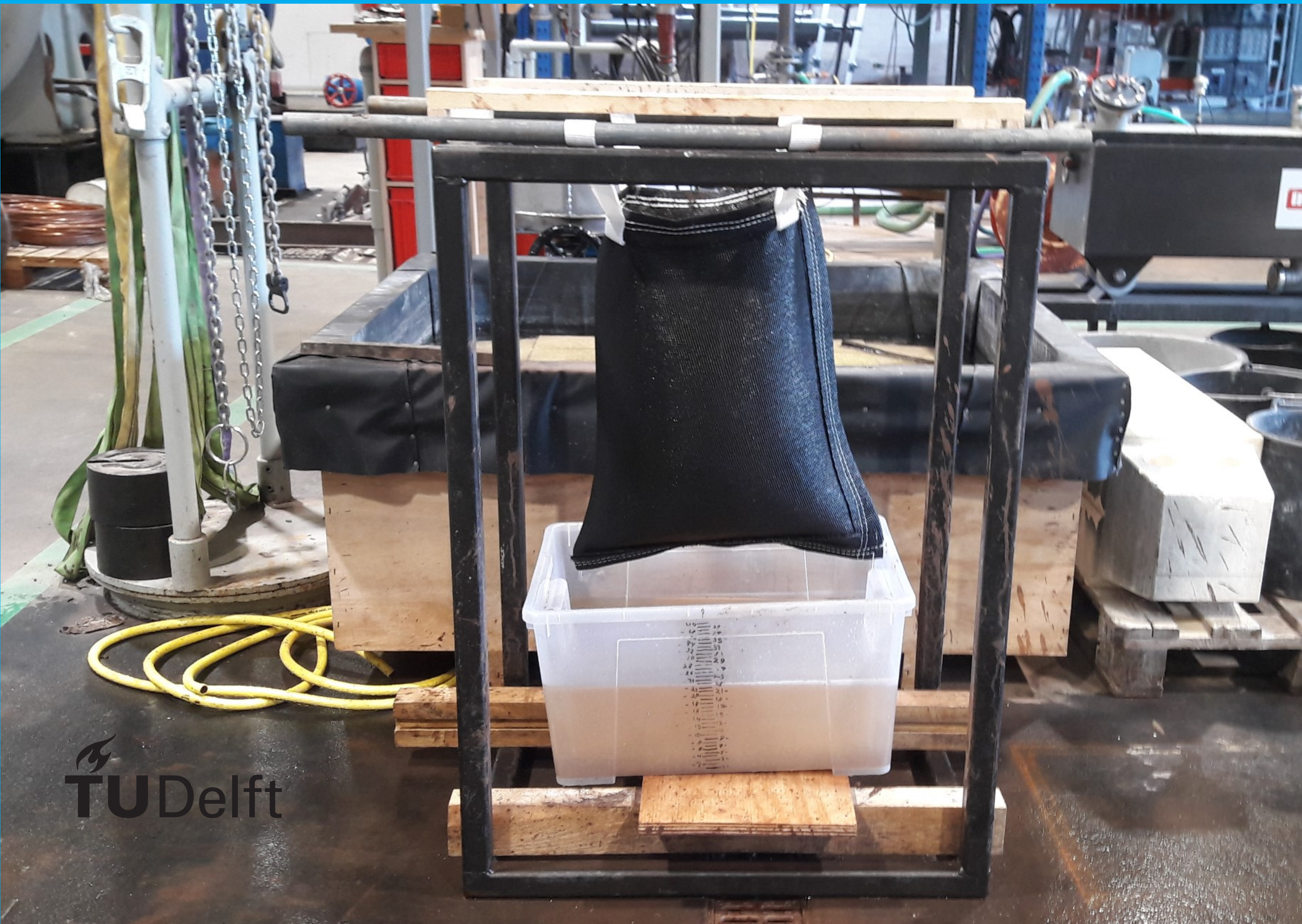
Geotextile tubes filled with mine tailings as a construction element

P. Bosman



Geotextile tubes filled with mine tailings as a construction element

by

P. Bosman

to obtain the degree of Master of Science
at the Delft University of Technology,
to be defended publicly on Friday November 22, 2019 at 2:00 PM.

Student number:	4319729	
Project duration:	February 18, 2019 – November 22, 2019	
Thesis committee:	Prof. dr. ir. M. Buxton,	TU Delft, supervisor
	Ir. M. Keersemaker,	TU Delft
	Dr. M. Bakker,	TU Delft
	Prof. dr. B. Lottermoser,	RWTH Aachen
	PhD. W. Solowski,	Aalto University

Abstract

Large quantities of mine tailings are stored in mine tailings storage facilities. These facilities are only costing money for mining companies. In addition, these tailings storage facilities are prone to failure. To prevent this from happening these facilities have to be reinforced. This research introduces a concept to use geotextile tubes filled with mine tailings as reinforcing construction elements. That way value is assigned to the mine tailings while also making the storage facilities safer. However, pumping mine tailings in geotextile tubes is not an option due to the small particle sizes which will flow through the openings in the geotextile fabric. In addition, the contaminations in the mine tailings will also be mobilised using this method. Therefore, this research investigates potential enhancement of mine tailings to make them suitable to be pumped in geotextile tubes. The potential enhancement is in the form of flocculation and remediation. While flocculation increases the dewatering properties of mine tailings, the remediation will prevent the contaminations from mobilising. Test performed in this research include a series of jar tests, cone tests, two different strength tests and hanging bag tests.

The results show that when flocculants are added to the tailings, the dewatering rate increases. The flocculant type which proved the most effective for these specific tailings type is a flocculant with an average molecular weight and average anionic value. Adding coagulant to the tailings further increases the dewatering rate of the tailings. When Ordinary Portland Cement is added, to stabilise and solidify the tailings, the dewatering rate increases even more. In addition, adding Ordinary Portland Cement to the tailings resulted in an increase of shear strength and unconfined compressive strength. After seven days of curing, the highest strength of curing was measured for the mixtures where fifteen mass percentage Ordinary Portland Cement was added. The mixture where the highest strength and the fastest dewatering rate was measured, was the mixture composes of mine tailings, flocculants, coagulants and fifteen mass percentage Ordinary Portland Cement.

With these results, it may be concluded that mine tailings can be used to fill geotextile tubes to form reinforcing construction elements if its dewatering behaviour is enhanced and its contaminants remediated.

Acknowledgements

First of all, I would like to thank everyone who participated in this project. A special thanks to Andrina Drost from IHC and Hugo Ekkelenkamp from NETICS to give me the opportunity to conduct my thesis on this very interesting subject. I want to thank everyone from IHC Mining & Tunneling and NETICS to make me feel part of their company and their team. A big thanks goes to my supervisor from Delft, namely Dr. Mike Buxton for his time and guidance throughout our discussion sessions. My thanks to my family and friends, who always supported me and allowed me to reach my goals. I would also like to thank Vale for giving me the samples to perform the experiments with, Kemira for supplying me with flocculants, Clariant for giving their coagulant and TenCate Geosynthetics for their geotextile fabric. And lastly, I would like to thank everyone from the European Mining Course including staff for giving me the possibility to study the thing I love. To my fellow EMC students I would like to say ’Glück Auf!’ and wish you the best in the near future and that we may see each other again.

Pieter Bosman
Delft, November 2019

Table of contents

List of Figures

List of Tables

Acronyms

Acronym	Definition
% Strength Ret.	% Strength Retention
°C	Degree Celsius
µm	micrometre
AMD	Acid mine drainage
BA	Bottom Ash
CAC	Calcium Aluminate Cement
CEM	Cement
coag	coagulant
do.	ditto
e.g.	exempli gratia
FA	Fly Ash
floc	flocculant
g	gram
g/m2	gram per metre squared
ha	hectare
hrs	hours
ISL	In Situ Leaching
kg/m^3	kilogram per cubic metre
kN/m	kilonewton per metre
kPa	kilopascal
L	litre
L/min/m2	litre per minute per metre squared
m	metre
m^3 /s	cubic metres per second
MFT	Mature Fine Tailings
min	minutes
ml	milliliter
mm	millimetre
Mt	megatonne
OPC	Ordinary Portland Cement
PC	Pozzolanic Cement
ppm	part per million
RHA	Rice Husk Ash
rpm	rounds per minute
s	second
S/S	Stabilisation & Solidification
SRC	Sulphate Resistant Portland Cement
t	tonne
Ts%	Solid mass divided by total mass in percentage
V	Volume
Wc%	Water content
wt%	Weight percent
XRD	X-ray powder diffraction
µg/L	microgram per liter

Part I

Introduction

1

Introduction

1.1. Motivation

In the last 25 years, there have been a couple of serious accidents with mine tailings storage facilities. In 1994, a tailings dam failed of the Harmony Gold mine. This tailings dam was situated 320 metre up-slope of Merriespruit, a suburb of Virginia in the Free State Goldfields of South Africa. More than 2.5 million tonnes of liquefied mine tailings moved down the slope, killing seventeen people [Van Niekerk and Viljoen, 2005]. In the next year, in 1995, a major environmental disaster occurred in Guyana when a tailings dam failed, spilling more than 1.5 million cubic metres of cyanide contaminated material into the Omai River [Ramraj, 2001]. Three years, in 1998, later disaster struck Spain when a tailings dam at the Los Frailes mine in Andalucia failed, releasing between 5 to 7 million cubic metres of acid tailings, pH of 2, into the river Guadiamar. As a result, an estimated 26 tonnes of fish died and over 4000-5000 ha of the riverbed and flood plains were covered in acid sludge of 1.5 m thick [Achterberg et al., 1999]. The following year, in 1999, a tailings dam failed at mine operated by Aural SA Company in Baia Mare, northwest Romania, releasing more than 100 thousand cubic metres cyanide containing tailings into the rivers Sasar, Lapus, Somes, Tisza and Danube. As a results, the water supply of 24 municipalities was interrupted and over 1240 tonnes of fish died [Baja Mare Task Force, 2000]. Even more recent, in the beginning of the year 2019, there was a big tailings dam failure in Brazil, resulting in 248 casualties and 22 missing individuals [British Broadcasting Corporation, 2019] and a big environmental impact on the lower regions of the flood area. The failures mentioned above are only five tailings dam failures of a total of 90 reported tailings dam failure related incidents in the last 25 years [WISE Uranium Project, 2019]. It is without a doubt that mine tailings dam failures prove very dangerous. Together with the large environmental impact tailings storage facilities have in general, with big water consumption, large areas used for storage and the acid mine drainage phenomenon [Kossoff et al., 2014], there is an increasing pressure on mining companies to reduce the amount of tailings storage facilities and to reduce the environmental impact these facilities have on the surroundings.

Nowadays, there are many different methods that can be used to reduce the volume of mine tailings produced, like dry-stacking, and to reduce the impact the mine tailings have on the surroundings, like acid mine drainage prevention [Kefeni et al., 2017]. One of these methods is to re-use mine tailings as a potential resource for construction material. This method can reduce the volume of mine tailings that have to be stored inside storage facilities and it can result in a potential revenue for the mining companies. However, the biggest problem of mine tailings in general is that mine tailings dewater very slow and that the geo-mechanical properties of the resulting dry material does not meet the requirements that are needed to be used in construction work. Therefore, the mine tailings have to be treated to make it suitable as a resource for construction purposes. With this in mind, this research focuses on the potential stabilisation of mine tailings in order to make it a suitable resource for construction purposes while retaining any contaminants initially present in the mine tailings.

1.2. Previous research

A lot of research has been done on the topic of re-using of mine tailings as a potential resource for construction material. Many of them are about using the mine tailings as an aggregate for cement production [Argane et al., 2015, Huang et al., 2013, Onuaguluchi and Eren, 2016, Ranade and Li, 2013, Zhao et al., 2014].This proved effective without any strength loss. In [Chen et al., 2011] it was researched whether it was possible to produce bricks from iron ore tailings. The re-use of mine tailings as road base was investigated in [Mahmood and Mulligan, 2007]. It proved even possible to make pigment for paint from iron ore tailings [Galvão et al., 2018]. From the examples above it becomes clear that mine tailings can be re-used in various ways, all with their own advantages and disadvantages. Although these researches focus in re-using the mine tailings, they do not tackle the problem of the unstable tailings dams.

Some research was done on reinforcing dams with soil bags [Mohri et al., 2009] but these bags were filled with imported material imported. [Howard and Trainer-tencate, 2011] investigated the construction of temporary walls using geotextile tubes. In that research, the material used to fill the tubes was locally dredged sludge, but it was not mine tailings. [Zheng et al., 2019] investigated whether mine tailings could be reinforced with basalt fiber to increase the stability of tailings dams but this research only investigated the concept and not the practical application.

It can be seen that multiple research papers have been written on either the re-use of mine tailing or the reinforcement of tailings dams. But none of researches mentioned above combine the two: re-using the mine tailings to reinforce the tailings dams. That is the gap in knowledge that this research will try to fill.

1.3. Aim and objectives

With the gap defined, it is now possible to specify the aim, the objectives and the corresponding research questions, all given below.

The aim:

> This research aims to identify mine tailings as a sustainable resource for construction with geotextile tubes by assessing the characteristics of mine tailings and geotextile tubes.

The hypothesis:

> It is hypothesised that stabilised mine tailings can be used as a sustainable resource for construction with geotextile tubes.

To test this hypothesis, the following objectives are defined:

1. Identify the worldwide occurrence of mine tailings with respect to origin, quantity, composition and potential hazard by means of literature study.
2. Characterise one type of mine tailings with respect to chemical composition and geomechanical properties and investigate the effect of potential enhancement of the mine tailings.
3. Investigate whether mine tailings are a suitable resource for construction with geotextile tubes.
4. Extrapolate the results from the samples to a wider range of mine tailings.

The relation between the aim of this research and its corresponding objectives is given in Figure 1.1.

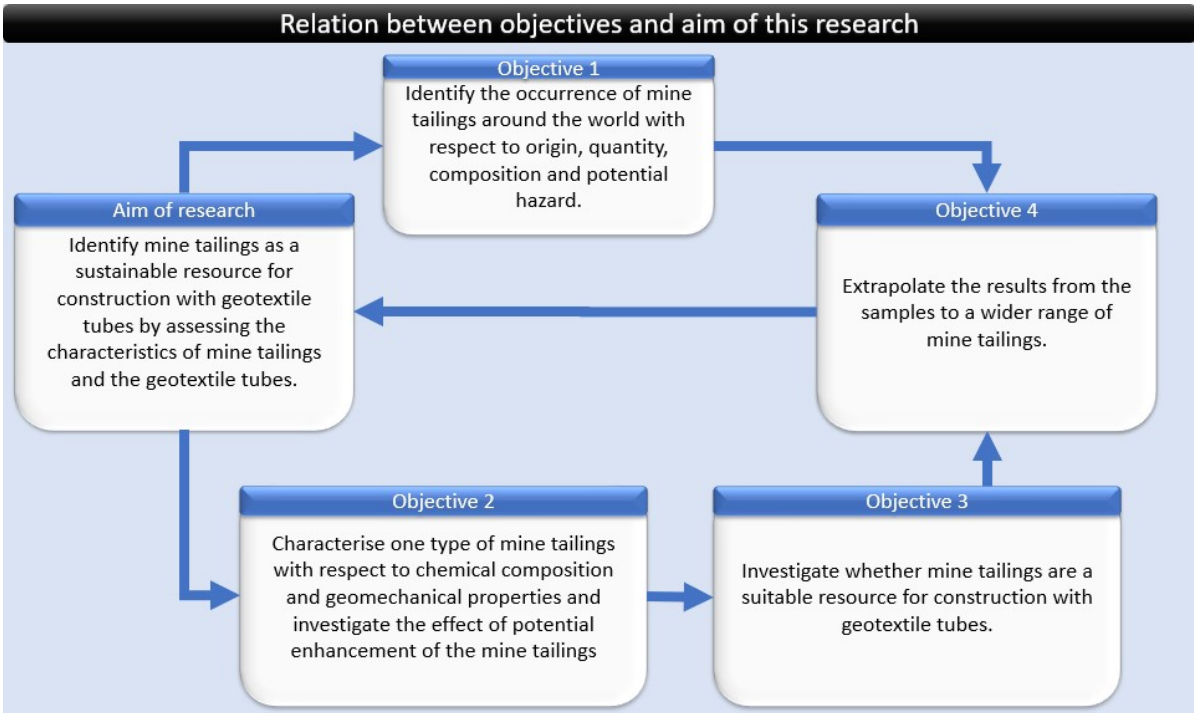


Figure 1.1: Relation between objectives and the aim of this research.

1.4. Research questions

In order to reach the aim, the main research question to be answered is:

Are mine tailings a sustainable alternative resource for construction with geotextile tubes?

To answer this question and to attain the objectives stated in the previous section, the following secondary research questions should be answered:

Objective 1:

1.1. Where in the world are mine tailings storage facilities located?
1.2. What is the quantity of mine tailings storage facilities and what is their mineralogical composition?
1.3. What are the potential hazards of mine tailings storage facilities?

Objective 2:

2.1. What are the geomechanical properties of the iron ore tailings within the tailings pond?
2.2. How are the geomechanical properties of mine tailings affected by potential remediation?
2.3. What are the geomechanical properties of the iron ore tailings after remediation?

Objective 3:

3.1. What are the geomechanical requirements of mine tailings in order to be suitable for use as resource for construction with geotextile tubes?
3.2. Do the geomechanical properties of treated mine tailings from research question 2.3 satisfy the requirements from research question 3.1?

Objective 4:

4.1. What are the similarities and the differences between the samples and mine tailings in general and how can the results be extrapolated to a wider range of mine tailings?

1.5. Scope and Limitations

This research is focused on treating mine tailings in such a way that they can be used as a sustainable resource in construction with geotextile tubes. The proposed construction application is to strengthen tailings dam walls by filling geotextile tubes with mine tailings. All other applications will not be considered. Therefore, this research will not include the dewatering and stabilisation methods that do not sufficiently enhance the properties of mine tailings to the extent that it can be used to fill geotextile tubes to strengthen tailings dam walls. This means that for the different dewatering methods, drying will not be included. For the stabilisation methods, this means that only the immobilisation method will be investigated. An overview of the scope of this research is given in Table 1.1.

Table 1.1: Scope of research.

Category	Included	Excluded
Application	Strengthening of dam walls by geotextile tube	Aggregates Substitute in cement mixing Geowall
Mine Tailings	Origin Quantity Mineralogy Geo-chemical behaviour Potential hazards	Chemical composition Biological behaviour
Dewatering	Sedimentation Filtering	Drying
Stabilisation	Immobilisation	Isolation Vitrification Permeable treatment walls Biological stabilisation Physical separation Extraction

1.6. Report outline

This report is build as follows:

Part 1: Introduction

Chapter 1: Introduction:
An introduction to the topic with an overview of the aim, the objectives, the research questions and the scope of this research

Part 2: Theory

Chapter 2: Mine tailings:
An overview of mine tailings: their origin, their quantity, their mineralogical composition and the potential hazards.

Chapter 3: Dewatering:
A extensive explanation of common dewatering methods and machinery.

Chapter 4: Stabilisation:
A brief introduction to several stabilisation methods.

Chapter 5: Analysis of literature:
Analysis of chapter two, three and four.

Part 3: Methodology

Chapter 6: Material used:
Overview of materials used during testing.

Chapter 7: Methodology:
Comprehensive explanation of test procedures used for this research.

Part 4: Results and Discussion

Chapter 8: Results:
Presentation of results according to the structure described in the methodology.

Chapter 9: Discussion:
Discussion of the results, their trends and the correlation with theory.

Part 5: Conclusion and Recommendations

Chapter 10: Conclusion:
Conclusion of the overall findings related to the scope of this research.

Chapter 11: Recommendations:
Recommendations regarding path forward based on the scope of this research.

Part II

Theory

2

Tailings

In this section the general aspects of mine tailings are described. This section starts with an introduction on tailings, explaining what tailings are and their occurrence and quantity around the world. This is followed by an explanation of the different types of mine tailings with respect to the geochemical origin.

2.1. Introduction

All mining projects in the world are aimed at the exploitation of ore. Whether this ore contains purely the desired element, depends on the elements and the environment of deposition. For example, alluvial gold deposits contain grains of pure gold while most metal deposits contain ore where the ore is found within different mineral structures. When the ore consists out of various minerals where the desired elements is found within a mineral structure, this ore must be processed in order to liberate the element. The processing of ore normally involves various steps of size reduction and separation of individual minerals. Figure 2.1 depicts a simplified flow-chart of the processing of ore.

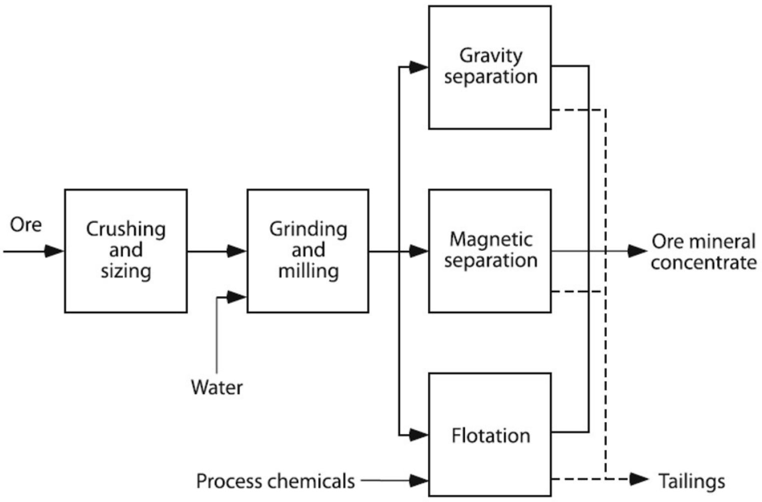


Figure 2.1: Simplified flow-chart of ore processing, to yield an ore mineral concentrate and tailings [Lottermoser, 2010].

Mineral processing generally results in two products: the mineral concentrate and a quantity of residual waste, called tailings. These tailings are normally produced in the form of a suspension and consist of a fine-grained sediment and water. Due to the various separation steps in the processing cycle, the sediments in the tailings are mainly fine gangue from which most of the valuable minerals are removed. Examples of these gangue minerals are silicates, oxides, hydroxides, carbonates and sulphides. Despite this, a certain amount of these valuable minerals end up in the tailings [Lottermoser, 2010]. These tailings are normally pumped in tailings storage facilities where the solids can settle in the suspension, separating the two faces.

2.1.1. Quantity

Putting an exact number to the tonnes of tailings stored within tailings storage facilities is difficult. The quantity of the tailings produced depends on type of the mine, the desired commodity, the grade of the ore, the efficiency of the processing plant and the method of deposition of the tailings. However, it is possible to give a rough indication of the tonnes of tailings produced annually. This can be done by combining the annual world production of metals and minerals and their corresponding average grades resulting in an approximation of the tonnes of tailings produced annually [Lottermoser, 2010].

Table 2.1: World production of several metals, minerals, ores and mineable minerals fuels in 2015.

Commodity	World Production	Unit	Ratio Commodity /Tailings	Tailings produced	Unit	Source ratio
Antimony	0.141	Mt	-	-	Mt	-
Arsenic Trioxide	36500	t	0.5%	7	Mt	[Lottermoser, 2010]
Bauxite	299	Mt	100.0%	299	Mt	do.
Beryl	5740	t	0.5%	1	Mt	do.
Chromite	28	Mt	0.5%	5572	Mt	do.
Cobalt	0.126	Mt	0.5%	25	Mt	do.
Copper	19.1	Mt	0.5%	3800	Mt	do.
Gold	3100	t	0.0002%	1550	Mt	do.
Iron Ore	2290	Mt	100.0%	2290	Mt	do.
Lead	4.95	Mt	0.500%	985	Mt	do.
Managese Ore	51.8	Mt	-	-	Mt	
Mercury	2470	t	0.5%	0.49	Mt	[Lottermoser, 2010]
Molybdenum	0.235	Mt	-	-	Mt	
Nickel	2.28	Mt	0.5%	454	Mt	[Lottermoser, 2010]
Palladium	215	t	-	-	Mt	
Platinum	189	t	-	-	Mt	
Other Pt-group metals	69.3	t	-	-	Mt	
Silver	27600	t	-	-	Mt	
Tin	0.289	Mt	0.5%	58	Mt	[Lottermoser, 2010]
Titanium (Ilmenite/ Leucoxene)	7.27	Mt	-	-	Mt	
Titanium (Rutile)	0.85	t	-	-	Mt	
Tungsten	89400	t	0.5%	18	Mt	[Lottermoser, 2010]
Vanadium	77800	t	0.5%	15	Mt	do.
Zinc	12.8	Mt	0.5%	2547	Mt	
Lignite	821.2	Mt	80.0%	205.	Mt	[Hoffmann and Huculak-Maczka, 2012]
Petroleum Oilsands	147.9	Mt	10.0%	1331	Mt	[Oil Sands Magazine, 2019b]
Uranium	0.0723	Mt	0.1%	72	Mt	[OECD/IAEA, 1999]
Peat	27.5	Mt	-	-	Mt	-

Table 2.1 shows the world production of several metals, minerals, ores and mineable mineral fuels for 2015. For example, for Arsenic Trioxide the ratio is 0.5%. This means that 0.5% of every tonne of mined ore is Arsenic Trioxide and the remaining 99.5% is waste material. A

ratio of 100% means one tonne of mined ore produces one tonne of tailings. When the ratio is kept blank, it means that no reliable value for the ratio between commodity and tailings could be found. The table shows that multiple tonnes of tailings are produced for every tonne of mined commodity. In this table industrial minerals are not included since it is difficult to specify waste and tailings for all industrial minerals. When all the tailings are added up, roughly 20 billion metric tonnes of tailings were produced in 2015 but this quantity will probably be higher when more metals and minerals are included in this calculation.

Moreover, the average grade of ore is decreasing, as shown in Figure 2.2 and Figure 2.3, while the demand for metals and minerals will increase in the coming 40 years [Bloodworth and Gunn, 2012]. It can be stated that the quantity of tailings produced annually will increase for the coming years.

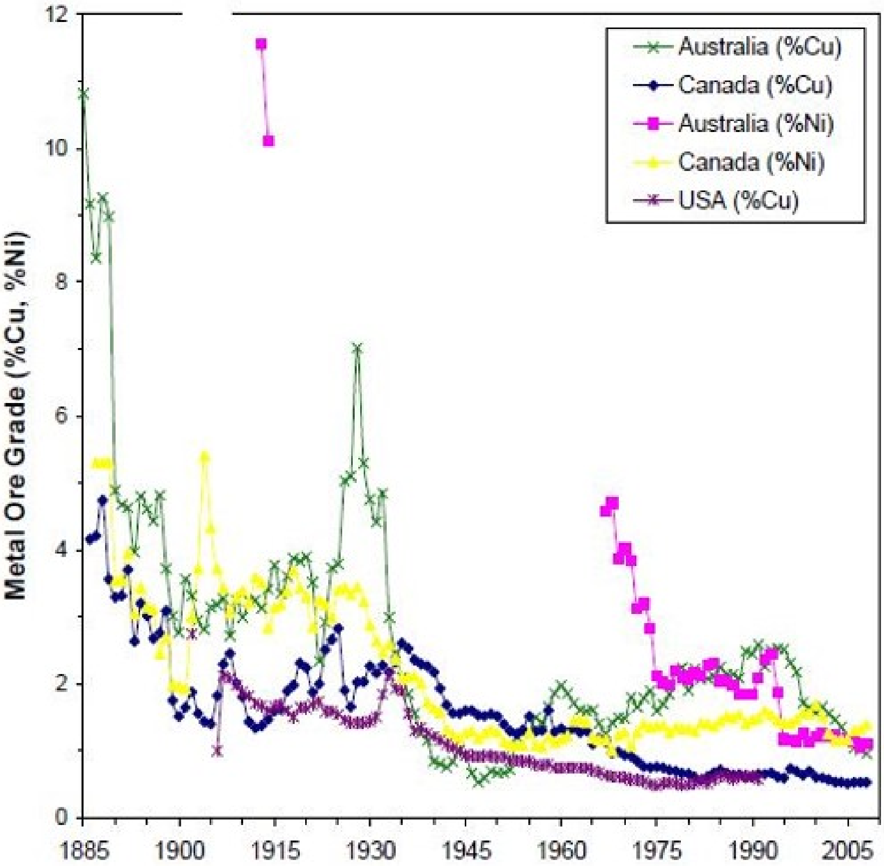


Figure 2.2: Average metal grade in % over time [Giurco et al., 2010].

