO3	24.5	6.3	4.6	210	127	9.50	3.40	15.60	0.70	8.40	62.1	5.7	14.10	0.0	0	0.0	0.1	0	0.0	0.0		100.000	0.000	-5	120	90.50	29.50	120.00	1.0001
NBH18	TSF 1	2012	Na-Ca-HCO3-Cl	24.6	6.5	4.0	254	159	9.72	3.76	25.10	1.09	15.70	69.5	11.0	20.94	3.0	3	63.0	0.1	0	0.0	30.7		88.000	45.500	-6	158	120.40	182.41	302.81	1.0001
NBH19	Heap Leach road to ADR	2009	Ca-Na-HCO3		6.6	5.3	321	183	17.35	2.39	12.48	0.77	4.50	115.1	3.3	1.48	5.0	18	6.0	0.1	0		12.0		20.000	42.500	-13	158	129.33	111.62	240.95	1.0001
NBH19	Heap Leach road to ADR	2010	Ca-Na-HCO3		6.7	4.9	252	167	14.28	3.54	11.17	0.84	2.45	106.3	4.2	2.70	5.0			0.0	0		15.5			110.333	-12	146	120.65	155.90	276.55	1.0001
NBH19	Heap Leach road to ADR	2011	Ca-Na-HCO3	24.5	6.7	4.7	257	161	25.22	2.87	14.03	0.80	6.88	93.3	1.0	3.26	6.0	10	39.0	0.1	1	1.0	27.5		51.6	63.1	10	150	110.63	184.39	295.02	1.0001
NBH19	Heap Leach road to ADR	2012	Ca-Na-HCO3	24.4	6.9	4.0	249	154	16.59	2.30	13.18	0.94	5.43	97.1	2.0	5.91	4.0	4	64.5	0.0	0	0.1	48.0		36.3	91.6	-8	144	114.83	241.86	356.69	1.0001
NBH19	Heap Leach road to ADR	2012	Ca-Na-HCO3	24.4	6.8 6.9	4.7 5.0	270	166	18.40 51.52	2.80 5.49	12.70 13.24	0.80 2.15	4.80 6.02	103	2.6 2.1	3.30 3.03	0.0 6.0	0	0.0 7.0	0.1 0.0	0	0.5	0.0		0.0	100.0 22.0	-5	150	114.00	135.60	249.60	1.0001 1.0002
NBH20	Phase 5 heap leach	2009	Ca-Na-HCO3		7.1	4.9	540	315	35.22	6.98	12.06	1.03	1.60	100.4	4.0	3.60	6.0	24	30.0	0.1	1		12.0		30.0	116.3	32	185	117.59	138.11	255.70	1.0002
NBH20	Phase 5 heap leach	2010	Ca-Mg-HCO3				426	283						146.2							1		24.0		23.5			211	161.40	226.13	387.53	
																											6					
NBH20	Phase 5 heap leach	2011	Ca-Na-HCO3	24.8	7.0	4.7	409	253	56.36	6.64	15.64	0.80	3.95	120.5	4.2	3.40	6.0		21.0	0.1	0	0.1	46.5		43.8	58.4	29	212	138.18	205.71	343.89	1.0002
NBH20	Phase 5 heap leach	2012	Ca-Na-HCO3	24.6	7.0	4.7	454	281	50.70	6.20	14.20	1.30	3.70	120.2	3.5	3.50	0.0	0	0.0	0.1	0	0.1	0.0		0.0	100.0	25	204	131.10	172.90	304.00	1.0002
NBH20	Phase 5 heap leach	2012	Ca-Na-HCO3	24.4	7.2	4.2	440	272	59.89	5.60	16.05	1.03	3.25	113.9	3.6	3.90	5.0	6	63.0	0.0	0	0.1	31.3		56.0	53.0	33	208	129.86	236.41	366.27	1.0002
NBH21	Phase 5 heap leach	2009	Ca-Na-HCO3		6.8	5.9	342	199	26.39	2.71	9.66	1.47	4.04	102.3	6.0	2.26	6.0	22	7.3	0.0	1		21.0		25.0	20.5	2	156	120.60	111.70	232.30	1.0001
NBH21	Phase 5 heap leach	2010	Ca-Na-HCO3-SO4		6.8 6.7	5.2 5.3	460	303	22.84 61.95	4.81 6.47	24.77	1.52 1.86	2.15 6.42	137.7	55.5 87.9	1.90 2.17	8.0 6.0		8.0		0				10.0 53.8	92.3 76.3	-14 10	252	205.25	154.70	359.95	1.0002 1.0003
NBH21	Phase 5 heap leach	2011	Ca-Na-HCO3-SO4				602	375			51.10			167.4					12.5		0							386	270.40	226.93	497.33	
				25																0.1		0.1	16.4									
NBH21	Phase 5 heap leach	2012	Ca-Na-HCO3-SO4	24.8	6.8 6.5	5.1 5.9	464	290	40.10	5.20 1.55	31.10	1.60 2.05	4.30 3.50	132.6	51.0 5.0	2.10 2.80	0.0 6.0	0	0.0	0.1 0.0	1	0.1	0.0		0.0 9.0	100.0	6	269 86	190.40	178.70 96.93	369.10	1.0002 1.0001
NBH22	Phase 5 heap leach	2009	Na-Ca-HCO3		6.4	5.3	226	148	8.36	1.14	17.37 11.01	1.18	11.60	45.3	22.5	11.96	9.0	9		0.1	0				54.3	58.5	16	109	62.60	104.75	159.53	1.0001
NBH22	Phase 5 heap leach	2010	Na-Ca-HCO3-SO4-Cl	24.9			130	81	9.00					40.5				14	13.0		0	0.1	10.5			44.6			95.56		200.31	
																											-21					
NBH22	Phase 5 heap leach	2011	Ca-Na-HCO3-SO4-Cl	24.6	6.5	4.8	110	68	8.92	1.13	7.16	1.43	9.90	39.6	14.1	3.60	4.0	3	13.0	0.1	0	0.1	11.0		50.0	63.0	-17	87	71.34	108.66	180.00	1.0001
NBH23	Phase 5 heap leach	2009	Ca-Na-HCO3		5.9	5.0	166	97	7.55	1.35	8.29	1.38	7.00	41.3	8.4	2.39	5.0	12	9.7	0.0	0		19.0		17.5	22.0	-9	78	64.12	81.97	146.09	1.0001
NBH23	Phase 5 heap leach	2010	Na-Ca-HCO3-Cl Na-Ca-HCO3-SO4		5.9 5.9	4.5 4.5	129 99	86	4.51 4.25	1.45 1.69	7.99 8.87	0.59 0.49	8.00 5.07	58.7 38.1	6.0 9.0	5.20 3.20	10.0		15.0 19.5	0.0 0.1	0		28.0 52.0			106.0	-32	93	87.90 62.37	163.82	251.72	1.0001 1.0001
NBH23	Phase 5 heap leach	2011						62									7.0				0					86.9	-13	71		174.11	236.48	
				24.9																		0.1			35.2							

