โลหะหนักและศักยภาพในการทำให้เกิดน้ำทิ้งสภาวะกรดจากของทิ้งจากเหมือง บริเวณพื้นที่เหมืองทองคำอัครา จังหวัดพิจิตร

นางสาว จุฬาลักษณ์ ชาญกูล

ศูนย์วิทยทรัพยากร
จุฬาลงกรณ์มหาวิทยาลัย

วิทยานิพนธ์นี้เป็นส่วนหนึ่งของการศึกษาตามหลักสูตรปริญญาวิทยาศาสตรดุษฎีบัณฑิต
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ลิขสิทธิ์ของจุฬาลงกรณ์มหาวิทยาลัย

HEAVY METALS AND ACID GENERATION POTENTIALS OF SOLID MINING WASTES FROM AKARA GOLD MINE, PHICHIT PROVINCE

Miss Chulalak Changul

A Dissertation Submitted in Partial Fulfillment of the Requirements
for the Degree of Doctor of Philosophy Program in Environmental Management
(Interdisciplinary Program)
Graduate School
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Thesis Title	HEAVY METALS AND ACID GENERATION POTENTIALS OF SOLID MINING WASTES FROM AKARA GOLD MINE, PHICHIT PROVINCE
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Field of Study	Environmental Management
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จุฬาลักษณ์ ชาญกูล : โลหะหนักและศักยภาพในการทำให้เกิดน้ำทิ้งสภาวะกรดจากของทิ้งจากเหมือง บริเวณพื้นที่เหมืองทองคำอัครา จังหวัดพิจิตร. (HEAVY METALS AND ACID GENERATION POTENTIALS OF SOLID MINING WASTES FROM AKARA GOLD MINE, PHICHIT PROVINCE) **อ.ที่ปรึกษาวิทยานิพนธ์หลัก** : **ผศ.ดร.จักรพันธ์ สุทธิรัตน์, อ.ที่ปรึกษาวิทยานิพนธ์ร่วม ศ.ดร. จี แพดมานาบาน, ดร.จันทรา ทองคำเภา** , 172 หน้า.

ลักษณะจำเพาะทางเคมีของของทิ้งจากเหมืองโดยเฉพาะอย่างยิ่งศักยภาพในการทำให้เกิดน้ำทิ้งสภาวะกรดและการละลายโลหะหนัก ของทิ้งจากเหมืองที่ใช้ในการศึกษาครั้งนี้ ประกอบด้วยหินทิ้งและกากตะกอนหางแร่ จากเหมืองทองคำขนาดใหญ่ที่สุดของประเทศ บริษัทอัคราไมนิ่งจำกัด จ.พิจิตร โดยหินทิ้งได้มาจากบ่อซีเอช และกากตะกอนหางแร่จะได้มาจากบ่อเก็บกักกากตะกอน โดยที่ตัวอย่างทั้งหมดถูกนำมาวิเคราะห์ถึงลักษณะจำเพาะทางเคมี การชะของโลหะหนักและศักยภาพในการทำให้เกิดน้ำเสียที่มีสภาวะเป็นกรด โดยข้อมูลทั้งหมดที่ได้รับจากการศึกษาทดลองสามารถใช้เป็นข้อมูลพื้นฐานในการจัดการสิ่งแวดล้อมได้ดี

หินทิ้งบางชนิดโดยเฉพาะ เช่น ซิลิซิไฟด์ ลาปิลี ทูฟ และ เชียร์ทูฟ มีศักยภาพในการทำให้เกิดน้ำทิ้งสภาวะกรดได้หากไม่ได้รับการจัดการที่ดี โดยที่หินชนิดอื่นๆและกากตะกอนหางแร่ ไม่แสดงถึงศักยภาพที่จะก่อให้เกิดน้ำทิ้งสภาวะเป็นกรด จากการศึกษาส่วนประกอบของตะกอนหางแร่พบว่าโลหะหนักส่วนใหญ่ เช่น โคบอลต์ ทองแดง แคดเมียม โครเมียม ตะกั่ว นิกเกิลและสังกะสี มีปริมาณที่ต่ำกว่าค่ามาตรฐานของโลหะหนักในดินเพื่อการเกษตรมีเพียงแมงกานีสที่มีค่าสูงกว่าค่ามาตรฐาน แต่ทั้งนี้เมื่อทำไปทดสอบความสามารถในการชะละลาย พบว่าแมงกานีสและตะกั่วถูกชะออกมาจนเกินค่ามาตรฐานในการอุปโภคบริโภคในทุกช่วงระดับพีเอช นอกจากนี้นิกเกิลสามารถชะออกมาจนเกินค่ามาตรฐานที่พีเอชต่ำกว่า 4 ดังนั้นทั้ง 3 ธาตุนี้จึงอยู่ในข่ายที่ควรเฝ้าติดตามระมัดระวังแม้กระทั่งในช่วงพีเอชที่เป็นกลาง ในขณะเดียวกัน การศึกษาถึงปริมาณโลหะหนักในหินทิ้งพบว่า อาร์เซนิค ปรอท แคดเมียม โคบอลต์ ทองแดง แมงกานีส นิกเกิลและตะกั่ว ถูกพบอยู่ในปริมาณที่มากกว่าค่ามาตรฐานของหินอัคนี เมื่อทดสอบถึงความสามารถในการชะของโลหะหนัก พบว่า ผลการชะละลายของแมงกานีส ตะกั่วและทองแดง สามารถชะออกมาได้จนเกินระดับค่ามาตรฐานของน้ำแม้ในพีเอชที่เป็นกลาง โดยผลจากการวิจัยนี้นำไปสู่แผนการดำเนินงานติดตามเฝ้าระวังปริมาณโลหะหนักในพื้นที่

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จุฬาลงกรณ์มหาวิทยาลัย

สาขาวิชา การจัดการสิ่งแวดล้อม ลายมือชื่อนิสิต ....
ปีการศึกษา 2551 ลายมือชื่ออ.ที่ปรึกษาวิทยานิพนธ์หลัก ....
ลายมือชื่ออ.ที่ปรึกษาวิทยานิพนธ์ร่วม ....

4789699220 : MAJOR ENVIRONMENTAL MANAGEMENT

KEYWORDS : GOLD MINE / MINE WASTE / ABA / NAG / HEAVY METAL

CHULALAK CHANGUL: POTENTIAL CONTAMINATION FROM MINING WASTE ON THE SURFACE WATER IN THE VICINITY OF THE AKARA GOLD MINE, PHICHIT PROVINCE. ADVISOR: ASST. PROF. CHAKKAPHAN SUTTHIRAT, Ph.D., CO-ADVISOR : PROF. G. Padmanabhan, Ph.D. , CHANTRA TONGCUMPOU, Ph.D., 172 pp.

Gold mining wastes were chemically characterized particularly in aspects of acid drainage formation and heavy metal leaching. Solid mining wastes contain waste rocks and tailings that were taken from the largest gold mine in Thailand, Akara gold mine in Phichit Province. Waste rock samples from C-H pit and tailing samples from tailing storage facility were collected to be representatives of waste material. All samples were tested for geochemical characteristics including leaching of heavy metal and acid generating potentials. All information gained from this study can provide such good idea for environmental management.

Two particular types of waste rocks (i.e., silicified lapilli tuff and sheared tuff) appear to have potential to generate acid mine drainage; therefore, good management plan must be taken with great care. Other rock types and tailing samples did not show signature to generate acidic drainage. From metal analyses of tailing, most toxic elements (e.g., Co, Cu, Cd, Cr, Pb, Ni and Zn) were found falling within the standard of Thailand Soil Quality Standards for Habitat and Agriculture. Only Mn appears to have higher content than the standard. However both Mn and Pb can leach out exceeding the surface water standard for agricultural at all pH condition, based on experiment leaching test. Ni can leach out exceeding the surface water standard at below pH 4. Thus, these 3 metals are needed to be monitor even at neutral pH condition. For waste rocks, most metals (e.g. As, Ag, Cd, Co, Cu, Mn, Ni and Pb) were found exceeding the average in igneous rocks. However, leaching experiment indicate the most metals have potential to leach out only at lower pH (pH2); besides, many metals show just little difference of leaching at pH 4 and 6.5. Mn, Pb and Cu appear to be the most concern in the leachate because these metals in all rock types yielded high concentration in the leachates at all pH conditions which are over the standards in most cases. These results lead to management and monitoring plan.

Field of Study : ENVIRONMENTAL MANAGEMENT Student's Signature Chulalak Changul.
Academic Year : 2008 Advisor's Signature
Co-Advisor's Signature

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CHAPTER I

INTRODUCTION

1.1 General Statement

Gold, one of the most favored precious metals, has long been a part of human history. Humans have utilized gold for ornamental purposes due to its beauty, rarity and durability (Kearney et al., 2008). Gold has a specific characteristic which is resistance to chemical reactions. Therefore, gold is an anti-corrosive and anti-rust material (DMR, 2008). For thousands of years, these outstanding characteristics have motivated humans to acquire gold. In addition, we also admire its beautiful yellow color that is the most popular for jewelry. Nowadays, gold appears to be a crucial indicator of national strength in both politic and economic. Hence, the gold reserve can actually imply the stability and wealth of country. Systematic gold explorations have occurred in Thailand for many decades. However, a few mines, including the Akara Gold Mine in Phichit province, are currently in operation.

Mineral processing and extraction processes usually produce enormous loads of crushed rock wastes and tailings. These materials may be a potential risk to the environment after some period of exposure to the natural weathering (Cidu et al., 1997). Acid mine drainage (AMD) is the most important environmental impact often concerned in the mining industry (Akabzaa et al., 2007). The process of acid mine drainage is usually accelerated when mining exposes metal sulphide in mineralized rock and mine wastes to oxygen and water, allowing rapid oxidation of the sulfides. Sulfide oxidation has the potential to produce sulfate, which may turn into sulfuric acid when it is dissolved by rain, leading to the production of acidity drainage. Subsequently, the acid drainage can cause metal leaching (ML) by dissolve heavy metals stored in waste rock and tailings such as lead, zinc, copper, arsenic, selenium, mercury and cadmium, into surface runoff and ground water (Nriagu, 1978; Smith and Skema, 2001; Piyush et al., 2007; Akabzaa et al., 2007; Bennett, 1969). Consequently, the high metal concentrations within acid mine drainage may in turn cause severe toxicological effects on aquatic

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ecosystems (Hazen et al., 2002). Metals remaining in solution may be directly toxic to biota or may enter the food chain with primary producers and subsequently be amplified to toxic levels (Bryan, 1976). Although some heavy metals are found in small quantities and some (likes copper and zinc) are required for a normal metabolism, when the quantities of these oligoelements increase, they become toxic and can cause damage and malfunctioning of human organs (Florea et al., 2005). Environmental protection should be carefully planned in order to eliminate and/or minimize any short and long-term environmental impacts that may occur. Otherwise, serious problems may occur that may be very difficult to remediate and extremely cost enormous.

An example of pollution from mining is in an abandoned gold mining belt, Beatrice, Sanyati Valley, Zimbabwe. The belt has been mined for gold since 1900, but was abandoned towards the year 2000. There are four mines along the belt. Each mine is located under 5 km from the Mupfure River, and so run-off from the mines and their dumps can easily reach the main river system. The dominant sulphides are arsenopyrite (FeAsS), pyrite (FeS_2), galena (PbS) and stibnite (Sb_2S_3). Gold is generally locked up in sulphides or occurs in close contact to the sulphides, so processing is associated with release of acid, arsenic and metals into streams that flow into the Mupfure River. Chemical results obtained show that the surface water is slightly contaminated with respect to Pb, Zn and Ni. Such contamination is attributed to the mine dumps (Ravengai et al., 2005).

In Thailand, there are many areas facing the environmental threat of mining activities. For example, cadmium contamination in the agricultural land of Mae Sot District of Tak Province in the northwestern region has caught the attention of many organizations. Lead contamination in stream sediment in Kitty creek of Thong Pha Phum District, located in Kanchanaburi Province, Western Thailand, has resulted from the failure of a tailing pond. Similarly, in the Southern Thailand, lead contamination in the Pattani River basin has been of critical concern for over the past ten years. Arsenic contamination, harming human health, has been disclosed in the vicinities of an abandon tin-mining area in Nakhon Si Thammarat Province. Acid mine drainage (AMD)

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in the Li District area of Lamphun Province, has been the main cause of manganese contamination in the reservoir water. Its manganese content exceeds the Thai standard for surface water.

Besides, the AMD can be distributed by the runoff passing mine, collapse of mining construction is also the reason of the contaminant distribution. For example, the case of collapsed waste mining pools at the Aznalcollar Mine, Spain in 1998, the tailing dam breakage about 5-6 million m3 of slurry composted of acid water, and finally divide metal sulphide and materials used in the refining/floating process were released directly into the agrio river, and consecutively into the Guadiamar river (a tributary of the Guadalquivir river) (Alvarez-Ayuso et al., 2008; Cabrera et al., 2008; Kraus and Wiegand 2006). Margui et al. (2004) reported that sludge spread along the river and subsequently the river overflowed affecting a nearby lowland area. Doñana National Park, which contains the most important biological reservoir in the world, was also affected by the flooding and consequent contamination of toxic spills from the Aznalcollar Mine. The soils were found to contain as high as 4000 mg kg^{-1} of Pb and 9000-15000 mg kg^{-1} of Zn (Conesia et al., 2007a).

Many cases mentioned above have been caused significantly by accidents and human errors; other unexpected problems have somehow occurred, despite the use of environmental impact assessments and routine monitoring are taken into consideration. Among these threats, AMD is the most often impact in many countries because of mismanaged operation. Therefore, impact of mining wastes should be taken into consideration with great care during the whole time of mine planning and operation. Enormous waste rock will be piled over the life of mine. As a result, some waste rock and tailing have a potential to acid generation.

A study on the natural geochemical background and ore mineralization has been used by economic geologists to explore and classify mineral deposits. It is also a key to understand the environmental signatures. Hence, mineralogical and chemical characteristics are one among many scientific tools that will lead to identification and classification of potential sources of such problems. Subsequently, appropriate prevention and mining plans can be designed based on these data. The best practice usually involves the integration of between acid generating waste and non-acid generating waste managements (Plumlee and Logsdon, 1999).

Gold exploration in Akara Gold Mine was commenced in July, 1995. It has been in operation for over ten years up to now; consequently, tons of wastes have been continuously produced and dumped within the area. Therefore, chemical testing, particularly detections of potential acid forming and heavy metals yields significant data for mining and prevention plans. In addition, experimentation on the leaching potentials of significant toxic elements provides useful information for developing appropriate management. A numerical model for predicting runoff is also studied in order to simulate contaminated surface water by particular metals. Finally, all information obtained are summarized and made recommendation applying to environmental protection plan and management strategies. That will be consulted to the mine to minimize threats for their operation.

1.2 Objectives

The main aim of this research project is to characterize all types of solid mining waste prior to propose the best practice of mining and waste management. Therefore, characterization of solid wastes (e.g., soil, rock and tailings) has to be carried out based on geochemical tests. Consequently, the following sub-objectives have been set:

Sub-objectives:

- To identify and quantify the heavy metals in mining wastes (e.g., soil, rock and tailing) of the Akara Gold mine, as they may be major sources of water resource contamination in the future.
- To estimate the potential of acid generation from the mining wastes

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- To predict area in which may become affected using a surface water numerical model capable to simulate the distribution of surface water and transportation of heavy metals.
- To develop and recommend a waste management plan for the mine

1.3 Hypotheses

1. Solid wastes produced from gold mining activities are of environmental concern due to their potential for causing surface and/or groundwater pollution.

2. The acidification of mining wastes at the Akara Gold Mine affects heavy metal mobility and cause environmental problem.

1.4 Scope of Study

1. The study area, Akara Gold Mine, is located along the border of Phichit and Phetchabun Provinces.

2. This study is focused on all types of solid wastes generated from mining activities.

3. The environmental concern of this work is focused on surface water quality.

1.5 Orientation of Thesis Report

This section provides the guidelines for the dissertation style. The dissertation will be divided into seven chapters which will describe for the whole work of this study. Summaries of each chapter are below.

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Chapter I: Introduction, background and motivation of the study are given and followed by the objective and scope of the study. That are already reported herein this chapter.

Chapter II: Literature reviews of gold mine operation and theoretical background, particularly on assessment of acid generation and heavy metal contamination from mining activities, as well as numerical model for surface run-off are summarized in the next chapter.

Chapter III: All methodologies used in this study and detail of the study area are revealed in Chapter III. Experimental methods, criteria and assumption used for numerical model are also described in detail.

Subsequently, results of the study are present in the format of manuscripts within three chapters including Chapters IV to VI. In addition, some results obtained from the mathematical model are given in Chapter VII. Conclusion and recommendations are also provided in the last chapter.

Chapter IV: Chemical characteristics and acid drainage assessment of mine tailings from the mine are described. This contains major chemical composition and heavy metals as well as their mineralogical characteristic. Moreover, assessment of potential to generate acid is also included. Experimental results of leaching tests for metal at different pH conditions are given at the end of the chapter for evaluation of possibility to the environmental impact.

Chapter V: Geochemical characteristics of waste rocks are reported in this chapter. It includes general information and characterization of all waste rock types. Major chemical compositions and heavy metal contents in waste rocks are initially revealed before leachable metals from waste rock at different pH conditions are also assessed from the experiment.

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Chapter VI: Acidic potential assessments of waste rocks are given in this chapter. It provides results of assessments of potential to generate acidic drainage from waste rocks. Different assessment procedures were used to confirm each other before waste rock management are planed and proposed.

Chapter VII: provides the overall conclusions and recommendation based on this study. The possible affected area is assumed using the available model before suggestion of the management plans are proposed within this chapter.

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CHAPTER II

THEORETICAL BACKGROUND

2.1 Gold and Mining Process

2.1.1 Gold

Gold has been one of the most popular precious metals since the prehistoric period. It is a chemical element assigned the symbol as “Au” and has atomic number of 79. It is a highly sought-after precious metal which, for many centuries, has been used as money, a store of value and in jewelry. Gold is dense, soft, shiny and the most malleable and ductile. Pure gold has an attractive bright yellow color. Nowadays, most of the world significant currencies are closely quoted to the gold value. In addition, gold demand has increased every year due to price increased and the expansion of the jewelry industry. There are records of gold use by humans dating backwards as far as 4500 years ago. There is a report stating that the Tutankhamen Cemetery contained royal property wholly made or glazed from gold. This evidence shows that Egyptians had a way to produce gold plates. Moreover, Egyptians possessed the arts of gold plating and alloying with other metals to achieve variations in the gold’s hardness and color (Dunn, 2006).

In the present days, gold is the best protection against both political and economic upheavals. Gold is still one of the most liquidfied and widely accepted forms of exchange because its value is stable and continually increasing. Gold forms the basis of the monetary standard used by the International Monetary Fund (IMF) and the Bank for International Settlements (BIS). Gold also offers protection against a weak currency or high domestic inflation levels, which are prevalent and persistent problems in the developing world. In addition, gold is inversely correlated to the US dollar, making it a good currency hedge. As an asset class, gold has all the advantages of being universally regarded as a form of currency, without what being subject to the economic and monetary policies of one particular country's government.

Gold mining involves the science, technology, and business of the discovery of gold. Gold may be found in many places, most common rocks but in very small quantities. More often it is found in greater quantities in veins associated with igneous rocks, rocks created by heat such as quartzite (The Shark Group, 2003).

In Thailand, gold is mostly imported with high value for all parts of the industries. For an example, based on data from the Department of Foreign Trade, Thailand imported a total amount of 8.11 tons of gold in August 2006, and its value was 152 US $ or 1.32% of all imported products. As a result, gold exploration has been taken place for many decades but a few potential areas have been found.

2.1.2 Gold Deposit in Thailand

Department of Mineral Resources reported the most significant gold occurrences in Thailand in 5 locations: (I) Doi Tung (Ban Pa EE), Chiang Rai Province, (II) Lampang , Phrae and Sukhothai Province, (III) Loei, Nong Khai and Udon Thani Province, (IV) Phichit, Phetchabun and Lop Buri Province and (V) East area of Thailand including (a) Prachin Buri and Sa Kaeo Province and (b) Chachoengsao, Chon Buri and Rayong Province (DMR, 2005). Among these occurrences, only the akara mine, however, has been operating using heavy machinery. The data from the Department of Mineral resource can be illustrated for the significant gold occurrence locations in Thailand as shown in Figure 2.1.

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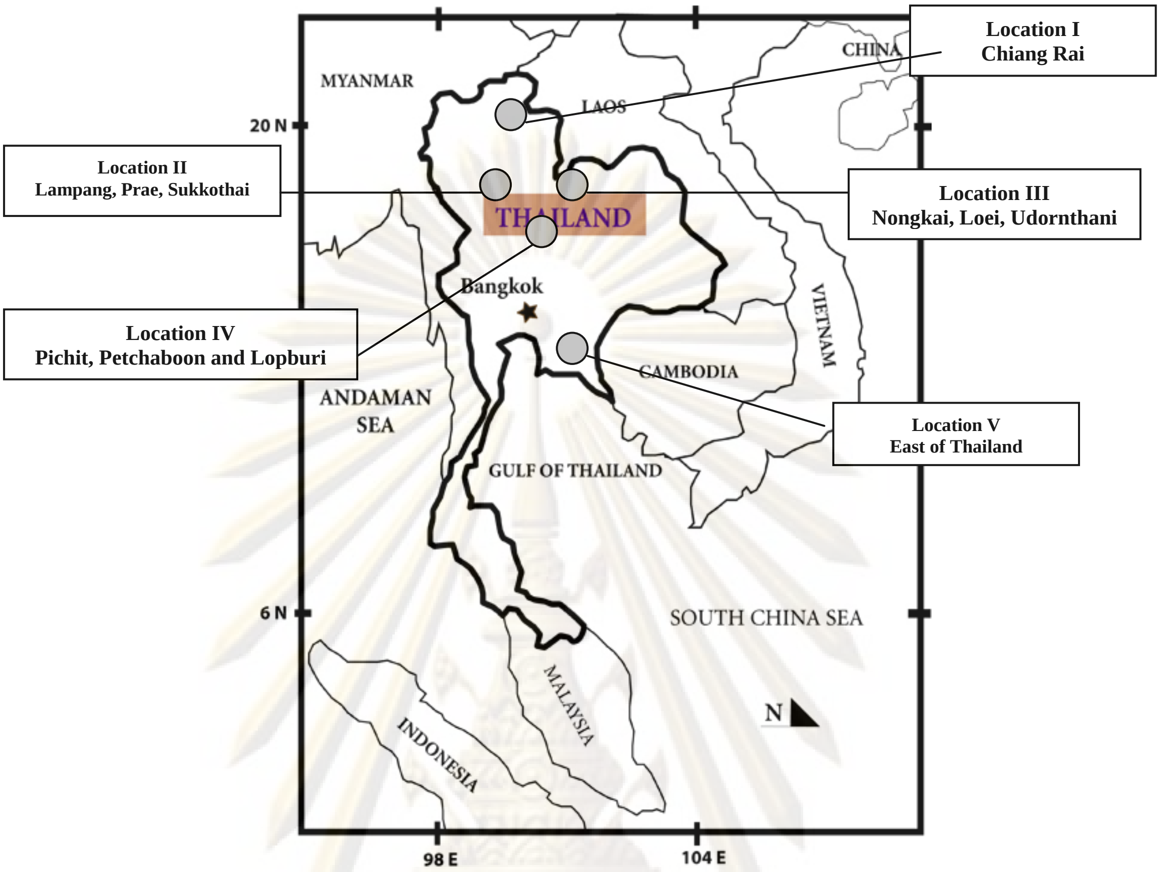

Figure 2.1 The significant gold occurrences in Thailand (modified after Department of Mineral Resources (DMR), 2005).

Before the beginning of mining operations, the companies have to explore the potential areas. Soil and rock samples are collected for chemical analyses because visible gold rarely occurs in the rock. Gold naturally originates deep within the earth in places called pockets, the cavities in the earth, filled with ore, or a rich deposit of gold. These pockets are filled with gold, heavy ore and quartz. If enough gold is discovered in the ore zone, the technological process of hard rock mining will begin (The Shark Group, 2003).

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2.1.3 Gold Mining Process

In general, a gold mine has 3 principle steps of operation including mining, mineral dressing and metallurgical extraction (or gold concentration).

Mining Process: also involves the exploitation of a mineral resource before the commercial resource is selected for further steps. It can be defined as the act of tunneling and digging out of the ground in order to recover one or more component part of the mined materials, which cover the ore body, from the ground in order to access one or more of the components of the mine minerals (Dunster et al., 1996). There are three types of mining, i.e., surface mine, underground mine and *in situ* (solution) mine. Regarding to surface mining, it can be divided into three subgroups including open pit, open cast and alluvial mining. However, this study is only focused on the open pit mining which is the operating type in the study site.

During the open pit mining, overburden and waste rock are firstly removed from the entire area of the final planned pit, often in very large quantities. Some crushing of overburden or waste rock may be required for efficient handing and/or use for construction within the site. Data for the U.S. in 1989 showed that surface mines produce eight times as much waste per ton of ore as underground mines. Overburden and waste rock are often used during the operation and closure of a mine (e.g., inert waste rock for building, solid of reclamation at closure). However, if they are contaminated with significant concentrations of potentially harmful minerals, they must be disposed as hazardous waste (Warhurst and Noronha, 1999).

Mineral Dressing: (ore processing) involves the size reduction and separation of the individual minerals in the ore-bearing hard rock (e.g., Au, Cu, Pb and Zn) and also industrial mineral deposits (e.g., phosphate and bauxite). The mineral processing is defined as the physical processing of minerals. It does not result in any chemical changes to the mineral component of the ore, but is a means of achieving the physical separation (and concentration) of different mineral phases (e.g., target mineral from gangue minerals or of one valuable mineral from another). Mineral processing method can be divided into two groups, size reduction and separation of mineral phases.

Size reduction is undertaken using crushers and grinding mill. Crushing and grinding are used to liberate economic and non-economic minerals from one another and thereby produce suitable feeds for subsequent processes in which separated mineral phases can be generated.

Separation of mineral can be achieved by employing differences in mineral characteristics base on particle size, particle density, magnetic properties, electrical properties and surface chemistry characteristics (floatation).

Metallurgical extraction aspires to destruct the crystallographic bonds in the ore mineral in order to obtain an element or compound. Hydrometallurgical methods of ore treatment are the most typical method applied for extractive metallurgy. Extractive metallurgy is based on Hydrometallurgy (e.g., Au, U, Cu, Zn, Ni and P) and Pyrometallurgy (e.g., Cu, Zn, Ni, Pb, Sn and Fe) and to a lesser degree on electrometallurgy (e.g., Al and Zn) (Lottermoser, 2003; Ripley et al., 1996). Ore is first crushed and processed according to the requirements of the subsequent process. A leaching agent is then used to extract to the valuable metal in the form of dilute metal-laden solution. In general, gold mine metallurgical extraction always uses cyanidation method. Cyanide (as a sodium or potassium cyanide solution) is used to dissolve gold. After passing through the gravity concentration process, gold is extracted from the ball mill cyclone overflow to the Carbon in Leach (CIL) unit by mixing sodium cyanide and alkaline solution to produce gold cyanide. Sodium cyanide solution is metered into the CIL distributor via a ring main system. The alkaline solution is added to ensure that free cyanide ions, which are essential for the reaction, are not lost as free cyanide gas. Quicklime (CaO) is added directly to the mill feed conveyer to achieve a pH suitable for cyanidation. The optimum pH is 10.3. The other two necessary components are oxygen and water. The electrochemical reaction of gold dissolution is described by Elsener's equation:

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

The general unit operations in gold mine are depicted in Figure 2.2. This unit is carried out in the stripping and goldroom area; acid washing of carbon, stripping of gold from loaded carbon, electrowinning of gold from pregnant solution and carbon regeneration. Then, wastewater from the processing plant is discharged along with waste solids (tailings) to the Tailing Storage Facility (TSF). Therefore, before the wastewater runs through the TSF, the wastewater will undergo a cyanide destruction process in order to reduce the cyanide concentration to an acceptable level prior to discharge to the TSF (Akara Mine 1999). The waste, generated from the extraction of gold and silver, is known as tailings. On the other hand, the final products are concentrates of multi-elements containing gold and other associated minerals, such as silver and copper.

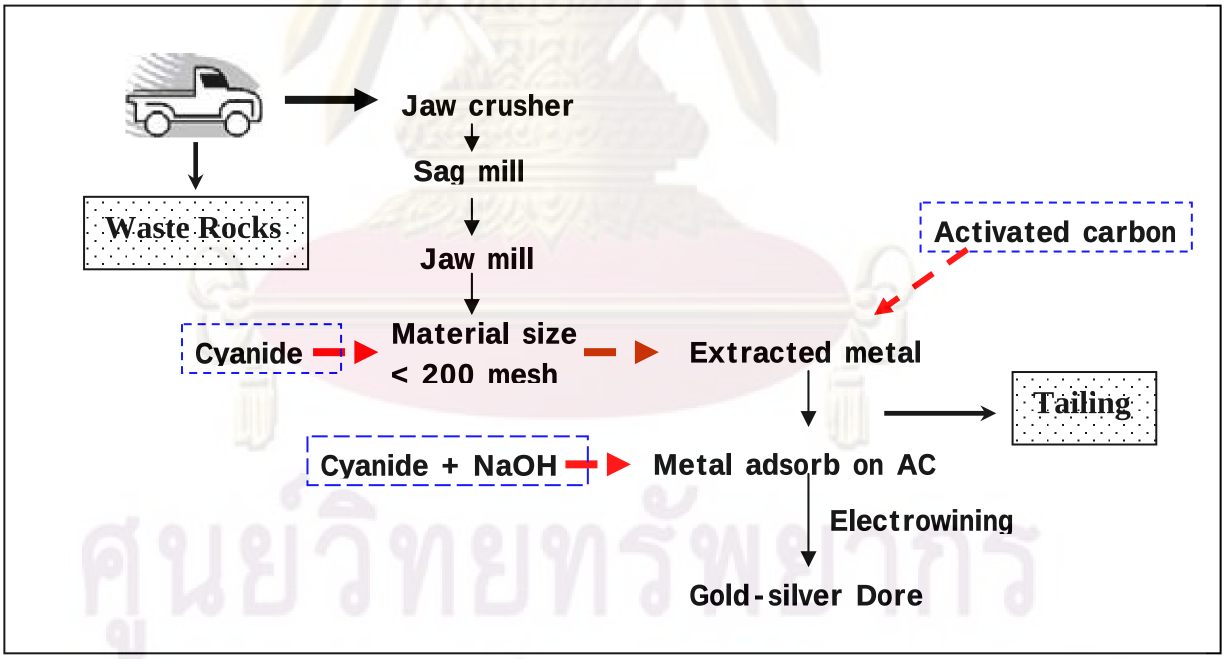

Figure 2.2 Mineral processing flowchart (modified after Akara Mine, 1999).

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2.1.4 Mining Wastes

All three principal activities of the mining activity produce waste. Mine wastes can be defined, based on their states, as solid, liquid and gaseous by-products generated from mining, mineral processing and metallurgical extraction. Regarding to solid wastes, they are unwanted, have no current economic value and accumulate at mine sites. Besides, liquid and gaseous wastes may leach to the environment and in turn impact to the surrounding area. Mine wastes are categorized and described in more detail as presented in Table 2.1.

Table 2.1 Operational mining steps and their produced solid, gaseous and liquid wastes (Lottermoser, 2003).

Activity	Types of wastes	Examples of wastes
Open pit mining, underground mining	Mine waste	Waste rocks, overburden, spoils, mining water, and atmospheric emissions
Mineral processing, coal washing, mineral fuel processing	Processing waste	Tailing, sludge, mill water, and atmospheric emissions
Pyrometallurgy, hydrometallurgy, electrometallurgy	Metallurgical waste	Slags, roasted ores, flue dusts, ashes, leached ore, process water, and atmospheric emissions

Mining wastes are heterogeneous geological materials that may consist of soil, sediment and metamorphic or igneous rock. As a result, the particle sizes of waste rock range from clay-size particles to boulder-size fragments. The primary sources of mine wastes are rock, soil and sediment from surface mining operations, especially in the open pit mining. On the other hand, tailings, sludge and waste water may be generated from mineral processing. Physical and chemical characteristics of mining wastes vary according to their mineralogy and geochemistry, type of mining equipment, particle size of the mined material and moisture content (Lottermoser, 2003).

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Waste Rock: It has become a more significant threat when open pit mining has replaced underground mining, particularly in developing countries; consequently, the volumes of waste would be increased. Waste rock is normally coarse (10 to over 20 cm in diameter) maybe with variable sulfide concentration. Sulfide minerals which commonly form as minor constituents in the earth's crust may contain in the large amount of waste rock dumps (Lottermoser, 2003). The mineral pyrite (FeS_2) tends to be the most common sulfide mineral present. When mining exposes sulfidic materials to an oxidizing environment, the materials become chemically unstable. A series of complex chemical weathering reactions are spontaneously initiated. This occurs because the mineral assemblages contained in the waste are not in equilibrium with the oxidizing environment. Although the oxidation of pyrite increases the solution's acidity, much of buffering of generated acidity is achieved through buffering reactions. Acid buffering is largely caused by the weathering of silicates, carbonates and hydroxides. Silicates which make up the majority of the minerals in the Earth's crush are the major buffering capacity in the environment. Carbonate minerals also play an important role in acid buffering reaction. Minerals such as calcite ($CaCO_3$), dolomite ($CaMg(CO_3)_2$) or magnesite ($MgCO_3$) will neutralize acid generated from sulfide oxidation.

Tailing: is solid residues produced during primary operations in the separation and concentration of minerals. It will be typical, finely-ground mine waste, with 100% finer than 150 microns. Tailings are a sand-like material, generally with no rocks mixed in. In the gold mine, their colors vary between light brown and dark grey. After gold is recovered using a cyanide leach method, the tailings are passed through a cyanide destruction circuit to reduce the total cyanide concentration. Finally, they are transported in slurry form to the pond via pipeline (Ramrez-Cordero and Canizares-Macas, 2009; Hutchison, 1992; Lottermoser, 2003).

Tailings often have high sulfide content (mainly in form of rejected pyrite, marcasite and pyrhotite) and are much finer than waste rock. Although tailings have much higher surface area than waste rock, the uniform and fine particle size lead to much lower permeability than that appear in waste rock pile. Therefore, the increased surface area available for oxidation and leaching reactions may be balanced by reduced contact with oxygen due to a slower moving waterfront and slower replenishment by oxygen rich water. Consequently, tailings often generate Acid Mine Drainage (AMD) more slowly than those generated by the coarse waste rocks (Mitchell, 1999).

2.2 Acid Mine Drainage and Example of Environmental Impact Site

2.2.1 Acid Mine Drainage

Acid Mine Drainage can occur as a result of the atmospheric oxidation of sulphide minerals exposed by mining activities to produce sulphuric acid and other oxidation products. Hence, it seems to be a significant environmental problem of mining operation, especially of opencast mines. Factor which increase metal leaching include rapid-weathering of metal-containing minerals, drainage conditions to increase solubility and high flow rates through contaminated materials (Price and Errington, 1998). In addition, runoff can also give to serious problems because it can permeate overburdened piles, dissolve some toxic metal from the heaps, and contaminate groundwater, eventually. AMD causes damage long after the mine operation has ended.

AMD is generated by the oxidation of sulfides in mine waste; the by-product is water with highly acidic, sulfide-rich drainage, leading to the mobilization of metals. In the mining process, there are 2 important types of wastes generated. The first one is tailings, final waste from the mining process. Tailing are the ore rock, which pass through the milling process and become extracted by cyanide. After being processed, tailing are composed of small particles, such as sand and silt. The other type is waste rock which is a non-economic material removed from a mine in order to access the ore body. Particle size of waste rock is generally large depending on rock type and blasting process.

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The acid production in a rock is tied directly to the amount of sulfides bound up in the rock in various form. Sulfides are crystalline substances that contain sulfur conbined with metal or semi-metals but no oxygen. The most general from of sulfide is “pyrite” (FeS_2). Other forms include Fe1-xSx, Fe3S4, FeS, CuFeS4, ZnS, PbS, HgS and CoAsS.

After these sulfide minerals in mines are exposed to the air and water, toxic metal ions and hydrogen ions are released into the environment. The sulfide ions are then oxidized into soluble sulfate ions.

