A Study of the Characteristics and Behaviour of Composite Backfill Material

By

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ABSTRACT

The history of mine backfill shows that in the past, considerable improvements in backfill-reliant mining technology were made when new fill systems were introduced. The present trend in mine backfill technology is towards the use of high-density fill systems. Tight filling and void reduction have also become essential requirements in engineering design of mines to ensure global stability. High-density fills have a low moisture content and are more competent products requiring less binder and time for stabilization, compared to conventional hydraulic backfill. Cemented rockfill and tailings/sand paste fills are two familiar high-density backfill systems in current use. In future there could be a high demand for low porosity high-density fills, as mines go deeper and ground stresses increase.

This study was an original attempt to investigate the characteristics and property of high-density composite fill systems. Composite fills are made up of derivatives of waste rock, tailings, sand and metallurgical by-products. Composite fills represent the future direction of backfill technology. As mines go deeper, the ore could be processed underground and the waste rock and tailings combined together to form a low-porosity competent fill product. The application of composite fill systems will also increase the material available for backfilling, provide more flexibility in backfill mix design and produce competent fill systems for ground support. It will also benefit the underground mine environment through effective utilization of mine wastes.

The fundamental basis of the work required the study and understanding of the characteristics and properties of cemented rockfill and paste backfill. The effects of sand addition to fine tailings as a means of reducing porosity and improving the mechanical properties of the fill product were also investigated. Additionally, a new concept of backfill, namely, Composite-Aggregate Paste (CAP) that consists of a mixture of fine tailings and graded coarse aggregates was introduced and the material properties were investigated.

RÉSUMÉ

L'histoire montre que l'adoption de nouveaux systèmes de remblayage nous a permis d'améliorer considérablement la technologie minière qui dépendait de ces systèmes. Actuellement, la tendance dans ce domaine penche vers l'utilisation de systèmes de remblayage de haute densité. Le remblayage étanche et la réduction des espaces vides sont ainsi devenus des exigences de premier plan pour les ingénieurs qui font la conception des mines avec un souci de stabilité globale. Les produits de remblayage de haute densité ont un faible contenu d'humidité et sont plus efficaces; ils requièrent moins de liant et moins de temps pour se stabiliser que le remblai hydraulique conventionnel. Les remblais cimentés en pâte constitués de résidus avec ou sans sable sont ces systèmes de remblayage de haute densité les plus communément utilisés maintenant. À l'avenir, la demande de remblais de haute densité et faible porosité pourrait être considérable, à cause de la profondeur de plus en plus grande des mines et de l'augmentation des pressions qui s'exercent sur les terrains.

Cette étude présente un nouvel effort pour étudier les caractéristiques et les propriétés des remblais composé de haute densité. Le remblayais composé est constitué de dérivés de débris de roche, de résidus miniers, de sable et de sous-produits métallurgiques. C'est vers ce type de remblais que s'oriente la technologie, car, les mines étant plus profondes, le minerai pourrait être traité en souterrain et les débris de roche et résidus miniers combinés pour former un produit de remblayage efficace à faible porosité. L'application des remblais composés augmente aussi la quantité de matériel de remblayage, permet une plus grande liberté dans la conception du mélange de matériaux de remblayage, et produit des systèmes de remblayage plus efficaces pour le soutènement. Elle améliore aussi l'environnement souterrain dans les mines par l'utilisation efficace des résidus miniers.

Essentiellement, ce travail a nécessité l'étude et la compréhension des caractéristiques et des propriétés des produits de remblayage de roche cimenté et de remblayage en pâte. Les effets de l'ajout de sable aux résidus fins pour réduire la porosité et améliorer les propriétés mécaniques du produit de remblayage a aussi été étudié. En outre un nouveau produit de remblayage, qui consiste en un mélange de résidus fins et d'agrégats classifiés de taille uniforme - le remblais en pâte composite - a été conçu et les propriétés du matériau ont ensuite été étudiées.

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LIST OF NOTATIONS AND ABBREVIATIONS

The following symbols are used in this document.

ANH	Anhydrite
a_p	Aggregate surface area for a constant water-cement ratio
B	Width of fill block (also, width of stope)
BMT(#)	Base Metal Tailings
C	Cement or binder content
c	Cohesion or cohesive strength
C^*	Ratio of cement to tailings or sand
C_c	Coefficient of curvature
C_g	Solids concentration of fill slurry placed underground
C_T	Classified tailings
C_u	Coefficient of Uniformity
C_v	Volume of cement in a unit dry volume of freshly placed backfill
C_w	Comminuted Waste
D_{10}	Grain size at 10% passing (also, the Effective grain size)
D_{30}	Grain size at 30% passing
D_{50}	Grain size at 50% passing
D_{60}	Grain size at 60% passing
D_{max}	Maximum particle size
D_{min}	Minimum particle size
e	Void ratio
e_{max}	maximum void ratio
e_{min}	minimum void ratio
F_s	Safety factor for fill stability
FA	Fly Ash
h	Height of cut

H, He	Effective height of fill block
L	Strike length of an exposed fill
M_c	Fill block weight minus the wall shear component
L/D	Length to diameter ratio
OPC	Ordinary Portland Cement
p	mean of minor and major principal stresses = $((\sigma_1 + \sigma_3)/2)$
PC	Portland Cement
PFA	Pulverized Fuel Ash
PBFC	Portland Blast furnace cement
PMT (#)	Precious Metal Tailings
q	One half of difference between major and minor principal stress = $((\sigma_1 - \sigma_3)/2)$
Q_o	Maximum load per unit area (of mine equipment)
UCS	Unconfined compressive strength
w/c	Water/cement (binder) ratio by weight
α	Angle of failure in fill block
η	Porosity
γ	Unit weight of fill
γ_r	Unit weight of rock overlying fill
φ	Angle of internal friction
σ_1	Total major principal stress
σ_3	Total minor principal stress
$(\sigma_1 - \sigma_3)$	Deviator stress
σ_x	Unconfined compressive strength
σ_t	Tensile strength of fill
τ	Shear stress

CHAPTER 1

1. INTRODUCTION

1.1 General

New cost-effective methods of improving operational efficiency are being sought by the mining industry worldwide. With regard to mining methods employing backfill, improvements have been made to various aspects of backfilling systems. These have included: different types of backfill systems; availability and supply of fill materials; fill preparation procedures including desired consistencies for transportation and placement; the use of alternative cementing agents; methods of backfill transportation, placement, consolidation/stabilization and stability analysis through modelling. The information has been presented in various conference proceedings on Mine Backfill and related subjects (Backfill Conference Proceedings, 1973-1998). These advancements have presently resulted in a growing demand toward the use of high-density cemented rockfill and paste backfill systems. Mining at greater depths will demand further innovations in high-density backfill technology. These should include the introduction and application of new fill systems complete with methods of preparation and placement.

This study was an original attempt to investigate the characteristics and properties of high-density composite fill systems. The work was motivated by the general interest in the mine backfill literature toward the application of low porosity composite and "aggregate" fills for ground support in mines. The laboratory investigations were carried out at the Mine Fill Support Systems Laboratory, Natural Resources Canada in Sudbury, Ontario between 1995 and 1997.

1.1.1 Definition of Mine Backfill

Mine backfill refers to any material that is used to fill mine openings for stability, environmental and other economic reasons. Backfilling serves several useful functions in the mining cycle. For example, backfill is used to improve safety and increase productivity. In terms of safety, it is used as an engineered structural product to control subsidence, to provide pillar support and to improve ground conditions in deep mines or in stressed zones. Backfilling provides an adequate working floor for workers and mine equipment and increases productivity by controlling ore dilution (Aitchison et al., 1973; Dickhout, 1973). In a mining operation, backfilling provides a means of disposing large quantities of waste products underground away from the surface environment (Amaratunga and Annor, 1989). It is also used to improve mine ventilation. In a more unique role, backfill is used to establish new mining methods (Annor and Clarke, 1988).