NBH23	Phase 5 heap leach	2012	Na-Ca-HCO3	25	6.0	4.0	132	81	7.00	1.00	9.00	1.00	6.00	44	8.0	4.00	0.0	0	0.0	0.0	0	0.0	0.0		0.0	0.0	-14	80	62.00	18.00	80.00	1.0001
NBH23	Phase 5 heap leach	2012	Ca-Na-HCO3	25	6.1	3.8	135	80	12.52	1.51	9.45	0.90	5.20	39.3	9.3	3.47	4.0	3	39.3	0.1	0	0.0	32.3		49.0	46.4	7	82	61.45	146.01	207.46	1.0001
NBH24	Phase 5 heap leach	2009	Na-Ca-HCO3-SO4		5.8	5.2	133	78	4.04	1.19	7.94	1.17	4.32	37.9	9.9	5.06	5.0	9	10.0	0.0	0		16.0		22.0	21.5	-20	72	62.14	70.78	132.92	1.0001
NBH24	Phase 5 heap leach	2010	Na-HCO3-SO4		5.9	5.0	97	68	1.77	1.13	7.12	0.41	3.55	33	8.0	5.90	4.0		15.5	0.0	0				20.0	126.7	-27	61	54.45	152.98	207.43	1.0000
NBH24	Phase 5 heap leach	2011	Na-HCO3-Cl-SO4 Na-HCO3-SO4	25.1	5.9	4.9	88	55	3.25	1.22	9.02	0.33	11.48	28.6	9.5	4.60	4.0		17.0	0.1	0	0.1	37.5		24.3	68.8	-23	69	58.42	137.33	195.75	1.0001
NBH24	Phase 5 heap leach	2012		24.8	6.0	3.7	89	56	2.01	1.23	9.02	0.46	4.38	25.3	7.8	4.11	3.0		89.0	0.1	0	0.1	37.7		78.5	57.9		55	44.74	202.58	247.32	1.0000
																											-11					
																		5														
NBH24	Phase 5 heap leach	2012	Na-HCO3-SO4-Cl	24.9	5.9	4.7	102	64	2.80	1.20	8.30	0.60	5.90	31.2	8.8	4.90	0.0	0	0.0	0.0	0	0.1	0.0		0.0	100.0	-20	64	51.00	113.30	164.30	1.0000
NBH26	Phase 5 heap leach	2009	Ca-Na-HCO3		5.9	5.0	240	136	9.27	3.23	10.49	1.74	10.03	75.4	12.7	1.04	6.0	9	8.3	0.1	0		13.0		27.0	34.0	-18	124	105.18	88.88	194.06	1.0001
NBH26	Phase 5 heap leach	2010	Na-HCO3-SO4		6.0 6.0	4.6 5.1	148	99	3.68 4.97	2.04 2.23	9.85	0.86 0.73	7.40 9.35	46.8 39.5	24.0 14.3	5.60 2.14	5.0 4.0		20.5	0.0 0.1	0		50.0 51.7		30.0 21.5	149.3	-31	101 90	88.80 69.63	236.57	325.37	1.0001 1.0001
NBH26	Phase 5 heap leach	2011	Na-Ca-HCO3-SO4-Cl				150	94			15.65								8.3		0					102.9	-5			202.67	272.30	
				25.1														16				0.1										
NBH26	Phase 5 heap leach Phase	2012	Na-Ca-Mg-HCO3-Cl	24.6 24.8	6.2 6.0	3.9 4.6	202	126	9.08 6.70	4.29 2.90	13.74 12.40	1.13 1.10	12.13	70.7 58.1	15.9 16.7	4.17 3.20	4.0 0.0	6	34.7	0.1 0.1	0	0.3 0.2	21.5 0.0		70.3 0.0	60.9	-13	132	107.08 87.90	151.68	258.76	1.0001 1.0001
NBH26	5 heap leach TSF 2	2012	Na-Ca-HCO3-SO4 Ca-Na-HCO3		7.7	5.2	185	114	44.56	2.62	19.42	3.96	9.70	122.4	21.3	3.96	4.0	0	0.0	0.1	0		30.0		16.0	100.0	-17	112	159.00	123.60	211.50	1.0002
NBH28		2009					522	300					7.35					6	7.7		0					14.0	11	226		128.35	287.35	
NBH28	TSF 2	2010	Ca-Na-HCO3-SO4		7.0	5.4	503	333	43.98	2.36	23.95	1.91	7.05	129.2	35.5	2.60	6.0		10.0	0.0	0		18.0		30.0	72.0	6	247	180.35	172.52	352.87	1.0002
NBH28	TSF 2	2011	Ca-Na-HCO3 Na-SO4	0	7.2	5.3	528	337	44.30	2.40	27.80	2.30	7.20	125.8	25.1	3.30	0.0	0	0.0	0.0	0	0.0	0.0		0.0	0.0	13	238 62	161.40	77.00	238.40	1.0002
NBH28	TSF 2	2012			6.9		560	378		2.16		1.12			18.6		7.0	6											25.60	49.42	75.02	1.0001
											40.14																67					
NBH30	TSF 2	2009	Ca-Na-HCO3-Cl		5.9	4.9	125	73	5.30	0.87	5.80	1.20	8.31	39	2.6	1.08	5.0	24	7.3	0.1	0		14.0		38.0	29.0	-20	65	55.96	87.80	143.76	1.0000
NBH30	TSF 2	2010	Ca-Na-HCO3 Na-Ca-HCO3-Cl		6.5 6.2	5.1 5.2	129	86	6.31 7.62	1.14 1.46	6.21 9.97	1.07 0.95	5.80 9.98	79.3 43	2.0 3.5	2.50 2.28	9.0 5.0		10.0 20.5	0.0 0.1	1				9.0	77.7 88.9	-36	105 80	98.60 63.76	103.11	201.71	1.0001 1.0001
NBH30	TSF 2	2011					121	75													1				34.3					246.66	310.42	
																											-5					
				25.2																		0.1	116.5									
NBH30	TSF 2	2012	Na-HCO3-Cl	24.8	6.2	4.2	121	76	3.68	1.44	10.06	0.94	10.67	40.1	4.4	3.17	4.0	4	32.5	0.1	1	0.1	21.0		84.7	37.8	-17	76	62.62	112.07	174.69	1.0001
NBH30	TSF 2	2012	Na-Ca-HCO3-Cl Ca-Na-HCO3	25	6.2 6.0	4.8 5.2	124	77	5.70	1.20 1.44	8.00 7.90	1.00 1.66	8.70 5.99	50.4 55.9	3.1 4.0	2.30 1.85	0.0 5.0	0	0.0	0.1 0.1	1	0.1	100.0		0.0	100.0	-20	81	64.70 72.74	216.60	281.30	1.0001 1.0001
NBH31	TSF 2	2009					190	111	13.89									114	15.0		0		20.0		41.3	20.5		93		194.52	267.26	
																											0					
NBH31	TSF 2	2010	Na-Ca-HCO3		6.4	4.8	165	109	6.88	1.78	9.21	0.91	4.90	81.3	5.0	4.20	7.0		17.0	0.0	0				60.0	70.0	-28	115 98	102.40	106.17	208.57	1.0001
NBH31	TSF 2	2011	Na-Ca-HCO3		6.2	4.3	161	100	9.73	2.20	13.27	1.10	8.63	49.9	8.8	3.07	5.0		15.0	0.1	0				21.0	70.8	0		75.59	291.53	367.12	1.0001
				25.1														5				0.1	174.0									
NBH31	TSF 2	2012	Na-Ca-HCO3	24.7	6.5	4.0	183	114	8.38	2.41	16.89	1.12	7.33	58.7	9.9	1.70	3.0	5	63.0	0.1	0	0.1	35.5		51.5	72.2	0	107	80.80	204.95	285.75	1.0001
NBH31	TSF 2	2012	Na-Ca-HCO3	24.9	6.3	4.6	175	109	9.70	2.00	11.80	1.20	6.70	61.4	6.9	2.70	0.0	0	0.0	0.1	0	0.1	100.0		0.0	100.0	-7	103	77.90	225.10	303.00	1.0001
NBH32	TSF 2	2009	Ca-Na-HCO3		6.4 6.4	4.9 4.8	253	151	23.27 17.74	1.91 1.42	9.20 9.92	2.40 1.73	6.17 4.05	88.7 96.3	11.6 20.5	2.89	5.0 7.0	12	7.0	0.0 0.0	0		19.0		25.5 22.0	4.0	-3	147	114.33	79.17	193.50	1.0001 1.0001
NBH32	TSF 2	2010	Ca-Na-HCO3-SO4		6.1	4.6	213	145	12.53	1.53	17.22	1.18	9.52	48.9	19.0	15.70	8.0		14.0	0.1	0				41.3	68.0	-23	168	143.55	113.09	256.64	1.0001
NBH32	TSF 2	2011	Na-Ca-HCO3-SO4				185	114								3.15			13.2		0		41.3			67.1	1	114	88.57	154.67	243.24	
				24.8																		0.2										
NBH32	TSF 2	2012	Ca-Na-HCO3-SO4	24.9	6.3	4.5	208	130	15.60	1.60	14.10	1.70	7.70	71.4	16.7	6.40	0.0	0	0.0	0.0	0	0.1	0.0		0.0	0.0	-7	136	102.30	33.50	135.80	1.0001
NBH32	TSF 2	2012	Na-Ca-HCO3-SO4-Cl	25	6.2	3.8	182	110	8.99	1.70	20.11	1.53	11.12	51.8	15.9	3.82	6.0	3	19.7	0.1	1	0.1	29.5		62.3	48.8	-1	116	88.74	134.28	223.02	1.0001