The primary requirements for acid generation are:

1. Sulfide minerals in the overburden
2. Water or a humid atmosphere
3. An oxidant (usually oxygen in form of O_2)

The process of acid generation lead to the formation of acid mine drainage (AMD). It is related to the oxidation of pyrite in waste or overburden from mine operation and precipitation of Fe hydroxides. There are four common chemical reactions that represent the AMD formed by pyrite:

$$2FeS_2 \quad + \quad 15O_2 \quad + \quad 14H_2O \quad \rightarrow \quad 2Fe(OH)_3^- \quad + \quad 4H_2SO_4 \qquad [1]$$

pyrite + oxygen + water ⟶ “yellow boy” + sulfuric acid

In the first equation, the important oxidant of pyrite is oxygen. Ferrous iron is released and sulfur is oxidized and changed to sulfide. This equation shows 2 moles of acidity generated for each mole of pyrite.

$$2FeS_2 \quad + \quad 7O_2 \quad + \quad 2H_2O \quad \rightarrow \quad 2Fe^{2+} \quad + \quad 2SO_4^{2-} \quad + 4H^+ \qquad [2]$$

pyrite + oxygen + water ⟶ ferrous iron + sulfate + acidity

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The second equation is the conversion of ferrous iron to ferric iron. It consumes one mole of acidity.

$$4Fe^{2+} + O_2 + 4H^+ \rightarrow 4Fe^{3+} + 2H_2O \qquad [3]$$

ferrous iron + oxygen + acidity → ferric iron + water

The third equation is a hydrolysis reaction, which splits the water molecule. These moles of acidity are the generated byproducts.

$$4Fe^{3+} + 12H_2O \rightarrow 4Fe(OH)_3^- + 12H^+ \qquad [4]$$

ferric iron + water → ferric hydroxide (yellow boy) + acidity

The fourth reaction is the oxidation of additional pyrites formed by ferric iron. The ferric iron generated in reaction steps 1 and 2 are the cycle and propagation of the overall reaction. They take place very rapidly and continue until either the ferric iron and pyrite is depleted. In this reaction, iron is the oxidizing agent, not oxygen.

$$FeS_2 + 14Fe^{3+} + 8H_2O \rightarrow 15Fe^{2+} + 2SO_4^{2-} + 16H^+ \qquad [5]$$

pyrite + ferric iron + water → ferrous iron + sulfate + acidity

While acidity is generating, the acid neutralization is also occurring if neutralizing minerals are available. The dissolution of carbonates such as calcite can neutralize the acidity generated by the oxidation of pyrite (Eq. (6)) and although slower to react, alumimo-silicates such as chlorite (Eq.(6)) can also contribute to the acid neutralization capacity (ANC) of the sample (Paktunc, 1999). The equations (6) and (7) are shown as following;

$$CaCO_3 + 2H^+ \rightarrow Ca^{2+} + H_2O + CO_2 \qquad [6]$$

$$Mg_5Al_2Si_3O_{10}(OH)_8 + 8H_2SO_4 \rightarrow 5Mg^{2+} + 2Al^{3+} + 3SiO_2 + 8SO_4^{2-} + 12H_2O \qquad [7]$$

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The subsequent problem generated from Acid Mine Drainage (AMD) likes Metal Leaching (ML) occurs when metals/heavy metals are leached from their source and flow with the runoff from rain or flooding. Acid Mine Drainage (AMD) and Metal Leaching (ML) can cause harmful effects on the environmental health of downstream ecosystems. The discharge of acidic water and high metal concentrations into natural reservoirs can cause the depletion of aquatic life. Hence, in a long term, they accumulate in the environmental metrics in the forms of sediment, soil, surface and groundwater. The metal contaminants can cause health effects on living things directly by touch or ingestion; and the heavy metals can also cause future problems by accumulating in the environment and affecting the food chain.

2.2.2 Example of Environmental Impact Site

There are several mining-related environmental problems. Examples of contaminated mining sites are given below;

Van dam (2008) studied the environmental impact of metals released from the decommissioned Mount Todd gold mine, located in the wet–dry tropics of northern Australia. The Todd gold mine consists of a large waste water inventory and an acid rock drainage problem, which has the potential to impact upon freshwater ecosystems of the Edith River catchment. Some tropical freshwater species i.e., duckweed, green alga, cladoceran, snail, green hydra, fish, etc., are determined to assess the toxicity of retention pond water from Mount Todd mine. The mining activities have resulted that the pond water was very toxic to all species, with the percentage dilution of water from the pond inhibiting 10% of organism response (IC10), or lethal to 5% of individuals (LC5), ranging from 0.007 to 0.088%. And the percentage dilution of the water inhibiting 50% of organism response (IC50), or lethal to 50% of individuals (LC50), ranging from 0.051% to 0.58%.

The Furtei gold mine in Sardinia, Italy lies in a sensitive environment, because of the close proximity to an inhabited area, and specifically to important water reserves and to agricultural activities (Pelo et al., 2009). The highest potential threat for the environment is mostly represented by mineralized rocks exposed in waste dumps and open pits that contain significant amounts of sulfides (mainly pyrite (FeS_2) and enargite (Cu_3AsS_4)). These materials have a high potential for acid drainage generation and release of toxic elements (notably Cu and As, but also Al, Ni, Co and Cd). It is reported that the drainage from this mine exhibited its acidity as low as pH 2, while for metals release, up to 5, 180 and 788 mg/L of As, Cu, and Al, respectively. However, leaching solutions and waters interacting with mineral assemblages of the propylitic alteration zone mainly composed of chlorite, quartz, and calcite, with relic magmatic plagioclase, neutralized and hence increased the pH, and lower metal loads. Leachates from cyanidation tailings show variable pH (between 6.2 and 9.7, depending on sulfide content in tailings); cyanide concentration varies between 110 μg/L and about 3 mg/L, whereas contents of toxic elements in leachates are, with the exception of Hg, within the limits of Italian regulations for nondangerous industrial wastes.

Chen et al. (2007) investigated the well water quality which was carried out in the rural area subject to irrigation with acidic mine water from the Guangdong Dabaoshan Mine, Southern China. The result of AMD affected on the groundwater. The pH of water was 2.9 ± 0.3 on average from monitoring result, which indicated that the acidity was more than 600 times higher than the permit limit set in the Chinese National Standard for Irrigation water quality. As a result, concentration of various metals of potential toxicity (e.g. Pb, Cu, Cd and Zn) was also extremely high.

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2.3 Potential Acid Formation

The potential for acidic mine generation is the primary water quality concern and this potential is largely determined by mine waste composition. Iron sulfide minerals are often present in mine waste and their oxidation leads to the production of acid.

Theoretically, sulfide bearing rock can produce acidic drainage when exposed to air and water. But, not all of exposed sulfide bearing rocks result in AMD. In some cases, acid drainage fails to occur when the sulfide minerals are non reactive, such as when the rock contains sufficient alkaline material to neutralize any of the acid generated. Although the acid formation is generated, it comes to contact with sufficient alkaline material and hence, pH will raise as a result of neutralizing reaction. Neutralized AMD may still contain metal concentrations but in lower amounts.

Regarding to the waste rocks, there are several methods proposed to predict AMD under various circumstances either static or kinetic conditions. However, the most common static test used worldwide is Acid-Base Accounting (ABA) as recommended by Kwong (2000). In addition, production and neutralization of AMD may occur due to complexities of physical, chemical and biological processes. Therefore only static method alone may not be able to provide completely accurate results. A combination of kinetic and static methods is usually applied to increase reliability of results. Therefore, in this study, kinetic humidity cell tests have been used to assess the AMD potential and confirm the ABA tests in several cases. However, this assessment is usually very time-consuming and expensive. Net Acid Generation (NAG) test has been proposed as a comparable test to kinetic humidity cell tests by Akabzaa et al. (2007). Advantages of NAG test, including lower cost, and being a rapid and simple technique have made it becoming a highly recommended method to evaluate the acid generating potential of rocks sample along with ABA method (Schafer 2000; Greenhill 2000).

2.3.1 Acid-Base Accounting Test (ABA Testing)

The acid-base accounting includes static laboratory procedures that assess the balance between acid generation processes (oxidation of sulfide minerals) and acid neutralizing processes (dissolution of alkaline carbonates). It is the most commonly-used static procedure for estimating/quantifying the potential of a mine waste's capacity (Furguson and Erickson, 1998). This procedure was developed at West Virginia

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University in the late 1960s. ABA tests are designed to measure the balance between the acid-generating potential (oxidation of sulfide materials) and acid-neutralizing potential in a sample. The values obtained from the acid-base account are referred to the maximum potential acidity (MPA) and the acid neutralizing capacity (ANC), respectively. The difference between the MPA and ANC value is referred to the net acid producing potential (NAPP) which is expressed in kg H_2SO_4/t of sample.

$$\text{NAPP} = \text{MPA} - \text{ANC}$$

After the MPA and NAC have been determined for a sample, these two values are necessary to combined to allow the comparison with the set of criteria. If NAPP is negative then it is likely that the material has sufficient inherent buffer capacity to prevent acid generation. Conversely, if the NAPP is positive then the material may be acid generating. The detail of the ANC and the MPA are defined as following;

Maximum Potential Acidity (MPA)

The MPA is the maximum amount of acid that could be generated by the sulfur contained within a sample. The calculation of NAP derived by assumes that all the sulfur measured in the sample occurs as pyrite (FeS_2) and the pyrite reacts under oxidizing conditions to generate acid according to the following reaction:

$$FeS_2 + 15/4\ O_2 + 7/2H_2O \Rightarrow Fe(OH)_3 + 2\ H_2SO_4$$

According to the reaction, the MPA of a sample containing 1 %S as pyrite would be 30.6 kilograms of H_2SO_4 per tone of material (*i.e.* kg H_2SO_4/t).

Acid-Neutralization Capacity (ANC)

Acid-Neutralization Capacity is typically evaluated by the titration type procedures based on the methodology of Sobek et al. (1978). The acid formed from pyrite oxidation will react to some extent with other acid neutralizing minerals contains within a sample. This existing acid buffering is computed in terms of the ANC. The ANC was determined by first giving each sample a fizz test rating in order to determine the amount and concentration of acid required to be used in the test. Each sample was then reacted with a known amount and standardized amount of acid (commonly HCl or H_2SO_4). The mixture was then heated and back titrated with NaOH to determine how much acid had been consumed by the sample. The ANC is expressed in the units of KgH_2SO_4/t of sample.

A major disadvantage of these ABA tests is that the assumptions intrinsic in the tests do not correspond to the real world situations. The ABA test assumes that:

(1) All sulfur in a sample will react to form acid.

(2) All material in a sample which consumes acid in the laboratory testing will generate alkalinity in the field.

(3) Pyrite oxidation rate is less than or equal to the rate of neutralization.

(4) Neutralization potential (NP) and %S below certain thresholds do not influence water quality.

Because these assumptions do not always hold the truth, researchers have questioned the ability for ABA to accurately predict post-mining drainage quality (diPretoro and Rauch 1988; Erickson and Hedin 1988). Nevertheless, on a field scale, researchers have found static tests to be quite useful. For example, Skousen et al. (2002) showed that the ABA is a good tool to assess overburden quality before mining and to predict post-mining drainage quality, and was correct in 50 out of 56 cases (i.e., 90%).

2.3.2 Net Acid Generation Test (NAG Test)

The NAG test was developed more than 20 years ago to evaluate the acid producing potential of a sample. The NAG test is usually used in association with the

NAPP to classify the acid generating potential of a sample. The NAG test directly measures the sample's ability to produce acid though sulfide oxidation. The NAG test uses H_2O_2 to complete the oxidation of the sulfide mineral(s) present. Hydrogen peroxide is added to the sample during the NAG test, so that both acid generation and acid neutralization reactions can occur simultaneously. The measurement of the solution's pH is taken after the reaction with hydrogen peroxide is complete. The acidity of the solution of the NAG test after the complete reaction is a direct measurement of the net acid generation of the sample.

The ABA and NAG test have been studies in several cases followed.

Paktune (2003) designed the mineralogical acid-base accounting program to calculate the neutralization and acid-generation potential of mine tailings and waste rock based on their mineralogical compositions. This method was used instead of ABA tests to screen mine tailings and waste rock to predict AMD.

Tran et al. (2003) studied the geochemistry of samples from mine sites in South Carolina, USA and Sudbury Ontario, Canada to determine the key implications for dump construction with regard to the control of acid mine drainage. The study consisted of investigations of the physical and hydrological aspects along the dumping sites as well as their geochemical and mineralogical constituents. The results revealed that from the study, each dumping sites can be evaluated the percentage of potential acid generated.

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Shu et al. (2001) studied the acid-forming potential of lead/zinc (Pb/Zn) mine tailings from a mine in Lechang City, Guangdong Province, China. Their experiment utilized the ABA and NAG to compare and confirm the results. They found that acidification occurred on the surface (0±20 cm depth) and had little effect at the deep layers of tailings. The amounts of heavy metals i.e, Pb, Zn, Cu, and Cd ,of acidified tailings on the surface (at 0±20 cm depth) were significantly higher than those of non-acidified tailings, which further revealed that acidification enhanced the mobility of heavy metals in the tailings.

2.4 Criteria for Classification of Mining Waste

NAPP Criteria

The method of MPA and ANC comparison is commonly used to set assessment criteria. There are various criteria for evaluation of AMD production potential, based on ABA test.

Weber et al. (2004) indicate the results of NAPP test that a negative NAPP indicates that the sample has a net neutralizing capacity and a positive NAPP indicates the sample has a net acid generation capacity.

While the criteria adapted from Hutchison and Ellison (1992) are shown in Table 2.2.

Table 2.2 Criteria for classifying the acid potential of mine waste[a].

Category	NAPP (kg H_2SO_4/ton)
Potentially acid generating	> 20
Non-acid generating	< -20
Zone of Uncertainty	Between -20 and +20

[a] Hutchison and Ellison (1992)

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NAG Test Criteria

The criteria of NAG test was suggested by EGI (2005) that if NAG pH higher than 4.5, it would be considered as non-acid forming whereas range of less than or equal to 4.5, it appears to be acid forming.

Classification of Rock Waste

Then the classification of mine waste is needed. Many previous papers have shown interest in the classification of mine waste and prediction of water quality in relation to AMD.

To protect a site from ARD, the proper management of rock waste is required. Best Practices in Environmental Mining Management in Australia (Environment Australia, 1997) suggested the classification of waste rock into 3 types:

1. *No net acid forming potential (NAF)*: A sample classified as NAF may, or may not, have a significant sulfur content but the content and reactivity of carbonate minerals and ANC within the material is more than adequate to neutralize all the acid that theoretically could be produced by any contained sulfide minerals. Material classified as NAF is considered unlikely to be a source of acidic drainage.

2. *Potentially net acid forming (PAF):* PAF material always has a significant sulfur content, the acid generating potential of which exceeds the inherent acid neutralizing capacity of the material. This means there is a risk that such a material, even if pH circum-neutral when freshly mined and processed, could oxidize and generate acidic drainage if exposed to atmospheric conditions.

3. *Uncertain net acid forming potential (UC)*: the uncertain classification is used when there is an apparent conflict between the results from the Net Acid Production Potential Test (NAPP) and Net Acid Generation Test (NAG).

2.5 Surface Water Model

2.5.1 Surface Water Model

To protect the problem that might be occur to the vicinity area. It needs to address the affected area where pollutants will distribute to. Modeling is frequently used to help build understanding of the transportation and distribution of water quality and water quantity problem. It is useful to quickly estimate the extent and severity of problem.

Watershed models simulate the hydrologic process by convert precipitation to streamflow. Precipitation is provided as input in watershed system. During the hydrological process that the runoff is being computed, water quality is changed. Some models consider only water quantities, whereas others simulate both water quality and water quality (Wurbs, 1995).

According to the condition of the area in this study that almost cover flat plain, the study is also focus on the transportation of heavy metal affected from AMD. The selected model should be able to simulate these 2 conditions; a) transportation of water over flat plain area, and b) the transportation of heavy metals along with water and/or sediment. Kitipan (2007) summarized the advantage and disadvantage of the models as shown in Tables 2.3 and 2.4.

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Table 2.3 Model advantage (Kitipan, 2007)

Model	ANSWERS	BASIN	CREAMS	HSPF	KINEROS	SWAT	CASC2D	TREX	AnnAGNPS
● Availability	X	X	X	X	X	X	X	X	X
● Distributed parameter/ semi-distributed model	X				X		X		X
● GIS integrated		X				X	X		X
● Flows and storage are represented in a simplified or conceptual manner				X			X		X
● Chemical transport and fate processes								X	
● Simulate event hydrology	X		X	X	X		X	X	X
● Two-dimensional overland routing approach							X	X	X

Table 2.4 Model disadvantage (Kitipan, 2007)

Model	ANSWERS	BASIN	CREAMS	HSPF	KINEROS	SWAT	CASC2D	TREX	AnnAGNPS
● Considerable amounts of spatial data are required	X			X	X			X	
● Scripted object-oriented interface		X						X	
● Lumped parameter			X	X					

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Two-dimension Runoff Erosion and Export (TREX)

TREX was selected to use in this study because TREX is a numerical modeling framework to simulate fate and transport of chemical across watersheds. It focuses on the event transport of metal in surface water. TREX code is designed to simulate multiple watershed outlets and also to allow channel network branching in the upstream and downstream directions. It does not consider groundwater flow process other than water loss at the surface by infiltration or channel transmission loss. TREX has consisted of 3 components that are important to this study namely; hydrology process, sediment transport and chemical transport (heavy metals) (Valluex, 2006).

Velluex (2005) developed a physical-based numerical modeling framework to simulate chemical transport and fate at the watershed scale. TREX (Two-dimension Runoff Erosion and Export) watershed model is a new modeling framework developed from CASC2D watershed model with chemical transport and fate process from the USEPA WASP and IPX series of stream water quality modeling, which is added to expanded CASC2D to create a new, fully distributed model to simulated contaminant transport and fate at the watershed scale.

A generalized conceptual framework for TREX is presented in Figure 2.3. At present, framework development focuses on the event transport of metals in surface water. Consequently, several possible processes in the general conceptual framework can be neglected because storm event are short-lived, lasting no more than a few hours. In particular, mass transfer and reaction process such as volatilization, biodegradation, hydrolysis and photo degradation can be neglected because the short time scale for simulations or because these process do not occur for metal. Other process such as dispersion and diffusion can also be neglected because at the time scale of event simulations transport processes are reasonable expected to be dominated by advection. At the event time scale, subsurface transport is also neglected. As a results, the transport and fate processes most important for the event simulation of metals are

- Advective water column transport
- Chemical partitioning between water (truly dissolved), dissolved organic compounds (DOC) (or other binding agent)(bound), and soil (particulate) phase;
- Transport (erosion, depositions, net burial) of solid and particulate chemicals;
- Infiltration of dissolved and bound (mobile) phase chemicals;
- External source and sinks of water, solids and chemicals.

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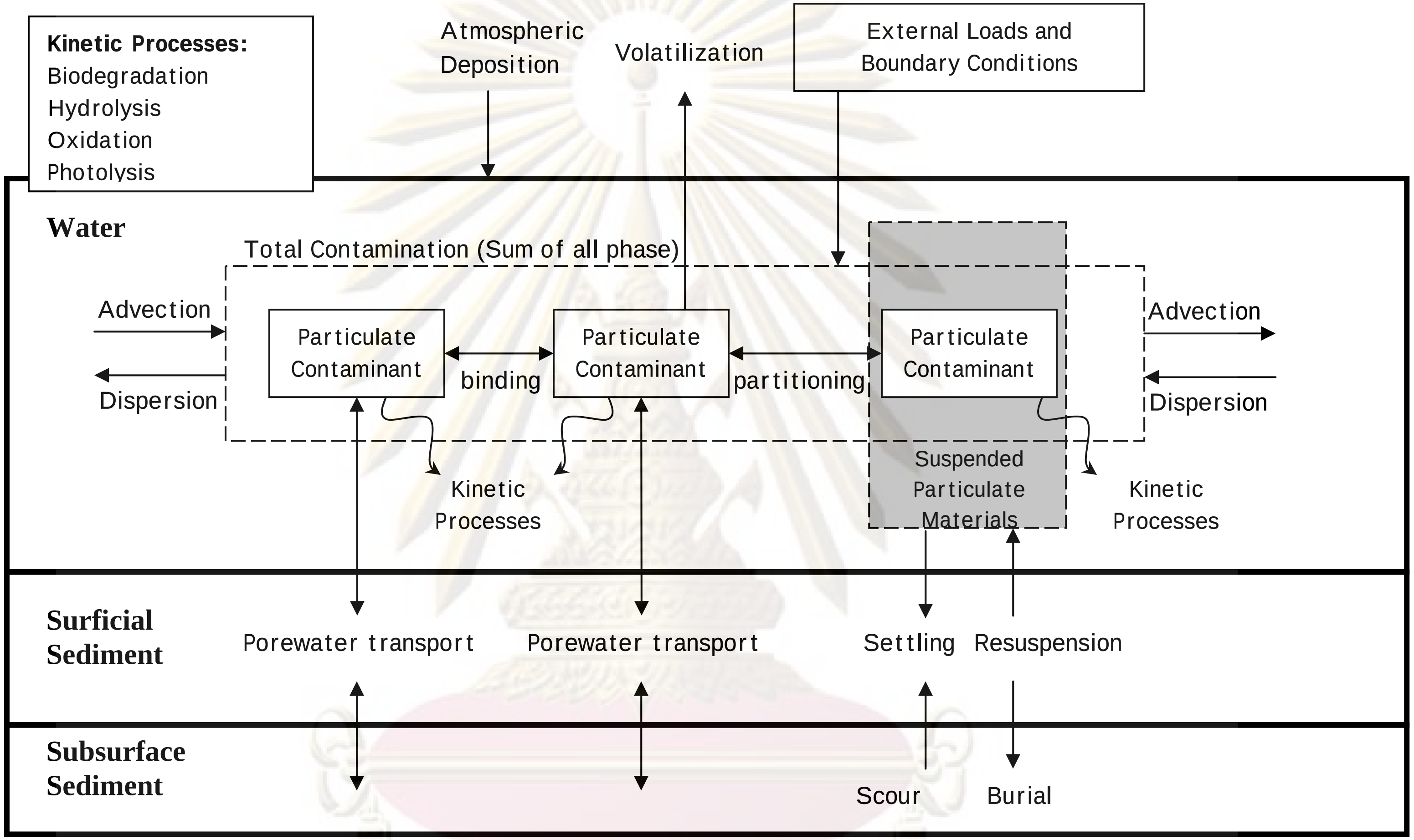

Partitioning between dissolved, particulate and DOC phases occurs in the sediments as conceptualized in the water column. Biodegradation, hydrolysis and oxidation can also occur in the sediment as conceptualized in the water column

Figure 2.3 Generalized Conceptual Model Framework (Valluex, 2005).

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TREX is generalizing watershed rainfall-runoff, sediment transport and contaminant transport modeling framework. It focuses on the event transport of model in surface water. This framework can simulates chemical partitioning and phase distribution, advection, erosion, deposition and dissolved phase infiltration in surface water, soil and sediment and chemical between upland area and stream water.

TREX consists of 3 main components which are hydrologic, sediment and chemical transport and fate process functional units. The organization of TREX component is present in Figure 2.4

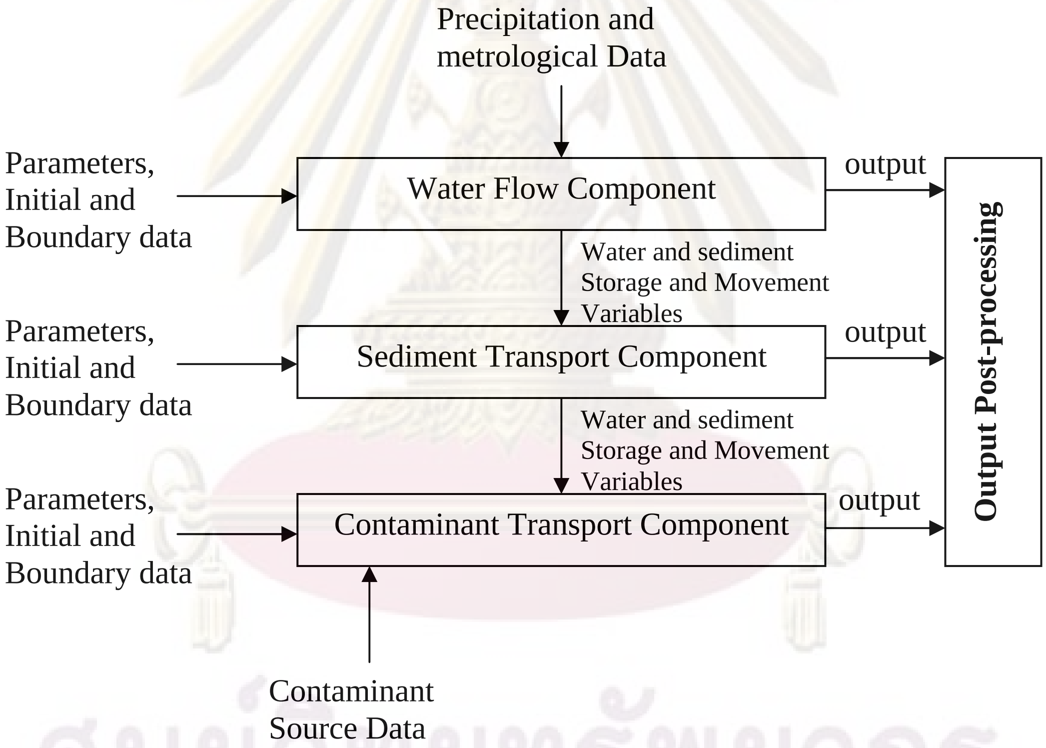

Figure 2.4 TREX Hierarchy and information flow (Valluex, 2005).

The description of each component will be summarized from TREX User's manual (Valluex et al., 2006b) as below;

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1. **Hydrology process**; can be viewed as submodel within itself as;

1.A) Rainfall and interception

Precipitation is the beginning of hydrologic cycle but not all rainfall can reach the land surface. Some may retention by plant/vegetable cover and some may return to the atmosphere by evaporation. For this reason, not all the precipitation can be used to account for the amount of water reaching the land surface, it needs to subtract by the fraction of the retention and evaporation. The net precipitation volume can be expressed as net (effective) precipitation

$$i_n = \frac{1 \partial V_n}{A_s \partial t}$$

Where

A_s = Surface area over which precipitation occur [L^3]

I_n = Net (effective) rainfall rate at the surface [L/T]

V_n = Net precipitation volume reaching the surface [L^3]

By $V_n = V_g - V_i$

V_i = interception volume [L^3]

V_g = Gross precipitation water volume [L^3]

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a) Infiltration and Transmission Loss

The transport of water from surface to subsurface called Infiltration. which affected from several factor including hydraulic conductivity, capillary action and gravity. The general equation can be express by:

$$f \quad = \quad K_h\left(1+\frac{H_c(1-S_c)\theta_c}{F}\right)$$

Where

f = infiltration rate [L/T]
K_h = effective hydraulic conductivity [L/T]
H_c = capillary pressure (suction) head at the wetting front [L]
θ_c = effective soil porosity [dimensionless]
φ = total soil porosity [dimensionless]
θ_r = residual soil moisture content [dimensionless]
S_r = effective soil saturation [dimensionless]
F = cumulative (total) infiltrated water depth [L]

Moreover, there is one more loss that similar to infiltration in overland areas, water in stream channels may be lost by subsurface by transmission loss. The transmission loss rate may be expressed as

$$t_1 \quad = \quad K_h\left(1+\frac{(H_w+H_c)(1-S_c)\theta_c}{T}\right)$$

where

t_1 = transmission loss rate [L/T]
K_h = effective hydraulic conductivity [L/T]
H_w = hydrostatic pressure head(depth of water in channel) [L]
H_c = capillary pressure (suction) head at the wetting front [L]
θ_e = effective sediment porosity [dimensionless]
φ = total sediment porosity [dimensionless]

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θ_r = residual sediment moisture content [dimensionless]

S_e = effective sediment saturation [dimensionless]

T = cumulative (total) depth of water transported by transmission loss [L]

1.B) Storage

Water may be stored in depressions on the land surface as small, discontinuous surface pool. Precipitation retained in such small surface depressions is depression storage. The depression storage depth represents a threshold limiting the occurrence of overland flow. When water depth is below the depression storage threshold, overland flow is zero.

1.C) Overland and Channel Flow

Overland flow can occur when water depth on overland plane exceeds the depression storage threshold. It governed by conservation of mass and conservation of momentum. The depth-discharge relationships on the overland flow are

$$q_x = \alpha_x h^{\beta}$$

$$q_y = \alpha_y h^{\beta}$$

$$\alpha_x = \frac{S_{fx}^{1/2}}{n}$$

$$\alpha_y = \frac{S_{fy}^{1/2}}{n}$$

where

α_x , α_y = unit discharge in the x- or y-direction

β = resistance coefficient for flow in the x- or y-direction

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n = manning roughness coefficient [T/L$^{1/3}$]

While Channel flowing watershed is defined as one dimension continuity equation from gradually-varied flow along a channel. The manning relationship may be used to describe flow resistance

$$Q = \frac{1}{n} A_c R_h^{2/3} S_f^{1/2}$$

R_h = hydraulic radius of flow = A_c/P_c [L]

P_c = wetted perimeter of channel flow [L]

2. Sediment transport

The main processes in the sediment submodel are:

<u>2.A) Advection-Diffusion</u>

For the overland planed in two-dimensions, the concentration of particles is governed by conservation of mass.

$$\frac{\partial C_s}{dt} + \frac{\partial \hat{q}_{tx}}{dx} + \frac{\partial \hat{q}_{ty}}{dy} = \hat{j}_e - \hat{j}_d + \hat{W}_s = \hat{J}_n$$

C_s = Concentration of sediment particles in the flow [M/L3]

$\hat{q}_{tx}, \hat{q}_{ty}$ = Total sediment transport area flux in the x- or y-direction

$\hat{j}_e$ = sediment erosion volumetric flux [M/L^{3}T]

$\hat{j}_d$ = sediment deposition volumetric flux [M/L^{3}T]

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$\hat{W}_s$ = sediment point source/sink volumetric flux [M/L^{3}T]

$\hat{J}_n$ = net sediment transport volumetric [M/L^{3}T]

2.B) Erosion

Erosion is the entrainment (gain) of material from a bottom boundary into a flow by the action of water. The erosion flux may be expressed as a mass rate of particle removal from the boundary over time and the concentration of particle at the boundary. In channels, sediment particles can be entrained into the flow when the exerted shear stress exceeds the stress required to initiate particle promotion. The relationship is considered a reasonable estimator of the total load:

$$C_w = 0.05\left(\frac{G}{G-1}\right)\frac{V_a S_f}{\left[(G-1)gd_p\right]^{0.5}}\left[\frac{R_h S_f}{(G-1)d_p}\right]^{0.5}$$

$$J_c = \frac{V_a C_t}{A_c}$$

C_w = Concentration o entrained sediment particle by weight at the transport capacity [dimensionless]

G = particle specific gravity [dimensionless]

V_a = advective (flow) velocity (in the down-gradient direction) [L/T]

S_f = friction slope [dimensionless]

R_h = hydraulic radius of flow [L]

g = gravitation acceleration [L/T^2]

d_p = particle diameter [L]

A_c = cross sectional area of flow [L^2]

C_t = concentration of entrained sediment particles at the transport capacity

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2.C) Deposition

Deposition is the loss of material entrained in a flow to a bottom boundary by gravity (Sedimentation). It is influenced by many factors including particle density, diameter and shape and fluid turbulence. The deposition flux can be expressed as an equation below

$$J_d = v_{se} C_s$$

J_d = deposition [M/L^2/T]
v_{se} = effective settling (deposition) velocity [L/T]
C_s = concentration of sediment particles in the flow [M/L^3]

3. Chemical Transport

While water and sediment move across the overland plane or along the channel network, it may transport the chemicals throughout a watershed. Chemical exist in the channel environment and on land surface in 3 phases:

(1) Dissolved in water.
(2) Bound with dissolved organic compounds (DOC) or other binding ligands or complexation agents and
(3) Particle-associated.

The phase in which the chemical are present can affect the pathway that chemical transport and interaction to the environment.

The main process in the chemical transport and fate submodel are :

(1) Chemical partitioning and phase distribution.
(2) advection-diffusion.
(3) erosion.
(4) deposition.

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(5) infiltration.

And (6) mass transfer and transformation process

(chemical reactions).

3.A) Chemical Partitioning and Phase Distribution

In general, chemicals have difference characteristic. Some are hydrophilic and some are hydrophobic. While chemical movements along with water, many chemicals are hydrophobic and readily partition between dissolved, bound, and particle-associated (particulate) phase. Partitioning to bound and particulate phases is a function of chemical affinity for surfaces and ion exchange (ionic chemical) or organic.

Chemicals may partition to all particle types (sorbents) present in a solution. The equilibrium partition (distribution) coefficient to any particle is defined as

$$\pi_{p\,n} = K_{p\,n} = f_{oc} K_{oc}$$

$\pi_{p\,n}$ = equilibrium partition (distribution) coefficient for particle "n" [L3/M]

$K_{p\,n}$ = equilibrium partition (distribution) coefficient for particle "n" [L3/M]

$f_{oc\,n}$ = fraction organic carbon of particle "n" [dimensionless]

K_{oc} = organic carbon normalized partition coefficient [L3/M]

The equilibrium partition coefficient can be used to describe the fraction of the total chemical that is associated with each phase as follows:

$$f_d = \frac{1}{1 + D_{oc}\pi_b + \sum_{n=1}^{N} m_n \pi_{px\,n}}$$

$$f_b = \frac{D_{oc}\pi_b}{1 + D_{oc}\pi_b + \sum_{n=1}^{N} m_n \pi_{px\,n}}$$

$$f_{p\,n} = \frac{m_n \pi_{px\,n}}{1 + D_{oc}\pi_b + \sum_{n=1}^{N} m_n \pi_{px\,n}}$$

$$f_d + f_b + \sum_{n=1}^{N} f_{p\,n} = 1$$

Where f_d = fraction of the total chemical in the dissolved phase [dimensionless]

f_b = fraction of the total chemical in the DOC-bound phase [dimensionless]

n = particle index = 1,2,3, etc.

$f_{p\,n}$ = fraction of the total chemical in the particulate phase associate with particle "n" [dimensionless]

3.B) Chemical Advection

Advection transports all chemical phases. For the two-dimensional flow in the overland plane, a chemical continuity equation analogous the sediment continuity equation can be written as:

$$J_{xc} = v_x \left(f_d + f_b + \sum_{n=1}^{N} f_{p\,n} \right) C_c = v_x C_c$$

$$J_{yc} = v_y \left(f_d + f_b + \sum_{n=1}^{N} f_{p\,n} \right) C_c = v_y C_c$$

Where J_{xc}, J_{yc} = chemical advective flux in the x- or y- direction [M/L^{2}T]

v_x, v_y = advective velocity in the x- or y- direction [L/T]

n = particle index = 1,2,3, etc

$v_{r\,n}$ = resuspension (erosion) velocity of particle "n"

[L/T]

f_d = fraction of the total chemical in dissolved phase in the water column [dimensionless]

f_b = fraction of the total chemical in the bound phase in the water column [dimensionless]

$f_{p\,n}$ = fraction of the total chemical in particulate phase associated with particle “n” in the sediment column [dimensionless]

C_c = total chemical concentration in the water column [M/L^3]

3.C) Erosion and Deposition of Particulate Phase Chemical

The factors that control particle transport between the water column and sediment bed can be expressed as equation in 2.b. Since particle phase chemicals move with the particles transported. Te erosion and deposition fluxes of chemicals are describes as

$$J_{ec} = \sum_{n=1}^{N} v_{rn} f_{p2\,n_n} C_{c2}$$

$$J_{dc} = \sum_{n=1}^{N} v_{se\,n} f_{p2\,n_n} C_{c1}$$

Where J_{ec} = chemical erosion flux [M/L^{2}T]

J_{dc} = chemical deposition flux [M/L^{2}T]

n = particle index = 1,2,3, etc

$v_{r\,n}$ = resuspension(erosion) velocity of particle “n” [L/T]

$v_{se\,n}$ = effective settling velocity of particle “n” [L/T]

$f_{p1\,n}$ = fraction of the total chemical in particulate phase associated with particle “n” in the water column [dimensionless]

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f_{p2n} = fraction of the total chemical in particulate phase associated with particle "n" in the sediment column [dimensionless]

C_{c1} = total chemical concentration in the water column [M/L^3]

C_{c2} = total chemical concentration in the soil/sediment column [M/L^3]

Input parameter

TREX main input files are divided into six grouping of related parameters. Inputs are typically specified in metric units (m, m/s, g/m^3, etc.)