1.2 RESEARCH JUSTIFICATION

In the past, the development of more efficient mining methods had often depended on the availability of better types of backfill and other improved support systems. For example, current improvements to conventional cut and fill stoping have benefited from the use of competent consolidated fills (Dickhout, 1973). These have resulted in increased stope sizes, mechanisation, safe and economical recovery of pillars and shortened stope cycle time.

Tight filling (Hedley, 1987, 1995; Grice, 1989; Yu, 1990; Hassani, 1993; Gürtunca and Gay 1993) is an essential requirement in mine design because it reduces convergence and enhances the global stability in a mine. As mines go deeper, ground instability problems can arise due to inadequate fill strength (Thomas et al., 1979), stope convergence (Hedley, 1987), fill segregation (Yu, 1989; Farsangi, 1996), and other causes including the break down of cement bond (Mitchell and Wong, 1982; Ouellet et al,. ,1998). Therefore, there is a critical requirement for backfill reliant mines to investigate alternative types of backfill systems. The new fill systems should rely on having abundant sources of fill material readily available near the mine workings (Stone, 1993; White and Robertson, 1998), and also economical means of preparing the material into a competent structural product (Annor and Clarke, 1988; Hassani, 1993; Stone, 1993) for underground support. For example, this will involve a careful selection and engineering of locally available materials (tailings, waste rock and metallurgical by-products) (Ross-Watt, 1989; Grice, 1989; McKinstry and Laukkanen, 1989; Hassani, 1993; Reschke, 1993; Petrolito et al., 1998; Bloss and Greenwood, 1998) as high-density composite fill products. It will also reduce the critical shortages of fill materials (Barrett et al., 1983) which are sometimes encountered at some mine sites.

A review of the literature shows that various opinions exist regarding the characteristics and behaviour of high-density backfill systems including total tailings and classified tailings paste fills. Paste fill is increasingly being recognised by industry as an effective high-density backfill system. This is because paste fill provides several advantages over the conventional hydraulic backfill systems (Arioglu, 1983; Blight and Clarke, 1983; Clarke, 1988; Aref et al., 1989; Landriault, 1992, 1995; Brackebusch, 1994; Chen and Annor, 1995; Ouellet et al., 1998; Pierce et al., 1998).

1.2.1 Concerns About Cemented Rockfill Applications

Cemented rockfill is a familiar and widely used backfill system by industry (Stone, 1993). It is often assumed to have properties that are similar to those of a weak concrete (Barrett, 1973; Berry, 1981; Arioglu, 1983; Yu and Counter, 1983, 1986, 1988; Yu, 1989; Quesnel et al., 1989; Reschke, 1993; Hedley, 1995; Farsangi, 1996; Farsangi et al, 1996). Cemented rockfill is usually porous and contains average void ratio of approximately 0.51 (Yu, 1990) which can be a disadvantage in terms of achieving a denser fill mass and also, utilizing more waste rock material for fill preparation.

1.2.2 Concerns About the Application of Paste Backfill

Compared to conventional hydraulic backfill, paste backfill provides better support properties and early stabilisation which often results in high strength and stiffness (Annor and Clarke, 1988; Hassani and Aref, 1988; Aref et al., 1989; Amaratunga and Annor, 1989; Bissonnette, 1995; Millette et al., 1995). For example, Hassani and Aref (1998) and Aref et al. (1989) have stated that paste fill has a shorter consolidation time and uses less binder, generally between 2 and 7 per cent. This can result in higher productivity and lower mine operating costs due to reduced binder requirements and early development of fill strength. Hedley (1995) has suggested that paste fill can be very resilient, depending on the binder composition ,and can provide effective control against rockbursts.

In spite of the suggested potential advantages associated with paste fill use at the present time, a large volume placement of paste fill in underground stopes relies on preparation from surface plants (Barrett et al., 1986; Lerche and Renetzeder, 1984; Bissonnette, 1995; Dodd and Paynter, 1995). It may be necessary to process the fill material underground (White and Robertson, 1998) as in the case of deep mining. Another major concern is that paste fill has become a generic term which is commonly used to represent a wide range of low-moisture content hydraulically placed backfill systems made from composite mixtures of various materials including mill tailings, sand, gravel, silts, clays, and other aggregate materials (Lidkea and Landriault, 1993) as well as mine and metallurgical by-products.

1.2.3 The Need for Composite Fill Systems

The new blended, composite or "aggregate" fills (Arioglu, 1983, 1984; Barrett et al., 1983; Grice, 1989; McKinstry and Laukkanen, 1989; Wingrove, 1993; Swan et al., 1993; Moss and Greenwood, 1998; Raffield et al., 1998) have originated from attempts to engineer and optimize uniformly graded fine or coarse fill materials to achieve a competent structural product. In most of the reported studies (Grice, 1989; McKinstry and Laukkanen, 1989; Wingrove, 1993; Moss and Greenwood, 1998; Raffield et al., 1998), the maximum aggregate size used had been limited to less than 20mm in diameter. The fine material had also consisted of classified hydraulic tailings to permit drainage after placement. The reported "aggregate" fills were mostly uncemented or weakly cemented (Swan et al., 1993; Wingrove, 1993).

The characteristics and behaviour of these new fills when placed as composite-aggregate paste ("CAP") fill merits further research, especially when aggregate sizes larger than 20mm and cemented full plant tailings paste mixtures are used. The understanding of composite fill behaviour is essential because the application of composite fill systems could become the future direction of high-density backfill technology when mining at greater depths.

It is proposed that the properties of composite aggregate paste (CAP) fill be considered fundamentally as consisting of the combined properties of two backfill systems: tailings paste fill and cemented rockfill. In this regard, a study on composite fill systems must also include the basic understanding of the characteristics and properties of tailings or sand paste backfills, as well as cemented rockfill.

1.2.4 Other Considerations

Consistency is defined in this study as the degree of firmness of a paste fill mixture, which can be expressed by either pulp density or slump (ASTM C-143). Various viewpoints exist (Lidkea and Landriault, 1993; Landriault, 1992, 1995) regarding how paste fill consistency should be determined. Presently, slump tests are used (ASTM C-143; Neville, 1987) to establish optimum pulp density ranges for paste backfill transportation. Generally, paste backfill is transported underground at approximately 178mm to 229mm (7" to 9") slump (Lerche and Renetzeder, 1984; Hassani and Aref, 1988; Aref et al., 1989; Landriault, 1992, 1995; Lidkea and Landriault, 1993; Brackebusch, 1994). The relationship between slump and moisture content for concrete is known to be influenced by the physical properties of the materials (Neville, 1987). Unlike concrete technology (Neville, 1987), there are no definitive studies in the published literature regarding the effects of material properties on paste fill pulp density, although low binder content paste backfill may not behave like concrete. This information is essential for understanding and establishing optimum mix design limits (Wingrove, 1993) for high-density composite backfill systems which essentially, involve the blending of sand, or tailings paste and rockfill aggregates.

Ideally, the behaviour of fill material in situ is best determined from field placement tests. Laboratory scale effect tests (Reschke, 1993; Yu, 1990; Yu and Counter, 1983), can however provide some indication of the potential underground behaviour of the material in the absence of a field test.

1.3 OBJECTIVES AND SCOPE OF STUDY

The main objective of the study is to extend the state of knowledge on backfill as a support system in underground mines. Specifically, an original attempt has been made in this study to investigate the characteristics and properties of high-density composite backfill as a new fill system.