NBH34	Primary Nursery	2009	Na-Ca-HCO3		6.2	4.9	141	81	5.13	2.01	6.15	1.73	3.96	47.9	1.3	4.37	4.0	14	8.0	0.1	0		21.0		46.5	92.5	-15	73	61.48	150.63	212.11	1.0001
NBH34	Primary Nursery	2010	Na-Mg-HCO3		6.6	5.2	118	79	3.17	2.28	8.41	0.87	1.95	53.8	1.0	2.70	5.0		8.0	0.0	0		10.0		8.0	133.7	-15	75	64.45	166.63	231.08	1.0001
NBH34	Primary Nursery	2011	Na-Ca-HCO3	24.1	6.4	5.8	88	55	3.47	1.55	7.06	0.42	4.57	33	1.3	4.46	4.0		22.0	0.0	0	0.0	31.0		34.5	96.9	-11	56	47.42	162.48	209.90	1.0000
NBH34	Primary Nursery	2012	Na-Ca-HCO3	24.5	6.4	5.2	107	66	3.40	1.80	7.20	0.90	3.90	40.8	1.4	3.90	0.0	0	0.0	0.0	0	0.0	0.0		0.0	100.0	-14	64	50.20	113.40	163.60	1.0000
NBH34	Primary Nursery	2012	Na-HCO3-Cl	24.9	6.5	4.9	81	50	1.83	1.39	7.11	0.54	5.22	28.5	2.0	3.99	3.0	8	45.3	0.1	0		25.5		43.3	94.7	-15	51	42.87	184.45	227.32	1.0000

