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Characterisation and applications of iron ore tailings in building and construction projects

Francis Atta Kuranchie
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Characterisation and Applications of Iron Ore Tailings in Building and Construction Projects

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This thesis is presented in fulfilment of the requirements for the
Degree of Doctor of Philosophy
May 2015



**School of Engineering
Faculty of Health, Engineering and Science
Edith Cowan University**

USE OF THESIS

The Use of Thesis statement is not included in this version of the thesis.

ABSTRACT

The mine tailings are generated as the wastes worldwide as a result of exploration, excavation, blasting, beneficiation and extraction of mineral ores. In Western Australia, due to the extensive mining activities and increasing low grade ores, there is generation of mine tailings in large quantities, which could lead to environmental and disposal problems. The common practice of handling the tailings are to store them in tailing dams or as stockpiles near mine sites. Limited quantities are sometimes used as backfills and other applications. The utilisation of tailings in building and construction projects, which may consume a large volume of wastes, have not been explored extensively so far. Additionally, the understanding of chemical composition-based utilisation of tailings has very limited investigation.

In the present research, a critical review of the literature was made focusing on the utilisation of mine tailings in large quantities. Experiments have been conducted by developing a methodology to characterise the tailings based on the relationship that exists between electrical resistivity and the relative density of the tailings in dry and wet conditions. The results show that the electrical resistivity of iron ore mine tailings produced in Western Australia in dry condition ranges from 11 k Ω m in a more dense state to 19 k Ω m in a very loose state, while that in fully saturated condition ranges from 20 Ω m for a very dense state to 31 Ω m in a very loose state.

The laboratory investigation has been conducted to utilise iron ore tailings to produce geopolymer bricks. The sized tailings were mixed with sodium silicate solution used as an activator to form a paste. The paste was moulded and cured for different durations. It was found that the geopolymer bricks produced from iron ore tailings could have a compressive strength as high as 50.35 MPa. This is either superior or similar to international standard specifications for conventional bricks. Additionally, the new bricks will be more economical than conventional bricks with potential cost reduction of 36.8%.

The research has also investigated the utilisation of iron ore mine tailings to replace conventional aggregates in concrete. 100% of both fine and coarse conventional aggregates were replaced with tailings in the mixed design. The concrete mix was casted into moulds and cured. It was found that the compressive strength of the concrete with tailings aggregates at 28 days was 36.95 MPa which shows an improvement of 11.56% over the concrete with conventional aggregates. Additionally, the new concrete met all other requirements for quality assessment of concrete.

Finally, the research has conducted investigation into load-settlement behaviour of iron ore tailings to be considered as a structural fill material. The experiment was conducted in a model test tank in the laboratory varying the relative density of the tailings. It was found that the load-bearing capacity is 22 times higher, and the stiffness is 13.5 times higher than their values for conventional fill materials.

The declaration page
is not included in this version of the thesis

ACKNOWLEDGEMENTS

A Chinese proverb and a quote from Albert Schweitzer respectively say, “when eating bamboo sprouts, remember the man who planted them” and “we should be thankful for those people who rekindle the inner spirit”. This Ph.D journey has come this far because of good guidance and valuable support I have received from many people for which I am thankful to them.

Firstly, I would like to express my heartfelt gratitude to my principal supervisor Associate Professor Sanjay Kumar Shukla, who gave me professional and personal guidance throughout this journey. He has been very inspirational to me throughout my Ph.D studies in all aspects and I have learnt a lot from him. His dedication, encouragement and professional supervision to this work is outstanding and immeasurable. It was a blessing to me to have Sanjay as my Ph.D supervisor. This study would not have reached this level without his intelligent guidelines and professional advice.

I am very much grateful to my associate supervisor Professor Daryoush Habibi, for his encouragement and support throughout my work. His suggestions and comments for all the papers have brought much improvement to this thesis. It was a great privilege to have him as one of my supervisors. Dr. Alireza Mohyeddin Kermani was also very helpful during my work on concrete and during our research meetings. His expert advice and quality comments has further improved this work. I greatly appreciate you for this dedicated help.

I would like to express my profound gratitude to all the technical staff at the school of Engineering, Edith Cowan University. In particular, Dr Mohamed Ismail, Dr Xiaoli Zhao, Adrian Styles and Slavko Nikolic for their technical support and expert advice in setting up the laboratory arrangements. Their valuable support to this work is highly appreciated. My special thanks to the late Jim Buchanan; may his soul rest in peace, his expert guidance and support in getting the components and setting up the resistivity experiment in the laboratory was very crucial to my work. He is no more but he will be forever remembered.

I also wish to acknowledge the valuable service rendered to me by all the administrative staff, particularly Muriel Vaughan and Myra Kendall. Their excellent service aided this work to be completed in timely manner and all of them have been the brain behind the success of this work. Their dedicated service is greatly appreciated.

I am also grateful to Greg Hudson and David Eastman of Mount Gibson Iron, Perth who assisted this work by providing the iron ore mine tailings for the research. I would also like to extend my appreciation to Lee Gough, Gary Wheeler, and Dave Farrar, all of

MinAnalytical Laboratory Services Australia Pty Ltd, for their assistance in conducting the XRF analysis with their facilities free of charge. My appreciation also goes to Dr. Aaron Dodd, a senior research officer at the University of Western Australia who provided his institution's facilities for conducting the XRD test on my behalf. I highly appreciate and very grateful to the sales and marketing manager of Coogee Chemicals Pty Ltd., Liane Lied-Cordruwisch, who supplied me with sodium silicate solution free of charge for the research.

I would like to thank Steve Hoban, Sam McDonald, Harley Davis and Dave Hodgson all of Metallurgy Property Limited, Perth; for providing me with mine tailings, distilled water, sodium silicate solution and also making their laboratory facilities available for me as well as conducting the XRF for me during the preliminary studies of this research. Their guide, kindness and willingness to support academic research are highly appreciated.

I am very grateful to Edith Cowan University, for giving me a Ph.D position and granting me International Postgraduate Research Scholarship (IPRS) to undertake the study and an ECU International Stipend (ECUIS) to cater for my living expenses throughout the period. I wish to express my sincere thanks to Prof Joe Luca, the ECU Graduate Research School and Miss Clare Ashby; ECU Scholarship officer, for their respective support.

I wish to thank my wife, Issabella so much for her understanding during this journey. Her support and encouragement have been crucial for this success. My children have also conducted themselves very well even when I have to be away from them for a long time because of the busy schedule the research demands. This went all the way to make this research a success. Yaa, Kwame and Oheneba your patience has paid off and you are welcome to spend more time with Dad now.

I wish to thank all my family members far away in Ghana and my special thanks also to my uncle Mr. Oheneba Afikura in Canada and all my friends; Dr. S. K. Appiah, Dr. K. Adusei, Dr. Patrick Aboagye-Sarfo, Kwasi, Eric, Esther, Monir and Wisdom for their continuing support and encouragement in this journey. Finally, I wish to thank all the people, I have had the opportunity to meet during my work. Without the help and the friendship of these people this success will never be complete. May God richly bless all those who have been part of this success, either directly or indirectly.

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LIST OF SYMBOLS

Basic SI units are given in parentheses.

B	Width of footing (m)
C_c	Coefficient of curvature (dimensionless)
C_u	Coefficient of uniformity (dimensionless)
D_{10}	Effective particle size (m)
D_{30}	Particle size corresponding to 30% passing (m)
D_{60}	Particle size corresponding to 60% passing (m)
D_r	Relative density (%)
k_s	Modulus of subgrade reaction (N/m^3)
q	Load-bearing pressure (N/m^2)
s	Settlement of the footing (m)
$\rho_{d\max}$	Maximum dry density (kg/m^3)
$\rho_{d\min}$	Minimum dry density (kg/m^3)
ρ	Electrical resistivity (Ωm)
ρ_d	Dry density (kg/m^3)
ρ_e	Experimental electrical resistivity (Ωm)
λ	Electrical resistivity correction factor (dimensionless)
a	Electrode spacing (mm)
ΔV	Potential difference (v)
I	Electrical current (Amps)
\vec{J}	Current density (dimensionless)
\vec{E}	Electric field (dimensionless)
q_u	Ultimate load-bearing capacity (N/m^2)
w_a	water absorption (%)
w_1	weight of brick after curing (kg)
w_2	weight of surface dry brick (kg)
R	Electrical resistance (Ω)
A	Surface area (m^2)
L	Length of wire across the brick (m)
T	Indirect tensile strength (MPa)
P	Maximum applied load (kN)
L_1	Length of cylinder (m)

D	Diameter (m)
D_f	Depth of foundation (m)
I_b	Bearing capacity improvement factor (%)
q_{uo}	Ultimate load-bearing capacity with respect to surface footing (Pa)
k_{so}	Modulus of subgrade reaction with respect to surface footing (N/m ³)
I_s	Modulus of subgrade reaction improvement factor (%)

LIST OF ABBREVIATIONS

WA	Western Australia
AMD	Acid mine drainage
CBR	California bearing ratio
SUDAS	Statewide urban design specifications
FE	Finite element
BS	British Standard
AS	Australian Standard
DEWHA	Department of the Environment, Water, Heritage and the Arts
USCS	Unified soil classification system
SP	Poorly graded sand
AC	Alternating current
DC	Direct current
XRF	X-Ray fluorescence
XRD	X-Ray diffraction
LOI	Loss on ignition
UCS	Unconfined compressive strength
NIS	Nigerian Standard Specifications
ASTM	American Society for Testing and Materials
SW-SM	Well-graded sand-silty sand
PennState Matse	Pennsylvania State University Department of Material Science and Engineering
AU\$	Australian dollars
CLSM	Controlled low strength materials
SW	Severe weathering
MW	Medium weathering
NW	No weathering
NAF	Non acid forming
ANC	Acid neutralising capacity
ESP	Exchangeable sodium percentage
CCD	Counter current decantation
BREFs	Reference document on best available techniques

CHAPTER 1

INTRODUCTION

This chapter explains the problem being considered in this thesis and the importance of finding sustainable solution to address it. It includes the aim and the specific objectives of the thesis. This is followed by the scope and an explanation of how this thesis has been structured for easy understanding and convenience of the reader.

1.1 GENERAL

Western Australia (WA) is one of the Australian States which is bounded by the Indian Ocean to the north and west (The great Australian Bight) and Indian Ocean to the south; the Northern Territory to the north-east and South Australia to the south-east. WA is Australia's largest state and the largest sub-national state in the world. The land mass of WA is 2.5 million square kilometers (a third of the Australia's continent). The population is approximately 2.3 million (approximately 10% of Australia's total population) and it is mainly concentrated in the Perth metropolitan area; the capital of WA. The state's economy relies mainly on three sectors namely energy, minerals and metals and agriculture (Berkel, 2007). The mining sector is the most crucial one in the state because of the extensive mining activities and the attraction of many mining industries. This has contributed to the generation of large volume of mine wastes in the state. It has therefore, drawn a lot of attention to the general public and the Government.

Currently, there is no proper reporting system in place to know the quantity of mine tailings generated in the state. There is also insufficient data to enhance the assessment of information concerning the nature and quantity of controlled tailings produced in WA (Australia Government Department of the Environment, Water, Heritage and the Arts, 2009). This has made the quantification of the tailings in WA very difficult. However, from Government reports and reports from some WA mining companies on their production, reliable estimations can be made. For instance, according to the Western Australian mineral and petroleum statistics digest 2008/2009 (WA Department of Mines and Petroleum, 2009), there was a production of 316 million tonnes and 136 tonnes of iron ore and gold in WA for the year 2008/2009 respectively. These figures are also the yearly average production of these resources in the state. It has also been reported that the stripping ratio (waste to ore) of iron

ore production in WA is 2:1 (Price, 2004) and that of gold production in WA is 2.3:1 (Evolution Mining, 2014). From the information above, it can be estimated that about 632 million tonnes of iron ore mine tailings and about 312.8 tonnes of gold tailings are produced yearly in WA. The above quantities of tailings generation, especially from the production of iron ore are so huge and it brings a lot of concern as to how well the tailings should be handled.

Mine tailings in WA have become an important issue to be considered because of its production in large quantities due to large scale mining activities in WA. For instance, the state has been enlisted among the top 20 mining jurisdictions in the world and it is currently ranked as 11th out of 93 top mining jurisdictions in the world (WA Department of Mines and Petroleum, 2009). Others in the ranking are New Brunswick, Finland, Alberta, Wyoming, Quebec, Saskatchewan, Sweden, Nevada, Ireland, Yukon and Western Australia in that order. The state holds a large amount of mineral deposits such as iron ore, alumina, gold and many other mineral deposits. The mineral sector is therefore, very crucial in WA’s economy and it gives the state high revenue from the mineral exports (Alexander, 1988). This gives an indication of the extent and capacity of mining activities associated with the state. Figure 1.1 shows the mineralogical map of WA and it describes the various processing centers and the reserves of the mineral deposit in Western Australia (Geosciences Australia, 2012).

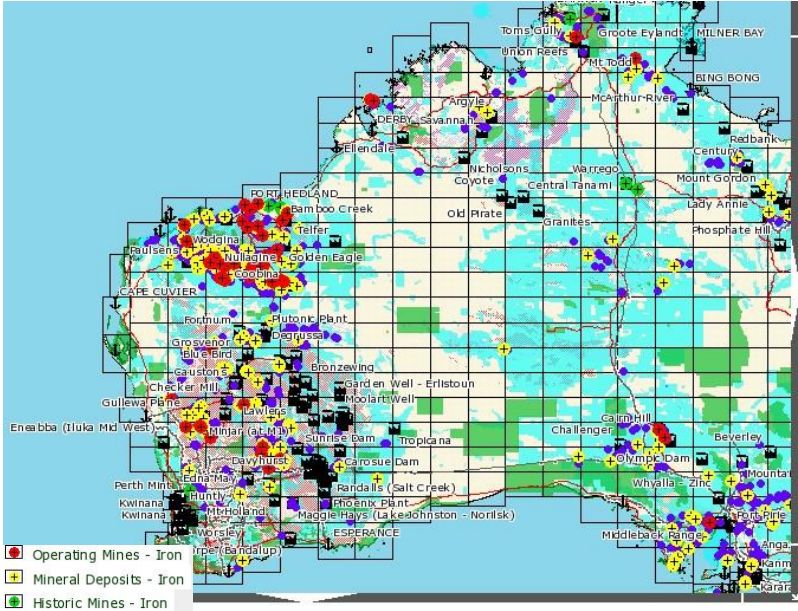


Figure 1.1: Mineralogical map of WA (after Geosciences Australia, 2012).

1.2 ORE PROCESSING AND GENERATION OF MINE TAILINGS

Each mining activity goes through a series of stages like exploration, excavation, beneficiation (crushing, grinding, and removal of gangue), extraction processes; hydrometallurgy which involves chemical leaching and use of reagents, electrometallurgy which involves electrolysis and pyrometallurgy which involves high temperature processes (Wills, 1992). Each of these stages in the mining operation is associated with the generation of tailings. This means that tailings generation cannot be dissociated from mining activities. Therefore the more WA Mines continue to grow and increase production mine wastes generation in the form of liquid, solid or gaseous emissions will continue to increase in the region. Some specific wastes generated from the mining processes are, waste rocks (rejects), mine overburden, fly ash and slag (Rampacek, 1982; Swane, 2009) and they are together termed as mine wastes or tailings. New processing technologies available currently make it possible to process and recover valuable minerals which could not be recovered previously. Despite the efforts and the current new technologies, wastes are still generated for example from the re-treatment of tailings.

The quantity of wastes produced from the mining activities depends on the type and grade of ore materials and the efficiency of the extraction or mining process used. What has enhanced the high volume waste generation from the mining activities is the treatment of low grade and complex ores which has become necessary as a result to meet the production volumes and market demands (Mudd, 2004).

1.3 MINE TAILINGS HANDLING AND POTENTIAL PROBLEMS

The disposal of the mine tailings continues to impose burden to the mining industries and the general public in terms of economic and environmental health respectively. Pollution concern is also one of the main issues to the mining industry. The industries concerned need to think of getting a near-by space for the tailings disposal through environmentally friendly and sustainable disposal methods. After successful disposal of the tailings, there is also the need to monitor and manage the tailings site (McKinnon, 2002). This ensures that the possibility of erosion, development of acid mine drainage (AMD) and dam failures are prevented (Yellishetty et al., 2008; Grangeia et al., 2011). One potential problem of the tailings is that they can release toxic and heavy metals into the environment. From the industries front, they are also confronted by mine wastes disposal cost which depletes a substantial amount of profit from production.

Previously, some mining companies employ direct discharge of their tailings into the sea (McKinnon, 2002). The current practice is that most of these tailings are pumped as a slurry with high pressure into tailings dam while others are stockpiled closer to mine sites (Wills, 1992). Very few of the mining companies send some part of their tailings to be used for land reclamation and as backfills to previously mined-out areas in open cast quarries and mines (Skarzynska, 1995).

The mine tailings handling problems in WA coupled with regulatory and legislation requirements, meeting community expectations and ensuring sustainable techniques of mining and mineral processing steps have made it necessary to consider a different ways of handling these wastes produced by the industries. Therefore, sustainable and economic options to handle mine tailings are very essential.

Among the various options available, reuse of these tailings is the best option of ensuring sustainable handling of the tailings in the region. Packey (2012) presented four economic scenarios on the benefits and profitability from mine tailings modification and its development into marketable products using a mathematical model. First, if the tailings are rejected, the firm will not engage in selling its modified wastes and its environmental rehabilitation cost remains the same. Second, the firm can decide to process the tailings into an acceptable product and give it away for free. This will be preferred because the decrease in the costs to rehabilitate the site is greater than the costs from producing the modified tailings product resulting in an overall decrease of the costs to the firm. Third, the market-based scenario where the tailings are modified into a marketable product with good price and this benefit the firm through a reduction of rehabilitation costs because of reduced volume of the mine tailings to handle and the extra income the tailings could generate. Lastly, there will be a generation of employment after mine closure through a modified tailings production into value added products.

Other available literature also report that some of these tailings could be used in civil engineering for building and construction; example (Skarzynska, 1995; Choi et al., 2009; Klauber et al., 2011; Kuranchie et al., 2014).

1.4 STATEMENT OF THE PROBLEM AND THE SIGNIFICANCE OF THE RESEARCH

WA is one of the major mining jurisdictions of the world and it has a lot of operating mining industries. This has resulted in the generation of large volume of mine tailings in the state. Space availability for the disposal of the tailings in the future will become a major problem

for the industries concerned. There are environmental pollution issues and the possibility of the disposed tailings causing acid mine drainage (AMD) and leaching of heavy metals to harm people and the environment.

Mine tailings that have been stored in tailings dam could have negative environmental impacts through geotechnical, physical and chemical instability. Pollution of ground and surface water by toxic substances such as dissolved metals, cyanide and sulfates are major negative impacts in which mine tailings stored in tailings dam bring to the environment. When the sulfates in particular are exposed to oxygen, they undergo chemical reaction through oxidation to form acid sulfate soils. This has potential of killing vegetation and seeping into ground water.

The mining industries concerned need to meet legislation requirements during mine wastes disposal from Governmental bodies. Mine wastes disposal, maintenance of tailings dam and monitoring add extra cost to the industries concerned and they need to reduce cost of production as well. The continuous accumulation of mine waste in the environment has contributed to the numerous environmental problems such as loss of land fertility, dust, erosion and effects on ecosystems. There could also be water and soil pollution. The worse situation is when tailings dams or heaps collapse due to high rainfall, rugged maintains and possible earthquakes which may have negative effects on the environment, human health and safety.

In other to eliminate the problems stated above, there is the need to explore and establish a sustainable reuse possibilities for mine tailings generated in WA. To increase mine tailings utilisation, there is the need to carry out more research in this area to find applications that will ensure high volume tailings utilisation.

The stakeholders in this industry have a part to play by involving themselves with researchers to implement the research outcomes that are readily feasible and commercialise the benefit of bringing a higher percentage of tailings into standard civil engineering applications for building and construction. The use of these mine tailings in civil engineering applications such as bricks manufacturing, concrete aggregates, embankment and structural fills are more attractive and with a greater feasibility.

Currently, utilisation of tailings is very poor. Any attempt to utilise the mine tailings will reduce the volume of tailings available and will further reduce the environmental effects to human and aquatic life. The positive side of incorporating tailings utilisation into mine operation systems is that it will open up an avenue of economic activity by enhancing the employability of rural communities, softening the impacts of mine closures on communities

and reduction of rehabilitation cost of mining companies after mine closure. The adoption of mine tailings utilisation will have sustainable contribution to the economy and environment as well. Therefore, the problem to solve is how to convert these tailings into value added products for civil engineering applications for building and construction projects. This will reduce or eliminate the problems mine tailings bring to the environment and human health. It will also ensure economic and environmental sustainability of the mining industries and provide cheaper alternative materials for building and construction projects.

1.5 AIM AND SPECIFIC OBJECTIVES OF THE RESEARCH

The aim of this research is *to characterise and identify sustainable and a high volume application possibilities for mine tailings in WA for building and construction projects to ensure economic and environmental sustainability.*

In an attempt to identify the best and feasible reuse options for mine tailings in WA for sustainable development, and to achieve the aim stated above based on the problem statement defined earlier, the research will have the following specific objectives;

- Analysing the literature to establish some reuse possibilities for mine tailings in WA
- Using electrical resistivity survey to develop a methodology for finding the electrical resistivity of WA iron ore mine tailings for its characterisation
- Investigating the feasibility of utilising WA iron ore mine tailings for the production of geopolymers
- Assessing the suitability of WA iron ore mine tailings as aggregates in concrete
- Considering WA iron ore mine tailings as an alternative materials for structural fills

1.6 RESEARCH SCOPE

The research specifically focuses on Western Australia (WA). This is because WA is the only state in Australia that has been enlisted as one of the top mining jurisdictions in the world due to its extensive mining operations. As a result, the state has accumulated high volumes of mine tailings and therefore an urgent attention is required to find sustainable handling for the tailings. In addition, proximity is another reason because Edith Cowan University is in Perth which is the capital of Western Australia. This afforded me the opportunity and it was easy for me to contact the officials in the mines for the obtaining the samples and other valuable information.

The research also focuses mainly on iron ore mine tailings. As mentioned earlier, there is generation of very high volume of iron ore mine tailings as compared to the others in the

state. The research also focuses on high volume utilisation of mine tailings. Therefore, iron ore mine tailings will immediately suite the aim of this research. Also the time limitation for the Ph.D journey could not permit me to look into other mine tailings though others could also have potential.

1.7 PUBLICATIONS BASED ON THE PRESENT WORK

During the progress of the research, attempts were made to prepare the thesis as research papers for submission to journals and conference proceedings to be considered for publication. The details of the published/accepted or submitted papers are as follows:

International Journals

1. **Kuranchie, F.A.**, Shukla, S.K. and Habibi, D. (2015). Utilisation of iron ore mine tailings for the production of geopolymer bricks. *International Journal of Mining, Reclamation and Environment*, UK, DOI:10.1080/17480930.2014.993834.
2. **Kuranchie, F.A.**, Shukla, S.K. and Habibi, D. (2015). Electrical resistivity of iron ore mine tailings produced in Western Australia. *International Journal of Mining, Reclamation and Environment*, UK, DOI: 10.1080/17480930.2014.941551.
3. **Kuranchie, F.A.**, Shukla, S.K., Habibi, D., Zhao, X. and Kazi, M. (2014). Studies on electrical resistivity of Perth Sand. *International Journal of Geotechnical Engineering*, UK, Vol. 8, No. 4, pp. 449-457, DOI 10.1179/1939787913Y.0000000033.
4. **Kuranchie, F.A.**, Shukla, S.K., Habibi, D. and Mohyeddin, A.K. (2015). Utilisation of iron ore tailings as aggregates in concrete. *Cogent Engineering*, UK (under review).
5. **Kuranchie, F.A.**, Shukla, S.K., Habibi, D. and Kazi, M. (2015). Load-settlement behaviour of a strip footing resting on iron ore tailings as a structural fill. *International Journal of Mining Science and Technology*, China (under review).

Conference Proceedings

6. **Kuranchie, F.A.**, Shukla, S.K. and Habibi, D. (2013). Mine wastes in Western Australia and their suitability for embankment construction. *Proceedings of the GeoCongress 2013: Stability and Performance of Slopes and Embankments III, ASCE Joint Committee on Sustainability Practices*, 3-7 March 2013, San Diego, California,

USA, GSP 231, Editors: C. Meehan, D. Pradel, M.A. Pando and J.F. Labuz, pp. 1450-1459, DOI: 10.1061/9780784412787.145.

7. **Kuranchie, F.A.**, Shukla, S.K. and Habibi, D. (2013). Study on mine wastes as potential resource for brick manufacturing in Western Australia. *Research, Development and Practice in Structural Engineering and Construction*, Editors: V. Vimonsatit, A. Singh and S. Yazdani, Research Publishing, Singapore, *Proceedings of the First Australasia and South East Asia Conference in Structural Engineering and Construction*, 28 November – 2 December 2012, Perth, Australia, pp. 819 – 823, DOI: 10.3850/9789810879204_AAE120319.

1.8 STRUCTURE AND THE ORGANISATION OF THE THESIS

The introduction is followed by the rest of the chapters in the thesis with each chapter aiming to achieve a specific objective as stated above. Chapter 2 is the general overview of significant and selected literature which is relevant and current to this study. This helped to identify the gaps in literature and what is needed to be done in order to close the gap.

Chapter 3 specifically analyses the literature to identify the reuse possibility of mine tailings for brick manufacturing in Western Australia. This chapter, except with limited modification in layout for consistency in the thesis was presented orally and published as a peer reviewed conference paper in the proceedings of the first Australasian and South-East Asia Structural Engineering and Construction Conference (ASEA-SEC 1) in November 28 – December 2, 2012, in Perth, Western Australia.

Chapter 4 analyses the literature to identify the reuse possibility of mine tailings as an embankment material in Western Australia. This chapter also presents a schematic view of a proposed embankment constructed with mine tailings. This chapter, except with limited modification in layout for consistency in the thesis was presented orally and published as a peer reviewed conference paper in the proceedings of the Geo-Congress 2013, Stability and performance of slopes and embankments III, March 3 – 6, 2013 in San Diego, California, U.S.A.

In chapter 5, a methodology was developed through a practical laboratory investigation to determine the electrical resistivity of a geo-material using electrical resistivity survey. In this chapter, Perth local sand commonly used for various construction works was used instead of iron ore mine tailings. This is because there is virtually no information and no established values of the electrical resistivity of iron ore mine tailings. Therefore, the local sand was used which has an established trend of the electrical resistivity values in the

literature. This made it possible to verify the methodology that has been developed by comparing the trend of the electrical resistivity values obtained using this methodology and the reported values in the literature on the local sand. This chapter, except with limited modification in layout for consistency in the thesis was published as a journal article in International Journal of Geotechnical Engineering, United Kingdom.

Chapter 6 is a practical laboratory investigation which utilised the verified methodology developed in chapter five to determine the electrical resistivity of iron ore mine tailings produced in Western Australia. This will provide a means to characterise the behavior and the engineering properties of the iron ore mine wastes during preliminary studies in building and construction projects. This chapter, except with limited modification in layout for consistency in the thesis has been published online ahead of print as a journal article in the International Journal of Mining, Reclamation and Environment, United Kingdom.

Chapter 7 is a practical laboratory investigation for the utilisation of iron ore mine tailings from WA for the production of geopolymer bricks. This chapter, except with limited modification in layout for consistency in the thesis has been published online ahead of print as a journal article in the International Journal of Mining, Reclamation and Environment, United Kingdom.

Chapter 8 is a practical laboratory investigation utilising WA iron ore mine tailings as aggregates in concrete. This chapter, except with limited modification for consistency in the thesis has been submitted to the Journal of Cogent Engineering, United Kingdom to be considered for publication and it is currently under review.

Chapter 9 is a practical laboratory investigation utilising WA iron ore mine tailings as an alternative material for structural fills. This chapter, except with limited modification for consistency in the thesis was submitted and it is currently under review at the International Journal of Mining Science and Technology, China.

Chapter 10 outlines the general conclusions from the previous chapters and spells out the contribution to knowledge added to the literature as a result of this research. This chapter also uses the conclusions to suggest potential future research path as a secondary consideration to this primary research.

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CHAPTER 2

LITERATURE REVIEW

Each chapter in this thesis has exhausted all the relevant literature on all the specific applications of the mine tailings as mentioned in the specific objectives. However, this chapter presents the general overview of the most current literature selected which is relevant to this research. This helped to identify the research gaps which this current research focuses on.

2.1 GENERAL

WA is endowed with a lot of mining resources. Some companies have chosen to focus on a specific type of mine while others have multiple focuses dealing in most of the mining resources. According to the Department of Mines and Petroleum, (WA Department of Mines and Petroleum, 2009) the state's mineral statistics and the most important mining resources in the state are:

- Iron ore mines
- Gold mines
- Petroleum wells
- Alumina mines
- Nickel mines
- Base metals mines
- Mineral sand mines
- Diamond mines

Iron Ore mining sector in WA is one of the most valuable resource sectors in the state. It accounts for 47% of the total value of the state's resources reaching a total production of 316 million tonnes in 2008/2009 year (WA Department of Mines and Petroleum, 2009). Before the 'boom era' toned down recently, due to the consistent reduction in the price of iron ore in the world market, this production tonnage has been the yearly average production in the state. Iron ore is the main finished product from WA iron ore mines. The finished product (iron ore) is exported to other countries for further processing to extract the steel or the iron. China dominates WA's iron ore exports reaching 64% or \$ 21 billion of the overall amount shipped in 2008/2009 year. The rest are patronized by Japan, South Korea, Taiwan and

Europe (WA Department of Mines and Petroleum, 2009). The patronage in the exports of WA iron ore is shown in the Figure 2.1.

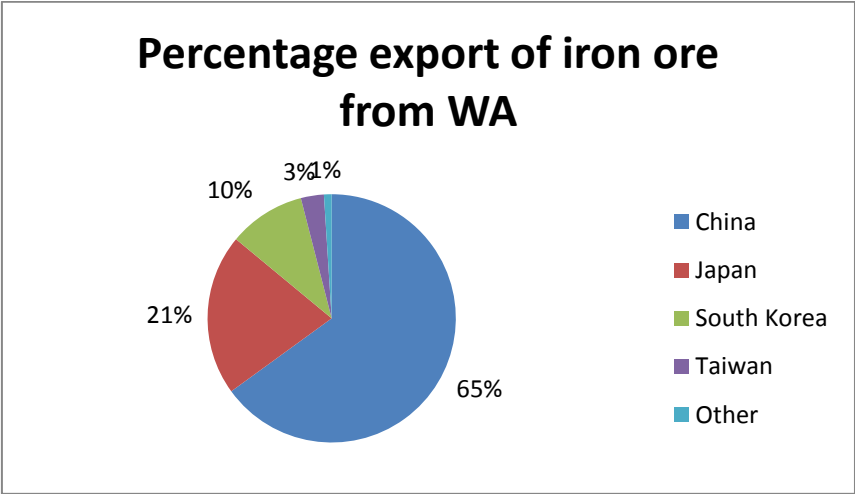


Figure 2.1: Iron ore exports in WA - total value \$ 33.4 billion (after WA Department of Mines and Petroleum, 2009).

The main producers of iron ore in the state are Rio Tinto iron ore, Hope Downs iron ore and BHP Billiton iron ore which together produce about 90% of the state’s iron ore. Rio Tinto owns the Hamersley iron ore pty. Ltd. which consists of six mines as Brockman, Marandoo, Mt. Tom price, Paraburdoo, Yandicoogina and Nammuldi. Hamersley iron ore pty. Ltd. also owns 60% of Channar mine and 54% Eastern Range mine and have further two mines; Western Turner Syncline Iron ore mine and Brockman 4 iron ore mine which are under construction. BHP Billiton Iron ore mine operates 7 mines including one of the largest single pit open-cut iron ore mines in the world which is Whaleback mine found in Newman. Other smaller producers of iron ore in the state are Cliffs Natural resources, Mount Gibson Mining ltd, Midwest Corporation ltd, Crosslands Resources ltd and Atlas iron ltd.

For the past 40 years all iron ore mined in WA have been hematite ore (Fe_2O_3); however, the state has other large resource of magnetite ore (Fe_3O_4) but hematite is always considered past magnetite with the reason that the concentration of hematite ore is considered more cost effective than magnetite ore as saleable products.

Gold industry is also one of the valuable resource sectors in the state. The state Gold production was 136 tonnes or 4.4 million ounces as a against 218 tonnes or 7 million ounces in the whole of Australia in 2008-09 year with the state’s output estimated to be around 62%

of Australia’s total Gold production. The main producers of Gold in Western Australia (WA) are;

- Golden Mile (Kalgoorlie consolidated Gold Mines pty Ltd (KCGM) – 20.0 tonnes in 2008 -09
- Telfer Gold (Newcrest Mining Limited) – 18.1 tonnes in 2008 – 09
- St. Ives (Gold Fields Ltd) – 13.4 tonnes in 2008 – 09
- Sunrise Dam (AngloGold Ltd) – 12.2 tonnes in 2008 – 09
- Jundee Nimary (Newmont Mining Corp) – 11.8 tonnes in 2008 – 09
- Kanowna Belle (Placer Dome Inc) – 7.0 tonnes in 2008 – 09
- Agnew (Goldfields Ltd) – 6.0 tonnes in 2008 – 09
- Marvel Loch (St. Barbara Limited) – 4.8 tonnes in 2008 – 09
- Plutonic (Barrick Gold Corp) – 4.4 tonnes in 2008 – 09
- Higginsville (Avoca Resources Limited) – 3.8 tonnes in 2008 – 09.

These companies listed above accounted for a total production of 75% of the state’s Gold production in 2008 – 09. Currently all Australia’s Gold are refined in Western Australia. Gold exports from the state amounted to \$ 16.8 Billion in 2008 – 09 of which 31% or \$ 5.2 billion was the actual production from Western Australia. The remaining 69% or \$ 11.66 billion (approximate) can be attributed to gold refined and exported in Western Australia but was actually produced from mining activities in other States, Territories and Overseas. United Kingdom Dominates the state’s Gold exports totaling 41% with others exported to other countries shown in the pie chart in Figure 2.2.

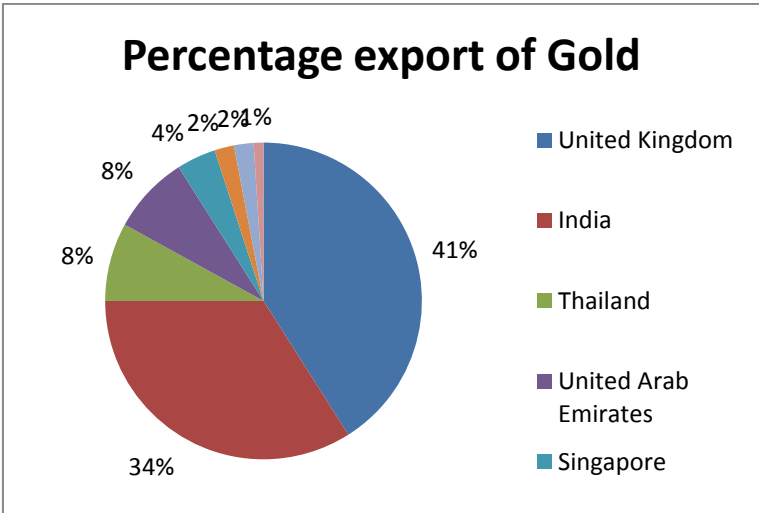


Figure 2.2: Countries patronising WA gold export (after WA Department of Mines and Petroleum, 2009).

The petroleum industry in the state is the second most valuable resource sector after iron ore. Western Australia accounts for 73% of natural gas and 64% of crude oil condensate production in the whole of Australia's petroleum production. Japan dominates the export of most of the Western Australia's petroleum products.

Alumina is the state's fifth largest resource sector and it accounts for 6% or \$4.6 billion of all mineral and petroleum sales in 2008 – 09 with a total production volume of 12.3 million tonnes in 2008 - 09. Currently, the state has two alumina producers namely Alcoa World Alumina Australia and Worsley Alumina Pty Ltd. Since Alumina comes from the refinery of bauxite, both companies have refineries closer to their bauxite mines with shipping facilities which makes it much economical to even treat low-grade bauxite.

Nickel comes sixth of value in the resource sectors of the state and a sale of \$3 billion was realized in 2008 – 09 with equivalent total production of 178 thousand tonnes. All Australia's Nickel output comes from Western Australia and in terms of production; BHP Billiton's Nickel West dominates.

Copper production in Western Australia may be divided into three categories namely, copper concentrate, copper cathode and copper by product mainly from nickel mining. The total copper production in the state amounted to 137,841 tonnes in 2008 – 09.

The state has other mining sectors such as mineral sands, diamonds, coal, salt, tin, tantalum, lithium, raw earths, manganese, chromites and other base metals such as lead, zinc and copper. Among all, the three sectors namely iron ore, Gold and Petroleum industries account for 83% or \$59 billion of all the state minerals and petroleum sales in 2009 – 10 and these three sectors currently form the backbone of WA's economy (WA Department of Mines and Petroleum, 2010). In total, WA contributes nearly 40% of the value of Australia's mineral exports (Alexander, 1988) and this value will continue to increase.

2.2 MINING PROCESSES AND WASTES GENERATION

Mining processes commence with the collection of ores by exploration, blasting and excavation processes. These processes generate wastes which are usually in the form of top soil, mine overburden wastes and waste rocks of variable sizes. The product that results from these processes is the ore which could be of variable types depending on the ore deposit in the area. The processes are shown in Figure 2.3. The ore then needs to be processed using favorable and efficient process for maximum liberation. These processes could involve comminution, concentration, upgrading and leaching (Wills, 1992). During comminution solid ore materials are reduced in size through crushing and grinding. This exposes the

mineral particles of interest and increases the surface area in order to be subjected to subsequent processes. The valuable minerals are embedded and are locked into the matrix or in strict combination with other materials. The comminution process helps to free the useful minerals of interest from the matrix. This process generates some other wastes as spoils or fines which the miner considers non useful.

The mineral ores need to have a higher percentage of metal or valuable mineral content in the ore. This is done through concentration and upgrading of the ore by separating valuable minerals from the wastes. Concentration and upgrading is done by washing, froth flotation, magnetic and chemical separation depending on the type of ore being treated. Iron ore processes for instance ends at this stage but however other ores such as gold needs to go through other processes such as leaching which is done by subjecting the concentrated and upgraded ore into cyanidation.

The processing of the ore stage, as elaborated above; generates wastes such as tailings, process waste water, slurries, leached residues, hazardous chemicals etc. as presented in Figure 2.3. The product from the ore processing stage is the bulk raw material. This needs to be processed further through smelting and refinery processes to arrive at the final marketable product. During smelting and refinery processes wastes such as slags and ashes are produced.

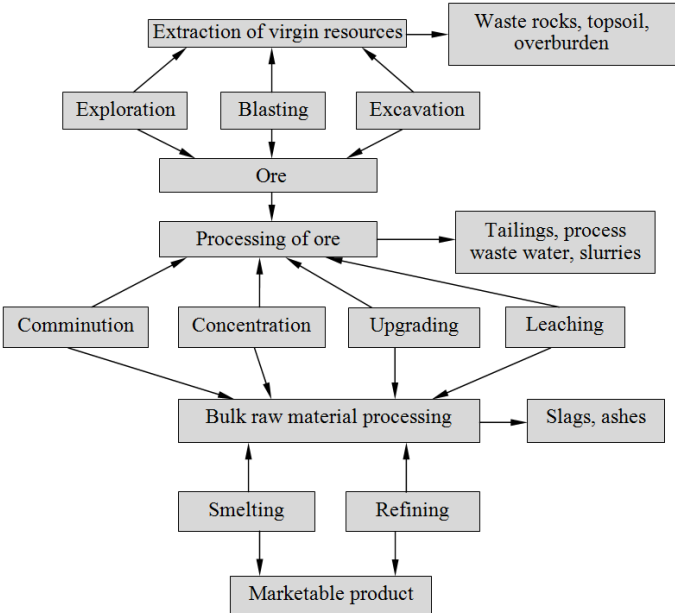


Figure 2.3: Mining processes and waste generation (after Yellishetty et al., 2008; Kuranchie et al., 2012).

2.3 CHARACTERISTICS AND USABILITY OF MINE WASTES

There are different types of mine wastes that are generated during mining activities and each of these wastes has unique characteristics and properties. The waste generated depends on the type of ore being treated and the treatment method being used.

2.3.1 Mine overburden wastes and top soil

Mine overburden wastes are the materials that lie above the ore of economic interest. They range from coarse to fine grain size and are normally non-toxic. They come about during the extraction of virgin raw materials as shown in Figure 2.3. These overburden materials normally needs to be removed in order to get access or locate the mineral ore of interest and when this happens the overburden becomes a waste material; a summary of the chemistry on mine overburden and fly ash is reported by (Behera & Mishra, 2012) as seen in Table 2.1.

Table 2.1: Physico-chemical properties of fly ash and mine overburden (after Behera & Mishra, 2012).