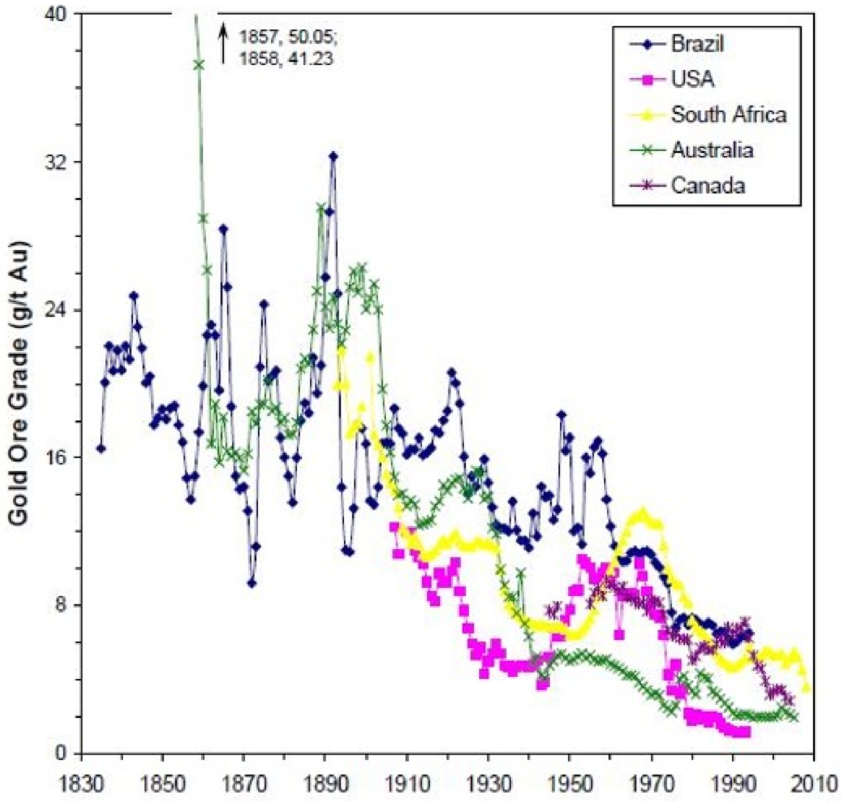


Figure 2.3: Average grade of gold ore in ppm over time [Giurco et al., 2010].

Approximately 3500 tailings storage facilities exist worldwide. In general, these dams are constructed and raised according to one of the designs given in Figure 2.4. It is estimated that more than 50% of these dams are constructed in an upstream design, where the dam crest, including the upstream edge of the crest, moves over beached tailings at any point in its sequential raising. Approximately 60% of all tailings dams failures were upstream constructed tailings dams [Davies and Martin, 2000]. Therefore, it can be stated that upstream designed tailings dam are prone to failure. This is one of the most important reasons why tailings dams have to reinforced.

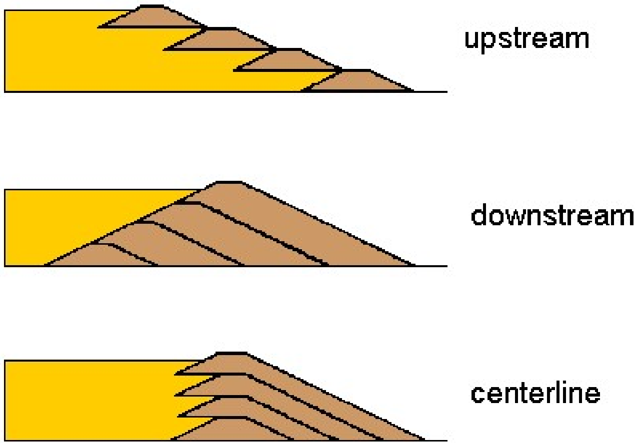


Figure 2.4: Types of sequentially raised tailings dams [Nilsson, 2001].

2.1.2. Chemical composition

The exact chemical composition of tailings within tailings storage facilities can differ from mine to mine. However, most tailings that are produced nowadays can be categorised in groups based on the origin of the ore [Lottermoser, 2010].

These groups are:
- Tailings based on sulphidic ore
- Cyanide based tailings of gold-silver ore
- Radioactive tailings of uranium ore
- Tailings of phosphate and potash ore
- Tailings of solid minerals fuels
- Tailings of liquid mineable mineral fuels
- Tailings of industrial minerals

In the following sections these groups are elaborated further: the most important minerals and chemical processes in these tailings are explained and the environmental concerns that come with storing the tailings in tailings storage facilities are highlighted.

2.2. Tailings based on sulphidic ore

The majority of mined metal ore consists of a combination of sulphide and a metal, e.g., Galena (PbS), Chalcopyrite (CuFeS), Sphalerite (Zns), Argentite (AgS) or Cinnabar (HgS). In addition, the ore contains a range of gangue minerals including silicates, oxides, hydroxides, phosphates, halides and carbonates. Sulphide ore can contain a wide range of minor and trace elements. Most of these minor and trace elements substitute the metals in sulphides. Minor and trace elements that can do this include: Ag, As, Bi, Cd, Co, Cr, Cu, Fe, Ga, Ge, Hg, Mn, Mo, Ni, Pb, Sb, Se, Sn, Ti, V, Zn. This is an overview and not every sulphide mineral can contain all of the trace elements. A more detailed table of the trace elements per sulphide minerals is given in Table 2.2. Mining of this ore is done via surface or underground mining methods. The ore is sent to the processing plant where the metals are extracted from the ore. Figure 2.1 shows an overview of the general processing steps that are performed in this plant. The first processing steps includes various stages of size reduction by crushing and grinding the ore to a desired size. Next, the metals are separated from the gangue minerals via various processing steps, depending on the metal that needs to be separated and the ore. In these separation steps, the gangue minerals are disposed of while the concentrate is processed further. Depending on the separation method used, the processing step is done wet or dry. When a dry method is used, the dry gangue material is placed in dry tailings storage facilities. When a wet method is used, the wet gangue material is further processed to dewater the slurry and then sent to the wet tailings storage facilities.

During the whole production cycle of sulfidic ore, the sulphide minerals are exposed to the environment. While the sulphide minerals were in equilibrium with its surroundings in the deposit, the moment the ore is extracted they are not in equilibrium anymore. Under the influence of oxygen, the sulphide minerals tend to react towards a new equilibrium. In this process, the sulphide minerals are oxidised, producing acid and sulphates, called Acid Mine Drainage [AMD]. In the presence of acid, heavy metals and metalloids that were initially in a solid form, are now dissolved and brought into solution. Many of the sulphide minerals react with oxygen and are oxidised but the most important sulphide mineral is pyrite (FeS_2). During the oxidation of pyrite, Fe^{2+} is produced. The Fe^{2+} is further oxidised to Fe^{3+}, bringing Fe^{3+} in the solution. This Fe^{3+} will react with other pyrite, resulting in a reactive system that sustains itself. In addition, this system produces an acid solution, dissolving metals and metalloids [Rimstidt and Vaughan, 2003].

Table 2.2: Mineral chemistry of metal sulphides [Vaughan and Craig, 1978].

Mineral Name	Chemical Formula	Minor and trace element substitution
Arsenopyrite	$FeAsS$	-
Bornite	Cu_3FeS_4	-
Chalcocite	Cu_2S	-
Chalcopyrite	$CuFeS_2$	As, Bi, Cd, Co, Cr, In, Mn, Mo, Ni, Pb, Sb, Se, Sn, Ag, Ti, V, Zn
Cinnabar	HgS	-
Cobaltite	$CoAsS$	-
Covellite	CuS	-
Cubanite	$CuFe_2S_3$	-
Enargite	Cu_3AsS_4	-
Galena	PbS	Ag, As, Bu, Cd, Cu, Fe, Hg, Mn, Ni, Sb, Se, Tl, Zn
Mackinawite	$(Fe,Ni)_9S_8$	-
Marcasite	FeS_2	As, Hg, Se, Sn, Ti, Tl, Pb, V
Melnikovite	Fe_3S_4	-
Millerite	NiS	-
Molybdenite	MoS_2	-
Orpiment	As_2S_3	-
Pentlandite	$(NiFe)_9S_8$	-
Pyrite	FeS_2	Ag, As, Au, Bi, Cd, Co, Ga, Ge, Hg, In, Mo, Ni, Pb, Sb, Se, Sn, Ti, Tl, V
Pyrrhotite	$Fe_{1-x}S$	Ag, As, Co, Cr, Cu, Mo, Ni, Pb, Se, Sn, V, Zn
Realgar	AsS	-
Stibnite	Sb_2S_3	-
Sphalerite	ZnS	Ag, As, Ba, Cu, Cd, Co, Cr, Fe, Ga, Ge, Hg, In, Mn, Mo, Ni, Sb, Se, Sn, Tl, V
Tennantite	$(Cu,Fe)_{12}As_4S_{[13}$	-
Tetrahedrite	$(Cu,Fe)_{12}Sb_4S_{13}$	-
Violarite	$FeNi_2S_4$	-

2.2.1. Products of sulphidic ore processing

All the waste that is produced during mineral processing is considered tailings. The tailings are usually in the form of a suspension of fine-grained particles and water. The fine-grained particles are mostly gangue minerals like silicates, oxides, hydroxides, carbonates and sulphides. The slurry contains the process chemicals used during processing. This includes the flotation reagents, modifiers, flocculants, coagulants, hydrometallurgical agents and oxidants. Moreover, the tailings can contain some valuables minerals if the processing is not 100% efficient, which it never is.

All the tailings are stored in tailings storage facilities. These are large facilities where the slurry is allowed to settle, separating the solids from the liquids. The water will start to flow on top of solids and can be decanted for reuse in the processing plant. The solids remain behind and form different layers based on settling velocity of the different particles.

2.2.2. Environmental concerns

The tailings storage facilities are generally not constructed to completion. They are raised gradually as the impoundment fills. The larger the tailings storage facilities get; the more material is stored in them. With this growth, the potential hazard of failure of these tailings storage facilities also increases as more material is released when the facilities fail. This can happen due to liquefaction, a rapid increase in dam wall height, foundation failure, excessive water levels due to high rainfall or excessive seepage. When these facilities fail, large volumes of contaminated material are released into the surrounding, contaminating the environment and causing major hazard for the local population [Achterberg et al., 1999]. In addition, the facilities contain a range of minerals. While some do not react with the environment, like silicates, other minerals do, like sulphides. The sulphides will oxidise under the influence of oxygen and produce AMD. This acid liquid may come in contact with the surrounding groundwater via seepage, potentially contaminating the groundwater with acid and mobilised metals and metalloids.

2.3. Cyanide based tailings of gold-silver ore

Gold-silver ores are mined by surface or underground mining. The gold and silver are extracted from the ore predominantly by cyanide leaching. In this process a cyanide solution is used to selectively dissolute gold and silver, creating a solution rich in gold and silver. This process is so efficient that the processing of low-grade ores has become profitable, and it has become possible to extract a few grams of gold from a tonne of rock. As a result, a large portion of waste rock produced in gold mining are contaminated with cyanide.

Cyanide

Cyanide is a general term which refers to a group of chemicals which combines carbon and nitrogen to form cyanide, (CN^-), compounds. These compounds can be classified into five general groups.

- Free cyanide
- Simple cyanide
- Weak complexed cyanide
- Moderately strong complexed cyanide
- Strong complexed cyanide

Cyanide

Free cyanides are defined as the forms of molecular and ionic cyanide released into aqueous solution by the dissolution and dissociation of cyanide compounds and complexes. The free cyanide can have two forms, the cyanide ion (CN^-) and hydrogen cyanide (HCN). How these two forms are distributed in the solution depends on the pH of the solution. In solution with a pH below 8.5 the free cyanide is present as hydrogen cyanide. This compound has a low boiling point and a high vapour pressure, resulting in a phase change at water/air interfaces. At these interfaces the hydrogen cyanide will change from a dissolved free cyanide to a cyanide gas (HCN(g)), which can be harmful for employees in the vicinity. Less harmfull is CN, which can be achieved by keeping the pH around 10.5. At this pH level, the majority of the free cyanide in the solution is the cyanide ion (CN^-). When the pH is higher than this value the efficiency of the gold extraction process decreases [Mudder et al., 1991].

Simple cyanide

Simple cyanides are defined as the salts of the hydrocyanic acid. The most used forms are potassium cyanide (KCN) and sodium cyanide (NaCN). The simple salts dissolve completely in an aqueous solution, producing free alkali cations and cyanide anions. The cyanide reacts with water to form a hydrogen cyanide and a hydroxide [Mudder et al., 1991].

Weak, moderately strong and strong cyanide complexes

The weak, moderately strong and strong cyanide compounds are more complex when dissolved into an aqueous solution. These compounds do not release a cyanide ion when dissolved but release a complex radical anion, consisting of a transition metal and cyanide ion. The definition of the complex anion, namely weak, moderately strong and strong, is dependent on the stability of the metal cyanide complex [Mudder et al., 1991].

2.3.1. Extraction of gold

The production of gold involves a two-stage process: an extraction stage and a recovery stage. In the extraction stage, the gold is dissolved using cyanide. In the recovery stage, the dissolved gold is recovered from the cyanide solution using cementation with zinc or adsorption onto carbon. The extraction stage starts with the production of a cyanide solution by dissolving sodium cyanide ($NaCN$) and lime ($CaCO_3$) in water. The extraction step can be done via two methods: the heap leaching process and the vat/tank leaching process.

In the heap leaching process, the gold ore is extracted and piled onto plastic lined pads. These plastic lined pads provide an impermeable base to prevent ground and surface water contamination. During the process, a dilute solution of sodium cyanate is sprayed onto the heap and allowed to percolate down through the ore. The cyanide solution moves through the heap and dissolves gold and silver during this movement. The resulting solution is rich in gold and silver and is captured at the bottom of the heap by the plastic pad. The solution is then channelled into a holding pond for further processing and gold extraction. After the gold and silver are extracted from the solution, the solution is recycled, recharged with additional cyanide, returned to the heap and sprayed over the pile again [Bahrami et al., 2007].

In the vat/tank leaching process the gold ore is placed in large enclosed tanks. In these tanks the gold is dissolved using a cyanide solution. After the gold is dissolved, it adheres to pieces of carbon. These pieces of carbon can be stripped from the solution. The processed ore is removed and the cyanide solution is recycled, in a similar was as the heap leaching process.

Both methods are based on the ability of cyanide to form highly soluble gold complexes. However, the cyanide anion not only dissolves gold and silver, it may also dissolve metals and metalloids when these are present in the ore. Metals and metalloids can include: As, Cd, Co, Cu, Fe, Hg, Mo, Ni, Pb, Zn. The dissolution of these metal and metalloids depends on the form in which they are present within the ore. When the ores are completely weathered and oxidized, the metals and metalloids are present as secondary minerals, which are insoluble in the alkaline process water. When the sulphide-rich ore is processed, sulphide oxidation, acid generation and a reduction in the solution pH can be expected. In addition, the sulphide minerals will react with the cyanide in the process water, resulting in the formation of thiocyanate (SCN^-) and metal cyanide complexes. Consequently, the reaction with the sulphide minerals will lead to a higher cyanide consumption during processing and a solution containing a range of cyanide species and complexes and significant metals and metalloids concentrations [Lottermoser, 2010].

2.3.2. Products of cyanide treatment

As previously mentioned, the production of gold results in a large quantity of tailings for every tonne of gold ore processed. These tailings products can be categorised into three groups:
- Process water
- Heap leach residues
- Tailings

Process water

Once the gold and silver are extracted from the cyanide solution, the solution is recycled and re-used in the processing of ore. However, when the concentrations of thiocyanate and metal cyanide complexes are too high, the solution is discarded from the process circuit because this process water can no longer be used efficiently. This discarded process water may be treated to reduce cyanide levels. After treatment, the process water is pumped into tailings storage facilities.

Heap leach residues

After the heap leaching process, the processed ore piles remain. These piles do not commonly show sulphide oxidation or acid mine drainage. This is partly due to the fact that sulphide ore is usually not processed via cyanide leaching. However, if sulphide ore is processed via cyanide leaching, the processing takes place in an environment with a very high pH, which in turn neutralises any acid produced during sulphide oxidation and acid mine drainage. The processed ore may contain carbonate minerals and therefore, the piles may be strongly alkaline after processing.

Tailings
When the ore is processed via the vat/tank leaching process, the resulting ore may be pumped to a tailings storage facility as a slurry. When the tailings settle in this facility, the pores of the tailings will be filled with a dilute cyanide solution and some of the solution may start to float on top of the tailings. The portion of the solution that starts to accumulate at the surface of the facility may be collected and reprocessed to extract more gold.

2.3.3. Environmental concerns

The three groups of tailings produced during gold extraction, all contain cyanide or cyanide related groups. It is well known that cyanide is harmfull for humans and animals [World Health Organization, 2004]. Therefore, any contact of humans and animals with cyanide while dealing with cyanide bases, must be avoided. This can be achieved with proper management of the facilities and preventing liquids containing cyanide and cyanide related groups from contaminating surrounding surface and groundwater.

Both the process water and the tailings are placed into tailings storage facilities. These facilities can contain large volumes of solids and liquids when used for a longer time. Failure of these tailings storage facilities is not uncommon [Davies and Martin, 2000]. When these facilities fail, large volumes of the solids and liquid are released into the environment, potentially harming the environment and the local population and wildlife.

The processed heap leach piles are large piles of solids. As mentioned before in this section, these piles can contain large volumes of carbonate minerals and can be strongly alkaline. This alkalinity prevents plants from growing on the waste piles due to a reduced availability of nutrients and an environment which most plants do not tolerate. When sodium cyanide is used as the major cyanide provider, the high sodium concentrations will have an adverse impact on the soil structure. The sodium will cause clay particles to disperse, resulting in a weak soil, susceptible to erosion. Moreover, these large piles are susceptible to failure, potentially moving large volumes of solids in very little time and harming surrounding employees and infrastructure.

2.4. Radioactive tailings of uranium ore

Uranium is extracted from uranium ore. Uranium ore can occur as reduced and oxidised species. Reduced uranium minerals consist of U^{4+} whereas the oxidised uranium minerals contain the U^{6+} ions. Uraninite (UO_{2+x}) is the most mined mineral uranium containing mineral in the world. However, uranium ore grades are expressed in terms of triuranium octoxide (U_3O_8), which is 85% elemental uranium. Triuranium octoxide (U_3O_8) is an oxidised form of uraninite. Uranium ore can contain variable levels of various metals and metalloids such as: As, Cu, Mo, Ni, Pb, Ra, Re, Sc, Se, Th, V, Y, Zr [Pichler et al., 2001].

Uranium is mined via three basic processes: surface mining, underground mining and In Situ Leaching [ISL] [Abdelouas et al., 2006]. The tailings produced from surface mining and underground mining are similar in mineral composition and geo-mechanical behaviour while the tailings produced from ISL differs from these two.

2.4.1. Extraction of uranium

When surface and underground mining methods are used to extract the uranium ore, the ore is first crushed and/or powdered to reduce the grain size and then leached. Sulphuric acid leaching is the most common method. Alkaline leaching is preferred when the ore contains limestone or similar basic constituents. This is because the acid consumption during leaching will be high when processing basic ore. The uranium solution is then purified and concentrated by ion exchange or solvent extraction. After concentration, the uranium is precipitated from the solution, filtered and dried, which produces a concentrate, called yellowcake [OECD/IAEA, 1999].

The ISL method leaches the uranium within the deposit. To do this, the deposit should be porous (sandstone), saturated with groundwater and confined between relatively impermeable layers. The uranium is leached from the deposit by injecting an oxidant and an acidic or alkaline leaching solution in the deposit. The solution moves through the ore body and oxidises and dissolves the uranium. At the extraction well, the uranium-bearing solution is extracted and transported to the processing facility. Here, the uranium is recovered via ion exchange, similar to surface or underground mining [OECD/IAEA, 1999].

2.4.2. Products of uranium extraction

When surface and underground mining methods are used, tailings are produced during the extraction of uranium from the ore. This happens in the processing plant. As mentioned before, the chemical characteristics of the ore determine whether acid or alkaline leaching is used to extract the uranium. Both methods result in the production of tailings, consisting of slurries of sands and clay-like particles, which are pumped to the tailings storage facilities for disposal. Due to the low grades of uranium in uranium ore, with an average of 0.1% [OECD/IAEA, 1999], the dry weight of the tailings produced is nearly equal to the dry weight of ore processed. Typiccaly, dry tailings are composed of 70 to 80 wt % sand-sized particles and 20-30 wt % finer-sized particles. An overview of the typical characteristics of uranium processing tailings is given in Table 2.3 [Oak Ridge National Laboratory, 1997]. The tailings contain traces of radium (Ra) and Thorium (Th). These traces are low level radioactive. In addition to Ra and Th, the tailings will also contain all the daughter nuclides of the radioactive U-235 and U-238, which were originally present in the uranium ore. These nuclides are redundant and are separated from the uranium during processing. Consequently, the tailings will contain multiple low-level radioactive elements.

The ISL method results in a smaller amount of solids and slimes in the tailings. This is because the uranium is leached from the ore within the deposit, resulting in the majority of the solids staying in the deposit. However, the chemical and mineralogical composition of the solids and liquids in the tailings for both the ISL method and the surface and underground mining method are equal.

The radionuclides, heavy metals and metalloids in the sand and slimes fraction are found in a wide range of minerals and chemical precipitates during the entire processing and disposal phase. The majority of the radionuclides, heavy metals and metalloids originate from the gangue and secondary minerals found within the ore. The remaining fraction can be found in the chemical precipitates which are formed during and after mineral processing and after disposal of the solids within the tailings storage facility. Depending on the environment of deposition, the solids are stable. However, when the environment of deposition changes, the stability of the solids may change as well resulting in the potential release of radionuclides, heavy metals and metalloids [Lottermoser, 2010].

2.4.3. Environmental concerns

As stated in Table 2.3, the tailings stored in tailings storage facilities can contain a range of elements, both radionuclides and trace elements. The radionuclides that are within these storage facilities are of major concern. Due to the long half-lives time of some radionuclides (Th-230, Ra-226), these radionuclides can remain within life-support systems and food chains for a very long time. Therefore, successful isolation of these radionuclides should be carried out to prevent them from entering the surrounding ecological systems, potentially harming the environment [OECD/IAEA, 1999]. When the tailings come in contact with oxygen, sulphides present in the tailings can oxidize, producing AMD. The acid that is produced during this process can dissolve trace elements that were present in the tailings. These mobilised metals can come in contact with surrounding ground and surface water, potentially

Table 2.3: Typical characteristics of uranium mill tailings

Tailings component	Particle size	Chemical composition	Mineralogical composition
Sands	>75	• SiO_2 with <1 wt% complex silicates of Al, Fe, Mg, Ca, Na, K, Se, Mn, Ni, Mo, Zn, U, V • U_3O_8 • Ra, Th, when acid leaching is used	Gangue minerals originally present in ore
Slimes	<75	• Small amount of SiO_2, mostly rather complex clay-like silicates of Na, Ca, Mn, Mg, Al and Fe present in ore • U_3O_8 • Ra, Th when acid leaching is used	Fine grained gangue minerals originally present in ore and clay minerals, oxides, fluorides, sulphates and amorphous substances
Liquids	Acid leaching	• pH 1.2 to 2.0: Na^+, NH^{4+}, $SO_4{}^{2-}$, Cl^- and $PO_4{}^{3-}$ • Dissolved solids +-10 wt % • U, Ra, Th	
	Alkaline leaching	• pH 10 to 10.5: $CO_3{}^{2-}$ and $HCO_3{}^-$ • Dissolved solids +- 10 wt% • Ra	

harming the environment and the local population [Fernandes et al., 1998]. Just as important are the tailings storage facilities themselves. When these facilities are filled up and are not monitored, failure of these facilities can occur, potentially releasing large volumes of solids and water into the surroundings, harming environment and the local population.

2.5. Tailings of phosphate and potash ore

Phosphate and potash have in common that they are both used as macronutrients for plants and agricultural crops. The ore from which they are mined differ both in origin and chemical composition, resulting in different tailings and associated environmental concerns.

2.5.1. Phosphate

Phosphate ore is found within sedimentary or igneous deposits. These deposits are mostly mined via surface mining and underground mining. The ore is usually crushed or grinded, scrubbed, washed with water and screened and/or separated via hydrocyclones. This process aims to separate the valuables from the gangue material, called beneficiation. The composition of the gangue material depends on the type of deposit, the associated minerals and the level of weathering of the minerals that has occurred within the deposit prior to mining of the ore. In general, the major impurities include organic matter, clay and other fines, siliceous material, carbonates and iron bearing minerals. The distribution of the impurities influences the above-mentioned sequential beneficiation steps. In addition to the mineral based impurities that are present in the ore, elemental impurities can also be found. Their origin lies in the most important phosphate mineral in phosphate ore, fluorapatite ($Ca_5(PO_4)_3F$). This mineral can contain many different elemental substitutions in its structure, as shown in Table 2.4. In general, these impurities stay in the mineral structure during the beneficiation steps. Elemental impurities also include trace elements present in the phosphate ore [Segura et al., 2016].

Table 2.4: Possible substitutions in the francolite structure

Constituent ion	Substituting ion
Ca^{2+}	Na^+, K^+, Ag^+ Mg^{2+}, Sr^{2+}, Ba^{2+}, Cd^2+, Mn^{2+}, Zn^{2+} Bi^{3+}, Sc^{3+}, Y^{3+}, REE^{3+} U^{4+}
$PO_4{}^{3-}$	$CO_3{}^{2-}$, $SO_4{}^{2-}$, $CrO_4{}^{2-}$ $CO_3 \cdot F^{3-}$, $CO_3 \cdot OH^{3-}$, $AsO_4{}^{3-}$, $VO_4{}^{3-}$ $SiO_4{}^{4-}$
F^-	OH^-, Cl^-, Br^- O^{2-}

The separated fine fraction, which consist of clay, fine-grained aluminium and iron phosphates, are used as backfill material or stored within tailings storage facilities. The siliceous material may require an additional beneficiation step such as froth flotation. After this step, the siliceous material can also be used as backfill material or may be stored. Carbonates are usually removed through calcination followed by slaking with water. Any iron minerals present, are in the form of magnetite, hematite and goethite. These minerals are removed through scrubbing and size classification or magnetic separation. Both the carbonates and the iron minerals can be used as backfill material or are stored within tailings storage facilities [UNEP and IFA, 2001].

In addition to the tailings that are produced during the separation steps, tailings are also produced during the processing of the phosphate ore. Here, phosphate bearing minerals within the phosphate ore is converted to phosphoric acid, an important intermediate product for the manufacture of phosphatic fertilisers. In this step, the elemental impurities present in the mineral structure, are mobilised and some are separated from the valuable phosphate (Table 2.4). The most common way to convert the phosphate ore to phosphoric acid is by the wet-process, also known as acidulation. In the wet-process, the ore reacts with sulphuric acid. In addition to the production of phosphoric acid, the process also produces a by-product known as phosphogypsum ($CaSO_4 \cdot$2H2O) [Rutherford et al., 1994]. Although phosphogypsum is used in Japan and Germany for the production of cement and other construction materials, in most other phosphate producing countries the phosphogypsum is pumped to tailings storage facilities for cooling and dewatering. The main reason for this is the elemental impurities that were initially present in the phosphate ore. These impurities are mobilised during the wet phosphoric acid process and are released into solution. During the formation of phosphogypsum, many of these impurities remain in the crystalline structure of phoshophysum. An overview of the impurities and how they are involved in the crystalline structure of phosphogypsum is given in Table 2.5.

Table 2.5: Impurities in phosphogypsum structure

Impurity	Found as
Cd, Sr, Mg, Na, K, Cl	Substitute into the gypsum structure
$CaPO_4$, $NaSO_4$, POF	Substitute into crystal structure of gypsum crystals
Ce	In combination with Na substitutes Ca in phosphogypsum
Trace elements	Unknown form, possibly formed under aerobic and acidic conditions
Ra-226, Po-210	Found in phosphogypsum Ra-226 precipitates as a sulphate ((Ba, Ra)SO_4)
U	• Dissolved in residual phosphoric acid as $UO_2{}^{2+}$ • $UO_2{}^{2+}$ substitutes Ca on surface of crystal • Absorbed on surface of gypsum as UO_2HPO_4

2.5.2. Environmental Concerns

The tailings that are produced and stored within tailings storage facilities, come with environmental concerns. These concerns are:

- Atmospheric contamination with fluoride or other toxic elements
  During the processing of fluorapatite by acidulation, in addition to phosphoric acid, hydrofluoric acid (HF) may also be formed. HF remains in the liquid phase during processing but may become gaseous under high pressure conditions. Some remaining gaseous fluorine (F) may end up in the tailings storage facilities but most processing plants remove the remaining fluorine before disposition of tailings in tailings storage facilities. When HF comes in contact with the skin, serious burns will be the result [Yamashita et al., 2001].