The specific objectives and scope of the study are seen as follows:

1. Examination of factors that affect the consistency of tailings and sand paste backfill in terms of mix design considerations for composite fill. The evaluation of paste fill consistency will be made as a function of the physical properties of the fill materials (tailings and sand).
2. Examination of factors that affect high-density composite backfill properties including the range of strength development as a function of particle size gradation, binder and moisture contents, curing environment and time.
3. Investigation of scale effects on the strength and deformation properties of the high-density composite fill systems relative to the cemented rockfill and paste backfill. This information will be used as a means of establishing the potential in situ behaviour of the high-density composite backfills relative to those of the cemented rockfill and the paste backfill.

4. 1.4 METHODOLOGY

The investigation consisted of the following major components:

1. Literature review
2. Laboratory studies
3. Analysis of the results
4. Discussions and conclusions

A flowchart of the research methodology is provided in Figure 1.1

1.4.1 Literature Review

The first element of research deals with the identification of potential properties of high-density (stiff) backfill systems from the literature. A comprehensive review of backfill materials, their properties and behaviour, optimization methods, quality control procedures for high-density fill preparation and placement, as well as engineering design models have been made from the literature survey. The material has provided the relevant background information for the study.

1.4.2 Laboratory Studies

This element of research has consisted of physical and mechanical properties determinations on backfill materials and stabilized high-density backfill products.

These investigations were conducted using full plant or "total" and classified mill tailings from six different sources of base metal and precious metal mines. Three different sources of alluvial sand materials, as well as waste rock aggregates were investigated. The broad selection of test materials was made in attempt to obtain a fair representation of the wide variety of backfill materials found in practice. The tests have been carried out as a function of specific gravity, size gradation, sand and/or coarse aggregate contents, binder type and composition, moisture content, and curing age and environment.

A fundamental examination of paste backfill consistency in terms of method of measurement for mill tailings and/or sand fills have been made with the selection of a suitable range of mix design limits for the composite aggregate paste (CAP) fill systems. Variable quantities of sand were used for mix design of the tailings/sand composite pastefill, depending on the size gradation of the tailings. A three step approach was followed in this regard.

i) Initially, the characteristics and properties of tailings and sand paste fills as well as cemented rockfill were established independently.

ii) Secondly, sand was added to the tailings in various proportions depending on the particle size distribution, in an attempt to improve the size gradation of the combined material. The characteristics and properties of the composite paste product with and without the binder were then evaluated.

iii) Next, fine tailings were added to graded rockfill aggregates in a fixed ratio and the characteristics and properties of the resulting composite-aggregate paste (CAP) fill were assessed.

With regard to the mechanical properties determinations, testing has included direct shear measurements, unconfined and triaxial compressive strength tests on uncemented as well as stabilized paste fill samples. These were done to provide information on the strength and deformation properties as well as the failure characteristics of the fill samples.

1.4.3 <u>Backfill Materials and Cementing Agents</u>

Both conventional and non-conventional backfill materials have been investigated as part of this study.

i) The conventional backfill materials have included:

classified mill tailings;

alluvial sand;

waste rock aggregates; and

a combination of mill tailings and coarse aggregates including sand.

ii) Full plant tailings or " total tailings" have also been studied under the category of a non-conventional fill material.

iii) Backfill preparation has also included the evaluation of binder composition, sample curing environment and time. Both singular and combined effectiveness of the most commonly used cementitious materials such as Ordinary Portland Cement (OPC), slag, fly ash, anhydrite, silica fume and other metallurgical by-products have been investigated as part of the evaluation in meeting the study objectives.

iv) The laboratory mechanical property test data has been developed using both small size and large size cylinders. Test specimen sizes have ranged between 50mm by 100mm cylinders and 457mm by 914mm cylinders. Cube samples with sizes ranging between 50mm and 150mm were also tested. This was done in order to evaluate the potential scale effects on the in situ properties of the composite backfills relative to those of the paste backfill and cemented rockfill.

1.5 OUTLINE OF THESIS

The investigation is as outlined in the following chapters:

(1) The functions of backfill in the mining cycle are introduced and a new original definition of mine backfill is provided in Chapter 1. The advantages of using high-density composite backfill systems and the justification for undertaking the research, as well as the study objectives and the outline of the thesis research program are also presented in this chapter.

(2) Chapter 2 contains a review of the literature on high-density backfill systems. These include mill tailings and sand paste fills, cemented rockfill and derivative fills including blended paste fills and composite-aggregate (CAP) fill. The chapter also contains a summary of optimization methods for improving backfill material properties, as well as relevant failure models for analyzing backfill stability and designing backfill

systems. The review contains information on the scale effects of mechanical properties of geotechnical materials such as rocks and soils.

(3) Chapter 3 outlines the physical property experiments and results on mill tailings and sand paste fills, cemented rockfill and composite backfills, including blended tailings/sand paste and rockfill/tailings composites. The chapter also contains a description of the sample preparation procedures as well as test methods used to determine the slump and pulp density of paste backfill.

(4) The mechanical property tests and results are presented in Chapters 4 and 5. The mechanical properties of the fill materials are determined as a function of binder type and content, and curing environment and time. The effects of specimen size on the mechanical properties were also determined for the three high-density backfill systems in order to provide a basis for comparing their potential fill behaviour in situ.

(5) Comparisons of the three high-density backfill materials properties as well as possible applications of the results of this study are provided in Chapter 6.

(6) The results of the research study are discussed and overall conclusions from the various segments of the investigations are summarized in Chapter 7.

(7) Chapter 8 contains the literature references cited in this study.

Detailed information on the laboratory tests are summarized in a series of appendices.

Outline of Research Program

A STUDY OF THE CHARACTERISTICS AND BEHAVIOUR OF COMPOSITE BACKFILL MATERIAL

LITERATURE REVIEW
- Identification of high-density fill properties and behaviour
- Identification of design models

↓

CHARACTERIZATION AND OPTIMIZATION OF HIGH-DENSITY FILL MATERIAL PROPERTIES

↓

MILL TAILING PASTE FILL

<u>Materials</u>
- Classified tailings
- Total tailings
  - ► Precious metal mine tailings (fine)
  - ► base metal mine tailings (medium)
  - ► base metal mine tailings (coarse)

<u>Physical Properties</u>
- ► particle size gradation (C_u, C_c, $-20\mu m$)
- ► specific gravity

<u>Engineering/Mechanical Properties</u>
- Paste consistency
  - ► sedimentation/consolidation
  - ► slump vs moisture content
- Uncemented product
  - ► shear strength at optimum moisture
- Cemented product
  - ► compressive strength
  - ► shear strength parameters
  - ► deformation modulus
  - ► porosity & void ratio
- Scale effects on material properties
- Binder alternatives

CEMENTED ROCKFILL

<u>Materials</u>
- Rockfill aggregates

<u>Physical Properties</u>
- ► particle size gradation (C_u & C_c)
- ► specific gravity

<u>Engineering/Machanical Properties</u>
- ► water-binder ratio
- ► compressive strength
- ► deformation modulus
- ► porosity and void ratio
- Scale effects on material properties

COMPOSITE AGGREGATE PASTE (CAP) FILL

<u>Materials</u>
- ► tailings
- ► alluvial sand
- ► rock aggregate

<u>Physical Properties</u>
- ► particle size gradation (Cu & Cc)
- ► specific gravity

<u>Engineering/Mechanical Properties</u>
- ► water-binder ratio
- ► compressive strength
- ► deformation modulus
- ► porosity & void ratio
- Scale effects on material properties

↓

Relative Comparisons of Material Properties

CHAPTER 2

2. PREVIOUS WORK

2.1 GENERAL

Many articles document the properties and behaviour of conventional hydraulic sand and tailings backfill systems. Recent advancements in conventional hydraulic backfill technology have resulted in the development of low moisture content, or high-density fill systems. Cemented rock fill and "paste fill" are the two most common high-density backfill systems which are in present use. While cemented rockfill can be considered as a form of weak concrete (Gonano et al., 1978; Blight and Clarke, 1983; Weaver and Luka,1970; Yu and Counter, 1983, 1986), paste fill (Lerche and Renetzeder, 1984; Hassani and Aref, 1988; Aref et al., 1989; Ouellet et al, 1998; Pierce et al., 1998) has become a generic term, which is commonly used to represent a wide range of low moisture content hydraulically placed backfill systems.