Property	Fly ash	Mine overburden
Specific gravity	2.16	2.6
Particle size analysis (%)		
Gravel (> 4.75 mm)	-	9.71
Sand (4.75-0.075 mm)	22.17	32.91
Silt (0.075-0.002 mm)	75.04	43.73
Clay (< 0.002 mm)	2.79	13.65
Consistency limits		
Liquid limit (%)	30.75	25.7
Plastic limit (%)	Non-plastic	15.04
Shrinkage limit (%)	-	13.44
Plasticity index (%)	-	10.66
Free swell index (%)	Negligible	20
Ph value	7.2	4.85
Loss on ignition (%)	2	10

Geochemical assessment was conducted on mine overburden (Swane, 2009) which reports that mine overburden could consist of coarse-grained particles, rock fragments and fine grain particles. The normal practice of handling mine overburden materials is to dispose

off into already mined-out pits. The same Geochemical assessment points out that the mine overburden are non acid forming (NAF) and with high acid neutralizing capacity (ANC). The mine overburden material was also observed to have low sulphur content and low metal concentrations as well as having low water solubility characteristics. The chemistry of mine overburden is such that it is marginally sodic and with marginal exchangeable sodium percentage (ESP) which could have structural stability problems due to potential dispersion. This problem could be overcome by subjecting it to further treatment by amelioration if it is intended to be used as a revegetation cover (Ye et al., 1999). However, the exchangeable sodium percentages and the sodicity characteristics of the mine overburden are not far from desirable ranges and therefore not considered extreme and it could be acceptable for use as a vegetation growth cover. The Geochemical assessment also shows that the mine overburden has moderate to high electrical conductivity and is alkaline in nature. However, in Table 2.1, the pH value of the overburden is reported as 4.85 which could be interpreted as acidic and it is due to the fact that the geochemistry of the mine overburden materials could be different in different locations.

2.3.2 Mine waste rocks

Waste rock is a material mined from an open cast pit or underground workings in order to gain access to the rock material hosting economic grades. It is normally defined to include topsoil, weathered and partially weathered overburden and primary or unweathered waste rock (WA Department of Resources, Energy and Tourism, 2009). Mine waste rocks do not contain minerals of interest or they contain minerals in very low concentrations that cannot be economically processed during and after mining. It is sometimes called coarse tailings. Waste rocks are generated during the extraction of virgin resources which could involve exploration, blasting and excavation as shown in Figure 2.3. Normally waste rocks are returned underground into mined-out pits for backfilling and others are left in waste rock dump in a stockpile.

The constituents of these mine waste rocks dumps have been studied by European Commission (2009), and it has been reported to contain trace amounts of heavy metals naturally present in the minerals such as cadmium, antimony and some trace elements such as selenium, mercury, arsenic and dissolved metals in the form of iron and copper. Some other possible constituents are sulphur, calcite or calcium carbonate (CaCO_3), dolomite ($\text{CaMg}(\text{CO}_3)_2$), Magnesite (MgCO_3) and some silicate minerals; quartz (SiO_2) depending on the nature and mineralogy of the ore. The chemistry and the composition of most waste rocks

from different ore locations share certain similar features and this applies to most metalliferous, coal and industrial mineral deposits (Hitch et al., 2010). To consider mine waste rocks for further use, characterisation of the waste rocks should be considered due to the potential environmental problems. Some constituents of the waste rocks might cause oxidation and the production of acid leading to acid mine drainage (AMD). The sulphide ores such as pyrite are common in most sulphide minerals and they are usually associated with many metallic ore deposits and have potential of producing acid due to sulphide mineral oxidation and this could potentially lead to acid mine drainage (AMD). The presence of calcite, dolomite, silicate and hydroxide minerals in the waste rocks can prevent oxidation of the sulphide minerals through buffering and consumption of hydrogen ions produced by the sulphide content in the waste rocks. This can render the waste rock non toxic to be considered for further use (Hitch et al., 2010). This makes it possible to consider many other applications of mine waste rocks compared to applications of the ore commodity mined itself.

Pilbara region of WA is where the majority of the state iron ore is mined and this region produces about 160 metric tonnes of iron ore mine waste rocks annually. From this quantity of iron ore waste rocks, about 50 metric tonnes comes from Hamersley iron ore which is owned by Rio Tinto iron ore group, about 105 metric tonnes from BHP Billiton and between 6 to 8 metric tonnes from Robe iron ore. The pH of this iron ore waste rocks in the Pilbara region of WA ranges from acidic (1.5) to alkaline (9.5) with lower electrical conductivity similar to ceramics and with poor fertility (Mulligan, 1996).

2.3.3 Mine tailings

Mine tailings that can also be called mine dumps, slimes, leach residues or slickens are the unwanted materials that are left over after separating the valuable minerals of economic interest from the gangue or wastes of an ore. Mine tailings are produced during ore processing stage which could include comminution, concentration, upgrading and leaching as shown in Figure 2.3.

In the earlier times in British Columbia, Cornwall, some parts of Britain and other parts of the world, mine tailings were thrown into the sea and rivers for a long time as means of disposal and it is still practiced in some mines. This was done to reduce the volumes and handling of mine tailings available. This practice was considered as damaging to the marine species and ecology in general and it is now strongly discouraged. This practice has the potential of causing water pollution because the tailings are normally contaminated with heavy metals, solids, mill reagents and sulphur compounds (McKinnon, 2002).

According to Wills (1992), the current practice of handling mine tailings is disposing it off into a tailings dam or in a tailings storage facility. In doing this there is a requirement to construct the tailings dam close to the mine site to reduce transportation cost and for convenience. This brings a limitation and problems in terms of site selection for tailings dam. Tailings are normally disposed off as slurry of high water content and could also contain coarse dry material (Wills, 1992). Low grade ores result in very large amounts of fine tailings. Therefore, the most acceptable and more sustainable way of dealing with mine tailings is to consider reuse options for them especially in construction and building where large volume could be used.

Tailings have other useful properties such as self cementing characteristics which remove the necessity of adding cement when it is being used to fill mined-out areas and for other similar applications. The self cementing characteristics of the tailings is due to its slimy nature and the large quantity of sulphides it contains which oxidizes on contact with the air to form hard cement-like crust.

To consider the mine tailings for reuse options it is normally ideal to think of some improvement before its applications. Ye et al. (1999) reports that mine tailings are often characterized by high concentrations of heavy metals such as lead, zinc, cadmium, sulphur, and arsenic, alkaline or acidic conditions and could contain some of the toxic chemicals used to process the ore. For instance to consider the mine tailings for revegetation applications ameliorative approach which rely on achieving the optimum conditions for plant growth by considering some improvement for the physical and chemical nature of the mine tailings. This could be achieved by using lime, organic matter such as pig manure mixtures with the tailings. The lime in this sense neutralizes and reduces the acidic conditions and the organic matter enhances the fertility of the tailings to support plant growth. Mine tailings can also be detoxified by reacting it with iron-rich counter-current decantation (CCD) wash water (Mckinnon, 2002).

To exploit the utilisation of mine tailings, a study was conducted by Roy et al. (2007) to investigate the feasibility of using gold mine tailings to make bricks. Four samples were prepared for this idea. The first was using mill tailings alone without any additive, the second was mill tailings and cement as an additive, the third was mill tailings with black cotton soil and the last was mill tailings with red soil. It was discovered that the mill tailings alone have low plasticity which could not make it suitable for making bricks without any additive. However, the mill tailings with the soil additive passed the test for bricks manufacturing and

it was also cheaper than the one with cement additive. The same experiment evaluated the physical and chemical properties of the gold tailings and established that the specific gravity of the gold mine tailings was as high as 2.75 due to the presence of iron, liquid limit was found to be non plastic, plastic limit was also non plastic, plasticity index was also non plastic and particle size ranges from 33% clay size particles, 17% silt size particles and 50% sand size particles. The major chemical composition of the gold mine tailings is presented in Table 2.2.

Table 2.2: Constituents of gold tailings (after Roy et al., 2007).

Constituents	Percentage (%)	
	Iron-deficient gold tailings	Iron-rich gold tailings
Calcium oxide	8.4	7.6
Silica	56.0	51.8
Aluminium oxide	11.9	8.2
Ferrous oxide	10.2	18.9
Magnesium oxide	8.6	6.3
Loss on ignition	2.0	3.9

Celik et al. (2006) considered one application using gold mine tailings as an additive in the manufacture of Portland cement. The results indicated that gold tailings between 5 to 15% as an additive could be feasible in Portland cement production if the gold tailings used in the cement are blended with silica fume and C type fly ash to attain the desired values of compressive strength (60 N mm^{-2}). The chemical composition and some physical properties of the gold tailings and other materials were also reported by (Celik et al., 2006) as shown in Table 2.3.

Das et al. (2000) also described a new development in managing iron ore tailings by converting them into value added products such as ceramic floor and wall tiles for building applications. They reported that iron ore particles below $150\mu\text{m}$ in size were discarded as waste tailings. They also tested constituents of the tailings from different locations and their mixture using standard techniques XRD (Siemens D 500) with Ni filter and Cu ($K\alpha$) radiation. The result of the test is presented in the Table 2.4. The iron ore tailings were found to contain high percentage of silica as can be seen in Table 2.4. The high silica content in the iron ore tailings is considered favorable in terms of the property and the raw material

requirements for the production of ceramic tiles. The study concluded that iron ore tailings up to 40% by weight can be considered for use as a part of raw materials for ceramic floor and wall tiles due to its high silica content. The ceramic tiles from the iron ore tailing materials were found to be superior in terms of scratch hardness and strength. The new tiles from the iron ore tailings maintain most of the other essential properties as the conventional raw materials used for ceramic tiles. The application of iron ore tailings in the ceramic tiles production was also found to be cost effective in comparison with the usual traditional clay for ceramic tiles production.

Table 2.3: Chemical composition and physical properties of tailings and other materials (after Celik et al., 2006).

Constituents (%)	Gold tailings	Silica fume	Fly-ash (Soma Unit VI)	Fly-ash (Seyitömer)	Clinker	Portland cement
SiO ₂	94.56	90.02	33.97	54.37	22.02	20.14
Al ₂ O ₃	1.67	-	19.53	19.46	5.9	5.79
Fe ₂ O ₃	1.87	0.3	4.53	11.17	3.61	3.34
CaO	0.39	0.33	32.29	4.64	65.1	65.8
MgO	0.27	2.36	1.48	5.58	1.21	0.84
SO ₃	0.09	0.85	4.95	1.15	0.24	2.69
Na ₂ O	0.31	0.29	0.61	0.74	0.23	0.43
K ₂ O	1.16	3.72	1.09	2.29	0.95	0.5
TiO ₂	0.11	-	0.64	0.8	-	-
P ₂ O ₅	-	-	0.185	0.075	-	-
Fineness (cm ² g ⁻¹)						
measured by Blaine method	-	14 000*	4020	3215	-	3550
Specific gravity (gcm ⁻³)	-	-	2.14	2.26	-	3.13

*Silica fume has 14000 cm²g⁻¹ specific area measured by BET method.

Table 2.4: Constituents of iron ore tailings from five locations and their mix (after Das et al., 2000).

Constituents	Samples (wt. %)					
	D1	D2	D3	D4	D5	D mix
SiO ₂	39.40	42.94	40.06	63.32	60.42	51.12
Al ₂ O ₃	1.36	1.42	1.33	1.37	1.42	1.22
Fe ₂ O ₃	55.61	52.05	55.32	32.31	34.81	44.36
CaO	0.12	0.08	0.25	0.36	0.33	0.22
Loss on ignition	3.42	3.40	2.91	2.56	2.33	2.95

2.3.4 Mine smelting slag wastes

Slags are waste glassy materials left over when metals are pyrometallurgically extracted from ores. Slags could come from the smelting of metals such as copper, gold and other metals. Study on copper slag by Gorai et al. (2003), reports that every tone of copper produced results in the generation of 2.2 tonnes of slag. Currently slags from industries are recycled or disposed in slag dumps or stockpiles. The study also reports that air cooled granulated copper slags have a number of applications due to favorable mechanical and physical properties such as soundness characteristics, good abrasion resistance and good stability. Copper slags contain very low content of lime (CaO) and also have pozzolanic properties. If the lime (CaO) is increased by the addition of sodium hydroxide (NaOH) it could show cementitious properties that can be considered to be used for partial or full replacement of Portland cement in concrete or as a part of raw material for cement manufacturing. Generally slags have favorable mechanical and physical properties that could be considered for use as an aggregate in asphalt paving applications, recovery of metal values, cement replacement in concrete, fill, ballast, abrasives, aggregates, glass, tiles etc.

A study by (Samet & Chaabouni, 2004) reports that slags are almost glassy and could have good reactivity. Therefore, their study considers the suitability of slags as a partial replacement of clinker in cement manufacturing. It was seen that slag has an acceptable extension of the setting time, an improvement of the rheological behavior and very good stability to expansion. Again, the compressive strength of the mixture with slag was seen as almost suitable and comparable to be accepted for use as additive in Portland cement. The optimum composition realized in the experiment was 61% clinker, 35% slag, 3% gypsum and

1% limestone. This will make it possible to replace part of the clinker and the cement raw materials with slag.

2.3.5 Mine fly ashes

Fly ash is a residue generated during combustion. It can also be found from the bottom of a furnace during smelting and it comprises of fine particles that rise with the flue gases. Fly ash ranges in size from 0.5 to 100 μm and mostly consists of silicon dioxide (SiO_2), aluminium oxide (Al_2O_3) and iron oxide (Fe_2O_3). It is highly heterogeneous and is made up of glassy particles. A review on fly ash by (Ahmaruzzaman, 2010), reports that the colour could range from tan to gray to black with abrasive nature and mostly alkaline and refractory in nature. There can be self cementing (Type C) and pozzolanic (Types F and C) materials present. They contain other essential elements such as phosphorus, potassium, calcium, magnesium, zinc, iron, copper, manganese, boron, and molybdenum which are essential nutrients for plant growth. The geotechnical properties of fly ash such as specific gravity (2.1 – 3.0), permeability, internal angular friction, consolidation behavior, specific surface area (170 – 1000 m^2/Kg) makes it suitable in its application as road construction materials, embankments and as fills. The fly ashes normally contain pozzolanic properties and have lime binding characteristics which makes it useful and suitable for its use as a part of raw material in cement manufacturing and as a constituent in concrete mixture. Fly ashes have other essential physicochemical properties such as bulk density, particle size, porosity, water retention capacity and surface area which make it suitable to be considered for use as an adsorbent.

Another study was conducted on fly ash for the application in agricultural purposes (Basu et al., 2009). It was found that in general, fly ash has low bulk density (1.01 – 1.43 gcm^{-3}), hydraulic conductivity and specific gravity (1.6 – 3.1 gcm^{-3}) with a moisture retention of 6.1% at 15 bar to 13.4% at 1/3 bar. In the same work, the constituents of fly ash was reported to include some plant nutrients such as phosphorus, potassium, calcium, magnesium and sulphur and some micro nutrients like iron, manganese, zinc, copper, cobalt, boron and molybdenum except organic carbon and nitrogen which is suitable to be considered for its application in agriculture purposes. Relatively, the fly ash may contain heavy metals and may be naturally radioactive which could be toxic to plant growth. However, it has been reported in the same study that research have proved that the effects of these heavy metals and the radioactivity content is in safe limit if the fly ashes are applied in optimum quantities.

2.3.6 Mine process wastewater

A study conducted into mine wastewater reports there are various sources of wastewater in a typical mine site (Dharmappa et al., 2002). The constituents and characteristics of mine wastewater were found to have some chemicals and elements that could be very toxic when it is considered for reuse. Treatment and further processing of the wastewater will undoubtedly be required before considering it for secondary applications. The study reports that wastewater in the mines comes from mine water, process wastewater, domestic wastewater and surface run-off. The various contaminants in the wastewater are categorized into five namely, physical being suspended solids, taste and odour, temperature, colour; chemical (organic) being soaps and detergents, oils and grease; chemical (inorganic) being heavy metals, acids, cyanides, dissolved salts, alkalis; biological being bacteria, viruses, small organisms and radiological being uranium, tritium and other radioactive substances from mine tailings. Some of the elements present in the waste water are sodium, potassium, calcium, manganese and magnesium which are non poisonous. Therefore, the process water when treated will remove the toxic chemicals and could be considered for other many applications such as irrigation, industrial process and in civil engineering constructions like concrete mixture.

2.4 ELECTRICAL RESISTIVITY/CONDUCTIVITY OF GEOMATERIALS

Electrical resistivity, (ρ) or conductivity (inverse of resistivity) of soil is an important parameter that gives an idea of the properties and the behaviour of the soil for engineering and other applications. The method is non-destructive and very sensitive and therefore, it gives a reliable and acceptable way for describing and knowing soil properties and behaviour without digging.

According to Samouelian et al. (2005), the electrical resistivity of soil is a measure of the water content in the soil, temperature of the soil, soil porosity and the constituents of the soil. The electrical resistivity decreases with an increase in water content, decreases with an increase in the temperature of the soil, fine particles of soil has greater electrical conductivity than coarse grained particles and soil with macro and meso porosity have high and low electrical resistivity respectively. Electrical resistivity also measures the soil salinity and this can be applied in agriculture (Adam et al., 2012). Electrical resistivity is also a measure of the corrosiveness of soil according to British standard BS – 1377 (British Standards, 1990). If the electrical resistivity is greater than 100 Ωm the soil is slightly corrosive, if it is between 50 and 100 Ωm the soil is moderately corrosive, if it is between 10 and 50 Ωm the soil is

corrosive and if it is less than 10 Ωm the soil is severely corrosive. This relationship between the soil parameters and the electrical resistivity help to know the important soil characteristics for engineering applications.

The theory of soil electrical resistivity is based on the ohm's law;

$$V = IR \quad (2.1)$$

where, V is the potential difference across the conductor, I is the current flowing through the conductor (Amperes) and R is the resistance of the conductor (Ohms). The electrical resistivity is dependent on the resistance (R), the length L and the cross sectional area A as seen in the relation in equation (2.2); where ρ is the resistivity of the conductor material (Ωm), L is the length of the conductor (m) and A is the cross sectional area (m^2).

$$R = \frac{\rho L}{A} \quad (2.2)$$

Four electrodes are commonly used for the measurement of soil electrical resistivity; two electrodes are used to inject current called current electrodes and the other two electrodes called potential electrodes that are used to record the potential difference (Δv) between the two electrodes. The electrical resistivity is also dependent on a factor shown below;

$$\rho = K \frac{\Delta v}{I} \quad (2.3)$$

where K is a geometrical coefficient and according to Munoz-Castelblanco et al. (2012), K depends on the size and the arrangement of the electrodes, I is the current and Robain et al. (2003) used direct current (DC) for the measurement. However, according to Yan et al. (2012), the use of alternate current (AC) with frequency range of 10 Hz and 10,000 Hz have the advantage of eliminating polarization and the electrolytic phenomena of electrodes that can cause changes in water content, soil structure and pore-fluid chemistry at low frequency during measuring.

Several methods exist for the measurement of electrical resistivity namely, Wenner method, Schlumberger method and driven Rod (3 pin) method. Of these, the use of four electrodes such as the Schlumberger and the Wenner methods are common. In the

Schlumberger method, the current electrodes are placed at the opposite ends of the array with potential electrodes placed centrally as seen in Figure 2.4. The spacing between the current and the potential electrodes are expanded by a factor n while measuring the increasing depths. This array has the highest sensitivity to both changes in the horizontal and vertical planes at low and high values of n respectively (Jones et al., 2012). The geometric constant K , for the Schlumberger array is calculated as;

$$K = \pi n (n + 1) a \tag{2.4}$$

where c in Figure 2.4 is given by $c = na$ and n is the factor that expands the spacing between the current electrodes and the potential electrodes.

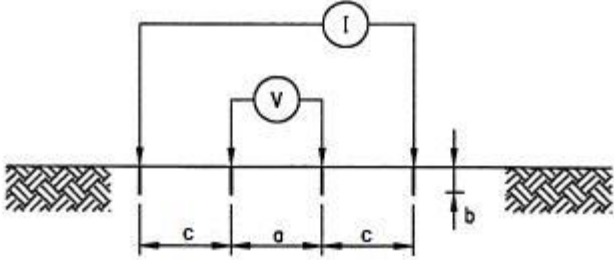


Figure 2.4: Schlumberger method of arrangement for soil electrical resistivity measurement

In the Wenner array, the four electrodes are equidistantly spaced in straight line at the soil surface with two outer electrodes serving as the current or transmission electrodes and the two inner electrodes serving as potential or receiving electrodes (Corwin & Lesch, 2005). In the Wenner method, all four electrodes are moved after each reading and it makes this method susceptible to lateral variation effects. This method also requires much time, long measuring cables, large free space and big electrode spacing. The diagram for the arrangement of the Wenner array is shown in Figure 2.5. The electrical resistivity in the Wenner array can be calculated using equation;

$$K = 2\pi a \tag{2.5}$$

where K is the geometric constant, and a is the inter-electrode spacing.

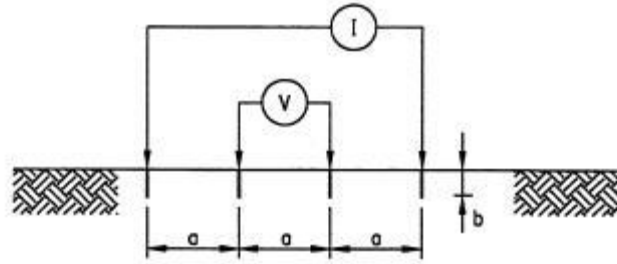


Figure 2.5: Wenner array arrangement of soil electrical resistivity measurement (after Adam et al., 2012)

Using soil electrical resistivity to know the engineering behavior and geotechnical properties of soils could have significant civil engineering applications. Soil electrical resistivity has been used and it has been adopted as a non-destructive way to assess the safety of embankments for delineating weak zones at the core of the embankment. Oh (2012) prepared a laboratory set up for testing the safety of embankment by making use of the electrical response of changes in the void ratio in the core unit of the embankment through electrical resistivity measurement. The work identified that the weak zones of the core material of the embankment could have either a higher or lower resistivity values rapidly if it is compared with the standard known values. This anomaly could help to identify the areas of the weak zone in the embankment.

(Oh & Sun, 2008) also investigated similar work making use of electrical resistivity to determine the strength of embankment. They reported that if the clay within the core of the embankment flows out during the process of piping or internal erosion, the vulnerable point in the core material will have high resistivity. Also according to Bhatti & Reidt, (1989), soils with low moisture absorption may exhibit potentially better thermal insulation characteristics and it can be applied in concrete.

According to Samouelian et al. (2005), the electrical resistivity has been applied in ground water exploration, landfill and solute transfer delineation, agronomical management and petroleum prospecting. Electrical resistivity is also applied in the design of lightning earthing system. Once the soil resistivity is known, the design of the earthing system can be made to achieve the desired earth resistance. Thermal conductivity or electrical resistivity of soil is also applied in thermal performance of buried pipelines and geothermal heat pumps (Haigh, 2012). Electrical resistivity measurement has also been applied in agriculture to measure the salinity of the soil. The electrical resistivity measured was converted to apparent

electrical conductivity and a decrease in apparent electrical conductivity is an indication of desalinization or a decrease in salt content of the soil (Adam et al., 2012).

2.5 USES OF MINE TAILINGS AND INVOLVED ENGINEERING

Mine wastes have comparable engineering and geotechnical characteristics to many civil engineering materials used for building and construction. This makes it feasible to replace some civil engineering materials with mine wastes in many applications.

2.5.1 Mine tailings for brick production

Roy et al. (2007) studied the feasibility of using gold mill tailings from Kolar Gold Fields, in Karnataka, India for bricks production. They mixed the gold tailings with ordinary Portland cement as one material and mixed the tailings with other soils as another material in a specified proportion, molded and finally fired at high temperatures. They observed that the cement-tailings bricks containing 20% of cement and having 14 days of curing met the required compressive strength but the cost was 2.4 times the traditional clay bricks. The soil-tailings bricks also met the requirements for compressive strength and other requirements with the cost being 0.72 times the cost of traditional clay bricks. This work is an improvement on the existing practice. However, this practice still has some drawbacks on the environment because cement was used which will still add more quantity of greenhouse gases to the environment. Also high volume utilisation of the tailings will not be realized as the tailings were not used in full since other soil was added to make it feasible.

Ahmari & Zhang (2012), investigated the feasibility of utilising copper mine tailings from Mission Mine operations of ASARCO LLC in Tucson, Arizona in the United States of America for the production of eco-friendly bricks based on the geopolymerization technology. The geopolymerization is the reaction undergone by aluminosilicate materials in a highly concentrated alkali hydroxide or silicate solution, forming very stable material called geopolymer having amorphous polymeric structures with interconnected Si-O-Al-O-Si bonds. In their process they mixed the copper mine tailings with sodium hydroxide (NaOH) solution and formed the bricks by compressing the mixture within a mould under a specified pressure and curing the bricks at slightly elevated temperatures. Their method did not follow the conventional one by using clay and shale and firing at high kiln temperatures. They checked the properties of their geopolymer bricks through water absorption, unconfined compressive strength and abrasion resistance by calculating the abrasion index of the bricks using the relation below;

$$\text{Abrasion Index} = \frac{100 X_{\text{absorption}} (\%)}{UCS (Psi)} \quad (2.6)$$

Table 2.5: ASTM Specifications for different applications of bricks (after Ahmari & Zhang, 2012).

Title of specification	of	ASTM designation	Type / grade	Minimum UCS (MPa)	Maximum water absorption (%)	Abrasion Index
Structural load bearing wall tile	clay	C 34-03	LBX ^A	9.6 ^C	16 ^E	NA
			LBX	4.8 ^D	16 ^E	NA
			LB ^B	6.8 ^C	25 ^E	NA
			LB	4.8 ^D	25 ^E	NA
Building brick		C62-10	SW ^F	20.7	17	NA
			MW ^G	17.2	22	NA
			NW ^H	10.3	No limit	NA
Solid masonry unit		C126-99	Vertical coring	20.7	NA	NA
			Horizontal coring	13.8	NA	NA
Facing brick		C216-07a	SW	20.7	17 ¹	NA
			MW	17.2	22 ¹	NA
Pedestrian and light traffic paving brick	and	C902-07	SX	55.2	8	Type 0.11 I ^J
			MX	20.7	14	Type 0.25 II ^J
			NX	20.7	No limit	Type 0.50 III ^J

^ALBX = load bearing exposed

^BLB = load bearing non-exposed

^C End construction use

^D Side construction use

^E Based on 1 h boiling water absorption

^F Severe weathering

^G Moderate weathering

^H Negligible weathering

^I Based on 5 h boiling water absorption

^J Types I, II and III are respectively subjected to extensive, intermediate and low abrasion.

They then compared the properties of their geopolymer bricks with the ASTM specifications for different applications of bricks. The ASTM specifications for different applications of bricks are shown in Table 2.5. They finally concluded that using 15M NaOH, initial water content of 16%, forming pressure of 0.5MPa and 90°C curing temperature for seven days of the copper geopolymer bricks meets the ASTM standard requirements for building bricks and other application of bricks except pedestrian and light traffic paving bricks. The extra advantages of the geopolymer bricks were that they are resistance to abrasion, require less energy for production because they do not require high temperature kiln firing and finally reduce the emission of greenhouse gases in the environment because of less energy requirements. Their work should therefore prompt researchers to try other alkali activators for a possibility of further cost reduction or to provide more convenient options.

Chen et al. (2011) investigated the possibility of making construction bricks by using hematite tailings from Western Hubei province of China. They mixed the hematite tailings with clay and class F fly ash in different proportions. The process included mixing, forming, drying and firing. They found out that with hematite tailings of 84%, forming water content 12.5 – 15%, forming pressure 20 – 25 MPa, firing temperature 980°C – 1030°C for 2 hours, the produced bricks had a water absorption of 16.54 – 17.93% and mechanical strength of 20 – 25 MPa and these and other properties of the newly produced bricks conformed to the Chinese Fired Common bricks standard GB / T5101 – 2003 (State General Administration of China for Quality and Quarantine, 2003).

Liew et al. (2004a) used a mixture of sewage sludge and clay to make bricks successfully. Their bricks came out with good qualities comparable to standard clay bricks. This work added some value to this material which is considered as waste.

The initiatives above to add value to waste materials through utilisation looks good but there can be more improvement if the use of clay is totally avoided by trying 100% of the mine wastes. This will ensure high volume utilisation of mine wastes and at the same time reducing the emission of the greenhouse gases by avoiding the mining of the clay.

Ferone et al. (2007) used weathered coal fly ash from ENEL SPA Power plant in Brindisi of Southern Italy to produced bricks based on the geopolymerization technology.

Sodium silicate solution and sodium hydroxide solution was used as the alkali activator. Different specimens were prepared in different proportions using mixtures of fly ash, sodium silicate solution and sodium hydroxide solution. The specimens were moulded in cylindrical polyethylene moulds and were cured in different durations and at different temperatures. They observed that at 60°C curing temperature for seven days the unconfined compressive strength (UCS) of the bricks increases. In their work, instead of mixing both the sodium silicate and the sodium hydroxide solutions as the alkali activator, sodium silicate solution alone could also be used which may provide the same strength and also reduce cost by using only one activator instead of mixing two activators.

Arioz et al. (2010) used the geopolymer technology on fly ash to produce bricks that met specific requirements. Their research also reported that the properties of a geopolymer paste depend on the type or source of material used. Since a number of studies have focused on fly ash for many years, other waste materials that contain silica and alumina such as iron ore mine wastes should also be investigated to bring more alternatives.

None of the above works on bricks which used geopolymerization technology researched into the setting time of the geopolymer mixture. This is also very important parameter to be considered because geopolymer concrete hardens quickly and therefore, information on the setting time of the mixture is very important to be considered.

2.5.2 Mine tailings as aggregates in concrete

Yellishetty et al. (2008) studied the use of iron ore mine tailings from Goa in India as an aggregate in concrete. They obtained the iron ore mine wastes from four different types of mine waste dumps in different companies in Goa, India and mixed them together. They made two types of concrete, one with the mine aggregate and the other with normal granite quarry aggregate in concrete and compared the properties of the two different concrete with different aggregate. The composition of the concrete was in different proportions using mine aggregate (12.5mm – 20mm in size), sand and cement as the binder. They concluded that the aggregate component of the mine wastes conforms to the Indian Standard Specifications for quality standards of aggregates. Their work is also an improvement for making concrete by adding some mine wastes to partially replace the coarse aggregate part. The use of mine wastes for 100% replacement for both fine and coarse aggregates will avoid the use of natural sand and natural granite quarry completely. Further study into this will bring much more improvement and economy in concrete production.

Bhatty & Reidt (1989), made pellets and slabs from sludge ash and used as lightweight aggregates in lightweight concretes. A concrete mixture was made with the pellets and slabs as aggregates, cement and sand in different proportions. They found out that the moderate strength concretes produced from the pellets have better strength characteristics than those made of slabs and other commercial aggregates mainly because of their shape (spherical), uniformity of size and low moisture absorptions. Again, their work made use of sand and a commonly known fly ash. The avoidance of the use of sand and the search for new waste materials such as iron ore wastes will contribute positively on the environment and provide more different alternatives.

A study was conducted into the suitability of using iron ore mine tailings from Goa in India as aggregates in making concrete and building material (Karpe, 2011). 40% by weight of the mine wastes with size of 12.5 – 20 mm was used as aggregate in the concrete mix. The concrete mixture contained Mine wastes as coarse aggregates, siliceous sand as fine aggregates and portable water with neutral pH and ordinary Portland cement. Based on the mixture, concrete blocks were made and cured for 28 days. Another concrete blocks were made with granite as the coarse aggregate instead of the mine wastes. The strength of the concrete with mine wastes aggregates was 225 kg/cm^2 and that of granite aggregate was 200 kg/cm^2 . The ratio used was 1:2:4 cement, sand and aggregate respectively. They concluded that the compressive strength of concrete made of mine wastes as aggregate was more than that of the concrete made of granite as aggregate and the mine wastes aggregates of the concrete conforms to Indian Specification Standards. Their work can be improved further if 100% of the mine wastes is used as the aggregates in instead of 40%. Also there can be further improvement in terms of the ecological footprint if the use of natural sand could be avoided.

Zhao et al. (2014) recommended 40% replacement of natural sand as fine aggregates only with iron ore mine tailings. In order to have more improvement further research into increasing the quantity of iron ore mine wastes as both fine and coarse aggregates in concrete will be necessary. This will contribute to the course of high volume utilisation of iron ore mine wastes for environmental and economical sustainability.

Alnuaimi (2012) also recommended the replacement of fine aggregates by 40% with copper slag in concrete. It was reported that, there was no major changes in the quality requirements of the concrete. This is also an improvement in concrete technology. However, from their recommendation 60% of natural sand will still have to be used in the concrete as

fine aggregates. Further improvement in this will add more value in concrete production for both environmental and economical sustainability.

2.5.3 Mine tailings utilisation in structural fills

Structural fills such as embankments, foundations, backfills, trenches and all other applications requiring filling in building and construction is a potential area where mine wastes could be applied in large volumes.

A successful pilot project was carried out using bauxite residue (tailings) for construction of road embankment (Kehagia, 2010). The embankment was designed 75 m long, 3 m high with crown width of 8 m. The 75 m long embankment consisted of three sections each with 25 m long. Section I was made of natural soil of A – 4 group, section II was a mixture of bauxite residue (40%) and soil of A – 1 group (60%) and section III a mixture of bauxite residue with fly ash (4%). These different sections were done for comparison study. Specimens of the constituent materials of the three sections were prepared and submitted to mechanical strength test. It was observed that excellent workability and performance of the bauxite residue material was recorded throughout the construction. Soil deformation problems due to insufficient compaction did not occur. After long operational period under traffic (15 – 20 trucks per day) no disintegration appeared on the body of the embankment. The highest percentage of the bauxite tailings used was 60%. If this quantity could be increased or using other mine wastes could increase this percentage it will further have positive impact on the environment.

Baykal et al. (2004) used snow added fly ash from Soma Thermal Power Plant located in the Aegean region of Turkey, cured for different days at 21 °C and used for the construction of highway embankment. The snow introduced extra water during the compaction of fly ash without causing any workability problem. The snow addition in the fly ash caused 30% increase in void ratio, 70% increase in shear strength. It was concluded that the new embankment have lighter weight, higher stability and higher insulation property.

Horiuchi et al. (2000) studied the use of fly ash slurry as a fill material. They concluded that the fly ash slurry is good for underwater fills, lightweight backfills and lightweight structural fills. They also reported that the fly ash slurry ensured higher stability in terms of sliding failure, high strength on the ground and long term durability in terms of creep failure. Looking into other mine waste materials for similar qualities will bring many options and economy in building and construction.

2.6 CONCLUDING REMARKS

The literature review shows that reuse of mine tailings have been explored partially but this cannot be sufficient looking at the numerous favourable applications that exists. It has also revealed that most of the applications of mine tailings have not been used in full, only partial replacement for conventional materials. For high volume utilisation of mine tailings, considering almost 100% replacement for conventional building and construction materials with mine wastes will further enhance environmental and economical sustainability.

Extensive research into this is therefore important to find some favourable and high volume reuse options for the mine wastes generated in Western Australia. This will provide economic gain for the mining industries whiles reducing the volume of mine wastes in state for environmental sustainability.

The application of mine tailings in building and construction (especially iron ore wastes) is very limited or almost none existing in the whole of Australia which is considered as a country with very high mining resources with high generation of mine wastes.

A suitable laboratory friendly methodology that could help to determine the electrical resistivity values for geo-materials are not common in the literature. In particular, the electrical resistivity values of some mine tailings (example iron ore mine tailings) is absent from the literature. To encourage and promote the comprehensive utilisation of iron ore mine tailings in mining destinations such as Western Australia, knowledge about the electrical resistivity of the iron ore mine tailings could help in the characterisation of their engineering and geotechnical parameters to be applied in building and construction projects.

This research will therefore seek to select a number of applications of mine tailings in building and construction that will ensure high volume utilisation of the tailings in Western Australia. The research also seeks to develop a suitable methodology to find the electrical resistivity of iron ore mine tailings for easy characterisation to be applied in building and construction projects.

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CHAPTER 3

STUDY ON MINE WASTES AS POTENTIAL RESOURCE FOR BRICK MANUFACTURING IN WESTERN AUSTRALIA

This chapter was presented orally and published by Research Publishing Services as a peer reviewed conference paper in the proceedings of the First Australasian and South-East Asia Structural Engineering and Construction Conference (ASEA-SEC 1), November 28 – December 2, 2012, Perth, Australia as mentioned in number 6 in Section 1.7. The details presented here are the same, except some changes in the layout in order to maintain a consistency in the presentation throughout the thesis.

3.1 ABSTRACT

Western Australia (WA) is known for its extensive mining activities, and therefore it is in the list of mining jurisdictions of the world. The types of mines in WA cover a wide range of minerals including iron and gold ores. The refining processes of the ores are associated with huge amount of waste generation. Some of the wastes generated are top soils, overburdens, waste rocks, tailings, waste water, ashes, slags, etc. The disposal of these wastes continues to create economic, environmental and legislative problems for the mining industries. To ensure sustainable handling of mine wastes so as to meet community expectations and legislative requirements, these wastes must be handled effectively. Among various options, reuse of wastes is the most sustainable way of countering these problems. This paper explores the potential of using mine tailings available in WA for brick manufacturing. It is found that the mine tailings alone have low plasticity and therefore if it is mixed with some selected soils, it could result in favorable properties that can suit brick manufacturing. This paper discusses the selection of raw materials for bricks from the large volumes of mine tailings in a cost-effective and sustainable way.

3.2 INTRODUCTION

To make mining activities more environment-friendly, economically feasible, and socially acceptable, it is important to use practices that are more sustainable in handling the wastes generated. The volume of mining wastes generated from mining and mineral processing

activities is one of the main pollution concerns for the mining industries and the community. Western Australia is one of the main mining jurisdictions of the world and they encounter the problem of mine waste handling. This state is blessed with large quantities of mineral deposits such as iron ore, alumina, gold and many other mineral deposits and therefore the state earns a high value from mineral exports (Alexander, 1988) The availability of mine ores has attracted much investment in the mining sector and it has resulted in many operating mining sites. The high extent of mining activities in the state has brought about significant volumes of mine wastes which need to be handled by the mining industries. Some of these wastes generated are gold and iron ore tailings, waste rocks, mine overburden, slags and fly ashes (Rampacek, 1982; Swane, 2009).

The disposal of these wastes needs to meet legislations and community expectations. In order to eliminate the problems, the available literature reports that some useful applications could be found for these mine wastes (Choi et al., 2009; Klauber et al., 2011). They could serve as raw materials for many other industrial applications, some without modification or with little modification. In this paper, the details of some mine tailings are presented, and an attempt is made to compare the characteristics of these tailings with the raw materials used for the production of bricks, which are used in civil engineering structures.

3.3 TYPES OF MINES IN WESTERN AUSTRALIA

According to the report of the WA Department of Mines and Petroleum (2009), a large number of mines exist in Western Australia. Considering iron ores, the document reports that iron ore mining is one of the top resource sectors in WA and that it accounted for 47% of the total value of WA's resources with a total production of 316 million tonnes in 2008/2009. The main finished product in this sector is the iron ore and after its production it is exported to other countries as the main raw material for the production of metallic iron.

Gold industry is also one of the important resource sectors in WA. The state gold production was 136 tonnes or 4.4 million ounces as compared with 218 tonnes or 7 million ounces in the whole of Australia in 2008/2009. In percentage terms, the gold production in WA was estimated to be 62% of Australia's total gold production.

Currently all Australia's gold are refined in Western Australia. Gold exports from the state amounted to \$ 16.8 Billion in 2008 – 09 of which 31% or \$ 5.2 billion was the actual production from Western Australia. The remaining 69% or \$11.66 Billion (approximate) can

be attributed to the gold that was refined in and exported from Western Australia but this gold was actually produced from mining activities in other states, territories and overseas.

Other mining sectors in the state are petroleum, alumina, nickel, copper, diamonds, coal, manganese and other minerals. Gold, petroleum and iron ore industries alone together accounts for 83% or 59 billion dollars of all WA minerals and petroleum sales in 2009 – 10 and this together forms the backbone of WA’s economy (WA Department of Mines and Petroleum, 2009).

3.4 MINING PROCESSES

Mining processes begin with collection of the ores by exploration, blasting and excavation processes. Different wastes are generated at this stage, namely top soil, mine overburden and waste rocks of different sources. After this process, the ore of interest is obtained. These processes are shown in Figure 3.1. The ore received at this stage needs further processing. The ore could be processed using the techniques like crushing, grinding, concentration, upgrading and leaching. In order to get the mineral ores having a higher percentage of mineral content concentration, the upgrading is performed after the size reduction process (Wills, 1992). The ore is then leached by subjecting the concentrated ore to Cyanidation. Iron ore process does not go through leaching. The processing stage generates some wastes such as tailings, slags, ashes and process waste water are generated as shown in Figure 3.1. The product from the leached ore is then smelted and sometimes made to undergo refinery if available before the final saleable product is obtained.

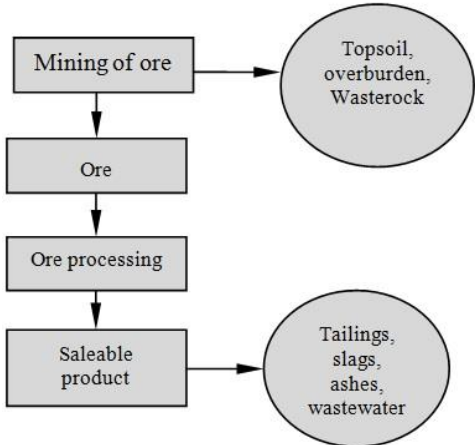


Figure 3.1: Mining processes and waste generation (after Yellishetty et al., 2008).

3.5 CHARACTERISTICS AND USABILITY OF TAILINGS

Mine tailings are the unwanted materials that are left over after separating the valuable minerals of economic interest from the gangue or wastes of an ore. Mine tailings are produced during ore processing stage which could include comminution, concentration, upgrading and leaching.