- Groundwater pollution with mobile anions, acidity, trace elements.
  During the acidulation process, many elemental impurities in the ore are mobilised and brought into solution. Some are found in relation with phosphogypsum (Table 2.4), where they are part of the crystal structure. Others remain in solution. However, in both cases, the impurities may contaminate the groundwater via seepage or through downward leaching of rainwater that infiltrates a tailings storage facility.

- Radon gas and other radioactive exposure
  As already mentioned in this section, the nuclides uranium (U), radium (Ra) and polonium (Po) may be found within the crystalline structure of phosphogypsum. As all of them are unstable, both the parent and the daughter radionuclides are found within phosphogypsum. When the phosphogypsum is stored in tailings storage facilities, these radionuclides are concentrated. Uranium, radium and polonium may come into solution and can contaminate the groundwater in much the same way as trace elements do, via seepage or leaching of rainwater. However, radon, a daughter nuclide of Ra-226, is a noble gas and may be released into the air when exposed to the atmosphere. Therefore, radon may be inhaled by people working or living close to the tailings storage facility. The concentration of radon in the atmosphere is much lower than the maximum limit allowed to be inhaled by humans and does not pose a threat to human safety. The other nuclides in the solution are harmful. They can accumulate downstream and emit high levels of radioactivity.

- Instability of tailings storage facilities
  For every tonne of phosphoric acid produced, approximately five tonnes of phosphogypsum is produced as by-product. All phosphogypsum produced worldwide is placed in tailings storage facilities, apart from Japan and Germany. These facilities are susceptible to failure, potentially releasing millions of tonnes of phosphogypsum in the environment, damaging local environment.

2.5.3. Potash

Potash is a general term that is used to refer to a group of potassium salts. This group includes carbonate, sulphate and chloride compounds. The potash deposits are, in general, evaporitic salt deposits. These deposits mainly consist of gypsum ($CaSO_4 \cdot 2\ H_20$), anhydrite ($CaSO_4$) and halite (NaCl). The most important potash mineral is sylvite (KCl) followed by carnalite ($KMgCl_3 \cdot 6\ H_2O$). Gangue minerals present in the deposit may include clay minerals, sulphides, carbonates, iron oxides and a range of evaporative salts such as kieserite ($MgSO_4$) and tachyhydrite ($CaMgCl_4 \cdot 12\ H_2O$). All of the gangue minerals are separated from the valuables and are disposed of. Depending on the business model of the mine, halite may also be separated from the gangue material and sold to the market. However, in the majority of the potash mines, halite is considered a gangue mineral and is disposed of [Lottermoser, 2010, Rauche et al., 2001].

Potash can be mined via two methods, underground mining and solution mining. Conventional underground mining is used in the majority of the potash mines and is based on extraction of the potash ore by mechanical means. Solution mining is based on the greater solubility of sylvite in brines at elevated temperatures than halite. In this process, brine is heated at the surface and injected into the orebody through well. The heated brine absorbs sylvite from the orebody and is then pumped back to the surface and placed into cooling ponds. In these ponds, the brine is cooled down and the dissolved potash precipitates. The brine is removed from the pond and can be reused again. The potash is recovered from the pond and is send to the processing plant. The biggest advantages of using solution mining is the greater depth that can be reached than with conventional underground mining.

For the underground mining methods, the general processing steps that are used to liberate the potash from the ore include size reduction, desliming, separation, drying, compaction and granulation. The order and the steps used are dependent on the characteristics of the ore. Size reduction is often done by grinding and crushing of the ore and desliming is done by intense agitation. The separation steps can be done by flotation, electrostatic separation, thermal dissolution-crystallisation and heavy media separation. Often several of these steps are used together to enhance recovery. After separation, the potash concentrate is dried to reduce the water content and then compacted and granulated to produce a uniform size potash product.

Brines and tailings are the major waste streams that are produced from the beneficiation steps. Brines are the liquid wastes from potash mines and are saline solutions that can be enriched in ore elements and compounds including calcium, potassium, sodium, magnesium, chloride and sulphate. The brines can be disposed of by:

- Injecting the brine into deep aquifers below the orebodies.
- Discharge into the ocean.
- Treatment and disposal into local rivers.
- Use it with or without solid residue as backfill material.

Tailings are the solid waste produced during processing of potash ore. It contains gangue minerals and salts produced during the beneficiation steps. These solids may be used together with brines as backfill material or can be stored at the surface in tailings storage facilities.

2.5.4. Environmental concerns

Storage of the solid waste produced by potash mines is the major concern. The piles can be become very large in size and can contain large portions of salt. This salt can dissolve relatively easily under influence from rainwater and may move to the surrounding area. These elevated levels of salt can be hamrful for the environment and should be contained.The other concern are the stacks themselves. When these waste piles fail, large volumes of solid material are moved within seconds, potentially harming the surrounding environment and population.

2.6. Tailings of solid minerals fuels

Solid mineral fuels are the different kinds of coal, lignite and peat. Coal can be mined using underground or surface mining methods whereas lignite and peat is mainly mined via surface mining. This section will discuss coal and the corresponding mineralogical and chemical composition. For lignite and peat, the mineralogical and chemical composition is roughly the same as for coal but the water content is higher.

The occurrence of coal can vary, from thin steeply dipping deposits, to thick flat deposits. However, regardless of the shape of the orebody, most of the coal deposits consists of a number of minerals which are: coal, quartz (SiO_2), clay minerals, pyrite (FeS_2), calcite ($CaCO_3$), siderite ($FeCO_3$) and in lesser quantities ankerite ($Ca(Mg,Fe)(CO_3)_2$). In addition to these minerals, coal deposits also contain minerals in smaller quantities such chalcopyrite ($CaFeS_2$), galena (PbS) and sphalerite (ZnS) and other less important minerals. When the layers are extracted, oxygen can come in contact with the sulphide minerals, oxidising the minerals, resulting in the production of sulphates and an acidic solution. Other Fe^{2+}-bearing minerals present in the mineral fuel layers that oxidise when in contact with oxygen are siderite and ankerite. Siderite will dissolve when in contact with acid, consuming the acidity. In the presence of oxygen, the released Fe^{2+} ion will oxidise to Fe^{3+}, consuming an acid atom. However, the hydrolysis of Fe^{3+} will generate three H^+ molecules. This means that, in general, siderite will produce more acidity then it consumes. On the other hand, when ankerite dissolves, the resulting reactions will consume more H+ molecules then they produce, making the oxidation reaction of ankerite a neutralisation reaction rather than an acid generation reaction [Younger, 2004].

Other elements that are found within coal layers are the trace elements: Ag, B, Bi, Ce, Cr, Dy, F, Ge, Ho, La, Mn, Nd, P, Pr, Re, Sb, Sm, Ta, Ti, V, Yb [Schweinfurth, 2009]. Before the coal is ready to be used, the extracted coal has to undergo several beneficiation steps. These steps include washing, crushing and grinding, size separation, desliming, gangue separation, blending and drying [Bethell, 2013]. The end product of the beneficiation is a blend of coal that is ready to be fed to the incineration plant. The first step of beneficiation is washing of the coal. Fine particles, mostly clay minerals, are brought in suspension and are removed from the process. This suspension is dewatered and stored in tailings storage facilities. During several separation steps, gangue minerals are separated from the coal. Trace elements that are inorganically bound are also separated here. These gangue minerals, such as quartz, calcite and several sulphides and the inorganically bound trace elements are placed in separate tailings storage facilities. When the coal is put into the incineration plant and burned, this burning also produces waste. The primary waste products are two types of ash, bottom and fly ash. Fly ash contains the light particles, (smaller than <100 μm) that are released during the incineration process. Bottom ash contains the heavier particles, (larger than >100 μm) such as slag [Kopsick and Angino, 1981]. In this ash, remaining gangue minerals are present, which were not initially separated from the coal during beneficiation. In addition, organically bound trace elements, of which the initial bonding was destroyed during incineration, are also found in this ash.

2.6.1. Environmental concerns

As stated above, three types of waste material are produced during the processing and incineration of coal, lignite and peat: a suspension of fine particles and water, gangue material and ash. All of these waste products need to be disposed of in an environmentally safe way. Therefore, they are placed in tailings storage facilities.

Suspension of fine particles and water
The slurry that is produced during the different beneficiation steps is pumped to tailings storage facilities. Here, the solids in the slurry settle, separating solids and water. The solids are used as backfill in the mine or placed on storage piles. These tailings storage facilities, both wet and dry are a potential threat to the environment due to their mineralogical composition. Sulphides that may be present in the coal deposit can react with oxygen, producing AMD. The acid that is produced during this process can in turn mobilise trace elements. These trace elements can be transported from the storage piles, into ground or surface water. This way higher concentrations of toxic trace elements, such as Hg and As, can be found in ground and surface water, potentially harming local population and the surrounding environment.

Gangue material
Gangue material is usually placed on storage piles. These piles can be very big and contain a range of minerals. These big piles are susceptible to failure, potentially harming the surrounding environment by the sudden movement of large volumes of solid material. In addition, gangue material can consist of a range of sulphides. Just as with the fine particles, the sulphides in the gangue material can react with oxygen, potentially producing AMD. The potential threats that this phenomenon can cause are similar as with the fine particles in

suspension. The low pH can mobilise trace elements, which in turn can contaminate ground and surface water potentially harming local population and the surrounding environment.

Ash
The ash that is produced during the incineration process is an aluminosilicate glass, consisting of Ca, Al, Fe and Si oxides with minor amounts of Mg, Ti, Na, K, S and some trace elements. While the various oxides make the matrix, the trace elements are found both absorbed onto the surface of the crystal or incorporated into the crystalline structure. The Ca, Al, Fe, and Si that are found in the ash, originate from the gangue minerals that remain in the coal after beneficiation. Calcium carbonate and clay minerals are the most abundant gangue minerals, followed by sulphides, chlorides and oxides [Kopsick and Angino, 1981]. When fly ash is considered suitable for industrial use, it can be re-used in cement and concrete products, improving the quality of such products [Dermatas and Meng, 2003]. However, if the fly ash is not suitable for industrial use, the fly ash must be stored. For bottom ash, the same applies. Both fly and bottom ash are used as backfill material, when the incineration plant is located close to the mine. When this is not the case, the two types of ashes are placed in a tailings storage facility. Both wet and dry storage occur although wet is preferred because of the size of the particles. Particles of fly ash are small, when this is stored dry, it can be moved by wind. This is not an issue when wet storage is used. However, wet storage can have its own complications. Similar to the fine particles in suspension and the storage of gangue material, the storage of ash in wet tailings ponds can produce AMD. This can also mobilise trace elements present in the storage facility. These trace elements can contaminate ground and surface water if mitigation measures are not in place.

2.7. Tailings of liquid mineable mineral fuels

Liquid mineable mineral fuels are the mineral fuels that are found in liquid form but are mined by conventional extraction methods. Oil sands deposits are an example of these liquid mineable mineral fuels. These deposits are loose sands or partially consolidated sandstones containing a mixture of sand, clay and water, saturated with bitumen. The bitumen content in oil sands range from 0 to 19% by total mass. The water content varies between 3% and 6% by total mass. The major clay components in oil sands are kaolinite (40-70%), illite (28-45%), and traces of montmorillonite [Yution, 2016]. These deposits can be extracted via surface mining. When the permeable layers containing oil are found deeper than 200 metres, steam extraction becomes an alternative and more efficient method to extract the bitumen.

2.7.1. Mining

In surface mining the oil sands are extracted via conventional mining methods and transported to the extraction plant. At the extraction plant, the oil sands are crushed and mixed with a mixture of hot water (50-80 °C), sodium hydroxide (NaOH) and steam. This mixture is agitated to separate the particles from each other. The sodium hydroxide is used to maintain a constant pH of the solution between 8.0 and 8.5, which aids in the separation of the bitumen from the mineral solids. The bitumen is separated from solids by a series of gravity separation steps. The solids are withdrawn as a concentrated suspension, called primary tailings, and are transported to a tailings storage facility. Smaller suspended solids and poorly separated bitumen droplets are both drawn off in a slurry, called middlings, from the middle of the separation vessel. These middlings are processed further to extract as much bitumen as possible. These steps include further agitation and flotation. The residual slurry, containing solids and bitumen, are transported to the tailings storage facilities.

2.7.2. In-situ extraction

When the oil sands deposit is found more than 200 metres deep, in-situ extraction becomes a more efficient method to extract the bitumen from the sands. This method uses steam to heat up and liquify the bitumen. The bitumen emulsion is then pumped to the surface and send to the separation plant. In this plant, the bitumen is separated from the water. The water is recycled and re-used as steam while the bitumen is sent to the refinery for further upgrading. This method is less efficient than surface extraction, with only 40-60% of the bitumen extracted from a deposit. However, this method does not produce any waste material and thus no tailings storage facilities are needed [Oil Sands Magazine, 2019a].

2.7.3. Environmental concerns

The tailings that are produced during the beneficiation steps in the extraction plant are a warm, aqueous suspension of sand, silt, clay and residual bitumen and has a pH of between 8 and 9. This suspension is pumped to a tailings storage facility. Once pumped into the tailings storage facility, the coarse particles settle rapidly, forming beached tailings. The fine fraction remains in suspension longer and flows to the middle of the storage facility. In the middle of this facility the energy of the particles is low and the fine particles settle. If the tailings are undisturbed for a longer time, three layers are developed. At the top, clear water accumulates, which can be recycled and re-used in the processing. Under this layer, the immature fine or thine fine tailings layer is formed, containing fine particles and water. The bottom layer is the mature fine tailings layer [MFT]. This layer contains silts and clays, fine sands, bitumen and water. In general, the MFT has a solid content of 30%. These tailings storage facilities become very large, holding large volumes of solids and liquid. Due to a low hydraulic conductivity and a high thixotropic strength, the settling time of the MFT is very long resulting in very large areas which are needed to allow the MFT to dewater and settle [Yution, 2016].

Due to bad engineering or natural causes, these facilities may fail resulting in large volumes of slurry released into the environment potentially harming local environment and population. In addition, the tailings can contain trace elements like As, Mo, Cu, Cr, Pb, Ni, Fe and V. These trace elements can be mobilised due to the pH of the suspension, potentially contaminating ground and surface water [Small et al., 2015].

2.8. Tailings of industrial minerals

The last group that should be included in this section are the tailings from industrial metals. The most important industrial minerals are bauxite and iron ore.

Iron ore is a rock type that consists of several iron-bearing minerals. The most important minerals are magnetite (Fe_3O_4), hematite (Fe_2O_3) and goethite ($FeO(OH)$). Depending on the deposit, several other minerals are also found in the ore, including quartz and several clay minerals. And in much smaller quantities, some minor and trace elements are also found in the ore such as Ba, Sr, Y, Sc, Zr, Be, V, Ni, Cy, Zn, Ga, Ge, Rb, Nb, Mo, Ag, In, Cs, Sn, Hf, Ti, Pb, bi, Th, U, Au, Ag, Br, Cr, Ir, Sb, Se and Fe^{3+} and Fe^{2+} ions [Kessler and Müller, 1988, Spier et al., 2008].

In general, iron ore is mined via surface mining methods. During the mining phase, the first step of separation is done. Iron ore is extracted and transported to the primary crusher while gangue material, like overburden, is sent to the waste piles. In the primary crusher, the iron ore is crushed to a particle size of 200 mm. This is followed by a series of secondary and tertiary crushers, crushing the ore to particle sizes ranging from 30 to 80 mm [Jankovic, 2015]. After crushing, the crushed iron ore is transported to the processing plant. Here, the ore is separated based on size. Three stockpiles are formed, one with granulated material, one for the sinter-feed size and one for the pellet-feed size. Granulated material is between 0.6 to 5 cm in diameter. Sinter-feed grains are up to 50 mm in diameter [Lu and Ishiyama, 2015] and the pellet-feed is the very fine iron ore powder that remains behind after separation is complete. The separation steps that take place in the processing plant include gravitational separation and magnetic separation. These steps are generally done with a wet feed to increase efficiency. Besides separating the iron ore based on size, the processing plant also separates valuables from gangue minerals. These gangue minerals are considered waste and are placed in tailings storage facilities. Here, the solids in the suspension are allowed to settle, effectively separating water and solids. Water will start to flow on top of the tailings storage facility and can be reused in processing. The solids that settle remain in the tailings storage facilities. Next to the primary gangue minerals present in the tailings storage facility, the other gangue minerals and elements are also found in the tailings storage facility.

2.8.1. Environmental concerns

Due to the various size reduction and separation steps, the particles that leave the processing plant are very fine and are in suspension. Due to these two properties, the material is mostly stored in wet tailings storage facilities. These storage facilities can become very large due to the large volumes of tailings produced every year. According to Table 2.1 2290 Mt tonnes of iron ore were produced in 2015 and for every tonne of iron ore produced, one tonne of tailings is produced as well. This results in 2290 Mt of tailings that needs to be stored in tailings storage facilities. These tailings storage facilities are susceptible to failure, potentially releasing large volumes of tailings into the environment. On top of that, as said before, the tailings contain elevated concentrations of minor and trace elements. Some of these elements, like As are particularly harmful to the local population when it finds its way into the local ecosystem.

3

Dewatering

In order to further process the mine tailings, dewatering first needs to take place. This makes the material easier and cheaper to handle. The main purpose of dewatering is reducing the amount of liquid in the suspension, decreasing the liquid-solids ratio. After the dewatering is finished, a dry concentrate remains for further processing steps. In this chapter, several dewatering methods will be discussed and explained. First the general dewatering methods are explained and how they are used. Next, the methods are explained in more detail.

3.1. Dewatering methods

Dewatering methods can be classified into three groups, based on their mechanism:
1: Sedimentation
2: Filtration
3: Thermal drying

The first method, sedimentation, is most efficient when the density difference between the liquid and the solids is large. This is the case when the carrier liquid is water. However, there are situations where sedimentation cannot always be used. These situations involve hydrometallurgical processes, where the carrier liquid may have high-grade leach abilities and the density may approach the density of the solids. In these cases, the two other methods can be used instead.

In general, dewatering involves a combination of the above-mentioned groups. First, the majority of water is removed via sedimentation and a thickened slurry remains with 50-65 wt% solids. The second step involves filtration, where a moist filter cake is produced, containing between 80-90 wt% solids. The last step in the dewatering process is thermal drying, here the final product of about 95 wt% solids is produced [Wills and Napier-Munn, 2007].

3.2. Sedimentation

Sedimentation is the first method of dewatering. Here, the rapid settling of solid particles in a liquid produces a clarified liquid and a thickened slurry. The clarified liquid can be decanted, and the thickened slurry can be processed further to decrease the water content. Sedimentation is mainly driven by settling rate of particles in the fluid. This settling rate is governed by Stokes' Law or Newton's Law [Stokes, 1880], shown in the equation below:

$$v = \frac{gd^2(D_s - D_f)}{18\eta} \tag{3.1}$$

Where v the terminal velocity of the particle (m s^{-1}), d is the particle diameter (m), g is the acceleration due to gravity (m s^{-2}, D_s = particle density (kg m^{-3}), D_f = fluid density (kg m^{-3}) and η is the fluid viscosity (N s m^{-3})

When dealing with very fine particles, with a diameter of a few micrometre, the settling rate is extremely low when the driving force is gravity. The settling rate can be accelerated by increasing the diameter of the solid particles or by using centrifugal sedimentation. An increase in diameter of the solid particles can be achieved via agglomeration or flocculation of the small particles into bigger lumps, called flocs, that settle more rapidly.
The flocculation process involves three principal steps:

- Destabilization of suspended fine particles
- Floc formation and growth
- Floc degradation

These three steps are in any flocculation process but the third step can be used to control the size of the flocs that are formed.

3.2.1. Destabilisation

The fine particles are often very stable and resist aggregation when in suspension. This is mainly caused by the electrical charge that develops when the particle is dispersed into aqueous media, called the zeta potential. This can be further enhanced by solvation effects or by the presence of protective adsorbed films on the particles surface. The particle charge generally originates from the electrochemical interactions between the solid and the surrounding aqueous solution and is dependent on the pH of that solution. At a certain pH level, the particles have no net charge and they lose their stability. The pH level where this phenomenon occurs is called the isoelectric point. Therefore, controlling the pH level of the solution is an effective method to destabilise particles. However, this becomes impractical when the isoelectric point lies within an inconvenient pH range. When this occurs, an alternative method for eliminating the electric charge on particles is to shield the charge of the particles from other particles in the surrounding. This can be done by adding relative high concentrations of ions, called coagulants, into the solution. With these ions, the electric double layers surrounding the particles are compressed, permitting them to come closer together till the point attractive forces are more dominant.

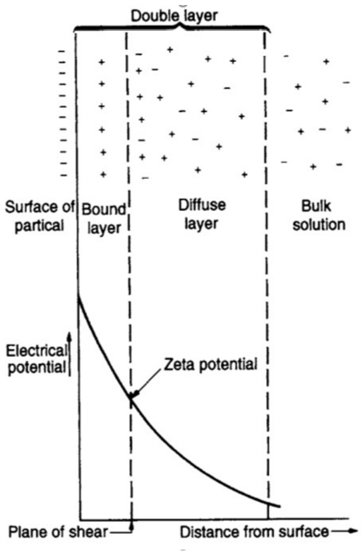


Figure 3.1: The zeta potential [Wills and Napier-Munn, 2007].

Hydrolysable cations such as Cu2+, Al3+ and Fe3+ are effective in dispersion destabilisation. They cause the double layer compression required for destabilisation. Moreover, the cations are highly surface-active and can reverse the zeta potential thus help attaining an isoelectric point. An schematic view of distribution of charges and the corresponding zeta potential is shown in Figure 3.1.

The result of the destabilisation step for a fine particle suspension are small flocs within the suspension. These flocs may grow to larger sizes but are fragile and highly susceptible to breakage in the presence of turbulence or shear. These flocs can reach an equilibrium size within a range of 5 to 10 µm when under vigorous agitation conditions. [Rattanakawin, 2005]. The equilibrium size means that the rate at which flocs develop and break are equal. However, for dewatering applications, either by sedimentation or filtration, larger and stronger flocs are preferred. This is where the next phase starts, namely the floc development.

3.2.2. Floc development

After the destabilisation step, the fine particles in suspension can adhere to each other. If sufficient fine particles have collided and adhered, small flocs begin to grow. The frequency of collisions is dependent on the motion of particles which is in turn dependent on three phenomena. The three phenomena are:

- Brownian motion arising from thermal energy in the suspension.
- Velocity gradients in mechanically agitated suspension.
- Difference in settling rate of individual particles or flocs.

[Kruyt, 1952] described the brownian motion as the dominant mechanism for very small particles. Brownian motion is considered the main driver of the flocculation process during the destabilisation phase and the initial formation of flocs. The effect of agitation becomes apparent later in the flocculation process where existing flocs aggregate to form larger flocs. During the sedimentation phase, the difference in settling rate can become important, if the sedimentation phase takes place under calm conditions. The comparison between the Brownian and shear flocculation is shown in Figure 3.2

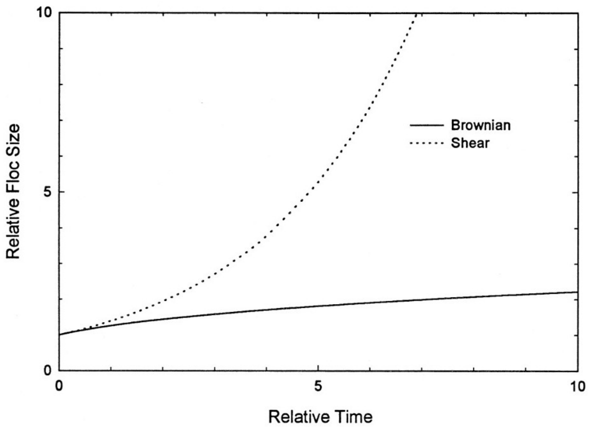


Figure 3.2: Comparison of long-term floc growth for Brownian and shear coagulation with the same initial rate [Hogg, 2000].

Floc development can be increased by adding flocculants, often polymers, which can form bridges between particles. The formation of these bridges can result in agglomeration of the particles. According to [Tarleton and Wakeman, 2007], the flocculation process can be influenced by a range of parameters, as described in Table 3.1. Depending on the desired floc size and strength, these parameters can be used to select the apprioprate flocculant.

3.2.3. Gravity sedimentation

The most commonly used dewatering technique in mineral processing is gravity sedimentation or thickening. This method is relatively cheap, has a high capacity and involves very low shear forces, which facilitates the flocculation of fine particles. The thickener is the vessel where thickening takes place. Here, the large flocs undergo low shear forces and can settle

Table 3.1: Some parameters that influence the flocculation process and their induced effects [Tarleton and Wakeman, 2007].

Parameter	Effects
Flocculant molecular weight increase	Poorer solubility and more viscous solutions
	Polymer chains more shear sensitive
	Higher unit cost and optimum flocculant dose
	Bridging flocculation favoured
	Larger more fragile flocs which often settle faster but give higher sediment volume and water retention during deliquoring
Flocculant dose increase	Improved flocculation up to the optimum, then deterioration
Increase in applied shear/stirring rate	Breakdown of longer polymer chains
	Irreversible floc degradation
	Smaller equilibrium floc size
Particle surface area increase	Greater flocculant consumption
	Ultrafines are susceptible to overdosing and thus restabilisation
Particle concentration increase	Does not always affect optimum dose
	Smaller and perhaps stronger flocs
	Local overdosing possible
Suspension pH	Non-ionics are little affected
	Ionisation and chain extension of anionics at alkaline pH — converse is true for cationics
	Alters particle surface charge
	Combined effects on flocculation are complex
Suspension ionic strength	Similar comments to suspension pH
	Can promote or hinder flocculation
	Excess salts lower solubility and coil polymer chains in solution
Polymer charge density increase	Extends polymer chains in solution under
	Decreases adsorption onto particles of the same charge sign
Temperature change	Complex effects

due to this condition. A sketch of a thickener is shown in Figure 3.3. The suspension is transported to the thickener via the feed well. The rate of the feed is adjusted to generate as little disturbance as possible. The large flocs that are formed in the thickener settle to the bottom of the vessel, from where they are guided to the centre of the thickener by rotating rakes. The thickened suspension is removed from the thickener at the bottom and the clarified liquid is removed at the top, where the clarified liquid is allowed to overflow at the edge of the vessel.

In Figure 3.4, a cross section of a thickener is shown. At the top, the clarified liquid is floating and allowed to overflow the vessel. Beneath the clarified liquid the concentration of solids is roughly the same as the concentration of the feed suspension. At the bottom of the vessel, the thickened slurry is allowed to accumulate and removed from the vessel. Above this thickened slurry, the concentration of solids is changing from the feed concentration to the thickened slurry concentration.

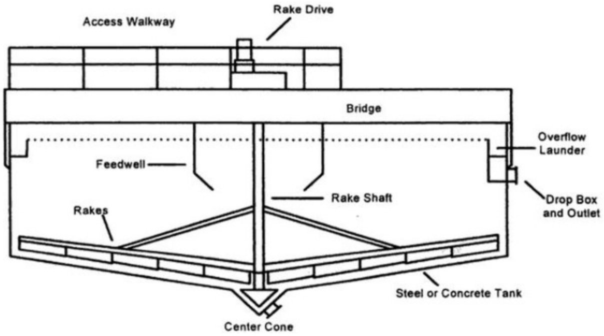


Figure 3.3: Sketch of a typical thickener [Wills and Napier-Munn, 2007].