Lidkea and Landriault (1993) have suggested that paste fill can be prepared from any types of materials. Presently mixtures and derivatives of various materials including mill tailings, sand, gravel, silts, clays, and other aggregate materials as well as, mine and metallurgical by-products can generally be classified as paste backfill. The characteristics of these blended or "aggregate" fills (Arioglu, 1983; Barrett et al.,1983; Grice, 1989; McKinstry and Laukkanen, 1989; Wingrove, 1993; Bloss and Greenwood, 1998) may resemble those of "engineered soils" (Nicholson and Wayment, 1964; Terzaghi and Peck, 1967; Weaver and Luka, 1970; Thomas et al., 1979; Arioglu, 1983; Grice, 1983; Barrett et al.,1986; Wingrove, 1993;), depending on the proportional amounts of the fill materials in the composite mixture. Studies in Soil Mechanics (Terzaghi and Peck, 1967) and Concrete Technology (Neville,1987) suggest however, that composite materials with similarities in physical appearance can sometimes have different and variable mechanical properties and behaviourial characteristics. Such products can also differ in terms of their transportation and placement behaviour.

The characteristics and behaviour of high-density fills are complex, the subject matter covers a wide range of topics in mining engineering, concrete technology, soil mechanics, rock mechanics, fine particle physics and fluid mechanics. The objectives and scope of the review have been limited to areas of mining engineering, soil mechanics, concrete technology and backfill technology with special emphasis on materials that relate to the preparation of composite fill systems.

2.1.1 Objectives and Scope of the Literature Review

The objectives of this literature review are to identify previously published works on high-density fill systems, to evaluate the properties and behaviour of these fill products and to determine the requirements for further work needed to meet the objectives of the proposed research study.

The scope and principal elements of the review include the following:

- High-density backfill identification criteria;
- Properties and behaviour of mill tailings paste backfill;
- Liquefaction potential of mill tailings paste backfill;
- Properties and behaviour of cemented rockfill;
- Properties and behaviour of "aggregate" fills;
- Relative comparisons of high-density backfill material properties;
- Optimization methods for backfill materials;
- Scale effects on mechanical properties of fill materials;
- Fill design rationale and predictive models

2.1.2 Definition of a High-Density Fill System

Paste fill, cemented rockfill and "aggregate" or "blended paste fill" for short, are the three types of high-density backfill systems being considered in this review.

Blight and Clarke (1983) describe high-density backfill or a "stiff fill" as it is commonly known, as consisting of a relatively rigid skeleton of waste rock having the interstices filled with cement or cemented tailings. On the other hand, Blight and Clarke (1983) consider a "soft fill" as normally composed of cemented tailings or sand. The preceding definition does not accurately describe most stiff fill systems since, it excludes cemented tailings, high density slurry and paste backfills as types of high-density fill systems. A high-density fill for this study is considered as a low water content product with high compressive strength and high deformation modulus, compared to values obtained with conventional cemented hydraulic fill. This new description of a stiff fill system, is based on the mechanical properties and behaviour of the fill mass, instead of the physical appearance of the fill product.

2.2 PROPERTIES AND BEHAVIOUR OF MILL TAILINGS PASTE BACKFILL

2.2.1 Definition of Mill Tailings Backfill Systems

Hydraulically placed mill tailings fills can exist in three consistencies (Millette et al., 1995, 1998) depending on the amounts of solids and water that are present in the fill. The consistencies are generally identified in terms of pulp density or settled density. The pulp density refers to the weight proportions of solids and water present in a given mass of fill. It is also considered in terms of water-to-solids (w/s) ratio (Clark, 1988). The settled density is the density at which the tailings material settles over a specified time period, generally, one-hour. The specific gravity, size, shape and the gradation of the tailings particles affect the settled density (Chen and Annor, 1995).

By definition, the pulp density of a conventional hydraulic fill is much lower than the settled density of the tailings material. The pulp density of a conventional hydraulic fill often ranges between 65% and 70 % (Millette et al., 1995, 1998)of dry solids by weight. For high-density slurry backfills, the pulp density is just below the settled density and it generally ranges between 71% and 76% solids by weight. Paste backfill on the other hand, has a pulp density, which is higher than the settled density. The solids composition of paste backfill generally ranges between 76 per cent and 84 per cent by weight (Millette et al., 1995). Paste backfill has also been described in terms of slump (ASTM C-143; Verkerk and Marcus, 1988; Lidkea and Landriault, 1993) as well as the water retention ability and percentage of minus 20μm particle size material present in the fill (Landriault, 1992; Brackebusch, 1994).

This later description uses transportation requirements as the criteria for identifying what constitutes a "paste fill" instead of the settling density of the fill materials. These reports tend to suggest that any composite mixture of aggregates qualifies as paste backfill provided the product contains more than 15% of the minus 20μm material particles. Paste fill has also been defined (Millette et al., 1995) as any hydraulically placed backfill that does not exude water.

2.3 PLACEMENT BEHAVIOUR OF MILL TAILINGS BACKFILLS

2.3.1 The Effects of Material Properties on Backfill Behaviour

The placement behaviour of mill tailings backfills is generally influenced by the properties of the constituent materials. Fill porosity and moisture content represented by water-to-solids (w/s) ratio have been suggested by Clark (1988), as sufficient parameters for describing the phase relationships of any backfill. Various investigators including Clark (1988), Lamos (1993) and Chen and Annor (1995) have identified some important factors, which define the three stages of mill tailings backfill consistency in a stope. Solids sedimentation, consolidation and drying are among the important processes that affect the placement behaviour of mill tailings backfills. All the above three processes are significantly influenced by the specific gravity of the solids and the porosity, or voids content of the fill. These processes transform the fill from a slurry to a paste-like consistency and finally, to a dry bulk solid (Clark, 1988; Chen and Annor, 1995). These processes can also affect the stability of the pastefill in situ (Ouellet et al., 1998) if the fill mass becomes saturated.

2.3.2 <u>Placement Behaviour of Gold Mill Tailings Backfill:</u>

Lidkea and Landriault (1993) have suggested that the density of pastefill can vary between 78 and 87 per cent solids by weight, depending on the particle size distribution of the material used. Lidkea and Landriault (1993) contend that, finer materials have greater surface area to be wetted, therefore they produce higher density than a coarser material at the same consistency. They have also suggested that it is possible to produce pastefill from any material however, they have recommend that prior test work should be done to determine the economic feasibility of using these materials. The following three pulp density ranges have been identified (Landriault, 1995; Landriault and Tenbergen, 1995) for total tailings paste backfill based on a 178mm (7-inch) slump:

Tailings Type	wt% solids content
Coarse	79 wt%
Medium	75 wt%
Fine	70 wt%

Brackebusch and Shillabeer (1998) have suggested however that both particle size distribution and the specific gravity of the minerals, affect the pulp density of paste mixtures. Brackebusch (1994) has suggested that the consistency of paste mixtures as measured in terms of slump cone may range from zero or slightly greater to nearly 300mm. Espley et al., (1970), concluded that the placement behaviour of mill tailings hydraulic fill is influenced by the specific gravity of the material, binder composition, as well as particle size distribution of the fill material.

2.4 MECHANICAL PROPERTIES AND BEHAVIOUR OF TOTAL TAILINGS PASTE BACKFILL

Various studies including those by Wayment, 1978; Hassani and Aref, 1988; Aref et al., 1989; Hunt 1989; Vickery and Boldt, 1989; Ross-Watt 1989; Landriault, 1992; Lidkea and Landriault, 1993, Udd and Annor, 1993; Hedley, 1995; Chen and Annor, 1995; Bissonnette, 1995; Ouellet et al., 1998; Pierce et al., 1998) have contributed to the knowledge base on the mechanical properties and behaviour of tailings paste backfill systems.