Group A	To specify general controls for the simulation such as the simulation type and the series of times steps to be used for numerical infraction
Group B	To specify parameters for hydrologic simulations
Group C	To specify parameters for sediment transport simulations
Group D	To specify parameters for chemical transport simulations
Group E	To specify parameters for environmental conditions such as air temperature and wind speed
Group F	To specify parameters for model output control

TREX is fully-distributed and is designed to be compatible with data from raster GIS sources. In particular, data describing elevation, soil types, and use, and contaminant distributions can be in processed in a GIS and used as model inputs. A summary of required input for each cell are shown in Table 2.5

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Table 2.5 Input summary

1) watershed elevation	2) soil type
3) land uses	4) grain size distribution of particle
5) effective hydraulic conductivity (K_h)	6) porosity
7) Soil erodibility factor	8) surface roughness (manning , n)
9) rainfall interception depth	10) land cover factor (c)
11) land management factor (P)	12) distribution coefficient of metals (log Kd)
13) land manage practice factor (P)	

Output summary

Model outputs are also designed to be compatible for use with GIS. As a result, transport and fate process most important for the event simulation of metal are:

- Advective water column transport
- Chemical partitioning between water (truly dissolved), dissolved organic compounds (DOC) (or other binding agents) (bound) and solid (particulate) phases.
- Transport (erosion, deposition, net burial) of solids and particulate chemicals
- Infiltration of dissolved and bound (mobile) phase chemical

TREX has a number of features specifically designed to aid visualization of chemical transport and fate simulation results. TREX can provide output in several different formats including point-in-time, point-in-space, cumulative-time, and mass balance report.

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CHAPTER III

SAMPLE COLLECTION AND METHODOLOGY

This research is focused on solid wastes obtained from all activities in the Akara Gold mine; they include especially waste rocks from mining operation and tailings from mineral processing. It should be reminded that potential environmental impacts on acidic leaching and heavy metal contamination from these mining wastes are the main concern under this study. Sample collection was then planed following the activities of the mine, focused particularly on solid wastes (i.e., rock waste and tailing), as reported in the previous chapter. At the end of the study, protection plan and recommendations would be suggested based on these data and application of numerical model of runoff.

Characterization and quantification of potential toxic elements were carried out using Inductively Coupled Plasma Optical Emission Spectroscope (ICP-OES). The sample powers were digested under EPA 3052 and EPA 3051 standard methods. EPA 3052 method was initially applied to analyze a total decomposition in waste rocks and tailings. Subsequently, EPA 3051 method was then applied in order to determine leaching potential of toxic metals. Some ore- initial material of tailings- and topsoil samples from undisturbed area were also collected and analyzed for multi-elements to compare the results with waste rock and tailing. Then, the solution samples were analyzed using Inductively Coupled Plasma Optical Emission Spectroscope (ICP-OES), model vista-mpx CCD simultaneous ICP-OES, Varian at NEC-EHWM.

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Undisturbed top soils were also studied for the multi-element analyses and used as a background concentration in the vicinity area. Ores are not classified as a mining waste although they have been stored onto the stockpile which is temporally exposed to the environment. It is however assumed that all ore materials will be taken into the mineral processing and none of them will be left after the mining is closed.

In addition, major compositions of all waste rock groups and tailings were determined in the form of oxide contents using X-Ray Fluorescence Spectrometer (XRF). X-Ray Fluorescence (XRF) spectrometer model Axios, PANalytical based at NCE-EHWM were initially engaged to quantify major and minor chemical compositions of the samples. This was taken as a key to understand their geologic characteristics and environmental signatures. These data are significantly applied to discussion part as well as environmental management plan and mining operation.

Potential Acid-Forming (PAF) tests were consequently evaluated for all waste rock types and tailing. Acid-Base Accounting (ABA) test and Net Acid Generation (NAG) test, two main worldwide acceptable methods, were performed in this study. These methods are assumed to assess the long-term potential of acid generation. They can predict the post-mining water quality at the site whether acid or alkaline.

The possible risk from the potential acidity source is not only on the mineral characterization of waste rocks and tailings, but also the quality of the potential leachate which is a consequently problem that may distribute the heavy metal direct to the environment. To address the metal drainage, the leaching test at different pH conditions were conducted by using the methods modified from the SW 486 method EPA 1312 method (Synthetic Precipitation Leaching Procedure, SPLP). The results from this study are expected to indicate the significant metal concentration that may pollute the environment (sediment, surface water, ground water, soil, etc.). This leaching test was simulated the acid drainage to assessed the leached metals in case of there are acid mine drainage occurring in the area.

The details for the study area and each stage of this study are described in the following sections.

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3.1 Study Area

Akara Gold Mine

The study area, Akara Gold Mine, is located along the boundary between Phichit and Petchabun Provinces, situated on the eastern edge of the Chao Phraya Basin along a gently undulating terrain, about 280 km north of Bangkok (Figure 3.1). This area comprises the mining and processing of gold ore over a period of fourteen year. Prior to the mining activity, it was a rural area. The land is generally flat countryside with two hills called Kho Mo in the north of the site area, and Khao Pong, near the northern boundary of the site. The vicinity area is mostly occupied by paddy field. Most local people living around the area obtain clean water from rain and groundwater. The Akara gold prospects are located within a zone of silicified volcanics over an area 7.5 km. by 2.5 km. which forms a slightly elevated landform surrounded by low lying rice paddy fields

Geologically, the mining area is situated along the Loei-Petchabun volcanic belt. The belt is occupied by Upper Permian and Triassic parallel zones of ancient island arc containing mafic to intermediate volcanic rocks (e.g., andesite and basaltic andesite) in association with marine sedimentary rocks. Gold-silver ores are found extensively along this belt but most occurrences appear to be invisible deposit with very rare small nugget found. Mineralization has been suspected as epithermal processes occurred in shallow to steeply dipping structures. According to exploration data, the highest commercial gold deposits in the study area are detected in gold-bearing vein, breccia and stock-work veinlets which are mostly characterized by quartz-carbonate veins (Hemmanee and Poot-heng 2006).

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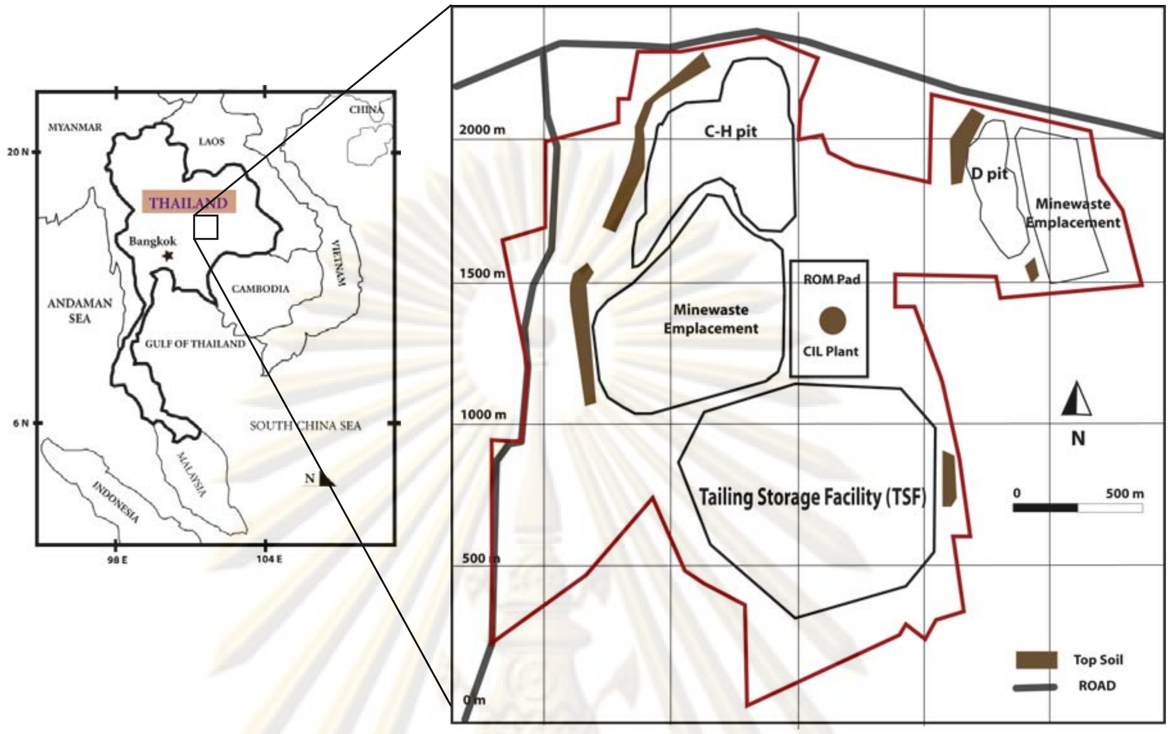

Figure 3.1 The locations of the pits at Akara Gold Mine and priority targets for potential extensions into existing resources.

The initial exploration of the area was based on porphyry copper, a epithermal gold model. There are two main pits separated by Khao Pong: C-H and D pits. The C-H pit, developed first, is the largest ore reserve. C-H Pit covers a maximum area of 167 Rai. C-H pit is mined to a depth of approximately 110 meters below the existing ground level. C-H is an active pit. D pit, developed after C-H, covers a maximum area of 42 Rai. Pit D is mined to a depth of approximately 70 meters below the existing ground level. There is an ore processing plant constructed within the mine site that processes the mined ore.

Ore grade gold values are confined to the quartz, carbonates veins, stockworks and breccias. Gold values occur over a wide range of grade, with the average being 2.5 g/t. The

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silver to gold ratio in the primary ore is 2.5:1 with more than 5:1 in the enriched oxide and transition zones.

Ore from this pit stored on an area of approximately 18 rai, from here it is fed into the ore processing plant. Mine wastes are transported to the dumping area. There are 2 mine waste emplacements to be used throughout the lifetime of this mine. The largest, approximately 315 rai, services the waste from the C-H pit. The mine emplacements 2 and 3 cover an area of approximately 86 Rai and 41 Rai, respectively. These areas service for the waste from the D pit. Top soil over the effected area is hauled in by a truck to the stockpile adjacent area to the pit and mine waste emplacement. Huge quantities of rock wastes will be left behind the mining end and may be able to generate the enormous environmental impact if great care is not taken into account.

There is a tailing storage located on the southern area of the mine site that stores the mine processing tailings. It covers an area of approximately 200 Rai. In the preliminary plan, the embankment was constructed to store a maximum of 9.75 million tones of tailings over the lifetime of the project. But now, since the project will be extended and tailing storage will be increased to a capacity of 19 million tones of tailings.

3.2 Sample Collection

Field survey and sample collection were conducted in June 2005 and March 2006.

Waste rocks: 5 types of waste rock, following classification of mining geologist, were collected with 5 samples (about 1 kg) of each groups around the C-H pit (see Figures 3.2 and 3.3). These waste rocks are classified as volcanic clastic rock, porphyritic andesite, andesite, silicified tuff, silicified lapilli tuff and shear tuff (Ore contact). After sample selection, they were sealed within plastic bag and transport to laboratory.

Ores: 2 types of ore-containing rocks (mineralized rocks) have been classified, based on the gold-silver proportion, as high grade ore (higher 1 g/ton gold) and low grade ore (0.8-1 g/ton gold) during the mining process. Both groups were collected from the stock pile at the Akara Gold mine. Five samples of each group were selected as representatives.

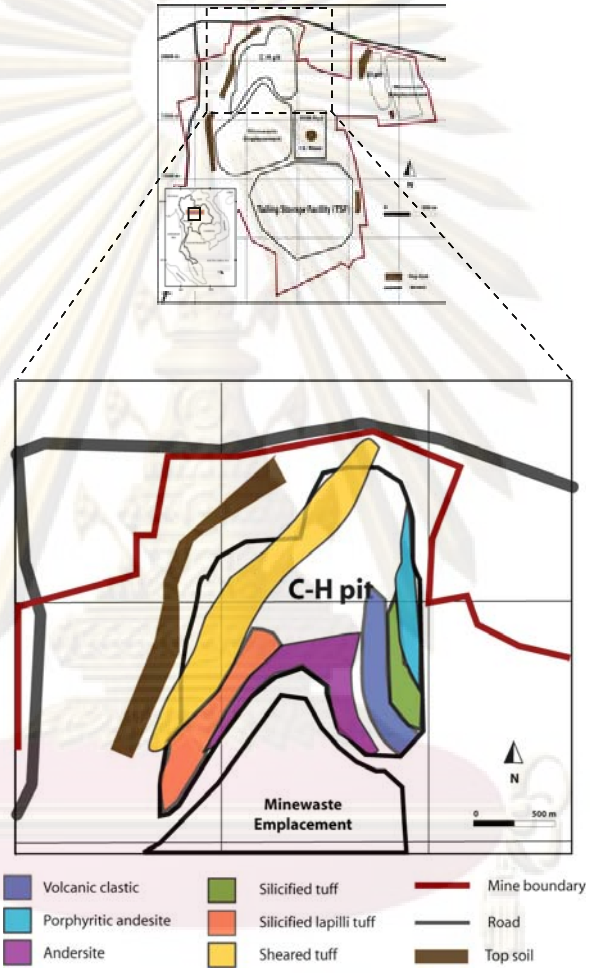

Figure 3.2 Map of C-H pits, and sampling location of each rock type; 1: Volcanic Clastic, 2: Porphyritic Andesite, 3: andesite, 4: silicified tuff, 5: silicified lapilli tuff and 6: Sheared tuff.

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Figure 3.3 Waste rock samples were collected around the C-H pit of Akara Gold Mine.

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Tailings: have been disposed within the tailings storage facility (TSF) covering the area of about 3.2 km^2. The locations of sampling point in tailing storage facility (TSF) are shown in Figure 3.4. All samples were taken using hand auger (Figure 3.5) and planed to represent whole tailings produced over the receding 3 to 4 years of mine production. However, old tailings placed beneath the storage are soaked with water which cannot be collected. Finally, tailing samples were collected from 13 different locations distributed over the TSF area. Three samples of all locations were taken on the surface, at depths between 0-1 m and 1-2 m, respectively. About 1 kg of moist tailings were then sealed in plastic container and transported to laboratory.

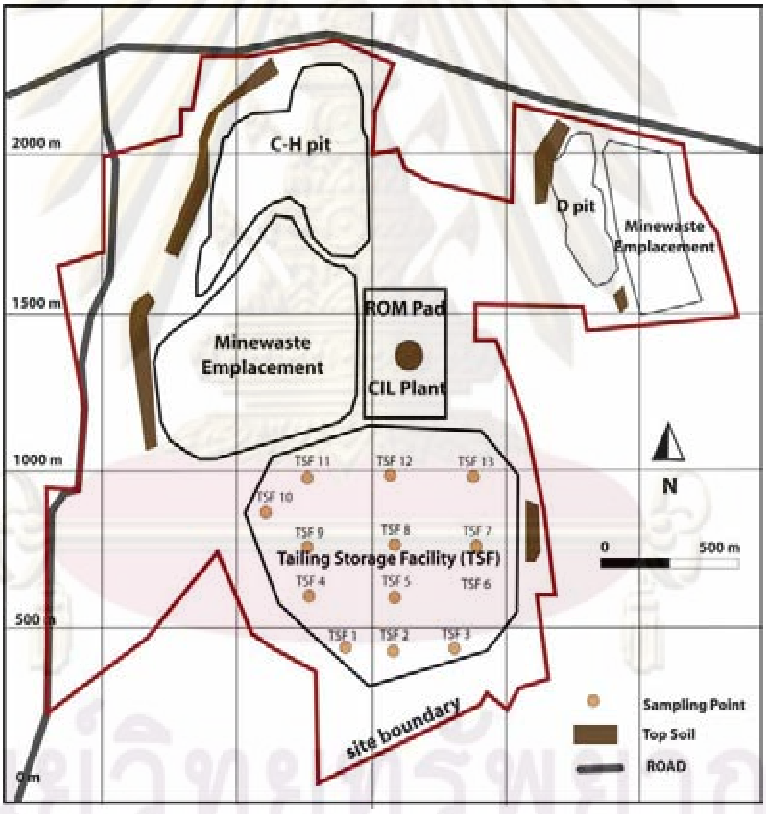

Figure 3.4 Map of the study site, showing the location of sampling point in tailing storage facility (TSF), Akara Gold Mine in Phichit-Petchabun provinces. (map modified from URS 1999).

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Figure 3.5 Tailing storage facility and sample collection using hand auger.

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Undisturbed Topsoils: are assumed to be representative of environmental background of the study. They were sampled from the undisturbed area nearby the mine site. These samples were randomly collected on from surface downward 30 cm depths at each point (Figure 3.6). All samples were then packed, transported back to the laboratory.

Figure 3.6 Soil sample collecting from the undisturbed area nearby the mine.

3.3 Sample Preparation

Tailings and Topsoils: all tailing and topsoil samples taken from the mine were delivered to the laboratory immediately for analytical preparation. They were air-dried for about 72 hours. Subsequently, the dried samples were crushed using agate mortar into fine powders and passing through 0.075-mm mesh prior to further experiments. Samples were homogenized, stored and sealed within polyethylene bag until analysis were take place. This sample storage procedure was suggested by Tessler et al. (1979).

Waste Rocks and Ores: Low grade ore (0.8-1 g/ton gold), high grade ore (>1 g/ton gold) and waste rock (mostly <0.8 g/ton gold) samples were crushed using jaw crusher before grinding with tungsten-carbine disk mill. Then passed through 0.075 mm mesh prior

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to further processes; however, all milled samples had be been stored in polyethylene bags until analysis taking place. This preserved procedure was recommended by Tessier et al. (1979). All plastic containers and glassware used in laboratory were cleaned with de-ionized water then immerging in 10% nitric acid for at least 12 hr before rinsing again with double de-ionized water. Crushing and milling machines are shown in Figure 3.5. All of these steps were carried out at Department of Geology, Faculty of Science, Chulalongkorn University.

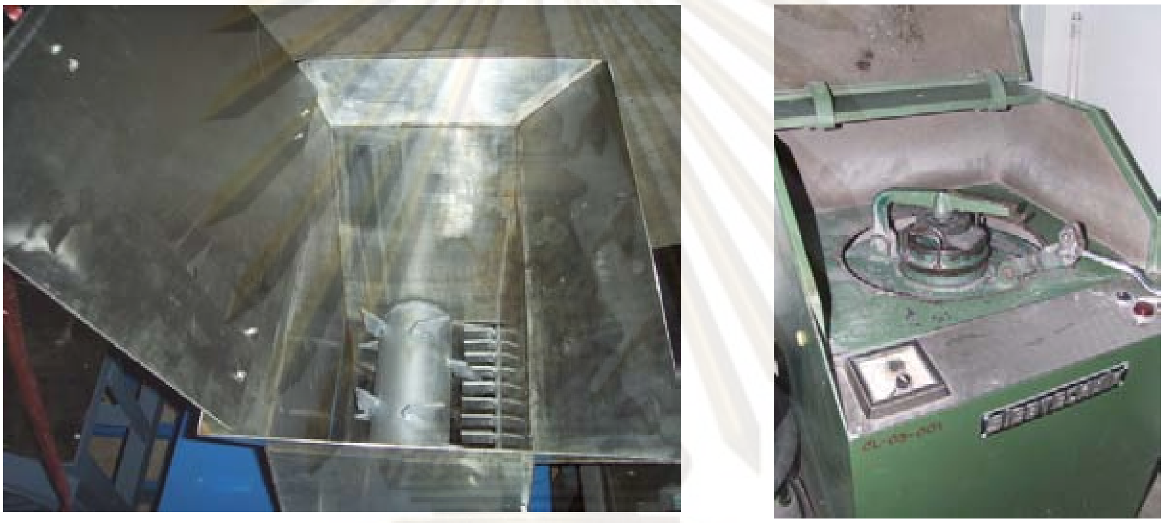

Figure 3.7 Jaw crusher and disc mill machines based at Department of Geology, Faculty of Science, Chulalongkorn University.

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3.4 Analytical Procedure

3.4.1 Major Element Analyses

X-Ray Fluorescence (XRF) spectrometer is a common technique used to determine the major and minor chemical constituents of rocks and minerals (Norrish and Chappell, 1967; Van Johnson and Maxwell, 1981) because its analytical technique is easily applied to solid samples and sample preparation is not complicated. In this study, samples were prepared as fused glass bead. 0.5 g powder sample was mixed with 6.5 g lithium tetraborate ($Li_2B_4O_7$) in ball milling process. Subsequently, mixed material was fused at high temperature. Eventually, major elements in oxide forms were obtained using a PANalytical ModelAxios system XRF spectrometer in calibration with geochemical standards. Condition of operation was set at 24 kV/100 mA for analyses of Na_2O, MgO, Al_2O_3, SiO_2, P_2O_5, K_2O and CaO and 60 kV/40 mA for analyses of Fe_2O_{3total} before calibration with reference standards including sediment and igneous rocks such as AGV-2, BHVO-2, COQ-1, SCo-1, SGR, SDO-1, STM-1, PCC-1 and JG-1 which are supplied by U.S. geological survey (USGS) and the geological survey of Japan (GSJ). Automatic calculation was then taken before weight percent oxides were reported.

3.4.2 Trace and Potential Toxic Element Analyses

Microwave-assisted acid solubilization has been recommended with proof as the most suitable method for digestion of complex matrices such as soil and rock. This method shortens digestion time, reduces risk of external contamination and uses smaller quantities of acid (Wang et al., 2004). Under this study, all samples were firstly carried out in forms of total concentrations which were digested following the EPA 3052 procedure. This method is appropriate for those applications requiring a total decomposition for research purposes (i.e., geological studies, mass balances, analysis of Standard Reference Materials) or in response to a regulation that requires total sample decomposition. On the other hand, EPA

3051 is applied to evaluate leaching potential of elements, particularly concern of toxic leaches, that maybe taken place under the reproductive acid mine drainage. All solutions obtained from both digestion procedures were subsequently analyzed by Inductively Coupled Plasma Optical Emission Spectroscope (ICP-OES) with geological standards were used for calibration.

EPA 3052 Procedure: Add a representative sample of up to 0.5 g into a digestion vessel and add 9 ml concentration nitric acid (HNO_3) and 3 ml hydrofluoric acid (HF) carefully into the digestion vessel. Close the Teflon cover. The vessel is sealed and heated in the microwave system. The temperature profile is specified to permit specific reactions and incorporates reaching 180 ± 5 ºC in approximately less than 5.5 min and remaining at 180 ± 5 ºC for 9.5 min for the completion of specific reactions. After cooling, filtrate the solution through a 0.45 μm pore diameter of Whatman filter paper. At the end, further dilute the filtrated solution in a 50 ml volumetric flask and keep it in a polyethylene container at 4 °C ICP.

EPA 3051 Procedure: The sample material of 0.5 g is transferred into a fluorocarbon microwave digestion vessel with 10 ml nitric acid. The temperature was risen to 175 °C for less than 5.5 minutes and remained between 170-180°C for the balance of the 10 min irradiation period. The sample is cooled and the supernatant diluted with 50 ml of deionized water. The solution was then filtered through a 0.45 μm pore diameter of Whatman filter paper. At the end, further dilute the filtrated solution in a 50 ml volumetric flask and keep it in a polyethylene container at 4 °C.

ICP Analysis: The solution samples obtained from both microwave digestion procedures were analyzed for the heavy metal elements by Inductively Coupled Plasma Optical Emission Spectroscope (ICP-OES). Geological standards such as Marine Sediment (MAG-1), Andesite (AGV-2) and Basaltic Hawaiian Volcanic Observatory (BHVO-1) were also digested under EPA 3052 to standard solutions then used to calibrate analyses of all

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samples. Bank and standard solutions prepared from AGV-2, BHVO-1, and MAG-1 were also systemically used per each sample lot of 10. Finally, concentrations of elements were computerized and reported.

3.4.3 Acid Potential Test

3.4.3.1 Acid-Base Accounting (ABA)

The values measured from the acid-base accounting are referred to the maximum potential acidity (MPA) and the acid neutralizing capacity (ANC), respectively. The difference between the MPA and ANC value is referred to the net acid producing potential (NAPP). The MPA of a sample is calculated from the total sulfur content that could be generated by assuming that all the sulfur occurs as reactive pyrite according to the following reaction:

$$FeS_2 + 15/4\ O_2 + 7/2\ H_2O \Rightarrow Fe(OH)_3 + 2\ H_2SO_4$$

According to above reaction, the MPA of a sample containing 1 %S as pyrite would be 30.6 kilograms of H_2SO_4 per ton of material (i.e., kg H_2SO_4/t).

$$\text{MPA (kg H2SO4/t)} = \text{(Total \%S)} \times 30.6$$

The ANC is commonly determined by the Modified Sobek method (sobek, 1978). The method involves reacting a known amount of sample of acid at pH less than 2 for 1 to 2 hours which it should follow precursor fizz test for fizz rating. Fizz rating can be done by add one or two drops of 1:3 HCl to the sample. The percentage of CaCO3 is indicated by a bubbling or audible “fizz”. Rate of the bubbling or “fizz” and the addible acid was shown in Table 3.1.

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Table 3.1 Volume and normality of hydrochloric acid used for each fizz rating

Fizz Rating	HCl	
	ml	Normality
None	20	0.1
Slight	40	0.1
Moderate	40	0.5
Strong	80	0.5

Then, a known amount of standardized hydrochloric acid (HCl) is added to an accurately weighed sample, allowing the sample time to react (with heating), then back-titrating the residual acidity to determine the amount of acid consumed by sample with standardized sodium hydroxide (NaOH) to determine the amount of unreacted HCl. The amount of acid consumed by reaction with the sample is then calculated and expressed in the same units as the MPA, kg H_2SO_4/t of sample.

3.4.3.2 Net Acid Generating Test (NAG Test)

NAG is used to verify the NAPP prediction. Direct measurement of NAG also provides an indication of the reactivity of the contained sulfides (Miller et al., 1994 and 1997).The test involved reaction of a sample with hydrogen peroxide (H_2O_2) to rapidly oxidize any sulfide minerals presented. Both acid generation and acid neutralization occur simultaneously during the NAG test. All parts of the acid generated by this oxidation were then consumed by alkaline constituents in the sample. If the sample after reaction has a pH of 4.5 or less that it is considered to be acid forming. The net amount of acid remaining after reaction is determined by titration to pH 7.0 and usually expressed as kg H_2SO_4/t of sample.

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3.5.4 Leaching Test

Samples were analyzed for leaching test in order to find how many metals will be leached if AMD is generated. In addition, this is aimed to fine out the amounts of leached metals that may transfer into the water with different pH drainages. The experiment was performed under SW 486 method EPA 1312 (Synthetic Precipitation Leaching Procedure, SPLP) with some modifications. SPLP was selected to use in this study because it synthesizes an aqueous solution intended to simulate rain/acid rain water (of sulfuric and nitric acids). The experiment was started by prepare the mixture of acid solution. Sulfuric (H_2SO_4) and nitric (HNO_3) acids were mixed under the ratio 60:40 by weight. The mixed acid was added to the deionized water until reaching pH 2. Then add the mixed acid (H_2SO_4 and HNO_3) 20 ml into 1 g of sample. Shake for 18 hrs under 30 rpm. Filtrate the leaching fluid from the solids using 0.45 μm pore size filter paper. Repeat all steps by adjust the pH of deionized water by the mixed acid to pH 4 and again with pure deionized water with no acid mixing. All the leaching fluids of all pH conditions were analyzed ICP-OES.

3.5 Numerical Modeling

3.5.1 TREX Model

TREX was used to trace the fate and transport of metals in the catchment. A model is created using the following procedures.

1) Data preparation includes DEM, landuse and soil shape files, land cover (crop types) and land use practices. Input data require soil and landuse shape files whereas output, dominant soil and land use ID’s in each cell are exported as ASCII file.

2) Delineation of watershed requires input data of DEM 90 m x 90 m to generate grids cells not larger than 30,000 to 50,000 cells in the model. TREX is fully-distributed and is

designed to be compatible with data from raster GIS sources. In particular, data describing elevation, soil types, land used and contaminant distributions can be processed in a GIS and used as model inputs.

TREX model is executed to generate results. Model outputs are also designed to be compatible with GIS system. The result can be exported in ASCII file. Then, it will be imported to GIS. The model can be calibrated using runoff, sediment loss and historical metal level in the area and then validated with a new set of validation data and measured metal level from the field. Within the study watershed, database was defined into 12 soil classes and 6 land use types.

The channels are segments connected from overland grid cell centering to grid cell center that represent rivers or creeks in the watershed. The location of channel cells within DEM is typically determined from stream network generation techniques using GIS system. Alternative tools to determine locations of channel cells and stream channel network in this research is ArcMAP 9.3.

In order to model channels, the users have to firstly specify a stream network that defines the location of cells containing channel segments. The topology of this network is then used to specify two maps to TREX that contains the connectivity information.

The first map is called a “link” map containing a grid of integers that denote channel locations for each grid cell within the watershed as well as how each channel segment or “river reach” is connected to the other. A “node” map is derived from a link map and contains integer numbers that designate the connectivity between each grid cell (and thus flow direction) within an individual link.

The project area boundary, stream channel, soil type and land use data for the studies are presented in Figures 3.8 to 3.11.

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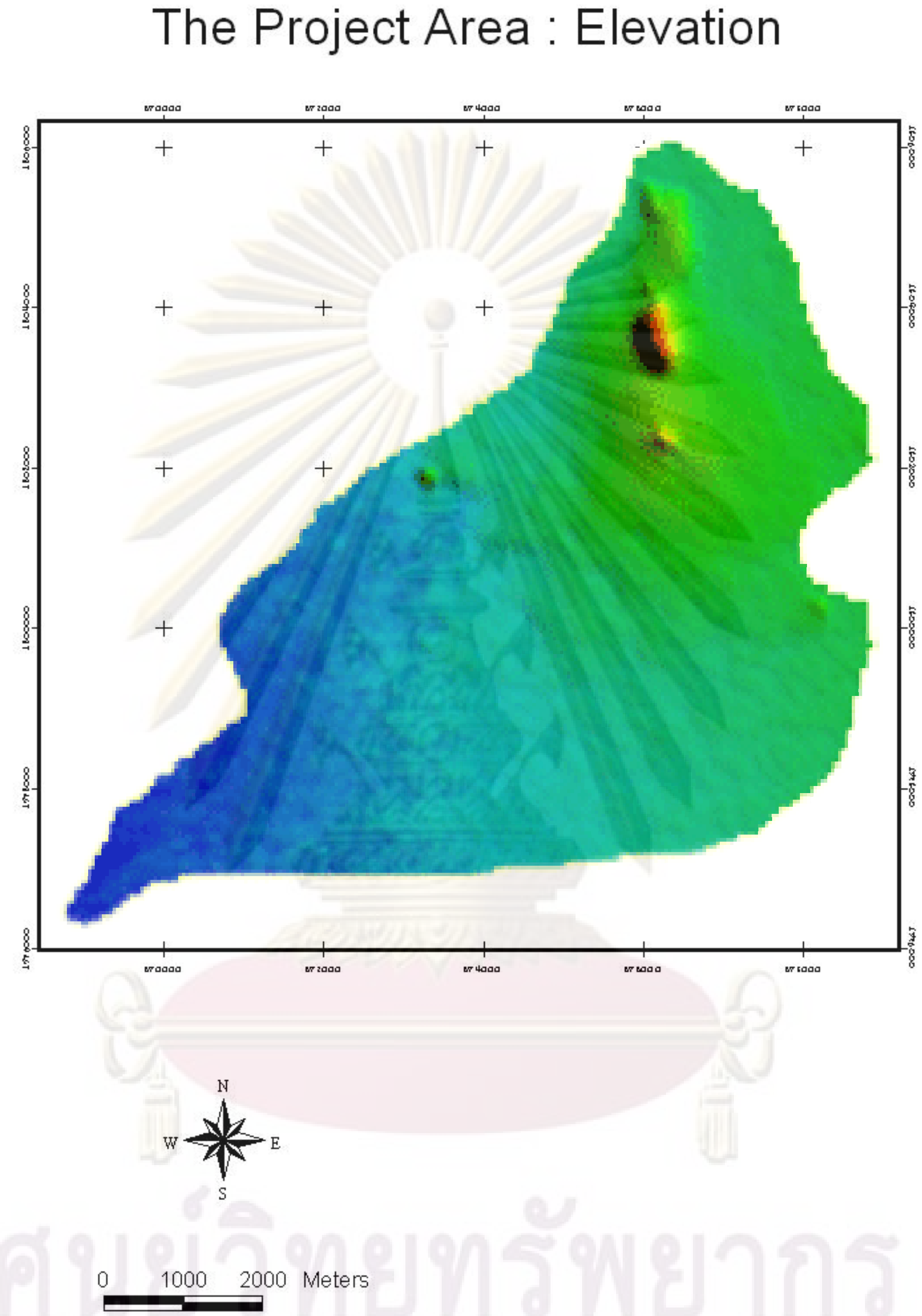

Figure 3.8 The Akara mine and its boundary showing project elevation.

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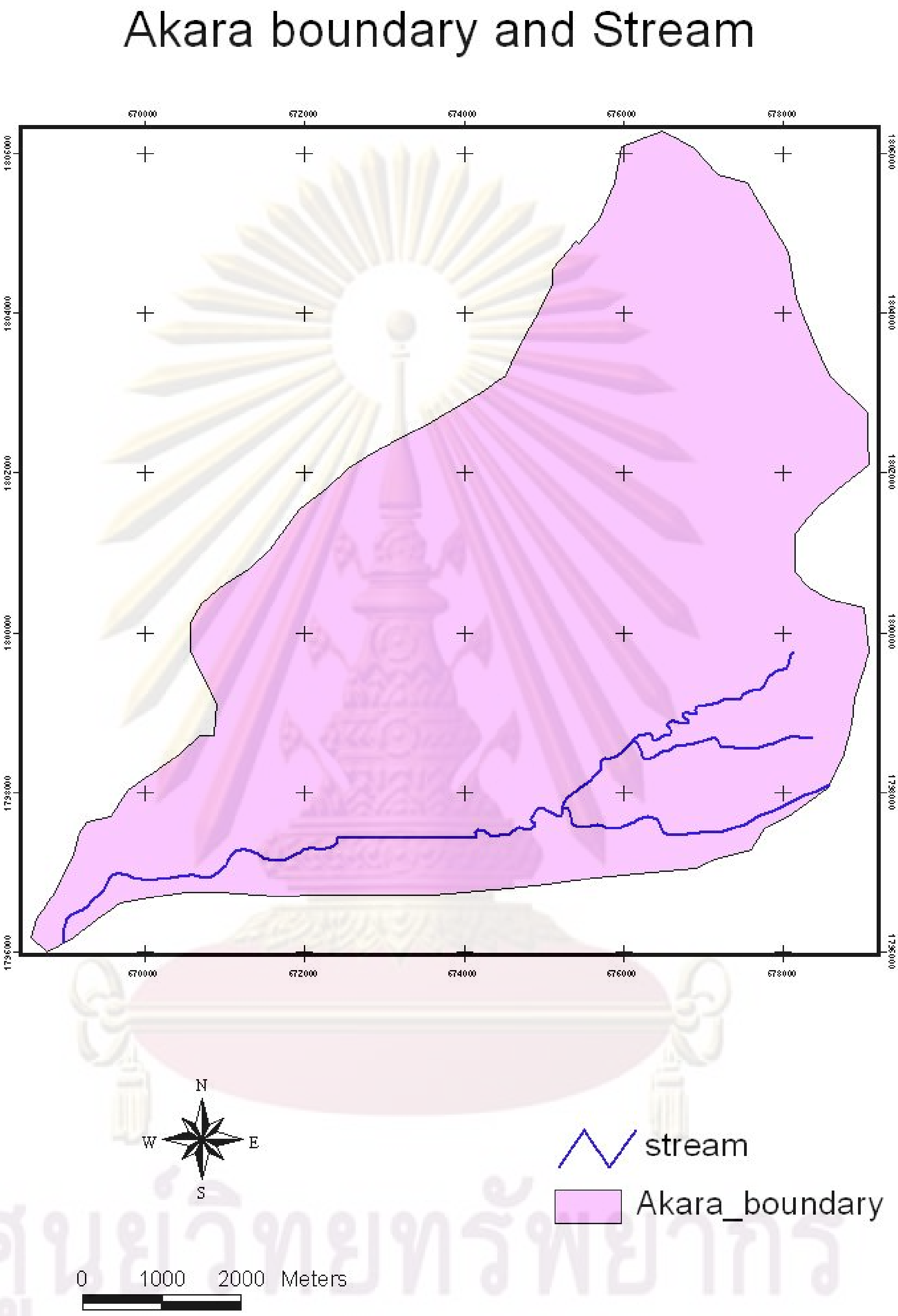

Figure 3.9 The Akara mine and its surrounding area showing project boundary, water channel/stream locations.