Tailings are normally contaminated with heavy metals, solids, mill reagents and sulphur compounds and this could cause environmental pollution if it is not handled properly (Waldichuk, 1978). According to Wills (1992), the current practice of handling mine tailings is disposing it off into a tailings dam or in a tailings storage facility. In doing this, there is a requirement to construct the tailings dam close to the mine site to reduce transportation cost and for convenience. This brings a limitation and problems in terms of site selection for tailings dam and tailings utilisation. Another limitation in the utilisation of mine tailings such as gold tailings is that it needs some modification or some additives to match engineering characteristics for construction applications. Tailings are normally disposed off as slurry of high water content and could also contain coarse dry material. Low grade ores result in very large amounts of tailings. This therefore makes reuse of mine tailings most acceptable and more sustainable way of handling it. It could also be reprocessed to recover additional valuable materials such as coarse tailing materials (20 – 30 mm) for used as railway ballasts and aggregates. Tailings have other useful properties such as self cementing characteristics which remove the necessity of adding cement when it is being used to fill mined-out areas and other similar applications. Celik et al. (2006) confirms this by using gold mine tailings as an additive in the manufacture of Portland cement. The results indicated that gold tailings between 5 to 15% as an additive could be feasible in Portland cement production if the gold tailings used in the cement are blended with silica fume and C type fly ash to attain the desired values of compressive strength of 60 N/mm².

To consider the mine tailings for reuse options, it is normally ideal to think of some improvement before its applications. The mine tailings are often characterized by high concentrations of heavy metals such as lead, zinc, cadmium, sulphur, arsenic, alkaline or acidic conditions, and they could contain some of the toxic chemicals used to process the ore (Ye et al., 1999). For instance, to consider the mine tailings for revegetation applications, ameliorative approach should be used. This relies on achieving the optimum conditions for plant growth by improving the physical and chemical nature of the mine tailings. This could be achieved by using lime, organic matter such as pig manure mixtures with the tailings. The

lime in this sense neutralizes and reduces the acidic conditions, and the organic matter enhances the fertility of the tailings to support plant growth.

Das et al. (2000) described a sustainable way of handling iron ore tailings by converting them into a new value added product as ceramic floor and wall tiles for building applications. The iron ore tailings were found to contain high percentage of silica and they concluded that iron ore tailings up to 40% by weight can be considered for use as a part of raw materials for ceramic floor and wall tiles due to its high silica content. The ceramic tiles from the iron ore tailings were found to be superior in terms of scratch hardness and strength. The new tiles from the iron ore tailings also maintain most of the other essential properties as the conventional raw materials of ceramic tiles. The application of iron ore tailings in the ceramic tiles production was cost effective in comparison with the usual traditional clay for ceramic tiles production. (Ahmari & Zhang, 2012) also confirmed that it is possible to make eco-friendly bricks to conform to ASTM requirements using geopolymerization technology by selecting appropriate preparation conditions.

3.6 RAW MATERIAL REQUIREMENTS AND CRITERIA FOR ASSESSMENT OF QUALITY BRICKS

Clay is the main raw material for brick manufacturing. For good-quality bricks, the clay should exhibit some specific properties and characteristics as per the type of civil engineering applications. The clay must have some plasticity that can allow it to be shaped or molded when it is mixed with water. Clay must have sufficient moisture content and air-dried characteristics so that it can maintain its shape after it is formed. Also, when clay is subjected to the optimum temperatures during firing, clay particles must fuse together (Bricks Industry Association, 2006). Previous studies concerning the assessment for the quality of bricks reports that a good brick should be hard, strong and durable. The optimum compressive strength of bricks may vary from 35 kg/cm² to 350 kg/cm² (Roy et al., 2007).

Water absorption is one of the main factors for the assessment of the quality of bricks. It basically determines the degree of burning of the bricks. Water absorption of good bricks should be less than 20% after 24 hours of immersion in water.

Linear shrinkage is one of the quality factors, and it is considered necessary for the determination of quality of bricks for civil engineering applications. The linear shrinkage is measured using the dimension of the bricks before and after burning. Good indication in this

sense is that the bricks after firing should be within $\pm 3\%$ of smoothness, they should have rectangular faces and should have sharp edges.

3.7 SUITABILITY AND IMPROVEMENT OF TAILINGS FOR BRICK MANUFACTURING

Roy et al. (2007) tested a sample of gold tailing and reported that it consisted of 33% clay, 17% silt and 50% sand; thus the tailing was having low plasticity from the point of view of brick manufacturing. The specific gravity was found to be 2.75 and it is attributed to the presence of iron in the tailings. This means that some additive will be required to enhance the plasticity characteristics of the tailings. The major chemical composition of the gold mine tailings is presented in Table 3.1.

Table 3.1: Constituents of gold mine tailings (after Roy et al., 2007).

Constituents	Light coloured (%)	Dark coloured (%)
CaO	8.4	50
SiO ₂	56.0	70
Al ₂ O ₃	11.9	90
Fe ₂ O ₃	10.2	119
MgO	8.6	135
Loss on ignition	2.0	155

The same study on the test of the tailings samples reports that mill tailings mixed with some red soil with plasticity between 10 and 14% was able to bind well with the tailings and produce a good plasticity to be considered for bricks manufacturing. Also the red soil mixed with the tailings could maintain the usual red colour of the traditional clay bricks and it was able to pass the required criteria for compressive strength, linear shrinkage and water absorption of the bricks. It was observed that the soil addition with the tailings bricks was more cost-effective than the one with cement additive.

3.8 CONCLUSIONS

Mine tailings can have several civil engineering applications. Studies show that they can be a good source of raw materials for brick manufacturing with some suitable modifications such as by addition of soil, lime or cement. This innovation will add value to brick manufacturing

through cost reduction by getting cheap mine tailings as raw materials and thus reduce the need to excavate new soil. This will further reduce CO₂ emissions, save land use for mine waste disposal and provide other environmental and economic benefits. The mining details presented for Western Australia indicate that there is a need to carry out analysis of the mine wastes so that they can be used effectively in large quantities, especially in brick manufacturing.

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CHAPTER 4

MINE WASTES IN WESTERN AUSTRALIA AND THEIR SUITABILITY FOR EMBANKMENT CONSTRUCTION

This chapter was presented orally and published by the American Society of Civil Engineers as a peer reviewed conference paper in the proceedings of Geo-Congress 2013, Stability and performance of slopes and embankments III, March 3 – 6, 2013, San Diego, California, U.S.A., as mentioned in number 7 in Section 1.7. The details presented here are the same, except some changes in the layout in order to maintain a consistency in the presentation throughout the thesis.

4.1 ABSTRACT

Western Australia (WA) is considered as a major mining jurisdiction because of its lucrative mining. The state holds large reserves of mineral deposits including iron and gold ores. The treatment processes of the ores are associated with large volumes of waste generation. Some of the wastes are mine tailings, waste rocks, fly ashes, and slags. The disposal of these wastes continues to create economic, environmental and land problems for the mining industries and associated communities. The increasing public awareness of the environmental impact of these activities is an incentive for establishment of alternative handling for these mine wastes. Among the various alternatives, reuse of wastes is the most sustainable way of solving the problem. This paper describes the wastes being generated in WA in large volumes and explores the potential of their use as a construction material for highway and railway embankments. It is found that the properties of some mine wastes are favorable for their use as highway and railway embankment materials in a cost-effective and sustainable way.

4.2 INTRODUCTION

The handling of mine wastes has become a major issue for most mining industries throughout the world. This process is seen as an additional cost to production without any direct revenue. The industries concerned need to think of a near-by space for their mine waste disposal with sustainable disposal methods that will not cause harm to the environment and the community. After a successful disposal, the waste sites need further monitoring and management. This is

done to prevent the possibility of erosion, acid mine drainage and dam failures (Yellishetty et al., 2008; Grangeia et al., 2011).

Western Australia (WA) is one of the mining jurisdictions of the world and it regularly faces the problem of mine waste handling. This state is endowed with abundance of mineral deposits such as iron ore, alumina, gold and many other mineral deposits and therefore it earns a high value from the mineral exports (Alexander, 1988). The availability of mine ores has attracted much investment in the mining sector and it has resulted in many operating mining sites. The high extent of mining activities in the state has brought about significant volumes of mine wastes which need to be handled by the mining industries. Some of the wastes generated could be in the form of tailings, waste rocks (rejects), mine overburden, fly ash and slags (Rampacek, 1982; Swane, 2009). The disposal of these wastes needs to meet legislations and community expectations. In order to eliminate the problems, the available literature reports some useful applications for these mine wastes (Choi et al., 2009; Klauber et al., 2011). Some of the mine wastes have applications in Civil Engineering (Skarzynska, 1995). In this paper, the details of some WA mine wastes are presented, and an attempt is made to compare the characteristics of these wastes with that of the construction materials used for highway and railway embankments.

4.3 TYPES OF MINES IN WA

According to the WA Department of Mines and Petroleum (2009), there are a large number of mines in WA. The iron ore mining is one of the top resource sectors in WA and it accounts for about 47% of the total value of WA's resources with a total production of 316 million tonnes during 2008-2009. The main finished product in this sector is the iron ore and after its production, it is exported to other countries as the main raw material for the production of metallic iron.

Gold industry is also one of the important resource sectors in WA. The state gold production was 136 tonnes or 4.4 million ounces as compared with 218 tonnes or 7 million ounces production in the whole Australia during 2008-2009. In percentage form, the gold production in WA was estimated to be 62% of the Australia's total gold production. Currently all Australia's gold ores are refined in Western Australia. Gold exports from the state amounted to \$16.8 billion during 2008 – 09 of which 31% or \$5.2 billion was the actual production from Western Australia. The remaining about 69% or \$11.66 billion can be attributed to the gold that was refined in and exported from Western Australia but this gold was actually produced from mining activities in other states, territories and overseas.

Other mining sectors in the state are petroleum, alumina, nickel, copper, diamond, coal, manganese and other minerals. Gold, petroleum and iron ore industries together account for 83% or 59 billion dollars of all the WA minerals and petroleum sales during 2009-2010 and this together forms the backbone of WA’s economy (WA Department of Mines and Petroleum, 2010). In total, the state contributes nearly 40% or more of the value of Australia’s mineral exports (Alexander, 1988).

4.4 MINING PROCESSES AND WASTE GENERATION

Mining processes begin with collection of the ores by exploration, blasting and excavation processes. Different wastes are generated at this stage, namely topsoil, mine overburden and waste rocks of different sources. After this process, the ore of interest is obtained. These processes are shown in Figure 4.1. The ore received at this stage needs further processing. The ore could be processed using techniques like crushing, grinding, concentration, leaching, and upgrading, which is done in order to get the mineral ores to have a higher percentage of mineral content (Wills, 1992). The ore is then leached by subjecting the concentrated ore to cyanidation. Iron ore process in WA does not go through leaching. The processing stage generates some wastes such as tailings, slags, ashes and process waste water as shown in Figure 4.1. The product from the leached ore is then smelted and sometimes made to undergo refinery if available before the final marketable product is obtained.

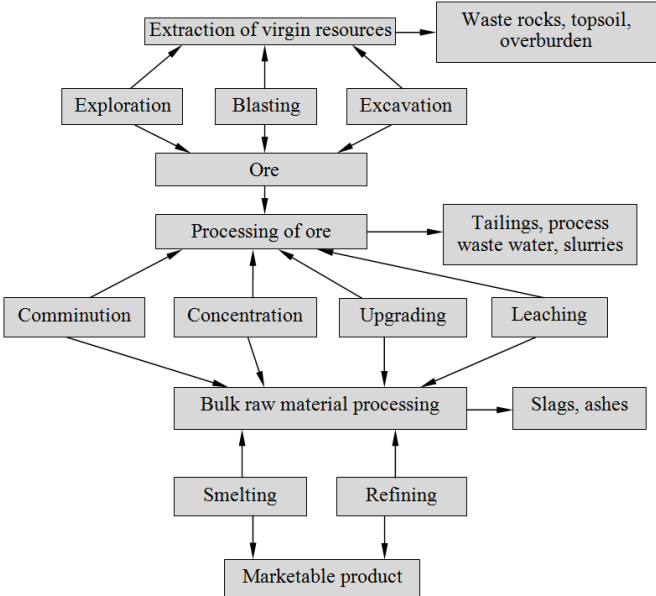


Figure 4.1: Mining processes and waste generation (Yellishetty et al., 2008; Kuranchie et al., 2012).

4.5 CHARACTERISTICS AND USABILITY OF MINE WASTES

4.5.1 Mine tailings

Mine tailings are the unwanted materials that are left over after separating the valuable minerals of economic interest from the gangue or wastes of an ore. Mine tailings are produced during ore processing stage which could include comminution, concentration, upgrading and leaching. Tailings are normally contaminated with heavy metals, solids, mill reagents and sulphur compounds and these could cause environmental pollution if they are not handled properly (Waldichuk, 1979). According to Wills (1992), the current practice of handling mine tailings is disposing it off into a tailings dam or in a tailings storage facility. In doing this, there is a requirement to construct the tailings dam close to the mine site to reduce transportation cost and also for convenience. This brings a limitation and problems in terms of site selection for tailings dam. Tailings are normally disposed off as slurry of high water content and could also contain coarse dry material (Wills, 1992). Low grade ores result in very large amounts of tailings. Therefore, this makes reuse of mine tailings most acceptable and more sustainable way of handling it. It could be reprocessed to recover additional valuable materials in the tailings and also to consider coarse tailing materials (20 – 30 mm) to be used as railway ballasts and aggregates. Tailings have other useful properties such as self cementing characteristics, which remove the necessity of adding cement when it is being used to fill mined-out areas and for other similar applications. The self cementing characteristics of the tailings is due to its slimy nature and the large quantity of sulphides it contains which oxidizes on contact with the air to form hard cement-like crust. A study by Celik et al. (2006) confirms this by using gold mine tailings as an additive in the manufacture of Portland cement. The results indicated that gold tailings between 5 to 15% as an additive could be feasible in Portland cement production if the gold tailings used in the cement are blended with silica fume and C type fly ash to attain the desired values of compressive strength of 60 N/mm².

To consider the mine tailings for reuse options, it is normally ideal to think of some improvement before its applications. Das et al. (2000) described a new development in managing iron ore tailings by converting them into value added products such as ceramic floors and wall tiles for building applications. They found that iron ore particles below 150 µm in size were discarded as tailings by the mines. They also tested the constituents of the tailings collected from different locations and their mix using standard techniques as XRD (Siemens D 500) with Ni filter and Cu (K α) radiation. The iron ore tailings were found to contain high percentage of silica. This high silica content in the iron ore tailings is considered

favorable in terms of the property and the raw material requirements for the production of ceramic tiles. It has been reported that that the iron ore tailings up to 40% by weight can be considered for use as a part of raw materials for ceramic floor and wall tiles due to its high silica content. The ceramic tiles from the iron ore tailing materials were found to be superior in terms of scratch hardness and strength. The new tiles from the iron ore tailings also maintain most of the other essential properties as the conventional raw materials used for ceramic tiles. The application of iron ore tailings in the ceramic tiles production was also found to be cost effective in comparison with the usual traditional clay for ceramic tiles production. Tables 4.1 and 4.2 present the major constituents of iron and gold ore tailings, respectively.

The mine tailings are often characterized by high concentrations of heavy metals such as lead, zinc, cadmium, sulphur, arsenic, alkaline or acidic conditions, and they could contain some of the toxic chemicals used to process the ore (Ye et al., 1999).

Table 4.1: Constituents of Iron Ore Tailings (after Das et al., 2000).

Constituents	Weight (%)
CaO	0.22
SiO ₂	51.12
Al ₂ O ₃	1.22
Fe ₂ O ₃	44.36
Loss on ignition	2.95

Table 4.2: Constituents of Gold Mine Tailings (after Roy et al., 2007).

Constituents	Iron-Deficient Gold Tailings (%)	Iron-Rich Gold Tailings (%)
CaO	8.4	50
SiO ₂	56.0	70
Al ₂ O ₃	11.9	90
Fe ₂ O ₃	10.2	119
MgO	8.6	135
Loss on ignition	2.0	155

For instance, to consider the mine tailings for revegetation applications, an ameliorative approach should be used. This relies on achieving the optimum conditions for plant growth by improving the physical and chemical nature of the mine tailings. This could be achieved by using lime, organic matter such as pig manure mixtures with the tailings. The lime in this sense neutralizes and reduces the acidic conditions, and the organic matter enhances the fertility of the tailings to support plant growth.

4.5.2 Mine waste rocks

Waste rock or overburden is a material that is mined from an open cast pit or underground workings in order to gain access to the rock material hosting economic grades. It normally includes topsoil, weathered and partially weathered overburden and primary or unweathered waste rock (Department of Resources, Energy and Tourism, 2009). Topsoil, sometimes cannot be considered to be a waste since it is a valuable resource for rehabilitation of the mine site upon closure.

Mine waste rocks do not contain minerals of interest or they may contain minerals in very low concentrations that cannot be economically processed during and after mining. It is sometimes called coarse tailings. Waste rocks are generated during the extraction of virgin resources which could involve exploration, blasting and excavation as shown in Figure 4.1. Waste rocks are normally returned underground into mined out pits for backfilling or they are left in waste rock dump in a stockpile.

The constituents of these mine waste rock dumps have been studied by the European Commission (2009) to know trace amounts of heavy metals naturally present in the minerals such as cadmium, antimony and some trace elements such as selenium, mercury, arsenic and dissolved metals in the form of iron and copper. Some other possible constituents are sulphur, calcite or calcium carbonate (CaCO_3), dolomite ($\text{CaMg}(\text{CO}_3)_2$), magnesite (MgCO_3) and some silicate minerals, depending on the nature and mineralogy of the ore. The chemistry and the composition of most waste rocks from different ore locations share certain similar features and this applies to most metalliferous, coal and industrial mineral deposits (Hitch et al., 2010). To consider mine waste rocks for further use, characterisation of the waste rocks should be considered due to the potential environmental problems. Some constituents of the waste rocks might cause oxidation and the production of acid leading to acid mine drainage. The sulphide ores such as pyrite are common in most sulphide minerals and they are usually associated with many metallic ore deposits and have potential of producing acid due to sulphide mineral oxidation and this could potentially lead to acid mine drainage (AMD). The presence of

calcite, dolomite, silicate and hydroxide minerals in the waste rocks can prevent oxidation of the sulphide minerals through buffering and consumption of hydrogen ions produced by the sulphide content in the waste rocks and this can render the waste rock non toxic to be considered for further use. This makes it possible to consider many other applications of the waste rocks compared to applications of the ore commodity mined initially.

Pilbara region of WA is where the majority of the state iron ore is mined and this region produces 160 metric tonnes of iron ore mine waste rocks annually. From this quantity of iron ore waste rocks, 50 metric tonnes comes from the Hamersley iron ore which is owned by the Rio Tinto iron ore group; and 105 metric tonnes from the BHP Billiton, and between 6 to 8 metric tonnes from the Robe iron ore. The pH of this iron ore waste rock in the Pilbara region of WA ranges from acidic (1.5) to alkaline (9.5) with lower electrical conductivity similar to ceramic materials and poor fertility Mulligan (1996).

4.5.3 Mine fly ashes

Fly ash is a residue generated during combustion. It can also be generated from mining extraction processes in bulk material processing like gold smelting and refinery as seen in Figure 4.1 and it comprises of fine particles that rise with the flue gases. Fly ash ranges in size from 0.5 to 100 μm and mostly consists of silicon dioxide (SiO_2), aluminium oxide (Al_2O_3) and iron oxide (Fe_2O_3) (Ahmaruzzaman, 2010). They are highly heterogeneous and are made up of glassy particles. The colour of fly ash could range from tan to gray to black, and it is abrasive, mostly alkaline and refractory in nature. Fly ash can be self cementing (Type C) or pozzolanic (Types F and C). They contain other essential elements such as phosphorus, potassium, calcium, magnesium, zinc, iron, copper, manganese, boron, and molybdenum, which are essential nutrients for the plant growth. The geotechnical properties of fly ash such as specific gravity (2.1 – 3.0), permeability, internal angle of friction, consolidation behavior, and specific surface area (170 – 1000 m^2/kg) make it suitable in its application as road construction materials and structural fills and hydraulic barriers in landfills (Ahmaruzzaman, 2010; Prashanth et al., 2001). The fly ashes normally contain pozzolanic properties and have lime binding characteristics which make it useful and suitable for its use as a part of raw material in cement manufacturing and as a constituent in concrete mixture. Fly ashes have other essential physicochemical properties such as bulk density (1.01 – 1.43 gcm^{-3}), hydraulic conductivity and specific gravity (1.6 – 3.1), particle size of diameter less than 0.010 mm, porosity, water retention capacity (6.1% at 15 bar to 13.4% at 1/3 bar) and surface area which make it suitable to be considered for use as an adsorbent.

Fly ashes are also used for agricultural purposes. Basu et al. (2009) found that in general, fly ash had low bulk density (1.01-1.43 g/cm³), hydraulic conductivity and specific gravity (1.6 – 3.1) with a moisture retention of 6.1% at 15 bar to 13.4% at 1/3 bar. In the same work, the constituents of fly ash was reported to include some plant nutrients such as phosphorus, potassium, calcium, magnesium and sulphur, and some micro nutrients like iron, manganese, zinc, copper, cobalt, boron and molybdenum except organic carbon and nitrogen which are suitable to be considered for its application in agriculture purposes. Relatively, the fly ash may contain heavy metals and may be naturally radioactive which could be toxic to plant growth. However, it has been reported that the effects of these heavy metals and the radioactivity content are within the permissible limits if the fly ashes are applied in optimum quantities.

4.6 SUITABILITY OF SOME MINE WASTES FOR EMBANKMENT CONSTRUCTION

The engineering properties of mine wastes are considered similar to soil materials. This is increasing the awareness for the use of mine wastes in place of natural soils as construction materials in embankments and other construction applications (Kamon et al., 2000). Particle size distribution of most tailings consists of mixed grained sizes. Roy et al. (2007) tested a sample of gold tailing and reported that it consisted of 33% clay-size particles, 17% silt-size particles and 50% sand-size particles and it had a specific gravity of 2.75. The percentage of clay-size particles in the tailings is significant to provide the required cohesive characteristics for the entire tailings.

To consider the mine tailings to be used in the construction of embankment, the tailings can be blended with the mine fly ashes. This will further enhance the compaction characteristics and lower the permeability of the entire tailing material to suit its usage for the embankment construction. According to SUDAS (2011), the optimum compaction parameters for highway and/or railway embankment material to be used in subgrade should be 95% of the Proctor density with over 4% of optimum moisture content. In terms of strength and stability, the compacted soil should have a California Bearing Ratio (CBR) greater than 10. All these parameters for the embankment material can be determined in accordance with the ASTM standards. The embankments with such parameters are considered good as a highway and railway embankments, which can support heavy loading without excessive deformation.

The resulting blend of the tailings and the fly ashes could be done in order to reach these optimum parameters for the embankment construction. The tailings contain lime as shown in Tables 4.1 and 4.2, and the lime will help neutralize and prevent the potential of acid mine drainage in the embankment. The fly ash also possesses some favorable properties for embankment such as its pozzolanic nature, high shear strength, ease of compaction, amenability to stabilization and self hardening. These properties will enhance the workability and performance of the embankments to be used as highway and railway embankments.

The subbase part of the highway and/or railway embankment can be made with local natural sand/soil which is porous and permeable to allow the required drainage from the embankment. The natural sand/soil can be lime treated if the percentage of the clay is found to be high. Mine waste rocks with some binder on the side slopes such as cement or slimy gold tailings which could also prevent erosion can then form the base course layer as the outer protective coarser layer of the embankment. This will allow the required drainage characteristics and also control erosion. The proposed typical cross-section of the embankment is shown in Figure 4.2.

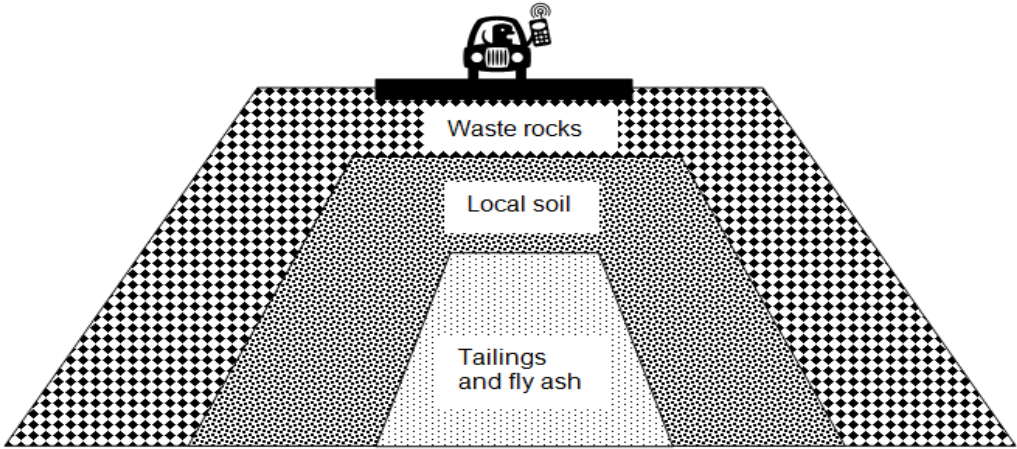


Figure 4.2: Schematic view of the proposed embankment constructed with mine wastes.

4.7 CONCLUSIONS

The sustainable use of mine wastes for highway and railway embankments is an attractive option for utilization of mine wastes in large volumes. This practice will reduce the need for excavating large quantities of natural soils and rocks for embankment construction, thus reducing carbon dioxide emissions, saving valuable land for mine waste disposal, and achieving other economic and environmental benefits.

The details of mine wastes presented for Western Australia indicate that there is a need to carry out analysis of the mine wastes so that they can be used effectively in significant quantities, especially as highway and railway embankment construction material in civil engineering projects.

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CHAPTER 5

STUDIES ON ELECTRICAL RESISTIVITY OF PERTH SAND

This chapter has been published as a Journal article in the International Journal of Geotechnical Engineering by Maney Publishing, United Kingdom as mentioned in number 1 in Section 1.7. The details presented here are the same, except some changes in the layout in order to maintain a consistency in the presentation throughout the thesis.

5.1 ABSTRACT

This paper presents experimental and numerical studies on electrical resistivity of Perth sand in its very loose state to very dense state. A simple laboratory experimental setup was developed to measure the electrical resistivity in terms of its apparent value, using the Wenner array of electrodes. The electrode spacing and depth were varied to investigate their influence on the resistivity values. The results indicate that an increase in electrode depth causes a decrease in resistivity; while an increase in electrode spacing results in an increase in resistivity for all the relative densities of sand. In view of limitations with respect to the geometry and electrode configuration of the laboratory setup, the values determined using the classical Wenner array expression, $\rho = 2\pi a(\Delta V / I)$ [I = current, ΔV = potential difference, a = equal electrode spacing, and ρ = electrical resistivity] required corrections by a suitable factor. To determine this correction factor for a specific electrode arrangement, say electrode depth = 150 mm, electrode spacing = 180 mm, as an example, in our resistivity box; finite element simulation was carried out using the commercial software COMSOL. The simulation results indicate that a correction factor of about 0.46 should be applied to the value calculated using the expression in order to obtain more realistic values. By employing this methodology, it was found that the electrical resistivity of dry Perth sand ranged from 60, 606 Ωm for the very dense state to 142, 857 Ωm for the very loose state.

5.2 INTRODUCTION

Sand is the most predominant soil in Perth and its surrounding areas (Byrne, 2009). Sandy soils are extensively used as structural fills and construction materials in Perth and its region because of their easy availability. Sandy grounds are common foundations for buildings, highway and runway pavements, railway tracks, and other structures. The feasibility of such

projects is greatly impacted by the ground conditions, and therefore it is often essential that a quick evaluation of the characteristics of the ground should be undertaken. For many engineering projects which do cover such large areas, a geophysical investigation, which utilises an electrical resistivity survey, is the most convenient method for estimating the properties of the target geomaterials; because this method is non-destructive, and it is less expensive in terms of time and money (Abu-Hassanein *et al.*, 1996; Coskun, 2009).

The variation of the electrical properties of the geomaterials easily provides their basic information and geotechnical characteristics (McCarter and Desmazes, 1997, Bryson, 2005; Yan *et al.*, 2012). Electrical resistivity has many other geotechnical and construction applications such as: 1) evaluation of the densification or extent of compaction of soil (McCarter, 1984; Seladji *et al.*, 2010; Yan *et al.*, 2012); 2) monitoring of the safety of dam embankments (Foster *et al.*, 2000; Oh and Sun, 2008; Sjö Dahl *et al.*, 2008; Oh, 2012); 3) drainage problems (Coskun, 2009); 4) monitoring the moisture content of municipal solid waste in landfills (Grellier *et al.*, 2007); 5) evaluation of mortar infiltration in buildings (Farook *et al.*, 2013); 6) application in railway engineering and other similar projects to detect defects (Barta, 2010; Sirieix *et al.*, 2013); and 7) detection of soil salinity in agricultural applications (Adam *et al.*, 2012). It has been established that if the core material within an embankment (normally clayey particles) flows out as a result of internal erosion or piping, the designated area of the core material will have high electrical resistivity (Oh and Sun, 2008). Coskun (2009) observed that the areas with high water content lowered the resistivity of the soil, indicating poor drainage zones. Grellier *et al.* (2007) reported that the electrical resistivity measurements of the landfill materials taken at a bioreactor landfill site could be used to assess the effectiveness of a recirculation system, which were installed. The electrical resistivity survey can therefore be satisfactorily used as a tool to facilitate the management of landfills. Electrical resistivity also measures the corrosive potential of soil, which has application for buried steel pipelines in industrial settings (BS – 1377). Normally the lower the resistivity, the higher will be the corrosivity.

This paper is distinct, in that, to the authors' understanding; no literature has previously been reported, specifically referring to the electrical resistivity of Perth's sandy soil. This study therefore, demonstrates a method through which the resistivity range of Perth sandy soils could be studied in order to evaluate its geotechnical properties to be applied in construction work in Perth region, and also in other similar parts of the world. The study will provide first-hand information, by enabling correct inferences to be made about the subsurface conditions of the sandy ground. In this study, an electrical resistivity measurement

apparatus was fabricated in the laboratory based on the concept of the Wenner electrode array, and this was used to determine the electrical resistivity of sand, which covers a significant part of the Perth region, at different relative densities. Finite element (FE) simulation was then carried out by utilising the COMSOL software, in order to assist experimental data processing, and provide correction for experimental limitations.

5.3 EXPERIMENTAL PROGRAMME

The design of the apparatus for measuring the electrical resistivity of dry Perth sand in the laboratory is based on the Wenner array experimental setup. Measurements of the voltage and current were taken directly from the apparatus. The resistivity was then evaluated using an equation derived from Ohm's law, which relates the current density \vec{J} and the electric field \vec{E} as (Walker, 2008):

$$\vec{E} = \rho \vec{J} \quad (5.1)$$

where ρ is the electrical resistivity of the material in Ωm . Equation (5.1) is commonly used as a fundamental equation for measuring the electrical resistivity of geo-materials. For the Wenner electrode array, as shown in Figure 5.1, the resistivity is expressed as (Telford *et al.*, 1990; Kearey *et al.*, 2002; Samouelian *et al.*, 2005, Shukla and Sivakugan, 2011):

$$\rho = 2\pi a \left(\frac{\Delta V}{I} \right) \quad (5.2)$$

where ΔV is the electrical potential difference in volts (V) between the two inner electrodes M and N , I is the induced electric current in amperes (A) in the medium between outer electrodes A and B , and a is the distance between equally spaced electrodes.

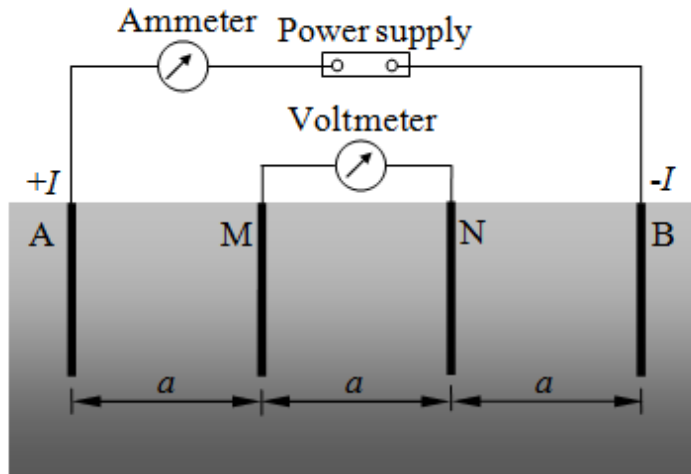


Figure 5.1: Wenner array of soil electrical resistivity measurement (adapted from Shukla and Sivakugan, 2011)

The soil materials, of which the ground is comprised, are generally inhomogeneous and anisotropic; therefore, the measured resistivity may vary with the relative positions of the electrodes in terms of their depth and spacing. Thus the measured resistivity is generally an average resistivity termed the *apparent* or *effective resistivity*.

5.3.1 Materials

The sand used for the various construction projects, known as ‘brickies sand’, was collected from a quarry site, about 40 km North from Perth city. It consisted of some natural fibres, which were removed before the tests. The properties of the dry sand are given in Table 5.1. The soil was mainly sand with 3.85 % fines, and it was used in the tests in its dry condition. All the properties were measured as per the relevant standards listed in AS: 1289 (2000). Figure 5.2 shows the particle-size distribution curve of the sand. The soil was classified as the poorly graded sand (SP) as per the Unified Soil Classification System (USCS).

Table 5.1: Physical properties of the dry Perth sand.

Property	Value
Fines (%)	3.85
Sand (%)	96.15
Specific gravity	2.66
Minimum dry density, ρ_{dmin} (kg/m ³)	1392
Maximum dry density, ρ_{dmax} (kg/m ³)	1598
Effective grain size, D_{10} (mm)	0.18
D_{60} (mm)	0.38
D_{30} (mm)	0.26
Coefficient of uniformity, C_U	2.11
Coefficient of Curvature, C_C	0.99
Soil classification as per USCS	Poorly graded sand (SP)

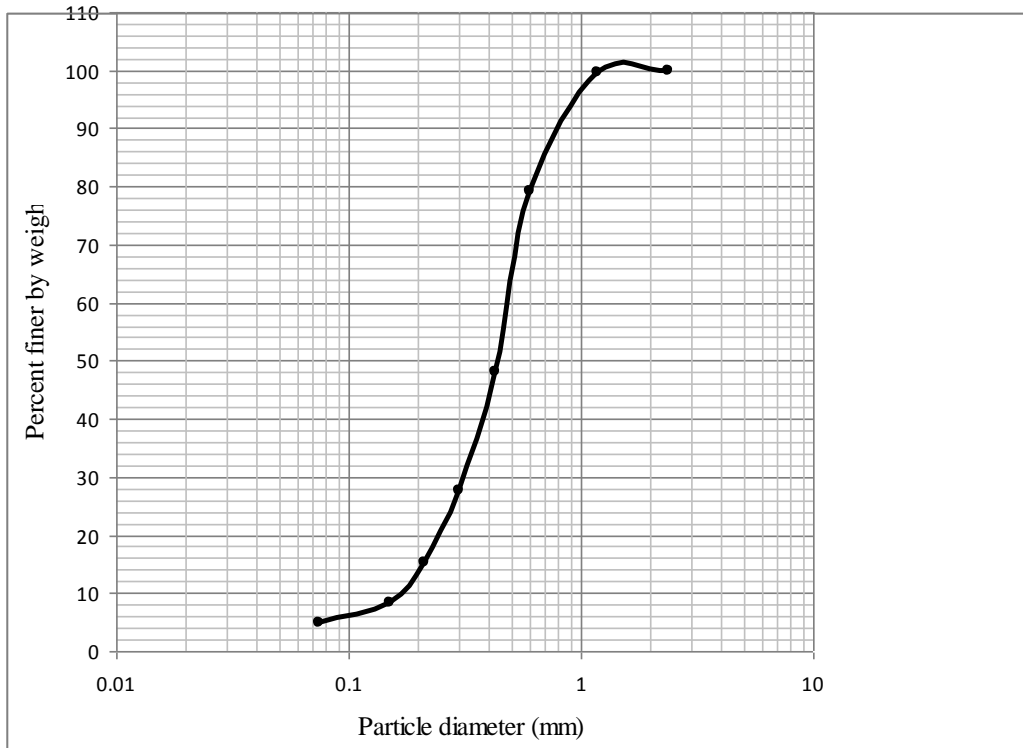


Figure 5.2: Particle-size distribution curve of the sand that represents a significant part of the Perth region.

5.3.2 Test apparatus

An electrical resistivity measurement box (length of 600 mm, width of 300 mm and height of 450 mm) was fabricated using a non-conducting transparent perspex sheet of 20 mm thickness. Four hot-galvanised steel rods with a diameter of 6 mm and a length of 50 mm

were used as the electrodes arranged within the sand in the Wenner array. The electric circuit uses two multimeters; one for measuring the current (ammeter), and the other for measuring the voltage (voltmeter). The current source was a fully charged 12 V battery with an in-built power inverter, which was capable of delivering 230 V alternating current with 50 Hz frequency and 400 W power. An alternating current (AC) was used in the study because it is able to overcome the problem of polarisation (Yan *et al.*, 2012). All the electrical units were mounted on a perforated wooden plank. The complete experimental apparatus is shown in Figure 5.3, which is similar to the apparatus described by Herman (2001).

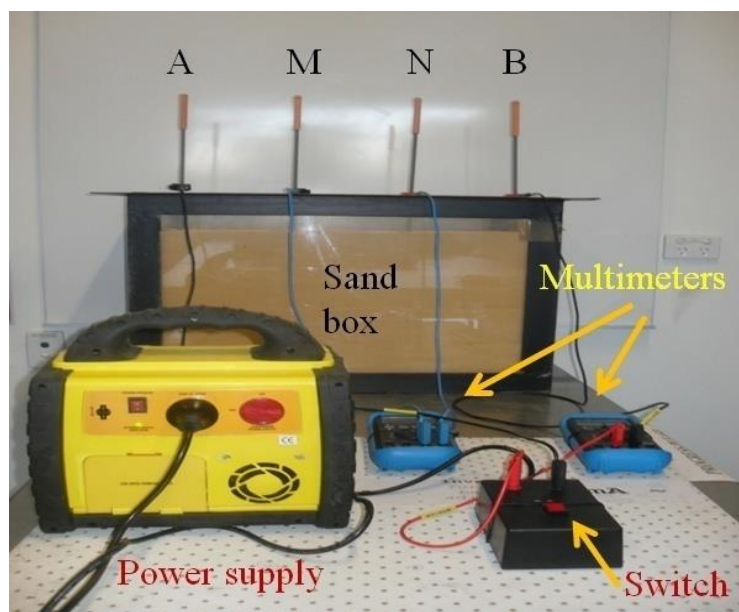


Figure 5.3: Laboratory resistivity experimental setup with connected accessories

5.3.3 Test procedure

In the test, the dry sand beds were created in the electrical resistivity box in a range of densities, from very loose to very dense states. The values of the dry density (ρ_d) corresponding to five different values of relative density (D_r) were as follows: 1392 kg/m³ ($D_r = 0\%$), 1439 kg/m³ ($D_r = 25\%$), 1488 kg/m³ ($D_r = 50\%$), 1541 kg/m³ ($D_r = 75\%$) and 1598 kg/m³ ($D_r = 100\%$). As the internal volume of the box was known, the dry mass of sand required to fill the box for each relative density could be calculated. The weighed dry sand was placed in the box by free-fall for $D_r = 0\%$, and by suitable compaction in four lifts for other dense conditions; so that homogenous sand beds were obtained within the box for each resistivity measurement test. The four electrodes *A*, *M*, *N* and *B* as in Figure 5.3 were

carefully pushed into the sand bed in such a way that significant disturbance could be avoided. A slow pushing method was employed to ensure good contact with the sand from an electrical resistivity point of view. Before the electrodes were inserted into the sand, the lower ends of the electrodes were moistened with tap water, in order to provide a good contact between the electrodes and the sand. The moistening of the lower end of the electrodes does not affect the measurement in any way, as only a small amount of tap water is used (Herman, 2001). The arrangement of the electrodes was kept as per the Wenner array shown in Figure 5.1. The electrode depth and spacing were varied from 100 to 300 mm and 100 to 180 mm, respectively. The two outer electrodes *A* and *B* were connected to a current source, along with a switch and an ammeter to form an electric circuit. The two internal electrodes *M* and *N* were connected to a voltmeter to measure the potential difference across them. All the connections were made using 16 gauge braided speaker wire with alligator clips at the lower end of the wire which connects the electrodes. The laboratory room temperature was controlled to approximately 20 °C, to avoid any variation in electrical resistivity due to temperature effects (Munoz-Castelblanco *et al.*, 2011). The voltage and current so measured were used to determine the apparent soil resistivity using equation (5.2) for all the cases of electrode depths and spacings. However, equation (5.2) was derived on the basis of ideal conditions, which are normally not met in field and laboratory experiments. Hence finite element (FE) simulations were carried out in order to obtain the required geometrical correction factors. This will be further discussed in the numerical simulation, results and discussion sections.

5.4 NUMERICAL SIMULATION

The derivation of equation (5.2), which provides an expression for the determination of apparent electrical resistivity using the measured voltage and current, is based on the following three key requirements and assumptions (Telford *et al.*, 1990; Adam *et al.*, 2012):

1. The electrodes are buried in a homogeneous, isotropic and semi-infinite medium,
2. The air above the medium has zero electrical conductivity, and
3. The four electrodes are located at the air-soil interface as point sources.