There are also conflicting reports in the available literature regarding the behaviour of backfills containing high amounts of fine material. Thomas (1981) and Boldt et al. (1993) determined that the compressive strength increased when fine material was added to conventional hydraulic fill (Figure 2.1). Ross-Watt (1989) also reported that compressive strength increased for paste backfill with increased content of fine materials. On the other hand, Clark (1988), showed that the presence of fine particles decreased the compressive strengths of total tailings paste backfill. Chen and Annor (1995) studied the properties of finely ground gold mill tailings and concluded that there were close similarities between the compressive strengths of cemented full plant tailings samples and cemented classified tailings samples. Similarly, Vickery and Boldt (1989) also investigated the engineering properties of dewatered total tailings including compressive and tensile strengths as described in Section 2.4.1.

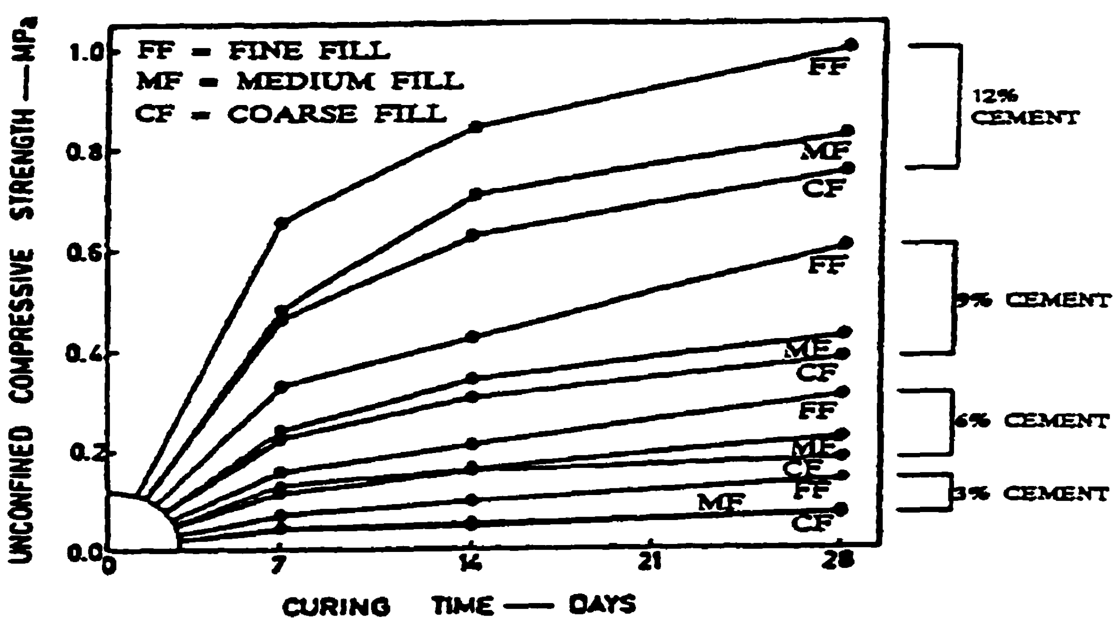


Figure 2.1 Tests - curing curves to 28 days for three fill grades (coarse, medium, and fine) and range of Portland cement addition level (after Thomas, 1981).

2.4.1. Engineering Properties of Dewatered Total Tailings Fill

Vickery and Boldt (1989) studied total tailings backfill in which the pulp density of the pastefill slurry was limited to 70 per cent solids by weight. The gradation of the total tailings ranged between 0.001 to 0.6 mm in diameter. Commercially available Class F fly ash super plasticizer, pit-run and ground smelter slag, kiln dust, and oil shale retorted waste were incorporated in the test mixes to determine their influences on the physical properties of the backfills. The material properties were determined after 28, 120, and 180 days of curing.

A 28-day compressive strength of 717 kPa was obtained for mill tailings paste fill containing oil shale retorted waste without cement. It was found that the cementing properties of the retorted wastes were greater for the finer grain size tailings. Also, the compressive strength of the total tailings aggregate improved when fly ash was used as a binder. The influence of the fly ash diminished however, after the amount of the minus 200 mesh (75 micron) size material increased. It was also reported that the addition of fly ash improved the compressive strength of total tailings backfill samples and resulted in increases of up to 98% over the strength gained by the use of cement alone. The ratio of compressive strength to tensile strength ranged between 4.4 and 4.8 for the range of studied tailings material.

2.4.2 Reported Examples of Backfill Properties (after Ross-Watt, 1989)

Ross-Watt (1989) reported on the backfilling practices at three mechanised base metal (Gold Fields Group) mines in South Africa. The three mines were Black Mountain Mineral Development Company (Pty) Limited (BMM), O'okiep Copper Company Limited (OCC) and Tsumeb Corporation Limited (TCL).

2.4.2.1 Black Mountain Mine

Three types of tailings material were used for backfilling at the Black Mountain Mine. These consisted of unclassified, classified and a mixture of classified tailings and dune sand. The mixtures of the above materials with various cement contents were placed at rates ranging between 300 and 400 tons per hour. The placement pulp density was in the range of 70% to

74%. A 14-day target strength of 700 kPa was obtained for the mixture containing 50% classified tailings and dune sand using a 1:13 cement ratio. It was noted that the backfill required sufficient resiliency to withstand the effect of pillar stope blasts. The cement ratio required for an unconfined compressive strength of 700 kPa was found to be too high to allow for resiliency of the placed fill against blasting. The high strength also resulted in vertical shrinkage cracks. The change to a compressive strength target of 400 kPa improved the properties of the fill in terms of its resistance to vertical shrinkage cracking. These findings suggest that fill design is sometimes influenced more by considerations other than strength and stiffness.

2.4.2.2 O'okiep Copper Company (OCC)

Ross-Watt (1989) reported further that at the O'okiep Copper Company, the mining practice required the use of a strong backfill to withstand mining of adjacent stopes. A free standing height of up to 150m was required. An unconfined compressive strength criterion of 850 kPa after 90 days was selected, based on test work and experience previously gained with cemented hydraulic backfill. A mixture of sand and tailings was used for backfilling. Test work showed that a backfill consisting of a ratio of 3.5:1 tailings to dune sand by weight, a 1:15 cement ratio and reticulated at a pulp density of 75% would provide the required strength. Copper reverbatory slag produced at the OCC smelter was investigated as an alternative binder, a readily available low cost substitute for Portland cement. The addition of 6% to 7% by weight of the modified slag to the backfill was shown to produce the required strength.

2.4.2.3 Tsumeb Corporation

Ross-Watt (1989) reported also on the properties of cemented backfill that was being used for all cut-and-fill stoping as a standard practise at Tsumeb Corporation. After substantial investigation and test work at Tsumeb Corporation, unclassified tailings was identified as the most suitable backfill material at the mine. The average strength of the fill was reported as follows:

Cement Ratio	UCS, kPa @ 7 days
1:8	2900
1:12	830
1:15	650
1:20	500
1:25	240
1:30	240
1:35	120

It was found that, in general, the strength increased with fines content of the tailings as shown below:

	Percent Minus 45 micron	UCS, kPa at 7 days
Secondary underflow	3.8	765
Primary underflow	15	800
Unclassified	45	2900

The test work indicated that the strength also increased with increasing pulp density, but it was difficult to consistently achieve pulp densities of over 75 per cent in practice.

The two types of backfill which were being used at the mine had the following specifications:

Cement Ratio	UCS (kPa) 24 hours	UCS (kPa) 7 days	Pulp Density %
1:12	84:3	830	75
1:25	35	240	75

The higher strength backfill provided a hard surface for mining purposes after 48 hours.