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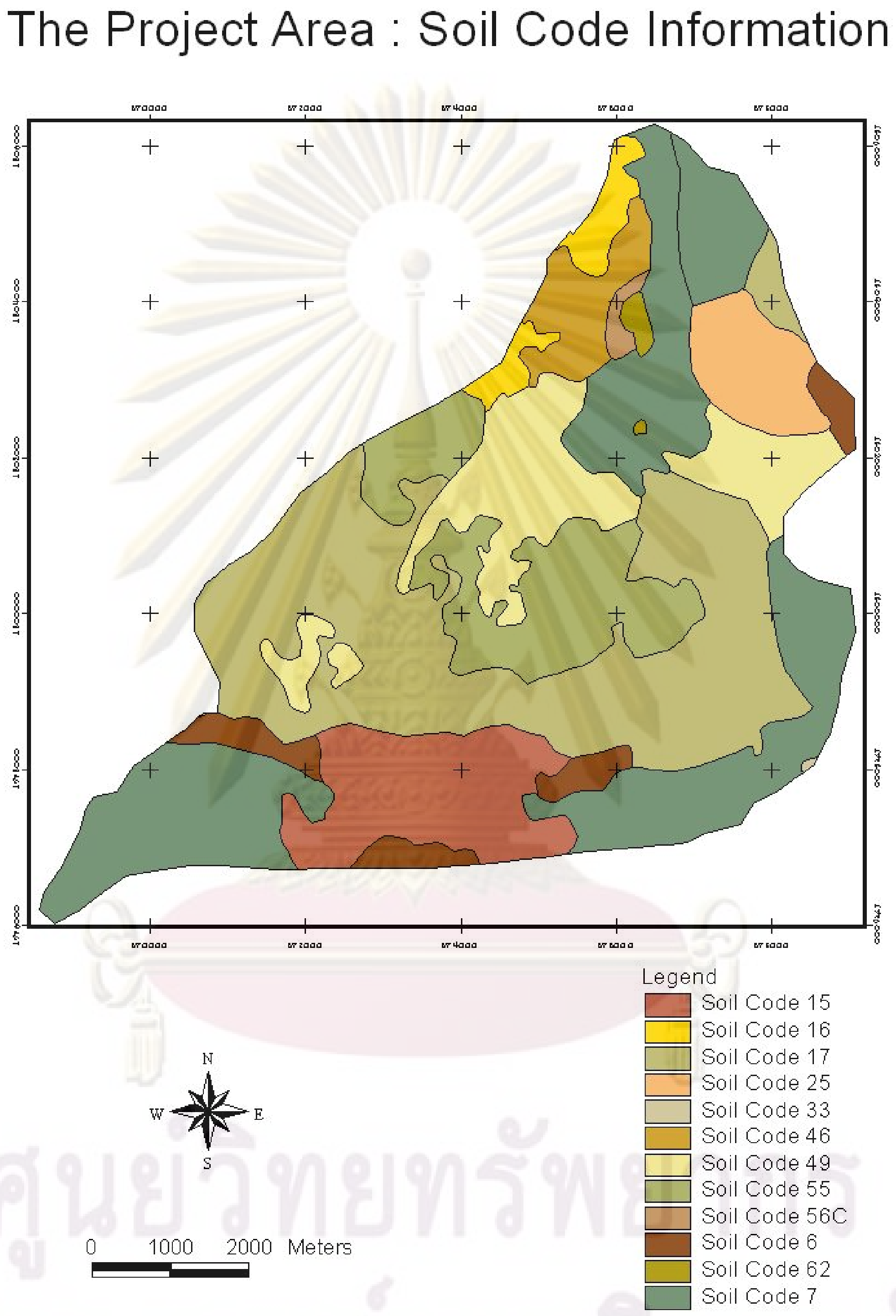

Figure 3.10 Soil types within the Akara mine project boundary.

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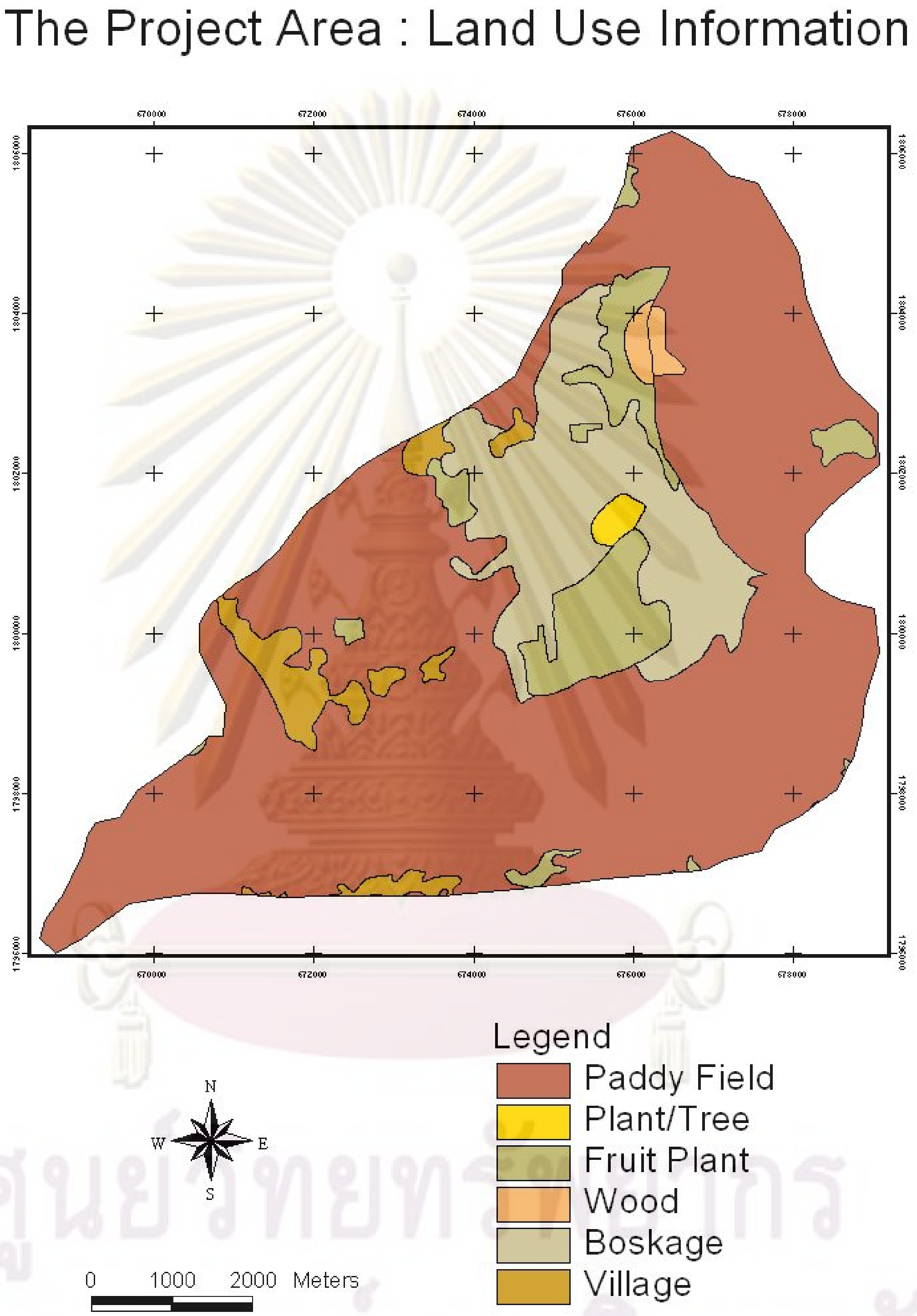

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Figure 3.11 Land use within the Akara mine project boundary.

Under the required data for TREX simulation, the explanation of the data was gathered. Within the watershed, the Land Development Department database defines 12 soil classifications to 12 soil codes and 13 land use types. These soil codes are combination with the data from AGIESU (AGricultural Information and Expert Systems Unit), Kasetsart university where includes the description details of grain size distribution. The details of grain size distribution (e.g., % sand, % silt and % clay), pH and soil texture are reported in Table 3.2.

Table 3.2 Grain size distribution for soil codes.

Soil Fraction	SOIL CODE											
	6	7	15	16	17	25	30	33	46	49	55	56
% sand	33.3	25.9	6. 8	23.7	52.7	42.7	26	27.7	38.3	47	25.6	75.3
% silt	35	43.8	57.7	65.3	33.8	39.9	24.6	57.9	29.6	34	40.9	15.5
% clay	31.8	30.3	34.5	15.2	16.2	18.9	49.5	14.5	38.6	15.8	33.5	12.7

Source: AGIESU, 2008

Then the soil triangle of basic soil textural classes suggested by the U.S. Department of Agriculture (USDA) and the U.S. Soil Conservation Service was used to classify soil texture. The soil classification type is determined by plotting the percentages of three individual soil particle classes found within the soil sample on the USDA Soil Textural Triangle (Figure 3.11). The point at the intersection of these three particle class percentages will fall within one of the soil classification types (see Figure 3.12). The soils in the vicinity of Akara mining were then classified as shown in Table 3.3.

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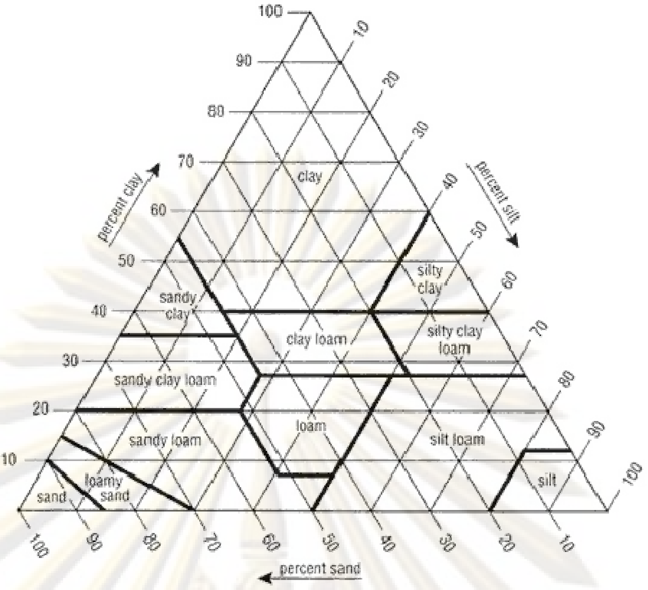

Figure 3.12 Soil Texture Diagram shows proportions of sand, silt and clay for soil classification (U.S. Deparment of Africulture, 1951).

Table3.3 Classification of Soil in the project boundary.

Soil code	Soil class	Soil code	Soil class
Soil code 6	Clay loam	Soil code 30	Clay
Soil code 7	Clay loam	Soil code 33	Silty loam
Soil code 15	Silty Clay loam	Soil code 46	Clay loam
Soil code 16	Silty loam	Soil code 49	Loam
Soil code 17	Sandy loam	Soil code 55	Clay loam
Soil code 25	loam	Soil code 56	Sandy loam

These classified soils are used to select the other appropriate parameters in modeling. The model is then expected to yield reasonably good results in terms of runoff hydrographs, sediment transport in suspension and metals at the other internal station of the watershed. Model parameters subject to calibration are effective hydraulic conductivity (Kh), roughness (Manning n), soil erodibility (K), land cover factor (C) and the chemical distribution (partition) coefficient (Kd). The parameters used in TREX to generate the metal distribution under this study are shown in Table 3.4.

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Table 3.4 Summary of model parameter values for the project boundary (after Wanielista et al., 1997).

Parameter	value	Application
Hydraulic conductivity (m/s)	0.046	Clay loam
	0.018	Silty Clay loam
	0.135	Silty loam
	0.510	Sandy loam
	0.260	Loam
	0.012	Clay
Minimum infiltration (in/hr)	0.09	Clay loam
	0.06	Silty Clay loam
	0.27	Silty loam
	1.02	Sandy loam
	0.52	Loam
	0.02	Clay
Effective Porostiy	0.390	Clay loam
	0.432	Silty Clay loam
	0.486	Silty loam
	0.412	Sandy loam
	0.434	Loam
	0.385	Clay
Manning's N	0.01 – 0.013	Concrete
	0.01 – 0.015	Asphalth
	0.012 – 0.030	Graveled surface
	0.012 – 0.033	Bare clay-loam
	0.39 – 0.63	Grass

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CHAPTER IV

CHEMICAL CHARACTERISTICS AND ACID DRAINAGE ASSESSEMENT OF MINE TAILINGS FROM AKARA GOLD MINE THAILAND

Abstract

Acid Mine Drainage (AMD) is a great concern in many abandoned mines because of its adverse effect on environment. In mining processes, many kinds of wastes are produced. These wastes may become eventually sources of environmental degradation. The focus of this study is the geochemical characterization of the end-processed tailings generated by Akara Gold Mine, the biggest gold mine in Thailand. Tailing samples were systematically collected for analyses of chemical and mineralogical compositions. As a result, their quantitative chemical analyses are slightly different from place to place but mineral components can not be clearly differentiated. For instant, it may be assumed that the end-processed tailings, which were mixture between high and low ore concentrates, would have similar mineral components. However, the little variation of chemical composition may be caused by the ore refining processes that are somehow varied in proportion of chemical additives, alkali cyanide and quick lime in particular. In addition, clay composition in ore-bearing layers may also influence alumina content of tailings, accordingly. Distribution of the tailings is not related to depth and distance of tailing storage pond due to disposal step has spreaded them over the pond during operation.

Total heavy metals of the tailing samples were analyzed on the basis of EPA 3052 method. Consequently, most toxic elements (e.g., Co, Cu, Cd, Cr, Pb, Ni and Zn) were found falling within the standard of Thailand Soil Quality Standards for Habitat and Agriculture. Only Mn appears to have higher content than the standard. In addition, leaching tests had also been tested even these tailings contain low metal concentrations. As a result, at pH 2 Mn can leach out exceeding the Thailand Surface Water Quality Standard for Agricultural (Mn < 1ppm) and the Thailand Industrial Effluent

Standard (< 5 ppm). Although leachate at pH 4 and neutral condition contains lower Mn than the Industrial Effluent Standard it still exceeds the surface water quality standard. Interestingly, Pb can be leached out exceeding both standards (0.2 ppm for the industrial effluent standard and 0.05 ppm for the surface water quality standard). For Ni leaching, its concentration is lower than the Industrial Effluent Standard at all pH conditions but it still exceeds the Surface Water Standard at pH 2 and 4. This information would be taken into consideration of the further environmental monitoring.

Acid generating potential of the tailings was estimated using acid-base accounting (ABA) and Net Acid Generation (NAG) tests. The results of ABA and NAG tests show that the tailing samples contain a high amount of sulfur. However, they also contain high acid neutralization capacity. Consequently, these tailings may not have potential to generate acid drainage; on the other word, they can be classified as non-acid forming (NAF) material. However, since these tailings contain some heavy metals (e.g., Ni, Mn and Pb) that are observed from leachates exceeding the standards at low pH; the AMD conditions may lead to heavy metal releasing. Therefore, preventions of oxidizing process and dissolution would be concerned with great care. In addition, Mn and Pb can also be leached at neutral conditions then both elements would be taken into consideration. Barrier of air and water, clay layer for example, would be placed over the tailings pond before covering of topsoil for re-vegetation. Native grass is recommended to grow for stabilization of surface and reducing erosion rate. Monitoring of water quality would also be carried out annually.

4.1 Introduction

Mining processes including raw material grinding, ore refining and solid waste disposal may be sources of contamination to the environment (Adriano et al., 2001). Enormous solid wastes with high risks to acid generation and heavy metal leaching may cause contamination of surface water and groundwater during mine operation or even long after mine closure (Conesa et al., 2007). Among the solid wastes, tailings are one of

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the most concerns, particularly when they have low pH and high concentrations of heavy metals (Shu et al., 2001). This eventually leads to Acid Mine Drainage (AMD) that usually contains high levels of heavy metals. Consequently, AMD significantly impacts water quality and natural ecosystems (Hu, 1998).

Sulfur-bearing tailings disposed on neutral or low alkaline conditions can be weathered within a few months or a few years; consequently, generation of severe acidity and acid mine drainage may occur (Robbed and Robison, 1995). The extent and degree of heavy metal contamination around the mining activity usually vary from place to place depending on geochemical characteristics of tailings (Jung, 2001). Management of tailing placement is extremely important to minimize the environmental impact. For example, Portman Bay area in Spain has suffered from major impact resulting of mining activity which millions of ton of mine tailings had been dumped into the bay over a long period and gradually filling the bay (Martínez-Sánchez, 2007).

During the past 30 years, various techniques have been developed to predict acid-forming potential of mine waste materials. However, two commonly used techniques are Acid-Base Accounting (ABA) procedure and hydrogen peroxide direct oxidation procedure called Net Acid Generation (NAG) test (Ferguson and Erickson, 1988; Paktunc, 1999; Jennings et al., 2000). Recently, both methods have been typically used to confirm results from each other.

The largest gold mine in Thailand, Akara Gold Mine, was selected for this study because it has been in operation for fourteen years with great care of environmental issues. Since AMD has been widely known as a major cause of environmental pollution, ABA and NAG tests have been taken into consideration with increasing trend in the near future. The main objective of this study is to evaluate the acid-forming potential of mine tailings at the gold mine using both ABA and NAG methods; apart from that, their chemical and mineral compositions were initially investigated before leaching experiments of heavy metal were carried out at various pH conditions. The results earned from the study could be very useful for environmental management and best practice of the mine operation.

4.2 Study Site

The Akara Gold Mine is located along the boundary between Phichit and Petchaboon provinces (Figure 4.1). Gold deposit of the area has been classified as an epithermal gold genesis. The mining has been expected to last for twenty years. During the operation, waste rocks have been burst and partly used for construction of a tailing storage facility (TSF) whereas the rests have been disposed within three emplacements. During ore processing, approximate 750,000 dry tons per year of tailings have been generated and deposited in the TSF.

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Figure 4.1 Sketch map of the study site, Akara Gold Mine along the boundary of Phichit and Petchaboon provinces, showing land utility including tailing storage facility (TSF) in which samples were collected for the study (map modified from URS 1999).

The TSF is located on the southern area of the mine (see Figure 4.1). In the preliminary plan, the embankment was constructed covering an area of about 320 hectares to store a maximum of 9.75 million tons of tailings over the lifetime of the project. However, since the company has proposed for extension of their concession on mining project, the tailing storage is then expected to be increasing its capacity to 19 million tons of tailing (Akara Mine, 1999).

The vicinity area is mainly flat plain with only two hills, namely Khao Mo and Khao Pong, located in the north of the mining site. The regional area is mostly occupied

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by agriculture land, paddy field in particular. Hence, AMD and its consequences would be a major environmental concern, especially in the case of flooding over the surrounding areas.

4.3 Ore Processing and Tailing Producing

Physical and chemical processes are taken to involve in ore dressing. After the crushing unit, crushed ore is transported directly to the semi-autogenous grinding (SAG) and followed by gravity concentration using a ball mill combined with six hydrocyclones. Crushed ore is socked with water before feeding into SAG. After passing through the gravity concentration process, gold is extracted from the ball mill cyclone and overflowed to the carbon in leach (CIL) unit in which mixture of sodium cyanide and alkaline solution are added to produce gold cyanide. The alkaline solution is added to ensure that free cyanide ions, which are essential for the reaction, are not lost as free cyanide gas. Besides, quicklime (CaO) is added directly to the mill feed conveyer to achieve an optimum pH of 10.3 suitable for cyanidation. The other necessary components are oxygen and water. The electrochemical reaction of gold dissolution is described by Elsener's equation:

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

The following unit operations are carried out in the stripping and goldroom area which include acid washing of carbon, stripping of gold from loaded carbon, electrowinning of gold from pregnant solution and carbon regeneration. Finally, wastewater from the process and solid waste, so-called tailings, is discharged into the TSF. Therefore, before the wastewater runs through to the TSF, it has to undergo a cyanide destruction process in order to reduce the cyanide concentration to an acceptable level prior to discharge to the TSF (Akara Mine, 1999). Tailing is typically dumped into low-lying areas. It is characterized by sand-like material with various colors between light brown and dark grey.

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4.4 Sample and Analytical Method

Tailing samples were collected systemically from 13 different locations (TSF 1 to TSF 13) distributed over the area of TSF (Figure 4.1). Most samples were collected at 3 different depths of 0, 1 and 2 m. Except at the storage edge, only one sample could be taken on the surface (0 m) due to shallowness of tailing. Hand auger was used for sampling; hence, slurry material, usually situated under 2 m depth could not be taken for investigation. Subsequently, samples were kept in clean plastic bags and transported to laboratory where all samples were air-dried, hand crushed and sieved through a 0.075 mm mesh prior to the experiment.

Analyses and experiments were carried out at the National Center of Excellence for Environmental and Hazardous Waste Management (NCE-EHWM) and Department of Geology, Faculty of Science, Chulalongkorn University. X-Ray Fluorescence (XRF) spectrometer (model Axios, PANalytical based at the NCE-EHWM) was initially engaged to quantify major and minor chemical compositions of the samples. Operational condition was set at 24 kV/100 mA for analyses of Na_2O, MgO, Al_2O_3, SiO_2, P_2O_5, K_2O and CaO and 60 kV/40 mA for analyses of Fe_2O_{3total} before calibration with reference standards including sediments and igneous rocks such as AGV-2, BHVO-2, COQ-1, SCo-1, SGR, SDO-1, STM-1, PCC-1 and JG-1 which are supplied by the U.S. Geological Survey (USGS) and the Geological Survey of Japan (GSJ). Automatic calculation was then taken before weight percent oxides were reported.

In addition, mineralogical phases present in the samples were identified using X-Ray Diffractometer (XRD) (model Bruker B8 advance) based at the Geology Department, Chulalongkorn University. Selective samples, which were distinguished from XRF results, were analyzed under operational condition of 40 kV/ 30 mA with 2° /min scanning rate. Peak searching was compared with database provided by the manufacturing before phase identification was individually designed.

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Total multi-elemental analyses of heavy metals and toxic elements containing in tailing samples were carried out following the EPA 3052 method. Samples, about 0.5 mg of sample powder, were digested by a mixture of 3 ml concentrated hydrofluoric acid and 9 ml concentrated nitric acid before making volume to 50 ml. All samples were carefully prepared to make sure that sample powders were totally digested without any solid left from the reaction. Subsequently, the solution samples were analyzed using Inductively Coupled Plasma Optical-Emission Spectroscope (ICP-OES), Varian model vista-mpx CCD, at the NEC-EHWM. Bank and standard solutions (prepared from AGV-2, BHVO-1 and MAG-1) were also systemically used per each sample lot of ten. Finally, concentrations of elements were computerized and reported.

Subsequently, leaching experiment was performed under SW 486, EPA method 1312 (Synthetic Precipitation Leaching Procedure, SPLP) with some modification of the methods. The experiment was started by preparing the mixed acid solution. Sulfuric (H_2SO_4) and nitric (HNO_3) acids were mixed under a ratio of 60:40 by weight. The mixed acid was added to the deionized water until reach pH 2 and 4, respectively. Then 1 g of sample was added into the prepared acid solutions and pure distilled water (pH $\approx$ 6.5) before shaking for 18 hrs under 30 rpm. Leachates of all pH conditions were analyzed for the potentially leached heavy metals by ICP-OES.

For acid formation potential, paste pH is an instant measurement of acidity or alkalinity of the sample but it cannot provide quantitative result. The test was prepared by 1 g powder sample mixed with 20 ml deionized water, and then tested with a pH probe after one hour. The paste pH results would initially indicate potential of acid mine drainage (Sobek et al., 1978; Noll et al., 1988; Weber, 2006); however, quantitative static methods must be carried out.

The most common static test has been known as acid-base accounting (ABA) (Ferguson and Erickson, 1988). This test was originally designed to evaluate the acid producing capability of coal-mine wastes. It is now applied to evaluate both coal-mine waste and metal-mine wastes (White et al., 1999). Recently, net-acid producing potential

(NAPP) has become the most usual measurement for acid-base accounting of the mine-waste samples; consequently, the samples will be classified as either potentially acid producing or non-acid producing. NAPP comprises two components, i.e., maximum potential acidity (MPA) and acid neutralization capacity (ANC) (Weber et al., 2004). NAPP is determined by subtracting the estimated ANC from MPA (Hughes et al., 2007; Coastech Research Inc., 1989).

The MPA is calculated from the content of total or reactive sulfur in the sample. In this study, total-S was determined using Carbon & Sulfur analyzer (Leco CS-400 analyzer). Although this analyzer may yield higher sulfur content involved by carbon component leading to overestimation of MPA value that would be more appropriate for environmental assessment. Moreover, Lapakko (2002) suggested that combustion furnace techniques, including LECO furnace, with subsequent quantification of the sulfur dioxide evolved, are capable for accurate determination of total sulfur and, therefore, the maximum acid production potential. It assumes that all sulfur is present as sulfide. The MPA is determined from the total sulfur multiplied by 30.6 which is amount of H_2SO_4 (kg/t) produced through pyrotite and pyrite oxidation (Parker and Robertson, 1999).

The ANC is commonly determined by the Sobek method. Known amount of standardized hydrochloric acid (HCl) is added to an accurately weighed sample, before boiling of the solution samples time is taken to complete reaction. Then, the solution is back-titrating with standardized sodium hydroxide (NaOH) to determine the amount of the remaining HCl. The amount of acid consumed by the reaction within the sample is then calculated and reported as kg H_2SO_4/t (Environment Australia 1997; Sobek et al., 1978).

Net Acid Generation (NAG), additional static test, is always used in association with the NAPP to confirm the classification of acid generating potential. NAG tests are based on the principle that hydrogen peroxide accelerates the oxidation of iron sulfide minerals. The acid consequently produces and dissolves neutralizing materials;

eventually, the net result of the acid production and neutralization can be measured directly from its pH (report as NAGpH) (Miller et al., 1990; White et al., 1999).

4.5 Results and Discussion

4.5.1 Tailing Characterization

Chemical and mineralogical characterizations were carried out using XRF and XRD, respectively, as reported above. XRF results of all tailing samples were grouped based on their depths at 0, 1 and 2 meters. This is to investigate homogeneity of tailings. Subsequently, all analyses are plotted as variation diagrams including SiO_2 versus the other oxides and MgO versus the other oxides in Figures 4.2 and 4.3.

SiO_2 appears to be the main composition of all tailing samples which fall within a wide range of 42–54 % (Figure 4.2). Although there is no significant relation between SiO_2 and other oxides, two groups of tailing can be distinguished. The first group contains lower SiO_2 (about 46-42 %) and Fe_2O_3 (8-16 %) contents (Group A) whereas the second one have higher SiO_2 (about 47-54 %) with higher Fe_2O_3 (> 16%) (Group B). However, both groups spread over all depths of sampling. This heterogeneous distribution pattern may be caused by the mixture of tailings and water, which appears to be different in raw material and chemical additives from time to time, is feeding into the pipeline situated around the edge of pond and systematically disposing through the middle of pond.

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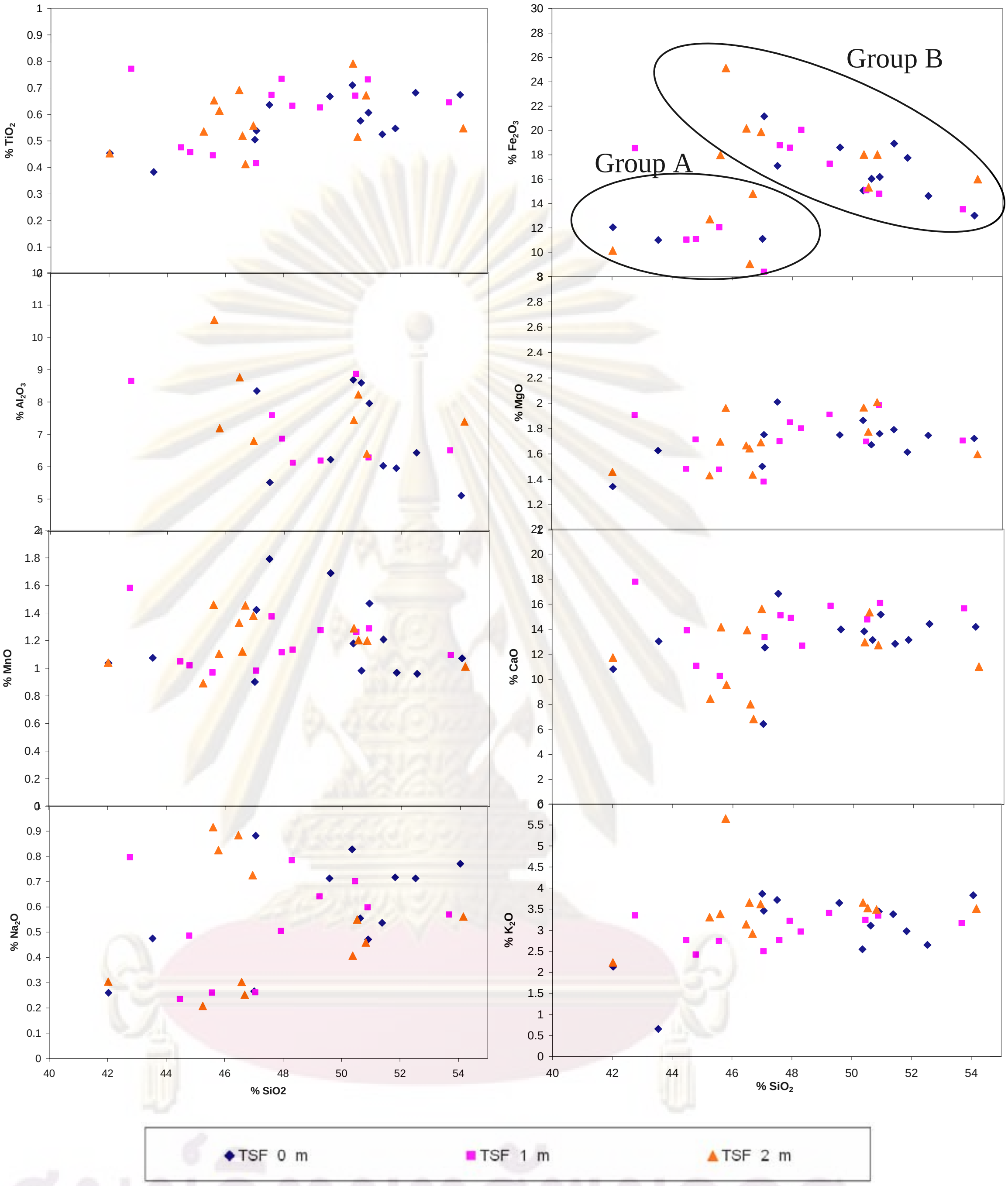

Figure 4.2 Variation diagrams, plotting of SiO_2 against other major and minor compositions showing two groups with higher SiO_2 and Fe_2O_3 group (Group A) and lower SiO_2 and Fe_2O_3 group (Group B).

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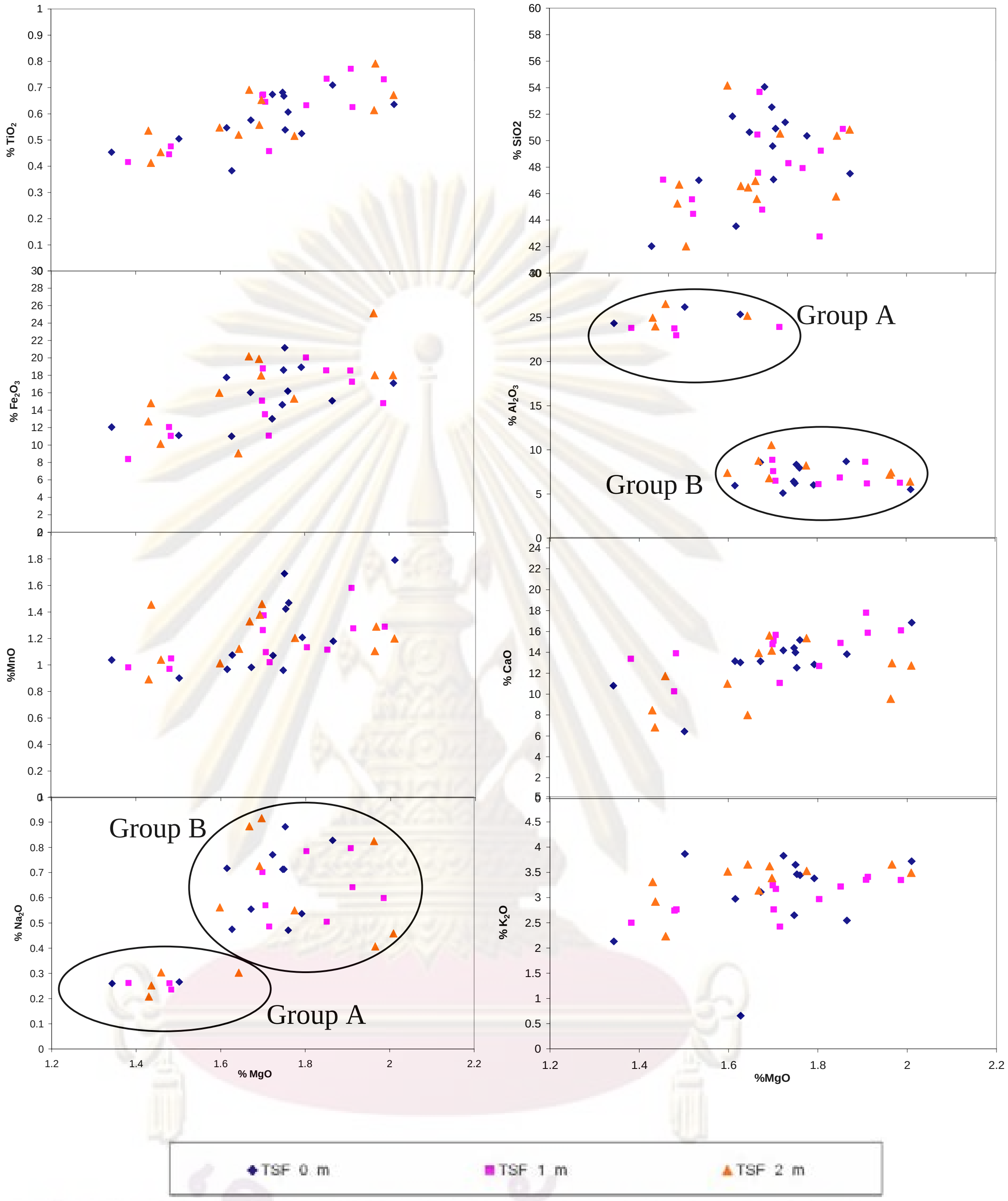

Figure 4.3 Variation diagrams, plotting of MgO against other oxides showing clearly two groups (A and B) of tailings with different Al_2O_3 and Na_2O contents.

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Although MgO contents range in a narrow range of 1.3–2 %, relationship of MgO against CaO, Fe_2O_3 and TiO_2 can be observed as positive relation which increasing MgO appears to have higher amounts of CaO, Fe_2O_3 and TiO_2, slightly (Figure 4.3). In addition, plots of Na_2O and Al_2O_3 against MgO also show two different groups. The first group (Group A) contains lower MgO (1.3-1.6%) with lower Na_2O (0.2 - 0.4 %) and higher Al_2O_3 (>20%) whereas the second group (Group B) contain higher MgO (1.6-2%) with higher Na_2O (0.5 – 0.9 %) and lower Al_2O_3 (about 5-10%). These two separated groups are concordant to those separated by SiO_2 and Fe_2O_3 as reported earlier. Consequently, Group A tailings appear to have lower SiO_2 and Fe_2O_3 whereas Group B tailings contain higher SiO_2 and Fe_2O_3.

However, both groups and their elemental relation may be occurred by different causes. MgO, Fe_2O_3 and TiO_2 seem likely due to original composition of the ore-bearing rock whereas Al_2O_3 may be caused by original composition or degree of alteration (clay content). On the other hand, Na_2O appears to have significant difference that may be due to amount of chemical additives (NaCN) added into the process. For CaO, its contents may be involved by initial composition as well as free lime additive added during the mineral process.

Hence, the representatives of both groups were selected for identification of mineral phases using powder X-Ray Diffractometer (XRD). Representative XRD patterns are shown in Figure 4.4.