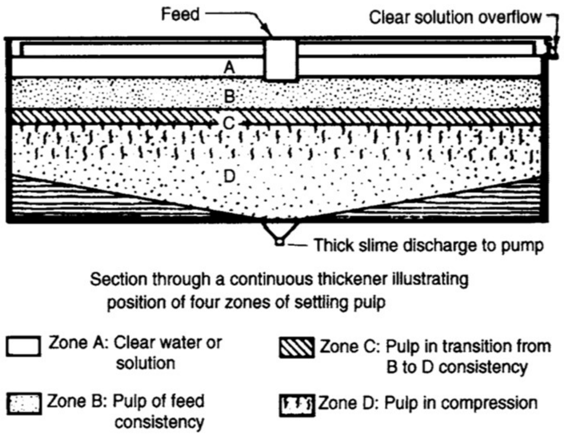


Figure 3.4: Cross section of the different layers inside a thickener [Wills and Napier-Munn, 2007].

3.2.4. Centrifugal sedimentation

Another approach to dewater a suspension by means of sedimentation is using centrifugal sedimentation. The process is similar to gravity sedimentation but here the driving force behind the separation of liquid and water is the centrifugal force. The devices that allow for centrifugal sedimentation are the hydrocyclone and the centrifuge. Both have their advantages and disadvantages.

Hydrocyclones are in general cheap and simple to install and use. However, the major disadvantage of these devices are the solids concentration and the relative proportions of overflow and underflow that results from the process. When dealing with fine particles smaller than ten m, the efficiency of the hydrocyclone drops rapidly, resulting in appearance of fine particles in the overflow. Flocculation of these fine particles is not possible because the shear forces acting on the particles in the hydrocyclone are very high, thus breaking up any agglomeration of particles. A cross-section of a hydrocyclone is shown in Figure 3.5.

Centrifuges are very expensive compared to hydrocyclones and much more complex to install. However, the clarification process is more efficient, and centrifuges are in general more flexible. With a centrifuge much greater solid concentrations can be achieved than with a cyclone. A cross-section of the most used centrifuge, the solid bowl scroll centrifuge, is shown in Figure 3.6.

The main component of the centrifuge is the horizontal rotating cylindroconical shell or bowl, inside which a screw conveyor of similar section rotates in the same direction at a slightly higher or lower speed. The feed suspension is fed to the centrifuge via the inlet which is connected to the centre tube of the revolving-screw conveyor. When the suspension leaves the feed pipe, the slurry is subjected to a high centrifugal force. This force causes the solids to settle at the inner surface of the bowl. Due to the rotation of the scroll, the solids are guided by the movement of the scroll towards the solid outlet. The centrifugal force continuously dewaters the solids as they move from the liquid zone to the outlet. The liquid accumulating in the centrifuge, can flow through the solid bed towards the liquid outlet. Here, the liquid can overflow and leaves the system.

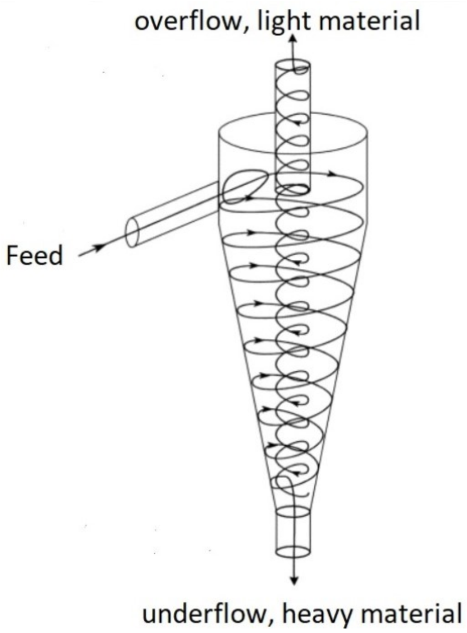


Figure 3.5: Cross-section of a hydrocyclone [Wills and Napier-Munn, 2007].

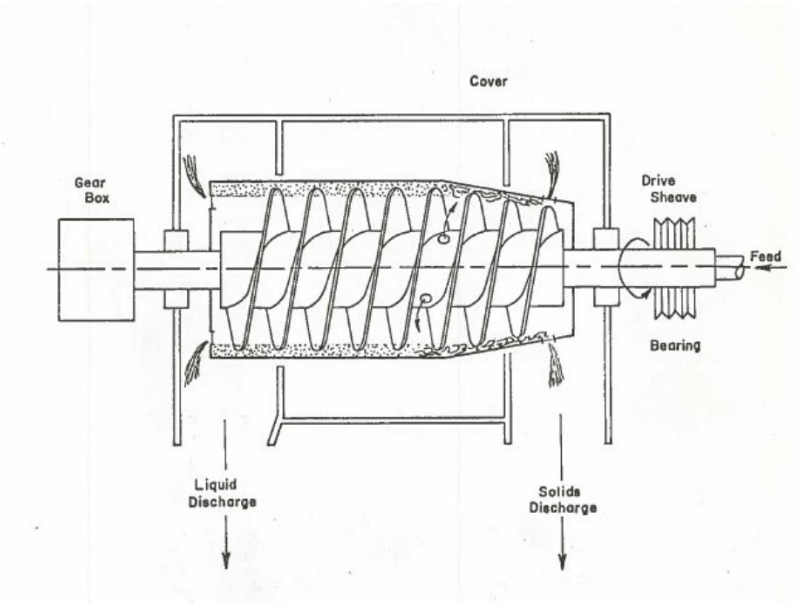


Figure 3.6: Cross-section of a solid bowl centrifuge [Wills and Napier-Munn, 2007].

3.3. Filtration

Filtration is another method of dewatering. In filtration, the solid is separated from the liquid by means of a porous medium which stops the solids from moving through the medium but allows the liquid to pass. Solids will accumulate at the filter. This accumulation of solids is called the filter cake. This filter cake causes an increase in resistance to flow and a decrease in the rate of filtration. Other factors that affect the rate of filtration are:

- The pressure difference between the feed site and the other site of the filter medium
- The area of the filtering surface
- The viscosity of the filtrate
- The resistance of the filter cake
- The resistance of the filter medium and initial layers of cake

Before the suspension is fed to the filtration system, flocculants may be added to the suspension to aid filtration. Fine particles in the suspension can clock the pores of the filter, increasing the resistance of that filter. With the flocculants, the concentration of fine particles in the suspension is reduced, decreasing the probability of fine particles clocking the pores. In addition, flocculants increase the relative number of voids between the particles, increasing the flow of the filtrate.

There are several types of filters that can be used to dewater slurry. However, filter systems are usually based on vacuum filtration or pressure filtration.

3.3.1. Vacuum filtration

Generally, vacuum filtration works as follows: within a rotating slurry tank a pressure difference is created between the filter surface and the feed suspension by means of a vacuum. This pressure difference causes the liquid to move through the filter surface while the solid remains behind, stopped by the filter. The solids accumulate to form a cake. When the tank continues to rotate the cake rises above the slurry and air is drawn to the cake, forcing out the liquid. The liquid, which could move through the filter, leaves the tank via the internal piping present in the tank. The filter cake is removed from the filter medium by either scraping of the cake or by removal of the vacuum within the tank. A sketch of a typical vacuum rotary-drum filter system is shown in Figure 3.7.

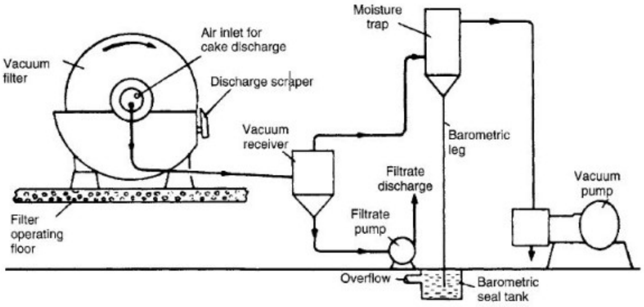


Figure 3.7: Sketch of a typical vacuum rotary-drum filter system [Wills and Napier-Munn, 2007].

3.3.2. Pressure filtration

Pressure filtration is another method which is often used. This system can work under higher pressure differences, which is required for filtering fine particles, than a vacuum filtration system. In the pressure filtration system, the pressure difference is used to force the liquid through the medium while the solids remain behind, forming the filter cake. The liquid is forced out of the voids between the particles by air, effectively dewatering the suspension. After the air blow cycle, the filter cake can be removed by a scraper or cleaning the filter medium using vibration and spray wash.

3.3.3. Geotextile tubes

Initially geotextile tubes were not used for the dewatering of slurry but for the prevention of erosion of beaches. For this purpose, the geotextile tubes were filled with locally available sand. The principle was that the sand could not leave the tubes because the pores in the fabric are too small but the liquid that would enter the tubes can. This would keep the sand in place while not disturbing the rest of the environment. The idea of capturing solids while releasing liquids proved to be usefull for other applications as well. Nowadays, geotextile tubes are mainly used to dewater dredging sludge and other applications where solids and liquids need to be separated in an easy and fast way. Combining these two applications has led to an increase in the use of geotextile tubes as dams and other coastal reinforcement applications. Sludge that is dredged from the local waterways can be pumped in the geotextile tubes. The use of tubes results in separation of solids and liquids, effectively dewatering the dredging sludge while also holding the solids in place, preventing the solids from re-entering the waterways. The tubes, now filled with relative dry solids, can act as coast protection, absorbing wave energy that otherwise will have eroded the coast.

Depending on the application of the tube and the properties of the fill material, the geotextile fabric can be ‘tuned’ to get the best results. The size of the openings can be made large (0.60 mm) or small (0.15 mm) [Howard and Trainer-tencate, 2011]. The material from which the fabric is made can also be changed. Some elements in the dredging sludge can react with the fabric itself, making it less resistant to erosion. In this case, proper choice of fabric material is important. In Figure 3.8, the four basic stages of geotextile tube dewatering are shown. In general, the dredging sludge is pumped directly from the dredging vessel or equipment into the geotextile tubes via pipes. After the tube is filled, the tube dewaters, and the solids consolidate. This process is repeated untill a desired fill height is reached. If desired, flocculants can be added to decrease the settling time of the material. In addition, adding flocculants prevents the openings in the fabric from clogging by particles which have sizes equal to the size of the openings. The working mechanics behind flocculants is given in section 3.2.1, Destabilisation.

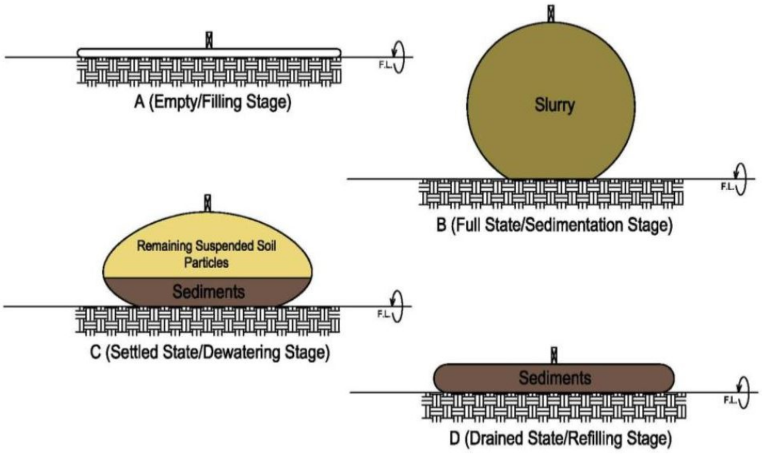


Figure 3.8: Four basic stages of geotextile tube dewatering [Kim et al., 2013].

A major advantage of using geotextile tubes as filtration method is that the geotextile tubes can develop strength when an appropriate textile is selected [Leshchinsky et al., 1996]. With this strength, the filled geotextile tube can be used as a construction element. This has been done already in many occasions, as shown in [Howard et al., 2018] and [Lawson, 2016].

3.4. Drying

The last step of the dewatering process is usually the drying of the concentrates. The aim of the drying process is to reduce the moisture content of the concentrate to about 5%. When the moisture content becomes less than 5%, dust losses may occur. The most often used machine for drying is the Rotary Thermal Dryer. This machine consists of a long cylindrical shell mounted on rollers and driven at a speed of maximum 24 rpm. The shell is at a slight slope, so the material moves from the feed to the discharge under the influence of gravity. The drying of the material is done by hot gasses or air which enter the shell either at the feed end or at the discharge end. The hot gasses may be in direct contact with the material or indirect, depending on the material and the gasses used. The most conventional location of feeding the gasses to the shell is at the feed end because this is more fuel efficient and has a greater capacity than at the discharge end.

An alternative method for drying is the tube press. Here, an hydraulic pressure of 100 bars is used to squeeze the water from the slurry. The water is allowed to leave the tube via a filter while the solids stay behind, similar to a filter system. The filtrate is removed from the tube by compressed air. This method can save up to 80% of the energy required by comparable thermal dryer. However, the tube press method is discontinuous while the rotary thermal dryer can be designed to operate continuous.

4

Remediation

A major issue of finding an useful application for mine tailings is the high level of heavy metals in the soil. These heavy metals can pose a real threat to the local flora, fauna and population if released to the environment. To prevent the release of these heavy metals to the environment, these heavy metals must be remediated. Several methods for remediation of heavy metals in soils and suspension exists. These technologies cane be divided in five categories of general approaches. These categories are: isolation, immobilisation, toxicity reduction, physical separation and extraction. In this chapter, the methods for the remediation of heavy metals in soils will be described [Evanko and Dzombak, 1997].

4.1. Isolation

Remediation via isolation attempts to contain the heavy metals within a designated area, to prevent the release of these heavy metals. This method is often used when other remediation methods proved to be technically or economically infeasible. Isolation is done either by capping the soil or by using surface barriers. Capping the soil uses an impermeable barrier to prevent the inflow from groundwater or the outflow from contaminated liquids. With subsurface barriers, it is possible to control the movement of groundwater at a contaminated site. The barriers are designed to reduce the movement of contaminated groundwater from the site, or to restrict the flow of clean groundwater through the contaminated site [Rumer and Ryan, 1995].

4.2. Immobilisation

The aim of immobilisation is to reduce the mobility of the heavy metals. This is done by changing the physical or leaching characteristics of the surrounding matrix. The most used method to change the physical characteristics is by restricting the physical contact between the heavy metals and the groundwater. Changing the leaching characteristics usually results in chemically altering the heavy metals to make them more stable with respect to dissolution in groundwater. The most used method for immobilisation is by solidification and stabilisation. Solidification means that a solidified matrix is formed that physically binds the heavy metals. Stabilisation is based on a chemical reaction that convert the heavy metals to a less mobile form. Another method to immobilise the heavy metals is by vitrification. Here, the mobility of the metal contaminants can be decreased by high-temperature treatment of the contaminated soil. This results in the formation of vitreous material, usually an oxide solid.

4.3. Toxicity reduction

To decrease the environmental impact of heavy metals in soils, a third method of remediation is to decrease the toxicity of the heavy metals. Decreasing the toxicity can be done by either chemical or biological processes. Chemical processes that reduce the toxicity of heavy metals are based on either oxidation, reduction or neutralisation reactions. Chemical oxidation changes the oxidation state of the metal atom through the loss of electrons. Chemical reduction does the opposite, changing the oxidation state of the metal atom by adding electrons. Changing the oxidation state of metals by oxidation or reduction can detoxify, precipitate or solubilise the metals [Kincaid, 1997]. The chemical processes that will decrease the toxicity of the heavy metals can either be done by adding additives to the suspension before pumping it in the depots, which will react with the heavy metals. Or by using permeable treatment walls, where the heavy metals that are in the groundwater can react with the material within the permeable walls. Both methods will produce a less toxic heavy metal.

Another method to decrease the toxicity of heavy metals is by using biological treatment. Here, biological processes are used to remediate the heavy metals. This is done via a variety of mechanisms, including adsorption, oxidation and reduction reactions and methylation [Igiri et al., 2018].

4.4. Physical separation

The fourth method for remediation of contaminated soils is physical separation. Physical separation attempts to separate the contaminated material from the surrounding soil matrix. This is done by using certain characteristics of both the metal and the soil with which separation can take place for example, density or magnetism. These techniques are most effective when the metal is either in the form of discrete particles or if the metal is adsorbed to soil particles that occur in a particular size fraction of the soil. Physical separation can be done via several techniques including screening, classification, gravity concentration, magnetic separation and froth flotation.

4.5. Extraction

The fifth method for remediation is by extraction of the contaminant. Here, extraction methods are used to extract the contaminated fraction from the rest of the suspension. Contaminant extraction can be done by using a solution that contains extracting agents which will be in contact with the contaminated suspension or by electrokinetic processes. The resulting extracted contaminants can either be treated or disposed of.

4.6. Applied remediation method

There are many ways to remediate mine tailings: isolation, immobilisation, toxicity reduction, physical separation and extraction. Because this research is focused on the useful application of mine tailings, isolation as a method for remediation is not considered here. Physical separation is also not considered here. This is due to the nature of the mine tailings. The solids in the suspension are very fine and separation based on physical properties is not possible. The last method of remediation that is not considered in this research is remediation via extraction. The mine tailings can contain very small concentrations of metals but consists of very big quantities of material that needs to be processed. This means that the extraction of these metals is not economically feasible, and thus will not be considered in this research. The remaining method that will be considered in this research is immobilisation.

The most common method for immobilisation is stabilisation and solidification, abbreviated to 'S/S'. Stabilisation involves the use of chemical reactions that will convert the contaminants to a less mobile form. During solidification, a solidified matrix is formed that physically binds the contaminants [Tajudin et al., 2016].

S/S is done with the use of additives. Depending on the contaminant different additives can be used. In Table 4.1, an overview is given of the different contaminants and the additives that can be used to stabilise and solidify them.
The additives in Table 4.1 can be used separately or can be combined to effectively treat the heavy metals in the suspension.

Contaminant	Additive
Pb	Ordinary Portland Cement (OPC) Quicklime, fly ash (FA), rice husk ash (RHA) Calcium Aluminate Cement (CAC), Sulphate Resistant Portland Cement (SRC), Pozzolanic Cement (PC)
Zn	OPC, CAC, PC, SRC
Cu	OPC, CAC, PC, bottom ash (BA), SRC
Fe	OPC, FA, BA
Mn	OPC
Cd	OPC, CAC, PC, FA, BA, SRC
Ni	OPC, CAC, PC, FA, BA, SRC
As	OPC, CAC, PC, SRC
Cr	OPC, RHA, FA, BA
Ba	OPC, FA, BA

Table 4.1: Contaminant and its corresponding stabilisation additive.

Vitrification is another approach towards the immobilisation of heavy metals. During the vitrification process, materials are converted into a glass or glassy substance through a thermal process. Organic contaminants may be destroyed via pyrolysis or combustion. The inorganic contaminants may be immobilised by incorporating them into the glass structure or by encapsulating them in the product glass. This is similar to the S/S method, where heavy metals are encapsulated by other particles. Due to limited access to resources in order to test the vitrification process, vitrification of contaminated waste will not be considered in this research [U.S. Environmental Protection Agency, 1992].

The toxicity and mobility reduction process aim, similar as the S/S method, to stabilise and restrict movement of contaminants. However, with the process of toxicity and mobility reduction, the contaminants only have a reduced toxicity and mobility while the aim of S/S method is to completely stabilise and solidify the contaminants within the matrix.

5

Analysis of Literature

In this chapter, the literature is reviewed to get an idea of what to expect when working with the iron ore tailings using the dewatering and remedation methods previously described. In the final section of this chapter the literature is analysed to predict the behaviour of the tailings using the chosen dewatering method with the selected remedation method.

5.1. Tailings

Section 2, Tailings, concluded that mine tailings can occur in many variations. Depending on the desired resource, certain mining and processing methods are more applicable than others. The different methods of mining and processing will result in different products, by-products and quantity produced. In addition, the range of gangue minerals found depends on the origin of the desired resource. In general, minerals like silicates, oxides, hydroxides, phosphates, halides and carbonates can be found in almost every mining project. Most of the time, these minerals are considered gangue minerals and are disposed of in tailings storage facilities. These facilities can be either dry or wet, depending on the material stored, the processing steps required to liberate the valuables and the environmental conditions. In general, wet storage facilities are preferred because the majority of the processing steps run more efficient when performed wet. In addition, when fine tailings are stored on dry tailings storage facilities, wind can cause the tailings to be mobilised, causing potential harm to the surroundings.

For this research, the focus lies on the iron ore tailings. As mentioned in section 2.8, Tailings of industrial metals, iron ore can contain a range of gangue minerals and elements. When separated from the valuables, the gangue minerals and elements are stored in the tailings storage facilities. The resulting tailings suspension is pumped to the storage facilities where solids and liquids separate. The liquids start to flow on top of the solids and can be re-used in the various processing steps. The solids settle in the facilities and remain there. Due to the various size reduction steps prior to processing, the solids are very fine (<16 um) [Ranade and Li, 2013], resulting in long settling times of the solids. The solids that are found in these facilities are the various gangue minerals originally present in the ore. These gangue minerals are mainly silicates and clay minerals and in much smaller quantities the various minor and trace elements mentioned in section 2.8, Tailings of industrial minerals

A major issue with these tailings is the small size of the solids in the tailings. Conventional separation of solids and liquids will take a long time or is very inefficient, resulting in high costs. The contaminants in the tailings are another issue. The levels of heavy metals and metalloids will be elevated due to concentration of the contaminants. Preventing the release of these heavy metals and metalloids in the environment has priority, resulting in expensive mitigation measurements. The maximum allowed concentrations of several elements in water bodies, as reported by [Segura et al., 2016], is given in Table 5.1.

Table 5.1: Maximum allowed concentrations of some metals in ppm

Elements	Maximum concentration (μg/l)	Maxium concentration (ppm)
As	10	0.01
Ba	700	0.7
Cd	1	0.001
Pb	10	0.01
Cu	9	0.009
Cr	50	0.05
Fe	300	0.3
Mn	100	0.1
Hg	0.2	0.0002
Ni	25	0.025
Ag	10	0.01
Se	10	0.01
Zn	180	0.18
Al	100	0.1
Be	40	0.04
Co	50	0.05
Li	2500	2.5
U	20	0.02

5.2. Dewatering

From Chapter 3, Dewatering, it can be concluded that there are various ways to dewater a suspension. Sedimentation is the most used, followed by filtration and drying. For this research, the decision has been made to use filtration in the form of geotextile tubes. Here, the suspension will be pumped in geotextile tubes. The water is forced outwards, through the pores in the geotextile fabric. The solids cannot move through these pores and are kept inside the geotextile tubes. Therefore, the liquids are separated from the solids and can be re-used in the processing while the solids are kept in place. To be able to design the geotextile tube and to make sure that the tubes are filled in a safe manner, some properties of the material should be known before the filling of the tubes should start. These properties are given in Table 5.2 [Besseling and Ekkelenkamp, 2017].

Table 5.2: Parameters that are important to be able to design geotextile tubes.

Property	Parameter
Grain size distribution	Clay fraction Silt fraction Sand fraction
Organic material	Organic material fraction
Acid level of material	Ph-value
Lime content	Lime content
Degree of compaction	Proctor density at field moisture content
Consistency	Atterberg limits Moisture content
Salt content	Chloride fraction
Friction	Undrained shear strength

5.3. Remediation

According to Chapter 4, Remediation, many methods exist to stabilise a contaminated soil. These categories are isolation, immobilisation, toxicity reduction, physical separation and extraction. All of them have their own advantages and disadvantages. For this research, the immobilisation method has been chosen to remediate the contaminants in the mine tailings and to be more specific the stabilisation and solidification method. This is considered the most effective method because it can be applied in situ while keeping in mind the intended reuse of the mine tailings as a construction resource. Which stabilisation additive to use depends on which metals need to be stabilised. In addition to the immobilisation of the contaminants, the curing of OPC has another important phenomenon, namely an increase in strength [Osbaeck and Johansen, 1989]. This second aspect, an increase in strength, is important when a potential reuse of the tailings is considered. With a higher strength, more construction options become available. However, the strength development is dependent on the hydration reaction that occurs when cement and water come into contact. In addition to the increase in strength, another important effect will result from using OPC for the stabilisation of the iron ore tailings, a pH increase. When OPC is added to water, the pH will increase to around 12.5 meaning that the slurry will be alkaline [NZRMCA, 2004].

5.4. Stabilised flocculated iron ore tailings in geotextile tube

When sections 5.1, 5.2 and 5.3 are combined, it is possible to predict the dewatering behaviour of iron ore tailings inside a geotextile tube as a construction element in three general effects. First, if the iron ore tailings are pumped directly in the tube, without any additive, the small size of the particles will have a negative effect on the dewatering behaviour of the tube. The openings in the fabric will clog resulting in a decrease in dewatering speed of the geotextile tube. This can be prevented by using a flocculant. With the flocculant, the fine iron ore tailings particles will form flocs. These flocs accumulate at the openings of the textile, leaving more space between the flocs. Through these voids water can continue to flow, dewatering the suspension in the tube. When coagulants are added to the tailings as well, the efficiency of the dewatering process will be even higher due to particles clogging together faster, as described in section 3.2, Sedimentation.

Secondly, if OPC is used to stabilise the iron ore tailings, it is expected that the contaminants present will be stabilised by the cement, forming a strong crystalline structure. In addition, it is expected that adding cement will increase the strength of the final slurry. However, it is impossible to predict what the final strength will be. In addition, the increase in pH resulting from using the OPC will also results in ionisation and chain extensions of the anionic flocculants and the opposite will happen for cationic flocculants as seen in Table 3.1. This effect is relevant because it will be mean that the anionic flocculants are more effective in alkaline environments.

Lastly, by pumping the iron ore tailings into geotextile tube, the use of the geotextile tubes will have two advantages. First, the tailings storage facilities are reinforced, making them safer and secondly, the geotextile tubes are filled with mine tailings from the tailings storage facilities, reducing the volume of mine tailings stored inside the storage facilities.

Part III

Methodology

6

Material Used

This chapter will describe all the material used. First, the sampling of the iron ore tailings will be described, followed by the description of the flocculant, the coagulant and the binder that was used to stabilise the material. This chapter ends with the description of the geotextile that was used.

6.1. Iron ore tailings

Experiments were performed using iron ore tailings. These samples were taken from a tailings dam from the VALE Vargem Grande iron ore mine in the state of Minas Gerais in Brazil. Ten drums of approximately 20 to 25 litre each were filled and sent to Royal IHC in Kinderdijk, the Netherlands. The location where the samples were taken is believed to be a representative location for the entire tailings dam. The exact location from where the samples were taken is given in Figure 6.1, encircled in green, with the scale given in the bottom right corner.

Figure 6.1: The exact location of the samples given in green, blue point: spot A, yellow point: spot B [Google., nd]

The transport of the sample and the vibrations caused by this transport, resulted in some separation of solids and liquids. Therefore, to get a representative sample of the actual iron ore tailings in the pond, the samples had to be homogenised. This was done by mechanically stirring the sample bins.

In Kinderdijk, four 500 ml jars were filled from four different drums. These jars were sent to AT MilieuAdvies B.V., were the material was tested for soil properties and metal concentrations. The results from this analysis are given in Appendix A with a summary of the results given in Table 6.1. Additionally, Clariant AG performed a mineralogical analysis by means of XRD analysis. The results of the XRD analysis are given in Appendix B.

Table 6.1: Summary of analysis iron ore tailings

Parameter	Value
Dry solids (%)	77.1
Organic material (%)	1.8
Calcite (%)	<0.5
Clay fraction (%)	16.05
Silt fraction (%)	71.25
Sand fraction (%)	12.7
pH	6.4
Arsenic (As) (ppm)	22.75
Cadmium (Cd) (ppm)	0.23
Chromium (Cr) (ppm)	27
Copper (Cu) (ppm)	11.85
Mercury (Hg) (ppm)	0.195
Nickel (Ni) (ppm)	21.75
Lead (Pb) (ppm)	20.25
Zinc (Zn) (ppm)	37.25
Barium (Ba) (ppm)	117.5
Cobalt (Co) (ppm)	15.75
Manganese (Mn) (ppm)	7100
Molybdenum (Mo) (ppm)	2.55

Taking in consideration the metal concentrations that are allowed in a water body in Brasil, shown in Table 5.1, it can be concluded that all metals concentrations are above the maximum allowed concentrations for a water body. This means that the iron ore tailings may not be discarded in water bodies. Furthermore, it means that specific measures had to be undertaken to prevent the material from entering water bodies.

6.2. Flocculant

The flocculant that were used in this research were produced by Kemira Oyj. The flocculants are described in more detail in Table 6.2. The relative molecular weights of the flocculants were chosen based on Table 3.1. For this research, the desired floc size needs to be a big as possible. That way, the dewatering process would be the most efficient. For this research, tests were performed with both anionic flocculants as cationic flocculants. This way, it was possible to find a flocculant that would work the best.