These three requirements cannot all be achieved practically in laboratory and field measurements. In particular, in order to meet the third requirement of achieving point sources in the experiments, Narayan (2011) recommended that the electrode depth in the medium should be at least 20 times lower than the electrode spacing. In view of the practical limitations of the laboratory conditions, the third requirement that was required for the

validity of equation (5.2) could not be directly achieved. However, in the present study, we ensured that the resistivity box was filled with sand in such a way as to achieve homogeneity and isotropy, and a correction was applied to the results to provide for a point-source equivalent electrode. Therefore, the apparent resistivity values determined using equation (5.2) with current and voltage measured experimentally in the present study was corrected in order to achieve more realistic values of electrical resistivity, by utilising a finite element (FE) simulation of the experimental model, which is demonstrated here for the case of electrode spacing of 180 mm with electrode depth of 150 mm only. The electromagnetic module (Direct current, DC, conductive mode) of COMSOL software was used in our simulation, in which the following general Laplace equation was applied as the starting point (Telford *et al.*, 1990):

$$\vec{\nabla}^2 V = 0 \quad (5.3)$$

where V is the electrical potential. It is noted that in our experiments an alternating current was used. However, due to its low frequency (50 Hz), the effect of both capacitance and inductance are insignificant, therefore, the errors in simulation resulting from using the DC scheme of COMSOL software is negligible. In the simulation process a model was set up and a digitisation based on the meshing grid was carried out. Equation (5.3) was used to resolve for V on each node, while for all other points in the space, the solution was evaluated using interpolation. The space distribution of the current density was then calculated using equation (5.4).

$$\vec{J} = -\left(\frac{1}{\rho}\right)\vec{\nabla}V \quad (5.4)$$

The 3D geometrical models, each composed of a box containing sand sample, and four metal electrodes arranged in the Wenner array as shown in Figure 5.3, were simulated with the software. The numerical model-I shown in Figure 5.4 (a), which was used for the initial simulation, has exactly the same dimensions as our experimental setup shown in Figure 5.3. Further simulations were carried out using three additional large-size sand box models, having different geometrical dimensions, but with electrodes the same as those used in the experiments carried out on the physical Wenner array.

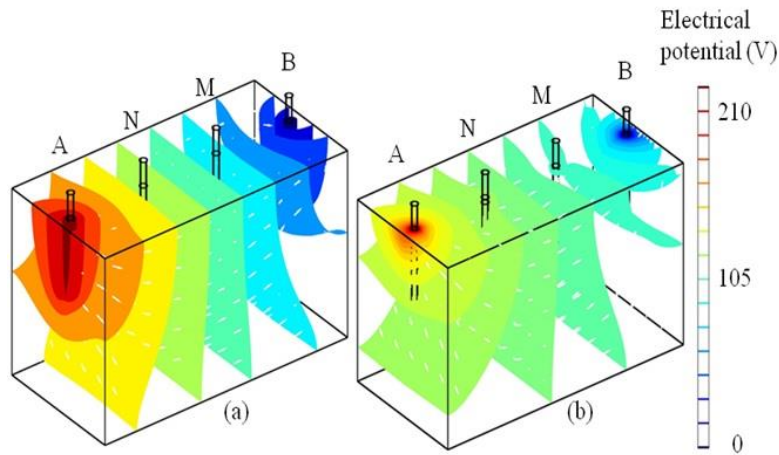


Figure 5.4: Simulated potential isosurface plots: (a) Normal size electrodes as experimental, and (b) with electrodes shortened

The purpose of the additional models used in the simulations was to investigate the effect of the size of the experimental apparatus on the resistivity values calculated from equation (5.2). The dimensions of all the geometrical models used in the simulations are summarised in Table 5.2. Again, in order to check the effects of not achieving the third requirement (i.e. electrodes being point sources), further FE models were then tested with the box of the same size as that of the experimental box, but with the electrodes shortened; Figure 5.4 (b), so that electrodes can be assumed to be point sources, according to Narayan (2011).

Table 5.2: The dimensions of the model boxes used in the simulation.

Model No	Length (m)	Width (m)	Height (m)
I	0.6	0.3	0.36
II	1.2	0.6	0.72
III	2.4	1.2	1.44
IV	4.8	2.4	2.88

For the boundary conditions, the top ends of the four electrodes i.e., *A*, *M*, *N* and *B* shown in Figure 5.5 were set to 230 V potential (*A*), electrical insulation (*M*), electrical insulation (*N*) and the ground (*B*), respectively.

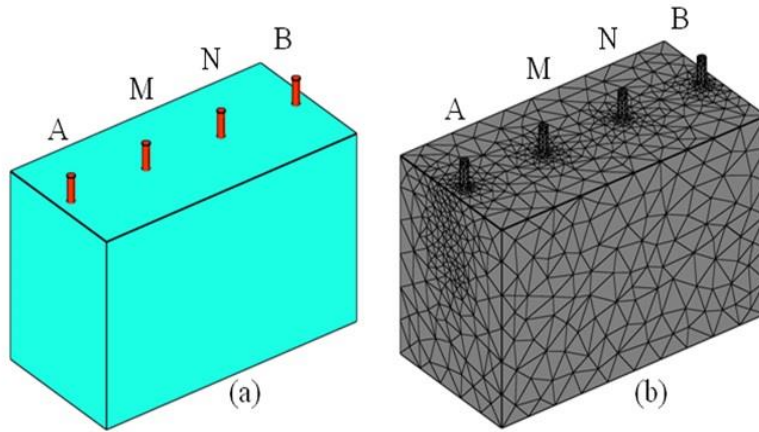


Figure 5.5: (a) Geometrical model, and (b) mesh used in the numerical simulation

All the six sides of the soil box, as well as the contact faces between the electrodes and the air (excluding the aforementioned four top ends), were set to electrical insulation. The mesh was generated by the software meshing tool using size control, so that the regions close to the electrodes, where large voltage and current gradient were expected, had denser mesh elements. The convergence test was carried by increasing the number of mesh point step by step, while comparing the simulation results. In each step the number of the mesh points is approximately doubled. Initially when the number of mesh points was low (in the order of 10^2), one step increase resulted in a variation of about 2 to 5%. With an increase of the mesh points, the variation in results diminished. The optimised mesh was obtained when the variation of the result of the last three simulations was narrowed down below 0.1%. Further increase in the mesh point only caused a very marginal improvement in accuracy, but a significant increase in the computational time was observed. The number of mesh element for the various models ranged from 3294 to 3958. Examples of the geometrical and mesh models used in the simulation are shown in Figures 5.5 (a) and 5.5 (b), respectively. In Figure 5.5, the four red-coloured rods are the conduction electrodes, labelled as *A*, *B*, *M* and *N* and the light green-coloured box represent the sand medium. In all the simulations, the sand in the box was assumed to be homogenous and isotropic and a constant conductivity was assigned over the complete sand domain. The contact resistance between the electrodes and the sand was neglected. In the first phase of the simulation, a parametric scan was carried out, using the resistivity (conductivity) as the parameter, with a 230 V voltage applied across the two outer electrodes *A* and *B*.

From the simulation results, the current through the two outer electrodes was evaluated by the boundary integration, and was compared with the value of current obtained

from the experiment. Finally the corresponding resistivity which results in a perfect match between the simulated current and the experimentally determined current was identified as the simulated resistivity.

5.5 RESULTS AND DISCUSSION

Based on the experimental study, Figure 5.6 shows the variation of apparent resistivity of sand with electrode depth for very loose to very dense states with relative density D_r equal to 0%, 25%, 50% 75% and 100%, respectively. It is observed that the apparent resistivity of sand decreases with an increase in electrode depth for any relative density. This trend may be attributed to the fact that if the electrodes are deep in the sand, more electric current passes through the sand medium for a given electric potential and electrode spacing; thus the resistivity decreases, as evident from equation (5.2). It is also noted that for any electrode depth, the resistivity is found to decrease as the relative density increases. For example, for electrode depth of 150 mm, the resistivity values of sand are 294 kΩm, 221 kΩm, 170 kΩm, 139 kΩm and 132 kΩm for D_r equal to 0%, 25%, 50%, 75% and 100%, respectively. The decrease in resistivity with an increase in relative density may be caused by more intimate contact between the sand grains when the sand is relatively dense; this means more ionic content per unit volume becomes available in dense state, allowing the current to pass freely. Figure 5.7 shows the variation of apparent resistivity of sand with electrode spacing for very loose to very dense states with relative density D_r equal to 0%, 25%, 50% 75% and 100%, respectively, based on the experimental study.

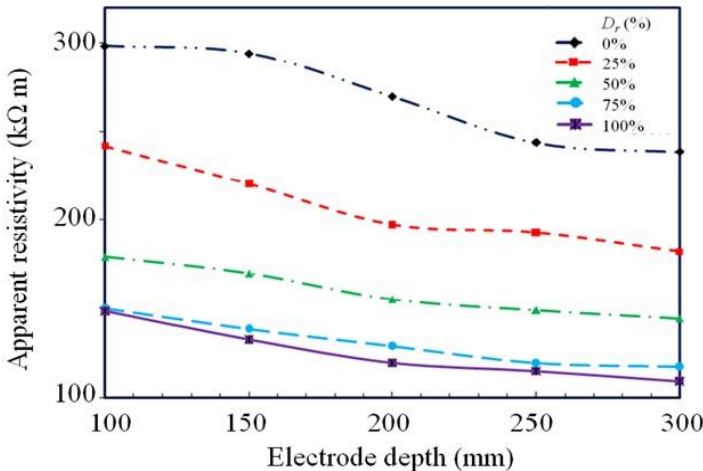


Figure 5.6: Variation of apparent resistivity of sand with electrode depth at constant electrode spacing of 180 mm.

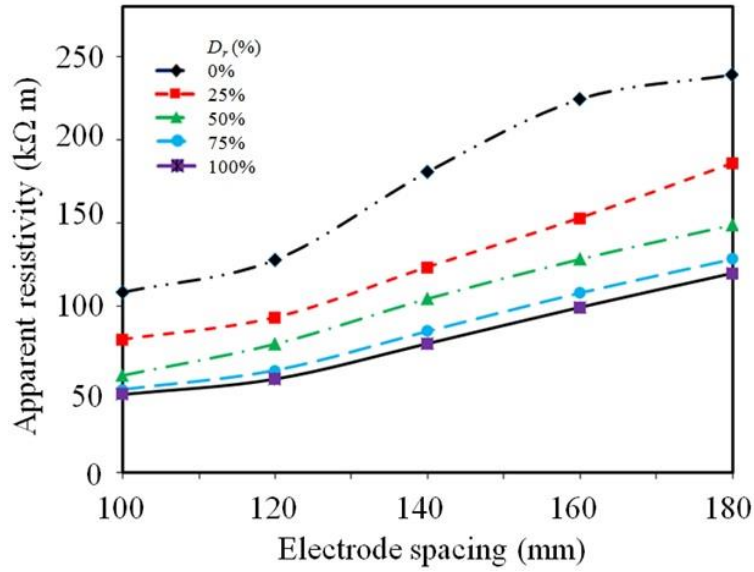


Figure 5.7: Variation of apparent resistivity with electrode spacing at constant electrode depth of 150 mm.

It is noticed that the apparent resistivity increases with an increase in electrode spacing for any relative density. It is likely that as the electrode spacing increases, less current passes through the sand medium because of higher resistance for a given electric potential and the point electrode as evident from equation (5.2). It is also observed that for any electrode spacing, the resistivity is found to decrease as the relative density increases. For example, for electrode spacing equal to 180 mm, the resistivity values of sand are 239 kΩm, 186 kΩm, 149 kΩm, 128 kΩm and 120 kΩm for $D_r = 0\%$, 25%, 50%, 75% and 100%, respectively.

When the experimental and the simulated resistivities were compared, it was observed that the experimental resistivity values differ from the simulated resistivity. It should be noted that, the simulated resistivity values refer to the modelled resistivity values that correspond to a perfect match between the simulated and the experimental current. For the range of samples of various relative densities, the simulated values of electrical resistivity were found to be about 50% of the experimentally obtained resistivities. The simulated values of the resistivity may be assumed to be more realistic values in view of the removal of the limitations regarding equation (5.2). The experimental resistivity (ρ_e) and simulated resistivity (ρ) values of sand are given in Table 5.3 and also shown in Figure 5.8. The last column of this table defines a factor λ as a ratio of the simulated resistivity to the experimental resistivity.

Table 5.3: Comparison of experimental and numerically simulated apparent resistivities with relative densities of sand at electrode depth of 150 mm and electrode spacing of 180 mm.

Relative density D_r (%) / dry density of sand	Voltage (V)	Current (mA)	ρ_e Ωm	ρ Ωm	$\lambda = \frac{\rho}{\rho_e}$
0 (1392 kg/m ³)	39.00	0.15	294,053	142,857	0.49
25 (1439 kg/m ³)	37.05	0.19	220,539	100,000	0.45
50 (1488 kg/m ³)	37.55	0.25	169,872	74,074	0.44
75 (1541 kg/m ³)	39.19	0.32	138,508	62,500	0.45
100 (1598 kg/m ³)	37.52	0.32	132,606	60,606	0.47

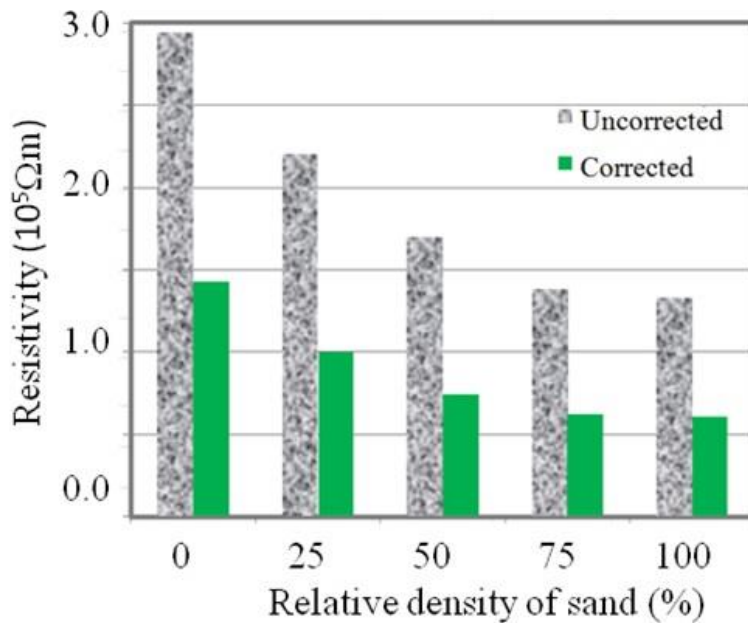


Figure 5.8: Experimental (or uncorrected) and simulated (or corrected) values of resistivity of sand at different relative densities.

Therefore,

$$\rho = \lambda \rho_e \quad (5.5)$$

Thus, the experimental value ρ_e , which is based on the ideal equation (5.2), may be corrected by multiplication with λ , which may be called the *resistivity correction factor*. In other words, equation (5.2) can be modified using equation (5.5) as:

$$\rho = 2\pi a \lambda \frac{\Delta V}{I} \quad (5.6)$$

In Table 5.3, it is noted that λ can be assumed to be approximately 0.46, which is an average value. This value can be used to correct the experimentally determined value based on equation (5.2) for the experimental setup similar to the one used in the present study. Figure 5.8 clearly provides the variation of resistivity values with relative density for both experimental and simulated resistivities. Since λ does not vary significantly with relative density, it can be stated that the correction factor is only determined by the geometrical configuration of the particular experimental setup, but is not linked to the relative density and other properties of the sand. As such, the final result of the correction factor for our particular experimental setup can be considered to be 0.46, by taking an average of the 5 values for the correction factors shown in column 6 of Table 5.3. This value can also be applicable to other types of soil, provided that the experimental setup of the same geo-metrical configuration is used. Herman (2001) did not introduce a correction factor to the geometrical setup; this work brings an improvement by the introduction of the correction factor to correct the geometry of the laboratory experimental setup based on the limitations in the laboratory apparatus.

In order to understand the difference in experimental and simulated resistivities, it is noted that the medium (sand) was assumed to have an infinite volume occupying the lower half of the space. Variation in the size of the sand box when conducting the simulations resulted in significant changes in the potential distribution, as presented in Figures 5.9 (a-d).

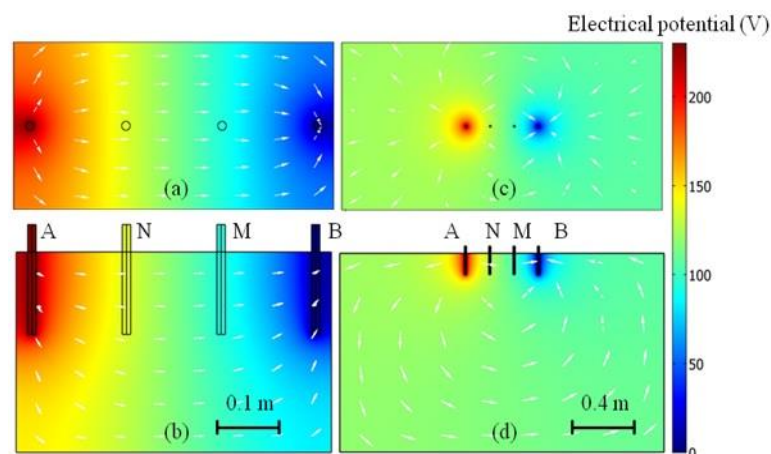


Figure 5.9: Simulated potential distribution resulted from the Wenner array.

Hence applying equation (5.2) results in an overestimation of the resistivity, because the cross section of the current path is reduced significantly in the simulation model, compared to that assumed with equation (5.2). The resistivity results obtained by simulating additional sand box models having different geometrical dimensions were compared. Examples of the models are given in Figures 5.9 (a-d), where Figure 5.9 (a) demonstrates the top view of model I, Figure 5.9 (b) the side view of Model I, Figure 5.9 (c) the top view of Model III, and Figure 5.9 (d) the side view of Model III. The white arrows indicate the direction of the current (not to scale). It was observed that the distribution of potential changed significantly due to the size effect. When the dimensions reached 4.8 m × 2.4 m × 2.88 m (Model IV in Table 5.2), further increases in the size of the box resulted only in marginal changes in the resistivity (< 2%). As expected, the difference between the simulated and the experimental resistivity was narrowed, and the simulation result is approximately 30% lower using Model IV.

Further, the third assumption for equation (5.2) requires all the electrodes to be point sources, instead of the metal rods as used in our experiments. Intuitively, application of equation (5.2) also in our experiments will result in an overestimation of the resistivity, since the finite length of the bar electrodes in the experiments and the simulation model will induce a larger current than that produced by point electrodes. This is evident in Figure 5.6, where an increase in electrode depth results in a decrease in resistivity. In addition, the numerical results have also indicated that the measured current between electrodes *A* and *B* is also dependent on the contact area between the electrodes and the sand, which is not reflected in equation (5.2). Finally, I carried out the simulation with model of a large box and point electrodes, which fulfils the assumptions of equation (5.2). With the finite surface area of the real electrodes taken into account, an excellent agreement, with difference below 5%, has been achieved between the simulation results and those obtained using equation (5.2) with experimental data. This verifies the modelling method, and validates the approach of using the resistivity correction factor in equation (5.6), in the case when the equation assumptions of equation (5.2) are not fulfilled in the experimental setup.

Figure 5.10 shows the variation of the simulated/realistic values of the apparent resistivity of sand with its relative density D_r . It was observed that the resistivity decreases significantly and nonlinearly, with an increase in D_r up to about $D_r = 75\%$, beyond which an increase in D_r does not cause any significant change. This observed variation is consistent with the general trends reported earlier (Seladji *et al.*, 2010; Dijkstra *et al.*, 2012). As can be seen from Table 5.3 and Figure 5.10, the simulated/realistic electrical resistivity ranges from

60,606 Ωm for a very high dense sand ($\rho_d = 1,598 \text{ kg/m}^3$, $D_r = 100\%$) to 142,857 Ωm for the very loose sand ($\rho_d = 1,392 \text{ kg/m}^3$, $D_r = 0\%$).

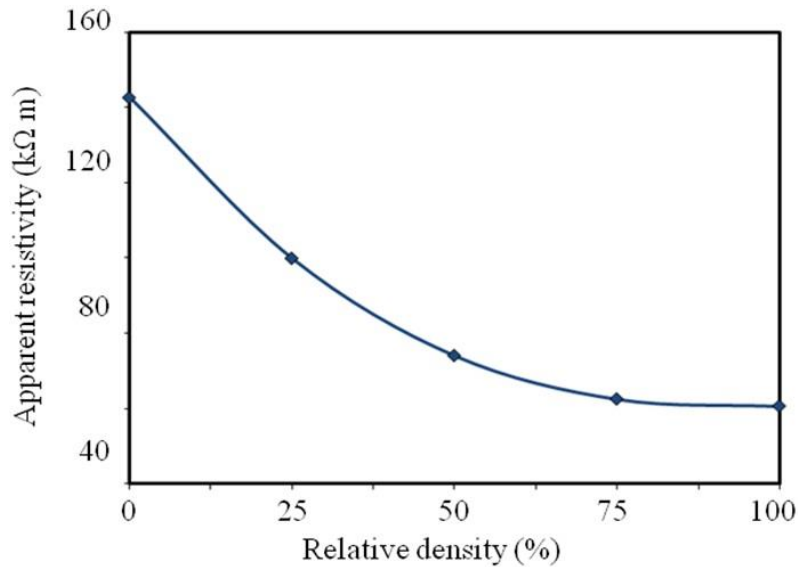


Figure 5.10: Variation of apparent resistivity with relative density at electrode depth and spacing of 150 mm and 180 mm, respectively.

The apparent resistivity of dry sands was previously reported for different places to be about 100,000 Ωm (Fukue *et al.*, 1999; Munoz-Castelblanco *et al.*, 2011). Thus the simulated values of the apparent resistivity of dry Perth sand at different compacted conditions as reported in Figure 5.10 are realistic, and they can be used by practising engineers working with dry Perth sand. In fact, Figure 5.10 can be used as a chart for determining the apparent resistivity of dry sands at any relative density for which the experimental or simulated resistivity value is unknown.

5.6 CONCLUSIONS

This paper presents the details of the experimental and numerical studies carried out for the determination of the electrical resistivity of dry Perth sand. The electrical resistivity experimental apparatus that follows the Wenner array was developed and the numerical simulation was carried out to correct the experimentally determined values, in view of the geometrical limitations of the apparatus. The effects of: 1) relative density of sand; 2) electrode depth; and 3) electrode spacing on resistivity were investigated. Based on the study presented in the previous sections, the following general conclusions can be drawn:

1. The value of electrical resistivity determined by using the small-scale laboratory apparatus depends greatly on the electrode depth and the spacing of electrodes. The results show that an increase in electrode depth causes a decrease in the resistivity while the increase in the electrode spacing results in an increase in the resistivity.
2. Equation (5.2), which is commonly used in the determination of electrical resistivity of geomaterials by the Wenner electrodes array, is based on several assumptions. These assumptions are difficult to achieve when conducting laboratory measurements. Equation (5.6) is presented as an improvement over equation (5.2), by incorporating the resistivity correction factor λ , which is found to be approximately 0.46 for the experimental setup used in this work. Being independent of types of soil, this value of λ can also be applicable to other types of soil, provided that the experimental setup of the same geometrical configuration is used.
3. If someone uses an experimental apparatus similar to the one developed in this study; it is essential that equation (5.6) rather than equation (5.2) should be utilized for calculating more realistic values of electrical resistivity by substituting the experimental current and voltage. In conjunction with the improved equation (5.6), the laboratory setup developed to determine the electrical resistivity under laboratory conditions, can be a cost-effective alternative to any commercial resistivity measuring apparatus.
4. The electrical resistivity of dry Perth sand was found to range from 60,606 Ωm to 142,857 Ωm for very dense to very loose conditions, respectively. This range is in close agreement to the resistivity values of sand reported in the literature for other places. It was also noted that the resistivity of sand decreases with an increase in its relative density.
5. Figure 5.10 can be used as a chart for determining the electrical resistivity of dry sand, which is similar to the dry Perth sand, by measuring the relative density of the sand only, and vice versa. This may help field engineers in characterizing the behavior of sand, especially when conducting a preliminary analysis, and when designing geotechnical and geo-electrical systems.

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CHAPTER 6

ELECTRICAL RESISTIVITY OF IRON ORE MINE TAILINGS PRODUCED IN WESTERN AUSTRALIA

This chapter has been published as a Journal article in the International Journal of Mining, Reclamation and Environment by Taylor and Francis Group, United Kingdom as mentioned in number 2 of Section 1.7. The details presented here are the same, except some changes in the layout in order to maintain a consistency in the presentation throughout the thesis.

6.1 ABSTRACT

This paper presents the details of an experimental study on the electrical resistivity of iron ore mine tailings produced in Western Australia. In the study, an experimental setup is developed based on the Wenner Array, and has been used for determining the electrical resistivity of the tailings at different relative densities in dry and fully saturated conditions. The apparent electrical resistivity of the iron ore mine tailings ranged from 11 k Ω m for a very dense state to 19 k Ω m for a very loose state in dry condition; while for the fully saturated condition, the resistivity ranged from 20 Ω m to 31 Ω m for very dense state to very loose state, respectively. The results are discussed for their practical applications such as identifying weak zones in tailing embankments, extent of corrosion in buried steel pipelines, and extent of degree of compaction in structural fills.

6.2 INTRODUCTION

The availability of large volumes of mineral deposits in Western Australia (WA) has resulted in many operating mines in the state. The major deposits are iron ore, gold, alumina and petroleum, which provide a high revenue from their exports to the state (Alexander, 1988). For instance, in 2008-2009 there was a total production of 316 million tonnes of iron ore (WA Department of Mines and Petroleum, 2009). The high mining activity has resulted in significant volumes of mine tailings in the state. The majority of the state's iron ore is mined in the Pilbara region and this region alone produces more than 160 million tonnes of iron ore mine tailings annually (Mulligan, 1996). According to Price (2004), on average for every tonne of iron ore, two tonnes of iron ore tailings are produced. This information indicates that about 632 million tonnes of iron ore mine tailings are produced every year in WA.

The continuous accumulation of the mine tailings in the state causes problems to the community concerning health issues and sustainability. Some of the problems relate to erosion, development of acid mine drainage (AMD) and dam failures, and high cost of rehabilitation to the mining industries upon mine closures. The problems have created the awareness of sustainability issues, and researchers are therefore focusing on improved ways in which the mine tailings can be managed or recycled effectively. Among the various options available, reuse of the mine tailings is receiving more attention and is the most sustainable solution to the problems mentioned (Packey, 2012). It has been reported that the mine tailings have significant applications in civil engineering such as brick manufacturing, aggregates for concrete, manufacturing of ceramic tiles, and construction materials for embankments and pavements (Skarzynska, 1995; Roy et al., 2007; Chen et al., 2011; Kuranchie et al., 2013).

In order to use the iron ore mine tailings for civil engineering applications, there is a need to find more convenient and practical ways of characterizing and predicting the engineering and geotechnical properties of the tailings so that the field engineers can rapidly make decisions for their applications. One of the simplest ways to achieve this task is the use of geophysical methods (Abu-Hassanein et al., 1996). Electrical resistivity survey is the most convenient geophysical method because it is non-invasive, sensitive and inexpensive, and moreover, it can be conducted quickly to characterize the material through the variation of the electrical properties. According to (McCarter & Desmazes, 1997), the application of electrical measurements for the determination of engineering properties of soil gives a clear and true characterization of the bulk material. The electrical characteristics of soil materials provide basic information and geotechnical properties of the soil. Sufficient information concerning the electrical resistivity of iron ore mine tailings, especially available in Western Australia, which can be beneficial when considering them for utilization in civil engineering applications, continues to be lacking in the literature. Therefore, this study focuses on determining the electrical resistivity values of the iron ore mine tailings, and establishing their relationships with geotechnical properties for practical applications.

6.3 METHODOLOGY

The experimental setup reported in this paper involves the construction of an electrical resistivity box using a transparent perspex material with the necessary accessories to measure the electrical resistivities of the mine tailings in the laboratory. Details of the materials, test procedures and the analysis of the test results are presented below. Further details about the methodology can be found in our earlier work (Kuranchie et al., 2014).

6.3.1 Materials

Iron ore mine tailings were obtained from the Extension Hill Operations in Perenjori, Western Australia, through the cooperation of Mount Gibson Iron Ore Limited. The physical properties of the iron ore mine tailings were determined following the relevant standards in AS:1289 (2000), and they are given in Table 6.1. Particle-size analysis was performed on the materials using mechanical sieving, and the results are presented in Figure 6.1. Similar properties of the local Perth sand are also included in Table 6.1 and Figure 6.1 for the purpose of a comparison.

Table 6.1: Physical properties of the iron ore mine tailings and local Perth sand.

Property	Iron ore mine tailings	Perth sand (Kuranchie et al., 2014).
Fines (%)	6.5	3.85
Sand and silt-size fraction (%)	70.5	96.15
Gravel-size fraction (%)	23.0	-
Specific gravity	2.65	2.66
Minimum dry density, ρ_{dmin} (kg/m ³)	1685	1392
Maximum dry density, ρ_{dmax} (kg/m ³)	1860	1598
Effective grain size, D_{10} (mm)	0.13	0.18
D_{60} (mm)	2.67	0.38
D_{30} (mm)	0.70	0.26
Coefficient of uniformity, C_U	20.54	2.11
Coefficient of curvature, C_C	1.41	0.99
Soil classification as per USCS	Well graded sand with silt (SW-SM)	Poorly graded sand (SP)

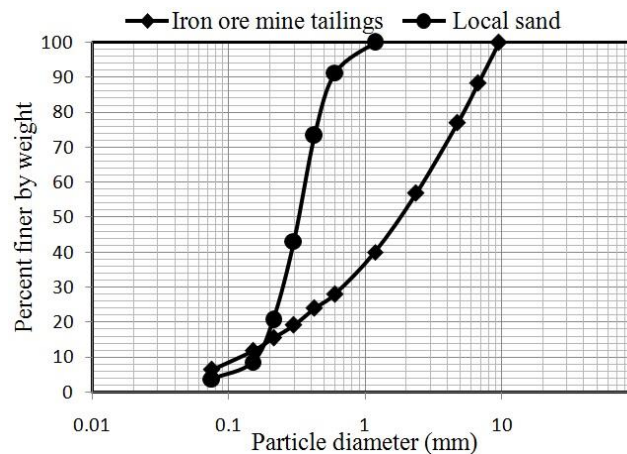


Figure 6.1: Particle-size distribution for the iron ore mine tailings and the Perth sand.

The chemical composition of the iron ore mine tailings was determined using the x-ray fluorescence (XRF) with AXIOS, and wavelength-dispersive XRF sequential spectrometer, and the results are given in Table 6.2. For a comparison, the results of the chemical compositions of some other iron ore mine tailings reported in the literature are also presented in Table 6.2.

Table 6.2: Chemical composition of the iron ore mine tailings from the current study and from others reported in the literature.

Constituents	Iron ore mine tailings from the current study* (%)	Iron ore mine tailings reported by (Chen et al., 2011) (%)	Hematite tailings reported by (Roy et al., 2007) (%)
CaO	0.03	0.12	6.20
SiO ₂	57.31	39.40	24.40
Al ₂ O ₃	9.58	1.36	10.95
Fe ₂ O ₃	25.13	55.61	-
MgO	0.08	-	0.99
SO ₃	0.16	-	-
Na ₂ O	0.04	-	0.28
K ₂ O	0.04	-	0.86
TiO ₂	0.61	-	0.42
Loss on ignition	6.67	3.42	6.95

*Courtesy: MinAnalytical Laboratory Services Australia Pty Ltd, Perth.

6.3.2 Resistivity apparatus

A box with internal dimensions of 600 mm length, 300 mm width and 450 mm height was constructed using a non-conducting transparent perspex material with a thickness of 20 mm. Four electrodes made up of hot-galvanised steel rods with a diameter of 6 mm and 50 mm length were arranged in the box containing the material to be tested in a Wenner array. Two multimeters, one measuring the current induced through the outer electrodes and the other measuring the potential difference across the inner electrodes were incorporated in the resistivity apparatus. The apparatus was completed by joining it with a switch to form a complete electrical circuit. The connections for the electrical circuit were made using 16 gauge braided speaker wire and alligator clips fixed at the lower end of the wires. Equation 6.1 which depicts the Wenner array expression was used for the calculation of the apparent resistivity using the measured current and the voltage values.

$$\rho = 2\pi a \left(\frac{\Delta V}{I} \right) \quad (6.1)$$

where a is the equal electrode spacing in m, ρ is the apparent electrical resistivity in Ωm , ΔV is the electrical potential difference in volts (V) between the two inner electrodes, and I is the induced electric current in amperes (A) in the material between the outer electrodes.

The supplied current source in the experiment was made up of a fully charged 12 V battery with an in-built power inverter capable of producing 230 V alternating current of 50 Hz frequency and 400 W power. The alternating current is recommended because it is able to eliminate the possible problem of polarisation (Yan et al., 2012). All the units were arranged on a perforated wooden material. The experimental setup is shown in Figure 6.2.

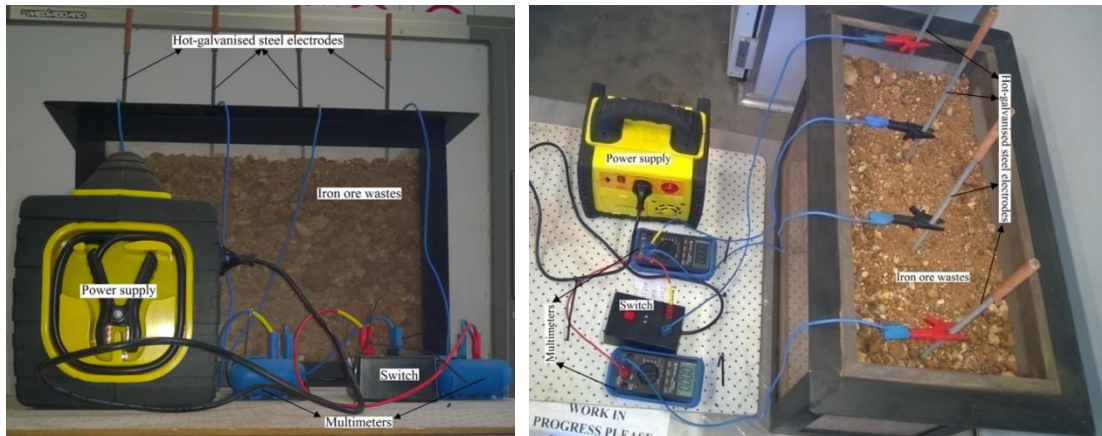


Figure 6.2: Resistivity apparatus: (a) front view, and (b) top view.

6.3.3 Electrical resistivity measurement procedure

A bed of the tailing material being tested was created in the resistivity box with different relative densities, from very loose to very dense conditions. The dry sand had already been tested with the same resistivity apparatus created in the laboratory (Kuranchie et al., 2014). The same procedure was used for the iron ore mine tailings both in dry and fully saturated conditions.

Five different conditions of the relative density (D_r) were determined for the iron ore mine tailings as $D_r = 0\%$ (1685 kg/m^3), $D_r = 25\%$ (1726 kg/m^3), $D_r = 50\%$ (1768 kg/m^3), $D_r = 75\%$ (1813 kg/m^3) and $D_r = 100\%$ (1860 kg/m^3). The densities given in the brackets were calculated based on the following equation using the maximum and minimum density values:

$$D_r = \left(\frac{\rho_{d \max}}{\rho_d} \right) \left(\frac{\rho_d - \rho_{d \min}}{\rho_{d \max} - \rho_{d \min}} \right) \times 100\% \quad (6.2)$$

where $\rho_{d \max}$ is the maximum density, $\rho_{d \min}$ is the minimum density and ρ_d is the particular density that corresponds to the relative density of interest for the experiment. With the known density and the volume of the resistivity box, the dry mass of the material needed to fill the box was determined. The dry mass of the material was placed in the box by a free fall for $D_r = 0\%$ and by appropriate compaction in four lifts for other relative densities to ensure homogeneity in the material beds.

For the fully saturated condition of the iron ore mine tailings, four small plastic pipes were placed at the four sides of the box with a glue. The box was then filled with the dry materials based on the criteria explained earlier. Afterwards, the distilled water was introduced to the bed through the plastic pipes. The distilled water was used to ensure that the geochemical composition of the materials was not altered. The wetting started from the bottom to the top until a fully saturated bed was obtained.

The lower ends of the electrodes were wetted with distilled water and slowly pushed into the material bed to avoid any significant disturbance and to provide good contact between the material and the electrodes. The electrode spacing and depth were maintained constant throughout the experiment at 180 mm spacing and 150 mm depth.

6.3.4 Correction to experimental values

The validity of the Wenner Array expression in equation 6.1 is based on the following three assumptions (Telford et al., 1990).

1. The four electrodes are located at the material interface as point sources,
2. The electrodes are buried in a homogeneous, isotropic and semi-infinite medium, and
3. The air above the medium has zero electrical conductivity.

Normally it is practically difficult to achieve all these three requirements in the laboratory environment. Therefore, a correction factor is required to correct the apparent resistivity values calculated from equation 6.1 in order to obtain realistic values of the apparent resistivity.

In our earlier work (Kuranchie et al., 2014), the COMSOL software was used to simulate the experimental values to obtain an appropriate correction factor. This correction factor was introduced as a multiplier in equation 6.1 and a new equation 6.3 was proposed, as given below, which contains the resistivity correction factor λ .

$$\rho = 2\pi a\lambda\left(\frac{\Delta V}{I}\right) \quad (6.3)$$

For the specific electrode geometrical configuration and the resistivity box used in our earlier work, the resistivity correction factor λ was evaluated as 0.46 (Kuranchie et al., 2014). Since the same resistivity apparatus and electrode geometrical configuration were used in the current work, this value of the resistivity correction factor was applied to all apparent resistivity values to provide correction and obtain realistic values for the apparent resistivities obtained from equation 6.1.

6.4 RESULTS, DISCUSSION AND POSSIBLE FIELD APPLICATIONS

Figure 6.3 compares the variation of apparent electrical resistivities of dry iron ore mine tailings and dry sand with their relative densities. It is observed that the electrical resistivity decreases as the relative density increases. In case of mine tailings, the rate of decrease is almost linear whereas the resistivity of sand decreases nonlinearly. It is also noted that the electrical resistivity ranges from 11 k Ω m for very dense (density = 1860 kg/m³, D_r = 100%) to 19 k Ω m for very loose (density = 1685 kg/m³, D_r = 0%) dry iron ore mine tailings. The decrease of resistivity for a relative density increase from 0% to 25% is about 21% while it is about 8% for a relative density increase from 50% to 75%. Practically the resistivity of tailings does not vary significantly when it is compacted at a relative density greater than 75%. The decrease in resistivity values with an increase in relative density may be attributed to the fact that at a higher relative density, the degree of compaction/densification of the iron ore mine tailings is high which means that there are more solid particles available in a given volume. The increase in solid particles bring about more intimate contact between each other, resulting in more ionic content per unit volume (Kuranchie et al., 2014); which brings about higher influence of surface conductance (Seladji et al., 2010). This allows more current to spread quickly and pass freely in the medium, resulting in a decreased resistivity as it is evident from equations 6.1 and 6.3 and in Figure 6.3.

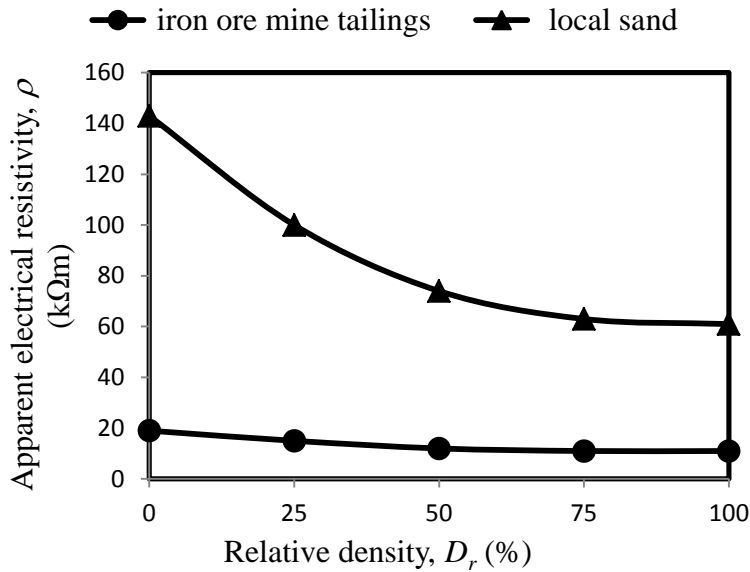


Figure 6.3: Variation of apparent electrical resistivity of iron ore mine tailings and Perth sand in dry condition.

It is noted in Figure 6.3 that the electrical resistivity of dry sand ranges from 61 kΩm for a very dense condition to 143 kΩm for a very loose condition (Kuranchie et al., 2014). Compared to these values, the resistivity values of tailings are much lower, which can be explained in relation to the difference in their chemical compositions and particle sizes (Samouelian et al., 2005). The iron ore tailings used in the current study had higher Fe_2O_3 content of 25.13% as seen in Table 6.2 as compared with the iron content reported for sand as (0.59 – 6.58%) (Yamada et al., 2012) and (1.15 – 3.98%) (Hasdemir et al., 2012). There is a possibility that the higher Fe_2O_3 content in the iron ore mine tailings could be a contributory factor for the lower resistivity than that for the sand. This is because the high iron content in the iron ore tailings will conduct more electric current, thus making the resistivity lower. Also, the iron ore tailings used in this study contain 6.50% fines as compared to only 3.85% in the sand (Table 6.1). The higher content of fines in the iron ore mine tailings will lower its resistivity than in sand with lower quantity of fines (Samouelian et al., 2005). These trends of variation are consistent with some resistivity studies on different types of earth materials reported in the literature (Seladji et al., 2010; Dijkstra et al., 2012; Kuranchie et al., 2014).