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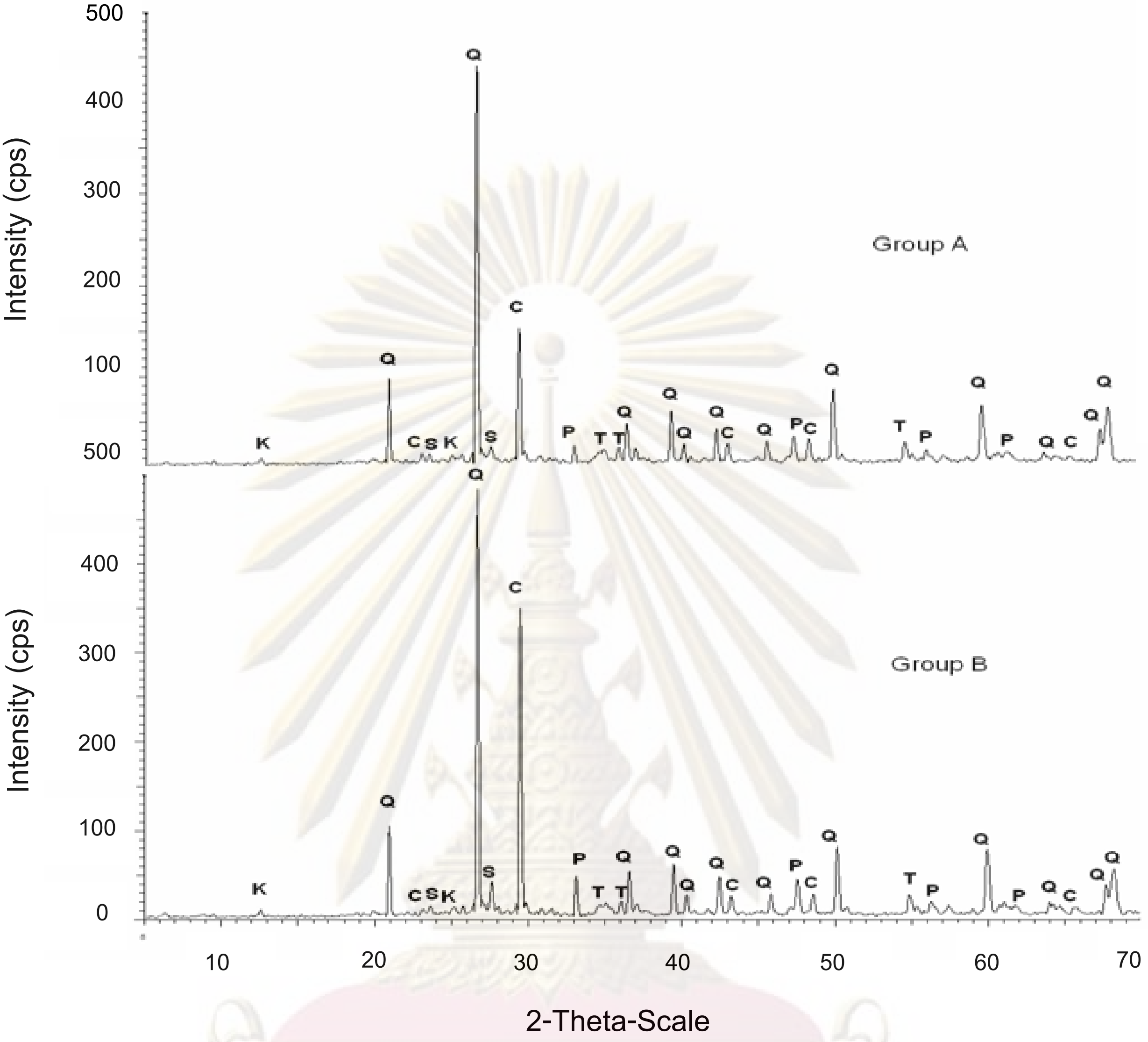

Figure 4.4 Representative XRD patterns of tailings Groups A and B from Akara gold mine. Q = quartz; S = alkali feldspar; C = calcite; P = pyrite; K = kaolinite; T = talc.

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As a result, there is not clearly different mineral phases present in both tailing groups. Quartz (SiO_2) appears to be the main component along with other minerals such as alkali feldspar, calcite ($Ca(CO_3)$), pyrite (FeS_2), kaolinite ($Al_2Si_2O_5(OH)_4$) and talc ($Mg_3(Si_4O_{10})(OH)_{2/3}MgO\ 4SiO_2\ H_2O$). Apart from calcite, the others seem to be original phases contained in the ore-bearing rocks which have been undergone alteration process (Akara Mine, 1999). They should remain from the mineral processing and have been left in the tailings. Calcite can also be part of the ore materials; however, it would be notify that quick lime is also added into the process. Although there is some difference of Na_2O content that is assumed to be effected by additives, no significant crystalline phase of Na or even CN present in the XRD pattern. Variations in content of the described minerals may lead to different chemical composition; besides, chemical additive may also modify the tailing compositions.

4.5.2 Total Heavy Metals

Total heavy metals contained in tailings analyzed by ICP-OES are summarized in Table 4.1. The result is compared to metal concentrations in ore material and the undisturbed topsoil as well as the Thailand Soil Quality Standards for Habitat and Agriculture (PCD, 2004).

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Table 4.1 Summary of total heavy metal concentrations in mine tailing samples from Akara Mine based on EPA 3052 method and ICP-OES analysis.

Depth		Elements (mg/kg)							
		Cd	Co	Cr	Cu	Pb	Mn	Ni	Zn
0 m	average	<0.05	9.41	15.11	88.63	41.86	1789.92	6.37	132.52
	Min-max	<0.05	1.8-15	12-18	77-114	22.58-92.31	1517-2782	4.6-8.2	79-231
(N=13)	standard deviation	<0.05	3.80	1.44	10.90	19.03	323.45	1.16	43.94
1 m	average	<0.05	11.62	14.92	88.73	35.35	2303.08	7.43	127.23
	Min-max	<0.05	6.5-17	14-16	81-98	22-46	1698-2731	5.4-10	89-177.5
(N=12)	standard deviation	<0.05	3.25	0.73	5.17	8.02	386.28	1.40	29.18
2 m	average	<0.05	10.49	16.16	97.58	51.09	2176.33	7.18	171.56
	min-max	<0.05	5.2-20	14-18	81-121	40-61	1788-2900	5.7-11	112-228
(N=12)	standard deviation	<0.05	3.87	1.18	13.09	8.37	328.15	1.53	34.61
all sample	average	<0.05	10.47	15.39	91.56	42.74	2081.68	6.98	143.46
	min-max	<0.05	1.75-19.62	12.14-18.09	77-121	21.9-92.3	1517-2900	4.58-10.99	78.6-230.9
(N=37)	standard deviation	<0.05	3.67	1.26	10.86	14.27	404.27	1.41	40.79
Low Grade Ore[a]		0.49	28.6	4.707	30.55	6.97	1456.18	2.17	47.62
High Grad Ore[a]		1.51	31.2	2.32	91.8	132.95	874.62	1.27	156.05
Undisturbed Topsoil			1.8027	7.584	54.946	0.8925	797.1	3.5993	63.36
Thailand Soil Quality Standards for Habitat and Agriculture (mg/kg) [b]		37	-	300	-	400	1800	1600	-

[a] Low grade ore (0.8-1 g/tonne gold); High grade ore (> 1 g/tonne gold) (Akara 1999)

[b] PCD, 2004.

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Concentrations of most elements varied slightly with depth. However, in general, concentrations of most elements are more or less falling within ranges of low grade and grade ores; only Cr and Ni appear to have higher than those in initial ore materials. In addition, all elements in tailings are higher than those contain in undisturbed topsoils. Regarding to Cr and Ni, their contents in topsoil are also higher than ore materials. Both elements may tend to be left in situ from both natural weathering and artificial extraction processes. However, comparing to the Thailand Soil Quality Standards for Habitat and Agriculture, they were found that most heavy metals are still within the standards. Only one exception, Mn concentrations exceed the standard (1800 mg/kg). This result is consistent with the water monitoring report by the Department of Primary Industries and Mines, Ministry of Industry. The report provides data of slightly increasing Mn concentration in deep and shallow well water in the vicinity of mine during the recent years. They reported the average concentrations of Mn well water in years 2005 to 2007 are 1.74 ± 5.38, 0.39 ± 0.55 and 0.77 ± 1.03 mg/l, respectively. The Thailand water quality standard limits the Mn concentration at 0.5 mg/l for both surface water and ground water (PCD 2008). However, excess of Mn in water can not be directly attributed to the human activities (mining). On the other hand, it may be due to the background Mn concentration of the area. In addition, though the Mn concentrations in tailing exceed the standard for Habitat and Agriculture, it is still, however, acceptable for the standard for other purposes that limits concentration of Mn at 3200 mg/kg. Moreover, Mn has no phytotoxicity; therefore, plant and grass can grow in this Mn-contained area.

4.5.3 Leaching Test

All of the tailing samples were undergone the leaching experiment at three different pH conditions (2, 4 and 6.5) to find out the metal that may leach to the environment when they expose to the acid mine drainage. Experimental procedure was described in the earlier part. Results of leachable metal of the tailing samples were plotted in comparison with the Thailand industrial effluent standard and the Thailand surface water quality standard for agricultural as presented in Figure 4.5.

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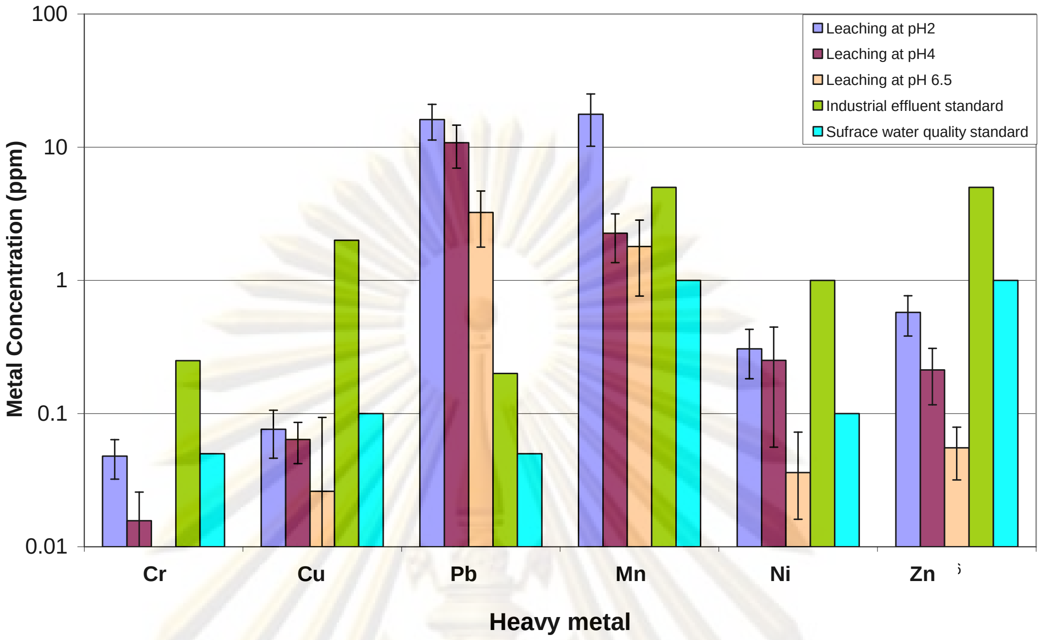

Figure 4.5 The Comparison between the metal concentrations analyzed from leachates of tailings at different pH conditions in comparison with the Thailand effluent standard and the Thailand surface water quality standard for agricultural.

Leached elements at different pH conditions (2, 4 and 6.5) are compared with the Thailand industrial effluence standard and the Thailand surface water quality standard. They reveal that Cr, Cu and Zn at all pH conditions are below both the industrial effluent standard and the surface water quality standard. On the other hand, Pb can be released over both standards at all experimental pH. While Mn exceed the surface water quality at all experiment pH but exceed the industrial effluent standard only at pH 2. For Ni it exceeds the surface water standard at pH 2 and 4 but still falls below the industrial effluent standard at all experimental pH. Concentrations of all elements in leachates decrease from pH 2 to pH 4 and 6.5, respectively. It can be concluded that pH play the most important role in the leaching process. This agrees well with the previous studies reported that metals have a tendency to leach more at extreme acidic pH conditions (Van der Sloot et al., 1997; and Jang et al., 2002).

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Among the concerned metals, Pb seem to be the most problematic metal because they can be released exceeding the standards at all pH even the neutral condition and Mn can be released exceeding the surface water quality at all experiment pH. There is no relationship between the total metal concentration in a solid waste and its ability of leaching; therefore, leaching tests are often used to determine the potential of solid waste that may contaminate ground water (Dungun, 2009). Mn, which highly contains in tailings and appears to leach out easily, should have tendency for contamination. However, its toxicity is not high as the other metals. On the other hand, Pb with more harmfulness has quite low concentration in the initial tailings but high potential to leach. Therefore, great care, however, must be taken during the whole period of operation as well as further monitoring plan.

4.5.4 Acid-Forming Potential

The results of paste pH from the Akara Gold Mine tailing samples are shown in Table 4.2. All samples are alkaline with pH values ranging from 7.64 to 8.96 with an average of 7.97. Hughes et al. (2007) stated that paste pH is an indicator of the net-acid producing potential (NAPP). Strongly negative NAPP samples usually yield neutral paste pH whereas strongly positive NAPP samples result acid paste pH. For instant, all tailing samples with paste pH of about 8 would imply strongly negative NAPP. The paste pH test may indicate the number of free hydrogen ions in the prepared sample but the pyrite oxidation reactions are time dependent. Thus the paste pH results provide little indication of the propensity of a sample to produce acid mine drainage (Kania, 2008).

The results of ABA and NAG tests are summarized in Table 4.2. For ABA test, tailings generally have moderate sulfur content, ranging from 2.32 to 4.48% S with an average of 3.15%. Thus, maximum potential acidity (MPA) value were calculated within the range of 70.99 to 137.08 kg H_2SO_4/t with an average of 95.51 kg H_2SO_4/t. For Acid neutralization capacity (ANC), its calculated values range from 98 to 172 kg H_2SO_4/t with an average of 136.33 kg H_2SO_4/t. In conclusion, ANC values are greater than MPA values in all samples, which indicate sufficient neutralizer available to react acid produced from sulfide oxidizing process. Consequently, negative NAPP, ranging from -9.912 to -87.01 kg H_2SO_4/t with an average of -39.82 kg H_2SO_4/t are yielded from calculation (Table 4.2). For NAG test, all NAGpH were measured and yield a range of 7.2-9.4 with an average of 8.19 which clearly indicates more neutralizer against acidity of the tailings.

Three categories of acid forming potential can be classified on the basis of NAPP and NAG results: 1) Potentially Acid Forming (PAF) when samples have positive NAPP and NAGpH $\leq$ 4.5; 2) Non-Acid Forming (NAF) when sample yield negative NAPP and NAGpH > 4.5; 3) Uncertain (UC) when they still show any conflict results between NAG and NAPP tests (EGI, 2005). Graphically, NAGpH and NAPP results of all tailing samples were plotted, based on the suggested criteria, for classification of acid forming potential (Figure 4.6). It shows clearly that all samples fall within the NAF quadrant. In addition, samples from different depths (e.g., at o, 1, and 2 m) show insignificantly different results.

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Table 4.2 Summary of acid potential tests for Akara Gold Mine tailings including, paste pH, acid-neutralization capacity (ANC), maximum potential acidity (MPA), net-acid producing potential (NAPP), net-acid generation (NAG), NAGpH, and total sulfur contents.

Depth (m)		Paste pH	Total S (%)	MPA (kg H2SO4/t)	ANC (kg H2SO4/t)	NAPP (kg H2SO4/t)	NAGpH	Class
0 m	average	7.98	3.31	101.34	137.15	-35.82	7.92	
	min-max	7.68 - 8.2	2.41 - 4.48	73.75 - 137.1	98 - 172	-75.92 - -9.91	7.2 - 9.1	NAF
(N=13)	Standard deviation	0.12	0.62	19.03	20.71	20.16	0.64	
1 m	average	7.94	3.12	95.55	135.67	-40.12	8.32	
	min-max	7.6 - 8.1	2.6 - 4.04	79.56 - 123.6	109 - 169	-85.07 - -12.3	7.6 - 9.3	NAF
(N=12)	Standard deviation	0.13	0.43	13.25	17.82	21.80	0.46	
2 m	average	8.01	3.08	94.20	136.25	-42.05	8.37	
	min-max	7.74 - 8.96	2.32 - 3.76	70.99 - 115.1	121 - 158	-87.01 - -27.94	7.3 - 9.4	NAF
(N=12)	Standard deviation	0.31	0.41	12.61	10.82	16.34	0.68	
all	average	7.98	3.17	97.15	136.38	-39.24	8.19	
sample	min-max	7.64 - 8.96	2.3-4.8	70.9 - 137	98 -172	-87 - -9.91	7.2 - 9.4	NAF
(N=37)	Standard deviation	0.20	0.50	15.27	16.62	19.22	0.62	

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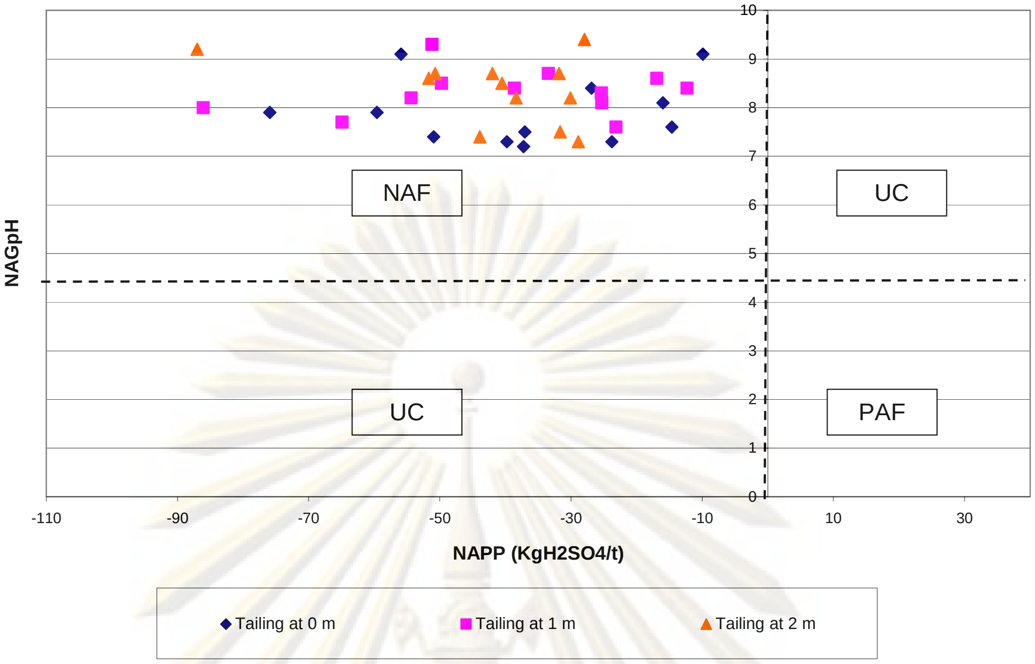

Figure 4.6 AMD classifications of the Akara Gold Mine tailings with three different depths, showing the comparison between NAPP and NAGpH criteria. (diagram suggested by EGI 2005).

Both MPA and ANC of these tailings are rather high compared to those of all waste rock types (as reported in Chapter 6). Gold bearing rocks in this mine usually contain dominant pyrite with subordinates of sphalertite and chalcopyrite (Akara Mine, 1999); these sulfides, particularly pyrite, are still left in the tailing as identified by XRD after gold extraction process. Consequently, they potentially yield high MPA contents that are in turn generating H_2SO_4 by oxidation reaction (Lei and Watkins, 2005). Although the MPA values in tailings are mostly high, their ANC are even more sufficient to neutralize the generated acid. For all prospect areas, the total sulfur increases progressively from the lowest level of oxide zone to the highest level of primary zone; in addition, the primary zone also contain high carbonate and calcium assays, corresponding to the occurrence of ore carbonates (Akara Mine, 1999). Moreover, calcium and sodium may be left from the additive (quicklime and NaCN) that are routinely added into the process. Therefore, these may contribute sufficient neutralization for these tailings.

4.6 Conclusions and Recommendations

Chemical and mineralogical composition show insignificant difference among the tailing samples although there is some chemical variation; besides, relation between the tailing samples and their distributions both in horizontal and vertical profiles are not clearly recognized. These may be a result of systematic disposal procedure and heterogeneous ore material as well as amount of additive consumed in the process from time to time. The other reason is that the water-soaked tailings have been feeding into the pipeline situated around the edge of pond and systematically disposing through the middle of pond.

Most of total heavy metal concentrations contained in the tailing samples are restricted within the Thailand Soil Quality Standards for Habitat and Agriculture except Mn appears to have higher content than the standard (1800 mg/kg). According to the water monitoring data, Mn concentration has been increasing slightly in the past 3 years (data from Polyuth et al., 2005; Polyuth and Wiwat, 2006; 2007). The causes of Mn increase however are not yet indicated from this study. However, leaching experiment indicates that Ni, Mn and Pb have potential to leach in acidic condition and even neutral pH for Pb and Mn. Thus, continuously careful water monitoring is needed although the contamination may be involved by the background of the area.

ABA and NAG tests conclusively prove that all tailing samples are Non-Acid Forming (NAF). Some previous works (Romano et al., 2003; Demer et al., 2007) suggest the utility of non-acid forming tailings as a cover material over other construction, such as reclamation process; however, it is not recommended for this case. Although the tailings in this study are classified as “NAF” and their pH were adjusted to maintain around 9-10 in cyanidation process before releasing to the TSF, high leaching rate of heavy metals (e.g., Pb, Mn) even at neutral pH are reported herein. It is not advisable to

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use these tailings as cover material to protect reactions from oxygen and water. In addition, tailings, themselves, need to be controlled avoiding the interaction with oxygen and water.

From a number of existing sites where tailings had been placed in lakes, for example in northern Canada, it can indicate that long-term submergence of potential acid-forming material is probably the most effective means of AMD control (Ritcey, 2005). However, the Akara Gold Mine is situated in the tropical zone; it is difficult to supply a large amount of water in order to submerge tailing facility for a long term period after the mine closure. Thus, the submerging of tailing is unsuitable strategy in this case. Alternatively, it is advisable to use clay cover as barrier of air and water infiltrations. Undisturbed top soil is recommended to be used for clay cover; this aims to be a fertilizer source for re-vegetation. Native grass species should be sown to stabilize the top soil surface and to protect soil erosion. In addition, recommended practices/managements for the Akara Gold Mine are following:

- Monitoring program of surface, deep well and shallow well waters should be performed every 6 months. However, in case of increasing concentrations of any metal (especially Mn and Pb), more frequent sampling (every 3 months) would be considered.
- pH plays a very important role for leaching of metals. Thus, pH is strongly recommended to be measured at all monitoring programs.
- Diversion of surface water flowing away from tailing storage should be constructed to avoid the erosion. Water diversion might be small ditch system lined around the tailing storage facility. These ditches should also be permanently topped by concrete or blasted rock to avoid collapse and erosion of the ditch.

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CHAPTER V

GEOCHEMICAL CHARACTERISTICS OF WASTE ROCKS FROM THE AKARA GOLD MINE, PHICHIT PROVINCE, THAILAND

Abstract

Geochemistry of the waste rocks collected from the C-H pit, Akara Gold mine, Pichit Province, Thailand was investigated to evaluate their potential of heavy metal impacting the environment. Six types of waste rocks have been classified, based on field experience of mining geologist, as volcanic clastic, porphyritic andesite, andesite, silicified tuff, silicified lapilli tuff and sheared tuff which the last two types are ore contact. The major compositions of waste rocks show that the major SiO_2 concentrations are increasing against decreasing of TiO_2, Fe_2O_3, Al_2O_3 and MgO. The results reveal the geological of waste rock is related to the alteration process of the area and showed the consistency in waste rocks. Different degrees of alteration had modified some rocks locally depending on geological structure likes silicified tuff, shear tuff and silicified lapilli tuff but not for the volcanic clastic, porphyritic andesite and andesite.

Regarding to heavy metals contained in these waste rocks comparing to the general igneous rocks As, Ag, Cd, Co and Mn are found with relatively high in all waste rocks whereas Cu and Pb enrich in silicified tuff and silicified lapilli tuff. The results of the totally metal in waste rock which were digest until no residual is chemically in the same trend to the leachable metals which were digested by only nitric acid. The trend can be plotted and given a relation equation. Therefore, the leachable metal by strongly acid and the leaching metal by the water pH condition are not related. Some of enrichment metals/leachable metals may not leach under water pH condition. Effect of AMD leaching were simulated by SW 486 – SPLP by modify at different pH conditions (2, 4 and 6.5). The results show the strongly relationship between pH and leached metal. The concentration of leached metals is increasing with decreasing pH. The leached test showed most metals such as Cd, Cu, Mn, Pb, Ni, Zn are leached exceed the Thailand

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standard of surface water for agricultural at pH 2 whereas there are only Pb and Mn are exceed the standard of industrial effluent. Pb and Mn become the most interested in this study together with Cu because these three metals can be leach since the pH is at neutral. The data suggest that the leaching formation should be strongly controlled and monitored even the AMD does not occurring yet. This information should be taken into consideration of the further environmental monitoring.

5.1 Introduction

Gold is rarely found on the Earth's surface; only trace amounts can be determinable in level of part per million (ppm) of host rock even in the economic deposit. Consequently, enormous waste materials are usually left behind the mining process. These materials may in turn have impacted the environment for long periods, particularly when no environmental impact reduction plan is implemented properly. As reported in many cases around the world, the effect of abandoned mines (including gold mine) may release acidic water/leachate, so called Acid Mine Drainage or AMD, to the environment. Moreover, AMD with its low pH also dissolves heavy metals from surrounding soils and rocks that leads to contamination of natural water supply and consequently threaten ecosystem in wide space (Paradis et al. 2000; Benzaazoua et at. 2008; Peppas et al. 2000).

The Akara gold mine is the first and biggest goldmine in Thailand and it is still under operation. Although, none of obvious environmental impacts are conclusively evidenced, some concerns have been initiated from both mining operator and some environmental agencies. Most aspects must be taken into account for monitoring and protection plans. Apart from AMD evaluation, investigation of the geochemical characteristics of waste rocks, including heavy metal contents and their potential to leach, would be the first step to develop the best practices of environmental protection. Therefore, this project was planed: (i) to characterize geochemical characteristics of all rock wastes; (ii) to estimate total heavy metal contents in these waste rocks; (iii) to evaluate their potential of heavy metal leaching. Finally, results from the study may be applied along with some AMD assumption to develop the best practices of environmental protection.

Waste rock samples were collected from C-H Pit at Akara Gold Mine (Figure 5.1). They are classified, based on the field investigation of mining geologist, into 6 types including volcanic clastic, porphyritic andesite, andesite, silicified tuff, silicified lapilli tuff (ore contact) and sheared tuff (ore contact). Analyses and experiments were carried out at the laboratory of National Center of Excellence for Environmental and Hazardous Waste Management (NCE-EHWM), Chulalongkorn University. Whole-rock analyses using XRF spectrometer (model PANalytical Axios) were initially carried out for major and minor compositions of the waste rock samples. The quantitative analyses were operated under conditions of 24 kV/100 mA (for Na_2O, MgO, Al_2O_3, SiO_2, P_2O_5, K_2O and CaO) and 60 kV/40 mA (for Fe_2O_3) before calibration with standard references (i.e., AGV-2, BHVO-2, COQ-1, SCo-1, SGR, SDO-1, STM-1, PCC-1 and JG-1) supplied by U.S. Geological Survey (USGS) and Geological Survey of Japan (GSJ). Finally, percent weight oxides were automatically calculated and reported. Results from this part were used for geochemical characterization prior to further experiments.

Subsequently, two standard procedures, EPA3052 and EPA3051, were applied to digest these samples prior to solution analyses using Inductively Coupled Plasma-Optical Emission Spectrometry (ICP-OES) for most heavy metal determinations, except arsenic using Atomic Absorption Spectrophotometer (AAS). Both digestion techniques were used to evaluate the total forms and leaching potential forms of all heavy metals, respectively. In addition, leaching experiments with assumed various AMD conditions were also taken place to evaluate leaching of heavy metals in different acidic levels. The detail of leaching procedure will be reported along with results later. All results are collected and discussed in term of geochemical characteristic and leaching potential that may in turn impact the environment; eventually, guidelines of the monitoring and protection plans will be recommended at the end.

5.2 Geology and Mining

Since 1993, Akara Mine has been operating in the “Chatree Gold Deposit” located on the eastern edge of the undulating Chao Phraya Basin. It is about 280 km north of Bangkok, the capital city of Thailand. Based on regionally geologic setting, this gold deposit is situated along the Loei-Petchabun volcanic belt which is bound by the upper Permian and Triassic parallel zones of ancient island arc containing dominant mafic to intermediate volcanic rocks (andesite and basaltic andesite in particular) in association with marine sedimentary rocks. Gold and silver ores are found in this area as very small nuggets to invisible particles. Their ineralization appears to have been related to epithermal process in shallow to steeply dipping structures. Gold-bearing veins are crucially characterized by breccia and stock-work veinlets which mostly consist of quartz and calcite, so-called quartz-carbonate veins (Hemmanee and Poot-heng, 2006). Ore reserves have been estimated for over 10 years of mining activity. However, more prospects have been discovered nearby the current mine and operation has been started in some part. Therefore, the mine may stay in operation longer than expected. Conventional open pit is processing before passing through the gold extraction process. Consequently, mining wastes particularly ore-barren rocks have been disposed with enormous amount. Moreover, this mine site is located in the tropical climate and surrounded by mostly cultivated land. Main crops are maize, sugar cane, peanut and seasonal fruits. The environmental impact must be concerned with great care.

Main rock types at C-H pit are petrologically classified as andersitic lithic breccia, plagioclase-pyroxene-phyric andesite and cystal-lithic breccia unit which these rock units are overlain by laterite and alluvium (Tangwattananukul, 2006). However, waste rock classification under this study follows the mining geologists’ routine identification because they will be easily for communication during the mine operation and practical environmental protection. On this basis, they are grouped as volcanic clastic, porphyritic andesite, andesite, silicified tuff, silicified lapilli tuff and sheared tuff which the last coupled types are ore contact which sample locations of each waste rock are shown in Figure 5.1.

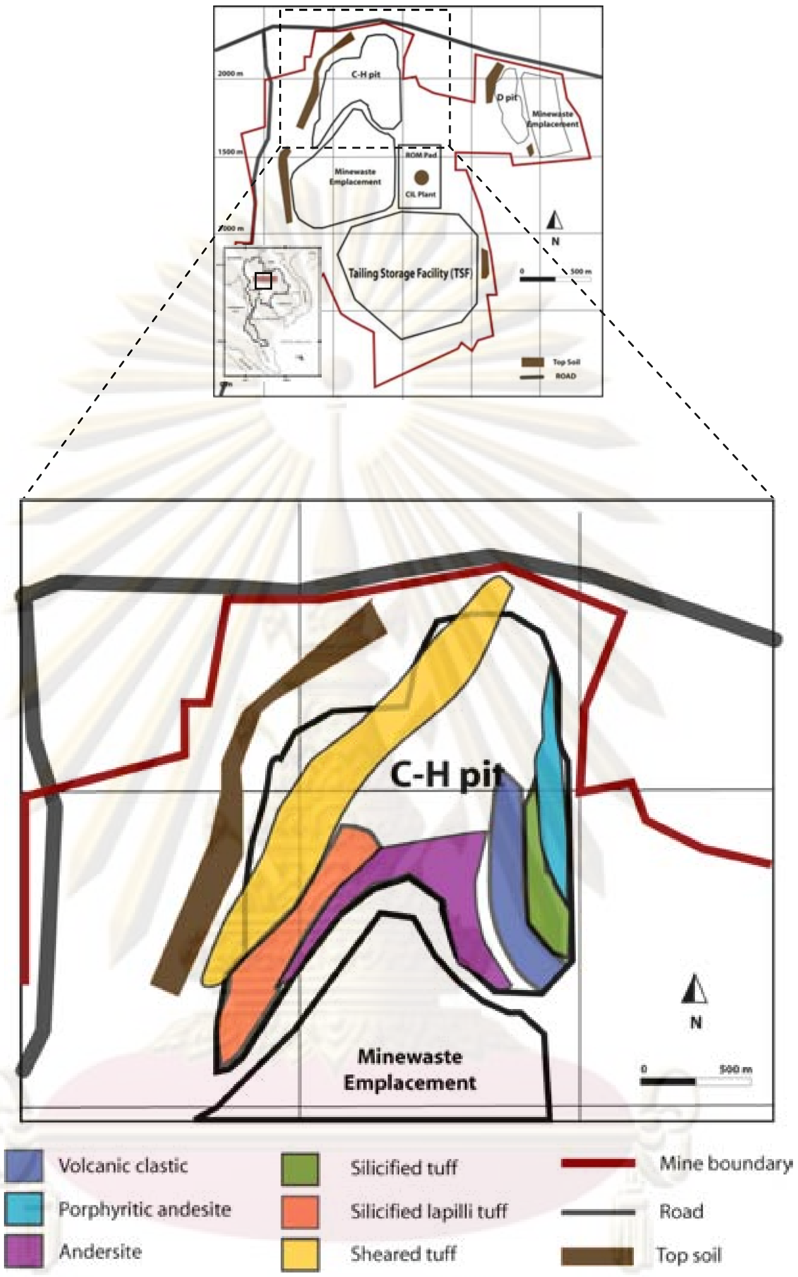

Figure 5.1 Sketched map showing location of the Akara Gold Mine and sample localities in C-H mining pit (modified after URS 1999). 1 = volcanic clastic; 2 = porphyritic andesite; 3 = andesite; 4 = silicified tuff; 5 = silicified lapilli tuff; 6 = sheared tuff.

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5.3 Geochemical Characterization

Major and trace analyses were carried out using XRF for rock powders and ICP-OES for EPA3052 solutions, respectively; representative analyses are summarized in Table 5.1. In addition, graphical presentation plotting of SiO_2 against other oxides of all samples is shown in Figure 5.2. Based on SiO_2 contents, all waste rock types can be categorized into 3 main groups including high-, medium- and low-silica contents. Silicified tuff contains extremely high SiO_2 range (71-76%) whereas porphyritic andesite, andesite, silicified lapilli tuff and sheared tuff fall within a medium wide rage of 50-65% SiO_2. On the other hand, volcanic clastic appears to have the lowest SiO_2 range of about 45-50% (Table 5.1 and Figure 5.2).

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Table 5.1 Representative chemical analyses of waste rock samples form Akara gold mine.