The above studies suggest that high early strength backfill can be produced with a careful choice of conventional binder quantities.

2.4.3 Reported Examples of Laboratory and In-Situ Paste Fill Properties Behaviour

Two recent studies on the properties and behaviour of paste backfill systems by Ouellet et al., (1998) and Pierce et al., (1998) are summarized below.

Ouellet et al. (1998) studied the physical and mechanical characterization of paste backfill by laboratory and in-situ testing. It was concluded from the study involving various confining pressures and cement contents ranging from zero and 6.5 per cent that cohesion is created by the effects of cement. Also that the cohesion can be destroyed depending on the sample load history. The uncemented samples were found to display purely frictional behaviour without cohesion. Ouellet et al. (1998) determined from their study that the observed apparent cohesion was a of function cement content and depended on the load history of the backfill. This finding is an agreement with the results of a previous reported study by Mitchell and Wong (1982) which concluded that the yield point on the stress-strain curves correspond to a transition from a linear to non-linear behaviour for these materials . Ouellet et al. (1998) have proposed further that during shearing, the cementation bonds are progressively broken until finally the backfill behaves as an uncemented material with no apparent cohesion.

Ouellet et al. (1998) concluded from the results of a pressure meter field study that the in-situ paste backfill was close to saturation.

The studies by Ouellet et al. (1998) have indicated that paste backfill displays a "contractant " or (non-dilatant) behaviour. Also, in situ testing of the fill six months after placement, revealed that the material was near a state of saturation therefore, the cement bond was the only thing producing cohesion.

The importance of having a good cement bond was stressed in the report. It was suggested that there must be sufficient cohesion to counteract the mining induced loading and also, to improve the liquefaction resistance of the fill mass and to ensure stability of an exposed backfill face in situ. The damage of the cement bond could also be caused by chemical reaction (Mitchell, 1983) within the fill, or rockmass convergence (Hedley, 1995). The exposure of a free face due to the extraction of secondary pillar have also been suggested as source of loading on the fill (Gürtunca and Gay 1993). The loading history of backfill can also affect the observed cohesion strength.

Pierce et al. (1998) also studied paste backfill properties and behaviour at the Golden Giant Mine. Test samples were prepared at a constant fill slump of 19cm (7.5in) and this corresponded to a pulp density of 75 per cent by weight. Three different types of binder were used in the study. These consisted of 50:50 blend of Normal Portland Cemented (Type 10) and type C Fly Ash (FA). Unconfined and triaxial tests samples consisted 50mm diameter by 120mm long. The samples were tested at 28, 56 and 112 days. The fill material was classified as sandy silt, and contained 50 per cent silt, 45 per cent sand and 5 per cent clay.

The coefficient of uniformity for the material was found to be 13. The weight per cent of material passing 20μm was 27 per cent.

The average physical properties of the test material were as follows: 25 per cent moisture content, 50 per cent porosity, bulk density of 2000kg/m^3 and 90 per cent degree of saturation. Laboratory tests were conducted on the pastefill material to determine the paste backfill behaviour under unconfined, confined and consolidated undrained triaxial compression loading. Shear strength parameters were determined using both consolidated drained and undrained triaxial tests.

Limit equilibrium analysis (Mitchell et al., 1982) was conducted based on the drained triaxial test parameters. The undrained shear strength of the pastefill was found to be less than the drained shear strength and that seemed to be of some concern. It was concluded that liquefaction potential of the Golden Giant pastefill with 3 per cent binder content would be low after 28 days curing period.

2.5 LIQUEFACTION POTENTIAL OF TOTAL TAILINGS PASTE BACKFILL

The performance of most soil-like (Geotechnical) materials including soil-cement and mill tailings backfills are controlled by their shear strength (Terzaghi and Peck, 1967). The build up of pore water pressure reduces the shear strengths of these materials (Terzaghi and Peck, 1967; Hassani and Aref, 1988; Ouellet et al., 1998). An evaluation of the liquefaction susceptibility of total tailings paste backfill should therefore be made under, variable loading conditions as a part of the engineering design requirements.

Hassani and Aref (1988) and Aref et al. (1989), evaluated and reported on the liquefaction potential of total tailings pastefill. The test program involved both laboratory and field investigations to define the physical and mechanical properties of the fill. Liquefaction potential was evaluated based on the principle of "steady state of deformation" approach (Poulos et al., 1985). The results of that investigation suggested that under the specified conditions, even weakly cemented tailings pastefill was not susceptible to liquefaction.

2.6 CEMENTED ROCKFILL AND COMPOSITE AGGREGATE PASTEFILL

2.6.1 Properties and Behaviour of Cemented Rockfill Systems

Rockfill has been described (Landriault, 1992) as any backfill material transported and placed in an underground workplace in a non-saturated state. The preceding definition suggests that fill systems that are used for underground civil construction such as sand and blended alluvial materials can also be considered as forms of rockfill. Mill tailings backfill can also be transported and placed in an underground workplace in a non-saturated state depending on the mining method and the mode of fill delivery or transportation.

For the purposes of this review and the study, a Rockfill system refers to the use of coarse waste rock for backfilling rather than a tailings or sand hydraulic fill system. A cemented rockfill may consist of sized or unsized cemented aggregates. It may also, be uncemented however, unless cemented the loose rock has limited ground support and negligible free standing potential. Quesnel et al., (1989) and Hedley, (1995) have both stated that cemented rockfill can provide good ground support in high stress areas and can also be effective in controlling rockbursts.

Yu (1990) reported on the various forms of rockfill systems in use at Kidd Creek Mines Limited. These rockfill types and their preparation are summarized below. Some of the described rockfill preparation methods are relevant to this research study because of their similarities to composite aggregate paste CAP backfill systems. Other reported studies in the available literature on the properties and behaviour of cemented rockfill for mining applications include the following: Barrett (1973), Gonano et al. (1978), Berry (1980), Arioglu (1983), Yu and Counter (1983, 1986, 1988), Yu (1989), Quesnel et al., (1989), Stone (1993), Farsangi and Hara (1993), Reschke (1993), Swan et al., (1993), Hedley (1995), Farsangi (1996), Farsangi et al., (1996) and Moss and Greenwood (1998). This part of the review covers the physical and mechanical properties of rockfill systems, their preparation and placement methods and the identification of requirements for improvement, in the context of the research study on composite fills.

2.6.2 Rockfill Type and Methods of Preparation and Placement

2.6.2.1 Cemented Rockfill

Cemented rockfill is composed of a mixture of aggregates containing various types and amounts of binder. The mixture generally produces a stiffer and higher strength fill with lower amounts of cementing agents, compared to conventional cemented hydraulic backfill (Reschke, 1993). With equivalent binder contents, cemented rockfill can generally produce unconfined compressive strengths that are higher than those of hydraulic fill. Compared to hydraulic fill

(Thomas et al., 1979), cemented rockfill normally develops a higher modulus of elasticity, cohesion and angle of friction.

It is reported that (Hassani et al., 1989; Yu,1990) in certain Canadian mining operations, the cemented rockfill product consisted of sized rockfill aggregates. The rock fill aggregates were generally mixed with cement slurry, usually 5% to 6% by weight of aggregates at a pulp density of 50-60%. The reported advantages of cemented rockfill include the fact that there are generally no drainage problems associated with its use (Berry, 1980; Stone, 1993; Reschke1993; Bloss and Greenwood 1998). Also, when correctly placed, cemented rockfill can produce a high quality fill product. The main disadvantages associated with cemented rockfill use, include problems with control over the segregation of the fill product and requirements for an extensive preparation plant and transportation system (Yu, 1990). Additionally, cemented rockfill generally contains high void ratio or porosity and therefore, may not be conducive to tight filling.