Figure 6.4 shows the variation of apparent electrical resistivity of iron ore mine tailings and sand with their relative densities in their saturated conditions. It is noticed that the apparent resistivity of fully saturated iron ore mine tailings ranges from 20 Ωm for a very dense state to 31 Ωm for a very loose state while for the fully saturated sand, it ranges from 82 Ωm for a very dense state to 125 Ωm for a very loose state. When compared with the dry case

(Figure 6.3), it can be noticed that the presence of water reduces the electrical resistivity significantly. It is also observed that the resistivity of iron ore mine tailings is lower than that of the sand for saturated case. It is possible that the higher iron content in the iron ore mine tailings, compared to sand, will bring about more conduction in the iron ore tailings and hence a decrease in the resistivity. It is interesting to note that the percentage difference in resistivities of mine tailings and sand are almost equal for dry and saturated conditions when they are in very loose and very dense conditions.

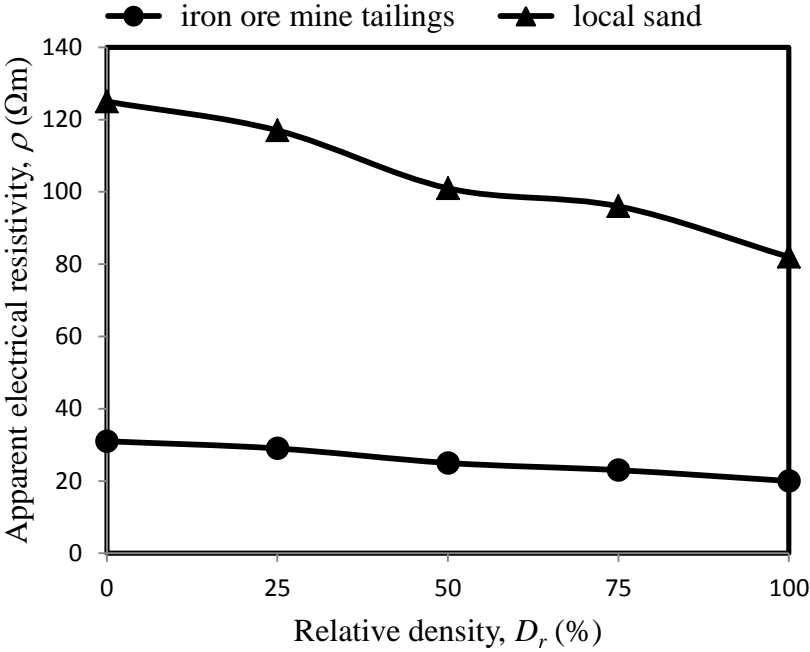


Figure 6.4: Variation of apparent electrical resistivity of iron ore mine tailings and Perth sand in fully saturated condition.

The resistivity of Fe_2O_3 has been reported to range from 3.5×10^{-3} to $10^7 \Omega m$ (Telford et al., 1990). This means that the resistivity values obtained for the iron ore mine tailings in this study are within the expected range and are therefore satisfactory. It should also be noted that some of the constituents of the iron ore mine tailings used in this study have very close values to other similar mine tailings as seen in Table 6.2. This is an indication that other similar iron ore mine tailings may have similar or closer resistivity values observed in this study.

For iron ore mine tailings for which the resistivity is unknown, Figures 6.3 and 6.4 can serve as the charts to determine the apparent resistivity at any relative density. The resistivity values established in this work for both dry and fully saturated cases of iron ore mine tailings

can be very beneficial in many applications such as use as a substitute for sand in structural fills constructed in railway and highway engineering. The use of resistivity values can also be used for the quality control during rehabilitation of mined out areas. In such applications, the measurement of apparent electrical resistivity can predict the respective relative densities, that is, the degree of compaction (McCarter, 1984).

Another example of application of the present study can be the location of steel pipes buried in the ground for different engineering projects using iron ore mine tailings as structural fills. In this application, the extent of corrosion of the steel pipes at different intervals of time can be predicted conveniently by measuring the resistivity values as demonstrated in this study (Sreedeeep, 2004). At lower resistivity, the extent of corrosion is generally high. This trend can be linked with the resistivity values observed in this work which can predict the degree of corrosion for any possible modifications to be done in such projects.

Finally, when the iron ore mine tailings are considered for the construction of embankments as reported earlier (Kuranchie et al., 2013), the weak zones of the embankment, if any, due to poor construction or internal erosion can be studied easily and non-destructively making use of the resistivity values of the iron ore mine tailings at various degrees of compaction. If there is any weak zone in the core of the embankment material as a result of piping or internal erosion, such zones will have a higher resistivity (Oh & Sun, 2008). The resistivity variation within an embankment can be compared with the values reported here, and thus this study can help to address any safety concern in the embankment.

6.5 CONCLUSIONS

This paper presents the details of an experimental study to determine the electrical resistivity of iron ore mine tailings produced in Western Australia. The work follows a methodology that has already been verified. The Wenner array of electrodes arrangement was used to determine the electrical resistivity at several relative densities. Based on the study presented the following general conclusions can be made:

1. The electrical resistivity of the iron ore mine tailings, both in dry and fully saturated conditions, depends on the relative density or the extent of compaction. The resistivity is in inverse relationship with the relative density. The resistivity decreases with an increase in relative density and vice versa.
2. The electrical resistivity of iron ore mine tailings produced in Western Australia in dry condition is found to range from 11 k Ω m in a more dense state to 19 k Ω m in a very

loose state while that in fully saturated condition it ranges from 20 Ωm for a very dense state to 31 Ωm in a very loose state.

3. The resistivity of iron ore mine tailings both in dry and fully saturated conditions has lower values compared with resistivity values for sand. This occurs mainly because the iron ore mine tailings have very high iron content compared to that of sand.
4. Figures 6.3 and 6.4 can be used as the engineering charts for determining the electrical resistivity of iron ore mine tailings by knowing its relative density. This may guide the field engineers to economically identify the behavior of the iron ore mine tailings when used in engineering projects such as the following:
 - Structural fills in railway or highway engineering projects, and in rehabilitation of mined out areas
 - Determining the extent of corrosion for buried steel pipe lines for modification and replacement
 - Studies on weak zones in embankments

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CHAPTER 7

UTILISATION OF IRON ORE MINE TAILINGS FOR THE PRODUCTION OF GEOPOLYMER BRICKS

This chapter has been published as a Journal article in the International Journal of Mining, Reclamation and Environment by Taylor and Francis Group, United Kingdom as mentioned in number 3 of Section 1.7. The details presented here are the same, except some changes in the layout in order to maintain a consistency in the presentation throughout the thesis.

7.1 ABSTRACT

This paper presents a methodology for making bricks, in a cost-effective and environmentally friendly manner, using the tailings produced from iron ore mines in Western Australia (WA). The study was based on the geopolymerization process, which is known to conserve energy by reducing the emission of greenhouse gases. The reduction is accomplished by avoiding the processes of high temperature kiln firing, traditionally utilised when making bricks from sandy soils with high clay content. In the present study, the sodium silicate was added to the mine tailings in powder form, as an activator for the formulation of the geopolymer bricks. The effects of the initial setting time, curing temperature, curing time, and activator content on the unconfined compressive strength (UCS), water absorption, and other durability properties of the bricks were investigated. X-ray diffraction analysis (XRD) was performed to investigate the phase composition of the geopolymer bricks. The bricks achieved an UCS as high as 50.53 MPa for the optimum values of the parameters. Technically the geopolymer bricks that were produced met both the American Society of Testing and Materials (ASTM) and the Australian Standards (AS) requirements for bricks. A cost analysis of the geopolymer bricks is also presented, and this shows that the cost of geopolymer bricks is lower than that of the commercial, fired clay bricks.

7.2 INTRODUCTION

Iron ore mining is one of the most significant players in the resource sector in Western Australia (WA), and it accounts for about 47% of the total value of all resources in the state (WA Department of Mines and Petroleum, 2009). During 2008-2009, there was a total

production of 316 million tonnes of iron ore (WA Department of Mines and Petroleum, 2009). Although, there is no proper reporting mechanism in place which can help to document the quantity of iron ore mine tailings produced in the state; approximations based on the information made available by some iron ore companies, show that for every tonne of iron ore produced, two tonnes of iron ore tailings are removed (Price, 2004). Based on current production levels, it can be estimated that about 632 million tonnes of iron ore mine tailings are produced yearly in WA (Kuranchie et al., 2014).

The continuous discharge and the under-utilization of iron ore mine tailings will worsen the environmental problems for the state in the future. These tailings cause oxidation of sulphide minerals, and bring about acid mine drainage (AMD), both of which harm the environment (Mudd, 2004). Other negative environmental impacts arising from tailings accumulation are: removal of vegetation cover, deforestation, land slope changes, increased risk of erosion, water pollution, and contamination of agricultural goods and risk of human health (Castro-Gomes et al., 2012).

In recent years, the utilization of mine tailings has been getting much attention globally; especially for use in civil engineering construction (Roy et al., 2007; Aigbodion et al., 2010). This practice can help reduce the emission of greenhouse gases; by avoiding the mining of virgin engineering materials, providing cheaper alternative materials for building and construction, and for natural resource conservation (Chen et al., 2011).

The conventional method for manufacturing bricks is to produce a sand-clay mixture, which is then fired in a high temperature kiln (Ahmari & Zhang, 2012). This method is not considered to be environmentally friendly. China has taken the initiative, by setting a policy requiring all bricks to be produced using solid waste materials, instead of normal soils (Chen et al., 2011). As the world is becoming environmentally conscious, a variety of other waste materials have been used elsewhere for brick production. Roy et al. (2007) studied the feasibility of using gold mill tailings from the Kolar Gold Fields, Karnataka, India for brick production. The gold tailings were mixed with both ordinary Portland cement, and other soils in specified proportions, depending upon the situational requirements. The mixtures were then moulded into the required shape, and then fired at high temperatures (750 °C - 950 °C). The research by Roy et al. (2007) reported that the cement-tailings bricks containing 20% of cement, and having 14 days of curing, met the required compressive strength, but the cost was 2.4 times the cost of traditional clay bricks. The soil-tailings bricks also met the requirements for compressive strength, with the cost being 0.72 times the cost of traditional clay bricks. Chen et al. (2011) investigated the possibility of making construction bricks, by using

hematite tailings from the province of Western Hubei, China. The hematite tailings were mixed with clay and class F fly ash, in various proportions. The bricks had a water absorption of 16.54 – 17.93 %, and mechanical strength of 20 - 25 MPa; and these properties conformed to the Chinese fired common bricks standard (GB / 75101-2003). Aigbodion et al. (2010) investigated the feasibility of using tin tailings from Jos Tin Mining in Nigeria, for the production of refractory bricks to be used as a furnace liner. The tin tailings were mixed with Arabic gum, cotton wool, and water. The mixture was moulded and fired at temperatures ranging from 200 °C to 1600 °C. Their study concluded that the refractory bricks were comparable to those required by the Indian fire clay refractory standards, for use as furnace lining. Anyakora (2013) used the waste sludge from the lower Usuma dam water treatment plant in Abuja, Nigeria to produce bricks. The sludge was mixed with clay in various proportions, after which the conventional brick making process was followed. They reported that their bricks conformed to both Nigerian standard specifications (NIS 74:1976), and the British standard specifications (BS 3921:1985), enabling the bricks to be used for various purposes in construction and building. Ingunza et al. (2011) made soft-mud bricks using a mixture of clay and sewage sludge from a septic tank cleaning company in Brazil. Their study concluded that bricks could be made by using 20% sludge and 80% clay mixture. (Lin & Weng, 2001) used a mixture of clay and incinerated sewage sludge ash, from southern Taiwan, to make bricks. They found that a range of 20 to 40% incinerated sewage sludge ash could be mixed with the clay to successfully produce bricks, if the bricks are fired at temperature of 1000 °C for 6 hours.

It is apparent that the various methods for utilizing waste materials for brick production elaborated above require either the inclusion of some clay or cement content, or else high temperature kiln firing. These processes bring some improvement over conventional bricks manufacture in terms of ecological benefit. However, they still have some negative effects on the environment; because of both the high energy consumption, and the emission of greenhouse gases arising from the high temperature kiln firing.

Currently researchers are exploring alternative ways to produce bricks in a more environmentally friendly fashion over the existing technologies. Geopolymerization technology is an alternative method which can be used for bricks production; and this is catching the interest of researchers, because of the significant environmental and ecological benefits associated with the process. The geopolymerization process can be defined as the reaction undergone between solid alumino-silicate minerals, with highly concentrated

aqueous alkali hydroxide or silicate solution. This technology was first introduced by Davidovits (Duxson et al., 2007).

According to (Duxson et al., 2007), the geopolymerization process is comprised of: 1) dissolving solid aluminosilicates in a highly concentrated alkali or silicate solution; 2) forming a silica-alumina oligomer as a gel followed by polycondensation of the oligomer; 3) forming a stable inorganic material; 4) reorganization; and 5) forming a strong bond with any undissolved solid materials in the polymeric structure. (Mohsen & Mostafa, 2010) reported that the main binding phase of the geopolymer is called amorphous alumino-silicate gel; which is made up of a three dimensional framework of alumina and silica, that is linked at the corners with oxygen atoms. Therefore, this can provide the binding characteristics required for brick making, to the clay. The geopolymerization process is presented in Figure 7.1.

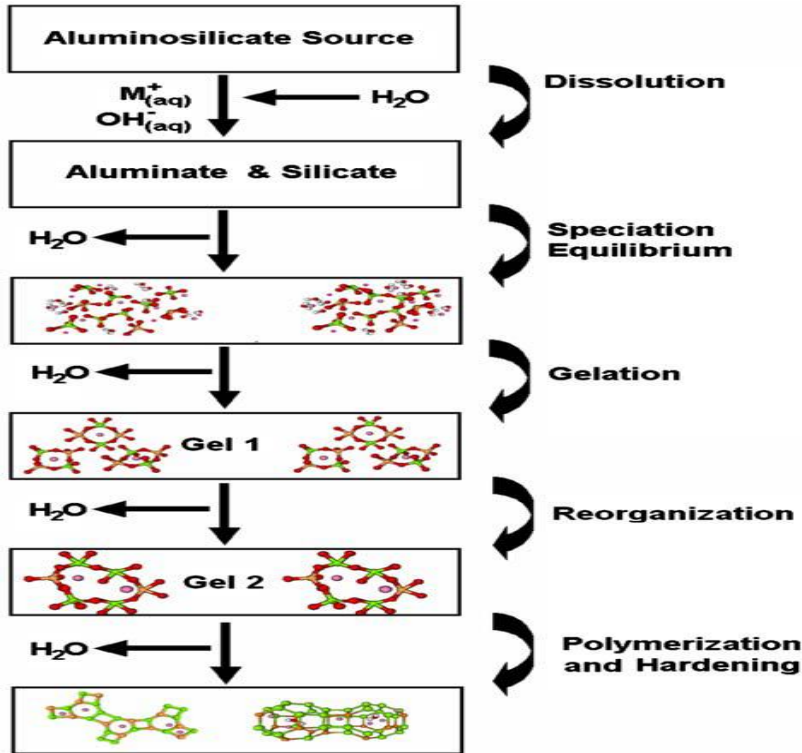


Figure 7.1: The geopolymerization process (Duxson et al., 2007).

This method shows additional advantages, such as; the development of higher mechanical strength and hardness, higher acid and fire resistance, higher adherence to aggregates and high bond strength, lower shrinkage, lower thermal conductivity, high surface characteristics, increased durability, reducing the movement of toxic and hazardous materials

such as heavy metals (lead, copper), and reduction of greenhouse gas emissions (Drechsler & Graham, 2005; Diop & Grutzeck, 2008; Ferone et al., 2011; Ahmari & Zhang, 2012). This technology has been used successfully by other researchers for bricks production utilising a variety of waste and other materials (Diop & Grutzeck, 2008; Komljenovic et al., 2010; Mohsen & Mostafa, 2010; Ahmari & Zhang, 2012).

Considering the large quantity of iron ore tailings in WA, and the tailings containing a substantial amount of silica and alumina; the state is in a favourable location for the application of large quantities of iron ore mine tailings for brick production.

To our knowledge, no previous practical studies have been conducted using the geopolymerization technology on WA iron ore mine tailings for brick production. In addition, the available papers detailed above, present the use of either a high concentration sodium hydroxide solution, or a mixture of high concentration sodium hydroxide and sodium silicate solutions as activators. The use of a sodium silicate solution alone, which could reduce the cost of alkali activator in the production of the geopolymer bricks, is missing from the literature.

This study attempts to focus on this direction, and thereby to fill this gap in the literature. The work will study the feasibility of using the iron ore mine tailings, which are produced in WA in large quantities, and sodium silicate solution for the production of environmentally friendly and value added geopolymer bricks. It is expected that mechanical properties similar to those of conventional clay bricks will be achieved. In the following sections, the suitable process conditions, mechanical strength, durability properties, and the raw materials for the production of the bricks are discussed.

7.3 EXPERIMENTAL PROGRAMME

The experimental study was done in the geotechnical engineering laboratory at the school of engineering, Edith Cowan University. It consisted of the following steps: 1) collecting the iron ore mine tailings; 2) grinding and sieving; 3) mixing of graded tailings with sodium silicate solution (activator); 4) making the bricks in a mould; and 5) curing the moulded bricks. The finished bricks were subjected to a range of quality control assessment tests based on, unconfined compressive strength (UCS), water absorption, resistance to salt attack, potential to effloresce, and the electrical resistivity of the bricks. These properties were compared with the requirements recommended by the relevant Australian Standards and the American Society for Testing and Materials (ASTM). X-ray diffraction (XRD) analysis was

carried out to study the difference in the phase composition of the original mine tailings and the finished bricks.

7.3.1 Materials

The materials used in this study include iron ore mine tailings and sodium silicate solution (Na_2SiO_3). The mine tailings were received in sizes ranging from fines ($< 75\mu\text{m}$), to gravels ($\leq 32\text{ mm}$), supplied by the Mount Gibson Iron, from their Extension Hill operations in Perenjori of Western Australia (WA). The particle size distribution of the mine tailings as supplied is shown in Figure 7.2. It contains about 6.50% fines, which was used after limited grinding.

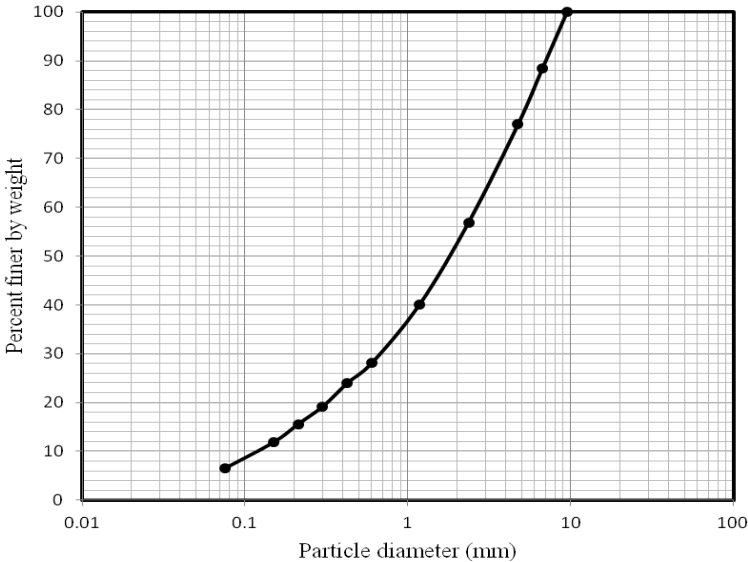


Figure 7.2: Particle-size distribution curve of the iron ore mine tailings used for brick manufacturing.

Other physical properties of the mine tailings were determined as per the relevant standards listed in AS 1289 (2000). Table 7.1 shows the various physical properties of the iron ore mine tailings. The tailings are made up of 70.5% sandy and silty-sized particles as seen in Table 7.1.

Table 7.1: Physical properties of the iron ore mine tailings.

Property	Value
Fines (clay and silt) (%)	6.5
Sand content (%)	69.5
Gravel content (%)	24.0
Specific gravity	2.65
Minimum dry density, ρ_{dmin} (kg/m ³)	1685.00
Maximum dry density, ρ_{dmax} (kg/m ³)	1860.00
Effective grain size, D_{10} (mm)	0.13
D_{60} (mm)	2.67
D_{30} (mm)	0.70
Coefficient of uniformity, C_U	20.54
Coefficient of Curvature, C_C	1.41
Soil classification as per USCS	SW-SM (Well graded sand-silty sand)

A total of 60 litres of sodium silicate (Na_2SiO_3) solution as an activator was received from Coogee Chemicals Limited and Metallurgy Property Limited, Western Australia. This solution consisted of 8.9% of Na_2O , 28.7% of SiO_2 , and 62.4% of H_2O by weight. The SiO_2 to Na_2O molar ratio was 3.58. The concentration of the sodium silicate in the solution was 40%. It should be noted that the composition and the technical specifications for the activator, were selected and designed based on recommendations in the literature (Silva & Sagoe-Crentsil, 2009; Villa et al., 2010; Heah et al., 2012). Consequently it was initially expected, that the geopolymer bricks produced in this study will have a high mechanical strength.

The analysis of the chemical composition of the iron ore mine tailings, as given in Table 7.2, were carried out using x-ray fluorescence (XRF) with an AXIOS wavelength-dispersive XRF sequential spectrometer. Table 7.2 also presents the chemical composition of some other mine tailings for the sake of comparison. The mine tailings used here have higher silica (SiO_2) content than the others reported earlier. The tailings also have a substantial amount of alumina (Al_2O_3), which together with the silica, makes them favourable for the geopolymerization reaction, regarding enhanced dissolution and mechanical strength. The tailings are made up of about 25% of iron oxide, which contributed to the light colour observed in the tailings. This is likely to favour the usual colour normally seen in traditional clay bricks.

Table 7.2: Chemical composition of the iron ore mine tailings.

Constituents	Iron ore mine waste from the current study (%)*	Iron ore mine waste (%) (Das et al., 2000)	Hematite waste (%) (Chen et al., 2011)
CaO	0.03	0.12	6.20
SiO ₂	57.31	39.40	24.40
Al ₂ O ₃	9.58	1.36	10.95
Fe ₂ O ₃	25.13	55.61	---
MgO	0.08	---	0.99
SO ₃	0.16	---	---
Na ₂ O	0.04	---	0.28
K ₂ O	0.04	---	0.86
TiO ₂	0.61	---	0.42
As	---	---	---
Loss on ignition	6.67	3.42	6.95

* Courtesy of MinAnalytical Laboratory Services Australia Pty Ltd.

7.3.2 Grinding and sieving of tailings

The mine tailings were first screened through a sieve of Australian mesh size of 212 μm . Other researchers have used sieves of various mesh sizes, such as 300 μm (Roy et al., 2007), 1.70 mm (Bektas et al., 2007), 250 μm (Diop & Grutzeck, 2008) and 120 μm (Mohsen & Mostafa, 2010). The geopolymerization process requires some level of fineness, so that there can be a complete dissolution of the alumino-silicate materials in the tailings. It is also reported that the brick making material should contain at least 30% of sandy and silty sized particles combined together (Somayayi, 2001). As a result, the mine tailings were screened below 212 μm , which provided the optimum fines for ensuring a complete dissolution of the alumino-silicate materials, and the polycondensation process.

The oversized material captured by the 212 μm sieve, was milled using a 12 litre capacity laboratory steel ball mill. In order to determine the optimum milling time, that would ensure that 90% or more of the tailings pass through a 212 μm sieve; individual 5 kg samples of the tailings were each milled separately, for selected durations of 15, 30, 45, 60, and 75 minutes. The purpose was to determine a grinding time, ensuring that at least 90% of the ground material passed through a 212 μm mesh sieve. Once this optimum grinding time was determined, the rest of the material would be milled for that specific duration.

Figure 7.3 shows the milling time against the percentage of material passing through the 212 μm mesh sieve. In general, the percentage passing the 212 μm sieve size increased when the milling time was increased. By milling the samples for 60 minutes, 72% passed, and milling it for 75 minutes, as high as 92% passed through the sieve used. Therefore, a milling

time of 75 minutes was selected as the optimum, and formed the basis for milling the rest of the samples throughout the work. Each ball mill discharge was screened through the 212 μm sieve and the undersize materials were used. The process was repeated several times, until enough materials were obtained for the study, such that 100% of the mine tailings used for the formulation of the bricks passed the 212 μm sieve size.

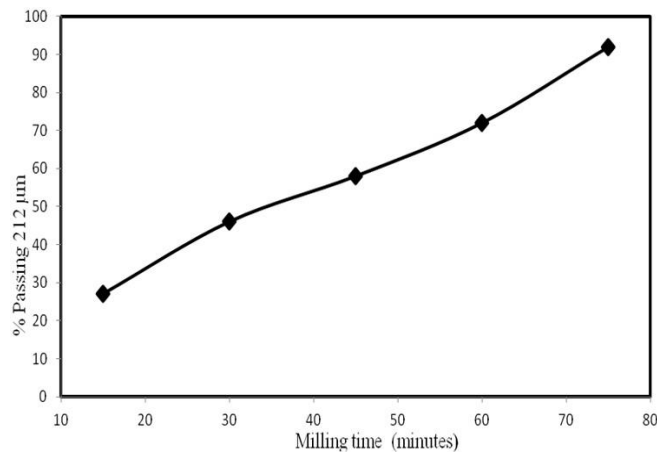


Figure 7.3: Milling time versus percent passing 212 μm .

7.3.3 Mixture preparation and brick making

A mould having a size of 11.5 cm length, 5.5 cm width, and 3.8 cm height was used to make the bricks in the laboratory. The volume of this mould is about one-eighth of the volume of the Australian standard mould for making bricks, because each dimension of the laboratory mould is set to half of the respective dimension of the Australian standard mould. A lower size mould was developed in the study, to reduce the quantity of materials to be handled in the laboratory, and also to reduce some practical difficulties.

For the sodium silicate content (activator) in the mixture, (Diop & Grutzeck, 2008) reported that a high mechanical strength of the tuff geopolymer bricks was obtained by using 20% of 12 M sodium hydroxide solution (activator). A high activator content generally enhances the dissolution process, and increases the strength of the geopolymer brick, depending on the particular type of materials used. From our preliminary studies on the iron ore mine tailings, we found that 20 to 30% of sodium silicate solution content could not provide the desired tailings mixture consistency needed to make the stable bricks conveniently. This could be because of the nature of the mine tailings. Therefore, the activator content was increased slightly, reducing the margin to 31%, and this provided a suitable

consistency for making the bricks. Increasing the activator content also resulted in a higher UCS than the value reported by (Diop & Grutzeck, 2008). Therefore, 31% by weight of the sodium silicate solution was initially used for this work, while the investigation into the effect of the other parameters was carried out.

The sodium silicate solution was added to the mine tailings as described above, and the mixture was stirred using a laboratory concrete mixer for about 10 minutes, until a homogeneous mixture was obtained.

Several researchers have reported that for a high mechanical strength, the density of the final fired clay bricks should range from 1.8 g/cm³ to 2.0 g/cm³ (Lin & Weng, 2001; Somayayi, 2001; Chen et al., 2011). In this work, an initial density of 2.1 g/cm³ was used for estimating the dry weight of tailings for one brick, in anticipation that after curing, the density of the resulting brick will reduce down to a value which is in the recommended range.

The mine tailings were compacted in the mould in a thick paste form. The volume of material required to fill the mould was determined to be 240.35 cm³ (11.5 cm × 5.5 cm × 3.8 cm = 240.35 cm³). These values are derived from the internal dimensions of the mould which was used to make the bricks, (11.5 cm length, 5.5 cm wide and 3.8 cm high). With the known volume and using the selected density mentioned above; the mass of the thick paste mixture (mine waste and activator solution) was determined by multiplying the density and the volume (2.1 × 240.35 = 504.7 g), to obtain the mass of the paste needed to form the brick. Mould oil was initially applied on the internal surfaces of the mould to reduce friction and allow easy release from the mould after casting. The brick was moulded from a 504.7 g mass of the paste, by a manual press, with the help of a tamper. Afterwards each brick was removed from its mould, and then allowed to sit for 24 hours at room temperature (22 °C – 24 °C). This process according to (Diop & Grutzeck, 2008), is called soaking, and it accelerates the dissolution process with time, which enhances mechanical performance. Figure 7.4 (a) explains the process of making the bricks, through compaction in the mould with the help of a tamper, and Figure 7.4 (b) shows some freshly prepared bricks.

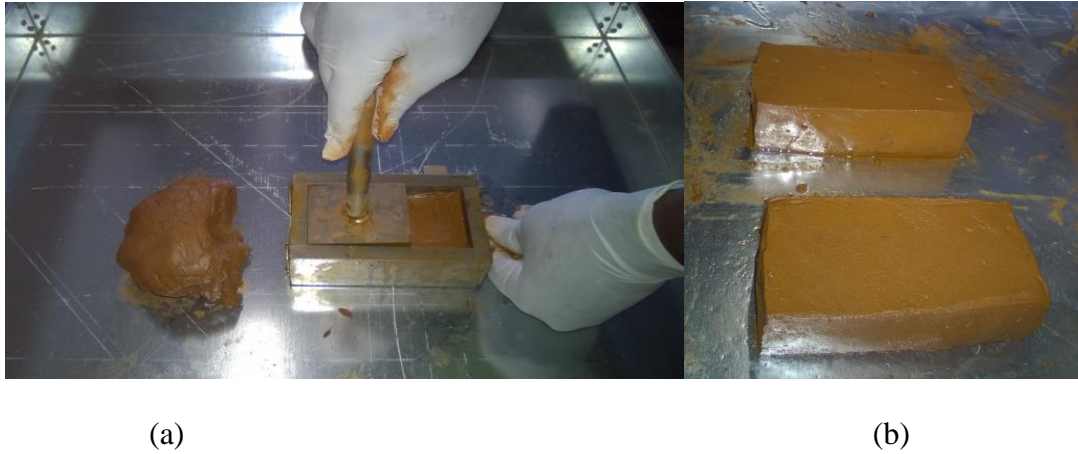


Figure 7.4: (a) Process of making the brick, and (b) freshly prepared bricks

After the 24 hours of soaking, the bricks were placed in the oven, and cured according to the curing schemes explained above.

7.3.4 Parameters investigated

During the geopolymerization process, the alumino-silicate minerals from the tailings material react with each other. This type of reaction is complex, and it is influenced by many factors during formation and curing, such as the following: 1) nature of the tailing material 2) type of activator (sodium hydroxide, potassium hydroxide, sodium silicate solutions or a mixture of these solutions) 3) content/concentration of activator; 4) particle size distribution; 5) initial setting time after the tailing material and the activator are mixed; 6) curing time; 7) curing temperature; etc. Thus, extensive study is needed to identify the optimum parameters for the different types of materials. For our investigation, the four most influential parameters for the geopolymerization process were chosen, which are: 1) effect of initial setting time before moulding; 2) curing temperature; 3) curing time; and 4) content of the sodium silicate solution (activator) in the mixture. These four parameters were selected, because in every geopolymerization reaction, the reaction time and the temperature will always be needed for curing; and also because the process requires thermal activation. Again, the nature of the process is such that the reaction takes place very quickly. As soon as the alumino-silicate minerals come into contact with the activator solution, the reaction takes place, and a hardened paste is obtained. Therefore, because of the nature of the reaction, the initial setting time of the mixture before moulding is worth investigating. Finally, because the reaction is solely between the material and the activator solution, ideally we should know the content, or

how much of the activator solution will be required to mix with the tailing materials, in order to obtain the optimum amount for a higher strength.

To investigate the effect of initial setting time before moulding, the mixture was allowed to sit for 5, 10, 15, 20, and 25 minutes for each cycle. For each of these times, seven bricks were prepared and cured. The curing conditions for this parameter were maintained at 3 days of oven curing, at a curing temperature of 80 °C. These parameters were selected, based on the recommendations in the literature on brick manufacturing with different materials and processes. For instance, according to (Mohsen & Mostafa, 2010) and (Ferone et al., 2011), the curing temperature of geopolymer bricks should be 100 °C or less. Anything above 100°C will reduce the mechanical strength. Again, (Diop & Grutzeck, 2008) cured tuff geopolymer bricks at 80 °C and the bricks came out stronger with good performance. Therefore, a temperature of 80 °C was chosen to conform to these recommendations.

To investigate the effect of curing temperature, 22 °C (average room temperature), 40 °C, 60 °C, 80 °C, and 100 °C were tested; while maintaining the other parameters constant as stated above. The effect of the content of the activator in the mixture was also investigated, by selecting the content of the sodium silicate solution in the mixture to be 25%, 31%, and 37%, based on the findings of the preliminary studies stated above. Other higher values of sodium silicate content were not investigated, because when the sodium silicate content was greater than 31%, the unconfined compressive strength (UCS) started to diminish. Finally, to know the effect of curing time, the bricks were cured for 1, 3, 7, 14, and 28 days; because these are the accepted periods typically employed when testing bricks. The same values (that were identified above), for each of the other parameters were used during the tests for the curing time.

7.3.5 Quality assessment tests

After curing, the resulting bricks were subjected to quality assessment tests based on unconfined compressive strength (UCS); and durability tests, such as water absorption, resistance to salt attack, potential to effloresce, and electrical resistivity. The UCS was tested using a compression testing machine in the civil engineering material testing laboratory, following the Australian Standard AS 4456.4 (2003). Three bricks were tested in each case for the UCS and the average of the UCS values was computed. The bricks were weighed before and after curing to determine the final water content. The water absorption test was

performed according to the Australian Standard AS 4456.14 (2003). The percentage water absorption (w_a) was calculated as:

$$w_a = \frac{w_2 - w_1}{w_1} \times 100\% \quad (7.1)$$

where w_1 is the weight of the brick after curing at the specified temperature mentioned above; and w_2 is the weight of the surface dry brick after immersion in water for 24 hours. The test to determine resistance to salt attack was performed using sodium chloride solution having a concentration of 14%, following Australian Standard AS 4456.10 (2003). The potential of the bricks to effloresce was tested following Australian Standard AS 4456.6 (2003).

The electrical resistivities of the tailings geopolymer bricks, as well as a commercial brick were also determined. In the measuring process, the brick was clamped with two probes attached to two small metal plates, directly facing both ends of the brick, to provide a medium for current flow. The other ends of the two probes were connected to a multimeter, with a capacity up to 40 M Ω , to display the resultant resistance of the brick. The resistivity of the brick was calculated using the resistance value recorded and the dimensions of the brick, using the following the expression:

$$\rho = \frac{RA}{L} \quad (7.2)$$

where ρ is the electrical resistivity (Ωm), R is the electrical resistance (Ω), A is the surface area where the current is flowing through the brick (m^2) and L is the length of the wire across the brick (m). The setup used for the resistivity measurement is shown in Figure 7.5. The steps used for the entire experimental study are summarised in the flow chart shown in Figure 7.6.

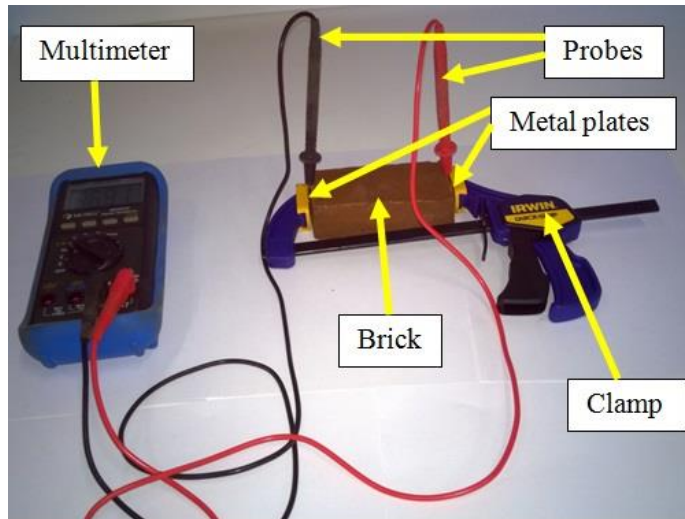


Figure 7.5: Setup for the measurement of electrical resistivity of bricks.

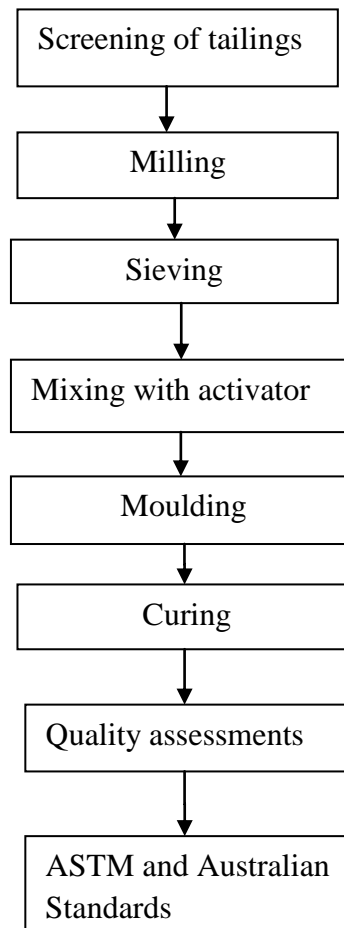


Figure 7.6: The steps used for the study.

7.4 RESULTS AND DISCUSSION

Figure 7.7 shows the variation of UCS with the initial setting time, which is the time interval between when the tailings and the silicate solution are mixed, and the time when the brick is moulded. It is observed that the UCS first increases from 5 minutes up to 15 minutes. The strength starts to decrease when the initial setting time is further increased to 20 and then 25 minutes. This is because at the initial stages between 5 and 15 minutes, there is an ongoing dissolution of the alumina-silicate materials in the mixture, which enhances the strength after it has been moulded and cured. The decrease in UCS after the initial setting time of 15 minutes and beyond is as a result of practical difficulties in the moulding.

When the initial setting time of the mixture becomes longer, the mixture is already partially cured, i.e. solid. It may be noted that as soon as the silicate solution mixes with the alumina-silicate minerals in the tailings, the geopolymerization process starts quickly, and soon a strong bond is formed, which causes the mixture to harden, without allowing sufficient time for the moulding process. Making bricks with the partially cured mixture was problematic, as it became difficult to completely fill the mould. Using the partially cured mixture resulted in voids within the moulded brick, reducing its overall strength. When the mixture was partially cured, it did not achieve full homogeneity; and dissolution of alumina-silicate materials was also slowed down, which consequently affected the strength of the bricks.

Table 7.3 shows the average water absorption and the average densities for the bricks, across a range of values for the initial setting times. Each of the water absorption values for the range of initial setting times was observed to be between 7 and 10%. For the initial setting time of 15 minutes, the brick had the minimum water absorption of 7.18%, and also resulted in the highest strength of 34 MPa. All other initial setting times met the strength and the water absorption requirements for the ASTM standard specifications (Table 7.4a), and Australian requirements (Table 7.4b), for several applications of bricks under varying weathering conditions. The values obtained for the strength of the bricks across the range of initial setting times were even better, and surpassed the minimum requirements of the bricks. The only exceptional case not meeting the ASTM standard was the specification for pedestrian and light traffic paving bricks, experiencing severe weathering conditions. Each of the water absorption values across the range of initial setting times, were either within the limit, or were better for all the various applications of the bricks listed in the ASTM standard specifications. Although, all the initial setting times met the requirements of the standards, the highest strength of 34 MPa was obtained at initial setting time of 15 minutes; which corresponded to

the lowest water absorption of 7.18% (Table 7.3). Therefore, an initial setting time of 15 minutes was seen as the optimum, and this was held constant for the rest of the tests, whilst evaluating the other parameters. In addition, an average density of less than 2.2 g/cm³ was obtained from tests conducted across the range of initial setting times (Table 7.3); and this density is within the acceptable range as per the requirements of the standards. When compared with concrete, which has been reported to have initial setting time of up to 1 hour (Piyasena et al., 2013); the optimum initial setting time of around 15 minutes is much shorter due to the nature of the geopolymerization reaction. The reaction causes the material to harden very quickly, but the setting time still agrees favourably to that of concrete, and is still in the range specified by the literature above.

Figure 7.8 shows how the unconfined compressive strength (UCS) changes with temperature. It is observed that the UCS is limited to 0.7 MPa, and 1.6 MPa, at average room temperature of 22 °C, and curing temperature of 40 °C, respectively. These values did not meet any of the ASTM, or the Australian requirements for bricks (Tables 7.4a and 7.4b). Further increasing the temperature to 60 °C resulted in a substantial increase in the UCS to 12.2 MPa. This goes to confirm that at least some level of thermal curing is required for the geopolymerization process, in order for the bricks to meet the requirements of the standards (Rajiwala et al., 2012). The highest UCS of 34 MPa, with oven curing of three days, was obtained at a temperature of 80 °C. A similar situation was also reported for geopolymer concrete based on fly ash (Rajiwala et al., 2012). However, when the temperature was increased further to 100 °C, the UCS was reduced to 19.6 MPa. This observed trend can be explained by the controlling factors of dissolution and polycondensation, which are significant factors in the geopolymerization process. When the curing temperature of the bricks is increased, the rate of dissolution and polycondensation of silica and alumina materials in the mixture increases; and results in an increase in the UCS of the bricks. However, at temperatures above the optimum, the strength of the geopolymer bricks decreases. The quick polycondensation and rapid development of the polymeric gel, hampers the dissolution of silica and alumina materials; resulting in a loss of strength. Similar trends have also been observed by other authors (Diop & Grutzeck, 2008; Mohsen & Mostafa, 2010).