	Volcanic clastic			Porphyritic andesite			Andesite			Silicified tuff			Silicified lapilli tuff			Sheared tuff		
Sample	No. 1	No. 2	No. 3	No. 1	No. 2	No. 3	No. 1	No. 2	No. 3	No. 1	No. 2	No. 3	No. 1	No. 2	No. 3	No. 1	No. 2	No. 3
SiO_2	45.94	47.23	47.00	62.61	62.72	61.11	66.20	62.23	59.95	75.81	73.41	71.41	54.88	51.06	52.92	57.3	51.44	57.02
TiO_2	0.68	0.71	0.60	0.58	0.57	0.60	0.41	0.59	0.57	0.17	0.10	0.11	0.56	0.65	0.69	0.40	0.42	0.63
Al_2O_3	14.58	16.35	12.63	13.67	12.79	13.76	13.07	11.54	11.39	10.86	10.18	10.46	14.79	16.32	13.85	11.85	11.44	12.96
Fe_2O_3	7.78	7.97	7.96	5.26	5.13	5.25	2.92	5.35	5.25	0.60	0.74	0.98	5.03	5.69	7.74	4.53	5.13	6.10
MnO	0.33	0.33	0.35	0.35	0.33	0.35	0.18	0.39	0.39	0.31	0.26	0.35	0.43	0.46	0.55	0.36	0.31	0.54
MgO	4.40	6.16	4.55	3.04	2.21	2.78	1.99	1.77	1.72	1.06	1.08	1.26	3.77	4.86	4.09	1.78	2.23	2.79
CaO	9.93	7.07	11.55	3.07	3.51	3.72	6.98	8.07	8.20	7.16	8.10	6.71	3.53	4.28	4.11	11.36	13.41	10.70
K_2O	6.54	6.85	4.61	7.90	7.46	7.53	1.37	3.82	4.76	0.46	0.43	0.53	10.19	10.40	10.77	0.13	0.17	0.24
Na_2O	0.91	1.07	0.71	1.11	1.31	1.72	1.31	0.83	0.85	0.78	0.67	0.73	1.85	1.97	0.79	0.64	0.64	1.00
P_2O_5	0.21	0.23	0.17	0.18	0.17	0.20	0.15	0.20	0.20	0.10	0.11	0.12	0.15	0.19	0.20	0.14	0.14	0.16
LOI	8.54	6.62	9.45	2.52	3.01	3.12	5.52	5.55	6.12	3.24	4.15	6.67	4.35	4.22	4.36	11.12	13.94	9.02
Total	99.84	100.59	99.58	100.29	99.21	100.14	100.10	100.34	99.40	100.55	99.23	99.33	99.53	100.10	100.07	99.61	99.27	101.16
Trace Composition (ppm)																		
Ba	298.38	508.04	104.34	49.09	40.79	69.12	999.27	825.13	874.89	673.95	1052.80	1032.00	1353.20	1644.30	1415.70	28.96	48.77	1223.10
Co	26.64	30.44	32.16	212.17	227.74	22.82	55.00	68.08	64.15	51.54	42.70	33.86	49.17	68.07	81.44	47.60	45.60	42.53
Cr	9.91	5.77	12.29	3.42	3.01	8.84	10.58	12.75	11.20	2.37	5.61	2.90	7.70	8.35	7.73	12.93	13.07	12.83
Cu	116.43	67.55	112.66	9.90	10.78	111.01	51.05	46.58	55.24	216.78	138.37	90.91	81.33	141.56	142.47	97.37	90.43	145.14
Ga	14.73	14.25	16.26	4.90	3.83	12.36	11.26	11.56	11.63	13.38	13.23	13.36	12.45	12.63	12.23	10.25	12.16	11.19
Nb	4.79	3.78	4.60	0.26	0.21	1.10	4.01	4.13	4.04	5.75	5.89	6.11	3.98	4.44	3.56	4.05	4.44	4.16
Ni	2.02	1.43	2.40	1.00	0.74	1.61	1.46	1.54	1.53	2.73	3.00	2.96	2.54	2.58	2.27	2.58	3.31	2.70
Pb	3.30	2.93	3.24	5.63	6.40	2.38	12.21	5.24	11.25	14.55	3.91	22.04	4.43	5.73	28.64	2.67	5.51	2.00
Sc	34.49	31.30	65.00	3.33	3.47	35.59	37.57	40.82	45.33	43.77	51.47	41.48	39.59	58.21	51.45	46.83	29.23	44.39
Sr	60.93	145.87	49.19	21.10	17.58	34.55	68.98	74.01	65.32	108.13	102.28	118.63	46.64	114.85	109.64	158.23	181.93	152.41
U	6.57	6.98	8.07	4.86	4.86	8.10	4.25	4.71	4.76	6.30	6.95	6.47	5.55	6.19	6.23	5.90	6.23	5.58
V	168.41	103.98	143.41	17.29	14.07	93.20	98.37	100.38	100.70	130.77	152.33	124.92	149.50	151.36	125.48	89.78	88.15	93.46
Y	11.18	9.51	13.60	1.30	1.47	13.48	14.13	11.92	14.95	19.04	19.14	19.83	11.69	12.35	10.78	6.73	8.02	8.44
Zn	25.50	27.37	32.94	88.42	98.79	29.80	20.25	22.20	22.85	55.93	37.14	39.90	39.85	32.42	101.43	21.17	27.04	26.85
Zr	31.83	19.02	22.41	362.94	305.19	269.77	33.10	38.21	34.55	90.41	88.42	84.60	62.48	52.05	61.64	32.41	34.68	36.59
La	4.97	5.21	8.74	8.20	6.90	6.47	9.29	8.58	6.98	16.85	11.85	15.07	6.74	7.09	7.73	6.44	7.11	6.65
Ce	12.41	12.13	17.45	40.29	33.73	30.80	17.17	18.81	15.45	33.89	31.42	31.11	20.31	14.28	22.49	22.80	22.14	24.99
Nd	8.70	7.04	9.04	1.60	1.46	3.09	8.17	8.45	8.17	11.81	11.92	12.21	7.94	8.53	7.34	2.58	3.31	2.70
Sm	3.18	1.75	4.43	3.14	3.59	2.48	2.84	3.17	2.69	2.74	2.77	4.04	2.06	2.06	2.10	1.77	2.33	3.74
Gd	4.39	4.52	5.14	0.50	0.62	5.15	3.51	3.35	3.68	4.81	5.52	5.22	4.52	4.83	4.05	4.33	4.66	3.91
Yb	0.85	0.64	0.96	0.19	0.15	0.59	0.94	0.80	0.86	1.04	1.05	0.95	0.80	0.92	0.78	0.69	0.73	0.70

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In general, TiO_2, Fe_2O_3, Al_2O_3 and MgO have decreasing trend against increasing of SiO_2. Although the other elements display fluctuate correlation, CaO contents of silicified lapilli tuff and porphyritic andesite appear to lower than the other rocks. Low K_2O contents of sheared tuff and silicified tuff with unusual high SiO_2 of silicified tuff may be caused by alteration process of the area. Discrimination diagram of LeBas et al. (1986), plots of SiO_2 against alkali content (Na_2O + K_2O), is applied to the waste rock because they are geologically related to volcanism of the area. Consequently, it clearly distinguishes these waste rocks with consistency (see Figure 5.3). Volcanic clastic samples fall within the field between tephrite basanite and trachy-basalt that would be due to various composition of fragments embedded in this rock type. Porphyritic andesite and andesite are typically fitted within their fields. However, these rocks are clearly characterized within mafic to intermediate geochemical compositions. On the other hand, silicified lapilli tuff is chemically similar to tephri-phonolite indicating alkali-rich composition. For silicified tuff and sheared tuff, they mostly fall out the common rock compositions that may confirm effect of alteration as mentioned earlier.

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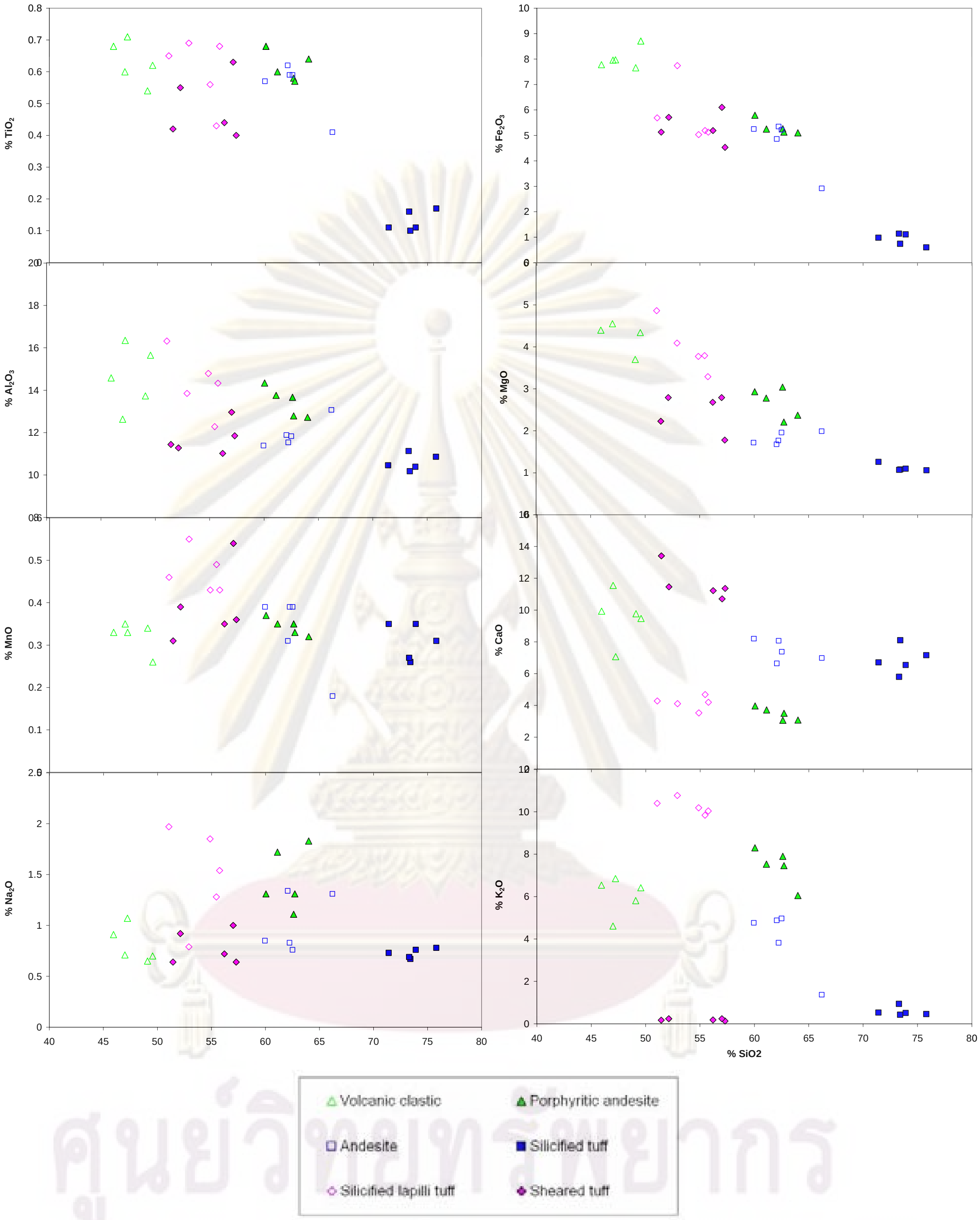

Figure 5.2 Variation diagrams, plotting of SiO_2 against other major oxides of waste rock samples from C-H pit of Akara gold mine showing some correlation of these waste rocks.

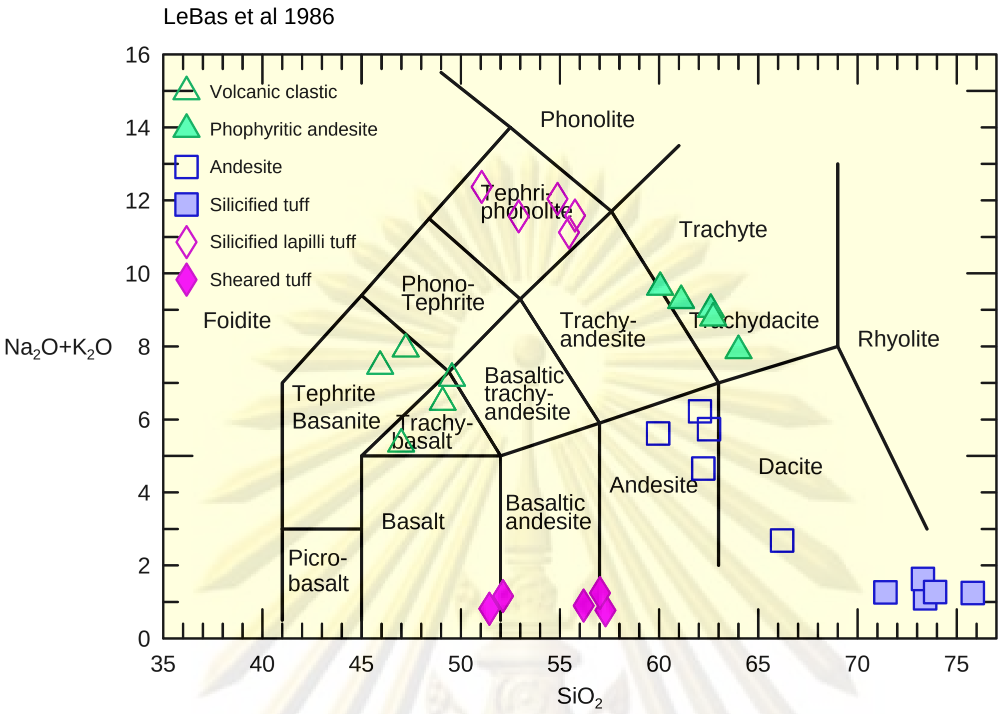

Figure 5.3 Alkali-silica discrimination diagram (LeBas et al. 1986) applied for waste rock samples from C-H pit of Akara gold mine which are related to volcanic vicinity.

Rare earth element (REE) contents of all rock waste samples are normalized against REE of chondrites (Sun and McDonough 1989) and plotted in spidergram (Figure 5.4). All 6 types of rock show quite consistently declination from LREE to HREE with gentle slopes; however, silicified tuff and sheared tuff show zigzag pattern around Ce, Nd and Sm which may be a result of alteration. Moreover, silicified lapilli tuff also slightly presents such zigzag pattern. As a result, similar depletion REE patterns of all waste rocks would indicate the close relations in initial provenance and magma process of all rock types; however, later alteration process which may have taken place during gold mineralization involved their compositions. Different degrees of alteration had modified these rocks locally depending on geological structure and rock types. Silicified tuff and sheared tuff appear to have highly effected whereas silicified lapilli may also be modified with lower degree. On the other hand the other groups, i.e., volcanic clastic, porphyritic andesite and andesite,

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seem to have no influence of the alteration process. In addition, it should also be notified that silicified lapilli tuff and sheared tuff are usually contacted by the mineralization zone.

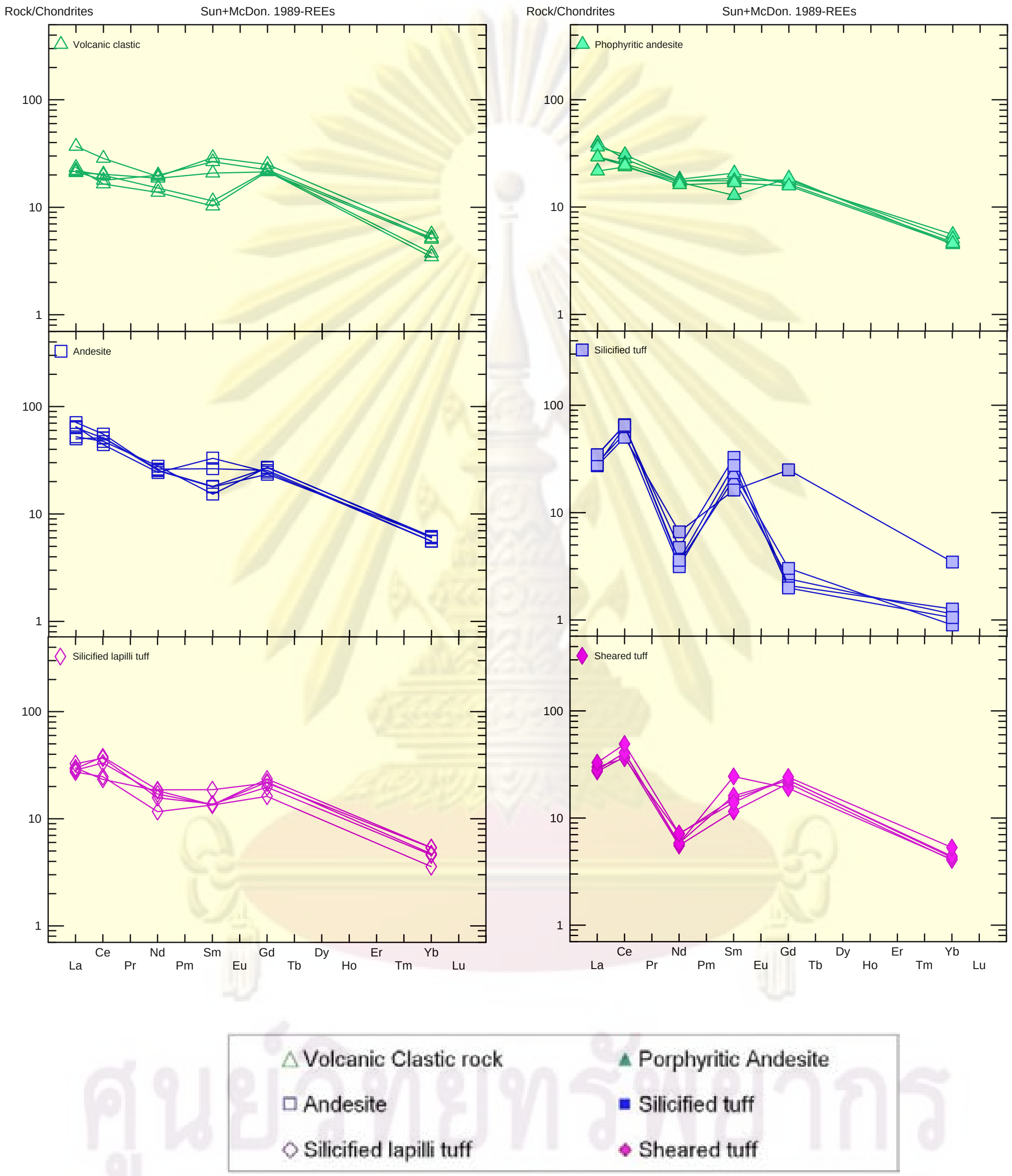

Figure 5.4 Chondrite-normalized REE patterns of waste rock samples (chondrite composition from Sun and McDonough, 1989) indicating similar initial provenance and process of all rock types with different degrees of alteration involved in the later stages.

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5.4 Heavy Metals

Selective heavy metals, possible harmful elements, are picked up to report in this part that will initially give idea of total form of these heavy metals of all waste rocks from the mine prior to consideration of their leaching potential forms. Total heavy metal compositions are statistically summarized in Table 5.2 with graphically plotted in Figure 5.5. Average heavy metal composition of natural igneous rocks reported by Hawkes and Webb (1962) is used for comparison. The average concentrations of arsenic (As), silver (Ag), cadmium (Cd), cobalt (Co), chromium (Cr), copper (Cu), manganese (Mn), nickel (Ni), lead (Pb) and zinc (Zn) in waste rock samples vary in the ranges of 3.59-22.06, 0.28-1.07, 0.16-0.28, 30.54-229.54, 3.25-14.44, 28.96-144.50, 1066.20-1412.20, 0.86-2.93, 0.35-18.96 and 19.94-66.25 mg/kg, respectively. Among these metals, there are seven metals (i.e., As, Ag, Cd, Co, Cu, Mn and Pb) that tend to have higher concentration than those in the natural igneous standards. As, Ag, Cd, Co and Mn are in all rock types are higher than the standards. Except porphyritic andesite and andesite, Cu appears with higher concentration in the other rocks whereas Pb is high only in silicified tuff and silicified lapilli tuff. As in this study exceeds the natural rock more than 5-10 times in andesite and silicified tuff. While the natural igneous contain about 2 mg/kg, As concentrations from this study are up to 22 mg/kg.

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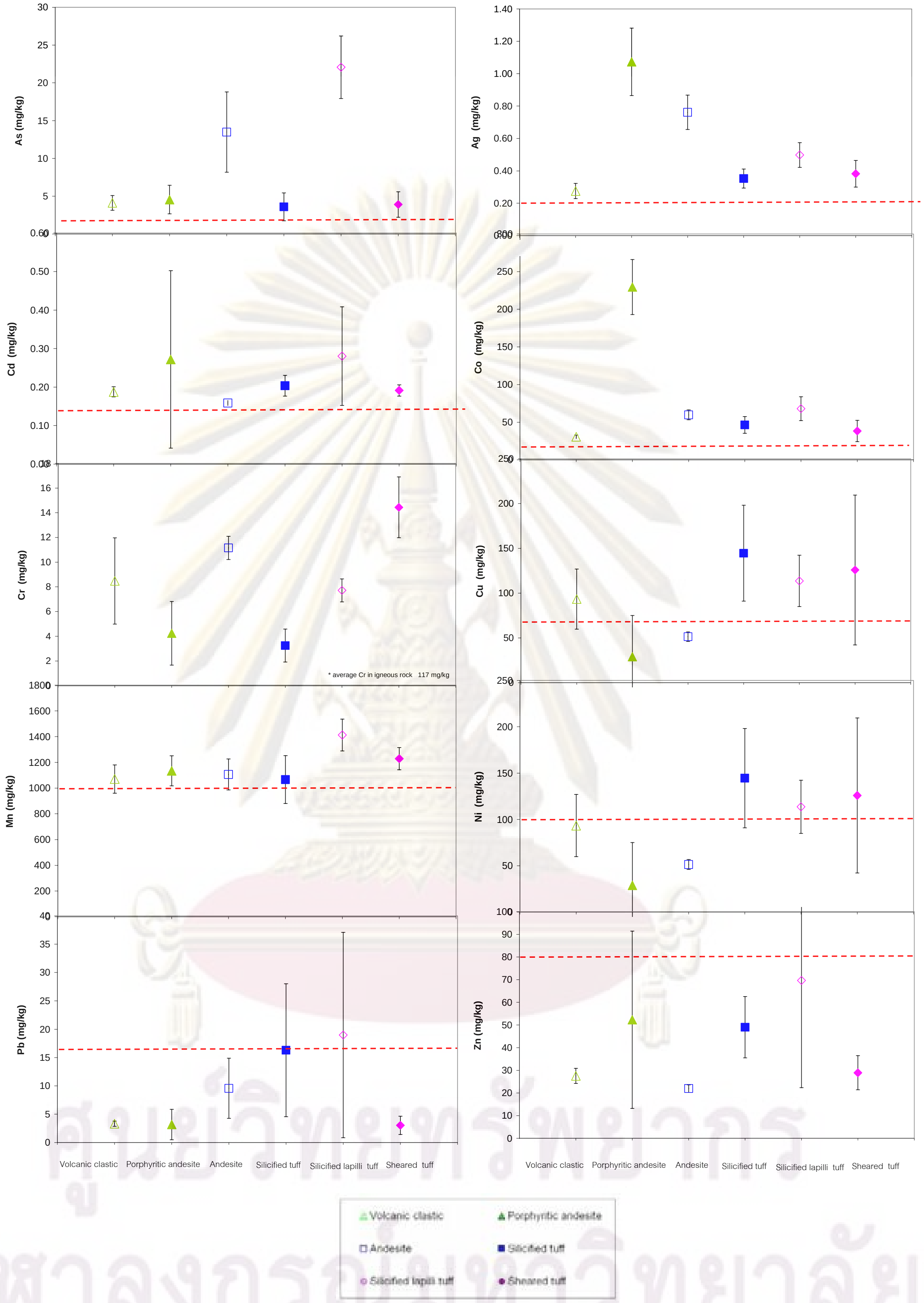

Figure 5.5 Statistic plots of total concentrations of some particular heavy metals analyzed from each waste rock type from C-H pit of Akara mine in comparison with natural igneous rock standards of Hawkes and Webb (1962) in red lines.

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Table 5.2 Statistics of total heavy metal analyses of waste rock samples from C-H pit in Akara Gold Mine.

Element Concentration (mg/kg)		Volcanic clastic	Porphyritic andesite	Andesite	Silicified tuff	Silicified Lapilli tuff	Sheared tuff
As	Range	2.93 – 5.32	1.54 - 6.48	8.82 – 21.10	1.68 – 6.60	17.00 – 27.60	1.70 – 5.47
	Average	4.11	4.55	13.48	3.59	22.06	3.90
	S.D.	0.98	1.88	5.31	1.84	4.13	1.68
Ag	Range	0.20-0.32	0.89-1.43	0.62-0.87	0.29-0.42	0.43-0.59	0.29-0.50
	Average	0.28	1.07	0.76	0.35	0.50	0.38
	S.D.	0.05	0.21	0.11	0.06	0.08	0.08
Cd	Range	0.18-0.20	0.04-0.54	0.15-0.17	0.18-0.25	0.17-0.44	0.17-0.20
	Average	0.19	0.27	0.16	0.20	0.28	0.19
	S.D.	0.01	0.23	0.01	0.03	0.13	0.01
Co	Range	26.64-32.33	202.82-293.00	52.43-68.08	33.86-62.96	49.17-85.73	13.10-47.6
	Average	30.54	229.54	59.73	46.40	68.00	38.24
	S.D.	2.31	36.59	6.43	11.20	15.84	14.22
Cr	Range	3.92-12.29	2.88-2.84	10.44-12.75	2.37-5.61	6.21-8.60	12.83-18.59
	Average	8.48	4.24	11.16	3.25	7.72	14.44
	S.D.	3.49	2.58	0.94	1.33	0.93	2.46
Cu	Range	47.64-122.45	1.00-111.00	45.67-58.11	90.91-216.78	81.33-142.47	37.31-258.98
	Average	93.35	28.96	51.33	144.53	113.63	125.85
	S.D.	33.57	46.07	5.38	53.57	28.72	83.67
Mn	Range	942.00-1229.00	978.00-1302.00	975.00-1293.00	804.00-1302.00	1234.00-1543.00	1103.00-1320.00
	Average	1070.40	1134.20	1105.80	1066.20	1412.20	1228.60
	S.D.	109.87	116.89	120.30	185.65	123.54	86.50
Ni	Range	1.27-2.40	0.45-1.61	1.31-1.54	2.73-3.12	1.75-2.60	2.58-3.35
	Average	1.85	0.86	1.45	2.93	2.35	2.93
	S.D.	0.48	0.47	0.10	0.15	0.36	0.37
Pb	Range	2.93-4.09	0.65-6.40	3.08-16.03	3.91-33.21	4.43-46.29	1.41-5.51
	Average	3.32	3.18	9.56	16.28	18.96	3.05
	S.D.	0.46	2.69	5.30	11.72	18.11	1.61
Zn	Range	24.45-32.84	8.92-83.42	18.80-21.60	32.80-61.42	33.91-124.99	16.52-34.65
	Average	27.33	42.33	19.94	46.40	66.25	23.12
	S.D.	3.32	32.13	1.40	11.28	42.19	7.40

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The total heavy metals contained in waste rocks may or may not release to reach the harmful level and impact the environment. Thus the assessment of leaching potentials of these metals is more suitable information for environmental protection plan. Because some rock contains heavy metals with high amounts but only some chemical form can be leached to the environment. That is due to chemical bonding in mineral host. Therefore, relation between total forms and leachable concentrations of individual metals are made and present graphically in Figure 5.6. As a result, good relationships between total forms and leachable forms of all concerned metals can be obtained from this study which linear equations are fit to all of the metals (see Figure 5.6). In most cases, the leachable concentrations are lower than the total concentrations. By taken the lowest and highest total concentrations of each elements into the appropriate equations, percentages of leaching potential of these heavy metals are calculated as 30.48-63.18% for As, 80.41-81.85%for Ag, 0-92.78% for Cd, 63.63-87.55% for Co, 73.38-73.45% for Cr, 91.06-100% for Cu, 87.90-99.67% for Mn, 85.28-93.51% for Ni, 0-82.78% for Pb and 44.29-76.53% for Zn, respectively. For instant, it is may conclude that in the worst case scenarios, a very strong acid pass through the waste rock materials, about 0-100 % of metals can be leach into the environment.

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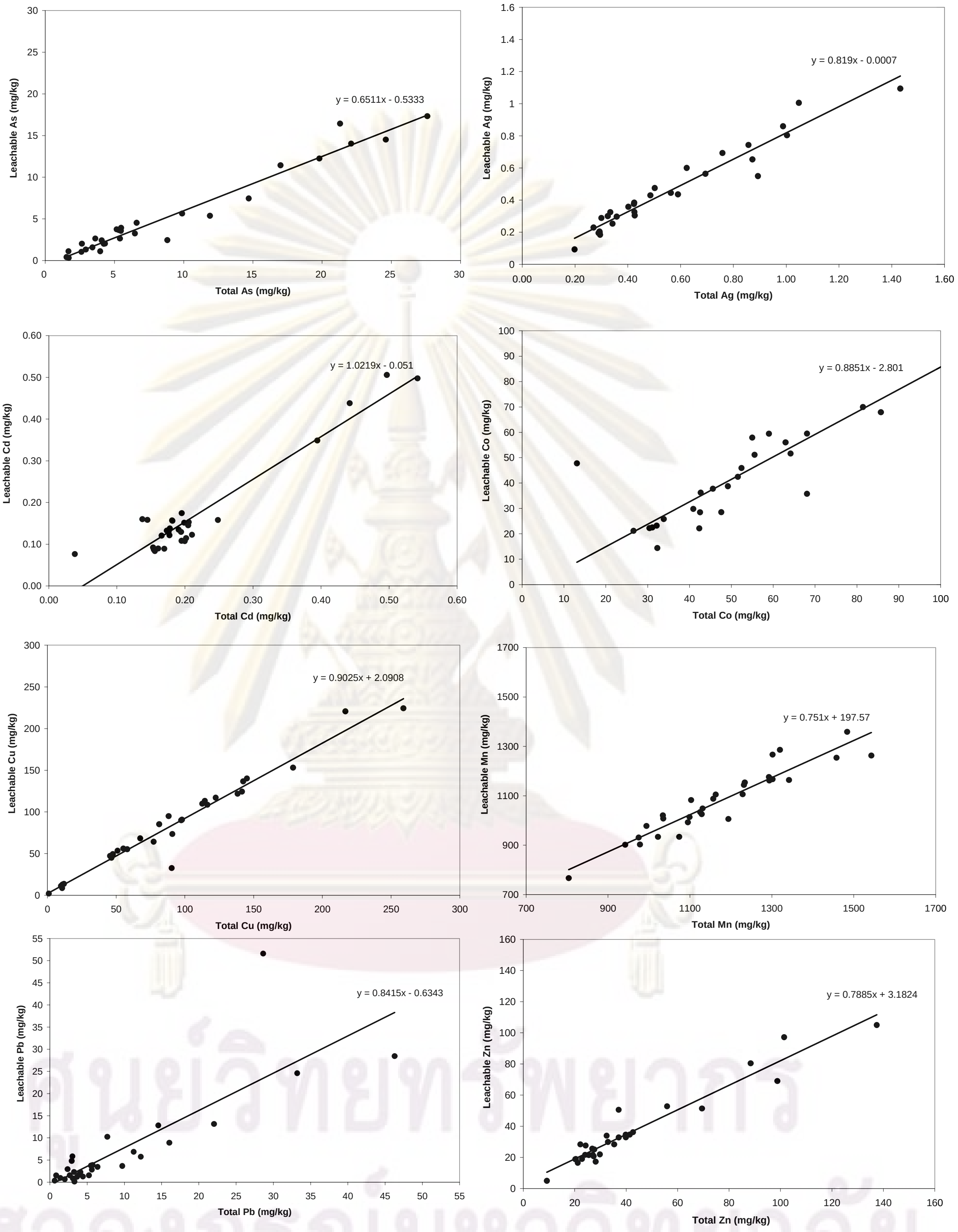

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Figure 5.6 Graphic correlations and best fit linear equations between total concentrations and leachable concentrations of selective metals analyzed from waste rock from the C-H pit, Akara Gold Mine, Phichit Province.

5.5 Experimental Leaching Test

Leachable metals are of the available potential sources of toxicity from the mine wastes. As reported in the previous part, some metals (e.g., As, Ag, Cd, Co, Cu, Mn and Pb) have potential to leach to the environment; however, such strong acidic condition may not happen in the reality if protection and monitoring plans are taken with great care.

Assessment of environmental impacts from land disposal or utilization of dredged materials also concerns the leaching behavior of these heavy metals in the waste material after disposal. The potential release of contaminants from waste materials can be assessed using various single and sequential chemical extraction techniques (Quevauviller et al. 1996). Dungan (2009) stated that there is no direct relationship between the total metal concentration and its leachability in a solid waste; hence, leaching tests are often used to determine the potential of a solid waste to contaminate natural water supply. An alternative leaching procedure, the synthetic precipitation leaching procedure (SPLP, SW-846 method 1312) which may offer more reliable conditions of a waste rock dump (Dungan and Dees 2009; Smith 1997), was designed to simulate the leaching test of heavy metals of waste rocks under this study. This method is supposed to provide a more realistic assessment of metal mobility under field conditions during precipitation event. Extraction at pH 4 has been chosen to simulate the most adverse existing environmental conditions. Exceptionally lower pH values may occur only in sulfide-rich environments. Therefore, three pH conditions of 2, 4 and 6.5 (DI water) were assigned for this experiment. The results of leaching tests at all pH conditions were compared with the Thailand industrial effluent standard and the standard of surface water quality for agriculture (PCD 2008).

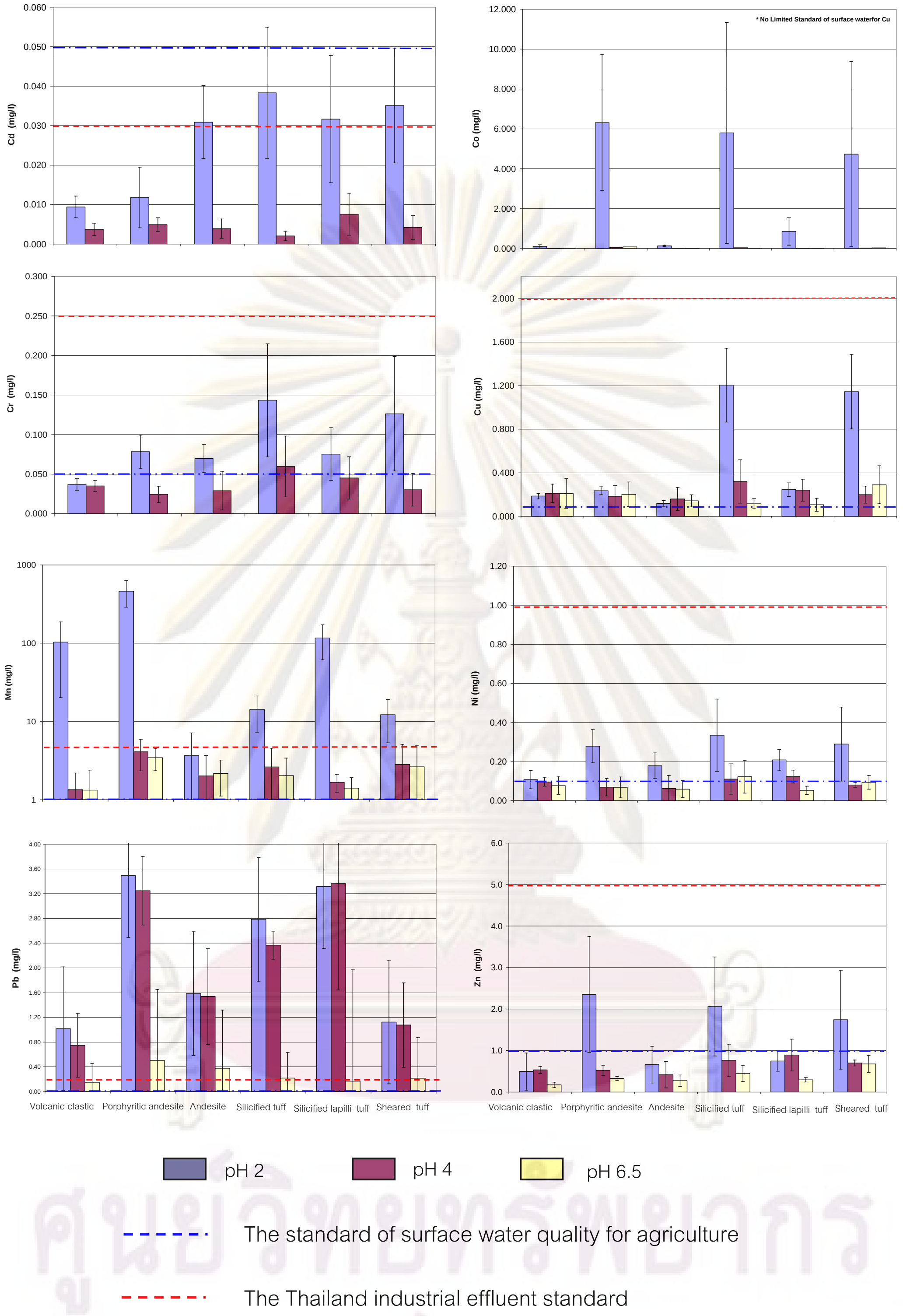

Figure 5.7 Leaching concentrations at difference pH conditions (2,4 and 6.5)of some metals exceeding the Thailand industrial effluent standard (dash red lines) and/or the standard water quality for agriculture (dash blue lines) (both standards suggested by PCD, 2008).

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In general, most metals have higher ability to leach out at lower pH (pH2); besides, many metals show just little difference of leaching at pH 4 and 6.5. Mn, Pb and Cu appear to be the most concern in the leachate because both metals in all rock types yield high concentration in the leachates at all pH conditions which are over the standards in most cases, particularly Pb, Zn, Ni, Cd and Cr are in turn exceeding the standards in some cases (see Figure 5.7). For As and Ag that are found with high concentration in the rocks (both total and leachable forms), their leachates at all pH conditions are lower than the standards. In addition, Co is also released at the pH 2 but there are no standard for Co in the Thailand industrial effluence standard for correlation.

For Cr, there are no Cr concentrations leached over both standards at pH 4 and 6.5 but it will increase until exceed the standard pH 2. Esakku et al. (2008) explained that Cr leachability in municipal solid wastes may have strong complex bonding in solid waste. Therefore, acidic condition must be high enough to remove Cr as comparable to the result of this study. .

In conclusion, Mn and Cu can be released exceeding the standard of water quality for agriculture of Thailand at all pH or even pH is at neutral. Thus, these metals may contaminate within neutral condition. Moreover, Pb has not much significant difference between the leaching capacity at pH 2 and pH 4 but the leaching capacity drop dramatically at pH 6.5. However, Pb appear to leach exceeding the Thailand industrial effluent standard at all pH.

The pH effect on leaching test found under this study agrees consistently with report of Esakku et al. (2008). They summarized that pH appears to be the significant factor influencing the release of metals. Metals are widely known to have a tendency to leach more at extreme acidic pH conditions (Van der Sloot et al., 1997; and Jang et al., 2002).