Station ID	Location	Sampling Date	Water Type	Temp °C	pH (lab)	DO mg/l	El. Cond. uS/cm	TDS mg/l	C a mg/l	Mg mg/l	Na mg/l	K mg/l	C l mg/l	HCO3 mg/l	SO4 mg/l	NO3 mg/l	CN_diss ug/l	Cd_diss ug/l	Cu_diss ug/l	Fe mg/l	Mn_diss mg/l	NH4 mg/l	Ni_diss ug/l	Ni_tot ug/L	Pb_diss ug/l	Zn ug/l	Electro Neutrality %: (Cations-Anions)/(Cations +Anions) %	Total dissolved solids mg/l	Sum of Anions (mg/l) mmol/l	Sum of Cations (mg/l) mmol/l	Sum of Ions mg/l	Calculated density based on the partial molal ionic volumes g/cm3
NBH35	TSF 3	2010	Ca-HCO3	24.7	6.6	4.9	561	344	107.61	3.85	10.13	1.44	6.43	104.1	11.9	3.12	13.0	7	16.0	0.1	0	0.2	47.8		21.6	117.5	47	250	138.90	311.82	450.72	1.0003
NBH35	TSF 3	2011	Ca-HCO3	24.8	6.6	4.6	478	295	83.20	2.90	9.50	1.10	7.20	103.5	8.9	2.90	0.0	0	0.0	0.1	0	0.1	0.0		100.0	100.0	39	220	122.80	197.10	319.90	1.0002
NBH35	TSF 3	2012	Ca-HCO3	24.8	6.6	4.2	396	245	58.86	1.97	8.81	0.74	7.95	102.9	6.0	2.64	6.0	5	46.0	0.1	0	0.1	30.3		88.5	53.3	25	191	125.76	205.34	331.10	1.0002
NBH36	TSF 3	2010	Ca-Na-HCO3	24.6	8.5	5.0	450	271	47.46	1.86	16.02	6.30	7.11	115.5	28.4	2.28	5.0		22.0	0.1	0	0.2	67.0		30.0	18.0	10	226	158.96	178.89	337.85	1.0002
NBH36	TSF 3	2011	Ca-Na-HCO3-SO4	24.6	7.4	4.3	337	208	33.04	3.55	15.42	7.27	6.67	91.8	39.0	3.96	4.0	5	43.3	0.1	0	0.1	23.3		72.5	35.0	4	202	145.82	166.32	312.14	1.0002
NBH36	TSF 3	2012	Ca-Na-HCO3-SO4	24.6	8.0	4.6	393	240	40.20	2.70	15.70	6.80	6.90	103.7	33.7	3.10	0.0	0	0.0	0.1	0	0.1	0.0		100.0	0.0	7	214	147.90	65.70	213.60	1.0002
NBH37	TSF 3	2010	Ca-Na-HCO3	24.8	6.3	5.0	133	81	10.01	1.65	8.92	1.34	7.31	38.3	4.4	9.24	4.0		19.3	0.1	0	0.1	36.0		12.5	63.8	0	82	63.31	141.23	204.54	1.0001
NBH37	TSF 3	2011	Ca-Na-HCO3	25.1	6.4	4.0	195	122	16.99	2.94	10.89	1.26	5.45	59.2	4.4	9.56	4.0	4	33.3	0.1	0	0.0	19.5		65.0	36.5	7	112	83.04	125.89	208.93	1.0001
NBH37	TSF 3	2012	Ca-Na-HCO3	25	6.4	4.5	164	102	13.50	2.30	9.90	1.30	6.40	48.7	4.4	9.40	0.0	0	0.0	0.1	0	0.1	0.0		0.0	100.0	4	97	69.10	127.40	196.50	1.0001
NBH38	TSF 3	2010	Ca-HCO3-Cl	25.1	7.2	4.8	310	184	40.42	3.93	10.89	2.66	24.61	115.9	9.3	4.60	7.0		20.3	0.1	0	0.1	68.0		43.7	54.3	0	213	161.49	200.67	362.16	1.0002
NBH38	TSF 3	2011	Ca-HCO3	25	6.8	4.1	311	192	44.17	3.41	8.95	1.30	5.50	120.1	4.4	6.91	5.0	5	25.7	0.1	0	0.0	39.3		90.8	46.4	11	195	142.14	173.97	316.11	1.0002
NBH38	TSF 3	2012	Ca-HCO3	25	7.0	4.4	310	188	42.30	3.70	9.90	2.00	15.10	118	6.8	5.80	0.0	0	0.0	0.1	0	0.0	100.0		100.0	100.0	5	204	145.90	258.00	403.90	1.0002
NBH39	TSF 3	2010	Ca-HCO3	25	7.9	4.8	398	244	57.84	4.24	10.57	3.05	4.70	94.4	9.1	2.40	6.0		26.3	0.0	0	0.1	16.0		57.5	29.3	33	187	116.91	147.39	264.30	1.0002
NBH39	TSF 3	2011	Ca-HCO3	25.1	7.2	3.9	374	230	56.45	3.75	10.25	1.15	5.58	106.4	6.6	2.95	7.0	3	43.0	0.1	0	0.0	22.7		85.3	43.9	26	194	128.74	183.71	312.45	1.0002
NBH39	TSF 3	2012	Ca-HCO3	25	7.5	4.3	386	237	57.10	4.00	10.40	2.10	5.10	100.4	7.8	2.70	0.0	0	0.0	0.1	0	0.1	0.0		100.0	0.0	29	190	116.30	73.70	190.00	1.0002
NBH40	TSF 3	2010	Ca-Na-HCO3	25.3	6.7	4.5	231	142	21.03	2.85	15.80	0.81	7.14	72.3	6.7	23.03	6.0		23.0	0.1	0	0.1	57.0		97.0	55.8	3	150	115.23	176.42	291.65	1.0001
NBH40	TSF 3	2011	Na-Ca-HCO3-SO4-NO3	25.1	6.3	4.3	238	149	9.74	2.54	21.61	1.23	4.45	60.3	21.4	27.06	10.0	3	41.0	0.1	0	0.0	22.0		93.3	42.4	-9	149	123.30	143.79	267.09	1.0001
NBH40	TSF 3	2012	Na-Ca-HCO3-NO3 Ca-	25.2	6.5	4.4	235	146	15.40	2.70	18.70	1.00	5.80	66.3	14.1	25.00	0.0	0	0.0	0.1	0	0.1	0.0		100.0	0.0	-3	149	111.30	38.10	149.40	1.0001
NBH42	TSF 3	2010	Na-SO4-HCO3	25	7.5	4.3	697	431	74.54	7.37	54.70	3.12	12.26	105.3	95.3	1.84	10.0		33.5	0.1	0	0.0	57.5		79.0	41.1	25	355	225.13	272.29	497.42	1.0003