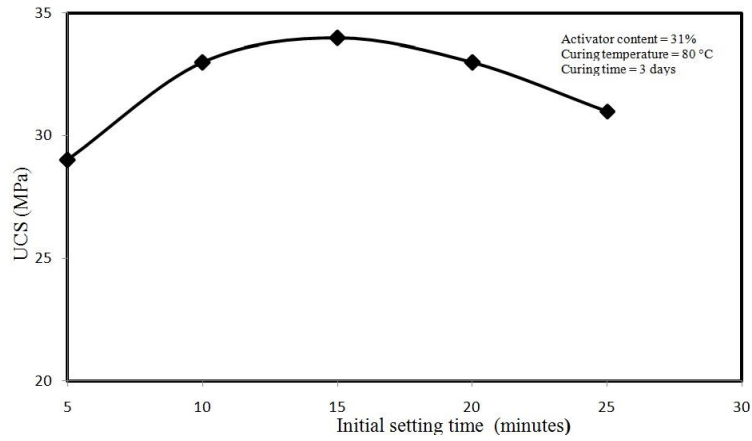


Figure 7.7: Variation of unconfined compressive strength (UCS) with initial setting time.

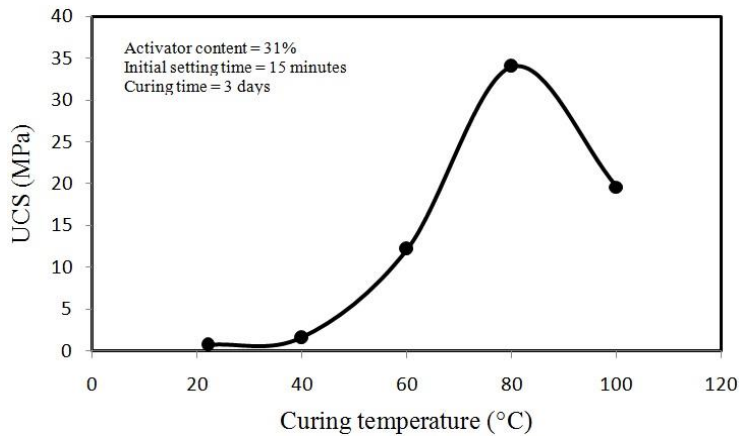


Figure 7.8: Variation of UCS versus curing temperature.

Table 7.3: Water absorption and density for the geopolymer bricks at different initial setting times with curing temperature of 80 °C, activator content of 31% and curing time of 3 days.

Initial setting time (minutes)	Water absorption (%)	Density (g/cm ³)
5	8.43	2.11
10	7.88	2.07
15	7.18	2.09
20	10.21	2.06
25	7.71	2.09

Table 7.4 (a): ASTM specifications for different bricks and load-bearing concrete units (Mohsen & Mostafa, 2010).

Type of specification	ASTM designation	Weathering conditions	Minimum compressive strength (MPa)	Maximum water absorption (%)
Building brick	C 62	SW*	20.7	17
		MW*	17.2	22
		NW*	10.3	No limit
Pedestrian and light traffic paving brick	C902	SW	55.2	8
		MW	20.7	14
		NW	20.7	No limit
Load bearing masonry units	C90	-	13.1	17

*SW indicates severe weathering, MW indicates moderate weathering and NW indicates negligible or no weathering

Table 7.4 (b): Typical UCS for Australian fire clay brick (Think Brick Australia, 2013).

State	Method of Manufacture	Typical UCS (MPa)	UCS range (MPa)
New South Wales	extruded	25	12 - 48
	pressed	16	9 - 34
Queensland	extruded	16	9 – 23
	pressed	13	6 - 23
South Australia	extruded	28	13 – 46
	pressed	19	14 - 24
Tasmania	extruded	23	8 - 48
Victoria	extruded	34	10 – 59
	pressed	26	14 – 41
Western Australia	extruded	18	11 - 33

At a temperature of 80 °C, all the requirements for the various specifications of bricks under both weathering and normal conditions were met. The only exception is the requirement for the brick to be used as a pedestrian and light traffic paving brick for severe weathering conditions; which requires an UCS of 55.2 MPa.

Table 7.5 shows the water absorptions and densities for the various curing temperatures. At an average room temperature of 22 °C and curing temperature of 40 °C, the water absorption could not be determined; as bricks immersed in water for 24 hours could not maintain their shape. At a temperature of between 60 °C to 100 °C, the water absorption values were within the requirements of the standards. The lowest water absorption was 8.41 % and it was obtained at a curing temperature of 80 °C, which coincided with the highest strength from the three days of curing. The densities were slightly higher than the optimum range (1.8-2.0) g/cm³, by few decimal points at temperatures from 25 °C to 60 °C. However, at temperatures of 80 °C and 100 °C, the average densities were 1.99 g/cm³ and 2.03g/cm³ respectively, and these values were within the range.

Figure 7.9 shows how the UCS varies according to the content of the activator solution used. (Diop & Grutzeck, 2008) used 20% of NaOH solution to combine with a tuff for brick making; and achieved a highest UCS value between 20 and 25 MPa. In our case, 25% and 31% of sodium silicate solutions resulted in UCS values of 30.58 MPa and 34.0 MPa, respectively. This means that in general, as the content of the activator solution increases, the UCS is also expected to increase; because dissolution is highest at higher activator content. However, when the content of sodium silicate solution reached 37%, the strength of the bricks reduced down to 24.78 MPa. This is because the optimum amount of the activator solution to be used is very important. When the content of the sodium silicate solution is above what is supposed to be the optimum, it affects the strength negatively. The reduction in strength occurs when the activator content (sodium silicate solution) is above the optimum value. There is a rapid and higher dissolution of the alumino-silicate species, and this causes alumino-silicate gels to crystallise at the early stages, which has negative consequences on the compressive strength of the bricks (Lee & Van Deventer, 2002). In addition, where the sodium silicate solution is substantially higher than the optimum value, the outcome is an increase in the SiO₂/Na₂O ratio. In this case, silicon is present in large quantities, which could result in the formation of several different polymeric structures (oligomers) or simple structures (monomers) (Duxson et al., 2005). The different structural formations could affect the strength of the geopolymer bricks in various ways. In this work, a

content of 31% of silicate solution is seen as the optimum, since it resulted in the highest UCS.

Table 7.5: Water absorption and density for the bricks at different curing temperatures with activator content of 31%, initial setting time of 15 minutes and curing time of 3 days.

Curing temperature (°C)	Water absorption (%)	Density (g/cm ³)
22	-	2.37
40	-	2.26
60	12.1	2.30
80	8.41	1.99
100	10.33	2.03

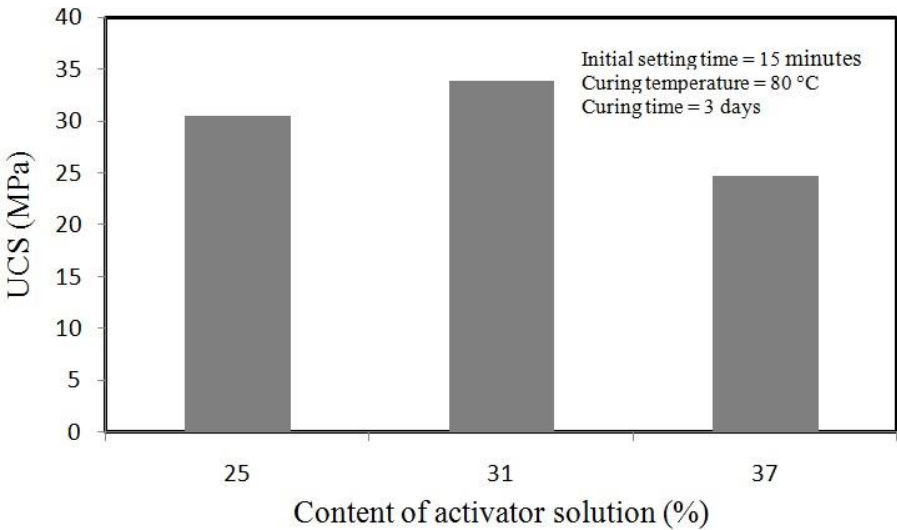


Figure 7.9: Variation of UCS versus content of activator solution.

Figure 7.10 shows how the UCS varies with curing time; while the initial setting time, content of activator, and temperature are maintained at the constant values of 15 minutes, 31%, and 80 °C, respectively. The UCS increases for the first 7 days, and then starts to decrease; if the curing time is extended out to 14, and then 28 days. The curing time that resulted in the highest UCS was 7 days (50.58 MPa).

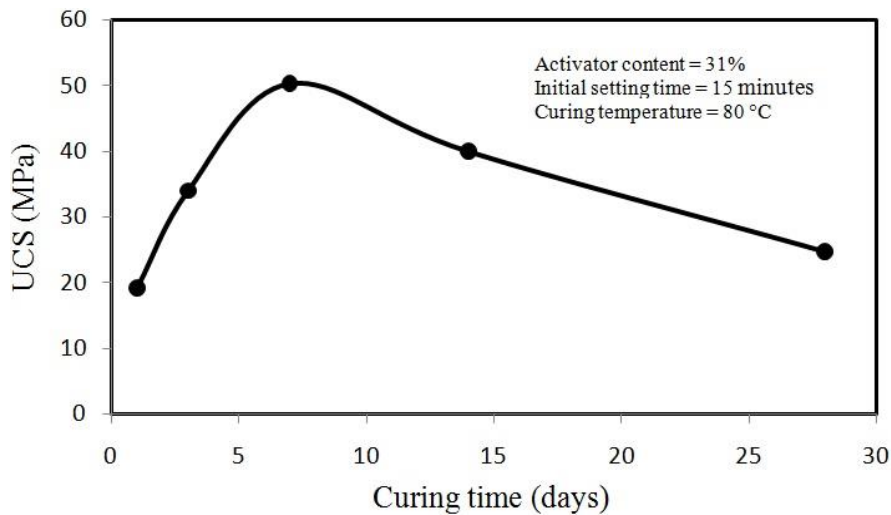


Figure 7.10: Variation of UCS versus curing time.

It should also be noted that even 1 day, and 3 days curing resulted in satisfactory UCS values of 19.18 MPa, and 34.00 MPa, respectively; which are within the standards for some applications as mentioned in Tables 7.4 (a) and 7.4 (b). Seven days of curing at the conditions stated above resulted in a very high UCS; which meets the requirements of all the specified applications for bricks, except pedestrian, and light traffic paving bricks, under severe weather conditions. When considering the factors of curing time, and curing temperature in isolation; UCS development is favoured by higher temperatures (80 °C), at moderate curing times of up to seven days. If the same higher temperature is maintained with longer curing times (up to 14, or 28 days), there is a negative effect on UCS development. It is understood that for 7 days of curing at 80 °C, the reaction of alumina-silicate materials goes to completion, and no further dissolution takes place; consequently stopping the formation of any more alumina-silicate gel in the mixture (de Vergas et al., 2011). If after this stage, the bricks are subjected to further curing, of say either 14, or 28 days, at the same higher temperature; crystallisation occurs which affects the strength negatively.

Table 7.6 shows the values of density and water absorption for a range of curing times. It can be seen that all the water absorption values are far below 17%, which is considered to be suitable for any type of brick application. In addition, all the densities are within the acceptable limits.

Table 7.6: Water absorption and density for the bricks at different curing times with curing temperature of 80 °C, activator content of 31% and initial setting time of 15 minutes.

Curing time (days)	Water absorption (%)	Density (g/cm ³)
1	8.72	2.08
3	8.41	1.99
7	9.00	2.13
14	8.90	2.11
28	7.86	2.06

Bricks that were produced using optimum values for each parameter, were tested for resistance to salt attack according to AS 4456.10 (2003). After 15 cycles of tests, bricks which had been cured for 3 days experienced a 0.2 g loss of mass, and bricks cured for 7 days experienced 0.1 g loss of mass. These mass losses were less than the threshold requirement of 0.4 g (Think Brick Australia, 2013). Therefore, the bricks can be classified as being a general purpose brick, and have an increased durability. When the bricks with the optimum parameters were tested for their potential to effloresce according to AS 4456.6 (2003), by immersion in water for 7 days to observe any salt deposits, nothing was found or there was a no observable efflorescence. The outcome of the efflorescence test is that the quality of the brick can be considered to be of a high standard.

Table 7.7 shows a summary of the electrical resistivity, the UCS, the water absorption, and the densities of the geopolymer bricks when utilizing the optimum parameters. For a comparison purpose, the values of electrical resistivity, the UCS, the water absorption, and the densities of a commonly used commercial brick are also given in Table 7.7.

The electrical resistivity of a brick is expected to be high, so as to avoid possible electrical shocks, or conduction of electricity through the building materials. (Telford et al., 1990) have reported that the average electrical resistivity for a fire clay is 30 Ωm. (PennState Matse, 2014) also reports that a typical electrical resistivity of fire clay brick is 1000 kΩm; while it has been reported to range from 0.0068 kΩm to 24.5 kΩm, for a firing temperature range of 300 °C to 1300 °C, respectively (White, 1932). The electrical resistivity of the commercial brick was found to be 1080 kΩm, while that for the geopolymer bricks produced

in this work was found to be 456 kΩm, 642 kΩm, and > 682 kΩm for one, three, and seven days of curing, respectively; as seen in Table 7.7. The commercial clay brick was expected to have a substantially higher value of electrical resistivity compared to the geopolymer bricks produced in this study, because of the high iron ore content of the tailings. The iron ore tailings are known to be more conductive than clay, and as expected, leading to a lower resistivity when the two bricks are compared. However, all the resistivity values reported at the various optimum parameter values for the geopolymer bricks produced in this study, were still high enough to be considered acceptable for each application of the bricks.

Table 7.7: Summary of the various properties of the bricks at different curing times with curing temperature of 80 °C, activator content of 31% and initial setting time 15 minutes.

Curing Conditions	Electrical resistivity (kΩm)	Compressive strength (MPa)	Water absorption (%)	Density (g/cm ³)
Commercial clay brick	1,080	25.72	7.70	2.18
Tailings brick at 1 day curing	456	19.18	8.72	2.08
Tailings brick at 3 days curing	642	34.00	8.41	1.99
Tailings brick at 7 days curing	> 682	50.35	9.00	2.13

The above discussions show that the UCS varies from 0.7 MPa to 50.35 MPa depending on the forming and curing conditions selected. Consequently if the appropriate forming and curing conditions are selected; geopolymer bricks can be produced from iron ore tailings, which meet all the requirements of the ASTM and the Australian Standards of bricks; except pedestrian and light traffic paving bricks (for use under severe weathering conditions), with a minimum UCS value of 55.2 MPa. For instance to make a building brick which can withstand severe weathering conditions, having a minimum UCS of 20.7 MPa and maximum water absorption of 17%; the following parameters could be chosen as the optimum forming conditions: 1) 15 minutes initial setting time; 2) 80 °C curing temperature; 3) 31% sodium

silicate solution content; and 4) 3 days curing time. These forming conditions would enable the resulting bricks to exceed the standard requirements, as the UCS for this selection is found to be 34 MPa. However, if a better performing brick is required, the oven curing time could be extended to 7 days; while maintaining the values of the other parameters, to achieve a UCS of 50.35 MPa.

Figure 7.11 (a) is the X-ray diffraction (XRD) pattern for a commercial clay brick. It is observed to have crystalline peaks consisting mainly of quartz (silica), mullite, and goethite. The peak with the highest intensity was quartz, followed by mullite, and then goethite in that order. The XRD pattern reveals that the particular clay used for this commercial brick is mainly quartz, with minor amounts of other minerals. In Figure 7.11 (b), which is for the original iron ore mine tailings, the XRD pattern is observed to contain crystalline peaks indicating a composition primarily of quartz, goethite, aluminian, birnessite, and sodian. The Mullite which is seen in the commercial clay brick is absent in the mine tailings. By contrast, the mine tailings contains birnessite and sodian, which are absent from the commercial clay brick. Our analysis has revealed that the iron ore tailings contain a large amount of silica. From the XRD patterns, it is observed that both the iron ore tailings and the resulting geopolymer bricks made from the tailings have crystalline phases. However, the main difference observed between the iron ore tailings and the finished geopolymer bricks, is the decrease in the intensity of the crystalline peaks, as seen in Figure 7.11 (c) for the finished geopolymer brick. This is observed in the birnessite and sodian that decreased from 43% in the unprocessed tailings, to 37% in the geopolymer bricks. This result may be due to the partial dissolution of the alumina-silicate minerals in the iron ore tailings; making it possible for the undissolved alumina-silicate solids to react with the partially dissolved alumina-silicate minerals as a polymeric gel. The polymeric gel in turn serves as a binder to further strengthen the bonds between the dissolved and undissolved solids, thereby contributing to a higher strength (Palomo et al., 1992). The bonds between the dissolved and the undissolved solids reorganise themselves during the reaction; and alter the structure from crystalline phases, to a final geopolymer with amorphous and semi-crystalline phases. When comparing Figures 7.11 (b) and 7.11 (c), it is seen that aluminian which was initially present in the iron ore tailings, does not appear in the final geopolymer brick. It is possible that the aluminian may have been completely dissolved in the geopolymeric gel, enhancing the strength development.

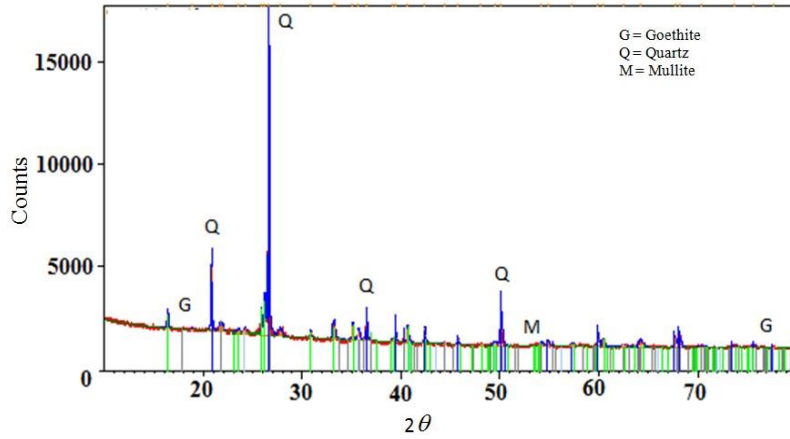


Figure 7.11 (a): XRD pattern of commercial clay brick.

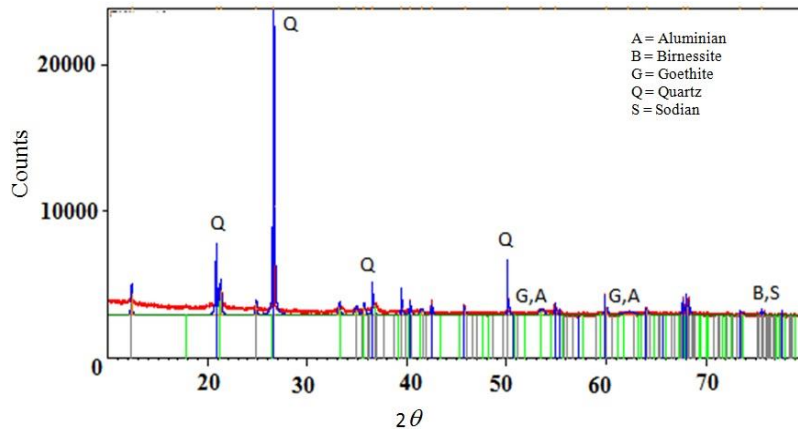


Figure 7.11 (b): XRD pattern of original iron ore mine tailings.

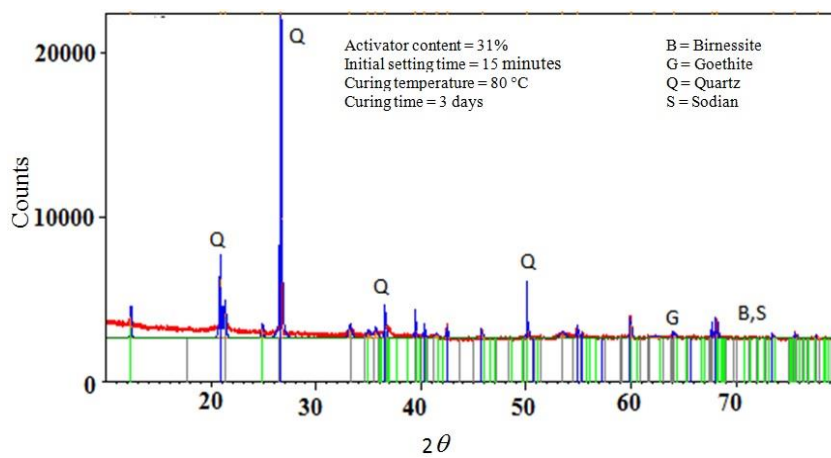
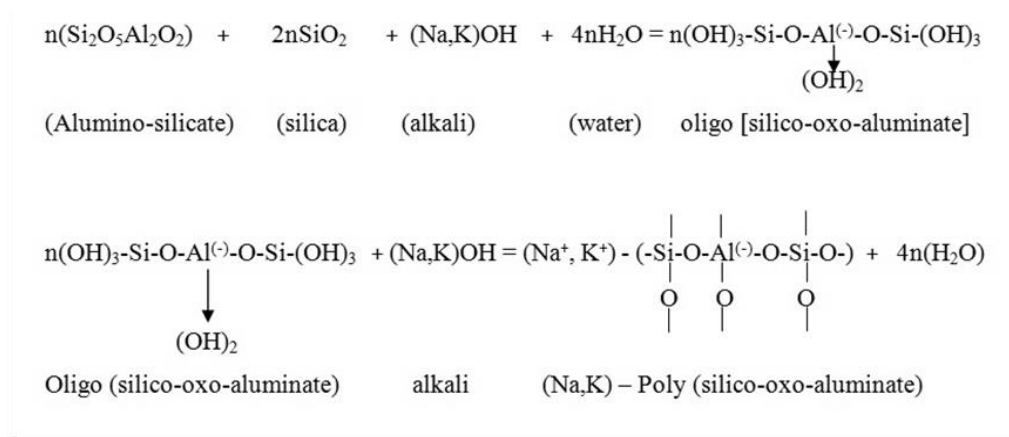


Figure 7.11 (c): XRD pattern of iron ore mine tailings geopolymer brick.

It is important to emphasize that the geopolymerization process is a chemical reaction between the constituent alumina-silicate materials; and as a consequence, the nature or extent of dissolution will influence the final strength of the geopolymer. The process results in the formation of a gel-like complex compound called poly silico-oxo-aluminate (Drechsler & Graham, 2005). This compound is comprised of crosslinked Si-O-Al-O-Si bonds, which crystallise to form a hardened and stable, amorphous or semi-crystalline structure. The chemical reactions that depict the process are shown below (Drechsler & Graham, 2005);



It should be noted that this study was carried out in a laboratory setting. It is anticipated that the strength development, and the properties of the tailings geopolymer bricks, could be enhanced further with the use of commercial manufacturing equipment.

7.5 COST ANALYSIS

A simplified cost analysis based on Western Australian conditions, and ignoring the cost of the iron ore tailings as they are currently considered to be a waste material, has been made for a full-size brick as detailed below:

Mass of tailings ground in each cycle in the ball mill = 5000 g

Time taken to grind 5000 g of tailings = 75 minutes

Required size of tailings needed = 212 μm

Density of brick required = 2.1 g/cm^3

Internal size of the standard mould:

Length = 23 cm, width = 11 cm, and height = 7.6 cm

Amount of the mixture of tailing material and sodium silicate required to produce one standard brick = $23 \times 11 \times 7.6 \times 2.1 = 4037.9$ g

The amount of the sodium silicate solution required to make one standard brick

$$= (31/100) \times 4037.9 \text{ g} = 1251.7 \text{ g}$$

Amount of tailing material required to make one standard brick

$$= 4037.9 \text{ g} - 1251.7 \text{ g} = 2786.2 \text{ g}$$

Amount of tailing material finer than 212 μm available in the tailings

$$= 15.6\% \text{ of } 2786.2 \text{ g} = 434.6 \text{ g}$$

Amount of tailing material finer than 212 μm obtained after grinding of the tailings

$$= 2786.2 - 434.6 = 2351.6 \text{ g}$$

The milling time for producing one standard brick

$$= (2351.6/5000) \times 75 = 35.3 \text{ min or } 0.588 \text{ h}$$

The laboratory ball mill used has a power rating of 0.15 kW or 150 W. The current electricity tariff in Australia set as k1/k2, combined residential/business properties is 27.0016 cents or AU\$ 0.270016 per unit (kWh) (WA Department of Finance, 2014). Therefore, the cost of running the ball mill for the

0.588 h to make one standard brick

$$= (0.15 \times 0.588 \times 0.270016) = \text{AU\$ } \mathbf{0.024}$$

The sodium silicate solution was obtained from Coogee Chemicals and the price tag was quoted as AU\$ 500 per 1,000,000 g (1 metric ton). In this work, 31% of the activator was found as being the optimum amount to make one brick from the tailings. Therefore, the cost of the sodium silicate solution to make one standard brick

$$= (1251.7/1000000) \times 500 = \text{AU\$ } \mathbf{0.626}$$

The power rating for the oven used is 3 kW or 3000 W. Therefore the cost for running the oven for one hour

$$= 3 \times 0.270016 = \text{AU\$ } 0.810$$

The oven has a capacity of 432.45 litres or 432450 cm^3 . Since one standard brick has a volume of $23 \times 11 \times 7.6 = 1922.8 \text{ cm}^3$, the total number of bricks that the oven can take at a time

$$= (432450/1922.8) = 225 \text{ bricks}$$

The oven can take 225 bricks at a time at the same cost per hour. Thus, the cost of oven drying for one standard brick for 1 hour

$$= (1/225) \times 0.810 = \text{AU\$ } 0.004$$

Thus, the cost of oven drying for one standard brick for 24 hours (1 day)

$$=24 \times 0.004 = \text{AU\$ } 0.096$$

Thus, the cost of oven drying for one standard brick for 3 days

$$=3 \times 0.096 = \text{AU\$ } 0.288$$

Thus, the cost of oven drying for one standard brick for 7 days

$$=7 \times 0.096 = \text{AU\$ } 0.672$$

From the above considerations, it can be estimated that the cost of producing one geopolymer brick from tailings, with 1 day of curing to obtain the strength of 19.18 MPa, is AU\$ (0.024 + 0.626 + 0.096) = AU\$ 0.746

Since the cost of one Western Australian normal brick tagged as handmade with the standard dimensions of 23 cm × 11 cm × 7.6 cm is AU\$ 1.18, according to the Midland Brick Company in Western Australia; the geopolymer brick will cost less by $(1.18 - 0.746) / 1.18 \times 100 = 36.8\%$.

If the brick is cured for three days to achieve a higher strength of 34 MPa, the cost of one geopolymer brick will be AU\$ (0.024 + 0.626 + 0.288) = AU\$ 0.938. The three days of curing will reduce the cost by $(1.18 - 0.938) / 1.18 \times 100 = 20.5\%$.

If higher specification bricks are desired, requiring a strength of 50.35 MPa; the curing time could be extended to 7 days, and the cost of this brick will be AU\$ (0.024 + 0.626 + 0.672) = AU\$ 1.322. This will cost more than the price of a commercially available brick by $(1.322 - 1.18) / 1.18 \times 100 = 12.0\%$.

7.6 CONCLUSIONS

The possibility of utilizing iron ore tailings produced at mine sites in Western Australia for the production of geopolymer bricks were investigated. This was done by producing a number of standard bricks using the tailings material, and studying their various properties, including unconfined compressive strength (UCS), durability properties such as water absorption, resistance to salt attack, and potential for efflorescence, as well as their electrical resistivity. The effects of four significant parameters, namely initial setting time, content of sodium silicate solution (activator), curing temperature, and curing time on the strength of the bricks were studied. Based on the study presented here, the following general conclusions can be drawn:

1. The strength of the geopolymer bricks made from iron ore tailings with sodium silicate solution, is influenced greatly by the curing temperature. The UCS increases as the curing temperature increases to a certain optimum point (80 °C), and then the UCS starts decreasing as the temperature increases further.
2. The optimum base parameters for the production of the geopolymer bricks are sodium silicate solution content of 31%, initial setting time of 15 minutes, and curing temperature of 80°C. In addition the curing time of one day for the brick results in an optimum UCS of 19.18 MPa, three days curing results in a UCS of 34.00 MPa, and seven days curing provides an optimum UCS of 50.35 MPa. When subjected to further curing above seven days, the UCS starts to decline.
3. The electrical resistivity of geopolymer bricks is lower than the commercial clay bricks due to the higher iron content associated with the iron ore mine tailings. However, the electrical resistivity of the geopolymer bricks is still high enough to be used for building construction. The electrical resistivity increases from a value of 456 kΩm for one day curing, to > 682 kΩm for 7 days curing.
4. The XRD pattern reveals that there is a decrease in the intensity (count) of the crystalline peaks in the final tailings geopolymer bricks. The primary cause of the decrease being the partial dissolution of alumina silicate minerals, producing a polymeric gel that reacts with the undissolved alumina silicate minerals, resulting in strong bonds. These strong bonds enhance strength development in the final geopolymer bricks.
5. The tailings geopolymer bricks met both the ASTM and the Australian Standards requirements for the specification of bricks. Even when the bricks were cured for seven days, the resulting bricks had superior qualities than both the ASTM and the Australian Standards requirements. This has provided an avenue for cheaper and alternative materials for civil engineering constructions.
6. The finished bricks are more economical than the commercial clay bricks, with a cost reduction of either 36.8% or 20.5%, when one day or three day curing times, respectively, are selected. If the curing for seven days is selected, the bricks produced exceed the cost of commercial clay bricks by 12.0%, but they have superior qualities over commercial clay bricks.
7. The new bricks have environmental benefits over the commercial bricks, because of a reduction in energy consumption. The reduction is achieved by adopting simple, low temperature oven drying, as opposed to high temperature kiln firing in the production

of commercial clay bricks. The quantity of mine tailings returned to the environment will be reduced; and the need to mine fresh clay materials for the production of bricks will be avoided, by substituting with mine tailings. This will lead to a reduction of greenhouse gases and a reduced ecological footprint.

8. With this new technology, there is the possibility of providing ongoing and guaranteed employment for the inhabitants of the mining communities, by eliminating the shocks and economic stand-still that the mining communities encounter upon mine closure.

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CHAPTER 8

UTILISATION OF IRON ORE TAILINGS AS AGGREGATES IN CONCRETE

This chapter has been submitted to the Journal of Cogent Engineering, of Taylor and Francis Group, United Kingdom and it is currently under review; as mentioned in number 4 of Section 1.7. The details presented here are the same, except some changes in the layout in order to maintain a consistency in the presentation throughout the thesis.

8.1 ABSTRACT

Sustainable handling of iron ore tailings is of prime concern to all stakeholders who are into iron ore mining. This study seeks to add value to the tailings by utilising them as a replacement for aggregates in concrete. A concrete mix of grade 40 MPa was prepared in the laboratory with water-cement ratio of 0.5. The concrete were cured for 1, 2, 3, 7, 14 and 28 days. The properties of the concrete such as workability, durability, density, compressive strength and indirect tensile strength were tested. A controlled mix of concrete was also prepared in similar way using conventional materials and the results were compared with the tailings concrete. It was found that the iron ore tailings may be utilised for complete replacement for conventional aggregates in concrete. The iron ore tailings aggregates concrete exhibited a good mechanical strength and even in the case of compressive strength, there was an improvement of 11.56% over conventional aggregates concrete. The indirect tensile strength did not improve against the control mix due high content of fines in the tailings aggregates but showed 4.8% improvement compared with the previous study where the conventional fine aggregates was partially replaced by 20% with iron ore tailings.

8.2 INTRODUCTION

Western Australia (WA) is endowed with a large reserve of iron ore. This has attracted a lot of iron ore mining activities for many years in the state. Iron ore resource is a very significant player of the economy of WA with an average yearly production of 316 million tonnes which accounts for about 47% of the total value of all resources mined in the state (W.A. Department of Mines and Petroleum, 2009).

The many years of large scale iron ore mining has resulted in the accumulation of huge quantities of iron ore tailings in the state which needs to be handled, disposed off and monitored properly. It has been reported that in WA on average, the production of 1 tonne of iron ore results in the generation of 2 tonnes of iron ore tailings (Price, 2004). It can therefore be estimated that about 632 million tonnes of iron ore tailings are generated yearly in WA. This quantity is so huge to the extent that if sustainable handling of these tailings is not found it could lead to adverse effects on the environment and human health. One of the common environmental problems associated with these tailings is the formation of acid mine drainage (AMD) which could be a potential source of surface and ground water pollution (Cassiano et al., 2012). These tailings also consume a lot of land that could be used for other purposes, and compromise the good looks of the environment in these areas (Yellishetty et al., 2008; Thomas et al., 2013). There could be a potential of erosion from these tailings dumps into the environment (Yellishetty et al., 2012). The current practice is that a smaller percentage of these tailings are returned to the mined out areas as backfills to fill the void created while majority of them is stockpiled in the environment or returned to a special tailings dam built for storage purposes.

In order to minimise the problems created by the mine tailings, one potential application area to explore is its utilisation in building and construction. This is because there is a greater potential in this sector where recycled wastes products could be considered as construction and building materials. An example is the use of these mine tailings as aggregates in concrete (Yellishetty et al., 2008; Thomas et al., 2013; Thomas & Gupta, 2013; Shetty et al., 2014).

Aggregates make up about 70% to 80% of a concrete mix (Thomas & Gupta, 2013; Shetty et al., 2014). As the natural granite quarries for aggregates are gradually decreasing, there would be the need for alternative materials to be used as natural aggregates in concrete. If mine tailings are considered as a partial or complete replacement of natural aggregates in concrete, majority of these tailings could be recycled and used sustainably, by turning these mine tailings into useful resource and providing cheaper alternatives in concrete production (Ugama et al., 2014). This will eliminate the need to mine virgin materials as concretes aggregates. The scarce resources could be conserved while at the same time providing sustainable solution to the handling of the tailings. This could make it possible for the mining industries concerned to generate extra income to defray their cost of production to maximise profit margin.

The processing activities associated with the iron ore beneficiation is such that it results in the tailings with particle sizes ranging from fine to coarse (Kuranchie et al., 2014a). If these tailings are segregated properly both fine and coarse aggregates for concrete could be obtained. Therefore, if the iron ore mining companies in WA could incorporate comprehensive utilisation of the tailings in their operation, it could lead to cleaner production and sustainable development in their operation. This could have a potential of reducing both cost and environmental impacts created by these tailings (Haibin & Zhenling, 2010).

The aim of this paper is to study the feasibility of using the iron ore tailings as both fine and coarse aggregates in making concrete. The paper evaluates the technical and the environmental characteristics of the concrete with the main concern of recycling and adding economic value to the iron ore tailings to be used as alternative cheaper materials for concrete aggregates.

8.3 EXPERIMENTAL PROGRAMME

In this section the standard process to make concrete was followed. Several tests were conducted to measure physical properties of concrete at different ages and the results were compared with those of normal concrete made using conventional aggregates.

8.3.1 Materials

For the normal concrete, conventional materials such as sand, cement and granite aggregates were used. For the tailings concrete being studied iron ore tailings were used together with the cement to make the new concrete.

The iron ore tailings were obtained from Mount Gibson Iron Extension Hill Operations in Perenjori of Western Australia (WA). The particle-size distribution curve of the original tailings as obtained is shown in Figure 8.1. The size of the tailings range from fines ($< 75\mu\text{m}$), to coarse ($\leq 32\text{ mm}$). Other physical and chemical properties of the tailings are presented in Tables 8.1 and 8.2, respectively (Kuranchie et al., 2014a).

Table 8.1: Physical properties of iron ore tailings (Kuranchie et al., 2014a).

Property	Value
Fines (clay and silt) (%)	6.5
Sand content (%)	69.5
Gravel content (%)	24.0
Specific gravity	2.65
Minimum dry density, ρ_{dmin} (kg/m ³)	1685.00
Maximum dry density, ρ_{dmax} (kg/m ³)	1860.00
Effective grain size, D_{10} (mm)	0.13
D_{60} (mm)	2.67
D_{30} (mm)	0.70
Coefficient of uniformity, C_U	20.54
Coefficient of Curvature, C_C	1.41
Soil classification as per USCS	SW-SM (Well graded sand-silty sand)

Table 8.2: Chemical composition of iron ore mine tailings (Kuranchie et al., 2014a).

Constituents	Amount (%)
CaO	0.03
SiO ₂	57.31
Al ₂ O ₃	9.58
Fe ₂ O ₃	25.13
MgO	0.08
SO ₃	0.16
Na ₂ O	0.04
K ₂ O	0.04
TiO ₂	0.61
As	---
Loss on ignition	6.67

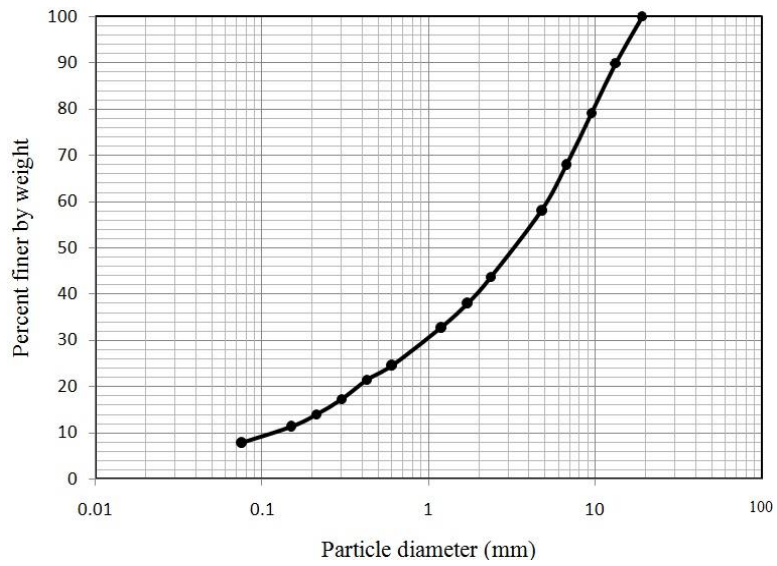


Figure 8.1: Particle-size distribution curve of tailings as received.



(a) Mine tailings fine aggregates (b) Mine tailings coarse aggregates



(b) Sand fine aggregates (d) Granite coarse aggregates

Figure 8.2: Normal and tailings aggregate.

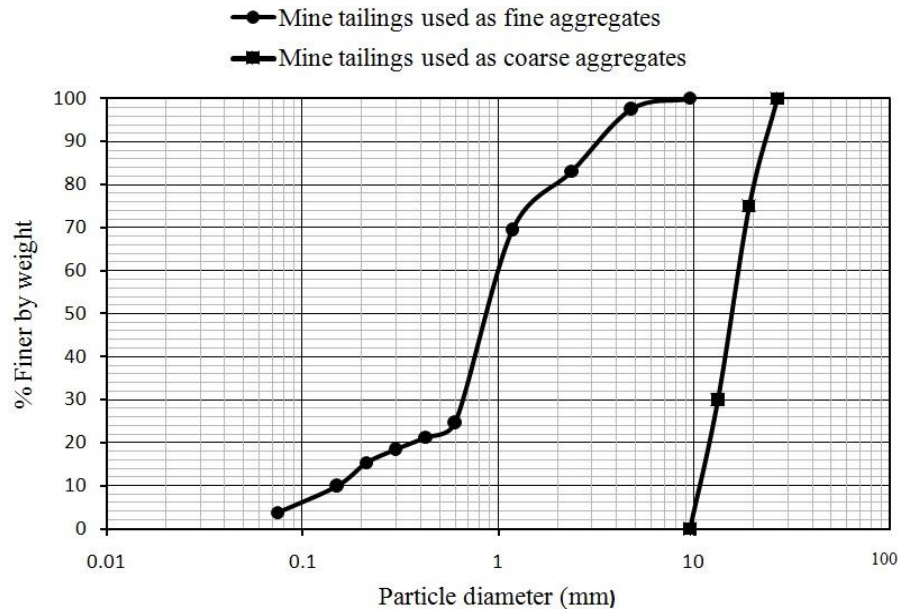


Figure 8.3: Particle-size distribution curves of the tailings used as aggregates.

The mine tailings were segregated using various sieve sizes. The particles ranging from 4.75 mm and below were used as fine aggregates while those ranging from 20 mm down to 10 mm were used as coarse aggregates in making the tailings concrete shown in Figure 8.2 (a) and (b). Figure 8.3 shows the particle-size distribution curves of the tailings aggregates and they conform to AS 2758.1. Ordinary general purpose Portland cement of grade of 43 which conforms to AS 3972 was used. This was obtained from Cockburn Cement Limited in Western Australia. The cement had a normal consistency of 29.5%, soundness of 1 mm with initial setting time of 2 hours and final setting time of 3.15 hours. The specific gravity of the cement was 3.1.

Commonly used sand for various construction purposes in Perth known as ‘brickies sand’, was obtained from a quarry site, 40 km North of Perth city. This was used as fine aggregates in the normal concrete as a control mix. This can be seen in Figure 8.2 (c). The physical properties of the sand are given in Table 8.3 (Kuranchie et al., 2014b). The sand was classified as poorly graded sand (SP) as per the Unified Soil Classification System (USCS).