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5.6 Discussion and Conclusions

Six types of waste rocks are routinely classified as volcanic clastic, porphyritic andesite, andesite, silicified tuff, silicified lapilli tuff and sheared tuff. Based on geochemical characteristics, all types of waste rock show good relation of TiO_2, Fe_2O_3, Al_2O_3 and MgO against SiO_2 which most rocks fall within mafic to intermediate vocanic rocks as in discrimination diagram of LeBas et al. (1986). However, higher alkali in silicified lapilli tuff, unusual high silica in silicified tuff and quite low alkali in shear tuff may be involved by the later alteration process. There are field evidences showing that silicified lapilli tuff and silicified tuff are ore contact zones. Moreover, REE patterns also indicate similarities of original provenance and initial magma process of these rocks but alteration may involve compositions of these rocks with different degrees. It agrees well with geologic setting of the area that has been believed to originate in the Loei-Petchabun volcanic belt and later mineralization took place involving epithermal alteration (Diemar et al., 2000; Tangwattananukul, 2006).

As, Ag, Cd, Co, and Mn contained in most waste rock types are found higher than the averages of natural igneous rocks reported by Hawkes and Webb (1962) besides, Cu and Pb are found high concentrations in some rock types. These heavy metal analyses show different amounts of total forms and leachable forms that can be fitted well by the linear equations. About zero to 100% of heavy metals in the rocks, depending of rock type and metal type, can be potentially leached out to the environment under strong acidic condition. However, experimental leaching tests at the different pH conditions (2, 4 and 6.5) reveal Cd, Cr, Cu, Mn, Pb, Ni and Zn have potential to leach at different levels in each rock types and different pH. Pb and Mn are the most crucial elements to concern because their ability to leach easier than the others in most cases. The other metals are also released but in a non-significant amount lower than the standards.

Overall data reveal that the waste rocks have high leaching potential of leaching even under the neutral condition in some cases, particularly Pb, Mn and Cu. However, pH condition appears to be the significant factors influencing the degree of metal release. More amount metals are increasingly leached against lowering of pH. On the other hand, the high pH condition results a lower leaching metal. The results of leaching experiment are a direct consequence of pH condition in drainage. Thus, the ongoing monitoring program should be closely applied to the mining area and surrounding. The local water chemistry with respect to pre-mining condition should be record to compare with the changing environment. The pH condition of the surface water should be strongly recommended to monitor. Mn and Pb which can be leached at the all pH condition should be recommended to follow up. The contaminated leachate water from mining site also should be restricted to migrate to the surrounding environmental. Finally, the obtainable data on the amount of toxic element in mine waste can be advantage in risk assessment. In the further study, the health risk assessment should be involved in the protection plan. If there are the presence a potential metal contamination of the food chain; however, in the absence of data on the amounts of toxic elements regarding contaminant transport from the mining site into then water, ground water, soils or local vegetation. Both kind of Ecological Risk Assessment and a Human Health Risk Assessment should be quantified. The applied of potential generated metal leaching and the relation of ordinary people behavior should be done site by site to evaluate for the acceptable of risk assessment.

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CHAPTER VI

ACIDIC POTENTIAL ASSESSMENT OF WASTE ROCKS FROM THE AKARA GOLD MINE IN THAILAND

Abstract

Acid mine drainage (AMD) is the environmental issue of most public concern against the mining industry. Characterization of mine waste rock in regard to acid generation potential is a necessary routine for mining operation that will ensure proper waste rock storage without adverse environmental impact in the future. This study was therefore conducted to estimate the potential of AMD generation in the largest operating gold mine in Thailand using acid base accounting (ABA) and net acid generation (NAG) tests. Six types of waste rock including volcanic clastic, porphyritic andesite, andesite, silicified tuff, silicified lapilli tuff and sheared tuff have been classified by mining geologists for mineral processing and waste dumping; hence, representative samples of these rock types were collected for this study.

Under various conditions, experimental results indicate that volcanic clastic and silicified tuff are potentially non acid forming (NAF) whereas silicified tuff and shear tuff are potentially acid forming material (PAF). The results indicate that it is possible that AMD generation may occur a long time after mine closure due to the lag time of dissolution of acid-neutralizing source. Acidic generation from some waste rocks may occur in the future based on environmental condition, particularly the oxidation of the sulphide mineral by the combination of oxygen and water. Therefore, proper design for waste rock dumping and storage and storage is necessary to reduce the risk of AMD generation in future. It is advisable to install a surface management system to control the overland flow direction away from waste dump area and tailing storage facility and install a second water storage pond next to the main storage pond to store the spilled water during storm and rainy season. A water quality monitoring plan needs to be put in

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place, focusing on the disturbed areas, such as water storage ponds and the mine pits in particular.

6.1 Introduction

Mineral processing and extraction usually produce enormous loads of crushed rock wastes and tailings. These materials may be a potential risk to the environment after some period of exposure to the natural weathering (Cidu et al., 1995). Acid mine drainage (AMD) is the most important environmental impact often concerned with the mining industry (Akabzaa et al., 2007). The process of acid mine drainage is usually accelerated when mining exposes metal sulphide in mineralized rock and mine wastes to oxygen and water, allowing rapid oxidation of the sulfides. Sulfide oxidation has the potential to produce sulfate, which may turn into sulfuric acid when it is dissolved by rain, leading to the production of acidity drainage. Subsequently, the acid drainage can dissolve heavy metals stored in waste rock and tailings such as lead, zinc, copper, arsenic, selenium, mercury and cadmium, into surface runoff and ground water (Nriagu 1978; Smith and Skema 2001; Piyush et al. 2007; Akabzaa et al. 2007; Bennett 1969). Consequently, the high metal concentrations within acid mine drainage may in turn cause severe toxicological effects on aquatic ecosystems (Hazen et al., 2002). Metals remaining in solution may be directly toxic to biota or may enter the food chain with primary producers and subsequently be amplified to toxic levels (Bryan, 1976). Although some heavy metals are found in small quantities and some (likes copper and zinc) are required for a normal metabolism, when the quantities of these oligoelements increase, they become toxic and can cause damage and malfunctioning of human organs (Florea et al., 2005).

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6.2 Study Site

The Akara Gold Mine was selected for this study because it is the biggest active and accessible gold mine in Thailand. The mine is located along the boundary between Phichit and Petchaboon provinces, situated on the eastern edge of the Chao Phraya Basin along a gently undulating terrain, about 280 km north of Bangkok (Figure 6.1). This project comprises the mining and processing of gold ore over a period of fourteen year. Open-pit mining is carrying on and generating loads of mining wastes. These waste rocks have been deposited onto two rock waste dumping sites D and CH) as shown in Figure 6.1.

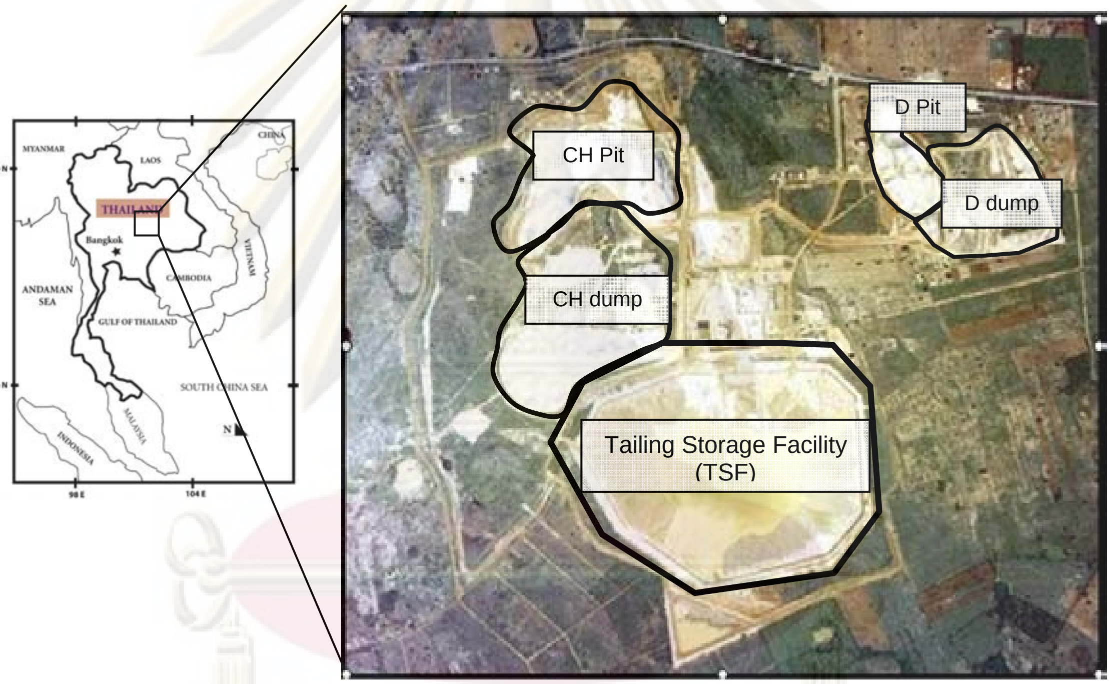

Figure 6.1 Map showing location of the Akara Gold Mine in Phichit-Petchaboon Province. C-H pit sampling site under this study and two rock waste dumping sites (D dump and C-H dump) revealed on the right map (URS 1999)

Geologically, the mining area is situated along the Loei-Petchabun volcanic belt. The belt is occupied by Upper Permian and Triassic parallel zones of ancient island arc

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containing mafic to intermediate volcanic rocks (e.g., andesite and basaltic andesite) in association with marine sedimentary rocks. Gold-silver ores are found extensively along this belt but most occurrences appear to be invisible deposit with very rare small nugget found. Mineralization has been suspected as epithermal processes occurred in shallow to steeply dipping structures. According to exploration data, the highest commercial gold deposits in the study area are detected in gold-bearing vein, breccia and stock-work veinlets which are mostly characterized by quartz-carbonate veins (Hemmanee and Poot-heng 2006). Huge quantities of rock wastes will be left behind the mining end and may be able to generate the enormous environmental impact if great care is not taken into account. Therefore, this research project was initiated to investigate the potential of acid formation of all rock wastes in the Akara Gold Mine. Rock samples were taken from C-H pit which is on going operation. Geologically, all types of waste rock exposed in this pit are similarly found in the other pits and also throughout the deposit.

The abbreviations used in the paper are listed below. The reader is advised to refer to the list first to avoid misunderstanding the terminology.

AMD	Acid Mine Drainage
ABA	Acid Base Accounting
NAG	Net Acid Generation
NAGpH	Final pH of NAG test
MPA	Maximum Potential Acidity
ANC	Acid Neutralization Capacity
NAPP	Net Acid Production Potential
PAF	Potential Acid Forming
NAF	Non Acid Forming
UC	Uncertain Classification

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6.3 Methods and Materials

AMD assessment actually requires estimation of potential for acid generation from mining waste materials. Regarding to the waste rocks, there are several methods proposed to predict AMD under various circumstances either static or kinetic conditions. However, the most common static test used worldwide is ABA as recommended by Kwong (2000). In addition, production and neutralization of AMD may occur due to complexities of physical, chemical and biological processes. Therefore only static method alone can not provide completely accurate results. A combination of kinetic and static methods is usually applied to increase reliability of results. Therefore, in this study, kinetic humidity cell tests have been used to assess the AMD potential and confirm the ABA tests in many cases. However, this assessment is usually very time-consuming and expensive. NAG test has been proposed as a comparable test to kinetic humidity cell tests by Akabzaa et al. (2007). Advantages of NAG test, including lower cost, and being a rapid and simple technique have made it becoming a highly recommended method to evaluate the acid generating potential of rocks sample along with ABA method (Schafer 2000; Greenhill 2000).

This study was focused on the waste rocks and determination of their acid producing potentials. All the waste rock samples were collected from C-H pit, the main operation of the Akara Gold Mine. ABA and NAG tests were applied for all rock samples. Finally, results of the study were used to screen geochemical characteristics of the waste rocks in the mine; subsequently, an environmental assessment and an environmental assessment and a feasible environmental protection plan are proposed in the discussion part.

Five samples of each waste rock type, based on geological field classification, were taken and kept in clean polypropylene bags before sending to laboratory for testing. The individual samples were crushed in a jaw crusher to less than 5 cm and then ground using ball mill to finer than 25 μm powder. Each sample was sieved through a 0.075 mm mesh prior to using them in the experiments. Eventually, these powder samples were ready for ABA and NAG tests.

ABA test (Weber et al., 2004) measures the balance between the MPA and the ANC. The difference between the MPA and the ANC values is described as the NAPP (expressed in kg H_2SO_4/t of sample)

$$\text{NAPP} = \text{MPA} - \text{ANC}$$

The MPA can be calculated directly from total sulphur by simple multiplication of total sulphur with 30.6 (Parker and Robertson, 1999). This calculation assumes that all the sulfur is contained in pyrite. A factor is determined from stochiometry of pyrite oxidation, calculated based on the assumption that the two molecules of acid will be produced for each mole of sulphur present. In this study, total sulphur contents were analyzed by the Leco high temperature combustion method. Average sulphur contents were determined twice for each sample. The ANC was determined by the modified Sobek method adopted by Environment Geochemistry International Pty Ltd (EGI). It was determined initially by a fizz test rating of the neutralization potential prior to determination of required amount of acid to react all of the calcium carbonate contained in the rock sample (Weber et al., 2004). However, the fizz test may be affected by personal bias, e.g., classification of the degree of effervescence (Paktunc et al., 2001); therefore, the fizz tests under study were carried out by only one researcher for all samples at the same time to minimize such bias. Subsequently, each sample was mixed with a known amount of standardized acid from fizz test, commonly using HCl or H_2SO_4. The mixture was subsequently heated and back titrated with NaOH to measure the acid remained from the reaction. Finally, the NAPP was then calculated by subtracting the ANC from the calculated MPA. The NAPP result is also expressed in the unit of kgH_2SO_4/t of sample (Weber et al., 2004).

There are various criteria for evaluation of AMD production potential, based on ABA test. Weber et al. (2004) suggested that a negative NAPP indicates a net neutralizing capacity, on the other hand, a positive NAPP indicates a net acid generation capacity. In addition, Lei and Watkins (2005) showed the criteria adapted from Hutchison and Ellison (1992) as shown in Table 6.1.

Table 6.1 Classification criteria for acid potential of mine waste based on ABA[a].

Category	NAPP (kg H2SO4/ton)
Potentially acid generating	> 20
Non-acid generating	< -20
Zone of Uncertainty	Between -20 and +20

[a] Adapted from Hutchison and Ellison (1992) cited in Lei and Watkins (2005)

The NAG test involves reaction between a rock sample and hydrogen peroxide which is assumed to oxidize rapidly most of sulphide minerals in the sample. During the NAG test, both acid generation and acid neutralization reactions can occur simultaneously; therefore, the final result actually represents a direct measurement of net acidic amount generated by the sample. 2.5 grams of rock powder sample was added by 250 ml of 15% hydrogenperoxide (H_2O_2); the mixture was placed inside a fume hood for 24 hours and then boiled for 1 hour. After cooling to room temperature, the final pH of NAG Test (NAGpH) was measured and the solution was titrated using 0.1 mol/l NaOH solutions until become pH 7 for determination of acidity. Finally, NAG was calculated in term of kg H_2SO_4/t (Finkelman and Giffin, 1986).

In general, the NAG test is always used in association with the NAPP to verify the acid generating potential of a sample. The NAPP and NAG test procedures are complementary in that the NAPP provides theoretical maxima for the acid forming and acid neutralizing reactions while the NAG is a direct measure of the net result of both types of reaction. The criteria of NAG test suggested by EGI (2005) are that NAGpH >4.5 would be considered as non-acid forming whereas range of $\leq$4.5 as acid forming.

6.4 Results and Discussion

The results of ABA and NAG tests are presented in Table 6.2; besides, calculations of net neutralization potential (NAPP = MPA - ANC) is also included. As a result, sulphur content detected from rock samples ranges from 0.1120 to 6.384 % whereas ANC of all samples ranges from 13.66 to 174 kg H_2SO_4/t. Most of the rock waste samples contain at least 1 % total S, with some rocks having up to 6.5 % total S. The highest sulphur-containing rock is sheared tuff (≈ 6.08±0.42 %) with the very low ANC value (≈ 38.7±16.3 Kg H_2SO_4/t). In contrast, silicified tuff containing the highest ANC (≈ 134±42 Kg H_2SO_4/t), also show the lowest % sulphur (≈ 0.77±1.08 %).

The assumption for computations of the variables used to predict acid generation from ABA tests is that all of the available acid generating and acid neutralizing materials are available to react (Akabzaa et al. 2007). The classification of acid generating potential under the ABA and NAG criteria for waste rock samples from the Akara Gold Mine is shown in Table 6.3.

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Table 6.2 Analyses of acid-forming potential of waste rocks from the Akara Gold Mine in Thailand.

Sample	% suphur By Leco	AP Kg H_2SO_4/t	ANC kg H_2SO_4/t	NAPP kg H_2SO_4/t	NAG pH
volcanic clastic rock (1)	2.28	69.77	95.15	-25.38	7.58
volcanic clastic rock (2)	0.99	30.29	63.04	-32.75	7.26
volcanic clastic rock (3)	3.84	117.50	252.7	-135.2	7.03
volcanic clastic rock (4)	2.13	65.18	88.82	-23.64	7.24
volcanic clastic rock (5)	1.18	36.11	62.11	-26	7.36
Porphyritic andesite (1)	0.76	23.26	30.03	-6.77	5.65
Porphyritic andesite (2)	0.79	24.17	30.75	-6.58	6.11
Porphyritic andesite (3)	0.81	24.79	32.61	-7.82	5.48
Porphyritic andesite (4)	0.45	13.77	28.26	-14.49	4.53
Porphyritic andesite (5)	0.62	18.97	24.84	-5.87	6.03
Andesite (1)	1.86	56.92	61.80	-4.88	6.5
Andesite (2)	1.95	59.67	68.70	-9.03	7.02
Andesite (3)	2.11	64.57	68.94	-4.37	7.08
Andesite (4)	1.15	35.19	25.16	10.03	4.14
Andesite (5)	1.07	32.74	84.32	-51.58	7.47
Silicified tuff (1)	0.29	8.87	114.5	-105.63	6.08
Silicified tuff (2)	0.11	3.37	96.69	-93.32	7.25
Silicified tuff (3)	2.71	82.93	185.1	-102.17	7.6
Silicified tuff (4)	0.33	10.10	100.8	-90.7	6.22
Silicified tuff (5)	0.43	13.16	174.0	-160.84	7.18
Silicified Lapilli Tuff (1)	2.54	77.72	33.23	44.49	2.55
Silicified Lapilli Tuff (2)	3.09	94.55	41.61	52.94	2.52
Silicified Lapilli Tuff (3)	2.72	83.23	32.92	50.31	2.86
Silicified Lapilli Tuff (4)	2.03	62.12	45.85	16.27	2.89
Silicified Lapilli Tuff (5)	2.05	62.73	22.05	40.68	2.61
Sheared tuff (1)	5.79	177.17	30.97	146.2	2.91
Sheared tuff (2)	6.37	194.92	13.66	181.26	2.78
Sheared tuff (3)	5.47	167.38	52.17	115.21	2.65
Sheared tuff (4)	6.37	194.92	49.69	145.23	2.94
Sheared tuff (5)	6.38	195.23	47.20	148.03	2.98

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Table 6.3 Classification of acid generating potential using various criteria for waste rock samples from the Akara Gold Mine.

Sample	ABA			NAG Test	
	NAPP (kgH_2SO_4/t)	Weber[a]	Hutchison[b]	NAG pH	EGI[c]
volcanic clastic rock (1)	-25.38	Net neutralizing capacity	NAF	7.58	NAF
volcanic clastic rock (2)	-32.75	Net neutralizing capacity	NAF	7.26	NAF
volcanic clastic rock (3)	-135.2	Net neutralizing capacity	NAF	7.03	NAF
volcanic clastic rock (4)	-23.64	Net neutralizing capacity	NAF	7.24	NAF
volcanic clastic rock (5)	-26	Net neutralizing capacity	NAF	7.36	NAF
Porphyritic andesite 1)	-6.77	Net neutralizing capacity	UC	5.65	NAF
Porphyritic andesite (2)	-6.58	Net neutralizing capacity	UC	6.11	NAF
Porphyritic andesite (3)	-7.82	Net neutralizing capacity	UC	5.48	NAF
Porphyritic andesite (4)	-14.49	Net neutralizing capacity	UC	4.53	NAF
Porphyritic andesite (5)	-5.87	Net neutralizing capacity	UC	6.03	NAF
Andesite (1)	-4.88	Net neutralizing capacity	UC	6.5	NAF
Andesite (2)	-9.03	Net neutralizing capacity	UC	7.02	NAF
Andesite (3)	-4.37	Net neutralizing capacity	UC	7.08	NAF
Andesite (4)	10.03	Net acid generation	UC	4.14	PAF
Andesite (5)	-51.58	Net neutralizing capacity	NAF	7.47	NAF
Silicified tuff (1)	-105.63	Net neutralizing capacity	NAF	6.08	NAF
Silicified tuff (2)	-93.32	Net neutralizing capacity	NAF	7.25	NAF
Silicified tuff (3)	-102.17	Net neutralizing capacity	NAF	7.6	NAF
Silicified tuff (4)	-90.7	Net neutralizing capacity	NAF	6.22	NAF
Silicified tuff (5)	-160.84	Net neutralizing capacity	NAF	7.18	NAF
Silicified lapilli tuff (1)	44.49	Net acid generation	PAF	2.55	PAF
Silicified lapilli tuff (2)	52.94	Net acid generation	PAF	2.52	PAF
Silicified lapilli tuff (3)	50.31	Net acid generation	PAF	2.86	PAF
Silicified lapilli tuff (4)	16.27	Net acid generation	UC	2.89	PAF
Silicified lapilli tuff (5)	40.68	Net acid generation	PAF	2.61	PAF
Sheared tuff (1)	146.2	Net acid generation	PAF	2.91	PAF
Sheared tuff (2)	181.26	Net acid generation	PAF	2.78	PAF
Sheared tuff (3)	115.21	Net acid generation	PAF	2.65	PAF
Sheared tuff (4)	145.23	Net acid generation	PAF	2.94	PAF
Sheared tuff (5)	148.03	Net acid generation	PAF	2.98	PAF

[a] Weber, et al. 2004

[b] Hutchison et at 1992 cited in Lei and Watkins (2005)

[c] Environmental Geochemistry International PTY LTD (EGI), 2005

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According to the classification summarized in Table 6.3, the results of ABA and NAG tests show consistency in 4 rock types: volcanic clastic rock, silicified tuff, silicified lapilli tuff and sheared tuff. Based on the criteria proposed by Weber et at. (2004) and Hutchison (1992), volcanic clastic rock and silicified tuff are similarly classified as “net neutralizing capacity” and NAF respectively whereas silicified lapilli tuff and sheared tuff are classified as “net acid generation” and PAF respectively. Shear tuff rocks contain the highest percent of sulphur (≈ 6.08±0.42 %) with rather low ANC (≈ 38.7±16.3 Kg H_2SO_4/t). Therefore, sheared tuff rocks are classified as PAF under all criteria. It should be noted that pyrite in sheared tuff was clearly visible during the field investigation.

Volcanic clastic and silicified tuff are classified as NAF under both ABA and NAG criteria. Silicified tuff rocks contain high ANC (> 100 kgH_2SO_4/t) and low MPA (≈10 kgH_2SO_4) that may lead to full neutralization capacity to all acidic generation. Although volcanic clastic rocks contain lower ANC than silicified tuff, their sufficient neutralization capacity can still neutralize MPA from sulphur leading to non-acidic potential.

Only porphyritic andesite and andesite yielded inconsistency between ABA and NAG tests. These two rocks were classified as UC under Hutchison criteria. Therefore, both rock types have moderate ANC values corresponding to low MPA values. By taking two from three criteria as NAF, both rock types should be more suitable to classify as NAF.

Although classification of acidic potential can be completely assessed, the most important issue is waste rock management in the real practice. In the Akara Gold Mine, mining geologists have routinely been separating all six types of rock waste into 4 main groups, 1) volcanic clastic rock, 2) porphyritic andesite, 3) sheared tuff and 4) the rest (including andesite, silicified tuff and silicified lapilli tuff) for disposal around the rock waste pile. Averaged tonnages of each waste rock type estimated for disposal during the mining period by the mine geologists are shown in Table 6.4.

Table 6.4 Tonnages of waste rock estimated for disposal

Rock Type	Average NAPP (kgH_2SO_4/t)	Volume (m^3)	Amount (ton)	Class
1. Volcanic clastic rock	-48.6	9,759,351	24,691,158	NAF
2. Porphyritic andesite	-8.3	11,029	29,227	NAF
3. Andesite	-11.9	6,040,449 (mixed)	15,825,976 (mixed)	NAF
4. Silicified tuff	-110.5			NAF
5. Silicified lapilli tuff	40.9			PAF
6. sheared tuff	147.2	575,328	1,461,333	PAF

The total mine wastes are around 16,386,157 m^3 occupied by volcanic clastic rock (about 24,000,000 ton) > mixed group (andesite, silicified tuff silicified lapilli tuff) > sheared tuff > porphyritic andesite, respectively. Based on the acid forming potential, andesite and silicified tuff are classified as NAF. Silicified lapilli tuff is suitable to be controlled as the PAF material. However, they are routinely mixed while gathering. Tonnages of individual materials are not available. However, the acidity of the mixed material can be calculated only if we have the tonnage estimation of individual materials. Therefore, two assumptions are made. The andesite and silicified tuff group is classified as NAF material and silicified lapilli tuff as PAF material. The ratio of mixing of these two groups (NAF:PAF) are set as 10:90 and 90:10 for calculating the acidic potential of the mixture. The acidic potential was calculated by multiplying the tonnage of waste rock with the average NAPP value from the ABA test in terms of kgH_2SO_4. The estimates of the amount of acid generation are shown in Table 6.5.

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Table 6.5 Estimated amounts of acid generation and neutralization (in form of H_2SO_4) that may occur after closure of mining activity.

Rock Type	Class	Mixing Ratio[a]	Amount (ton)	Net Acidity kgH_2SO_4	Mixing Ratio[a]	Amount (ton)	Net Acidity kgH_2SO_4
1. Volcanic clastic rock	NAF	-	24691158	$-1.2x10^9$	-	24691158	$-1.2x10^9$
2. Porphyritic andesite	NAF	-	29227	-2.4×10^5	-	29227	-2.4×10^5
4. Andesite	NAF	0.1	1582597.6	$-1.9X\ 10^8$	0.9	14243378	-1.7×10^9
3. Silicified tuff	NAF						
5. Silicified lapilli tuff	PAF	0.9	14243378	5.8×10^8	0.1	1582597.6	6.4×10^7
6. Sheared tuff	PAF	-	1461333	2.15×10^8	-	1461333	2.15×10^8

[a] mixing ratio ; the ratio between Andesite+Silicified tuff (NAF) : Silicified lapilli tuff (PAF)

In both ratio, the neutralization capacity of only volcanic clastic rock ($-1.2x10^9$ kgH_2SO_4) is sufficient to neutralize the acidity from silicified lapilli tuff and shear tuff. Although 100% of mixed group is occupied by only silicified lapilli tuff, possibly generating about 6.4 $x10^8$ kgH_2SO_4, it is still deficient to cause AMD. Consequently, based on the results of this study, AMD will not be generated as far as the neutralization capacity from NAF materials is still higher than the acidity from PAF materials. However, due to the complicated nature of acid generating and lag time of dissolution of neutralizing source, the dissolution rate of acid is much slower than neutralization (Lei and Watkins, 2005); consequently, AMD may occur after a long period of time if protection plan is neglected or practiced with carelessness. Good management of waste rock placing in dumping site is necessary to reduce the risk of AMD generation for a long time into the future after mine closure.

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6.5 Conclusions and Recommendations

The results of the geochemical testing of the waste rock from Akara Gold Mine, Thailand indicated two types of waste rock that having the potential to generate the acidity (classify as PAF): silicified lapilli tuff and sheared tuff. Of most concern to the environment is the sheared tuff based on its high acid forming potential with low neutralization capacity and high total S content (5-6 % S). In contrast, there is silicified tuff which has high neutralization capacity (ANC > 100 kgH_2SO_4/t) in the project area. With very huge amount of volcanic clastic rock having moderate neutralization (ANC 60 to 250 kgH_2SO_4/t) there is sufficient capacity to neutralize the acidity generated by PAF materials, shear tuff and silicified lapilli tuff. However, there is a long-term possibility after the mine's closure that AMD may be generated due to the lag time of dissolution of acid-neutralizing source. Therefore, a long term monitoring and good runoff management should be considered in addition to good practice of waste rock placing management. Based on this study, the following recommendations are suggested to improve disposal practices and control AMD generation includes:

- Covering the PAF material with NAF material and then the clay layer would be designed to minimize the potential for water or oxygen ingress. Control placement of NAF and PAF material within the dump could significantly reduce the overall AMD load. Storage of oxidation product and mineral surface will also slow the oxidation rate due to the mineral armor.

- Install the surface management system to control the overland flow direction away from waste dump area and tailing storage facility (closed to waste dump area; see Figure 1). Thus, only the rain fall can pass the waste dump.

Therefore, the use of ABA and NAG tests should be encouraged as routine tools to assess acid generating potential, particularly in environmental impact assessment of mining projects during active mining and/or before opening the new mine to support a circumspect mine waste management plan. These would be the first strategy of the project to prevent the acid generation. Hence, the cooperation between the mine environmental section and field geologists is also very important. The results of estimated acid generating potential of waste rock from the environmental section need to be properly communicated to the practice section for implementation of sound management of waste rock placing in dumping sites.

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CHAPTER VII

CONCLUSIONS AND RECOMMENTDATIONS

Thirty waste rock samples and thirty-seven tailing samples collected from the Akara Gold Mine are the main materials for this study. Both types of materials are the main wastes that will be left after closure of the mine. These may cause a serious environmental impact since they have capacities to generate acidic drainage and heavy metal leaching. The quality of drainage from mine wastes is a function of mine waste composition which is variable from place to place. Therefore, the study of mine waste characteristics is very important for environmental management plans. The future quality of drainage must be predicted for monitoring design. Thus, this study is focused on their chemical characteristics particularly in aspect of potential to acid drainage formation and heavy metal leaching drainage. All information gained from this study provides such good idea for environmental management. Actually, a mathematic model, TREX, was also applied based on parameters yielded from the study and intended to yield transport of some metals but it has unexpected error. Even communication with the model provider could not solve the problem. However, changing in concentration of the source still gives an idea of transportation. All findings will be summarized and environmental management will be recommended eventually.

7.1 Environmental Impacts of Tailings

Tailing samples were collected systemically from 13 different locations distributed over the area of tailing storage facility (TSF) for investigations of geochemical characteristics and the acid generating potentials. Result reveals that their chemical compositions are slightly different from place to place but mineral components can not be clearly differentiated. The essential minerals of all tailing samples are quartz (SiO_2), alkali feldspar, calcite ($Ca(CO_3)$), pyrite (FeS_2), kaolinite ($Al_2Si_2O_5(OH)_4$) and talc ($Mg_3(Si_4O_{10})(OH)_{2/3}MgO\ 4SiO_2\ H_2O$). Distribution pattern of their chemical compositions are unsystematically spreading over the TSF due to circulation of disposal system.

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Although all heavy metal contained in tailings are higher than those of undisturbed topsoils in the area, most of them are still lower than the Thailand Soil Quality Standards for Habitat and Agriculture. Except Mn, its concentrations appear to have higher content than the standard. Based on experimental leaching test, only Mn and Pb seem to be the most problematic elements because they can be released with concentrations over the industrial effluent standard and the surface water quality standard at all experimental pH (2, 4 and 6.5). On the other hand, the other metals (e.g., Cr, Cu and Zn) are mostly below both the standards at all pH conditions.

In addition, tests of acid forming potential clearly demonstrate that all tailing samples are classified as non acid forming (NAF) materials. Although the tailing samples contain a high amount of sulfur, they also contain high acid neutralization capacity. Thus, their acid neutralization capacities are much more sufficient to suppress the generated acid. Consequently, these tailings have no potential to generate acid drainage. However, the tailings are needed to be controlled avoiding the interaction with oxygen and water due to its potential to leach of Mn and Pb under acid and neutral conditions.

7.2 Environmental Impacts of Waste Rocks

Waste rocks under this study have been classified, based on field experience of mining geologist, as volcanic clastic, porphyritic andesite, andesite, silicified tuff, silicified lapilli tuff and sheared tuff. Geochemistry of these waste rocks was studied; subsequently, results reveal similar geologic signature of all waste rocks which appear to have undertaken some alteration process with different degrees. Based on silica content, these rocks can be categorized into three main groups including high-, medium- and low-silica contents. Discrimination diagram of SiO_2 against alkali contents (Na_2O+K_2O) (proposed by LeBas et al., 1986) was applied to these rocks. Volcanic clastic samples fall within the field between tephrite basanite and trachy-basalt that would be due to various compositions of fragments embedded in this rock type.

Porphyritic andesite and andesite are typically fitted well with their fields. On the other hand, silicified lapilli tuff is chemically similar to tephri-phonolite. Silicified tuff and sheared tuff, they mostly fall out the common rock compositions that may be caused by alteration process. The similar depletion REE patterns of all waste rocks would indicate the close relations in initial provenance and magma process of all rock types; however, later alteration process which may have taken place during gold mineralization involved their compositions.

Regarding to heavy metals, all rock types contain As, Ag, Cd, Co and Mn higher than the natural igneous standards whereas Cu and Pb in some waste rocks are found higher than the common igneous rocks. However, these metals have available forms to leach lower than the total forms; only some metals, particularly Cd, Co, Cu, Mn, Ni, Pb and Mn, can be leached to the environment. In addition, good relationships between total forms and leachable forms of all concerned metals can be constructed as reported in Chapter 5. In most cases, the leachable concentrations are lower than the total concentrations by about 44-100%.

Experimental leaching tests at pH 2, 4 and 6.5 were carried and compared with the Thailand industrial effluent standard and the standard of surface water quality for agriculture. The results reveal that some metals (e.g., As and Ag), high total forms in the rocks, may be leached out with lower concentrations than the standards. The pH condition appears to be the most significant factor influencing the release of metals. Most metals have higher ability to leach out when pH is dropped. Cd, Cr, Cu, Mn, Pb, Ni and Zn have potential to leach at different levels in each rock types and different pH. Pb, Mn and Cu become the most concern because these metals can be leach even at neutral pH.

Assessments of acid producing potentials show both NAF and PAF potentials in these waste rocks. Volcanic clastic, porphyritic andesite, andesite and silicified tuff are classified as NAF materials whereas silicified lapilli tuff and sheared tuff are classified as PAF materials. It should notified that the ore contacted materials, silicified lapilli tuff and

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sheared tuff, significantly contain a high percent of sulfur leading to potential of acidity generation over the neutralization capacity. These PAF materials need a good environmental management for placing in proper dumping site which try to reduce the risk of AMD generation for a long time after mining closure.

7.3 Contaminated Runoff Modeling

Contamination of runoff over the study have been tried to simulate using TREX model. The model was expected to yield reasonable results in terms of runoff hydrographs, sediment transport (both suspension and metals) in the watershed. However, result of TREX simulating appears to have error all the times. Results show some changing of metal concentrations with inconsistent trend but no transportation of sediment from the initial grid cell could be present. Although this error was discussed with the model provider, Dr. Mark L. Velluex, from Colorado State University, it could not be solved. That may be cause by nature of the area and lacking of some information. The model calibration required a lot of data that there are no records in the boundary of the study area. For examples, the runoff volume and peak flow in the stream have never been investigated by any agency. In addition, vicinity of the mining site is mostly occupied by flat plain in which topography has no difference and flowing rate must be low.

However, TREX is still very advantage tool to fully distribute watershed model with respect of metal source remediation. It allows the user to identify key source areas, such as waste pile, within the watershed. Hence, TREX is flexible to specify several directions of metal leaching from a waste pile. Therefore, chemicals can move in different paths: (1) as non-reactive particle that move like soil and sediment; (2) as sorbed chemicals that undergo equilibrium partitioning and redistribute themselves between dissolved, DOC-bound or particulate phases; or (3) as reactive particles that can be dissolved at a specified rate and where the dissolved chemical mass resulting from the dissolution process undergoes equilibrium partitioning. The user can also use

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point source and distributed load as approach to specify the result of a leaching process.

However, to assess the potential affected area, the elevation data are considered. The study boundary was selected to cover Sai Yang Rung Basin which is the closest waterway from the mine (about 3 kilometers). The factor that influences the direction of surface runoff is the elevation of surface area. Typically, water flows from a higher point of upgradient to a lower point of downgradient because of natural gravity. According to study boundary in Figure 7.1, it is found that the elevation of the basin decreases from north-east to south-west.