2.6.2.2 Consolidated Sand-Rockfill

This is reported (Yu, 1990) as a combination of consolidated rock fill with varying amounts of sand added to it, usually 5-10% of the rockfill aggregates by weight. With the same cement content as consolidated rock fill, a cement sand slurry is introduced simultaneously with the rockfill to fill the voids within the aggregates. This enhances the stability of the fill for both gravity loading as well as, resistence to blast vibration during excavation of adjacent stope or pillars. The reported advantages of using a consolidated sand rockfill system include the following: raise layout is not critical, the product is denser and has relatively good mobility; it also tends to segregate less than consolidated rockfill. Additionally, consolidated sand-rockfill has a lower angle of repose than conventional cemented rockfill. On the other hand, a relatively good access to stope is required for fill placement. Slurry runoff and control can also be a problem.

2.6.2.3 Consolidated Sand Waste Fill

This fill system (Yu, 1990) involves the use of mine wastes. Generally, waste rock is left in place and consolidated by adding a cement/sand slurry mixture which percolates through it. It is reported that the proportion of cement by weight of sand is approximately 18%. The overall cement content is about 5% of total aggregates. The pulp densities of the cement and sand slurries often range between 55-60% and 65-70% respectively. Fill consolidation is achieved through the use of a higher amount of binder in the mixture. Some reported advantages of this fill system are that the slurry mixture is very mobile and a good stope access is not essential for fill placement. Additionally, it is not necessary to remove waste from the stope. It also permits consolidation to be targeted to specific areas of a stope, such as individual walls. Some cited disadvantages associated with the use of this system, include a requirement for bulkhead construction and it is often difficult to control the flow direction of the sand slurry.

2.6.4 Consolidated Sand Fill

Consolidated sand fill (Yu, 1990) consists of a lean cement-sand slurry mixture with about 5 to 10 per cent cement content. The cement sand slurry is placed after the majority of the stope has been filled with cemented rockfill. It is generally used to tightly fill the remaining void beneath the stope back and the rockfill, thus providing effective roof support. The main advantage of this rockfill product is that the slurry is very mobile and good stope access is not required. Also, the placed product has a low angle of repose. The disadvantages include a requirement for bulkhead control. There are also problems with slurry runoff and control, after fill placement.

2.6.4 CEMENTED ROCKFILL PREPARATION AND PLACEMENT

Various methods (Barrett, 1973; Yu and Counter, 1983; Yu, 1989) are used for preparing and placing cemented rockfill. Preparation methods include surface plants and underground mixing systems. It is reported that at Kidd Creek Mines Limited (Yu and Counter 1983, 1986, 1988), Yu (1989), a simple mixing system suited to conveyor transportation is used to ensure adequate mixing and to minimize segregation of the rockfill product. This consists of a baffled slide or chute, a spray header for the slurry which is pumped from a holding tank, and a conveyor carrying the aggregates. The conveyor discharges aggregates on to an inclined baffle. A spray header mounted over the top of the baffle sprays the binder slurry onto the aggregates as they enter the baffle.

Alternate methods for mixing cemented rockfill aggregates, when trucks are employed for rockfill placement, is by simply spraying of the slurry on top of the rock in the bucket or tray. This system allows the slurry to sufficiently mix with the rock during transport while final mixing takes place as the fill is dumped. Similar to concrete technology (Neville,1987), the key for producing a competent cemented rockfill product has been identified as (Swan, 1985; Yu, 1990; Lidkea and Landriault, 1993) to thoroughly coat and bind all the aggregates with binder slurry during transportation, mixing and placement.

2.6.5 SEGREGATION

Product segregation often occurs when placing cemented rockfills. Differential settling of the fill material causes cemented rockfill aggregates to separate during backfilling. The severity of segregation is a function of the fill raise orientation and the stope geometry. It has been reported that (Barrett, 1973; Berry, 1980; Yu and Counter, 1983, 1986, and 1988) also that during segregation a zone of fine aggregate tends to occur near the impact area, by consuming most of the cement paste and leaving a low cement content rockfill at the perimeter of the fill cone. The measured fill strength in the impact zone has been reported (Yu and Counter, 1983 and 1986) to be higher than in other parts of the stope.

Severe segregation is also reported to occur when stopes are filled by conveyors (Berry, 1980; Yu and Counter, 1983 and 1986; Yu, 1989). This has been attributed to the impact velocity caused by the speed of the belt and the subsequent free fall of the aggregates. In contrast, when a stope is filled by mobile vehicles, only the largest particles have the momentum to travel to the stope wall. The rest of the material fills the stope by progressive slumping resulting in a more uniform product.

Barrett (1973) reported that different cemented rockfill structures were formed in a stope at Mount Isa Mine. The formed structures varied radially from the centre of impact zone towards the stope walls. The coarser rock fragments and the hydraulic fill were reported to have migrated to the outer edges of the fill with an open textured zone alongside the pulverized impact area. The outermost zone comprising of 15 per cent of total volume was found to be as strong as normal cemented fill.

2.6.6 AGGREGATE ATTRITION

Aggregate degradation (Barrett, 1973) or attrition (Yu, 1989) results from the breaking down the rockfill material as they are transported to a stope. It is reported (Yu, 1989) that the attrition of aggregates is proportional to the depth to which the material is transported. Yu, 1989 suggested that aggregate attrition results in excessive generation of fine materials and therefore, should be taken in to account in cemented rockfill mix design computations. It is reported that the introduction of excess fines can often result in a higher demand for additional binder to coat the extra fine material (Barrett 1973; Barrett et al., 1989).

It is apparent from the above description that material segregation and aggregate attrition can have a significant effect on cemented rockfill placement and properties. These parameters are generally considered in cemented rockfill mix preparation, and should therefore be included in the engineering design of composite fill systems.

2.6.7 PHYSICAL AND MECHANICAL PROPERTIES OF CONSOLIDATED ROCKFILL

2.6.7.1 Laboratory Determined Rockfill Properties and Quality Control Methods

Farsangi (1996) has outlined some of the quality control measures used to achieve superior rockfill engineering properties. It is suggested that, quality control measures should be exercised during fill preparation at the backfill plant, during fill transportation and over fill placement in the stope. Some of the recommended procedures included the proper weighing and sizing of fill aggregates, and the use of cementing agents, to ensure that consistent bulk densities are maintained for the fill batches. Additionally, it is recommended that care must be taken to ensure that there is sufficient binder slurries to adequately coat the rock aggregates. Careful control of moisture within the mass of aggregates has also been recommended.

Farsangi (1996) reported that the use of mine water for mixing reduced compressive strength of the cemented rockfill by as much as 60% compared to when potable water was used for mixing. It is reported that in tests performed at the Kidd Creek Mine, it was noted that rockfill manufacturing using 60% fly ash and 40% normal Portland cement binder and recycled underground process water, produced only half the rockfill strength when prepared using similar cementing agent mixed with potable water. The use of recycled process water for mixing rockfill mixtures was also shown to produce lower cohesive strengths, binder or cement hydration; it also produced excessive water bleed-off from the placed rockfill. Regular physical sampling monitoring to detect any significant deviations in the quantity and quality of fill materials used in the fill batching was also recommended.

It has been suggested (Barrett, 1973; Yu and Counter, 1983; Yu, 1989) that the mechanical properties of cemented rockfill are very different from those of cemented paste fill. Also, the segregation(Stone,1993; Reschke, 1993; Bloss and Greenwood, 1998) of cemented rockfill products, can result in a large range of in-situ densities existing in the cemented rockfill masses. It is reported that typical in-situ bulk densities can be 10-20% lower than those measured in the laboratory. Rockfill can have a moisture content ranging from 2% to 5% with an average porosity of 34%. The reported compressive strength of cemented rockfill in situ have been shown to vary from 1.3 to 11 MPa (Yu, 1990).