Natural granite quarry of size 10 mm and 20 mm, which were obtained from commercial quarry, were mixed and used as the coarse aggregates in the control mix. This is shown in Figure 8.2 (d).

8.3.2 Mix design and preparation of specimens

A concrete mix design of grade 40 MPa and a water cement ratio of 0.5 was prepared in the laboratory. The mix design recommended by Marsh (1997) was followed and it conforms to AS 1012.2. In the tailings aggregates concrete, i.e. where 100% of the aggregates consisted of iron ore tailings as described above, 18 cylindrical mould specimens of size 100 mm internal diameter and 200 mm length were prepared for 1, 2, 3, 7, 14 and 28 days curing duration for compressive strength test. Other 18 cylindrical moulds of size 150 mm internal diameter and 300 mm length were prepared from the same mix with the same curing duration for the indirect tensile strength test. The mixes were prepared at a controlled indoor temperature range of 25 to 30 °C. The process was repeated using conventional aggregates to prepare the control mix which was used for comparison purposes.

After preparation the small moulds were covered with their lids and plastic sheets were used to cover the big moulds. The specimens were kept in their moulds for 24 hours before de-moulding. The specimens were then placed in the concrete curing chamber for the various curing durations explained above. Some quality assessment tests such as the density and the slump tests were performed on the fresh concrete whiles after curing, other quality assessment tests on hardened concrete such as compressive and split tensile strength were also conducted.

8.3.3 Tests for fresh concrete

Density and slump tests were conducted on the fresh concrete for both the control mix and the tailings aggregates mix. Density of both the fresh tailings aggregate concrete and the normal concrete were determined following usual laboratory procedures such that it conformed to AS 1012.5 for determining the density of a fresh concrete.

In determining the slump, a standard slump cone with dimensions, height of 310 mm, upper diameter of 198 mm and lower diameter of 100 mm was used. The concrete was filled in the cone in 3 layers. Each layer was compacted using a metal rod for 25 times to ensure good compaction. The concrete was carefully de-moulded from the cone. The cone was then placed upside down close to the concrete and the difference in heights was measured as the slump using a ruler. The procedure according to AS 1012.3.1 was followed. Figure 8.4 shows the procedure for the slump test.



Figure 8.4: Determination of the slump of a fresh concrete.

8.3.4 Tests for hardened concrete

Compressive strength test, indirect tensile strength test and acid resistance and alkalinity test were performed after the concrete have been cured for the various curing duration of the concrete specified. The acid resistance and alkalinity test was limited to the tailings aggregate concrete while all other test were conducted on both the normal aggregate and tailings aggregate concretes. Figures 8.5 (a) and 8.5 (b) show some examples of how the tailings and the normal aggregates concretes specimens look respectively, after they have been cured. The tailings aggregates concretes have a brownish colour and this was due to the brown colour of the iron ore tailings.



(a) Tailings aggregate concretes

(b) Normal aggregate concretes

Figure 8.5: Tailings and normal aggregates concrete.

Compressive strength was tested for the smaller cylindrical concrete specimens following AS 1012.9 with a compression testing machine. Three specimens were tested for each curing scheme and the average of the values was used. The compressive strength was calculated by dividing the maximum load (force) observed from the compression machine by the cross-sectional area of the specimen tested and the unit was expressed in megapascals (MPa).

Indirect tensile strength was tested for the beams in the bigger cylindrical moulds. This test was consistent with AS 1012.10. Similarly, three specimens were tested for each curing duration and the average of the values was used. The test was done using a compression machine in the laboratory. The indirect tensile strength of the specimen was calculated using the formula

$$T = \frac{2000P}{\pi LD} \quad (8.1)$$

where T is the indirect tensile strength in MPa, P is the maximum applied load indicated by the compression machine in kN, L is the length of the cylinder in mm and D is the diameter of the specimen.

The concrete cubes after the various curing days were dried for 24 hours in the oven at a temperature of 105 °C. The specimen was cooled down to room temperature and was broken with a hammer to separate the mortar from the concrete. The dried mortar was grinded using a ball mill and sieved below 150 µm. 10 g of the undersize was mixed with distilled water and stirred. The pH of the resulting solution was taken using a digital pH meter. This methodology was recommended by (Thomas & Gupta, 2013).

8.4 RESULTS AND DISCUSSION

The results for the various tests on both fresh and cured concrete have been summarised in this section and it has been discussed in detail. The results include both the normal aggregates and tailings aggregates concrete.

In the absence of experimental results the density of a fresh concrete may be considered as 2400 kg / m³ as a recommended by AS 3600. In this work, the density of the normal concrete is found to be 2345 kg/m³ and the density of the tailings concrete is 2362

kg/m³. The difference is due to the slightly higher specific gravity of the tailings. The two densities are comparable with those reported in (Thomas & Gupta, 2013).

Slump test is used to measure the workability and consistency of a fresh concrete (Marsh, 1997). In this work the slump for the normal concrete was calculated as 80 mm and that of the tailings concrete was 20 mm. Slump ranging from 0 to 25 mm is considered as having low workability.

Compressive strength is the main mechanical property of a concrete that is normally specified in supply of concrete. Figure 8.6 shows the compressive strength of both normal and tailings aggregates concrete. It was observed that the compressive strength for all the curing ages were higher in iron ore tailings aggregates concrete than in the normal concrete as the control mix. At age of 28 days the compressive strength of the tailings aggregates concrete was 36.95 MPa while that of the control mix was 33.12 MPa, which shows an improvement of 11.56% of tailings aggregates concrete over normal concrete. This improvement of strength of the iron ore tailings aggregates concrete may be attributed to the chemistry associated with the chemical composition of the iron ore tailings. It has been reported that iron compounds have the potential to accelerate cement hydration (Yellishetty et al., 2008) and this could be the main factor for the improvement of compressive strength observed in the iron ore tailings aggregates concrete which has higher percentage of iron compounds.

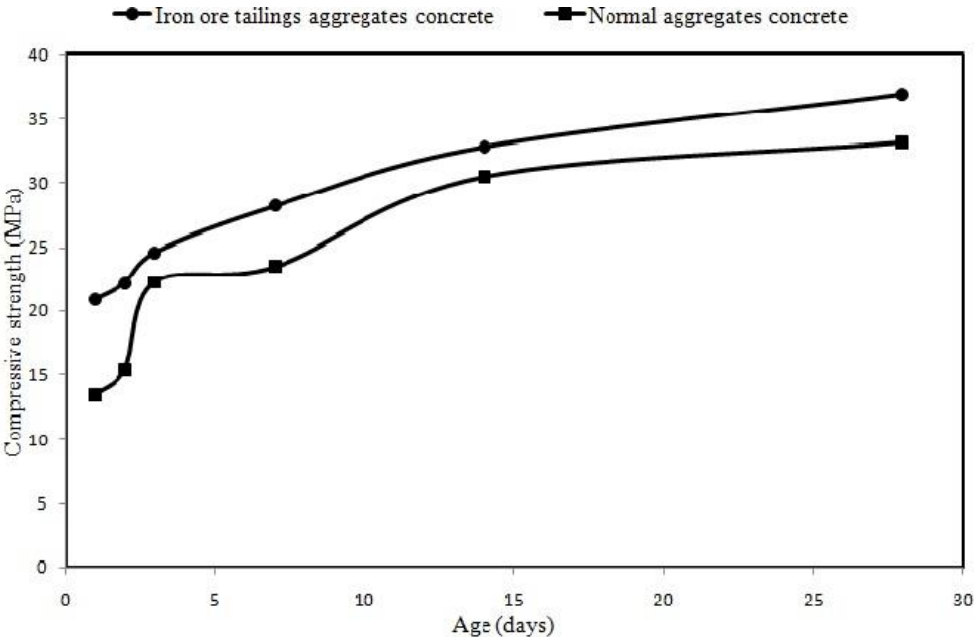


Figure 8.6: Compressive strength of normal and tailings aggregates concrete.

Tensile strength of concrete is another crucial mechanical property of concrete. This shows the strength of how the aggregates are bonded to the other materials in the concrete. Concrete structures are susceptible to tensile cracking and it becomes an important factor especially, when the concrete is intended to be used for the design of highway and airfield slabs (Neville, 2012). Therefore, the results from the indirect tensile strength test will investigate the influence of the aggregates structure on adhesion and the strength of the bond between the concrete materials.

Figure 8.7 shows indirect tensile strength of both normal and tailings aggregates concretes for various curing ages from 1 to 28 days. It can be observed that the indirect tensile strength favourably increases with the aging for both normal (control mix) and tailings aggregates concrete. From Figure 8.7 it is also observed that tensile strength for one day curing for tailings aggregates concrete was slightly higher than the control mix. As the concrete continues to age, the tensile strength of the normal aggregates concrete became higher than the tailings aggregates concrete. At age 28 days, the control mix achieved indirect tensile strength of 3.36 MPa while that of the tailings aggregates concrete was 2.82 MPa.

It is noted that, while the compressive strength improved, the tensile strength did not improve in the iron ore tailings aggregates concrete as compared with the control mix, although the compressive strength is directly proportional to the tensile strength. The reason may be attributed to the differences in the quantity of fines found in both materials used as fine aggregates.

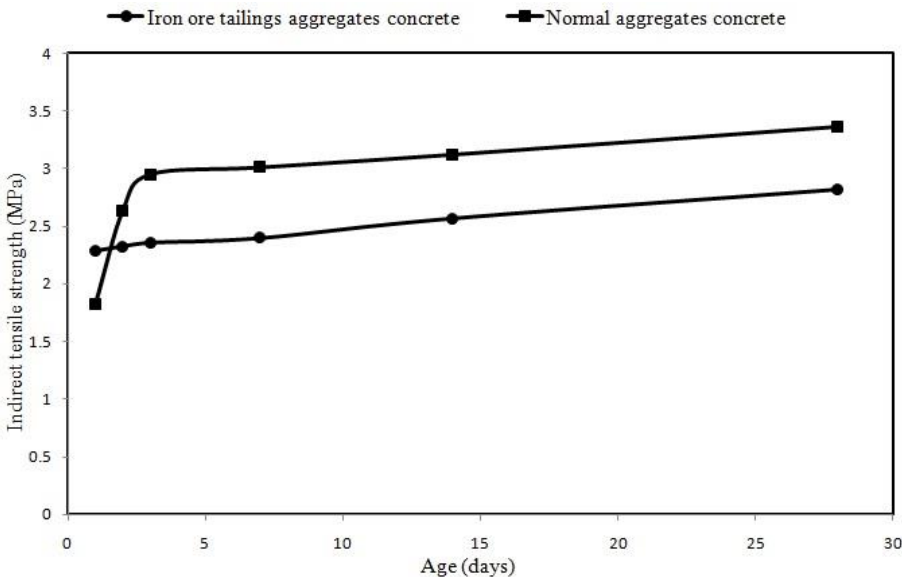


Figure 8.7: Indirect tensile strength of normal and tailings aggregates concrete

Table 8.3: Physical properties of sand (Kuranchie et al., 2014 b).

Property	Value
Fines (%)	3.85
Sand (%)	96.15
Specific gravity	2.66
Minimum dry density, ρ_{dmin} (kg/m ³)	1392
Maximum dry density, ρ_{dmax} (kg/m ³)	1598
Effective grain size, D_{10} (mm)	0.18
D_{60} (mm)	0.38
D_{30} (mm)	0.26
Coefficient of uniformity, C_U	2.11
Coefficient of Curvature, C_C	0.99
Soil classification as per USCS	Poorly graded sand (SP)

From Tables 8.1 and 8.3, it can be seen that iron ore tailings contain 6.5% fines while the natural sand used in this work contains only 3.85% fines. It should be noted that, in this work 100% of the iron ore tailings were used as fine aggregates in the tailings aggregates concrete. Therefore, this particular concrete will have more fines as compared with the control mix. The higher content of fines in the iron ore tailings aggregates concrete will increase the demand for water in the mix and this will reduce the bond strength existing in the aggregate-cement paste leading to a reduction in tensile strength (Adedayo & Onitiri, 2001; Ugama et al., 2014). However, the tensile strength exhibited by the iron ore tailings aggregates concrete in this work increased favourably with aging and there is 4.8% improvement as compared with similar work where only 20% of natural sand fine aggregates was replaced by iron ore tailings as reported earlier by (Ugama et al., 2014).

Mine tailings may contain some sulphide minerals and heavy metals, and there is a possibility that these minerals may come into contact with other oxidising agents such as oxygen and this will lead to the production of some acidic contents (Hitch et al., 2010; Thomas & Gupta, 2013). When the iron ore tailings aggregates concrete are exposed to these extreme conditions, disintegration of the concrete can occur due to acid attack. Also the ferrous content of the iron ore tailings is subject to corrosion which could also have negative effects on the long term durability of the concrete. Alkalinity and acid resistance test is therefore a means to check the long term durability of the tailings concrete against acid attack and corrosion. If the pH of the resulting solution is above 7 it means it is in alkaline condition and there is a potential that acid attack and potential for corrosion will be very low.

Figure 8.8 shows the pH values of resulting solution of the iron ore tailings aggregates concrete for the various curing ages. The pH values for the various ages of the concrete ranges from 11.8 to 12.5. These values can be termed as having high alkaline conditions and will therefore, make acid attack and potential for corrosion very low. This is a good sign of long term durability of the iron ore tailings concrete.

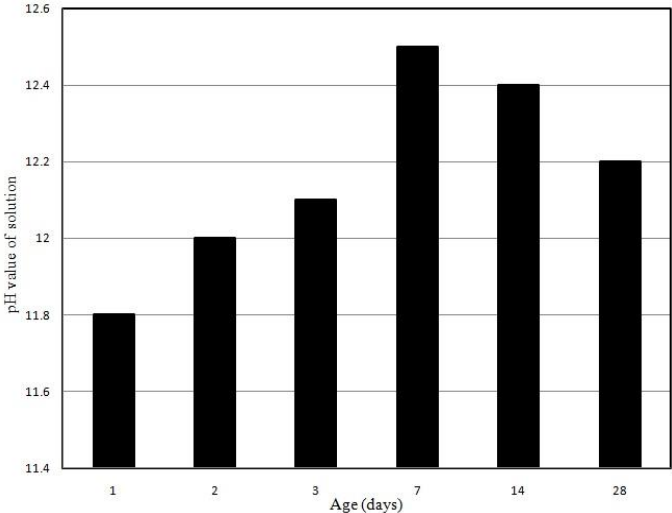


Figure 8.8: pH values showing the alkalinity of iron ore tailing aggregates concrete.

8.5 ECONOMIC AND ENVIRONMENTAL IMPLICATION

The cost of managing and monitoring mine tailings is very important to the life of every mine. This is because of some stringent regulations and legislations put in place by the appropriate Governmental authorities for the mining companies to adhere to in dealing with their tailings. As part of this, mining companies set aside specific budget for this purpose which is normally included in their cost of production. Comprehensive utilisation of mine tailings in this scenario will provide extra income to offset this cost for the mining companies upon mine closure. According to Packey (2012), the incorporation of utilising mine tailings in the operations of the mining companies could reduce rehabilitation costs and minimise the effects of mine closures on mining communities and bring about new economic activity.

Based on the results from this study, and the fact that aggregates forms about 70% to 80% of concrete, iron ore mine tailings in Western Australia could be considered as aggregates materials for concrete production where huge volumes of iron ore tailings could be utilised. The utilisation of the iron ore tailings could have positive effects on the environment because the quantity of iron ore tailings could be reduced tremendously. This could also lead

to the reduction of adverse environmental effects and render the operations of the iron ore mine companies more sustainable in state.

8.6 CONCLUSIONS

This study evaluated the possibility of completely replacing conventional aggregates in concrete with iron ore mine tailings produced in Western Australia to attain the same or better outputs for technical specifications. In the study, both conventional and tailings aggregates were used for comparison. Concrete mix design of 40 MPa and water cement ratio of 0.5 was used. Based on the results from this study, the following general conclusions could be made:

1. The compressive strength of the concrete with iron ore tailings aggregates at 28 days was 36.95 MPa which shows an improvement of 11.56% over the concrete with conventional aggregates. This is mainly because of favourable chemical composition of the iron ore tailings.
2. The split tensile strength exhibited by the concrete with tailings aggregates was 2.82 MPa at 28 days and this is slightly lower than concrete with conventional aggregates by 16% due to higher quantity of fines in the iron ore tailings as compared with the natural sand in the control mix. However, the tensile strength increased favourably with aging and there was still 4.8% improvement on the tensile strength as compared with similar study reported earlier.
3. The concrete with tailings aggregates has a low potential of corrosion and a low potential to acid attack due to high pH values of their resulting solution.
4. The utilisation of iron ore tailings as aggregates in concrete could have positive environmental implications to the mining companies and the mining communities and will provide cheaper alternative materials to bring about economy in concrete production.

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CHAPTER 9

LOAD-SETTLEMENT BEHAVIOUR OF A STRIP FOOTING RESTING ON IRON ORE TAILINGS AS A STRUCTURAL FILL

This chapter has been submitted to the International Journal of Mining Science and Technology, of Elsevier/SciencDirect Publication, China and it is currently under review; as mentioned in number 5 of Section 1.7. The details presented here are the same, except some changes in the layout in order to maintain a consistency in the presentation throughout the thesis.

9.1 ABSTRACT

This study presents a laboratory investigation of the load-settlement behaviour of a strip footing resting on iron ore tailings used as a homogeneous structural fill material in a model test tank. The strip footing was placed at various depths within the iron ore tailings. The relative density of the tailings was varied as $D_r = 50\%$, 70% and 90% . An incremental load was continuously applied on the footing while observing the settlement until there was a failure. The results obtained for the tailings were compared with that for sandy soil. The results show that the load-bearing capacity and the stiffness in terms of modulus of subgrade reaction increase with an increase in footing embedment depth as well as relative density of tailings. Compared to the load-settlement behaviour of Perth sandy soil, the tailings fill has significantly higher load-bearing capacity and stiffness. For example, for $D_r = 70\%$ and footing embedment depth ratio of 0.5, the load-bearing capacity and stiffness for the tailings are 22 times and 13.5 times higher. Therefore, the replacement of sandy soil with iron ore mine tailings in structural fills in construction of embankments, foundations and similar structures will be cost-effective and will contribute to environmental sustainability in construction.

9.2 INTRODUCTION

Building and construction works are normally associated with filling. Conventional fill material, commonly the local soil is used for this purpose. According to (Ash Development Association of Australia, 2012), a structural fill is an earthen material specifically engineered or designed to provide a strong and a stable base to support structures. Structural fills are

applied in engineering projects such as embankments, foundations, slopes, backfills, bridging layers, retaining functions, trenches, highway and railway engineering projects etc. The fill material is subject to meet some requirements such as good maximum dry density to enhance good compaction, high mechanical strength to allow less settlement when subjected to loading and consistent particle size distribution to allow good drainage characteristics (Industrial Resources Council, 2012). There is therefore, an increase demand for the natural soil or simply to mine enough of this soil to sustain the various construction works seen around. This excessive demand for the local sand has push prices forward and it contributes to the budget which makes construction very expensive in modern times. The identification of high quality and cheaper alternative materials as structural fills will incorporate cost effectiveness in modern construction (Industrial Resources Council, 2012).

Western Australia (WA) is one of the top mining jurisdictions in the world. The state has very active and extensive mining activities that have existed for long period of time, especially iron ore mining. This has given rise to the generation of huge quantities of iron ore tailings to be handled. It is estimated that about 632 million tonnes of iron ore tailings are produced in the state annually (Kuranchie et al., 2014). It has also been reported that these tailings have similar engineering properties compared with conventional materials that could be acceptable as civil engineering and general purpose construction materials (Yellishetty et al., 2008).

It is noted that the activities of the mining companies themselves involve some kind of filling. For instance, the construction of access roads to transport goods at the mine site, construction of tailings dams to store the tailings, and backfills. This therefore, makes it possible for the mine tailings to be considered for filling purpose internally by the mining companies and externally for engineering projects that require filling.

Structural fills could therefore be one of the potential applications of the iron ore tailings where huge quantities could be used to reduce significantly, the amount of iron ore tailings produced in the state (WA).

This study seeks to understand the load-settlement behaviour of iron ore tailings produced in Western Australia, for its intended use as a structural fill to replace the conventional earthfilling materials for enhanced environmental sustainability.

9.3 REVIEW OF PAST WORKS

Several conventional and waste materials have been explored in fill applications especially in embankments. Kehagia (2010) demonstrated the reuse potential of bauxite tailings as an

embankment material, and reported that the bauxite tailings were having good workability and compacted well which was good to be used as an embankment material.

In mining operations, filling is also a common habit that is adopted by most mines especially during backfilling. Normally, part of the mine tailings are returned to the voids created from the mined-out areas as a backfill material. This happens mostly when open-pit mining technique is adopted which is common in some Western Australian Mines. During mine closure and rehabilitation, the common practice is that the backfilled tailings are covered with layers of soil as a growing medium of planted vegetation cover. This makes the surface safe for earthmoving machines, example bulldozers to travel and work (Hossain & Fourie, 2013). Normally the layers on top of the tailings are stronger than the backfilled tailings. Hossain & Fourie (2013) studied the performance of the ground when these machines operate on the sand embankment over these mine tailings. They reported that placing a thicker sand layer on top of the mine tailings may prevent near crest failure.

Other studies have also focused on the use of fly ash from coal thermal plants as a fill material. Baykal et al. (2004) used a mixture of fly ash and crushed ice that is common in cold regions as an embankment material. The mixture was compacted together and they noted that the crushed ice introduced some moisture to the fly ash and this strengthened the material with time which led to the increase in compressive and tensile strengths in the embankment.

Trivedi & Sud (2007) conducted a laboratory plate load test experiment to analyse the load settlement behaviour of a compacted ash intended to be used as a structural fill. They observed that the settlement on footing on compacted dry ash is higher than on ash compacted wet and at lower degree of compaction; less than 90%, a shear failure may occur.

Chiaro et al. (2013) used a blend of coal wash and steel slag as a structural fill material for the reclamation of Port Kembla outer harbour near Wollongong in Australia. The parameters they considered specifically for the port reclamation were shear strength, bearing capacity, permeability, swelling and particle breakage. Their research concluded that steel slag content from 30% to 45% in the blend met all the expected requirements and the blend material could be used successfully for the project.

Trivedi & Sud (2005) presented an experimental investigation for footing on coal ash subjected to various loads. Their finding was that the failure of the ash fills is dependent on material characteristics of the ash, size and depth of the footing, and the settlement ratio.

Alizadeh et al. (2014) used a mixture of type 1 portland cement, class F fly ash, fine aggregates and water; together known as controlled low strength materials (CLSM) as a structural fill for bridge abutment to replace conventional earthfill materials. The important

parameters they considered were flowability, compressive strength and bond strength. They have reported that the higher temperatures accelerate early strength but lower the strength gain in the later stages when the CLSM material is aging. They recommended that, in using CLSM, their findings could be used as an initial guide in choosing raw materials and their proportions in structural fills for bridge abutment based on minimum strength requirements.

From the review above, it can be concluded that the use of iron ore tailings which is produced in large quantities in Western Australia (WA) as a fill material is very limited or simply unavailable. Most studies have concentrated extensively on fly ash as a fill material in waste utilisation. However, the absence of significant lime and clay minerals in iron ore tailings makes it fit as granular and cohesionless geo-material which could satisfy most geotechnical design requirements for structural fill (Trivedi & Sud, 2005; Chiaro et al., 2013).

The study will therefore investigate the load-settlement behaviour of iron ore tailings which is available in large quantity in Western Australia (WA) as a large volume civil engineering application in structural fills. In this case, the majority of the iron ore mine tailings could be recycled and reused for economy and environmental sustainability in construction.

9.4 EXPERIMENTAL WORK

A number of model tests on both surface and embedded footings on the iron ore tailings bed were carried out. This laboratory model was developed by (Kazi et al., 2014) at the Geotechnical Engineering Laboratory, Edith Cowan University. The experimental procedure of the model tests are presented in the following sections.

9.4.1 Materials

The iron ore tailings were collected from Mount Gibson Iron, Extension Hill, Perenjori (WA). Figure 9.1 shows the particle-size distribution curve of the tailings received. Size analysis of the tailings show that it ranges from fine ($75\mu\text{m}$) to coarse ($\leq 32\text{ mm}$). The physical and chemical properties of the tailings are shown in Tables 9.1 and 9.2, respectively (Kuranchie et al., 2014). As per the Unified Soil Classification System (USCS), the mine tailings were classified as well graded sand-silty sand. The tailings consisted of 6.5% fines and it was used in its air-dried form. The mine tailings were sieved and the material $\leq 19\text{ mm}$ was used for the experiment.

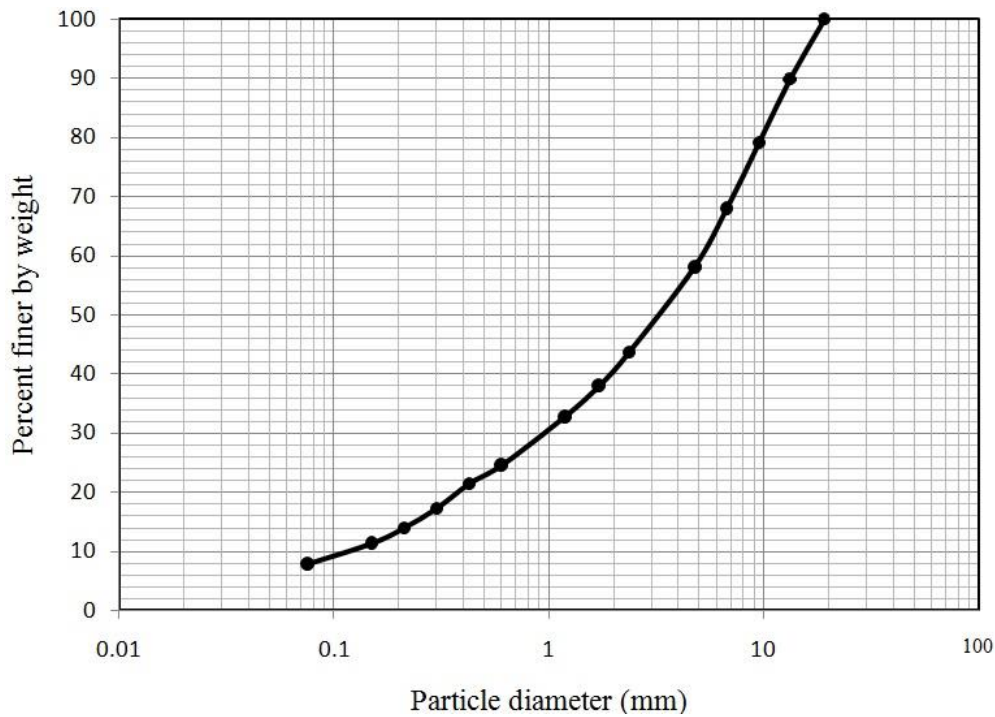


Figure 9.1: Particle-size distribution curve of iron ore tailings.

Table 9.1: Physical properties of iron ore mine tailings (Kuranchie et al., 2014).

Property	Value
Fines (clay and silt) (%)	6.5
Sand content (%)	69.5
Gravel content (%)	24.0
Specific gravity	2.65
Minimum dry density, ρ_{dmin} (kg/m ³)	1685.00
Maximum dry density, ρ_{dmax} (kg/m ³)	1860.00
Effective grain size, D_{10} (mm)	0.13
D_{60} (mm)	2.67
D_{30} (mm)	0.70
Coefficient of uniformity, C_U	20.54
Coefficient of Curvature, C_C	1.41
Soil classification as per USCS	SW-SM (Well graded sand-silty sand)

Table 9.2: Chemical properties of iron ore mine tailings (Kuranchie et al., 2014).

Constituents	Iron ore mine waste from the current study (%)
CaO	0.03
SiO ₂	57.31
Al ₂ O ₃	9.58
Fe ₂ O ₃	25.13
MgO	0.08
SO ₃	0.16
Na ₂ O	0.04
K ₂ O	0.04
TiO ₂	0.61
As	---
Loss on ignition	6.67

9.4.2 Test details

The tests were conducted in a laboratory model tank with internal dimensions of 1200 mm length, 400 mm width and 800 mm height. This was made from 25 mm thick Perspex material which is buttressed with high strength steel metal frame to absorb the load reaction.

The footing on which the load was applied is made up of a strip footing consisting of 40 mm thick rigid steel plate with length and width 390 mm and 80 mm respectively. The base of the footing was intentionally made rough by cementing a thin layer of the mine tailings to it using epoxy glue to prevent slipping on the surface of the tailings bed. The detailed laboratory test arrangement is shown in Figure 9.2, with (a) the model plan, (b) the cross section A and (c) the cross section B.

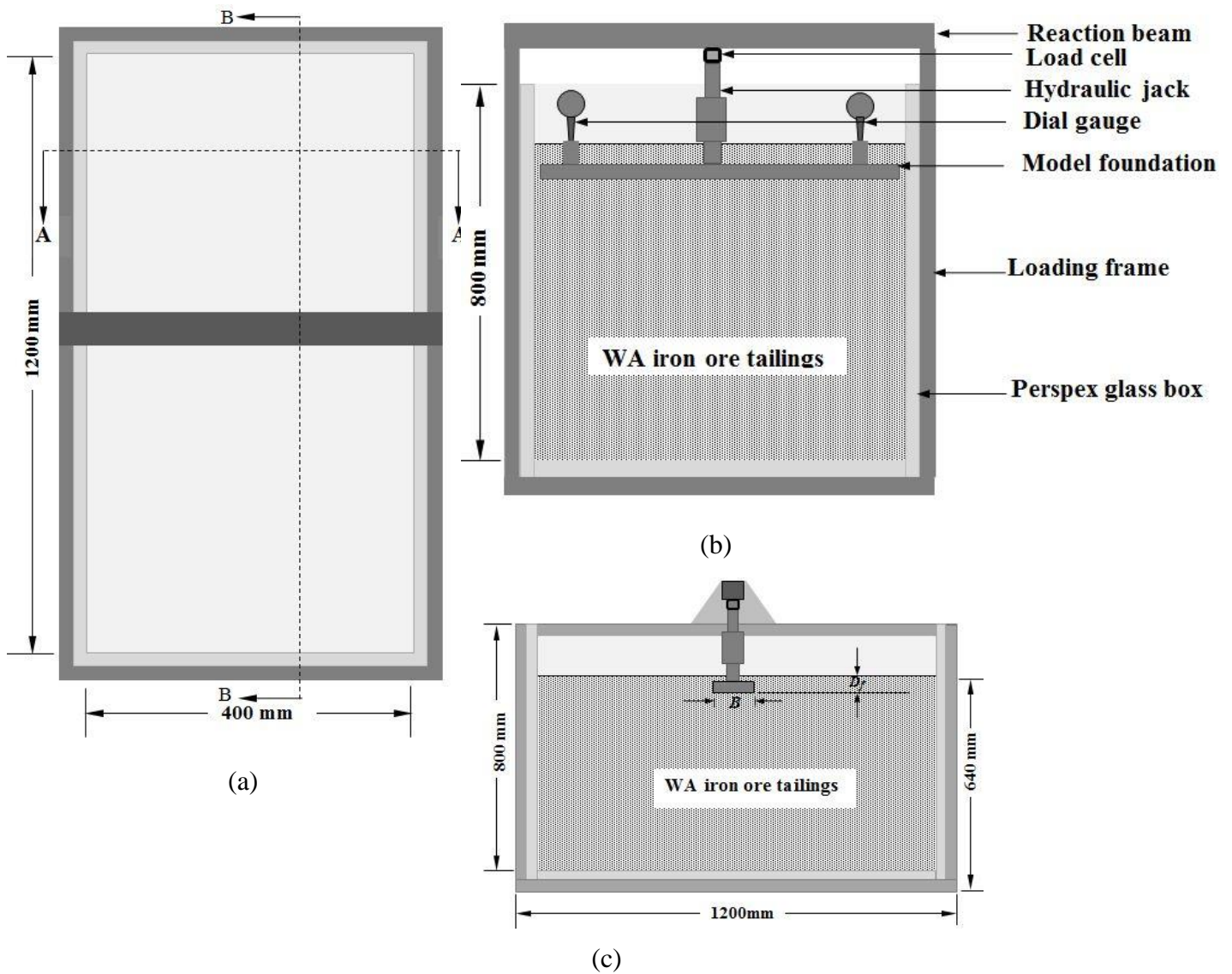


Figure 9.2: Laboratory test arrangement: (a) model plan, (b) cross section A, (c) cross section B.

The tests were conducted by varying the relative densities, D_r of the mine tailings. The relative densities used were $D_r = 50\%$, 70% , and 90% and these had equivalent densities of 1944.69 kg/m^3 , 1997.30 kg/m^3 and 2042.83 kg/m^3 , respectively. The model tank was filled with the mine tailings in four layers with each layer made up of 140 mm high. The compaction was done with a 25 mm vibratory poker to obtain the desired density. In order to have a homogeneous tailings bed in the model tank, care was taken by ensuring uniform compaction for all the individual layers to achieve the same density.

The top surface of the tailings bed was levelled and the footing was placed on top of the tailings bed for the case of surface footing with $D_f = 0$, where D_f is the depth of foundation as shown in Figure 9.2 (c). For the embedded footing, foundation depths of $D_f = 0.5B$, B , and $1.5B$ were used where B is the width of foundation as shown in Figure 9.2 (c). For the embedded case, the footing was placed on the tailings bed during the filling process and it was levelled afterwards where the footing was buried in the tailings bed. For both surface and embedded footing the footing was placed at the centre of the tailings bed to avoid eccentric loading. For each embedded or surface footing, the experiment was repeated for all the three relative densities.

An incremental compressive load of 0.78 kN which results in the pressure of 25 kN/m² was applied on the footing with the help of hydraulic jack with a capacity of 100 kN against the reaction beam. The load increments were observed using a load cell with 100 kN capacity. Two dial gauges that were located at equal distances from the centre of the footing were used to measure the settlement of the footing. The average reading from the two dial gauges were used. After each incremental load was applied, the settlement of the footing was allowed to become steady again before another load was applied. Incremental loads were applied continuously while noting the settlements until failure was observed within the tailings bed or the footing tilted where no more load could be taken by the footing. Figures 9.3 (a) and (b) show the arrangement with the embedded footing load test at initial stage of loading and at failure respectively. At the end of each test the model tank was completely emptied and tank refilled for other relative densities. This ensured standardisation throughout the tests for the investigation. The results from the mine tailings were compared with the results of similar test work using Perth local sandy soil (Kazi et al., 2014), which is the conventional earthfilling material used in Western Australia (WA).

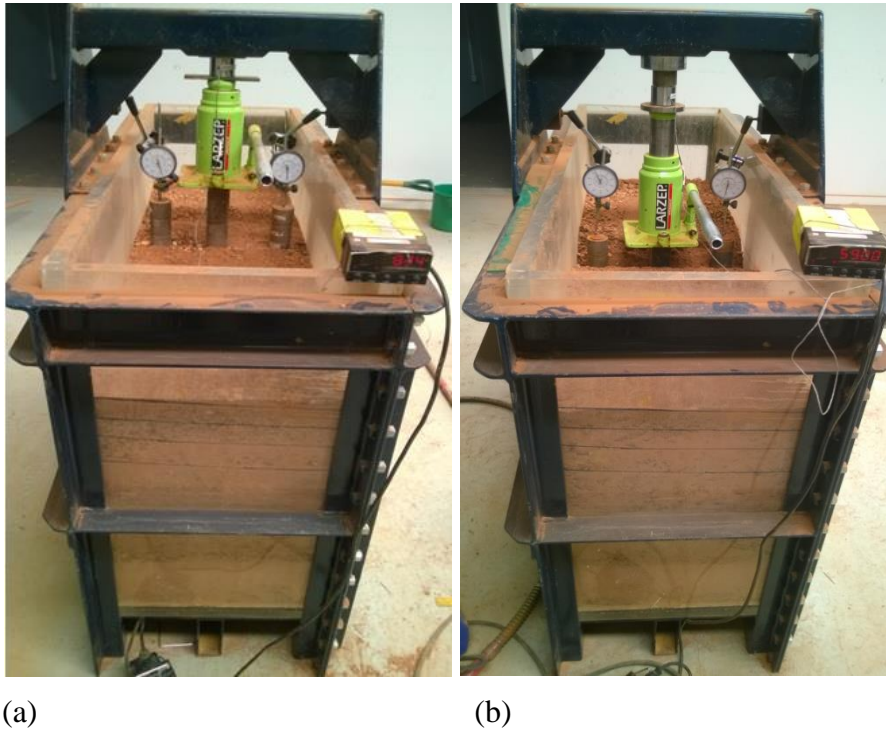


Figure 9.3: Model footing load test in the laboratory: (a) initial loading for embedded footing, (b) failure for embedded footing.

9.5 RESULTS AND DISCUSSION

The results of the load-settlement tests were plotted for all the various relative densities, $D_r = 50\%$, 70% and 90% . The parameters investigated were settlement (s) which is normalised by the width of the footing (B), load-bearing capacity and the modulus of subgrade reaction (k_s).

Figures 9.4 (a) to (c) show the variation of load-bearing pressure, q with settlement ratio (s/B) at different embedment depths $D_f/B = 0.0, 0.5, 1$ and 1.5 for relative densities $D_r = 50\%$, 70% and 90% , respectively. It may be noted that $D_f/B = 0.0$ refers to the surface footing. It is observed that as the load-bearing pressure increases, the settlement of the footing also increases for all the embedment depths, at any relative density until there is a failure. However, the load-bearing capacity increases with the embedment depth for all relative densities. The load-bearing capacity is also higher as the relative density increases from 50% to 90% for all embedment depths. This is because when the embedment depth increases or when the relative density increases there is much greater strength in the tailings bed and it is able to withstand loading more than at lower embedment depth or at lower relative density. The failure observed in the case of 70% and 90% relative densities is typically in a shear

failure mode by visible bulging as shown in Figures 9.4 (b) and (c). The failure for the 50% relative density appeared to be a local shear failure without any clear bulging, Figure 9.4 (a).

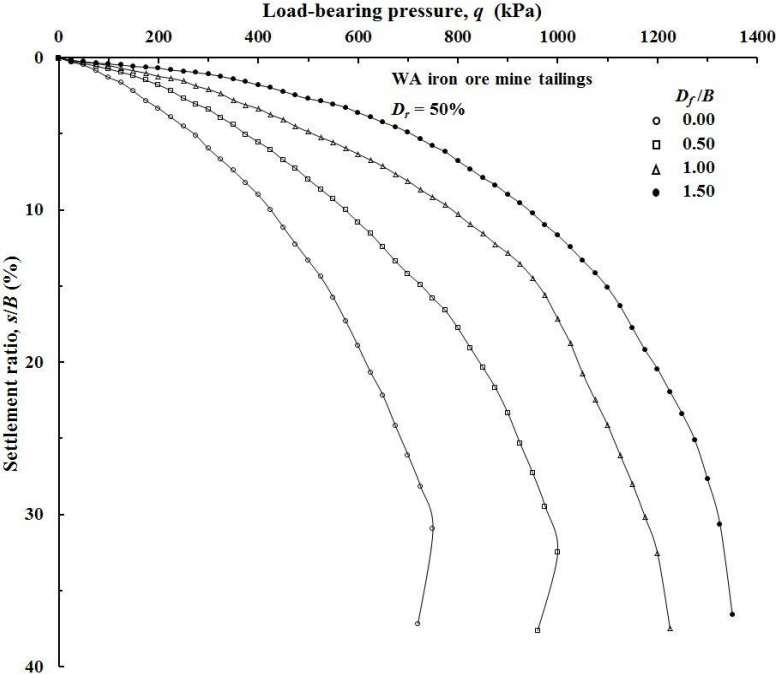


Figure 9.4 (a): Load-bearing pressure (q) versus settlement ratio (s/B) for $D_r= 50\%$.

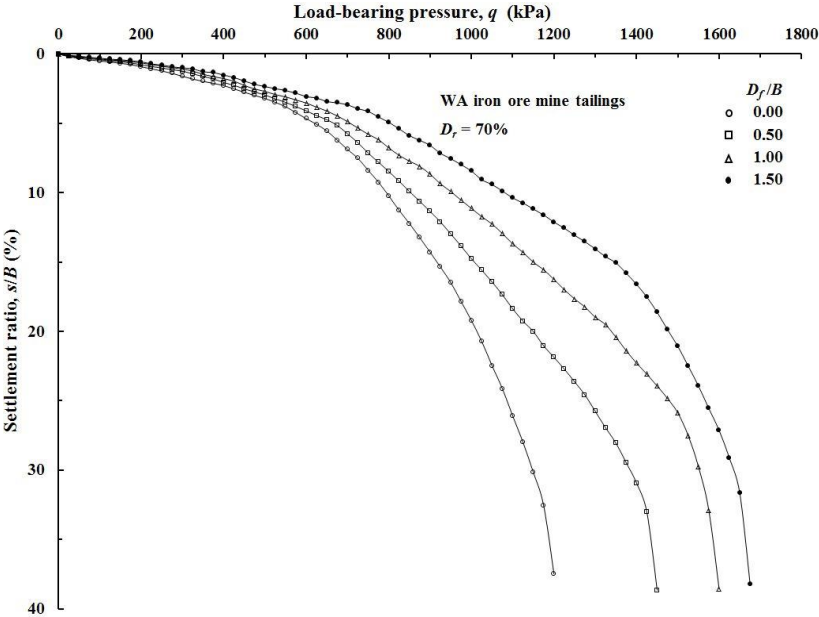


Figure 9.4 (b): Load-bearing pressure (q) versus settlement ratio (s/B) for $D_r= 70\%$.