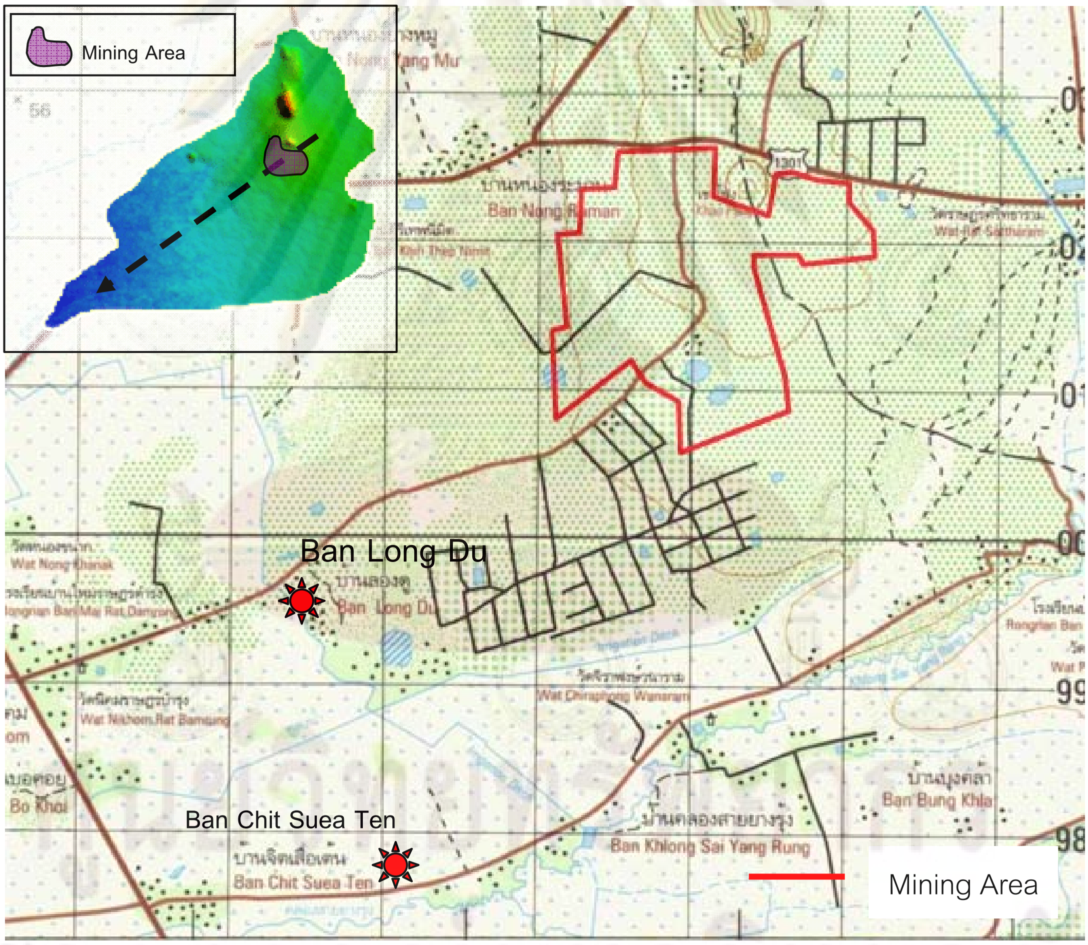

Figure 7.1 Estimate the flow direction following the decreasing of elevation in the study boundary and possible affected area.

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Thus, it may conclude that the runoff flows from north-east to south-west as shown by dash arrow in Figure 7.1. It is possible that area on the west of the mine may be affected from the mine drainage, particularly in case of over flooding from the mine. From this assumption, Ban Long Du may be the highest risky area. Monitoring should be greatly concerned especially in the rainy reason when it may have flooding spill over the mine.

7.4 Recommendations

Alternative strategies for the environmental and mining management at Akara Gold Mine must concern the control of the acid generation of some rock waste. Mine waste emplacement, area management, technologies and some suggestions for the monitoring plan are summarized below.

7.4.1 Mine Waste Emplacement

To improve the management techniques to control AMD, controlling the problem at the initial source is strongly recommended. Suppression of acid generation can be achieved by removing three essential components (i.e., sulfide, air and water) supporting the acid-generating process.

- *Rock Waste Segregation and Blending*: This would include thoroughly blending the acid-generating rock (silicified lapilli tuff and sheared tuff) with enough net neutralizing potential rock. Neutral pH levels within the waste system can be maintained. Waste rock should be placed restrictively to waste rock pile. Oxidation of sulfide minerals and dissolution of carbonate minerals are surface reaction and therefore, the rate of these reaction are dependent on the reactive surface area (Lapakko, 2002). Mixture of different size materials will be helpful to lower the pore volume of waste dump; then

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availability of air and water will be decreased. PAF materials are recommenced to maintain a large size for reducing the surface area whereas NAF material with a smaller size should be placed between the PAF layers. Covering the PAF material with NAF material followed by clay and topsoil layer before growing the native grass to reduce erosion would minimize the potential ingress of water and oxygen.

- *Covers and Caps*: Soil, clay, and synthetic material covers can alternatively be placed over the acid-generating rock to minimize oxygen diffusion (through air or water) into the sulfides mass and to eliminate water infiltration into concentrate, tailing or waste rock that are exposed to weathering.

7.4.2 Area Management and Construction

Waste dump and tailing storage facility are located on the western side of mine. The C-H dump which has a high slope gradient may cause high erosion. This is an important issue to be managed.

- *Construction Design:* Terraces or steps would be constructed along hill slope of C-H dump to slow down runoff. Lower runoff flow will reduce the water force that can erode the hillside.

- *Collection and Treatment of Contaminants*: Over spills from the waste dump should be diverted to the constructed wetlands in order to pre-treat before discharging the treated water to the environment. Wetland or settling pound and drainage system around the mine should be improved to prevent mining waste water and suspension releasing to the natural water system. Size of pond should be big enough to catch all amounts of water and suspension in the rainy season.

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- Diversion system of runoff flowing away from tailing storage should be constructed to avoid de-construction of the facilities. Diversion system might be small ditch system lined around the tailing storage facility. These ditches should also be permanently topped by concrete or blasted rock to avoid collapse and erosion.

7.4.3 Monitoring Program

The monitoring and environmental plans should be continuously improved to increase the efficiency of environmental management. Furthermore, long-term monitoring in the area should be concerned as suggested below.

- Monitoring program of surface, deep well and shallow well waters should be performed every 6 months. However, in case of increasing concentrations of any metal, especially Mn, Cu and Pb, more frequent sampling (every 3 months) would be considered.

- Install the monitoring stations in the down gradient direction. In this study, it is recommended to install the monitoring station/well between the direction to Long Du District, Ban Chit Suea Ten and vicinity as present in Figure 7.1.

- It is recommended to collect sediment in Klong Sai Yang Rung in which located at the downstream of the mine for a routinely metal analysis.

- Not only soil sample in the project area, but also soil sample outside the project area should be monitored continuously.

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- Some water level gage stations in the stream should be installed to study further the relation between rainfall and flow rate in the area. That will be useful for modeling.

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Appendixes

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Appendix A

Table A-1 Averages of heavy metal in natural igneous rock and sand stone. (Hawkes and Webb, 1962)

Heavy metal	Averages in	
	Igneous rock (mg/kg)	Sand stone (mg/kg)
Arsenic (As)	2	75-225 *
Silver (Ag)	0.2	0.4
Cadmium (Cd)	0.13	0.3 **
Chromium (Cr)	117	10-100
Cobalt (Co)	18	1-10
Copper (Cu)	70	10-40
Iron (Fe)	4.65	3.1
Lead (Pb)	16	10-40
Manganese (Mn)	1000	385
Nickel (Ni)	100	2-10
Sulfur (S)	900	2200
Zinc (Zn)	80	5-20

NOTE : * Averages of heavy metal in black shale

** Averages of heavy metal in shale

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Table A-2 The Thailand industrial effluent standard and The standard of surface water quality for agriculture (PCD, 2008)

parameters	The Thailand industrial effluent standard	The standard of surface water quality for agriculture
pH	5.5 – 9	5-9
Zinc (Zn)	< 5.0 mg/l	< 1.0 mg/l
Cr (hexavalent)	< 0.25 mg/l	< 0.05 mg/l
Cr (Trivalent)	< 0.75 mg/l	< 0.5 mg/l
Copper (Cu)	< 2.0 mg/l	<0.1 mg/l
Cadmium (Cd)	< 0.03 mg/l	< 0.05 mg/l
Lead (Pb)	< 0.2 mg/l	< 0.05 mg/l
Nickel (Ni)	< 1.0 mg/l	< 0.1 mg/l
Manganese (Mn)	< 5 mg/l	< 1.0 mg/l
Arsenic (As)	< 0.25 mg/l	< 0.01 mg/l

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Appendix B

Table B-1 Total heavy metal concentrations in mine waste rock samples from Akara Mine based on EPA 3052 method and ICP-OES analysis.

Sample	ELEMENTS (mg/l)							
	Cd	Co	Cr	Cu	Mn	Ni	Pb	Zn
volcanic Calstic	0.18	26.64	9.91	116.43	1125	2.02	3.30	25.50
volcanic Calstic	0.20	30.44	5.77	67.55	1022	1.43	2.93	27.37
volcanic Calstic	0.20	32.16	12.29	112.66	942	2.40	3.24	32.94
volcanic Calstic	0.18	31.13	10.49	122.45	1229	2.14	3.02	27.61
volcanic Calstic	0.18	32.33	3.92	47.64	1034	1.27	4.09	24.26
Porphyritic Andesite	0.50	212.17	3.42	9.90	1129	1.00	5.63	88.42
Porphyritic Andesite	0.54	227.74	3.01	10.78	1302	0.74	6.40	98.79
Porphyritic Andesite	0.14	202.82	8.84	11.01	1099	1.61	2.38	29.80
Porphyritic Andesite	0.14	293.00	3.07	12.11	978	0.45	0.84	35.34
Porphyritic Andesite	0.04	211.96	2.88	1.00	1163	0.51	0.65	9.21
Andesite	0.16	55.00	10.58	51.05	975	1.46	12.21	20.25
Andesite	0.15	68.08	12.75	46.58	1095	1.54	5.24	22.20
Andesite	0.16	64.15	11.20	55.24	1131	1.53	11.25	22.85
Andesite	0.17	58.99	10.44	58.11	1293	1.31	3.08	24.07
Andesite	0.15	52.43	10.81	45.67	1035	1.40	16.03	20.45
Silicified tuff	0.25	51.54	2.37	216.78	1302	2.73	14.55	55.93
Silicified tuff	0.21	42.70	5.61	138.37	1157	3.00	3.91	37.14
Silicified tuff	0.19	33.86	2.90	90.91	994	2.96	22.04	39.90
Silicified tuff	0.18	40.92	2.78	178.72	1074	3.12	33.21	42.67
Silicified tuff	0.20	62.96	2.60	97.87	804	2.83	7.71	69.47
Silicified lapilli tuff	0.17	49.17	7.70	81.33	1342	2.54	4.43	39.85
Silicified lapilli tuff	0.18	68.07	8.35	141.56	1543	2.58	5.73	32.42
Silicified lapilli tuff	0.39	81.44	7.73	142.47	1234	2.27	28.64	101.43
Silicified lapilli tuff	0.44	85.73	6.21	114.53	1484	1.75	46.29	137.45
Silicified lapilli tuff	0.21	55.57	8.60	88.24	1458	2.60	9.72	37.09
Sheared tuff	0.20	47.60	12.93	97.37	1320	2.58	2.67	21.17
Sheared tuff	0.20	45.60	13.07	90.43	1103	3.31	5.51	27.04
Sheared tuff	0.17	42.53	12.83	145.14	1232	2.70	2.00	26.85
Sheared tuff	0.20	13.10	14.78	77.31	1194	2.72	1.41	28.14
Sheared tuff	0.19	42.36	18.59	258.98	1294	3.35	3.65	41.44

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Table B-2 Total heavy metal concentrations in mine waste rock samples from Akara Mine based on EPA 3051 (Nitric Acid) method and ICP-OES analysis.

Sample	ELEMENTS (mg/l)							
	Cd	Co	Cr	Cu	Mn	Ni	Pb	Zn
volcanic Calstic	0.12	21.15	5.36	108.67	1034	1.83	0.15	21.45
volcanic Calstic	0.15	22.25	4.04	68.34	934	1.35	4.80	20.74
volcanic Calstic	0.15	23.25	7.84	109.79	902	2.20	2.29	29.84
volcanic Calstic	0.13	22.52	6.07	117.11	1106	1.94	5.82	25.08
volcanic Calstic	0.16	14.39	3.02	49.38	1021	1.07	2.27	27.56
Porphyritic Andesite	0.51	192.54	0.81	11.23	1026	0.64	2.81	80.42
Porphyritic Andesite	0.50	193.33	1.10	8.67	1167	0.71	3.45	69.02
Porphyritic Andesite	0.16	184.47	6.94	12.91	1014	2.14	2.93	21.93
Porphyritic Andesite	0.16	257.00	0.96	13.78	903	0.27	1.53	28.34
Porphyritic Andesite	0.08	182.84	0.68	2.04	1105	0.30	0.35	4.92
Andesite	0.08	57.91	7.48	53.53	931	1.24	5.73	18.80
Andesite	0.09	59.51	10.56	45.11	993	1.42	1.54	28.32
Andesite	0.09	51.65	9.32	56.06	1049	1.50	6.85	19.01
Andesite	0.09	59.47	8.69	55.28	1176	1.29	0.84	21.60
Andesite	0.09	45.91	6.52	47.20	1008	1.37	8.90	18.95
Silicified tuff	0.16	42.47	1.88	220.66	1267	2.75	12.81	52.77
Silicified tuff	0.15	36.23	3.12	121.84	1088	3.11	1.68	32.80
Silicified tuff	0.13	25.80	2.02	73.46	978	3.17	13.15	32.83
Silicified tuff	0.14	29.82	2.02	153.09	934	3.09	24.58	36.19
Silicified tuff	0.17	56.03	4.22	90.74	767	3.58	10.24	51.42
Silicified lapilli tuff	0.13	38.77	6.68	85.31	1164	2.65	1.28	34.56
Silicified lapilli tuff	0.16	35.75	7.98	124.40	1263	2.32	3.87	33.91
Silicified lapilli tuff	0.35	69.97	6.63	136.68	1154	1.90	51.60	97.20
Silicified lapilli tuff	0.44	67.90	5.19	113.31	1359	1.54	28.45	104.99
Silicified lapilli tuff	0.12	51.15	7.75	94.95	1254	2.25	3.65	50.57
Sheared tuff	0.11	28.52	10.49	89.99	1286	2.09	1.54	16.52
Sheared tuff	0.11	37.77	11.61	32.68	1083	2.35	3.74	21.59
Sheared tuff	0.12	28.48	11.09	140.14	1144	2.54	0.66	25.58
Sheared tuff	0.11	47.78	9.53	64.32	1006	1.98	0.94	17.24
Sheared tuff	0.13	22.14	11.23	224.36	1162	2.75	1.22	34.65

Table B-3 Total heavy metal concentrations in mine tailing samples at surface depth from Akara Mine based on EPA 3052 method and ICP-OES analysis.

Sample	Elements (mg/l)							
	Cd	Co	Cr	Cu	Pb	Mn	Ni	Zn
T1-0	<0.05	7.3	14.15	78.11	32.15	1950	6.26	148.5
T2-0	<0.05	8.97	14.32	101.7	92.31	2782	8.24	199.8
T3-0	<0.05	10.62	12.14	78.38	23.06	1555	5.93	84
T4-0	<0.05	11.06	17.25	86.43	34.19	1805	5.46	111.8
T5-0	<0.05	15.48	15.5	94.13	35.37	1745	8.14	105.6
T6-0	<0.05	12.34	14.8	82.37	43.87	1579	4.58	131.9
T7-0	<0.05	7.76	15.55	81.57	54.13	1517	5.4	162.5
T8-0	<0.05	1.75	17.54	77.05	22.58	1778	4.9	78.6
T9-0	<0.05	4.44	15.04	82.24	32.32	1817	7.02	106
T10-0	<0.05	8.93	15.03	89.71	34.24	1667	6.29	117.6
T11-0	<0.05	9.17	15.09	114	34.69	1812	6.17	134.5
T12-0	<0.05	9.34	13.68	88.06	40.5	1664	6.9	111.1
T13-0	<0.05	15.11	16.35	98.39	64.75	1598	7.58	230.9

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Table B-4 Total heavy metal concentrations in mine tailing samples at depth 1-2 m from Akara Mine based on EPA 3052 method and ICP-OES analysis.

Sample	Elements (mg/l)							
	Cd	Co	Cr	Cu	Pb	Mn	Ni	Zn
T2-1	<0.05	15.71	15.54	83.12	31.22	2668	9.32	119.9
T3-1	<0.05	9.75	15.84	98.38	43.18	2417	10.12	173.2
T4-1	<0.05	10.75	14.23	80.99	31.3	2491	6.29	105.8
T5-1	<0.05	10.1	13.78	87.46	37.75	2691	7.81	133.7
T6-1	<0.05	9.92	14.81	88.05	44.16	2354	7.6	138.7
T7-1	<0.05	17.27	14.42	96.18	46.27	2005	7.42	177.5
T8-1	<0.05	13.2	14.2	87.13	28.8	1717	6.77	109.2
T9-1	<0.05	6.45	14.41	84.9	36.02	1698	5.44	107.1
T10-1	<0.05	13.14	16.01	94	43.78	2731	8.49	156.3
T11-1	<0.05	9.58	14.87	86.42	21.9	2484	7.33	89
T12-1	<0.05	8.42	15.25	90.02	35.22	2556	7.07	119.8
T13-1	<0.05	15.09	15.68	88.15	24.61	1825	5.5	96.5

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Table B-5 Total heavy metal concentrations in mine tailing samples at depth 1-2 m from Akara Mine based on EPA 3052 method and ICP-OES analysis.

Sample	Elements (mg/l)							
	Cd	Co	Cr	Cu	Pb	Mn	Ni	Zn
T2-2	<0.05	13.25	16.62	86.22	52.44	2900	10.99	159.7
T3-2	<0.05	19.62	17.87	113.5	63.98	1788	7.5	201.5
T4-2	<0.05	9.37	16.79	89.75	39.7	1788	7.5	136.3
T5-2	<0.05	10.2	16.19	98.69	50.46	2020	5.66	188.6
T6-2	<0.05	10.04	15.27	81.52	45.12	2242	7.21	132.5
T7-2	<0.05	11.99	16.05	102.3	46.97	1941	7.09	192.3
T8-2	<0.05	7.1	18.09	113.6	60.01	2550	6.04	211.8
T9-2	<0.05	5.23	13.83	90.24	41.1	2264	6.04	112.2
T10-2	<0.05	13.12	16.6	100.3	64.13	2261	8.97	228.3
T11-2	<0.05	9.64	15.7	81.18	55.48	1904	5.77	164
T12-2	<0.05	5.68	15.94	92.65	44.1	2086	6.32	156.8
T13-2	<0.05	10.64	15.01	121	49.59	2372	7.02	174.7

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Appendix C

Table C-1 Major elements in mine tailing samples at surface depth from Akara Mine.

Sample	Na_2O	MgO	Al_2O_3	SiO_2	P_2O_5	K_2O	CaO	TiO_2	MnO	Fe_2O_3	LOI	TOTAL
	%											
TSF 01 0	0.54	1.79	6.03	51.39	0.17	3.38	12.83	0.53	1.21	18.93	3.46	100.24
TSF 02 0	0.27	1.50	26.18	47.01	0.26	3.87	6.43	0.51	0.90	11.11	4.25	102.27
TSF 03 0	0.26	1.34	24.34	42.03	0.22	2.13	10.81	0.45	1.04	12.05	6.36	101.04
TSF 04 0	0.48	1.63	25.35	43.54	0.24	0.66	13.03	0.38	1.08	11.01	4.38	101.76
TSF 05 0	0.83	1.87	8.69	50.36	0.17	2.55	13.83	0.71	1.18	15.08	5.85	101.11
TSF 06 0	0.71	1.75	6.43	52.53	0.16	2.65	14.43	0.68	0.96	14.63	6.01	100.93
TSF 07 0	0.77	1.72	5.11	54.06	0.17	3.83	14.19	0.67	1.07	13.02	7.14	101.75
TSF 08 0	1.21	2.01	5.51	47.51	0.17	3.72	16.85	0.64	1.79	17.10	3.85	100.35
TSF 09 0	0.88	1.75	8.34	47.06	0.20	3.46	12.53	0.54	1.42	21.16	3.94	101.30
TSF 10 0	0.47	1.76	7.96	50.91	0.15	3.44	15.18	0.61	1.47	16.19	3.02	101.15
TSF 11 0	0.71	1.75	6.22	49.59	0.36	3.65	13.99	0.67	1.69	18.61	3.63	100.86
TSF 12 0	0.72	1.62	5.95	51.84	0.15	2.98	13.15	0.55	0.97	17.75	5.10	100.77
TSF 13 0	0.56	1.67	8.59	50.64	0.15	3.11	13.15	0.58	0.98	16.04	5.14	100.60

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Table C-2 Major elements in mine tailing samples at depth 0-1 m. from Akara Mine.

Sample	Na_2O	MgO	Al_2O_3	SiO_2	P_2O_5	K_2O	CaO	TiO_2	MnO	Fe_2O_3	LOI	TOTAL
	%											
TSF 02 1	0.26	1.48	23.76	45.57	0.24	2.74	10.27	0.45	0.97	12.06	4.11	101.92
TSF 03 1	0.51	1.85	6.87	47.93	0.16	3.22	14.90	0.73	1.12	18.57	5.23	101.09
TSF 04 1	0.80	1.91	8.65	42.76	0.16	3.35	17.79	0.77	1.58	18.55	4.09	100.42
TSF 05 1	0.49	1.72	23.93	44.79	0.26	2.42	11.08	0.46	1.02	11.09	3.52	100.76
TSF 06 1	1.02	1.70	7.60	47.58	0.16	2.77	15.12	0.67	1.38	18.80	3.65	100.44
TSF 07 1	0.64	1.91	6.19	49.25	0.18	3.41	15.87	0.63	1.28	17.27	4.64	101.26
TSF 08 1	0.60	1.99	6.28	50.89	0.17	3.35	16.10	0.73	1.29	14.80	5.73	101.93
TSF 09 1	0.79	1.80	6.12	48.30	0.20	2.97	12.69	0.63	1.13	20.05	7.23	101.91
TSF 10 1	0.24	1.48	22.97	44.47	0.23	2.77	13.91	0.48	1.05	11.05	3.14	101.79
TSF11 1	0.70	1.70	8.87	50.46	0.15	3.25	14.78	0.67	1.26	15.09	4.94	101.88
TSF 12 1	0.57	1.71	6.51	53.68	0.16	3.17	15.68	0.65	1.10	13.53	5.02	101.76
TSF 13 1	0.26	1.38	23.82	47.05	0.23	2.50	13.38	0.42	0.98	8.40	3.74	102.16

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Table C-3 Major elements in mine tailing samples at depth 1-2 m. from Akara Mine.

Sample	Na_2O	MgO	Al_2O_3	SiO_2	P_2O_5	K_2O	CaO	TiO_2	MnO	Fe_2O_3	LOI	TOTAL
	%											
TSF 02 2	0.30	1.64	25.19	46.58	0.27	3.66	8.01	0.52	1.12	9.06	5.02	101.37
TSF 03 2	0.83	1.96	7.19	45.79	0.15	5.65	9.57	0.61	1.11	25.13	3.73	101.72
TSF 04 2	0.55	1.78	8.23	50.54	0.35	3.53	15.36	0.52	1.20	15.34	3.35	100.74
TSF 05 2	0.46	2.01	6.40	50.83	0.18	3.49	12.74	0.67	1.20	18.03	4.93	100.95
TSF 06 2	0.30	1.46	26.52	42.02	0.25	2.24	11.74	0.45	1.04	10.15	5.22	101.39
TSF 07 2	0.88	1.67	8.77	46.47	0.15	3.14	13.94	0.69	1.33	20.18	3.91	101.13
TSF 08 2	0.25	1.44	23.98	46.69	0.24	2.92	6.82	0.41	1.46	14.81	2.02	101.04
TSF 09 2	0.73	1.69	6.79	46.96	0.14	3.62	15.62	0.56	1.38	19.89	3.94	101.32
TSF 10 2	0.21	1.43	24.97	45.25	0.25	3.31	8.46	0.54	0.89	12.72	3.11	101.13
TSF11 2	0.41	1.97	7.45	50.38	0.17	3.66	12.97	0.79	1.29	18.03	3.57	100.68
TSF 12 2	0.56	1.60	7.39	54.17	0.18	3.52	11.01	0.55	1.01	16.00	4.81	100.80
TSF 13 2	0.92	1.70	10.54	45.61	0.15	3.39	14.17	0.65	1.46	17.99	4.28	100.85

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Appendix D

Table D-1 Acid potential tests for Akara Gold Mine tailings at surface depth including, paste pH, acid-neutralization capacity (ANC), maximum potential acidity (MPA), net-acid producing potential (NAPP), net-acid generation (NAG), NAGpH, and total sulfur contents.

Sample	Paste pH	ANC (kg H_2SO_4/t)	Total S (%)	MPA (kg H_2SO_4/t)	NAPP (kg H_2SO_4/t)	NAGpH	Class
T1-0	7.92	123	2.72	83.23	-39.77	7.3	NAF
T2-0	7.98	138	3.92	119.95	-18.05	8.20	
T3-0	8.01	151	3.27	100.1	-50.94	7.4	NAF
T4-0	8.04	152	3.14	96.08	-55.92	9.1	NAF
T5-0	7.95	141	4.13	126.4	-14.62	7.6	NAF
T6-0	8.07	152	3.02	92.41	-59.59	7.9	NAF
T7-0	7.92	98	2.68	82.01	-15.99	8.1	NAF
T8-0	8.08	142	3.43	105	-37.04	7.5	NAF
T9-0	7.94	172	3.14	96.08	-75.92	7.9	NAF
T10-0	8.01	111	2.41	73.75	-37.25	7.2	NAF
T11-0	7.99	147	4.48	137.1	-9.91	9.1	NAF
T12-0	8.2	145	3.86	118.1	-26.88	8.4	NAF
T13-0	7.68	111	2.85	87.21	-23.79	7.3	NAF

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Table D-2 Acid potential tests for Akara Gold Mine tailings at depth 0-1 m including, paste pH, acid-neutralization capacity (ANC), maximum potential acidity (MPA), net-acid producing potential (NAPP), net-acid generation (NAG), NAGpH, and total sulfur contents.

Sample	Paste pH	ANC (kg H_2SO_4/t)	Total S (%)	MPA (kg H_2SO_4/t)	NAPP (kg H_2SO_4/t)	NAGpH	Class
T2-1	7.84	148	3.06	93.64	-54.36	8.2	NAF
T3-1	8.07	149	4.04	123.6	-25.38	8.3	NAF
T4-1	8.01	113	2.6	79.56	-33.44	8.7	NAF
T5-1	7.92	133	3.52	107.7	-25.29	8.1	NAF
T6-1	7.98	109	3.16	96.7	-12.3	8.4	NAF
T7-1	7.96	128	3.63	111.1	-16.92	8.6	NAF
T8-1	8.04	148	3.21	98.23	-49.77	8.5	NAF
T9-1	8	146	2.65	81.09	-64.91	7.7	NAF
T10-1	8.13	169	2.71	82.93	-86.07	8	NAF
T11-1	7.87	143	3	91.8	-51.2	9.3	NAF
T12-1	7.64	118	3.1	94.86	-23.14	7.6	NAF
T13-1	7.84	124	2.79	85.37	-38.63	8.4	NAF

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Table D-3 Acid potential tests for Akara Gold Mine tailings at depth 1-2 m including, paste pH, acid-neutralization capacity (ANC), maximum potential acidity (MPA), net-acid producing potential (NAPP), net-acid generation (NAG), NAGpH, and total sulfur contents.

Sample	Paste pH	ANC (kg H_2SO_4/t)	Total S (%)	MPA (kg H_2SO_4/t)	NAPP (kg H_2SO_4/t)	NAGpH	Class
T2-2	8.01	134	2.69	82.31	-51.69	8.6	NAF
T3-2	7.87	143	3.76	115.1	-27.94	9.4	NAF
T4-2	7.92	125	2.65	81.09	-43.91	7.4	NAF
T5-2	7.83	121	3.01	92.11	-28.89	7.3	NAF
T6-2	8.01	141	2.95	90.27	-50.73	8.7	NAF
T7-2	7.93	136	3.46	105.9	-30.12	8.2	NAF
T8-2	7.89	123	2.98	91.19	-31.81	8.7	NAF
T9-2	8.04	158	2.32	70.99	-87.01	9.2	NAF
T10-2	8.96	136	3.12	95.47	-40.53	8.5	NAF
T11-2	7.74	150	3.53	108	-41.98	8.7	NAF
T12-2	7.94	136	3.41	104.3	-31.65	7.5	NAF
T13-2	8	132	3.06	93.64	-38.36	8.2	NAF

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Appendix E

NAG Test

The NAG test is used to refine the results of the theoretical ABA Tests. It provides a direct assessment of the potential for a material to produce acid after a period of exposure to the air and weathering. This procedure is aimed to determine that how much the net acid remaining , if any, after complete oxidation of the materials with hydrogen peroxide and allowing complete reaction of the acid formed with the neutralizing components of the material.

Sample Preparation

Drill core and bulk rock samples should be crushed to nominal 4 mm and a sub sample pulverized to approximately 200 Mesh (<75 µm). Tailing and process residue samples can be tested 'as received'.

Reagents

Reagent 1: H_20_2 - BDH 'Analar' Analytical Reagent 30% w/v (100 V), or equivalent, diluted 1:1 with deionized H_20 to 15%. (Refer to Note 1).

Reagent 2: NaOH -0.50 M Standardized Solution.

Reagent 3: NaOH - 0.10 M Standardized Solution.

Procedure

The procedure has a step by following this;

1. Add 250 ml of 15% H_2O_2 to 2.5 g of sample powder (tailing or pulverized waste rock) in a 500 ml wide mouth conical flask
2. Cover conical with a watch glass and place in well ventilated area or in a fume hood .Notice that before commencing test, the H_2O_2 is at room temperature.
3. Allow sample to react until 'boiling' or effervescing ceases. Heat sample on hot plate and gently boil until effervescence stops or for a minimum of 2 hours.

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(Do not allow sample to boil dry – add deionized water if necessary. pH and Eh (if possible) values should be recorded approximately every 15 minutes for the first 2 hours and periodically for the next 24 hours, or until values have reached equilibrium)

4. Allow solution to cool to room temperature then record final pH (NAGpH).
5. Rinse the sample that has adhered to the sides of the flask down into the solution with deionized water.
6. Add deionized water to give a final volume of 250 mL.
7. Titrate solution to pH 4.5 while stirring with appropriate NaOH concentration based on final NAG solution pH as follows:

NAG Solution pH Reagent NaOH Concentration

NAG Solution pH	Reagent	NaOH concentration
>2	3	0.1 M
<2	2	0.5 M

Calculation

Net Acid Generation

NAG = 49 x V x M/W

Where;

NAG = net acid generation (kg H_2SO_4/tonne)

V = volume of base NaOH titrated (ml)

M = molarity of base NaOH (moles/l)

W = weight of sample reacted (g)

NOTE: If NAG value exceeds 25 kg H_2SO_4 per tonne, repeat using a 1.00 g samples.

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Appendix F

Acid-Base Accounting (NAPP Test)

The amount of neutralizing bases, including carbonates, present in overburden materials is found by treating a sample with a known excess of standardized hydrochloric acid. The sample and acid are heated to insure that the reaction between the acid and the neutralizers goes to completion._The calcium carbonate equivalent of the sample is obtained by determining the amount of unconsumed acid by titration with standardized sodium hydroxide.

Chemicals

1. Carbon dixide-free water. Heat distilled water just to boiling in a beaker. Allow to cool slightly and pour into a container equipped with ascarite tube. Cool to room temperature before using.

2. Hydrochloric acid (HCl) solution, 0.1 N, certified grade (Fisher So-A-54 or equivalent).

3. Sodium hydroxide (NaOH), approximately 0.5 N: Dissolve 20.0 g of NaOH pellets in carbon dioxide-free water and dilute to 1 liter. Protect from CO_2 in the air with ascarite tube. Standardize solution by placing 50 ml of certified 0.1 N HCl in a beaker and titrating with the prepared 0.5 N NaOH until a pH of 7.00 is obtained. Calculate the Normality of the NaOH using the following equation:

$N_2 = (N_1V_1) / V_2$,

Where;

V_1 = Volume of HCl used.

N_1 = Normality of HCl used.

V_2 = Volume of NaOH used.

N_2 = Calculated Normality of NaOH

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4. Sodium hydroxide (NaOH) approximately 0.1 N: Dilute 200 ml of 0.5 N NaOH with carbon dioxide-free water to a volume of 1 liter. Protect from CO_2 in air with ascarite tube. Standardize solution by placing 20 ml of certified 0.1 N HCl in a beaker and titrating with the prepared 0.1 N NaOH until a pH of 7.00 is obtained. Calculate the Normality of the NaOH.
5. Hydrochloric acid (HCl), approximately 0.5 N: Dilute 42 ml of concentrated HCl to a volume of 1 liter with distilled water. Standardize solution by placing 20 ml of the known Normality NaOH prepared in a beaker and titrating with prepared HCl until a pH of 7.00 is obtained.

   Calculate the Normality of the HCl using the following equation:

   $N_1 = (N_2V_2) / V_1$,

   Where;

   V_2 = Volume of NaOH used.

   N_2 = Normality of NaOH used.

   V_1 = Volume of HCl used.

   N_1 = Calculated Normality of HCl.
6. Hydrochloric acid (HCl), approximately 0.1 N: Dilute 200 ml of 0.5 N HCl to a volume of 1 liter with distilled water. Standardize solution as before, but use 20 ml of the known Normality NaOH
7. Hydrochloric acid (HCl), 1 part acid to 3 parts water: Dilute 250 ml of concentrated HCl with 750 ml of distilled water.

Materials

1. Flasks, Erlenmeyer, 250 ml
2. Burst, 100 ml (one required for each acid and one for each base).

3. Hot plate, steam bath can be substituted.
4. pH meter (Corning Model 12 or equivalent) equipped with combination electrode.
5. Balance, can be read to 0.01 g.

Procedure

1. Place approximately 0.5 g of sample (less than 60 mesh) on a piece of aluminum foil.
2. Add one or two drops of 1:3 HCl to the sample. The presence of $CaCO_3$ is indicated by a bubbling or audible "fizz."
3. Rate the bubbling or "fizz" in step 2 as indicated in Table 2.
4. Weigh 2.00 g of sample (less than 60 mesh) into a 250 ml Erlenmeyer flask.
5. Carefully add HCl indicated by Table 2 into the flask containing sample.
6. Heat nearly to boiling, swirling flask every 5 minutes, until reaction is complete. NOTE: reaction is complete when no gas evolution is visible and particles settle evenly over the bottom of the flask.
7. Add distilled water to make a total volume of 125 ml.
8. Boil contents of flask for one minute and cool to slightly above room temperature. Cover tightly and cool to room temperature. CAUTION: Do not place rubber stopper in hot flask as it may implode upon cooling.

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Table 2: Volume and Normality of Hydrochloric Acid Used for Each Fizz Rating (Sobek et al., 1978)

Fizz Rating	HCl (ml)	HCl (Normality)
None	20	0.1
Slight	40	0.1
Moderate	40	0.5
Strong	80	0.5

9. Titrate using 0.1 N NaOH or 0.5 N NaOH (concentration exactly known), to pH 7.00 using an electrometric pH meter and burette. The concentration of NaOH used in the titration should correspond to the concentration of the HCl used in step 5. NOTE: Titrate with NaOH until a constant reading of pH 7.0 remains for at least 30 seconds.

10. If less than 3 ml of the NaOH is required to obtain a pH of 7.0, it is likely that the HCl added was not sufficient to neutralize all of the base present in the 2.00 g of sample. A duplicate sample should be run using the next higher volume or concentration of acid as indicated in Table 2.

11. Run a blank for each volume or normality using steps 5, 7, 8, and 9

Calculations

1. Constant (C) = (ml acid in blank) / (ml base in blank).

2. ml acid consumed = (ml acid added) - (ml base added x C).

3. Tons $CaCO_3$ equivalent / thousand tons of material = (ml of acid consumed) x (25.0) x (N of add).

BIOGRAPHY

My name is Miss Chulalak Changul. I was born on January 22, 1979 in Uthaithani, Thailand. I graduated the Bachelor degree (B. Eng) from Kasetsart University in Water Resource Engineering (2000). After that I worked for 1 year under the King project “Hydrodynamic Flow Measurement Math model in Chow Phaya river basin by ISIS”. Then I continued my study for Master degree of Environmental Engineering (M. Eng) in Department of Environmental Engineering at King Mongkut’s University of Technology Thonburi, Bangkok (2003). In 2004, I continued my study in Ph. D. International program at National Center of Excellence for Environmental and Hazardous Waste Management (NCE-EHWM), Chulalongkorn University, Thailand.

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