Table 6.2: Flocculant type and the corresponding description

Flocculant	Description
A-150	High molecular weight with high anionic value
A-120	Average molecular weight with average anionic value
XD-5500	High molecular weight with high cationic value
C-494	Average molecular weight with average cationic value

6.3. Coagulant

The coagulant used in this research is the Invoque MC700, produced by Clariant AG. Their coagulant was based on a calcium carbonate suspension. Clariant AG also performed their own tests resulting in a recipe that could be used for the iron ore tailings. They concluded from their own tests that their MC700 coagulant could be used efficiently in combination with a flocculant. In addition, they concluded that flocculants with a medium moleculair weight with a medium anionicity and flocculants with a high moleculair weight with a high anionicity would give the best results with respect to turbidity and filtration time. Furthermore, they concluded that the settling time takes longer for flocculants with high anionicity.

6.4. Binder

For this research, OPC was used as the stabiliser. The OPC was classified to as CEM I. This means that the cement will contain up to 5% minor additional constituents. The decision was made to use this type of cement to reduce the additional contaminants that will be added to the slurry, in comparison to using other types of cement. In addition, this type of cement is considered to be the 'standard' cement [Zhang, 2011].

6.5. Geotextile

The geotextile used in this research is the standard GT-500 geotextile from Ten Cate. This textile is very versatile and is applied worldwide in many different projects. It was decided to use this standard geotextile because the focus of this research was more on the additives and less on the filtering medium. The specifications of the geotextile are given in Table 6.3.

Table 6.3: Properties of the GT-500 geotextile [Howard and Trainer-tencate, 2011].

Property	Value
Material	Polypropylene
Biological Degradation	Inert
Chemical, Alkali, and Acids	Resistant
Wide Width Tensile Strength-MD (kN/m)$_a$	70
Wide Width Tensile Strength-CD (kN/m)$_b$	96
Wide Width Tensile Elongation-MD (%)$_c$	20
Wide Width Tensile Elongation-CD (%)$_c$	20
Factory Seam Strength (kN/m)	70
Apparent Opening Size - AOS (mm)	0.43
Water Flow Rate (L/min/m2)	813
Mass per Unit Area (g/m2)	585
UV Resistance at 500 hr (% Strength Ret.)	80

[a] Properties in Machine Direction
[b] Properties in Cross Direction
[c] Maximum Value

7

Methodology

In this chapter, the methodology is explained. To test the hypothesis of this study, a series of experiments were performed, focussed on flocculating, binding and dewatering behaviour of iron ore tailings in geotextile tubes. First, the experimental procedure is explained. This is followed by a description of the preparation of the flocculant solution, the preparation of the slurry, the jar tests and the geotextile cone tests. Furthermore, the preparation of the material for the strength tests, the torvane shear strength tests and the penetrometer tests are elaborated. This chapter ends with explanation of the method to perform the hanging bag tests.

7.1. Experimental procedure

Figure 7.1 shows the series of tests used to determine the flocculating, binding and dewatering behaviour of the iron ore tailings. To test the flocculation behaviour of mine tailings in geotextile tubes, first the separation of solids and water had to be investigated. This was done with a series of jar tests with different additives and different dosing of those additives. To investigate the interaction between the iron ore tailings, the additives and the geotextile, a series of geotextile cone tests were performed. From these geotextile cone tests, the dewatered material was taken and used in a series of strength tests to determine the binding properties of the iron ore tailings. The results from the jar, the geotextile cone and the strength tests were used to choose three mixtures which were used in the large-scale dewatering tests. The large-scale dewatering tests were done in the form of the hanging bag tests. In the following section, the different tests are elaborated further, and the results of the tests are given in Chapter 8, Results.

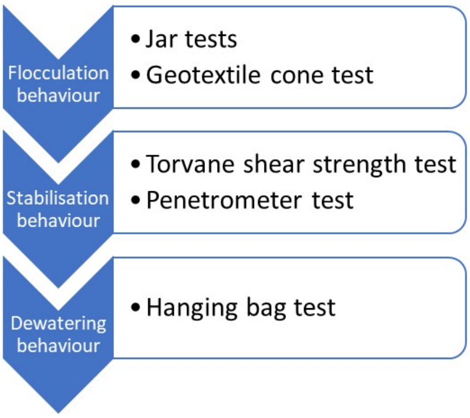


Figure 7.1: Tests used to determine the flocculating, stabilising and dewatering behaviour of iron ore tailings.

7.1.1. Preparation of polymer solution

The flocculants that were used to flocculate the iron ore tailings had to be prepared first. The flocculants provided by Kemira were in a dry form and had to be made into a solution first. To be able to dose the flocculant precisely, a flocculant solution with a concentration of 0.1% was used. Flocculant solutions with a higher concentration would make dosing of the flocculants less accurate due to the viscosity of the flocculant solution. The higher the concentration of the flocculant solution, the higher the viscosity of the solution, making it harder to handle the solution properly.

Procedure:

1. The required dry polymer was weighed. (for a 1000 ml of polymer solution at a concentration of 0.1%, 1.0 g of dry polymer was weighed.)
2. The weighed dry polymer was placed in a clean and dry beaker.
3. The required mass of the demineralised water in a beaker was weighed. (for 1000 ml of polymer solution at a concentration of 0.1%, weigh 1000 g of demineralised water.)
4. The water was directly poured on the dry polymer in a beaker and stirred vigorously for at least 15 seconds with a spoon.
5. The polymer solution was aged for at least 24 hours. This way, the unwinding of the polymer inside the solution was ensured.
6. Before the polymer solution was used, the polymer solution was stirred for at least 15 seconds.

7.1.2. Preparation of slurry

To be able to work with the samples, the samples had to be diluted first. The dilution of the samples was done by adding demineralised water to the samples. First, the required original sample was weighed on a scale. Second, water was added to the samples and the mixture was stirred. The dilution factor was test dependent. For the jar test, a total mass percentage of solids of 10% was used. This was considered a good value to start testing with, based on [Howard et al., 2012]. For the geotextile cone test, both strength tests and the hanging bag test, a total mass percentage of solids of 30% was used. This value was chosen as a representative value for practical purposes and was based on [Howard et al., 2012]. When a Ts% of 30% would be used, the corresponding density would be around 1200 kg/m^3, and based on literature [Shin and Oh, 2003], this is a good density value to fill geotextile tube with.

Procedure:

1. The desired quantity of original sample was weighed.
2. How much water should be added was calculated.
3. Required amount of water was added.
4. Mixture was stirred till a homogenous mixture was reached.

7.1.3. Jar tests

The jar tests were done according to an adjusted version of ASTMD2035-1 [ASTM International., 2019]. Here, the separation of solids and liquids were tested by putting a mixture of additive and iron ore tailings in a jar and then stirring the mixture for a certain amount of time. The stirring assured that the additive blended effectively with the iron ore tailings. After stirring, the mixture was put aside after which the separation of the solids and the liquids had taken place. By measuring the time at which the first floc, the first settlement and final settlement had taken place, an indication could be giving about the settling rate of the mixture. When the dosing was changed but the rest was kept the same, an ideal dosing of the additive could be found. Letting the slurry settle for two minutes and then measuring the boundary between the solids and liquids was considered to be a representative time for the settlement speed of the solids. This value was chosen at random but was kept constant for all the jar tests performed.

Procedure:

1. For a wet additive, the required volume of additive was measured in a beaker.
2. For a dry additive, The required additive was weighed in a bowl on a scale.
3. A beaker was filled with 1000 ml of sample.
4. The additive was added to the 1000 ml of sample.
5. The slurry was poured from one beaker into another beaker, This was done at least seven times. When more pouring was required for the first flocs to form, the required quantity was recorded.
6. After at least seven pours, The beaker was placed on the horizontal surface and the time til the first flocs settle was measured and recorded.
7. After two minutes of settling, the boundary between the liquids and solids inside the beaker, as shown in Figure 7.2 was measured and recorded.
8. The liquids were poured carefully into another beaker till the first solids also started to flow from the beaker.
9. The beaker was placed on a horizontal surface and the solids settled for at least 10 seconds.
10. Step 8 and 9 were repeated at least three times and the final volume of liquids was recorded.
11. The volume of liquids that was inside the second beaker was measured and recorded
12. The relative floc size of the solids and the relative clarity of the liquids were observed and recorded.
13. Tests were repeated with different doses and flocculants.

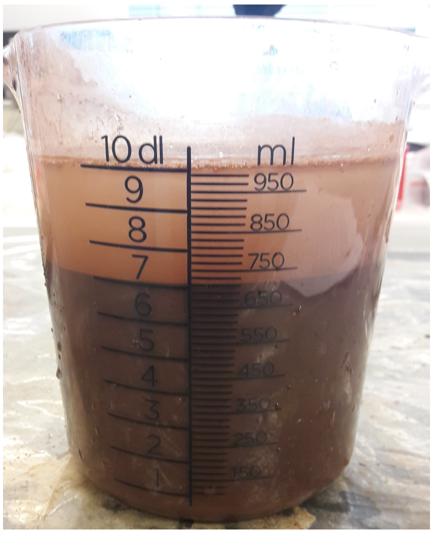


Figure 7.2: Boundary between solids and liquids

7.1.4. Geotextile cone test

With the cone tests, the dewatering behaviour of the iron ore tailings mixture combined with a geotextile could be tested. This test was done according to the Ten Cate Geotube RDT Test TenCate [2019]. They standardised their method to test the slurry-geotextile interaction. In this test, a mixture of additives and iron ore tailings were poured on a piece of geotextile fabric. Liquids were allowed to move through the geotextile while solids remained behind. The liquids that flowed through the fabric, the effluent, could be captured and chemically tested. The solids that remained on top of the geotextile fabric could be tested as well, but here the focus lay more on the geotechnical properties such as the Atterberg limits and final water content. When this test was repeated with different additives an ideal combination could be found which could be used for further testing. The decision to dewater the slurry for 15 minutes was based on initial testing with good flocculated mixtures. These mixtures had zero free water flow after 15 minutes, meaning that the dewatering process was finished. To compare the different mixtures, all mixtures were dewatered for 15 minutes. This way it was possible to compare the dewatering efficiency of the different mixtures.

Procedure:

1. Steps 1 till 6 from the jar tests were repeated.
2. After the seventh pour, The entire slurry was poured inside the geotextile cone as shown in Figure 7.3.
3. The slurry dewatered for 15 minutes.
4. Effluent was captured from the geotextile cone and the volume of the liquids after 15 minutes of dewatering was measured and recorded.
5. The clarity of liquids was observed and recorded.

Figure 7.3: Slurry poured into geotextile cone

7.1.5. Preparation of the samples for strength test

After the geotextile cone tests was performed and the flocculated iron ore tailings were dewatered for a minimum of 15 minutes the solids that remained on the geotextile were placed in a mould as shown in Figure 7.4. This way the solids could be placed inside a mould to enable further testing.

Procedure:

1. Solids were poured in the mould.
2. The mould was shaken so air entrappend in the solids could escape.
3. The surface was flattened with a spoon to ensure a horizontal surface.
4. The mould was labeled with the used mixture, the day it was placed and the exact time of completion of step 3.
5. The mould was placed in a regulated environment to ensure constant curing of the sample.

Figure 7.4: Pouring of dewatered slurry in mould.

7.1.6. Shear strength

Shear strength was measured by using the Torvane Shear Apparatus and was done according to ASTM D8121/D8121M – 19 ASTM International. [2018]. The aim of this test was to measure the shear strength of the stabilised iron ore tailings. This value was required to be able to model the consolidation behaviour of the geotextile tube when filled with the flocculated and stabilised iron ore tailings. This test was done at three different time moments. For the Torvane shear strength test, the mixture was cured for seven days. This was based on former research where cement was added to a slurry with the aim to measure the strength development of that slurry [Howard et al., 2012].

Procedure:

1. The vane size used was recorded.
2. The scale pointer was moved to read zero.
3. The handheld shear vane was pressed into the soil surface such that the axis of the post was normal to the soil surface.
4. The blades of the vane penetrated the soil until the face of the vane was touching the soil surface

5. The thumb of one hand was placed against the back of the vane. Only sufficient force to the axis of the handheld shear vane was applied to maintain contact between the face of the vane and the soil surface without penetrating the face of the vane into the soil surface.
6. The tips of the fingers of the free hand were placed on the edge of the knob. The vane was rotated clockwise using one smooth motion.
7. The value indicated by the pointer to the nearest half division in kilograms per square centimetre was recorded.
8. Steps one till seven were repeated on a total of three different locations.
9. The procedure was repeated on day two and day seven of curing.

7.1.7. Unconfined compressive strength

Unconfined compressive strength was measured by the pocket penetrometer. With this device it was possible to measure the unconfined compressive strength of the stabilised iron ore tailings and was done according to [Humboldt, 2015]. This test was done at three different time moments as well. With these values it was possible to say something about the unconfined compressive strength of the stabilised iron ore tailings while curing in a regulated environment. For the penetrometer test, the decision was made to let the mixture cure for seven days. This decision was based on former research where cement was added to a slurry with the aim to measure the strength development of that slurry [Howard et al., 2012].

Procedure:

1. The red ring down was slid against the handle of the penetrometer.
2. The penetrometer was placed onto the soil.
3. The penetrometer was pushed into the soil so the groove on the tip is even with the level of the soil.
4. The scale was read to determine unconfined compressive strength in kilograms per square centimetre. The reading was taken from the lower side of the red ring, closest to the handle.
5. The value was recorded and converted to kPa.
6. Steps one till five were repeated on a total of three different locations.
7. The procedure was repeated on day two and day seven of curing.

7.1.8. Hanging bag tests

While the jar test and the geotextile cone test were performed on small scale with a few litres of slurry in total, the hanging bag test simulated reality by scaling up the size of the test. This test was done according to ASTMD7701-11 [ASTM International., 2011]. The aim of this test was to simulate the dewatering behaviour of a slurry in a geotextile tube. The scale of this test made the test as close to reality as possible without an actual geotextile tube being filled with the material. For this test, a large geotextile bag was hung in a frame. Under the bag a pan was placed to capture the effluent. Next, the bag was filled with the mixture of additives and the iron ore tailings chosen from the cone test. This test could measure the quantity of liquids flowing out of the bag against time and the solids that remain behind.

Procedure:

1. The test set-up was prepared.
2. The bag was hung in the test set-up.

3. A collector pan was placed underneath the test set-up.
4. The slurry was prepared by mixing 40 litres of slurry with the pre-determined quantity flocculant and binder from the jar and the geotextile cone test for 1 minute by a mechanical mixer.
5. The mixture was poured Immediately in the geotextile container through a funnel attached to the geotextile container and the stopwatch was started.
6. The quantity of free water flow at predetermined time intervals (2, 5, 10, 15, 30, 45, 60 minutes followed by 24 and 48 hours was measured and recorded.
7. A sample was taken from the collector pan and tested the effluent on metals and metalloids.
8. A sample was taken from the geotextile container and tested for water content and dry solids content.

7.2. Limitations of testing

In this section the limitations of testing will be described. The structure will be according to the tests discussed in this chapter, starting with the jar test. Secondly, the limitations of the geotextile cone test will be discussed. This is followed by the limitations of both the strength tests and the hanging bag test.

7.2.1. Jar test

For the jar tests, it was assumed that the mineralogical composition and geo-mechanical properties of the iron ore tailings within the sample drums was constant for all drums. No further XRD analysis was performed besides the two performed by Clariant AG as given in appendix B. In addition, no further analysis was performed for the geo-mechanical properties of the samples besides the ones performed by AT-Milieu. Moreover, it was assumed that the pre-made flocculant solution was homogeneous and that the flocculants within the solution were active for the duration of the tests. Furthermore, it was assumed that the concentration of the coagulant within the coagulant solution was constant for the entire solution. For the volume of decanted liquids, it was assumed that only liquids were decanted and that no solids were found in the decanted liquids. Lastly, it was assumed that the flocculation process was 100 % successful meaning that all flocculants were used to form flocs.

7.2.2. Geotextile cone test

For the geotextile cone test, it was assumed that the mineralogical composition and geo-mechanical properties of the iron ore tailings within the sample drums was constant for all drums. In addition, it was assumed that the pre-made flocculant solution was homogeneous and that the flocculants within the solution were active for the duration of the tests. Furthermore, it was assumed that the concentration of the coagulant within the coagulant solution was constant for the entire solution. Moreover, it was assumed that the composition of the OPC was constant throughout the tests and that the cement was un-hydrated. For the geotextile fabric that was used for testing, it was assumed that the material properties as described in Table 6.3 were constant for all the geotextiles cones used for testing. Lastly, it was assumed that the flocculation process was 100 % successful meaning that all flocculants were used to form flocs.

7.2.3. Strength test

For the strength tests it was assumed that the accuracy of the torvane shear strength device was the same for all tests. In addition, it was assumed that the accuracy of penetrometer was constant for all tests performed. Furthermore, for the curing process it was assumed that the room temperature and humidity remained constant for the full curing cycle. Lastly, it was assumed that the disturbances in the surface as a result from previously performed measurements would not influence the curing process and the strength of the mixtures at future measurements.

7.2.4. Hanging bag

For the hanging bag the limitations of testing were almost the same as for the jar test and the geotextile cone test. It was assumed that the pre-made flocculant solution was homogenous and that the concentration of the coagulant within the coagulant solution was constant throughout the solution. In addition, it was assumed that the composition of the OPC was constant for all tests and that the cement was un-hydrated. Furthermore, it was assumed that the geotextile fabric from which the hanging bag was made, had the same properties, as shown in Table 6.3, for all the hanging bag test. Moreover, it was assumed that the humidity and the temperature within the test location remained constant throughout the entire dewatering period.

Part IV

Results and Discussion

8

Results

In this chapter the results from the all the experiments will be presented. They will be discussed in the order they were performed, as presented in Chapter 7, Methodology.

8.1. Flocculation behaviour

The flocculation behaviour was measured by performing the jar tests and the geotextile cone tests. The results from the jar tests are presented first, followed by the results from the geotextile cone tests.

8.1.1. Jar Test

A total of 37 jar tests were performed with four different flocculants. In Table 6.2, the different flocculants and their description are given. Per flocculant type, a series of 9 tests were performed, expect for the C-494 flocculant where 10 jar tests were performed. As mentioned in Chapter 7, Methodology, four properties were measured and noted during these tests: the boundary between solids and liquids after two minutes of settling, volume of liquids after decanting the liquids three times, the relative turbidity of the liquids and, if formed, the relative size of the floc.

Table 8.1: Results of the jar test for the A-150 flocculant type.

Mass (g)	V floc added (ml)	ppm floc added	boundary liquids/ solids after 2 min	Volume liquids after decanting three times	Relative turbidity of liquids	Relative flocsize	Other observations
1047.4	0.0	0.0	790	470	- -	None	
1047.4	1.1	11	710	530	- -	none	
1047.4	2.1	20	650	570	-	none	
1047.2	3.1	30	330	700	-	small	
1047.7	4.0	38	280	770	+	medium	
1047.6	5.2	49	250	800	-	large	Flocs instant settle
1047.5	6.1	58	270	800	-	large/very large	Flocs instant settle
1047.5	7.0	67	270	815	- -	large/very large	Flocs instant settle
1047.5	8.0	76	300	800	- -	very large	Flocs instant settle

In Table 8.1, the results from the jar tests for the flocculant type A-150 are given. The first three coloured columns, show the initial values used for the test. Furthermore, the four properties that were measured are shown. The concentration of flocculant added, ranges from 0 to almost 80 ppm. The boundary between the liquids and the solids, after two minutes of settling, ranges from 200 ml to 790 ml. The volume of the liquids after decanting three times, ranges from 470 ml to 850 ml. The relative turbidity of the liquids obtained by decanting the liquids three times is also given in the Table. The observed turbidity of the liquids is given relative to other concentrations of flocculant. The size of the flocs was also measured relative to other concentrations of flocculant. The largest flocs were formed when a concentration of

76.4 ppm of flocculant was added.

Table 8.2: Results of the jar test for the A-120 flocculant type.

Mass (g)	V floc added (ml)	ppm floc added	boundary liquids/ solids after 2 min	Volume liquids after decanting	Relative turbidity of liquids	Relative flocsize	Other observations
1047.4	0.0	0.0	790	470	- -	none formed	
1047.4	1.0	10	700	510	- -	none formed	
1047.2	2.0	19	430	650	-	small	
1047.2	3.0	29	270	770	-	medium	
1047.2	4.1	39	230	800	+	medium/large	Flocs instant settle
1047.4	5.1	49	220	850	-	large	Flocs instant settle
1047.4	6.0	57	200	850	-	very large	Flocs instant settle
1047.0	7.0	67	230	850	- -	very large	Flocs instant settle
1047.4	.0 8	76	230	840	- -	very large	Flocs instant settle

In Table 8.2, the results are given for the jar tests with the flocculant type A-120. Again, the first three coloured columns, show the values used for the test. In addition, the four properties that were measured are shown. For the A-120 flocculant, the highest settlement rate was at a flocculant concentration of 57.3 ppm of flocculant and the boundary between the liquids and the solids was at 200 ml. The largest volume of decanted liquids was found at three separate concentration of liquids namely at concentrations of 48.7, 58.3 and 66.9 ppm of flocculant added. The lowest relative turbidity was found at a flocculant concentration of 39.2 ppm and the largest flocs were formed at concentration higher than at least 57.3 ppm.

Table 8.3: Results of the jar test for the XD-5500 flocculant type.

Mass (g)	V floc added (ml)	ppm floc added	boundary liquids/ solids after 2 min	Volume liquids after decanting	Relative turbidity of liquids	Relative flocsize	Other observations
1047.4	0.0	0.0	790	470	- -	none formed	
1047.3	1.1	11	690	550	- -	none formed	
1047.4	2.0	19	660	600	-	none formed	
1047.5	3.1	30	300	770	-	small	
1047.7	4.0	38	270	780	+	small	
1047.3	5.1	49	230	850	-	medium	Flocs instant settle
1047.4	6.0	57	220	850	-	medium	Flocs instant settle
1047.6	7.1	68	230	850	- -	large	Flocs instant settle
1047.5	8.1	77	240	830	- -	large	Flocs instant settle

In Table 8.3, the results from the jar tests for the flocculant type XD-5500 are given. The first three coloured columns, show the values used for the specific test. For the XD-5500 test series, the four properties were measured as well. The fastest settlement was observed at a flocculant concentration of 57.3 ppm. The largest volume for decanted liquids was found at three separate flocculant concentrations, 48.7, 57.3 and 67.8 ppm. For these three flocculant concentrations the volume of decanted liquids was 850 ml. The relative turbidity was found to be the lowest at a flocculant concentration of 38.2 ml. The relative size of the flocs formed was largest at concentrations of both 67.8 and 77.3 ppm added.

In Table 8.4, the results from the jar tests for the flocculant type C-494 are given. Again, the first three columns, show the intital values of the slurry and the flocculant. The four properties described earlier in this chapter were also measured and observed for this flocculant type. For this test series, the fastest settlement was found to be at flocculant concentrations of 58.2, 66.8 and 86.9 ppm. The largest volume of decanted liquids was 870 ml at a flocculant concentration of both 58.2 and 66.8 ppm. The lowest relative turbidity was found a flocculant concentration of both 48.7 and 58.2 ppm. The largest flocs were formed at a concentration of 76.4 and 86.9 ppm flocculant.

Table 8.4: Results of the jar test for the C-494 flocculant type.

Mass (g)	V floc added (ml)	ppm floc added	boundary liquids/ solids after 2 min	Volume liquids after decanting	Relative turbidity of liquids	Relative flocsize	Other observations
1047.4	0.0	0.0	790	470	- -	none formed	
1047.1	1.0	10	650	570	- -	none formed	
1047.3	2.1	20	470	610	-	none/small	
1047.4	3.1	230	320	750	-	small	
1047.1	4.0	38	250	800	-	small	
1047.3	5.1	49	230	820	+	small/medium	
1047.3	6.1	58	200	870	+	medium	Flocs instant settle
1047.5	7.0	67	200	870	-	medium	Flocs instant settle
1047.5	8.0	76	220	850	- -	large	Flocs instant settle
1047.2	9.1	87	200	850	- -	large	Flocs instant settle

In Figure 8.1, the results from Table 8.1, 8.2, 8.3 and 8.4 are combined for the boundary between the liquids and the solids after two minutes of settling for different concentrations of added flocculant. This graph shows that when more flocculants are added to the mixture, the boundary between the liquids and the solids is lower, until a flocculant concentration of around 40 ppm. When concentrations higher than 40 ppm are used, the boundary between the liquids and the solids remains constant.

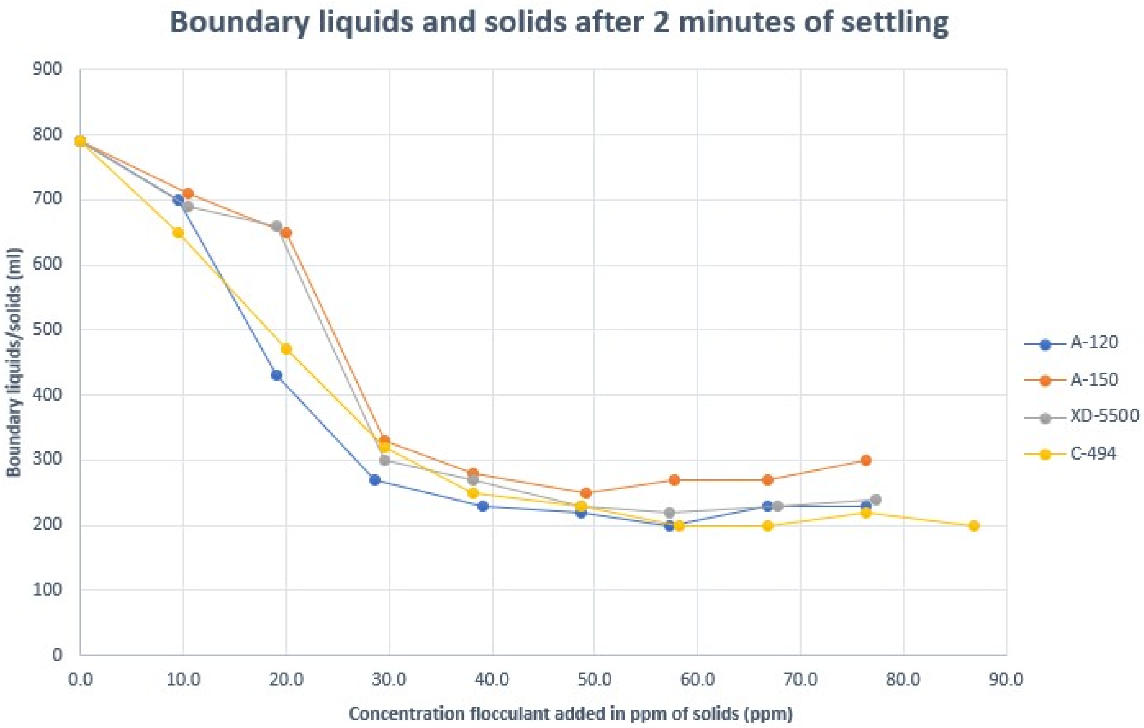


Figure 8.1: Combined graph of jar test for the boundary between liquids and solids for all flocculant types.

In Figure 8.2, the results from Table 8.1, 8.2, 8.3 and 8.4 are combined for the volume of decanted liquids. This graph shows that at higher concentrations of added flocculants, the volume also increases. The volume of liquids increases till flocculant concentrations of 50 ppm and higher are added. After this, the volume of decanted liquids remains constant.