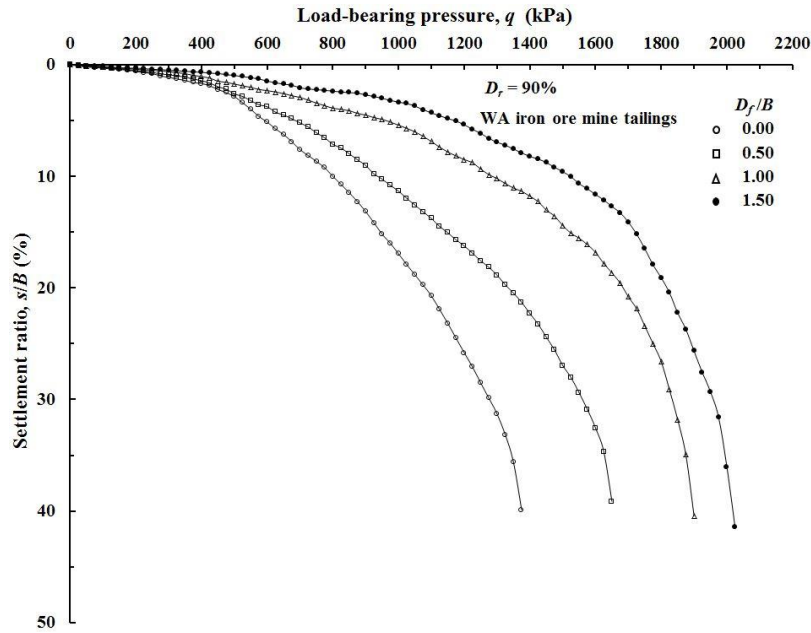


Figure 9.4 (c): Load-bearing pressure (q) versus settlement ratio (s/B) for $D_r=90\%$.

The ultimate load-bearing capacity, q_u is the highest load the material can withstand before failure and it is taken from the point corresponding to the load-bearing pressure where the curve bends and extends straight on the load-bearing pressure-settlement curve (Terzaghi et al., 1996; Shukla, 2014). Figure 9.5 shows variation of ultimate load-bearing capacity with embedment depth ratio, D_f/B for relative densities, $D_r=50\%$, 70% and 90% respectively, and it compares the results from the iron ore tailings and a similar results for Perth local sandy soil (Kazi et al., 2014).

From Figure 9.5 it is observed that as the embedment depth ratio increases, the ultimate load-bearing capacity also increases for both the iron ore mine tailings and Perth local sandy soil for all relative densities. The increase in ultimate load-bearing capacity is more significant for embedment depth ratio from 0 to 1. Embedment depth ratio greater than 1 does not yield any much increase in the ultimate load-bearing capacity for both iron ore mine tailings and Perth local sandy soil. This means that if the depth of foundation, D_f is placed at a distance deeper than the width of the foundation, B it will not be much profitable and it will not bring any significant enhancement to the ultimate load-bearing capacity. In Figure 9.5 it is also observed that the ultimate load-bearing capacity of iron ore mine tailings is much greater than that of the local soil for all embedment depths and for all relative densities. For example at $D_r = 70\%$ and an embedment depth of 1, the ultimate load-bearing capacity is 416.7% better in iron ore mine tailings than in sand. It may be noted that the differences in the specific

gravities and the chemical compositions of the two materials may be a contributory factor. Iron ore tailings appear to be heavier and it may contribute positively to the strength within the material bed to withstand heavy loading than the local sandy soil.

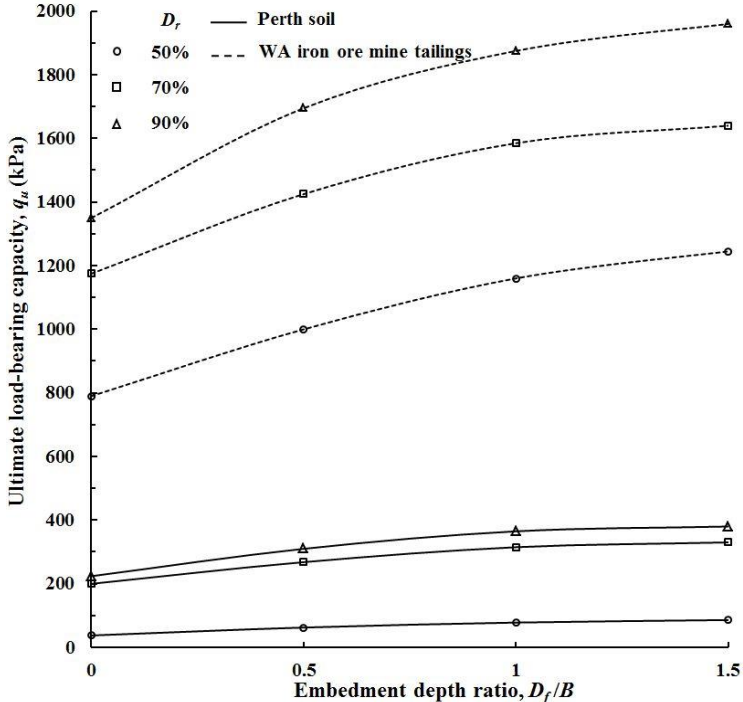


Figure 9.5: Comparison of ultimate load-bearing capacity of iron ore mine tailings and Perth local sandy soil with embedment depth ratio at different relative densities.

The modulus of subgrade reaction, (k_s) is the uniform pressure per unit settlement on the footing at 1.25 mm settlement (Kazi et al., 2014). It is a measure of the stiffness or the rigidness and the reaction of the subgrade when it is subjected to loading. Figure 9.6 shows the variation of the modulus of subgrade reaction with embedment depth ratio for both iron ore mine tailings and Perth local sandy soil at different relative densities. It is observed that in the case of the Perth local sandy soil as the embedment depth ratio increases, the modulus of subgrade reaction also increases up to an embedment depth ratio of 1, beyond which the modulus of subgrade reaction remains almost constant. For the iron ore mine tailings the modulus of subgrade reaction increases with the embedment depth ratio and even after an embedment depth ratio of 1, there is still an appreciable level of increase in the modulus of subgrade reaction for all relative densities. The increment in the level of modulus of subgrade reaction in the iron ore mine tailings is much better than in the Perth local sandy soil for all relative densities and for all embedment depth ratios. For example at a $D_f = 70\%$, the

improvement on the stiffness as a result of using iron ore tailings is as high as 1350% of 13.5 times and 933.3% or 9.33 times for embedment depth ratio of 0.5 and 1 respectively. This means that the iron ore mine tailings bed will be stiffer than the Perth local sandy soil when it is subjected to the same level of loading.

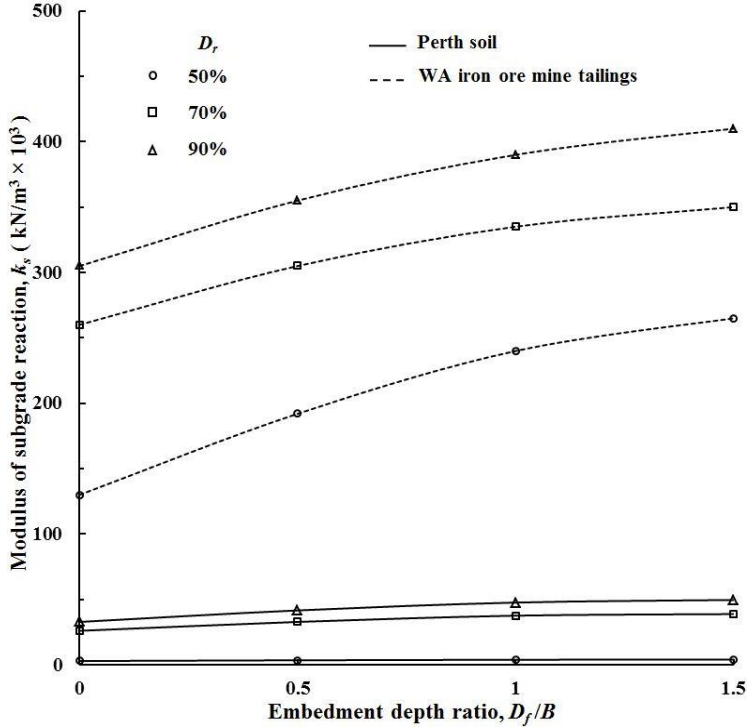


Figure 9.6: Comparison of modulus of subgrade reaction of iron ore mine tailings and Perth local soil with embedment depth ratio at different relative densities.

In distinguishing between the surface footing and the embedded footing on the improvement of the ultimate load-bearing capacity, a parameter called the bearing capacity improvement factor, (I_b) is used. This is defined as:

$$I_b = \left(\frac{q_u - q_{uo}}{q_{uo}} \right) \times 100 \tag{9.1}$$

where q_{uo} is the ultimate load-bearing capacity with respect to the surface footing. Figure 9.7 shows the variation of bearing capacity improvement factor with embedment depth ratio for iron ore mine tailings and Perth local soil at different relative densities. It is noted that as embedment depth ratio increases the bearing capacity improvement factor also increases for

all relative densities for both iron ore mine tailings and Perth local soil. The rate of increase on the bearing capacity improvement factor becomes slightly lower when the embedment depth ratio is greater than 1 for both materials. For instance, at relative density, $D_r = 50\%$, the bearing capacity improvement factor for the Perth local sandy soil is 20%, 30% and 35% for embedment depth ratios of 0.5, 1 and 1.5 respectively and that for the iron ore mine tailings is 340%, 360% and 370% for embedment depth ratios of 0.5, 1 and 1.5 respectively. For higher relative densities i.e. $D_r=70\%$ and 90% the trend is similar but with a slightly higher improvement values on the bearing capacities for higher relative densities for all embedment depth ratios. The improvement on the bearing capacity is highest for the very dense compaction, $D_r = 90\%$ for the highest embedment depth ratio $D_f/B = 1.5$, when the surface and the embedded footings are compared for both materials. It is also observed from Figure 9.7 that the improvement on the bearing capacity for a relative density from 50% to 70% is much wider than the improvement from 70% to 90% for all embedment depth ratios for both materials.

Similarly, the improvement on the bearing capacity as a result of embedment depth ratio is wider from 0.5 to 1 than from 1 to 1.5 for all relative densities for both materials. This means that as the relative density and the embedment depth ratio continues to increase, with time the improvement factor on the bearing capacity will not be significant. Therefore, for cost considerations and for convenience, the compaction, i.e. $D_r = 70\%$ and embedment depth ratio of $D_f/B = 1$, will be more appropriate and could also provide an acceptable improvement factor on the bearing capacity. It is also noted that the bearing capacity improvement factor for the iron ore mine tailings is much higher than the Perth local soil for all relative densities.

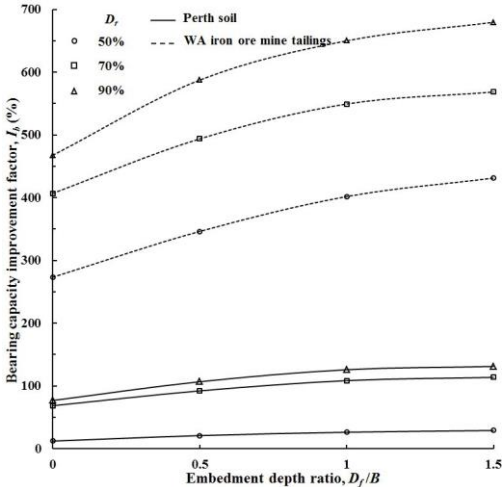


Figure 9.7: Variation of bearing capacity improvement factor with embedment depth ratio of iron ore mine tailings and Perth local soil at different relative densities.

Figures 9.8 (a) to (c) show the effects of using iron ore mine tailings instead of Perth local sandy soil on the improvement factor of the bearing capacity with embedment depth ratio for the relative densities $D_r = 50\%$, 70% and 90% respectively. It is noted that the iron ore mine tailings brings a lot of improvement on the bearing capacity as compared with using Perth local sandy soil. The improvement is much higher for embedment depth ratio of 0 to 0.5 than all other embedment ratios for all relative densities. The improvement as a result of iron ore tailings decreases as the embedment ratio increases for all relative densities. For instance, at $D_r = 50\%$, the difference in the improvement factor as a result of using the iron ore mine tailings over Perth local sandy soil is as high as 2,200% or 22 times for embedment depth ratio from 0 to 0.5. Similarly, at $D_r = 70\%$, the difference in the improvement factor as a result of using the iron ore mine tailings over Perth local sandy soil is as high as 2250% or 22.5 times for embedment depth ratio from 0 to 0.5 and it is 1300% for embedment depth ratio from 0.5 to 1. This trend is similar at higher relative densities i.e. $D_r = 90\%$. The higher specific gravity and the chemical composition of the iron ore tailings might have played a role in this bearing capacity improvement as iron materials is known to be heavier and stronger.

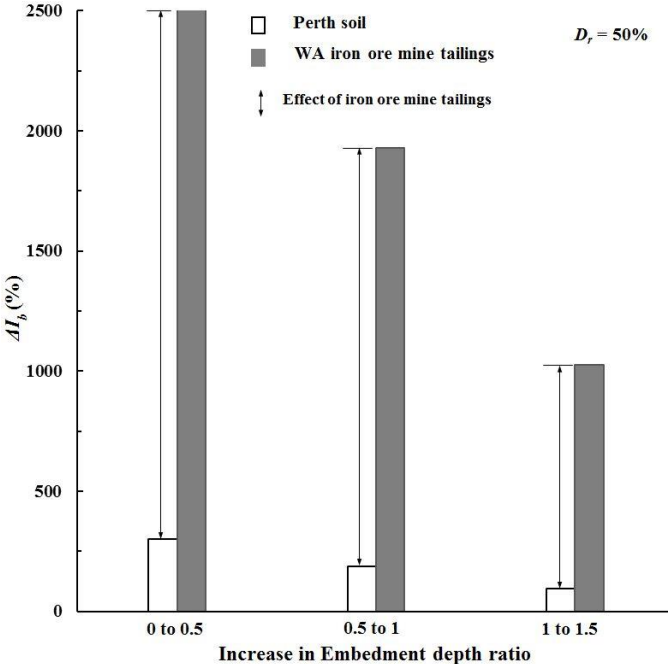


Figure 9.8 (a): Comparison of bearing capacity improvement factor of iron ore mine tailings and Perth local soil with increase in embedment depth ratio at $D_r = 50\%$.

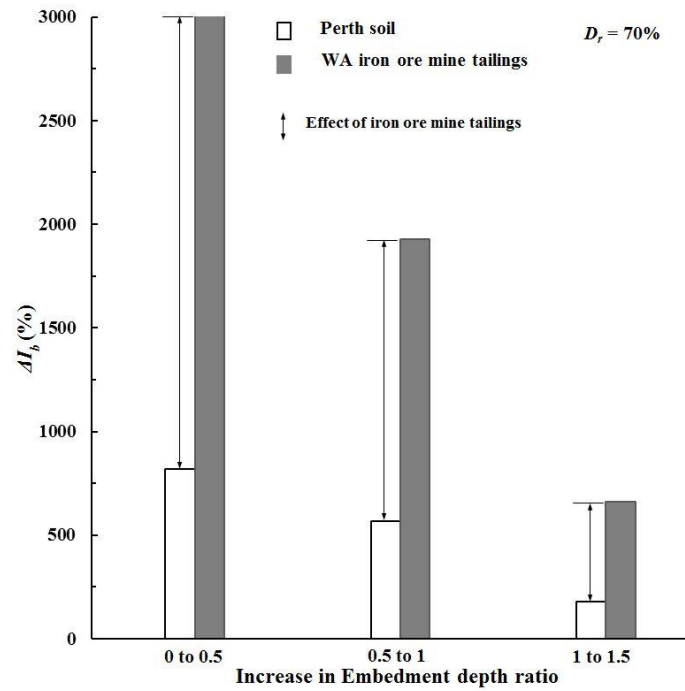


Figure 9.8 (b): Comparison of bearing capacity improvement factor of iron ore mine tailings and Perth local soil with increase in embedment depth ratio at $D_r = 70\%$.

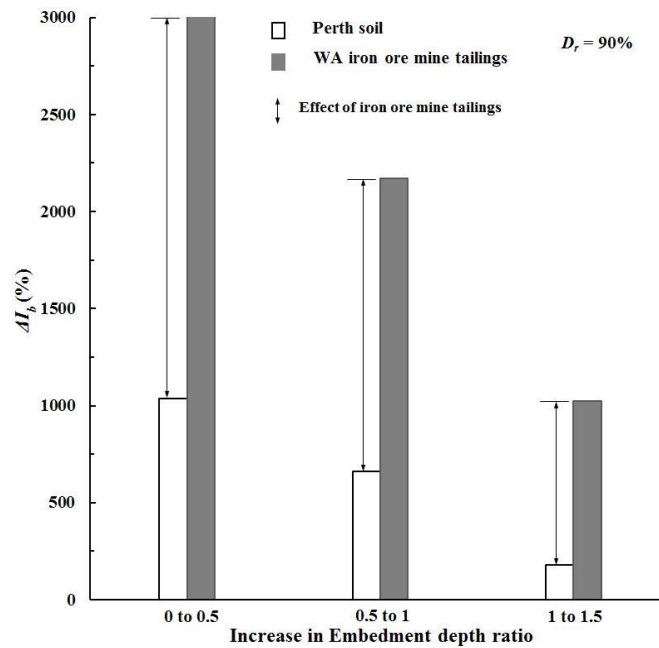


Figure 9.8 (c): Comparison of bearing capacity improvement factor of iron ore mine tailings and Perth local soil with increase in embedment depth ratio at $D_r = 90\%$.

Similarly, in distinguishing between the surface footing and the embedded footing on the improvement of the modulus of subgrade reaction, a parameter called the modulus of subgrade reaction improvement factor, (I_s) is used. This is defined as:

$$I_s = \left(\frac{k_s - k_{so}}{k_{so}} \right) \times 100 \quad (9.2)$$

where k_{so} is the modulus of subgrade reaction with respect to the surface footing. Figure 9.9 shows the variation of modulus of subgrade reaction improvement factor with embedment depth ratio for iron ore mine tailings and Perth local soil at different relative densities. It is observed that as the embedment depth increases the modulus of subgrade reaction improvement factor also increases. The level of increase from a lower embedment depth to a higher embedment depth is not very much in the case of Perth local soil but it is very significant for iron ore mine tailings for all relative densities. For instance, at $D_f/B > 1$ the level of increase in the subgrade reaction improvement factor is very low for the Perth local soil while it is still significant in the case of iron ore mine tailings for all relative densities. It is also observed that the level of increase on the subgrade reaction improvement factor from relative density of 50% to 70% is wider than from 70% to 90% for both materials. The improvement factor on the modulus of subgrade reaction is highest at a very dense compaction, i.e. $D_r = 90\%$.

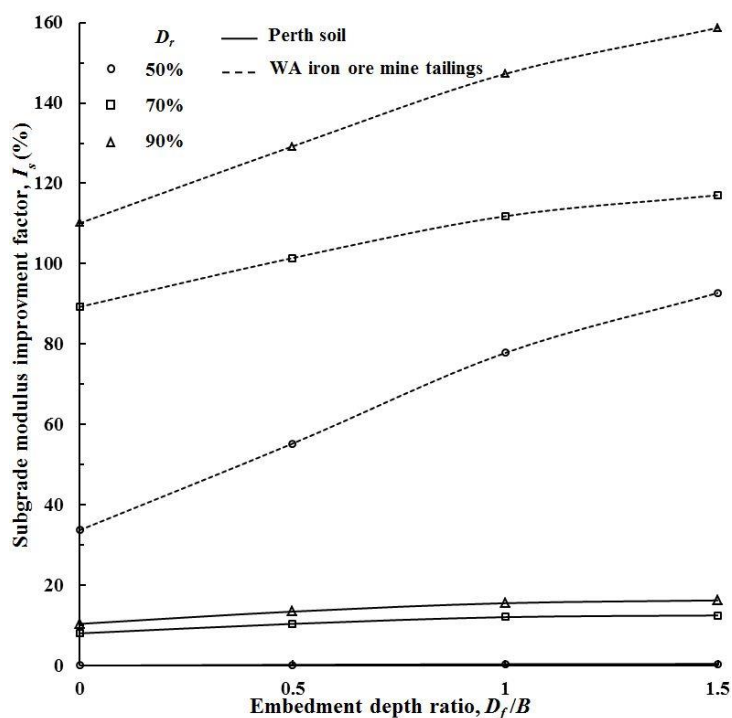


Figure 9.9: Variation of modulus of subgrade reaction improvement factor with embedment depth ratio of iron ore mine tailings and Perth local soil at different relative densities.

Figures 9.10 (a) to (c) show the effects of using iron ore mine tailings instead of Perth local soil on the improvement factor of the subgrade reaction with embedment depth ratio for the relative densities, $D_r = 50\%$, 70% and 90% respectively. Similar to the case of bearing capacity improvement factor, the iron ore mine tailings bring a lot of improvement on the modulus of subgrade reaction over the Perth local soil. For instance, At $D_r = 50\%$, the improvement factor on the modulus of subgrade reaction of iron ore tailings over Perth local soil is 740%, 552% and 255% for increase in embedment depth ratio of 0 to 0.5, 0.5 to 1 and from 1 to 1.5 respectively. For $D_r = 70\%$, it is 470%, 300% and 140% for increase in embedment depth ratio of 0 to 0.5, 0.5 to 1 and from 1 to 1.5 respectively. Lastly, for $D_r = 90\%$, it is 500%, 360% and 220% for increase in embedment depth ratio of 0 to 0.5, 0.5 to 1 and from 1 to 1.5 respectively. Therefore, the improvement factor on the modulus of subgrade reaction is higher for using iron ore mine tailings than the Perth local soil especially for embedment depth ratio of 0 to 0.5. Similarly, this could be attributed to the higher specific gravity and the chemical composition of the iron ore mine tailings, with higher iron contents known to be heavier and stronger that can further enhance the improvement behaviour of the modulus of subgrade reaction.

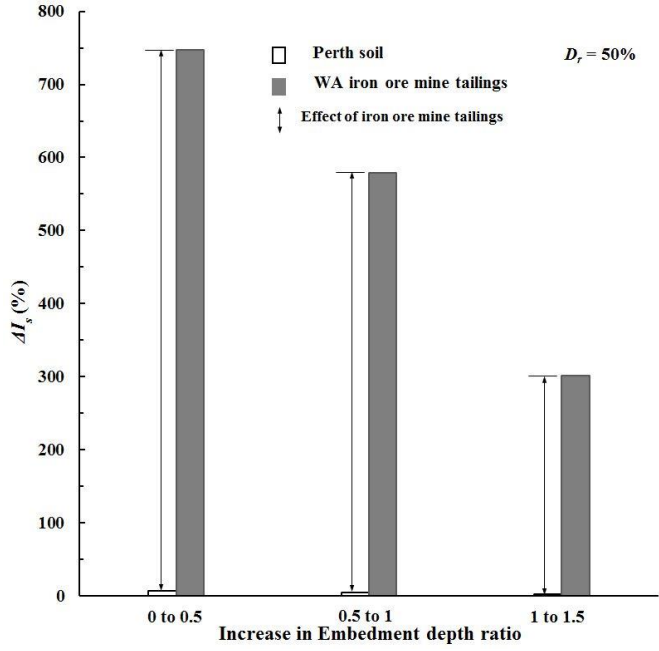


Figure 9.10 (a): Comparison of modulus of subgrade reaction improvement factor of iron ore mine tailings and Perth local soil with increase in embedment depth ratio at $D_r = 50\%$.

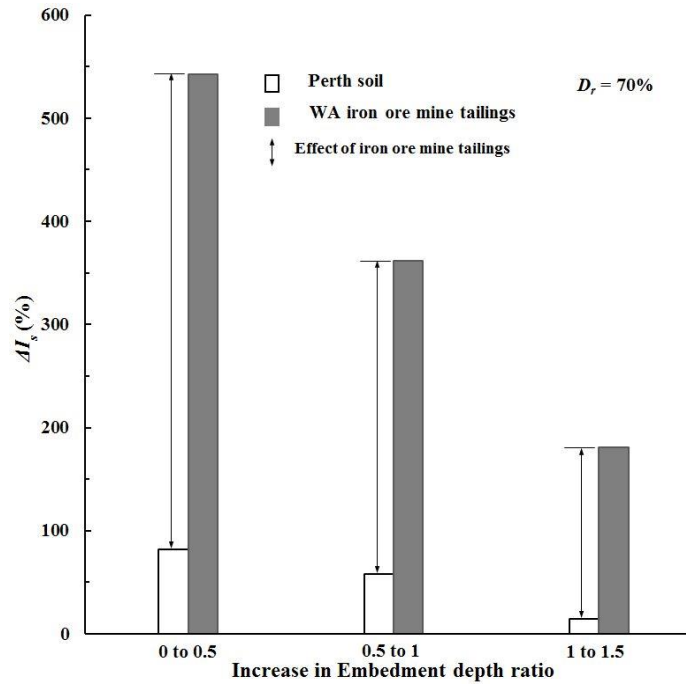


Figure 9.10 (b): Comparison of modulus of subgrade reaction improvement factor of iron ore mine tailings and Perth local soil with increase in embedment depth ratio at $D_r = 70\%$.

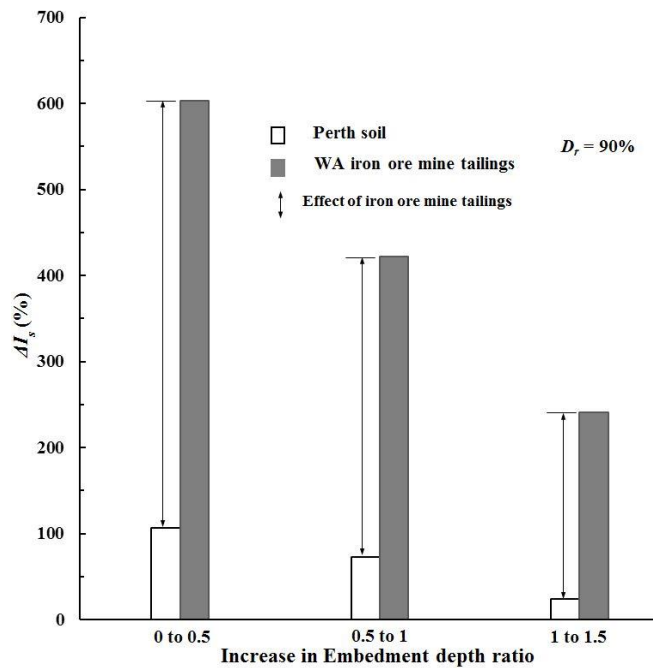


Figure 9.10 (c): Comparison of modulus of subgrade reaction improvement factor of iron ore mine tailings and Perth local soil with increase in embedment depth ratio at $D_r = 90\%$.

9.6 CONCLUSIONS

In this study, iron ore mine tailings from Western Australia (WA) was used as a structural fill material in a laboratory scale investigation using a model test tank. The aim was to replace

conventional earthfilling materials such as the Perth local soil with iron ore mine tailings. The load-bearing pressure with its corresponding settlements for the tailings bed at variable relative densities for both surface and embedded footings were investigated. The results for the iron ore mine tailings were compared with the results of a similar study using Perth local soil. Based on the results from the study, the following general conclusions could be drawn:

1. The load bearing capacity increases with an increase in embedment depth and also increases with increase in the relative density.
2. The ultimate load-bearing capacity increases with an increase in embedment depth for all relative densities. The increase in the ultimate load-bearing capacity is more significant at embedment depth ratio from 0 to 1, beyond this; the increase in the ultimate load-bearing capacity is not significant.
3. The stiffness in terms of the modulus of subgrade reaction, of the tailings bed is stronger and it improves more as the embedment depth of the footing increases for all relative densities.
4. The bearing capacity and the modulus of subgrade reaction improvement factors are higher in very dense compaction, $D_r = 90\%$, than other relative densities but the improvement factors for other relative densities are still high and it increases at higher relative densities.
5. For cost considerations and for convenience, compaction of $D_r = 70\%$ and embedment depth ratio of $D_f/B = 1$ could be chosen because there is very little improvement when these parameters are exceeded. Therefore, this compaction will compromise between cost, convenience and strength of the fill.
6. The use of iron ore mine tailings as a structural fill material in place of Perth local sandy soil brings superior improvement in terms of the load-bearing capacity and stiffness. The improvement on the stiffness as a result of using iron ore tailings is as high as 1350% or 13.5 times and that on the load-bearing capacity is 2200% or 22 times for $D_r = 70\%$ for embedment depth ratio of 0.5. This trend is similar if the relative density or the embedment depth ratio is increased further.
7. The iron ore mine tailings will make it cost effective in constructions requiring structural fills and leads to significant reduction in greenhouse gas emissions for environmental sustainability.

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CHAPTER 10

SUMMARY AND CONCLUSIONS

This chapter explains briefly the problem being addressed and outlines the methodology that was used. The overall findings have also being presented in this chapter. It goes further to point out the main novel findings this research has produced. Finally, it makes some recommendations for future research trajectories based on the experience from this research.

10.1 SUMMARY

The generation and handling of mine wastes by mining firms is known to be a global problem. The impact is much felt by all stakeholders in the mining jurisdictions. The problem is seen to be in two folds. First, it is a major problem to mining firms both economically and legislatively.

In economic wise, the mining firms have to dispose off their wastes at a cost without any revenue. The firms need to secure a convenient land preferably closer to the mine site to reduce transportation cost. If this land is exhausted, they need to continue to search for a more convenient one for waste disposal. These associated costs deplete off a substantial part of the firm's profit.

Legislatively, the firms need to meet some stringent regulatory requirements in their mine wastes disposal which is normally enforced by many regulatory and governmental bodies. Non conformance to these regulations and legislations bring negative consequences to the mining firms.

Secondly the mine wastes are key contributors to the numerous environmental problems. The mine waste dumps could oxidise and become acidic due to their contacts with potential substances in the environment. The acidic substances could be washed into water bodies when it rains and could cause harm to aquatic as well as human life. This is known as acid mine drainage (AMD). The mine wastes could also release some dust and other dangerous substances which could cause air pollution. There are many environmental problems caused by these mine wastes but common ones are elaborated above.

Comprehensive utilisation of mine wastes, especially in building and construction applications are seen to have the greatest potential in finding a sustainable solution to these problems. This is because; it will ensure high volume utilisation of mine wastes to

significantly reduce the quantity of mine wastes in the environment or possibly to eliminate them completely.

Many researchers have attempted to solve this problem and some have come out with very interesting ideas to deal with the problem. Unfortunately, these previous research works have some short falls in the sense that most of their works focus on partial or very small percentage utilisation of the wastes. Their practises normally follow conventional methods which does not any add significant value to the wastes and to impact positively to the environment. Looking at the numerous potentials these mine wastes could offer, it could be argued that the potentials have not been explored extensively. Also the previous research on this is undoubtedly not enough and in particular, it is almost non-existent in Western Australia with high mineral resources.

In an attempt to find sustainable and reliable solutions to the problems, this research first analysed the literature comprehensively; to find and proposed a means for reuse possibilities for these mine wastes, such as production of bricks and construction of embankments. Again, the research sought to develop a laboratory friendly and a simple methodology to characterise these mine wastes to be applied in construction works. The research went further to select some potential applications of the mine wastes where high volume utilisation of the mine wastes could be realised to add value to it. In doing this, after the characterisation, the research conducted practical laboratory investigations for utilising the mine wastes for geopolymer bricks production, concrete making and structural fills. The selected applications have the potential of ensuring high volume utilisation of the mine wastes because the engineering properties of the mine wastes are seen as comparable to many conventional construction materials.

10.2 CONCLUSIONS

Based on the current study, the following general conclusions are made from each of the individual applications and from the analysis of the literature.

10.2.1 Bricks production using mine wastes based on literature analysis

1. Mine tailings can have several civil engineering applications. Studies show that they can be a good source of raw materials for brick manufacturing with some suitable modifications such as by addition of soil, lime or cement.

2. The innovation will add value to brick manufacturing through cost reduction by getting cheap mine tailings as raw materials and thus reduce the need to excavate new soil.
3. This will further reduce CO₂ emissions, save land use for mine waste disposal and provide other environmental and economic benefits.
4. The mining details presented for Western Australia indicate that there is a need to carry out analysis of the mine wastes so that they can be used effectively in large quantities, especially in brick manufacturing.

10.2.2 Embankment construction using mine wastes based on literature analysis

1. The sustainable use of mine wastes for highway and railway embankments is an attractive option for utilisation of mine wastes in large volumes.
2. This practice will reduce the need for excavating large quantities of natural soils and rocks for embankment construction, thus reducing carbon dioxide emissions, saving valuable land for mine waste disposal, and achieving other economic and environmental benefits.
3. The details of mine wastes presented for Western Australia indicate that there is a need to carry out analysis of the mine wastes so that they can be used effectively in significant quantities, especially as highway and railway embankment construction material in civil engineering projects.

10.2.3 Development of a methodology through electrical resistivity survey to characterise geo-materials

1. The value of electrical resistivity determined by using the small-scale laboratory apparatus depends greatly on the electrode depth and the spacing of electrodes. The results show that an increase in electrode depth causes a decrease in the resistivity while the increase in the electrode spacing results in an increase in the resistivity.
2. Equation (8), which is commonly used in the determination of electrical resistivity of geomaterials by the Wenner electrodes array, is based on several assumptions. These assumptions are difficult to achieve when conducting laboratory measurements. Equation (12) is presented as an improvement over equation (8), by incorporating the resistivity correction factor λ , which is found to be approximately 0.46 for the experimental setup used in this work. Being

independent of types of soil, this value of λ can also be applicable to other types of soil, provided that the experimental setup of the same geometrical configuration is used.

3. If someone uses an experimental apparatus similar to the one developed in this study; it is essential that equation (12) rather than equation (8) should be utilized for calculating more realistic values of electrical resistivity by substituting the experimental current and voltage. In conjunction with the improved equation (12), the laboratory setup developed to determine the electrical resistivity under laboratory conditions, can be a cost-effective alternative to any commercial resistivity measuring apparatus.
4. The electrical resistivity of dry Perth sand was found to range from 60,606 Ωm to 142,857 Ωm for very dense to very loose conditions, respectively. This range is in close agreement to the resistivity values of sand reported in the literature for other places. It was also noted that the resistivity of sand decreases with an increase in its relative density.
5. Figure 19 can be used as a chart for determining the electrical resistivity of dry sand, which is similar to the dry Perth sand, by measuring the relative density of the sand only, and vice versa. This may help field engineers in characterizing the behavior of sand, especially when conducting a preliminary analysis, and when designing geotechnical and geo-electrical systems.

10.2.4 Characterisation of iron ore mine tailings using electrical resistivity survey

1. The electrical resistivity of the iron ore mine tailings, both in dry and fully saturated conditions, depends on the relative density or the extent of compaction. The resistivity is in inverse relationship with the relative density. The resistivity decreases with an increase in relative density and vice versa.
2. The electrical resistivity of iron ore mine tailings produced in Western Australia in dry condition is found to range from 11 k Ωm in a more dense state to 19 k Ωm in a very loose state while that in fully saturated condition it ranges from 20 Ωm for a very dense state to 31 Ωm in a very loose state.
3. The resistivity of iron ore mine tailings both in dry and fully saturated conditions has lower values compared with resistivity values for sand. This occurs mainly because the iron ore mine tailings have very high iron content compared to that of sand.

4. Figures 22 and 23 can be used as the engineering charts for determining the electrical resistivity of iron ore mine tailings by knowing its relative density. This may guide the field engineers to economically identify the behavior of the iron ore mine tailings when used in engineering projects such as the following:
 - Structural fills in railway or highway engineering projects, and in rehabilitation of mined out areas
 - Determining the extent of corrosion for buried steel pipe lines for modification and replacement
 - Studies on weak zones in embankments

10.2.5 Utilisation of iron ore mine tailings for the production of geopolymer bricks

1. The strength of the geopolymer bricks made from iron ore tailings with sodium silicate solution, is influenced greatly by the curing temperature. The UCS increases as the curing temperature increases to a certain optimum point (80 °C), and then the UCS starts decreasing as the temperature increases further.
2. The optimum base parameters for the production of the geopolymer bricks are sodium silicate solution content of 31%, initial setting time of 15 minutes, and curing temperature of 80°C. In addition the curing time of one day for the brick results in an optimum UCS of 19.18 MPa, three days curing results in a UCS of 34.00 MPa, and seven days curing provides an optimum UCS of 50.35 MPa. When subjected to further curing above seven days, the UCS starts to decline.
3. The electrical resistivity of geopolymer bricks is lower than the commercial clay bricks due to the higher iron content associated with the iron ore mine tailings. However, the electrical resistivity of the geopolymer bricks is still high enough to be used for building construction. The electrical resistivity increases from a value of 456 kΩm for one day curing, to > 682 kΩm for 7 days curing.
4. The XRD pattern reveals that there is a decrease in the intensity (count) of the crystalline peaks in the final tailings geopolymer bricks. The primary cause of the decrease being the partial dissolution of alumina silicate minerals, producing a polymeric gel that reacts with the undissolved alumina silicate minerals, resulting in strong bonds. These strong bonds enhance strength development in the final geopolymer bricks.
5. The tailings geopolymer bricks met both the ASTM and the Australian Standards requirements for the specification of bricks. Even when the bricks were cured for

seven days, the resulting bricks had superior qualities than both the ASTM and the Australian Standards requirements. This has provided an avenue for cheaper and alternative materials for civil engineering constructions.

6. The finished bricks are more economical than the commercial clay bricks, with a cost reduction of either 36.8% or 20.5%, when one day or three day curing times, respectively, are selected. If the curing for seven days is selected, the bricks produced exceed the cost of commercial clay bricks by 12.0%, but they have superior qualities over commercial clay bricks.
7. The new bricks have environmental benefits over the commercial bricks, because of a reduction in energy consumption. The reduction is achieved by adopting simple, low temperature oven drying, as opposed to high temperature kiln firing in the production of commercial clay bricks. The quantity of mine tailings returned to the environment will be reduced; and the need to mine fresh clay materials for the production of bricks will be avoided, by substituting with mine tailings. This will lead to a reduction of greenhouse gases and a reduced ecological footprint.
8. With this new technology, there is the possibility of providing ongoing and guaranteed employment for the inhabitants of the mining communities, by eliminating the shocks and economic stand-still that the mining communities encounter upon mine closure.

10.2.6 Utilisation of iron ore mine tailings as aggregates in concrete

1. The compressive strength of the concrete with iron ore tailings aggregates at 28 days was 36.95 MPa which shows an improvement of 11.56% over the concrete with conventional aggregates. This is mainly because of favourable chemical composition of the iron ore tailings.
2. The split tensile strength exhibited by the concrete with tailings aggregates was 2.82 MPa at 28 days and this is slightly lower than concrete with conventional aggregates by 16% due to higher quantity of fines in the iron ore tailings as compared with the natural sand in the control mix. However, the tensile strength increased favourably with aging and there was still 4.8% improvement on the tensile strength as compared with similar study reported earlier.
3. The concrete with tailings aggregates has a low potential of corrosion and a low potential to acid attack due to high pH values of their resulting solution.
4. The utilisation of iron ore tailings as aggregates in concrete could have positive environmental implications to the mining companies and the mining communities and

will provide cheaper alternative materials to bring about economy in concrete production.

10.2.7 Utilisation of iron ore mine tailings as structural fills

1. The load bearing capacity increases with an increase in embedment depth and also increases with increase in the relative density.
2. The ultimate load-bearing capacity increases with an increase in embedment depth for all relative densities. The increase in the ultimate load-bearing capacity is more significant at embedment depth ratio from 0 to 1, beyond this; the increase in the ultimate load-bearing capacity is not significant.
3. The stiffness in terms of the modulus of subgrade reaction, of the tailings bed is stronger and it improves more as the embedment depth of the footing increases for all relative densities.
4. The bearing capacity and the modulus of subgrade reaction improvement factors are higher in very dense compaction, $D_r = 90\%$, than other relative densities but the improvement factors for other relative densities are still high and it increases at higher relative densities.
5. For cost considerations and for convenience, compaction of $D_r = 70\%$ and embedment depth ratio of $D_f/B = 1$ could be chosen because there is very little improvement when these parameters are exceeded. Therefore, this compaction will compromise between cost, convenience and strength of the fill.
6. The use of iron ore mine tailings as a structural fill material in place of Perth local sandy soil brings superior improvement in terms of the load-bearing capacity and stiffness. The improvement on the stiffness as a result of using iron ore tailings is as high as 1350% or 13.5 times and that on the load-bearing capacity is 2200% or 22 times for $D_r = 70\%$ for embedment depth ratio of 0.5. This trend is similar if the relative density or the embedment depth ratio is increased further.
7. The iron ore mine tailings will make it cost effective in constructions requiring structural fills and leads to significant reduction in greenhouse gas emissions for environmental sustainability.

10.3 CONTRIBUTIONS TO KNOWLEDGE

This research has added value to iron ore mine wastes by providing sustainable options of utilisation for economy in building and construction especially for brick making, concrete

production and in fill applications. Western Australian miners have got the option of looking into a different dimension of economic activity after mine closure or during the down turn of mining activities. The research has also developed a laboratory friendly methodology of characterising all geo-materials and in particular, iron ore mine wastes for engineering applications.

10.4 FUTURE RESEARCH TRAJECTORIES IDENTIFIED BASED ON THIS RESEARCH

- Further research to identify the practicalities and the mode of commercialisation to these research findings
- Further research into different alkali activators for the production of geopolymer bricks and comparison based on cost effectiveness
- Further research into the engineering and the geotechnical properties of other different types of mine wastes
- Further research into several different types of mine wastes for applications in building and construction

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