NBH42	TSF 3	2011	Na-Ca-SO4-HCO3	25	7.3	4.6	680	423	54.96	4.60	67.42	1.57	12.90	134.5	123.9	2.75	5.0	3	57.0	0.0	0	0.0	16.0		64.4	33.1	8	403	279.49	237.84	517.33	1.0004
NBH42	TSF 3	2012	Ca-Na-SO4-HCO3	25	7.4	4.5	688	427	64.80	6.00	61.10	2.30	12.60	119.9	109.6	2.30	0.0	0	0.0	0.1	0	0.0	0.0		100.0	0.0	16	379	244.80	134.50	379.30	1.0003
NBH43	TSF 3	2010	Ca-Na-SO4-HCO3	25.3	7.6	5.1	786	481	74.26	4.69	83.70	7.13	15.31	114.8	220.5	2.79	12.0		16.2	0.1	0	0.1	52.8		90.6	31.8	6	524	366.20	270.83	637.03	1.0005
NBH43	TSF 3	2011	Na-Ca-SO4-HCO3	25.1	7.6	4.5	824	510	66.94	4.10	94.68	3.35	18.95	101.4	174.6	3.05	6.0	4	47.3	0.1	0	0.0	27.3		41.0	29.6	14	468	304.71	277.28	581.99	1.0004
NBH43	TSF 3	2012	Na-Ca-SO4-HCO3 Ca-	25.2	7.6	4.8	805	496	70.60	4.40	89.20	5.20	17.10	108.1	197.5	2.90	0.0	0	0.0	0.1	0	0.1	0.0		100.0	0.0	10	496	326.40	169.60	496.00	1.0005
NBH44	TSF 3	2010	Mg-Na-HCO3	24.9	7.1	4.5	324	199	38.11	7.96	10.94	0.82	6.34	76.8	5.1	3.32	5.0		19.5	0.1	0	0.1	21.3		48.0	42.0	31	150	96.75	141.21	237.96	1.0001
NBH44	TSF 3	2011	Ca-Na-HCO3	25.2	7.0	4.5	298	189	37.98	4.44	11.04	0.72	5.48	80.9	10.2	14.09	5.0	3	37.0	0.0	1	0.1	25.8		56.8	37.4	18	166	115.88	158.15	274.03	1.0001
NBH44	TSF 3	2012	Ca-Mg-Na-HCO3 Ca-	25.1	7.1	4.5	311	194	38.00	6.20	11.00	0.80	5.90	78.8	7.7	8.70	0.0	0	0.0	0.1	1	0.1	0.0		100.0	0.0	25	158	101.30	56.70	158.00	1.0001
SBH01	South close to ADR pregnant pond	2009	Na-HCO3-Cl		6.8	3.9	398	225	19.57	3.71	11.29	4.92	15.84	82.3	6.7	5.05	6.0	23	5.0	0.1	0		26.0		13.7	14.0	-3	150	115.86	107.68	223.54	1.0001
SBH01	South close to ADR pregnant pond	2010	Ca-HCO3	24.3	6.9	3.7	368	246	24.96	2.61	7.73	5.59	7.25	132.7	7.2		7.0		30.0	0.5	0		10.0		23.0	98.5	-13	188	154.18	179.54	333.72	1.0001
SBH01	South close to ADR pregnant	2011	Ca-Na-HCO3 Ca-	24.5	7.2	3.3	346	214	42.47	3.14	15.60	5.29	13.05	128	8.0	10.50	7.0	5	10.3	1.4	0	4.2	13.5		18.0	68.9	9	232	167.64	168.50	336.14	1.0002
SBH01	pond South close to ADR	2012	HCO3	24.5	7.1	3.1	365	228	46.31	3.07	12.45	5.40	9.20	86.2	3.4	11.13	4.0		51.7	0.1	0	4.9	21.5		49.4	113.3	29	183	114.60	259.01	373.61	1.0002
SBH02	pregnant pond	2009	Na-HCO3-Cl		6.0	5.4	180	104	2.15	0.67	16.22	0.45	6.62	37.3	3.3	2.80	5.0	33		0.1	1		23.0		30.0	38.0	-1	70	55.05	114.03	169.08	1.0001
SBH02	South Heap Leach	2010	Na-HCO3		6.1	5.2	147	99	1.39	0.73	16.34	0.61	6.05	64.8	1.0	1.70	6.0	8	22.0	0.1	1				20.0	124.3	-18	94	79.55	174.24	253.79	1.0001
	South Heap Leach																															
SBH02	South Heap Leach	2011	Na-HCO3	24.1	6.4	4.9	143	89	2.91	1.02	19.82	0.49	5.85	35.8	1.7	2.42	5.0	9	14.3	0.3	1	0.1	16.0		31.0	103.0	16	71	50.81	167.05	217.86	1.0001
SBH02	South Heap Leach	2012	Na-HCO3-Cl	24.8	6.2	4.9	139	86	1.63	0.90	19.03	0.63	7.75	29.9	1.7	5.17	3.0		60.0	0.1	1	0.1	21.0		41.0	119.6	11	68	47.63	223.65	271.28	1.0001
SBH03	South Heap Leach	2009	Ca-Na-HCO3		5.9	5.0	257	145	13.33	2.50	8.12	2.58	9.23	76.7	14.4	3.50	10.0	49		0.2	1		36.0		11.0	34.5	-17	131	113.86	146.69	260.55	1.0001
SBH03	South Heap Leach close to SP5/Galla	2010	Ca-HCO3-Cl		6.0	4.6	184	125	6.99	1.88	6.89	1.62	11.05	95.2	13.5	1.20	10.0		25.0	2.2	1		24.0		16.0	159.3	-43	139	130.95	226.39	357.34	1.0001
SBH03	South Heap Leach close to SP5/Galla	2011	Na-Ca-HCO3-Cl	24.3	6.0	4.4	133	84	6.53	1.84	7.59	1.04	7.38	45	9.0	2.73	9.0	21	9.0	0.6	0	0.3	29.5			127.6	-16	82	73.45	204.82	278.27	1.0001
SBH03	South Heap Leach close to SP5/Galla	2012	Na-Ca-HCO3	24.5	5.9	3.6	129	80	5.12	1.84	7.31	1.57	6.80	50.3	10.1	5.70	6.0		73.5	0.0	0	0.2	20.0		80.2	139.2	-24	90	78.91	249.24	328.15	1.0001
SBH04	South Heap Leach close to	2009	Na-Ca-HCO3		6.2	4.7	139	93	4.99	0.93	6.41	1.56	4.05	63.8	7.0	2.20	5.0	5	24.0	0.1	0				9.0	99.3	-34	91	82.05	142.52	224.57	1.0001
SBH04	BR3	2010	Ca-Na-HCO3		6.5	4.7	250	138	15.68	2.14	14.81	1.17	5.17	75.1	8.4	2.18	6.0		13.3	0.2	0				90.0	102.3	2	126	97.20	279.36	376.56	1.0001
	South Heap Leach close to BR4			24.1																		0.1	129.5									
SBH04	South Heap Leach close to	2011	Na-Ca-HCO3	24.5	6.5	4.3	194	120	10.52	1.54	12.72	0.81	5.10	74	7.2	2.67	4.0	4	59.0	0.0	0	0.0	41.5		63.7	85.9	-11	115	93.25	216.25	309.50	1.0001
SBH05	BR5 South Heap Leach	2009	Ca-Na-HCO3		5.7	5.5	137	85	6.05	1.79	6.71	0.44	4.87	55.7	2.0	3.63	3.0	39	5.0	0.0	0		39.0		11.0	102.5	-21	81	69.20	200.53	269.73	1.0001
SBH05	South Heap Leach	2010	Na-HCO3-Cl		5.4	5.1	49	34	0.94	0.37	5.31	0.22	4.25	33.2	1.6	2.70	4.0	6	10.0	0.0	0				19.0	87.7	-40	49	45.75	110.55	156.30	1.0000
SBH05	South Heap Leach	2011	Na-HCO3	24.1	5.6	4.7	47	30	1.19	1.26	10.25	1.95	3.57	16.5	1.0	4.66	4.0	10	16.0	0.1	0	0.1	53.0		32.0	200.1	15	41	30.37	293.90	324.27	1.0000
SBH05	South Heap Leach	2012	Na-HCO3-SO4-Cl	24.6	5.6	4.1	46	28	0.58	0.43	5.72	0.28	3.33	13.7	5.0	4.33	3.0	2	90.0	0.0	0	0.0	25.0		63.6	133.3	-21	34	29.67	257.41	287.08	1.0000
SBH06	South Heap Leach	2009	Na-HCO3-Cl		4.9	5.3	29	17	0.71	0.15	2.17	0.19	3.48	12.3	1.0	2.20	3.0	31	9.0	0.1	0		51.0		23.7	31.0	-40	22	21.98	125.28	147.26	1.0000
SBH06	South Heap Leach	2010	Na-HCO3-NO3		5.0	5.2	23	16	0.27	0.11	1.72	0.19	1.55	22.5	1.4	5.00	5.0		45.0		0		15.0		19.5	80.7	-66	33	35.45	143.01	178.46	1.0000