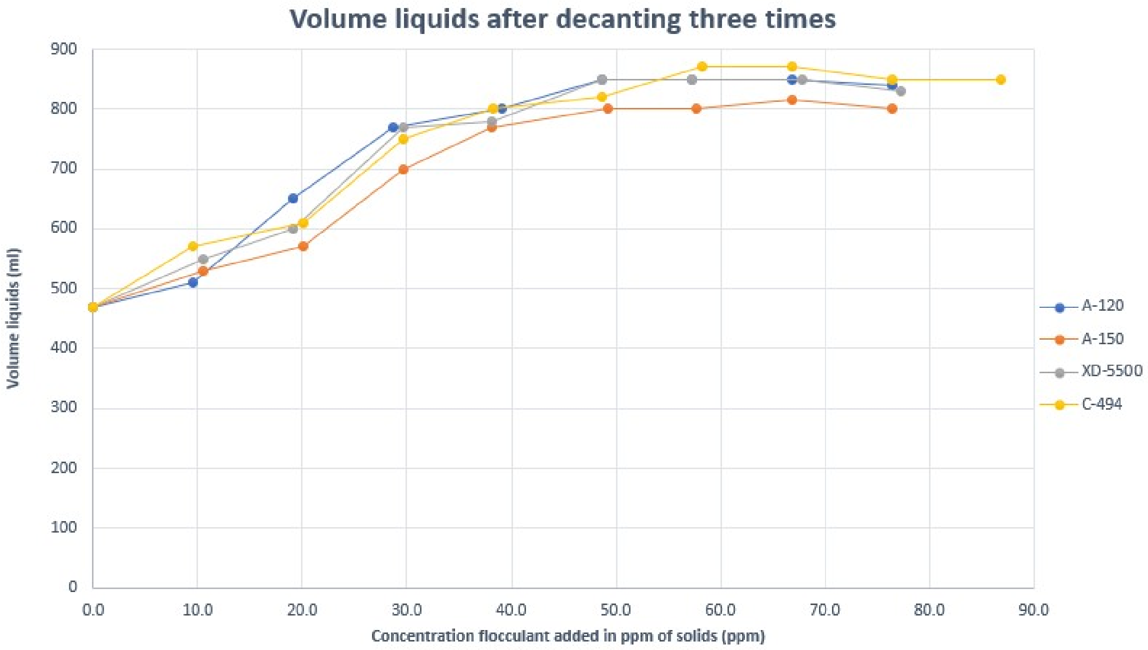


Figure 8.2: Combined graph of the jar test for decanted liquids for all flocculant types

8.1.2. Geotextile cone test

The geotextile cone test is a method to measure the efficiency of the flocculation process. Below, the results are given for the geotextile cone tests with different mixtures. A total of twelve geotextile cone tests were performed, all with different mixtures. The Tables below should be interpreted as follows. In the first column, the mixture is described. The different mixtures used included, only OPC (%OPC), a combination of flocculant and OPC (floc/% OPC) or a mixture of flocculant, coagulant and OPC (floc/coag/% OPC). The percentage OPC in the mixture indicates the mass percentage of OPC added based on the total mass of the slurry.

Table 8.5: Results from geotextile cone test for mine tailings + OPC mixtures.

Mixture	m slurry (g)	V floc (ml)	ppm floc	V coag (ml)	ppm coag	m OPC (g)	mass % OPC	V effluent (ml)	Observations
0% OPC	1155.9	0	0	0	0	0	0%	300	Solids found effluent
5% OPC	1158.2	0	0	0	0	57.3	4.95%	300	Solids found effluent
10% OPC	1158.2	0	0	0	0	115.5	9.972%	300	Solids found effluent
15% OPC	1159.2	0	0	0	0	173.2	14.94%	300	Solids found effluent

In Table 8.5, the results of the geotextile cone tests are given for the mixtures without added flocculants. The first eight columns show the initial values. For this tests series, the volume of flocculant and coagulant added is zero. The last two columns, coloured in white, show the results from the geotextile cone test As can be seen in this Table, the total volume of liquids that flowed through the geotextile cone after fifteen minutes is 300 ml for all the mixtures. However, solids were found in the effluent, meaning that they flowed through the geotextile as well.

In Table 8.6, the results are given for the geotextile cone test for the mixtures of mine tailings, flocculants and OPC. Again, the first eight columns show the initial values and the last columns show the results from the test. For this test series the largest volume of effluent after fifteen minutes of dewatering was 460 ml for the mixture with five mass percentage OPC added. In addition, it was observed that for all mixtures the flocs formed during flocculation were relatively large.

Table 8.6: Results from geotextile cone test for mine tailings + flocculant + OPC mixtures.

Mixture	m slurry (g)	V floc (ml)	ppm floc	V coag (ml)	ppm coag	m OPC (g)	mass % OPC	V effluent (ml)	Observations
floc/0% OPC	1159.2	50.0	144	0	0	0	0%	450	Large flocs
floc/5% OPC	1158.1	130.0	374.0	0	0	57.3	4.95%	460	Large flocs
floc/10% OPC	1159.9	160.0	459.8	0	0	115.5	9.958%	450	Large flocs
floc/15% OPC	1156.8	170.0	489.9	0	0	173.9	15.03%	390	Large flocs

Table 8.7: Results from geotextile cone test for mine tailings + flocculant + coagulant + OPC mixtures.

Mixture	m slurry (g)	V floc (ml)	ppm floc	V coag (ml)	ppm coag	m OPC (g)	mass % OPC	V effluent (ml)	Observations
floc/coag/0% OPC	1159.2	60.0	173	35.0	101	0	0%	500	Large flocs
floc/coag/5% OPC	1158.7	130.0	374.0	35.0	101	57.3	4.95%	400	Medium flocs
floc/coag/10% OPC	1158.9	160.0	460.2	35.0	101	115.5	9.966%	500	Large flocs
floc/coag/15% OPC	1158.9	170.0	489.0	35.0	101	173.3	14.95%	470	Large flocs

In Table 8.7, the results are given for the geotextile cone test for mixtures with mine tailings, flocculant, coagulant and OPC. The first eight columns, show the initial values with which the testing was performed. Again, the last two columns show the results from the tests. From this Table, it can be seen that the highest volume of effluent after fifteen minutes of dewatering was found at two separate mixtures namely the mixture with zero and the mixture with ten mass percentage of OPC added. Furthermore, it was observed that for three mixtures the flocs formed were relatively large while for one mixture, with five mass percentage OPC added, the flocs formed were medium sized. For this mixture the volume of effluent was also measured to be lowest of the four, with 400 ml after fifteen minutes of dewatering.

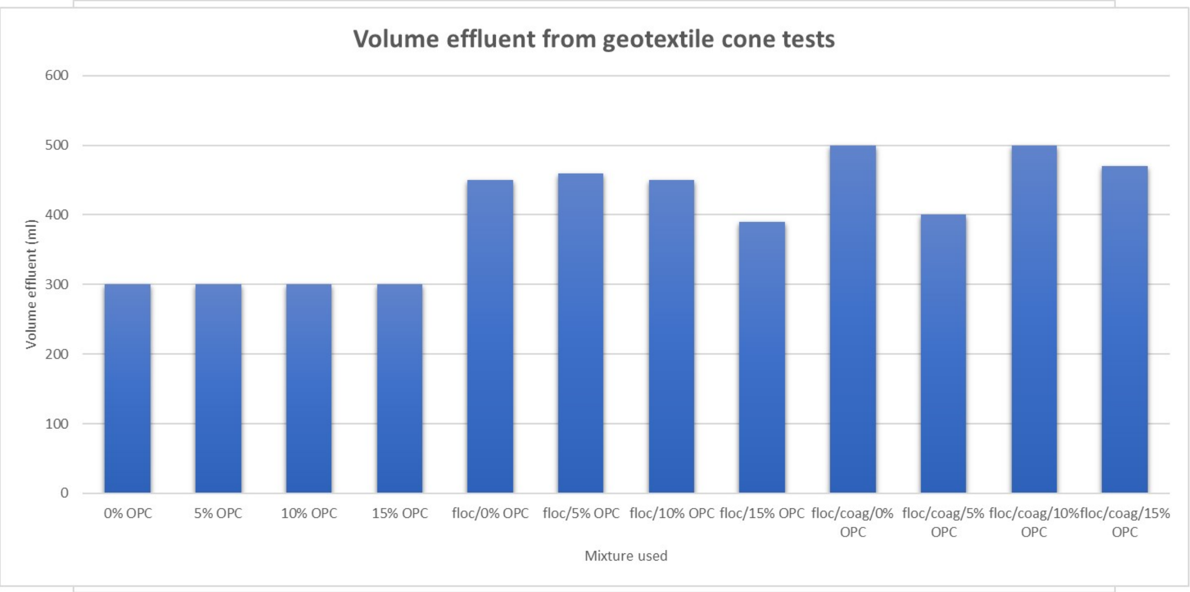


Figure 8.3: Volume of effluent from the geotextile cone test for different mixtures

In Figure 8.3, the results from Table 8.5, 8.6 and 8.7 are combined. This Figure shows that by adding the flocculant, the volume of effluent from the geotextile cone tests is higher than when no flocculant is added.

8.2. Stabilisation behaviour

The stabilisation behaviour of the iron ore tailings was measured by performing two types of strength tests. The Torvane shear strength test and the penetrometer test. In this section the results from these tests will be given. A total of 108 Torvane shear strength tests were performed to measure the shear strength. In addition, 108 tests were performed with the penetrometer to measure the unconfined compressive strength. In the first column, the mixture is described, similar to the tables used in the geotextile cone test section. The top row states the days of curing at which the testing was done, and the second number indicates which test it was, as a minimum of three strength tests were done each day. For example, day 2.3, means the test was done at day two of curing and it was the third measurement of the testing sequence.

8.2.1. Shear strength

In this section the results from the torvane shear strength test will be presented.

Table 8.8: Results from the torvane shear strength test for the mixture with iron ore tailings and OPC.

Shear strength (kPa)	day			day			day		
	1.1	1.2	1.3	2.1	2.2	2.3	7.1	7.2	7.3
0% OPC	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
5% OPC	0.40	0.40	0.60	2.0	1.8	1.6	9.0	8.0	11
10% OPC	1.6	2.0	2.0	3.6	3.4	3.2	30	33	35
15% OPC	1.8	2.0	2.6	4.4	4.6	5.0	36	34	34

In Table 8.8, the results are shown for the Torvane shear strength tester for mixtures of mine tailings and cement. It can be seen that the mixture with only mine tailings does not have any shear strength. The mixture with the highest shear strength was the mixture with 15% OPC added, although the mixture with only 10% added to the mine tailings almost had the same shear strength.

Table 8.9: Results from the torvane shear strength test for the mixture with iron ore tailings, flocculant and OPC.

Shear strength (kPa)	day			day			day		
	1.1	1.2	1.3	2.1	2.2	2.3	7.1	7.2	7.3
floc/0% OPC	0.40	0.40	0.40	0.40	0.40	0.40	2.0	2.0	2.0
floc/5% OPC	3.0	3.0	3.0	8.0	7.0	7.0	20	20	21
floc/10% OPC	9.0	7.0	8.0	19	20	20	35	47	48
floc/15% OPC	11	11	11	30	30	32	75	65	70

In Table 8.9, the results are shwon for the Torvane shear strength tester for mixtures of mine tailings, flocculant and cement. In this Table, the highest shear strengths are measured after seven days of curing for all mixtures. In addition, it should be noted that the mixture with zero percent of OPC, still has a shear strength of 2 kPa.

In Table 8.10, the results from the Torvane shear strength tester for mixtures of mine tailings, flocculant, coagulant and cement are shown. Again, the highest shear strengths are measured after seven days of curing. It should be noted that the mixture with only flocculant and coagulant still produces a shear strength of 2 kPa minimum.

Table 8.10: Results from the torvane shear strength test for the mixture with iron ore tailings, flocculant, coagulant and OPC.

Shear strength (kPa)	day			day			day		
	1.1	1.2	1.3	2.1	2.2	2.3	7.1	7.2	7.3
floc/coag/0% OPC	0	0	0	0.40	0.40	0.40	2.0	2.4	2.0
floc/coag/5% OPC	2.6	2.4	2.6	3.0	2.0	4.0	20	16	23
floc/coag/10% OPC	9.0	9.0	9.0	18	16	16	46	35	42
floc/coag/15% OPC	10	8.0	12	25	24	24	75	80	75

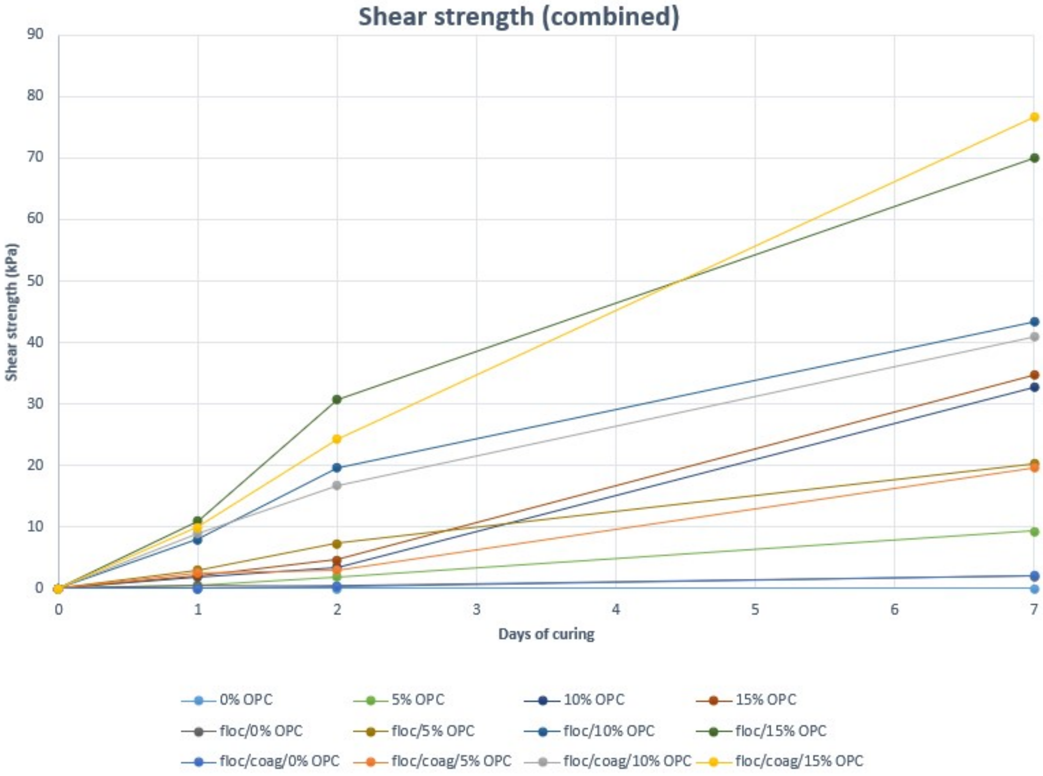


Figure 8.4: Combined graph for the shear strength of all mixtures.

In Figure 8.4, the combined results of the torvane shear strength test for all mixtures are presented. In this graph it is visible that the mixtures with flocculant and 15% OPC added have a higher shear strength than the mixtures with 15% OPC without flocculant.

8.2.2. Unconfined compressive strength

In this section the results from the penetrometer test will be presented.

Table 8.11: Results from the penetrometer test for the mixture with iron ore tailings and OPC.

Unconfined compressive strength (kPa)	day			day			day		
	1.1	1.2	1.3	2.1	2.2	2.3	7.1	7.2	7.3
0% OPC	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
5% OPC	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
10% OPC	0.00	0.00	0.00	1.25	1.25	1.25	8.75	5.00	6.75
15% OPC	0.00	0.00	0.00	1.25	1.25	2.50	10.0	12.5	13.8

In Table 8.11, the results are given for the penetrometer for mixtures of mine tailings and cement. This Table shows that both the mixtures with 0% and 5% OPC added do not have any unconfined compressive strength. The highest unconfined compressive strength was measured after seven days of curing, with the mixtures with 15% OPC added. Important thing to notice is the lack of unconfined compressive strength for all mixtures after one day of curing.

Table 8.12: Results from the penetrometer test for the mixture with iron ore tailings, flocculant and OPC.

Unconfined compressive strength (kPa)	day			day			day		
	1.1	1.2	1.3	2.1	2.2	2.3	7.1	7.2	7.3
floc/0% OPC	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
floc/5% OPC	0.00	0.00	0.00	0.00	0.00	0.00	2.50	2.50	2.50
floc/10% OPC	1.25	1.25	1.25	2.50	2.50	2.50	12.5	13.8	13.8
floc/15% OPC	2.50	2.50	2.50	5.00	5.00	5.00	26.3	27.5	27.5

In Table 8.12, the results are shown for the penetrometer for mixtures of mine tailings, flocculant and cement. What stands out is that the mixture of mine tailings and polymer does not produce any unconfined compressive strength, not even after seven days of curing. The highest unconfined compressive strength is achieved after seven days of curing for the mixture of mine tailings, flocculant and 15% of OPC. Another important thing to notice is the value of the mixture with mine tailings, flocculant and 5% OPC. This mixture does not have any unconfined compressive strength after two days of curing but has an unconfined compressive strength of 2.5 kPa after seven days of curing.

Table 8.13: Results from the penetrometer test for the mixture with iron ore tailings, flocculant, coagulant and OPC.

Unconfined compressive strength (kPa)	day			day			day		
	1.1	1.2	1.3	2.1	2.2	2.3	7.1	7.2	7.3
floc/coag/0% OPC	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
floc/coag/5% OPC	0.00	0.00	0.00	0.00	0.00	0.00	2.50	2.50	2.50
floc/coag/10% OPC	1.25	1.25	1.25	2.50	2.50	2.50	12.5	13.8	13.8
floc/coag/15% OPC	2.50	2.50	2.50	6.25	5.00	6.25	26.3	27.5	27.5

In Table 8.13, the results from the penetrometer for mixtures of mine tailings, flocculant, coagulant and cement are shown. For the mixture with zero mass percentage of OPC added, the unconfined compressive strength was measured to be zero at every tests moment. After seven days of curing, the mixture with fifteen mass percentage OPC has the highest unconfined compressive strength of the four mixtures. It should also be noted that the mixture with five mass percentage of OPC added, does not have any unconfined compressive strength for the first two days of curing. However, after seven days of curing, unconfined compressive strength was measured at 2.5 kPa.

In Figure 8.5 the results for the penetrometer test for all mixtures are presented. In this Figure, it is visible that the mixtures with flocculant and 15% OPC have the highest unconfined compressive strength. In addition, it can be seen that the unconfined compressive strength for the mixtures with only 15% OPC added and the mixture with flocculant, coagulant and 10% OPC added have almost the same unconfined compressive strength after seven days of curing. Important to notice is that four mixtures do not have any unconfined compressive strength after seven days of curing and are not visible in this Figure because they are all positioned on the x-axis. These four mixtures are: 0%OPC, 5%OPC, floc/0% OPC, floc/coag/0% OPC. In addition, the unconfined compressive strength of the mixtures floc/5% OPC and floc/coag/5% OPC are exactly the same for day one, day two and day seven of curing. Therefore, these two mixtures are exactly aligned and only the graph for the floc/coag/5% OPC mixture is visible.

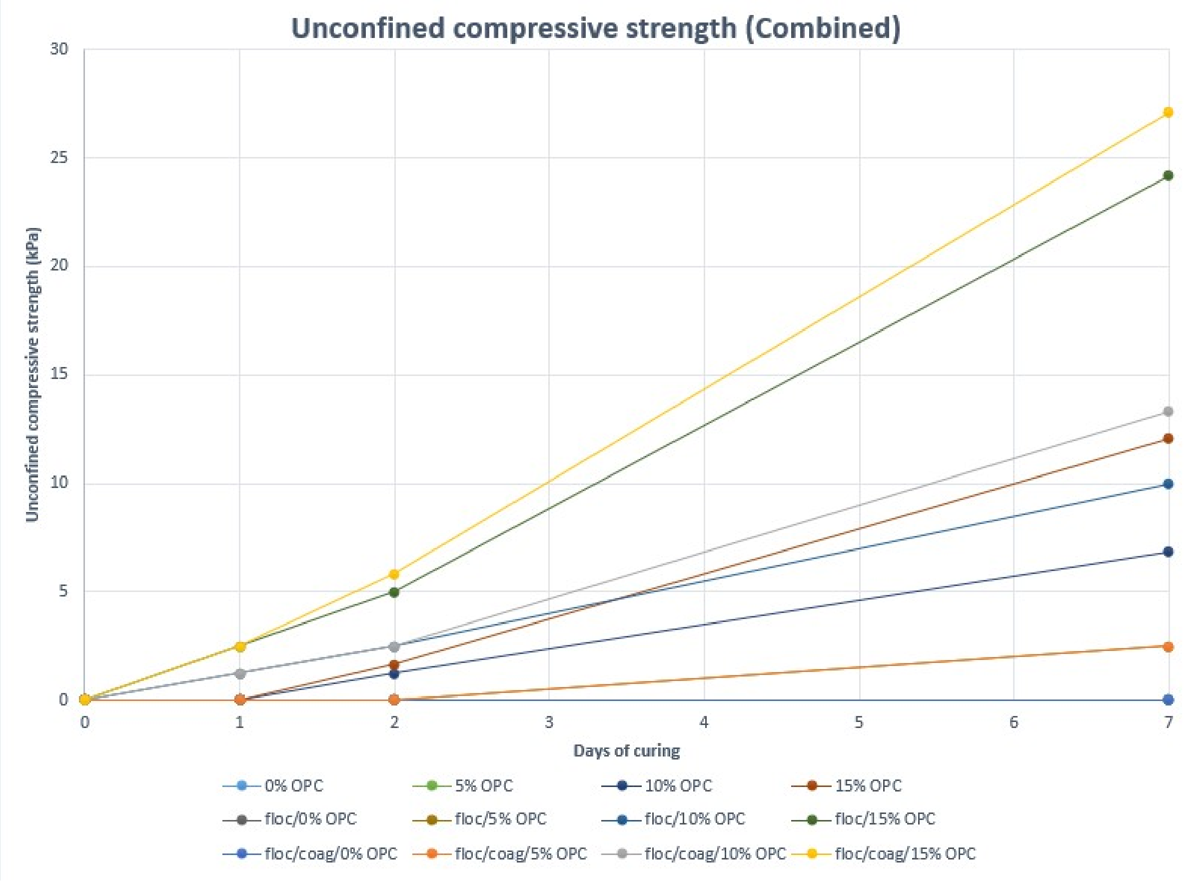


Figure 8.5: Combined graph for the unconfined compressive strength strength of all mixtures.

8.3. Dewatering behaviour

In the tables below the results are given for the hanging bag test for the five selected mixtures described in Chapter 7. Table 8.14 gives the results for the mixture with only mine tailings. In Table 8.15 the results are shown for the mixture with mine tailings and flocculant. Table 8.16 gives the results from the hanging bag test for the mixture with mine tailings, flocculant and 15% OPC. Table 8.17 shows the results for the mixture with mine tailings, flocculant and coagulant. The results for the mixture with mine tailings, flocculant, coagulant and 15% OPC is given in Table 8.18. It is visible that the total volume of effluent is different for every mixture. Comparing these final volumes would be difficult. Therefore, the decision was made to present the volume as percentages of the total volume instead of just volume of effluent. From the five mixtures that were tested, the highest volume of effluent after 48 hours of dewatering was measured at the mixture with both flocculant and coagulant added. The total volume of effluent for this mixture was 27 litre. In addition, it is visible that for all five mixtures the flow rate decreases as time passes.

Table 8.14: Results from the hanging bag for the mixture with mine tailings.

t (s)	V effluent (L)	% Effluent	Flow rate (m^3/s)
120	10.0	37%	8.33E-05
300	11.2	41%	6.67E-06
600	11.6	43%	1.33E-06
900	12.0	44%	1.33E-06
1800	13.0	48%	1.11E-06
2700	13.6	50%	6.67E-07
3600	14.0	52%	4.44E-07
86400	26.6	99%	1.52E-07
172800	27.0	100%	4.63E-09

Table 8.15: Results from the hanging bag for the floc/ 0% OPC mixture.

t (s)	V effluent (L)	% Effluent	Flow rate (m^3/s)
120	18.9	73%	1.58E-04
300	20.0	77%	6.11E-06
600	21.0	81%	3.33E-06
900	22.0	85%	3.33E-06
1800	23.0	88%	1.11E-06
2700	23.5	90%	5.56E-07
3600	23.9	92%	4.44E-07
86400	26.0	100%	2.54E-08
172800	26.0	100%	0.00

Table 8.16: Results from the hanging bag for the floc/ 15% OPC mixture.

t (s)	V effluent (L)	% Effluent	Flow rate (m^3/s)
120	15.0	65%	1.25E-04
300	17.1	74%	1.17E-05
600	18.9	82%	6.00E-06
900	19.9	87%	3.33E-06
1800	21.0	91%	1.22E-06
2700	21.9	95%	1.00E-06
3600	22.1	96%	2.22E-07
86400	23.0	100%	1.09E-08
172800	23.0	100%	0.00

Table 8.17: Results from the hanging bag for the floc/coag/ 0% OPC mixture.

t (s)	V effluent (L)	% Effluent	Flow rate (m^3/s)
120	20.1	74%	1.68E-04
300	21.4	79%	7.22E-06
600	22.8	84%	4.67E-06
900	23.3	86%	1.67E-06
1800	24.0	89%	7.78E-07
2700	24.6	91%	6.67E-07
3600	25.0	93%	4.44E-07
86400	27.0	100%	2.42E-08
172800	27.0	100%	0.00

Table 8.18: Results from the hanging bag for the floc/coag/ 15% OPC mixture.

t (s)	V effluent (L)	% Effluent	Flow rate (m^3/s)
120	18.0	78%	1.50E-04
300	19.4	84%	7.78E-06
600	20.4	89%	3.33E-06
900	21.1	92%	2.33E-06
1800	22.1	96%	1.11E-06
2700	22.3	97%	2.22E-07
3600	22.4	97%	1.11E-07
86400	23.0	100%	7.25E-09
172800	23.0	100%	0.00

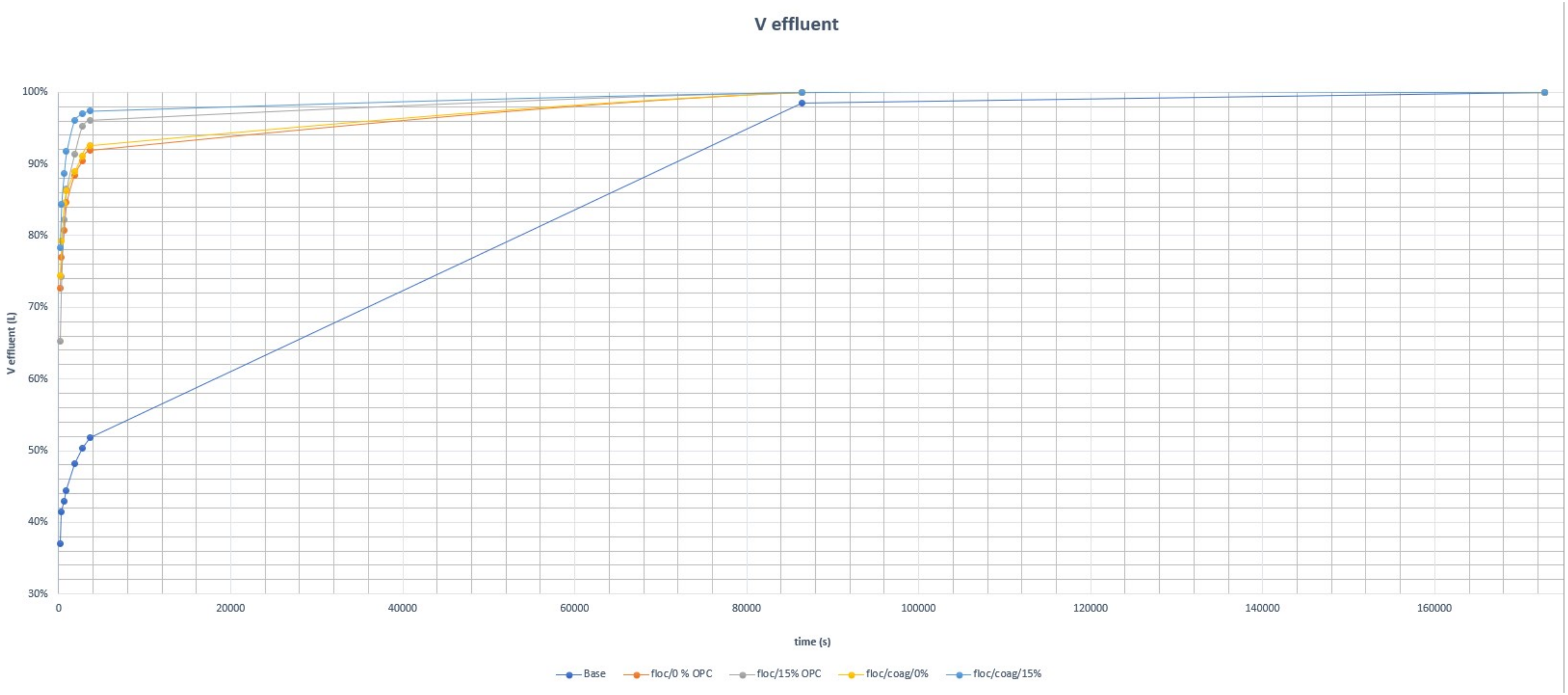


Figure 8.6: Percentage effluent of total effluent against time for the five mixtures.

Figure 8.6 shows the volume percentages of effluent from the hanging bag test for the five mixtures. It is visible in figure 8.6 that the mixture with only tailings has the lowest volume of effluent after one hour of dewatering. For this mixture only 52 % of the total volume of effluent had flowed from the bag. On the other hand, both the mixture floc/15% OPC and floc/coag/15% OPC have the highest volume of effluent after one hour of dewatering. For these mixtures, respectively 95% and 97% of the total volume of effluent had flowed from the hanging bag. In addition, it is visible that for four mixtures, the maximum volume of effluent was reached after 24 hours of dewatering. The mixture where this was not the case was the mixture with only mine tailings, where 98% of the total volume of effluent was measured after 24 hours of dewatering.

The final dry solids and water content for all the mixtures was also measured and are presented in Table 8.19. It can be seen in this table that the dry solids content for the mixture with only mine tailings is lower that for the other mixtures. In addition, the mixture with the highest dry solids content was measured to be the mixture with only flocculant added.

Table 8.19: Dry solids and water content of the mixture after the hanging bag test.

Mixture	Dry solids content (%)	
Base	61%	64%
floc/0 % OPC	66%	52%
floc/15% OPC	63%	60%
floc/coag/0%	64%	56%
floc/coag/15%	64%	57%

9

Discussion

In this chapter the results from chapter 8, Results will be discussed. The order of the discussion will be according to the order of testing described in chapter 7, Methodology. First, the results from the jar test will be discussed, followed by the results from the geotextile cone test and both strength tests. This chapter ends with the discussion of the results from the Hanging Bag test.

9.1. Jar test

The results from the jar test can be summarised in two graphs. These graphs were already presented in chapter 8, Results. In figure 8.1, the boundary between the liquids and the solids for different flocculant types and their corresponding concentrations is presented. In this graph, it is visible that most of the flocculant types flocculate the material with a similar rate, resulting in the boundary found at similar volumes for the corresponding flocculant concentrations. However, the flocculant type A-150 and XD-5500 show slightly higher volumes than the A-120 and the C-494 type. When the A-120 and the C-494 flocculant type are inspected closer, the boundary between the solids and the liquids is found at smaller volumes for the A-120 flocculant then for the C-494 flocculant type when small concentrations of flocculant are added (15 to 56 ppm). This means that the A-120 flocculant is better in flocculating the iron ore tailings at small concentrations that the C-494 flocculant type. In addition, from table A-120 and table C-494 it can be read that the relative size of the flocs for high flocculant concentrations (50 to 90 ppm) is very large for A-120 flocculant versus medium to large for the C-494 flocculant.

In figure 8.2, the volume of the decanted liquids for different flocculant types and their corresponding concentrations is presented. In this graph, the A-150 flocculant type has smaller volumes of decanted liquids than the A-120, the C-494 and the XD-5500 flocculant type. A smaller volume of decanted liquids means that the more water remains in the solids resulting in a higher water content. For the C-494 and the XD-5500 flocculant, the volume of decanted liquids is almost equal for increasing flocculant concentrations. This means that the volume of liquids that remain in between the solid particles is also equal for the two flocculant types, resulting in similar dewatering efficiency. For the A-120 flocculant type, the volume of decanted liquids for small flocculant concentrations (15 to 5 ppm) is higher that then the other three types, resulting in less liquids in between the solids particles. This means that the final water content will be lower for this flocculant type at these flocculant concentrations, resulting in a higher dewatering efficiency. However, for higher concentrations (50 to 80 ppm), the volume of the decanted liquids is roughly the same as for the XD-5500 flocculant type. In addition, the volume of decanted liquids for the C-494 flocculant is even higher than the volume when the A-120 and the XD-5500 flocculant types at concentrations higher than 55 ppm.

Both graphs show that the A-120 flocculant is more efficient in accelerating settlement at low flocculant concentrations and while the C-494 flocculant is more efficient at higher concentrations. Although the use of the C-494 flocculant is preferred at higher concentrations, the relative size of the flocs at these concentrations make its use less desirable. The aim of the flocculation process was to get the flocs as big as possible and the relative floc size of the flocs formed with the C-494 flocculant was large. However, the size of the flocs formed by the A-120 flocculant was very large. Therefore, it can be stated that the A-120 flocculant was the most optimal flocculant type for this type of tailings.

9.2. Geotextile cone test

The result from the geotextile cone tests show that when flocculant is added to the mixture, the volume of effluent after fifteen minutes of dewatering is higher than when flocculant is not added. These results show that adding flocculant increases the dewatering behaviour of the iron ore tailings. More important is the addition of coagulant to the mixture. As described in Chapter 3, Dewatering, the addition of coagulants should improve the dewatering behaviour of any slurry. For the iron ore tailings this holds true. Only for the floc/coag/5% OPC mixture this is not true. However, in Table 8.7, it is stated that the flocs for this mixture was relative medium sized. This could indicate that the flocculation of the particles was not hundred percent effective. In turn, this could have led to less effective dewatering, and thus less volume of the effluent because more liquids are retained between the solid particles. However, in general, the results show that by adding both flocculant and coagulant, the volume of effluent from the geotextile cone test will higher than when they are not added.

9.3. Strength tests

The results from both strength test can be summarised in two graphs. One for the shear strength and one for the unconfined compressive strength. These graphs are given in chapter 8, Results in figures 8.4 and 8.5. To be able to say something about the stabilisation behaviour of the mine tailings and the corresponding strength development both the results from the torvane shear strength tests and the penetrometer test should be combined and analysed together. For both graphs the highest values was achieved by the floc/coag/15%OPC mixture. The second highest value is achieved by the mixture with floc/15% OPC. This indicates that adding flocculant and OPC to the mixture will results in the mixture developing strength. Only adding OPC to the mixture will also develop strength but this strength will just be a bit more than half the strength developed by the mixture with both flocculant and OPC. Comparing the results for the mixtures with 10% OPC, the mixture with only OPC has developed more than 75% of the strength that was measured at the mixture with flocculant and OPC. However, the mixture with only 5% OPC will develop half the strength of the mixture with flocculant and OPC. Therefor, it can be stated that it will be impossible to predict what the strength will be of mixtures with percentages of OPC that lie between the ranges of 5 to 10% and 10 to 15% mass percentage OPC added.

In addition, it is visible that during the first seven days of curing, both the shear strength and the unconfined compressive strength will increase. However, it was not measured what the shear strength and the unconfined compressive strength was after more days of curing. So, it is impossible to predict what the final strength will be.

9.4. Hanging bag test

The results from the hanging bag test show that adding flocculant, coagulant and OPC, increases the dewatering rate of the mine tailings. The dewatering rate of the mixture with floc/coag/15% OPC was the highest followed by the mixture with floc/15% OPC as close second. The mixture with floc/coag/0% OPC dewatered faster than the mixture with floc/coag/0% OPC. When these observations are combined, it is evident that adding flocculants to a mixture will increase its dewatering rate. This dewatering rate can be increased even more by adding coagulant. This is conform to the literature stated in Section 3.2.1, Destabilisation.

Lastly, the addition of cement not only resulted in a development of strength but it also resulted in a increased dewatering rate.

However, a very important thing should be noted. The total volume of effluent for the mixtures with 15% OPC added was on average 3.5 litres less than for the mixtures without OPC added. This difference can be caused by a combination of factors. First, the total volume of slurry that went into the hanging bag was less for the mixtures with OPC added than for the other three. This was done to prevent the bags from overflowing, losing parts of the slurry. Another explanation for the effluent volume decrease is the hydration of OPC. During the hydratation reaction, cement reacts with water to form hydrates. The water that has reacted to form hydrates will not flow from the bag.

Part V

Conclusion and Recommendations

10

Conclusion

In this chapter the conclusions related to this research will be presented. The conclusion will be presented with regard to the research questions stated in the Introduction.

Based on the literature review performed for this research it can be concluded that mine tailings storage facilities are found all over the world, always situated close to a mine site. These tailings facilities contain a great range of minerals and metals. Its exact composition is very dependent on the type of ore, the associated gangue minerals and the method of extraction. In general, it can be stated that the gangue mineral will be a combination of silicates and various clay minerals. In addition, the exact quantity of the tailings within a tailings storage facility can vary a lot. Depending on the mineral that is extracted and its corresponding grade, the ratio of tailings produced per tonne of ore can differ from 1:1 (industrial minerals) to 500000:1 (gold ore at 2 ppm). Lastly, it is clear from the literature review that tailings storage facilities are dangerous constructions. For the wet tailings storage facilities, failure of the dams can result in large quantities of contaminated slurry flowing downstream. For the dry tailings storage facilities, failure of tailings dams can result in large landslides, potentially harming the local population and environment. Therefore, it is clear that risk mitigation measurements should be done to de-risk the tailings storage facilities.

The analysis performed showed that the mine tailings consist of mainly hematite and silicates. In addition, the analysis shows that the mine tailings were reasonable dry (Ts%: 77.4%). In order to pump these tailings inside a geotextile tubes this means that the mine tailings must be slurrified. The mine tailings were diluted to have a final dry solids content of 30%. With that propertie it was possible to pour the slurry successfully inside the geotextile bags. The applied properties enhancement and remediation methods will result in the slurry being flocculated with flocculants and stabilised and solidified using Ordinary Portland Cement. This will mean that the final dry solids content will be higher than the initial 30%. The dry solids content of the mixture after 48 hours of dewatering was measured between 61% and 66%. Therefore, it can be stated that the flocculation and remediation methods applied, results in a dry solids content higher than the initial dry solids content. In addition, the remediation of the mine tailings resulted in a strength development. This strength can be used to asses whether construction with geotextile tubes is possible.

The requirements for the mine tailings to pumped in geotextile tubes are as follows: the grainsize of the mine tailings should be large enough so that the grains do not pass the pores inside the geotextile tube. In addition, the mine tailings should be pumpable. This means that it should be possible to transport the tailings via pumping. This, to ensure that the tailings can be pumped inside a geotextile tube. And lastly, if contaminants are found in the mine tailings, remediation of the mine tailings via the stabilisation/solidification method needs to be possible.

The mine tailings with which the tests were performed had a grainsize that was too small for geotextile tubes to be used. In order to prevent the flushing out of particles, the mine tailings were flocculated, stopping the mine tailings from flowing through the pores. In addition, the slurry was made pumpable by slurrification, making it suitable for transport by pumping. Lastly, the contaminants in the mine tailings were stabilised and solidified by the Ordinary Portland Cement. This remediation method proved successful in stabilising the contaminants. In addition, the curing of the Ordinary Portland Cement meant that strength was developed. Based on the four statements above, it can be concluded that the enhanced properties of the mine tailings are suitable to be used as a resource for construction with geotextile tubes.

The flocculants and remediation method applied in this research is not guaranteed to work for other types of mine tailings without proper research. For this research to be applied on a wider range of mine tailings a few things should be known about the target mine tailings. The mineral composition and contaminants that require remediation should be investigated first. With this, it is possible to choose the correct flocculant and the remediation method that suits the contaminants the best. When the remediation method also develops strength, the possibility of building constructions with the geotextile tubes becomes possible. Therefore, it can be concluded that the findings of this research can be used as basis from which suitable flocculation and remediation methods can be selected.

It is now possible to answer the main research question asked in Chapter 1, Introduction.

Are mine tailings a sustainable alternative resource for construction with geotextile tubes?

Based on this research, it can be concluded that mine tailings are a sustainable alternative resource for construction with geotextile tubes. In addition, the hypothesis stated in the Introduction was tested with this research and is found to be true.

11

Recommendations

Based on this research and the conclusions drawn in Chapter 10, Conclusion, it is now possible to give recommendations and how to move forward.

First, the test performed in this research were all performed in the laboratory. How the mine tailings and the geotextile tube will react when larger scales are applied, is difficult to predict. Therefore, further testing should be done on a larger scale, to investigate the behaviour of the mine tailings and the geotextile tube.

Secondly, as described in Section 7.2, Limitations of testing, there were a couple of assumptions made during testing. These assumptions should be checked and validated for different conditions.

Thirdly, the different geotextiles available on the market were not investigated in this research. Further investigations on the geotextile fabric, opening size and general strength of the material should be performed. With that, it should be possible to choose a geotextile that works the best in combination with the flocculant and Ordinary Portland Cement. In this research, the flocculant dosing was made optimal for a specific opening size. When different opening sizes would be used, the required floc size and therefore the dosing of the flocculant will be expected to change.

Fourthly, proper design of the geotextile tubes must be done in order to use a geotextile tube as a construction element. In this research, the design of the geotextile tube was not considered. However, when the design of the tube would be considered, the requirements of the material with which the tube is filled, might change. This could result in different dosing for the flocculants and Ordinary Portland Cement.

Fifthly, leaching of the contaminants is also not investigated in this research. Even though, the concentration of the contaminations in the effluent were measured to be lower than in the mine tailings, they could still be mobilised by leaching. To measure this, leaching tests should be performed. With those tests, the potential mobilisation of the stabilised contaminations can be measured.

Sixthly, it may be interesting to investigate whether reprocessing of the tailings before pumping them in geotextile tubes can be economically and technically feasible. A big obstacle to reprocess tailings inside a tailings storage facility is the necessity to rehandle the material. This will cost money and is, in most cases, undesirable. However, to fill the geotextile tubes, the tailings must be pumped from the tailings storage facilities to the side where the geotextile tubes are placed. Therefore, the tailings must be rehandled anyway. Thus, it might be interesting to investigate reprocessing of the tailings, since the material is already being transported.

Finally, the flocculation process should be performed carefully. When too much shear is applied during mixing, the flocs will break because they are fragile. The flocculation process in this research was not 100% perfect. To get perfect flocculation specially designed machinery is needed. With such a machine the flocculation process might be more effective. Therefore, the effect of using a machine to flocculate the mine tailings should be investigated.

Bibliography

Abdelouas, A., Lutze, W., and Nuttall, H. (2006). Uranium: mineralogy, geochemistry and the environment. *Elements*, 2:335–341.

Achterberg, E. P., Braungardt, C., Morley, N. H., Elbaz-Poulichet, F., and Leblanc, M. (1999). Impact of Los Frailes mine spill on riverine, estuarine and coastal waters in Southern Spain. *Water Research*, 33(16):3387–3394.

Argane, R., Benzaazoua, M., Hakkou, R., and Bouamrane, A. (2015). Reuse of base-metal tailings as aggregates for rendering mortars: Assessment of immobilization performances and environmental behavior. *Construction and Building Materials*, 96:296–306.

ASTM International. (2011). D7701 − 11 Standard Test Method for Determining the Flow Rate of Water and Suspended Solids from a Geotextile Bag. Technical report, ASTM International., West Conshohocken, Pa.

ASTM International. (2018). D8121/D8121M − 19 Standard Test Method for Approximating the Shear Strength of Cohesive Soils by the Handheld Vane Shear Device. Technical report, ASTM International., West Conshohocken, Pa.

ASTM International. (2019). D2035-19 Standard Practice for Coagulation-Flocculation Jar Test of Water. Technical report, ASTM International., West Conshohocken, Pa.

Bahrami, A., Hosseini, M. R., and Razmi, K. (2007). An investigation on reusing process water in gold cyanidation. *Mine Water and the Environment*, 26:191–194.

Baja Mare Task Force (2000). The Cyanide Spill at Baia Mare, Romania. Technical report, UNEP/OCHA, Geneva.

Besseling, E. and Ekkelenkamp, H. H. M. (2017). *Handboek Baggerspecie in geotextiele tubes.* SBR Curnet, Delft.

Bethell, P. J. (2013). Economic factors affecting coal preparation: plant design worldwide and case studies illustrating economic impact. In *The coal handbook*, chapter 14, pages 445–466. Woodhead Publishing Limited.

Bloodworth, A. and Gunn, G. (2012). Minerals: The future of the global minerals and metals sector: Issues and challenges out to 2050. *Geosciences*, 15:90–96.

British Broadcasting Corporation (2019). Brazil dam disaster firms to face criminal charges. https://www.bbc.com/news/world-latin-america-49772778.

Chen, Y., Zhang, Y., Chen, T., Zhao, Y., and Bao, S. (2011). Preparation of eco-friendly construction bricks from hematite tailings. *Construction and Building Materials*, 25(4):2107–2111.

Davies, M. P. and Martin, T. E. (2000). Upstream constructed tailings dams - A review of the basics. In *Tailings and Mine Waste*, volume 7, pages 3–16. Balkema, Rotterdam.

Dermatas, D. and Meng, X. (2003). Utilization of fly ash for stabilization / solidification of heavy metal contaminated soils. *Engineering Geology*, 70:377–394.

Evanko, C. and Dzombak, D. A. (1997). Remediation of Metals-Contaminated Solid and Groundwater. Technical report, GWRTAC, Pittsburgh, PA.

Fernandes, H. M., Franklin, M. R., and Veiga, L. H. (1998). Acid rock drainage and radiological environmental impacts. *Waste Management*, 18:169–181.

Galvão, J. L. B., Andrade, H. D., Brigolini, G. J., Peixoto, R. A. F., and Mendes, J. C. (2018). Reuse of iron ore tailings from tailings dams as pigment for sustainable paints. *Journal of Cleaner Production*, 200:412–422.

Giurco, D., Prior, T., Mudd, G. M., Mason, L. M., and Behrisch, J. C. (2010). Peak Minerals in Australia: a review of changing impacts and benefits. Technical report, Institute for Sustainable Futures University of Technology, Sydney, Australia.

Google. (n.d.). Location of samples taken from Vargem Grande mine. https://www.google.com/maps/place/Belo+Horizonte,+State+of+Minas+Gerais,+Brazil/@-20.1764757,-43.8755897,4090m/data=!3m1!1e3!4m5!3m4!1s0xa690cacacf2c33:0x5b35795e3ad239
19.9166813!4d-43.9344931.

Hoffmann, K. and Huculak-Maczka, M. (2012). The utilization possibility of waste lignite as a raw material in the process of obtaining humic acids preparations. *Polish Journal of Chemical Technology*, 14(4):1–6.

Hogg, R. (2000). Flocculation and dewatering. *International Journal of Mineral Processing*, 58:223–236.

Howard, I. L., Carruth, W. D., Sullivan, W. G., Bilberry, A. C., Cost, T., Badran, W. H., and Jordan, B. D. (2012). Development Of An Emergency Construction Material For Disaster Recovery. Technical report, Mississippi State University, Oak Ridge, Tennessee.

Howard, I. L., Vahedifard, F., Williams, J. M., and Timpson, C. (2018). Geotextile tubes and beneficial reuse of dredged soil : applications near ports and harbours. *Proceedings of the Institution of Civil Engineers – Ground Improvement*, 171:244–257.

Howard, L. and Trainer-tencate, E. (2011). Use of Geotextile and Geomembrane Tubes to Construct Temporary Walls in a Flooded Area. Technical report, Mississippi State University, Oak Ridge, Tennessee.

Huang, X., Ranade, R., Ni, W., and Li, V. C. (2013). Development of green engineered cementitious composites using iron ore tailings as aggregates. *Construction and Building Materials*, 44:757–764.

Humboldt (2015). Product manual Pocket Style Soil Penetrometer. Technical report, Humboldt, Schiller Park, Illiniois.

Igiri, B. E., Okoduwa, S. I. R., Idoko, G. O., Akabuogu, E. P., Adeyi, A. O., and Ejiogu, I. K. (2018). Toxicity and Bioremediation of Heavy Metals Contaminated Ecosystem from Tannery Wastewater: A Review. *Journal of Toxicology*, pages 1–16.

Jankovic, A. (2015). Developments in iron ore comminution and classification technologies. In *Iron Ore: Mineralogy, Processing and Environmental Sustainability*, chapter 8, pages 251–282. Elsevier Ltd., Amsterdam.

Kefeni, K., Msagati, T. A. M., and Mamba, B. (2017). Acid mine drainage: Prevention, treatment options, and resource recovery: A review. *Journal of Cleaner Production*, 151:475–493.

Kessler, W. and Müller, G. (1988). Minor and Trace-element Data of Iron Oxides from Iron-formations of the Iron Quadrangle , Minas Gerais, Brazil. *Mineralogy and Petrology*, 39:245–250.

Kim, H., Jamin, J. C., Won, M., Sung, H., and Lee, J. (2013). A study on the behavior of dredged soil stratified in a transparent geobag. *Advances in Structural Engineering and Mechanics*, pages 2980–2987.

Kincaid, T. (1997). *Alternatives for Ground Water Cleanup.* National Academy Press, Washington, D.C.

Kopsick, D. A. and Angino, E. E. (1981). Effect Of Leachate Solutions From Fly And Bottom Ash. *Journal of Hydrology*, 54:341–356.

Kossoff, D., Dubbin, W. E., Alfredsson, M., Edwards, S. J., Macklin, M. G., and Hudson-Edwards, K. A. (2014). Mine tailings dams: Characteristics, failure, environmental impacts, and remediation. *Applied Geochemistry*, 51:229–245.

Kruyt, H. R. (1952). *Colloid Science.* Elsevier Publishing Company, Amsterdam.

Lawson, C. . R. (2016). Geotextiles in marine engineering. *Geotextiles*, pages 435–482.

Leshchinsky, D., Ling, H. I., and Gilbert, P. A. (1996). Geosynthetic Tubes For Confining Pressurized Slurry: Some Design Aspects. *Journal of Geotechnical Engineering*, 122(8):682–690.

Lottermoser, B. G. (2010). *Mine Wastes: Characterization, Treatment and Environmental Impacts.* Springer Science & Business Media, Berlin, 3th edition.

Lu, L. and Ishiyama, O. (2015). Iron ore sintering. In *Iron Ore: Mineralogy, Processing and Environmental Sustainability*, chapter 14, pages 395–433. Elsevier Ltd, Amsterdam.

Mahmood, A. A. and Mulligan, C. N. (2007). Investigation of the use of mine tailings for unpaved road base construction. *Proceedings of the Annual International Conference on Soils, Sediments, Water and Energy:*, 12(11):105–115.

Mohri, Y., Matsushima, K., Yamazaki, S., Lohani, T. N., Tatsuoka, F., and Tanaka, T. (2009). New direction for earth reinforcement: Disaster prevention for earthfill dams. *Geosynthetics International*, 16(4):246–273.

Mudder, T. I., Botz, M. M., and Smith, A. (1991). *Chemistry and Treatment of Cyanidation Wastes.* Mining Journal Books Ltd, London.

Nilsson, Å. (2001). Safe dam constructions. *Safe Tailings Dam Constructions*, pages 20–29.

NZRMCA (2004). Alkali Content of Concrete Mix Water and Aggregates. Technical report, NZRMCA, Tawa, New Zealand.

Oak Ridge National Laboratory (1997). Spent Nuclear Fuel and Radioactive Waste Inventories, Projections, and Characteristics. Technical report, U.S. Department of Energy, Oak Ridge, Tennessee.

OECD/IAEA (1999). Environmental activities in uranium mining and milling. Technical report, Nuclear Energy Agency, Paris.

Oil Sands Magazine (2019a). In-Situ Bitumen Extraction. https://www.oilsandsmagazine.com/technical/in-situ.

Oil Sands Magazine (2019b). Oil Sands 101: Process Overview. https://www.oilsandsmagazine.com/technical/in-situ.

Onuaguluchi, O. and Eren, Ö. (2016). Reusing copper tailings in concrete: Corrosion performance and socioeconomic implications for the Lefke-Xeros area of Cyprus. *Journal of Cleaner Production*, 112:420–429.

Osbaeck, B. and Johansen, V. (1989). Particle Size Distribution and Rate of Strength Development of Portland Cement. *Journal of the American Ceramic Society*, 72(2):197–201.

Pichler, T., Hendry, M. J., and Hall, G. E. M. (2001). The mineralogy of arsenic in uranium mine tailings at the Rabbit Lake in-pit facility, northern Saskatchewan, Canada. *Environmental Geology*, 40(4-5):495–506.

Ramraj, R. (2001). The Omai disaster in Guyana. Technical report, Winston-Salem State university, Winston-Salem, NC.

Ranade, R. and Li, V. (2013). Feasibility Study of Developing Green ECC Using Iron Ore Tailings Powder Cement Replacement. *Journal of Materials in Civil Engineering*.

Rattanakawin, C. (2005). Aggregate size distributions in sweep flocculation. *Journal Science Technology*, 5(27):1095–1101.

Rauche, H. A. M., Fulda, D., and Schwalm, V. (2001). Tailings and Disposal Brine Reduction – Design Criteria for Potash Production in the 21 st Century. In *Tailings and Mine Waste '01*, pages 85–91. Balkema, Rotterdam.

Rimstidt, J. D. and Vaughan, D. J. (2003). Pyrite oxidation: A state-of-the-art assessment of the reaction mechanism. *Geochimica et Cosmochimica Acta*, 67(5):873–880.

Rumer, R. R. and Ryan, M. E. (1995). *Barrier Containment Technologies for Environmental Remediation Applications*. John Wiley, Hoboken, New Jersey.

Rutherford, P. M., Dudas, M. J., and Samek, R. A. (1994). Environmental impacts of phosphogypsum. *Science of the Total Environment, The*, 149(1-2):1–38.

Schweinfurth, S. P. (2009). An Introduction to Coal Quality. In Pierce, B. S. and Dennen, K. O., editors, *The National Coal Resource Assessment Overview*, chapter C, pages 1–16. U.S. Geological Survey Professional Paper, Reston, Virginia.

Segura, F. R., Nunes, E. A., Paniz, F. P., Paulelli, A. C. C., Rodrigues, G. B., Braga, G. U. L., Filho, W. R. P., Barbosa, F., Cerchiaro, G., Silva, F. F., and Batista, B. L. (2016). Potential risks of the residue from Samarco's mine dam burst (Bento Rodrigues, Brazil). *Environmental Pollution*, 218:1–13.

Shin, E. C. and Oh, Y. I. (2003). Analysis of geotextile tube behaviour by large-scale field model tests. *Geosynthetics International*, 10(4):134–141.

Small, C. C., Cho, S., Hashisho, Z., and Ulrich, A. C. (2015). Emissions from oil sands tailings ponds : Review of tailings pond parameters and emission estimates. *Journal of Petroleum Science and Engineering*, 127:490–501.

Spier, C. A., de Oliveira, S. M. B., Rosière, C. A., and Ardisson, J. D. (2008). Mineralogy and trace-element geochemistry of the high-grade iron ores of the Águas Claras Mine and comparison with the Capão Xavier and Tamanduá iron ore deposits, Quadrilátero Ferrífero, Brazil. *Mineralium Deposita*, 43:229–254.

Stokes, G. G. (1880). *Mathematical and physical papers Volume 5*. Cambridge University Press, Cambridge.

Tajudin, S. A. A., Azmi, M. A. M., and Nabila, A. T. A. (2016). Stabilization/Solidification Remediation Method for Contaminated Soil: A Review. *IOP Conference Series: Materials Science and Engineering*, 136(1).

Tarleton, E. S. and Wakeman, R. J. (2007). Pretreatment of suspensions. *Solid/Liquid Separation*, pages 126–151.

TenCate (2019). TenCate Geotube RDT Test. Technical report, TenCate, Commerce, Georgia.

UNEP and IFA (2001). *Environmental Aspects of Phosphate and Potash Mining*. United Nations Environment Programme, Paris.

U.S. Environmental Protection Agency (1992). Vitrification Technologies for Treatment of Hazardous and Radioactive Waste. Technical report, U.S. Environmental Protection Agency, Washington, D.C.

Van Niekerk, H. J. and Viljoen, M. J. (2005). Causes and consequences of the merriespruit and other tailings-dam failures. *Land Degradation and Development*, 16(2):201–212.

Vaughan, D. J. and Craig, J. R. (1978). *Mineral Chemistry of Metal Sulfides*. Cambridge University Press, Cambridge.

Wills, B. A. and Napier-Munn, T. (2007). *Wills' Mineral Processing Technology: An Introduction to the Practical Aspects of Ore Treatment and Mineral Recovery*. Elsevier, Amsterdam, 7th edition.