APPENDIX 11

Explanation of groundwater Piper diagrams

Explanation of Piper Diagrams

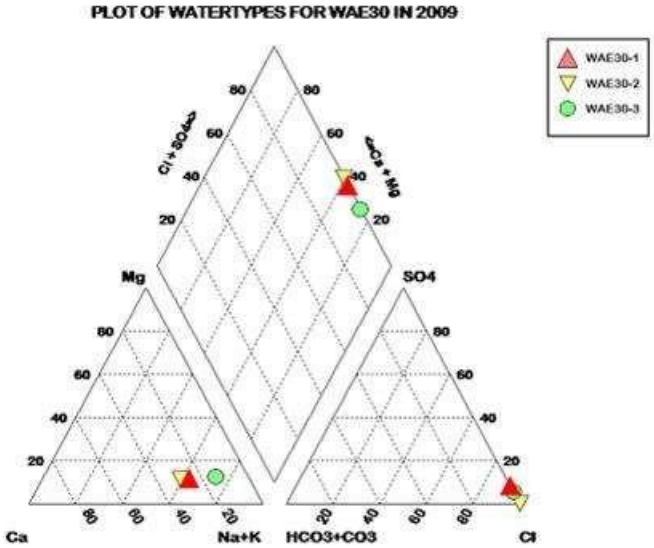
Piper diagrams show the relative concentrations of six to seven ions in solutions, in this case, the cations Ca, Mg, and Na+K, and the anions Cl, SO_4, and HCO_3. In most natural waters, these ions make up 95 to 100% of the ions in solution. The Piper diagram includes two trilinear diagrams, one for anions (on the lower right) and one for cations (on the lower left). For each sample, the information from each trilinear diagram is projected up into the central quadrilateral. Therefore, each sample will plot in each frame of the Piper, once representing cations, once representing anions, and once representing the combination. For each constituent, the concentration (in mg/l) is converted to chemical equivalents (meq/l) based on the valence and atomic weight. Then the percentages of each ion relative to the total are calculated, and plotted on the Piper diagram. Each trilinear diagram shows the relative percentages of the three ions. Each corner on the triangles represents 100% of the ion shown at that corner.