WISE Uranium Project (2019). Chronology of major tailings dam failures. https://www.wise-uranium.org/mdaf.html.

World Health Organization (2004). Hydrogen Cyanide and Cyanides: Human Health Aspects. Technical report, World Health Organization, Geneva.

Yamashita, M., Yamashita, M., Suzuki, M., Hirai, H., and Kajigaya, H. (2001). Iontophoretic delivery of calcium for experimental hydrofluoric acid burns. *Critical Care Medicine*, 29(8):1575–1578.

Younger, P. L. (2004). Environmental impacts of coal mining and associated wastes: A geochemical perspective. *Geological Society Special Publication*, 236:169–209.

Yution, Y. (2016). Dewatering Behaviour of Fine Oil Sands Tailings: An Experimental Study. Technical report, Delft University of Technology, Delft.

Zhang, H. (2011). Cement. In *Building Materials in Civil Engineering*, chapter 4, pages 46–423. Woodhead Publishing, Cambridge.

Zhao, S., Fan, J., and Sun, W. (2014). Utilization of iron ore tailings as fine aggregate in ultra-high performance concrete. *Construction and Building Materials*, 50:540–548.

Zheng, B., Zhang, D., Liu, W., Yang, Y., and Yang, H. (2019). Use of basalt fiber-reinforced tailings for improving the stability of tailings dam. *Materials*, 12(8):1–15.

A

Certificate of analysis of iron ore tailings

eurofins
— analytico®

AT MilieuAdvies B.V.
T.a.v. Niels van Wely
Lopikerplein 2A
2871 AN SCHOONHOVEN
NETHERLANDS

Analysecertificaat

Datum: 17-Jul-2019

Hierbij ontvangt u de resultaten van het navolgende laboratoriumonderzoek.

Certificaatnummer/Versie	2019100765/1
Uw project/verslagnummer	AT19069
Uw projectnaam	Netics analyses 2019
Uw ordernummer	
Monster(s) ontvangen	09-Jul-2019

Dit certificaat mag uitsluitend in zijn geheel worden gereproduceerd.
De analyse resultaten hebben alleen betrekking op het beproefde object.

De grondmonsters worden tot 4 weken na datum ontvangst bewaard en watermonsters tot 2 weken na datum ontvangst. Zonder tegenbericht worden de monsters nadien afgevoerd.
Indien de monsters langer bewaard dienen te blijven verzoeken wij U dit exemplaar uiterlijk 1 werkdag voor afloop van de standaardbewaarperiode ondertekend aan ons te retourneren. Voor de kosten van het langer bewaren van monsters verwijzen wij naar de prijslijst.

Bewaren tot:
Datum: Naam: Handtekening:

Wij vertrouwen erop uw opdracht hiermee naar verwachting te hebben uitgevoerd, mocht U naar aanleiding van dit analysecertificaat nog vragen hebben verzoeken wij U contact op te nemen met de afdeling Verkoop en Advies.

Met vriendelijke groet,

Eurofins Analytico B.V.

Ing. A. Veldhuizen
Technical Manager

eurofins — analytico®

Analysecertificaat

Uw project/verslagnummer	AT19069	Certificaatnummer/Versie	2019100765/1
Uw projectnaam	Netics analyses 2019	Startdatum	10-Jul-2019
Uw ordernummer		Rapportagedatum	17-Jul-2019/11:02
		Bijlage	A,B,C
Monsternemer		Pagina	1/4
Monstermatrix	Waterbodem (AS3000)		

	Analyse	**Eenheid**	**1**	**2**	**3**	**4**
	Bodemkundige analyses					
S	Droge stof	% (m/m)	79.3	75.6	77.9	75.6
S	Organische stof	% (m/m) ds	1.1	1.4	1.9	2.4
Q	Gloeirest	% (m/m) ds	97.6	97.7	97.0	96.7
	Calciet (CaCO3)	% (m/m) ds	<0.5	<0.5	<0.5	<0.5
	Calciet (CaCO3)	g/kg ds	<5.0	<5.0	<5.0	<5.0
Q	Korrelgrootte > 2 mm	% (m/m) ds	<0.1	<0.1	<0.1	<0.1
S	Korrelgrootte < 2 µm	% (m/m) ds	19.7	12.2	15.2	12.5
Q	Korrelgrootte < 2000 µm	% min. delen	100.0	100.0	100.0	100.0
Q	Korrelgrootte < 1000 µm	% min. delen	99.2	98.3	98.6	97.8
Q	Korrelgrootte < 500 µm	% min. delen	98.1	97.4	97.7	95.5
Q	Korrelgrootte < 250 µm	% min. delen	96.8	96.2	96.8	92.2
Q	Korrelgrootte < 125 µm	% min. delen	95.5	94.5	96.0	88.5
Q	Korrelgrootte < 63 µm	% min. delen	88.6	87.4	92.5	80.7
Q	Korrelgrootte < 50 µm	% min. delen	87.4	85.9	91.0	78.2
Q	Korrelgrootte < 32 µm	% min. delen	85.7	82.0	86.6	73.3
Q	Korrelgrootte < 16 µm	% min. delen	79.6	73.7	70.7	61.5
Q	Korrelgrootte < 8 µm	% min. delen	61.2	59.2	45.7	43.3
Q	Korrelgrootte < 2 µm	% min. delen	18.3	21.5	11.5	12.9
	Metalen					
S	Arseen (As)	mg/kg ds	23	22	22	24
S	Cadmium (Cd)	mg/kg ds	0.26	<0.20	0.23	0.23
S	Chroom (Cr)	mg/kg ds	24	27	27	30
S	Koper (Cu)	mg/kg ds	14	9.4	11	13
S	Kwik (Hg)	mg/kg ds	0.18	0.22	0.17	0.21
S	Nikkel (Ni)	mg/kg ds	28	14	20	25
S	Lood (Pb)	mg/kg ds	19	20	21	21
S	Zink (Zn)	mg/kg ds	49	28	35	37
S	Barium (Ba)	mg/kg ds	130	110	130	100
S	Kobalt (Co)	mg/kg ds	17	11	16	19
Q	Mangaan (Mn)	mg/kg ds	9100	6000	6700	6600

Nr.	**Monsteromschrijving**	**Datum monstername**	**Monster nr.**
1	Vale Brazil A1	09-Jul-2019	10820458
2	Vale Brazil A2	09-Jul-2019	10820459
3	Vale Brazil B3	09-Jul-2019	10820460
4	Vale Brazil B5	09-Jul-2019	10820461

kwaliteitswaarborg bodembeheer SIKB

Q: door RvA geaccrediteerde verrichting
A: AP04 erkende verrichting
S: AS SIKB erkende verrichting
V: VLAREL erkende verrichting
M: MCERTS erkend

Dit certificaat mag uitsluitend in zijn geheel worden gereproduceerd.

Eurofins Analytico B.V. is ISO 14001: 2015 gecertificeerd door TÜV en erkend door het Vlaamse Gewest (OVAM en Dep. Omgeving), het Brusselse Gewest (BIM), het Waalse Gewest (DGRNE-OWD) en door de overheid van Luxemburg (MEV).

TESTEN
RvA L010

Analysecertificaat

Uw project/verslagnummer	AT19069	Certificaatnummer/Versie	2019100765/1
Uw projectnaam	Netics analyses 2019	Startdatum	10-Jul-2019
Uw ordernummer		Rapportagedatum	17-Jul-2019/11:02
		Bijlage	A,B,C
Monsternemer		Pagina	2/4
Monstermatrix	Waterbodem (AS3000)		

	Analyse	Eenheid	1	2	3	4
S	Molybdeen (Mo)	mg/kg ds	2.8	<1.5	1.9	4.0
	Minerale olie					
	Minerale olie (C10-C12)	mg/kg ds	<3.0	<3.0	<3.0	<3.0
	Minerale olie (C12-C16)	mg/kg ds	<5.0	<5.0	<5.0	<5.0
	Minerale olie (C16-C21)	mg/kg ds	<5.0	<5.0	<5.0	<5.0
	Minerale olie (C21-C30)	mg/kg ds	<11	<11	<11	<11
	Minerale olie (C30-C35)	mg/kg ds	<5.0	<5.0	<5.0	<5.0
	Minerale olie (C35-C40)	mg/kg ds	<6.0	<6.0	<6.0	<6.0
S	Minerale olie totaal (C10-C40)	mg/kg ds	<35	<35	<35	<35
	Organo chloorbestrijdingsmiddelen, OCB					
S	alfa-HCH	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	beta-HCH	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	gamma-HCH	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	delta-HCH	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	Hexachloorbenzeen	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	Heptachloor	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	Heptachloorepoxide(cis- of A)	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	Heptachloorepoxide(trans- of B)	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	Hexachloorbutadieen	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	Aldrin	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	Dieldrin	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	Endrin	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	Isodrin	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	Telodrin	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	alfa-Endosulfan	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
Q	beta-Endosulfan	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	Endosulfansulfaat	mg/kg ds	<0.0020	<0.0020	<0.0020	<0.0020
S	alfa-Chloordaan	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	gamma-Chloordaan	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	o,p'-DDT	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	p,p'-DDT	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010

Nr.	Monsteromschrijving	Datum monstername	Monster nr.
1	Vale Brazil A1	09-Jul-2019	10820458
2	Vale Brazil A2	09-Jul-2019	10820459
3	Vale Brazil B3	09-Jul-2019	10820460
4	Vale Brazil B5	09-Jul-2019	10820461

Q: door RvA geaccrediteerde verrichting
A: AP04 erkende verrichting
S: AS SIKB erkende verrichting
V: VLAREL erkende verrichting
M: MCERTS erkend

Analysecertificaat

Uw project/verslagnummer	AT19069	Certificaatnummer/Versie	2019100765/1
Uw projectnaam	Netics analyses 2019	Startdatum	10-Jul-2019
Uw ordernummer		Rapportagedatum	17-Jul-2019/11:02
		Bijlage	A,B,C
Monsternemer		Pagina	3/4
Monstermatrix	Waterbodem (AS3000)		

	Analyse	Eenheid	1	2	3	4
S	o,p'-DDE	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	p,p'-DDE	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	o,p'-DDD	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	p,p'-DDD	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	HCH (som) (factor 0,7)	mg/kg ds	0.0028 1)	0.0028 1)	0.0028 1)	0.0028 1)
S	Drins (som) (factor 0,7)	mg/kg ds	0.0021 1)	0.0021 1)	0.0021 1)	0.0021 1)
S	Heptachloorepoxide (som) (factor 0,7)	mg/kg ds	0.0014 1)	0.0014 1)	0.0014 1)	0.0014 1)
S	DDD (som) (factor 0,7)	mg/kg ds	0.0014 1)	0.0014 1)	0.0014 1)	0.0014 1)
S	DDE (som) (factor 0,7)	mg/kg ds	0.0014 1)	0.0014 1)	0.0014 1)	0.0014 1)
S	DDT (som) (factor 0,7)	mg/kg ds	0.0014 1)	0.0014 1)	0.0014 1)	0.0014 1)
S	DDX (som) (factor 0,7)	mg/kg ds	0.0042 1)	0.0042 1)	0.0042 1)	0.0042 1)
S	Chloordaan (som) (factor 0,7)	mg/kg ds	0.0014 1)	0.0014 1)	0.0014 1)	0.0014 1)
S	OCB (som) LB (factor 0,7)	mg/kg ds	0.015 1)	0.015 1)	0.015 1)	0.015 1)
S	OCB (som) WB (factor 0,7)	mg/kg ds	0.017 1)	0.017 1)	0.017 1)	0.017 1)
S	Pentachloorbenzeen	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
	Polychloorbifenylen, PCB					
S	PCB 28	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	PCB 52	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	PCB 101	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	PCB 118	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	PCB 138	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	PCB 153	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	PCB 180	mg/kg ds	<0.0010	<0.0010	<0.0010	<0.0010
S	PCB (som 7) (factor 0,7)	mg/kg ds	0.0049 1)	0.0049 1)	0.0049 1)	0.0049 1)
	Fenolen					
S	Pentachloorfenol	mg/kg ds	<0.0030	<0.0030	<0.0030	<0.0030
	Polycyclische Aromatische Koolwaterstoffen, PAK					
S	Naftaleen	mg/kg ds	<0.050	<0.050	<0.050	<0.050
S	Fenanthreen	mg/kg ds	<0.050	<0.050	<0.050	<0.050
S	Anthraceen	mg/kg ds	<0.050	<0.050	<0.050	<0.050

Nr.	Monsteromschrijving	Datum monstername	Monster nr.
1	Vale Brazil A1	09-Jul-2019	10820458
2	Vale Brazil A2	09-Jul-2019	10820459
3	Vale Brazil B3	09-Jul-2019	10820460
4	Vale Brazil B5	09-Jul-2019	10820461

Q: door RvA geaccrediteerde verrichting
A: AP04 erkende verrichting
S: AS SIKB erkende verrichting
V: VLAREL erkende verrichting
M: MCERTS erkend

Analysecertificaat

Uw project/verslagnummer	AT19069	Certificaatnummer/Versie	2019100765/1
Uw projectnaam	Netics analyses 2019	Startdatum	10-Jul-2019
Uw ordernummer		Rapportagedatum	17-Jul-2019/11:02
		Bijlage	A,B,C
Monsternemer		Pagina	4/4
Monstermatrix	Waterbodem (AS3000)		

	Analyse	Eenheid	1	2	3	4
S	Fluorantheen	mg/kg ds	<0.050	<0.050	<0.050	<0.050
S	Benzo(a)anthraceen	mg/kg ds	<0.050	<0.050	<0.050	<0.050
S	Chryseen	mg/kg ds	<0.050	<0.050	<0.050	<0.050
S	Benzo(k)fluorantheen	mg/kg ds	<0.050	<0.050	<0.050	<0.050
S	Benzo(a)pyreen	mg/kg ds	<0.050	<0.050	<0.050	<0.050
S	Benzo(ghi)peryleen	mg/kg ds	<0.050	<0.050	<0.050	<0.050
S	Indeno(123-cd)pyreen	mg/kg ds	<0.050	<0.050	<0.050	<0.050
S	PAK VROM (10) (factor 0,7)	mg/kg ds	0.35 1)	0.35 1)	0.35 1)	0.35 1)
	Fysisch-chemische analyses					
	Meettemperatuur (pH-CaCl2)	°C	21	21	21	21
Q	Zuurgraad (pH-CaCl2)		6.4	6.6	6.2	6.4

Nr.	Monsteromschrijving	Datum monstername	Monster nr.
1	Vale Brazil A1	09-Jul-2019	10820458
2	Vale Brazil A2	09-Jul-2019	10820459
3	Vale Brazil B3	09-Jul-2019	10820460
4	Vale Brazil B5	09-Jul-2019	10820461

Q: door RvA geaccrediteerde verrichting
A: AP04 erkende verrichting
S: AS SIKB erkende verrichting
V: VLAREL erkende verrichting
M: MCERTS erkend

Akkoord
Pr.coörd.
MC

Dit certificaat mag uitsluitend in zijn geheel worden gereproduceerd.

Bijlage (A) met deelmonsterinformatie behorende bij analysecertificaat 2019100765/1

Monster nr.	Boornr	Omschrijving	Van	Tot	Barcode	Monstername ID/Monsteromsch.
10820458	Vale Brazil	A1	0	1	J1050704	Vale Brazil A1
10820459	Vale Brazil	A2	0	1	J1050689	Vale Brazil A2
10820460	Vale Brazil	B3	0	1	J1050695	Vale Brazil B3
10820461	Vale Brazil	B5	0	1	J1050694	Vale Brazil B5

Bijlage (B) met opmerkingen behorende bij analysecertificaat 2019100765/1

Opmerking 1)

De toetswaarde van de som is gelijk aan de sommatie van 0,7*RG

Bijlage (C) met methodeverwijzingen behorende bij analysecertificaat 2019100765/1

Analyse	Methode	Techniek	Methode referentie
Droge Stof	W0104	Gravimetrie	Cf. pb 3210-1 en cf. NEN-EN 15934
Organische stof (gloeiverlies)	W0109	Gravimetrie	Cf. 3210-2a/b en cf. NEN 5754/EN 12879
Calciet (CaCO3)	W0177	Volumetrisch	Gw. NEN-EN-ISO 10693
Korrelgrootte > 2 mm (natzeving)	W0105	Zeven	Cf. NEN 5753
Korrelgrootte < 2 µm (lutum) sedimentatie	W0173	Sedimentatie	Cf. pb 3210-3 en cf. NEN 5753
Korrelgrootte (fractie < 2000 µm)	W0174	Laserdiffractie	Cf. ISO 13320-1
Korrelgrootte (fractie < 1000 µm)	W0174	Laserdiffractie	Cf. ISO 13320-1
Korrelgrootte (fractie < 500 µm)	W0174	Laserdiffractie	Cf. ISO 13320-1
Korrelgrootte (fractie < 250 µm)	W0174	Laserdiffractie	Cf. ISO 13320-1
Korrelgrootte (fractie < 125 µm)	W0174	Laserdiffractie	Cf. ISO 13320-1
Korrelgrootte < 63 µm (MD) laser	W0174	Laserdiffractie	Cf. ISO 13320-1
Korrelgrootte (fractie < 50 µm)	W0174	Laserdiffractie	Cf. ISO 13320-1
Korrelgrootte < 32 µm (minerale delen)	W0174	Laserdiffractie	Cf. ISO 13320-1
Korrelgrootte < 16 µm (minerale delen)	W0174	Laserdiffractie	Cf. ISO 13320-1
Korrelgrootte (fractie < 8 µm)	W0174	Laserdiffractie	Cf. ISO 13320-1
Korrelgrootte (fractie < 2 µm)	W0174	Laserdiffractie	Cf. ISO 13320-1
Metalen (8) (As, Cd, Cr, Cu, Hg, Ni, Pb, Zn)	W0423	ICP-MS	Cf. pb 3210-4/3250-1 & NEN-EN-ISO 17294-2
Barium (Ba)	W0423	ICP-MS	Cf. pb 3210-4/3250-1 & NEN-EN-ISO 17294-2
Kobalt (Co)	W0423	ICP-MS	Cf. pb 3210-4/3250-1 & NEN-EN-ISO 17294-2
Mangaan (Mn)	W0423	ICP-MS	Cf. NEN-EN-ISO 17294-2
Molybdeen (Mo)	W0423	ICP-MS	Cf. pb 3210-4/3250-1 & NEN-EN-ISO 17294-2
Minerale Olie (C10-C40)	W0202	GC-FID	Cf. pb 3210-6 en cf. NEN 6978
OCB (25)	W0262	GC-MS	Cf. pb 3220-1 en gw. NEN 6980
OCB som AP04/AS3X	W0262	GC-MS	Cf. pb 3220-1 en gw. NEN 6980
Pentachloorbenzeen	W0262	GC-MS	Gelijkw. NEN 6980
PCB (7)	W0262	GC-MS	Cf. pb 3210-7 en gw. NEN 6980
Pentachloorfenol	W0267	GC-MS	Cf. pb 3260-1 en gw. NEN-EN 14154
PAK som AS3000/AP04	W0271	GC-MS	gw. NEN-ISO 18287
PAK (10) (VROM)	W0271	GC-MS	Cf. pb. 3210-5 & gw. NEN-ISO 18287
Zuurgraad (pH-CaCl2)	W0524	Potentiometrie	Cf. NEN-ISO 10390

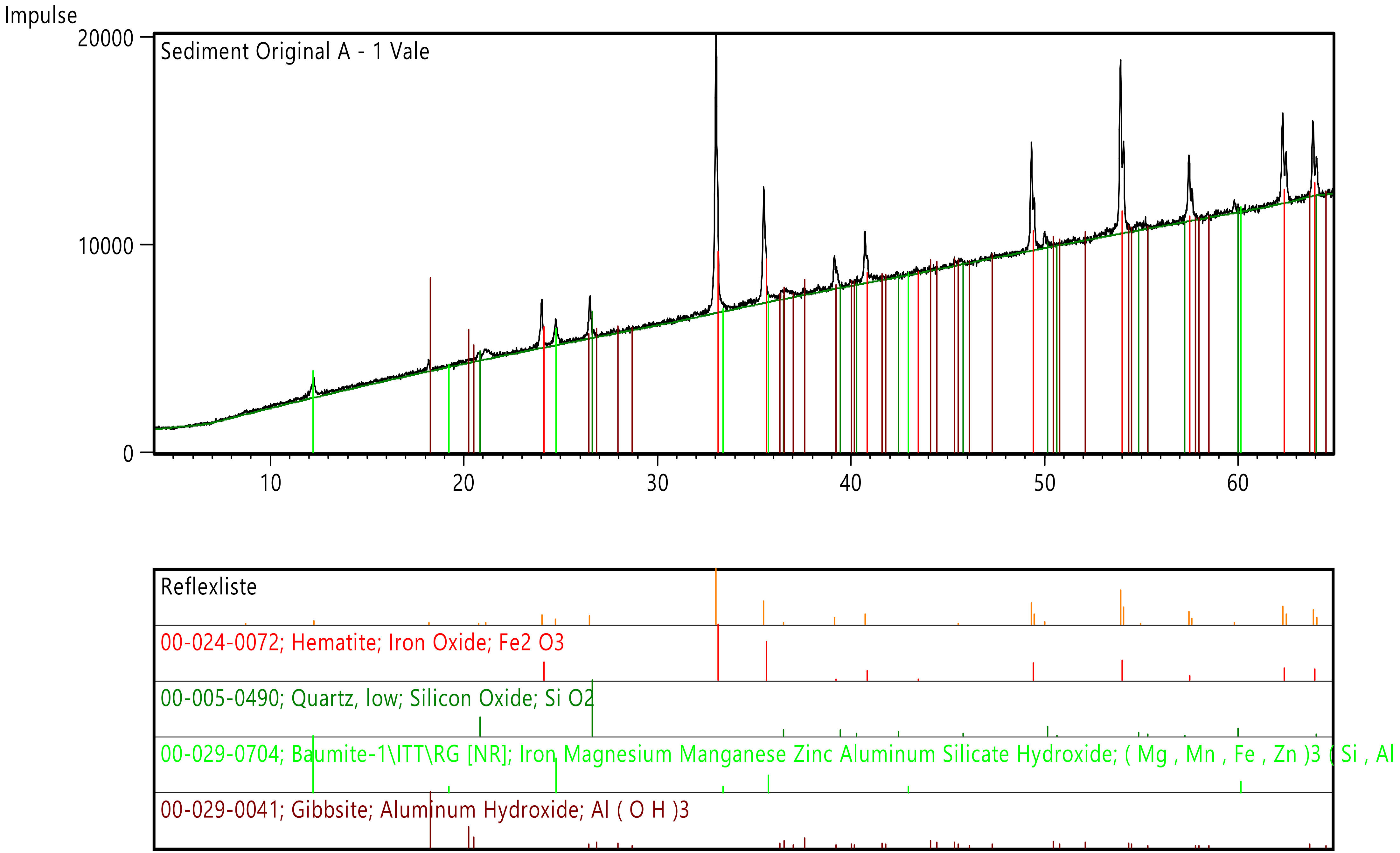
Bijlage (C) met methodeverwijzingen behorende bij analysecertificaat 2019100765/1

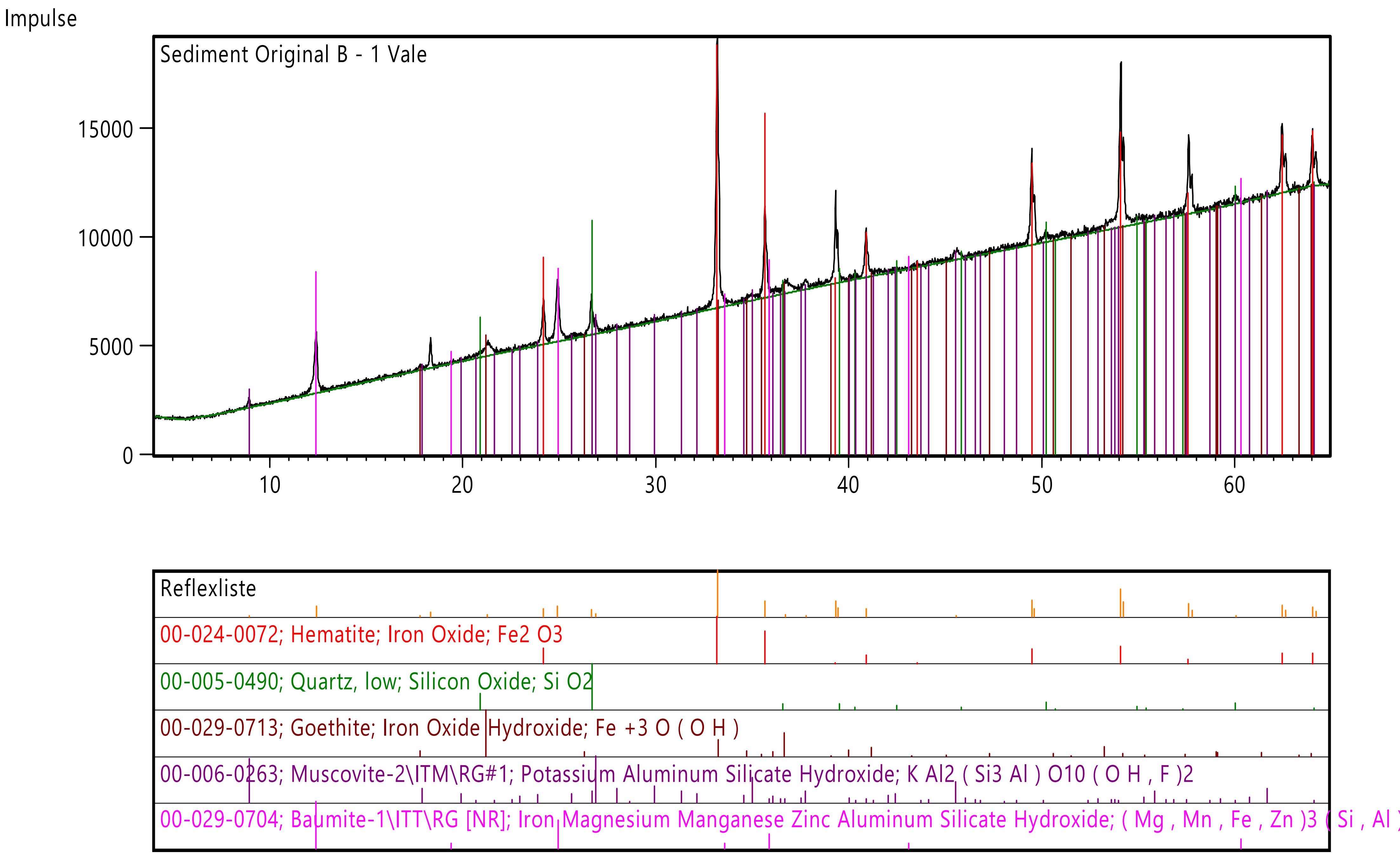
Analyse	Methode	Techniek	Methode referentie

Nadere informatie over de toegepaste onderzoeksmethoden alsmede een classificatie van de meetonzekerheid staan vermeld in ons overzicht "Specificaties analysemethoden", versie juni 2016.

B

Results from XRD


Impulse
20000
10000
0
Sediment Original A - 1 Vale
10
20
30
40
50
60
Reflexliste
00-024-0072; Hematite; Iron Oxide; Fe2 O3
00-005-0490; Quartz, low; Silicon Oxide; Si O2
00-029-0704; Baumite-1\ITT\RG [NR]; Iron Magnesium Manganese Zinc Aluminum Silicate Hydroxide; (Mg , Mn , Fe , Zn)3 (Si , Al
00-029-0041; Gibbsite; Aluminum Hydroxide; Al (O H)3


Impulse
Sediment Original B - 1 Vale
15000
10000
5000
0
10
20
30
40
50
60
Reflexliste
00-024-0072; Hematite; Iron Oxide; Fe2 O3
00-005-0490; Quartz, low; Silicon Oxide; Si O2
00-029-0713; Goethite; Iron Oxide Hydroxide; Fe +3 O (O H)
00-006-0263; Muscovite-2\ITM\RG#1; Potassium Aluminum Silicate Hydroxide; K Al2 (Si3 Al) O10 (O H , F)2
00-029-0704; Baumite-1\ITT\RG [NR]; Iron Magnesium Manganese Zinc Aluminum Silicate Hydroxide; (Mg , Mn , Fe , Zn)3 (Si , Al