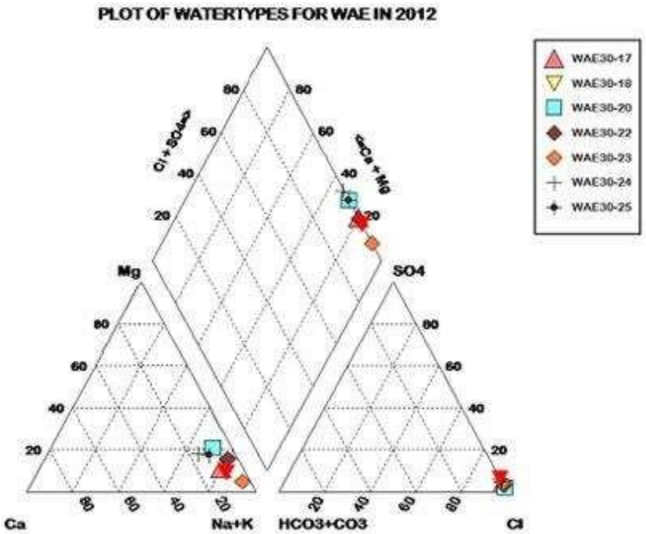
The axes of the trilinear and quadralinear diagrams are read in varying directions. Cations (in the lower left) are read with values increasing clockwise along each leg of the trilinear diagram from 0% to 100% with the axis restarting at 0% at each apex of the triangle. Ca is on the bottom axis ranging from 0% on the right end to 100% on the left end of the axis. Mg is on the left leg of the trilinear diagram ranging clockwise from 0% on the bottom of the leg to 100% on the top of the trilinear diagram. Similarly, Na plus K are added together and represented on the right side of the cation trilinear and increase in a clockwise direction. Anions in the lower right trilinear are read in a counter-clockwise direction along each axis in a similar fashion. In this case, Cl is represented along the bottom leg, SO_4 along the right leg, and HCO_3 along the left leg of the trilinear diagram.

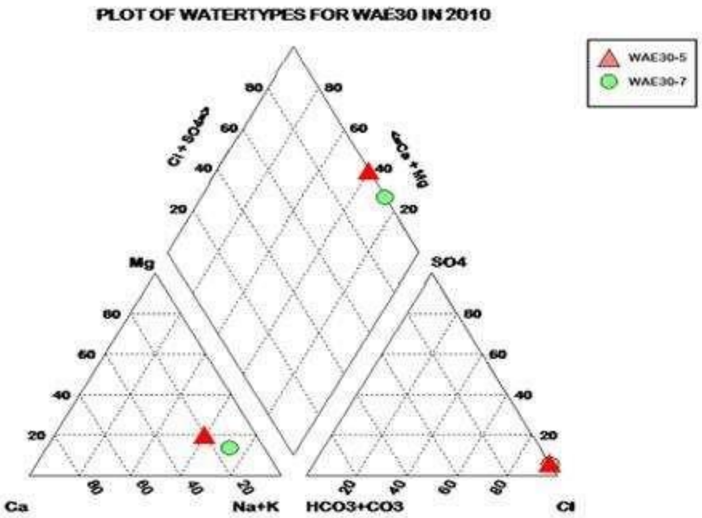
The location of the data point in each of the trilinear diagrams is then projected up into the quadrilateral, and plotted where the two projections intersect. The upper right side of the quadrilateral diagram is the sum of Ca plus Mg. The upper left side is the sum of Cl plus SO_4.

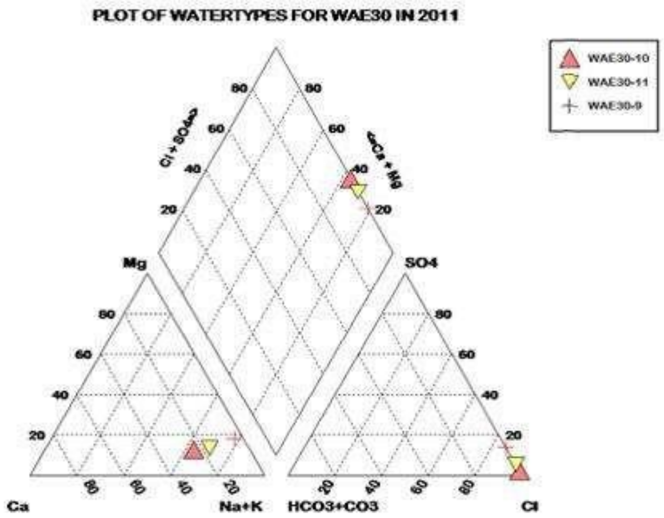
APPENDIX 12

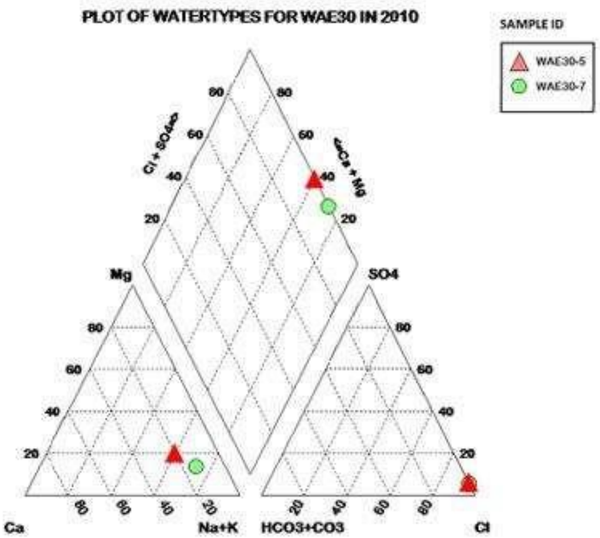
Piper diagrams for Community groundwater

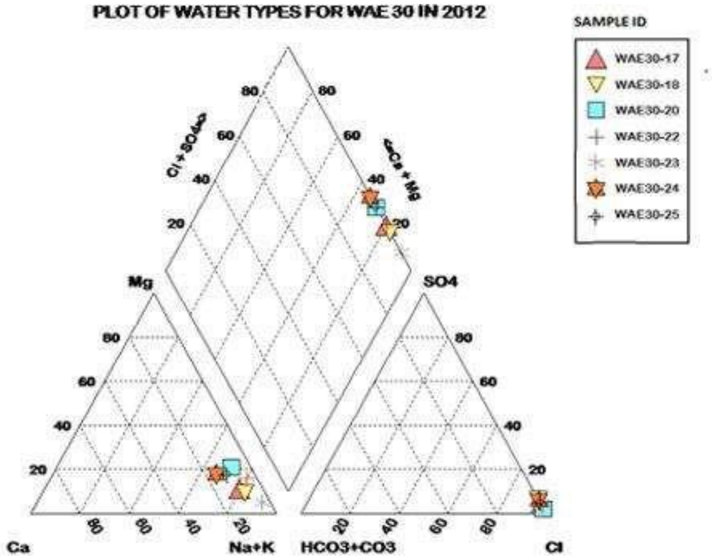

PLOT OF WATERTYPES FOR WAE30 IN 2009

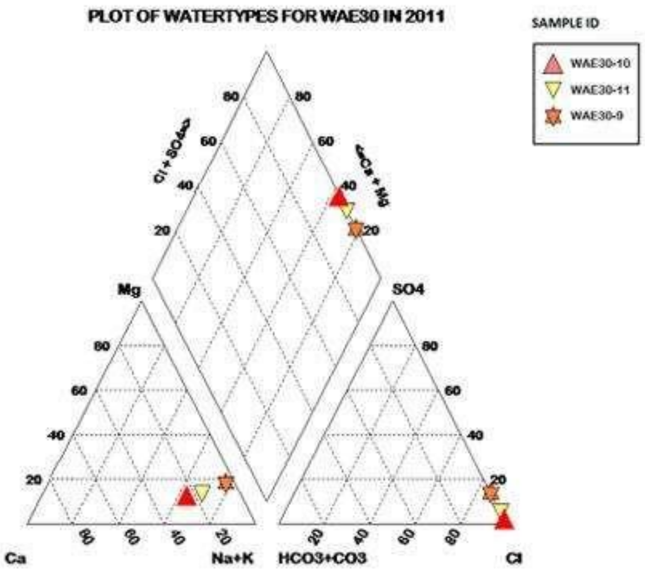

PLOT OF WATERTYPES FOR WAE IN 2012

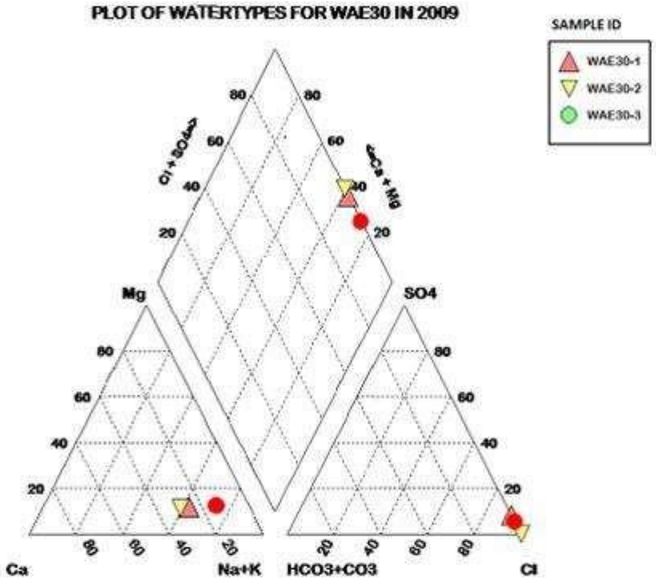

PLOT OF WATERTYPES FOR WAE30 IN 2010


PLOT OF WATERTYPES FOR WAE30 IN 2011


PLOT OF WATERTYPES FOR WAE30 IN 2010
SAMPLE ID
WAE30-5
WAE30-7
Cl + SO4
Ca + Mg
Mg
SO4
Ca
Na+K
HCO3+CO3
Cl
20
40
60
80


PLOT OF WATER TYPES FOR WAE 30 IN 2012
SAMPLE ID
WAE30-17
WAE30-18
WAE30-20
WAE30-22
WAE30-23
WAE30-24
WAE30-25
Cl + SO4
Ca + Mg
Mg
SO4
Ca
Na+K
HCO3+CO3
Cl
20
40
60
80


PLOT OF WATERTYPES FOR WAE30 IN 2011
SAMPLE ID
WAE30-10
WAE30-11
WAE30-9
Cl + SO4
Ca + Mg
Mg
SO4
Ca
Na+K
HCO3+CO3
Cl
20
40
60
80


PLOT OF WATERTYPES FOR WAE30 IN 2009
SAMPLE ID
WAE30-1
WAE30-2
WAE30-3
Cl + SO4
Ca + Mg
Mg
SO4
Ca
Na+K
HCO3+CO3
Cl
20
40
60
80