2.6.7.2 In-Situ Investigation of Cemented Rockfill Properties

At most mine sites, large scale field tests (Gonano and Kirby, 1973) are usually conducted to simulate the actual stope fill operation. Yu and Counter (1983) reported on large-scale cemented rock fill dump tests at Kidd Creek Mines. Table 2.3 summarizes some of the measured physical properties of core specimens and samples taken from the dump tests, together with results from tests on the placed fill. The compressive strengths of tested core specimens ranged between 6 and 10.3 MPa. However, it was later determined from chemical analysis that the cores contained higher cement content, 7.3% when compared to the average of 5% in the laboratory cast samples. This indicated that (Yu ad Counter, 1983) the distribution of cement in the fill piles was probably not uniform.

It is interesting to note from Table 2.1 the differences between the field and laboratory determined compressive strengths and modulus of elasticity values. The 8.4cm cored samples showed higher strength and deformation values than the 15.6cm cast samples. The higher strengths have been attributed to differences in cement composition. No consideration was given to the differences between the two sample sizes in the interpretation of the field test results by the investigators.

Table 2.1 Some physical properties of CRF from drop tests and underground fill specimens (after Yu and Counter, 1983)

Sample	Density g/cc	Curing Period	Ave. Comp. Strength MPa	Elastic Modulus MPa
Three cores 15cm dia.	2.0	28 days	6.9	2.0
Three cylinders 15cm dia.	2.5	28 days	6.1	-
Nine cores 8.4cm dia. (from 838 stope)	2.3	3 months	10.3	3.1
1235 stope; Dump Cone	2.4	4 yrs	11.0	3.8
Fines Layer	-	"	8.5	-
Mid. agg. Layer	-	"	3.5	-
Coarse Agg. Layer	-	"	Estimated 1.3	-

2.7 PROPERTIES AND BEHAVIOUR OF COMPOSITE BACKFILL SYSTEMS

2.7.1 Fill Description and Preparation

A composite-aggregate paste (CAP) backfill may be described as a heterogeneous mixture of materials ranging from cobble-size waste rock aggregates, down to clay size particles. The particle dimensions can cover a wide range of sizes that are similar to glacial till soils (Terzaghi and Peck, 1967; Peck et al., 1974). It is expected that when cemented, a high-density composite fill system could have properties that are similar to a well compacted glacial till soil. The present application of composite backfill in the mining industry has probably resulted from attempts to optimize particle size gradation of backfill materials to improve their strength and stiffness. Various types of composite fill systems are presently being used by the mining industry

industry world wide. The composite fills are sometimes identified by other names including "aggregate fill" (Berry, 1980; Corson et al., 1980; Arioglu, 1983; Barrett et al., 1983; Grice, 1989; McKinstry and Laukkanen, 1989; Raffield et al., 1998; Moss and Greenwood, 1998) and "alluvial/sand or blended paste fill" (Lidkea and Landriault, 1993; Chen et al., 1998). There is however, no uniformity in the description of the fill properties in the available literature.

The following are seen in this study as some probable forms of composite-aggregate paste backfill systems:

- Mill tailings paste containing agglomerates
- Mill tailings paste containing alluvial material (sand or gravel)
- Rockfill aggregates containing mill tailings paste
- Mill tailings paste containing metallurgical by-products (slags)

For the purposes of this review and study, a "conventional" paste fill refers to mill tailings paste backfill containing up to 10% coarse (minus 20μm) aggregates. Various descriptions of blended tailings/sand paste backfill systems are provided in the available literature (Corson et al., 1980; Arioglu, 1983; Landriault, 1992; Lidkea and Landriault, 1993; Landriault and Tenbergen, 1995). These are also considered in this review as types of composite backfill systems. For example, Corson et al. (1980) described a method of preparing a form of composite/aggregate fill in place whereby cemented sand slurry was introduced into a pile of waste rock, and was consolidated by introducing as a means of filling the voids between the rock aggregates.

Grice (1989) describes a project in which cemented aggregates fill consisting of (30%) aggregates, was added to hydraulic fill slurries and successful delivered to stopes. Initially, aggregate was tried as a replacement for rockfill at Mount Isa Mines. Dry aggregate and cemented hydraulic fill (in a ratio of 1:1) was introduced at the top of the stope in a similar manner as cemented rockfill method. The method of placement was discontinued because of inadequate penetration of the hydraulic fill. It is also reported that the cemented aggregate fill segregated when it was placed in a stope.

Wingrove (1993) also describes a "tailings-aggregate backfilling project at South Deep Mine in South Africa". Two types of aggregates were considered. The selected backfill material consisted of aggregate produced underground from development waste mixed with classified tailings. The second selection consisted of mixture aggregates produced underground from development waste and a rod-mill product. The maximum aggregate size of 20mm was selected to permit transportation in a pipeline. Wingrove (1993) reported that the acceptable ratio of pipeline diameter to maximum particle size of 5:1 was used to avoid bridging during hydraulic transportation.

The following mix tailings and aggregates (T/A) ratio by weight were used. T/A (50:50. 60:40 and 70:30). Other material ratios consisting of development waste aggregates (CCW/AG) 80:20, 60:40, 40:60 and 30:70 were also tried. The composite materials were either uncemented or cemented. Large scale placement tests were carried out under simulated stope conditions using two tests paddocks. Geotextile bags were installed inside the paddocks. A backfill blend of CCT/AGG and CMW/AGG of 30:70 were finally used. Placement relative densities were 2.15 and 2.18 respectively.

It was concluded from the simulated placement tests that both types of aggregate and composite waste mixtures displayed relatively good drainage and they were also found to be stable. However, when the geotextile bag was cut, it was reported that neither of the two backfill materials was found to be completely stable one day after placement. Solids loss of about 1 percent of the total placed solids was noted. Also, approximately 2 per cent shrinkage of the material was observed. An optimum blend for both types of backfill were similar and consisted of 60 to 70% aggregates 30 to 40 percent classified tailings or cumulated waste. The stiffness of the Tailings/Aggregate (T/A) was found to be flexible, it was also found to depend on the composition of the fine material. The CMW/AGG did not show much flexibility in terms of stiffness changes. The rate of strength gain after placement was found to be higher for the CCT/AGG backfill than for the CMW/AGG. The in situ shear strength of the backfill was measured using a shear vane. The shear strength of the placed backfill was estimated to be 22kPa after one day of placement.

Swan et al. (1993) studied the compressibility and stiffness of cemented and uncemented rockfill aggregates containing some tailings. The ratio of tailings-to-rockfill aggregates ranged between 5/95 and 20/80 per weight percentage. It was reported that the stiffness of the fill can be controlled by varying the porosity.

2.7.2 Agglomerated Fine Mill Tailings as Coarse Aggregates in "CAP" Backfills

The coarse fraction of a composite backfill system may consist of agglomerates or palletized fine tailings material. Agglomeration can be used to reduce milling wastes and also, as a means of converting wastes into a more useful backfill material. Various investigators including Archibald and Nantel (1986), Amaratunga and Annor (1989), and Boldt et al. (1990) have studied the use of agglomerated fine tailings to supplement underground backfilling materials. Archibald and Nantel (1986) considered immediate freezing of water and tailings pellets as potential backfill aggregates in cold climates. Amaratunga and Annor (1989), outlined a concept of agglomeration of fine tailings and showed how it could be used to supplement backfill materials. They also presented the results of preliminary experimental study of the concept. The environmental and economic benefits to the mining industry in terms of savings on surface tailings disposal requirements were also outlined in the study. They concluded that the proposed concept of mill tailings agglomeration could benefit the mining industry in many ways. It would supplement the aggregate material used for backfill, reduce the waste disposal requirements, conserve land and protect the environment. It could also eliminate the constraints placed upon fine grinding for the liberation of valuable minerals (Amaratunga and Annor, 1989), especially for gold ore.

2.7.3 Mechanical Properties of Composite-Aggregate Paste (CAP) Fill Systems

Because of confusion in terminology, there are no distinctive listings of properties of composite backfills in the published literature. The properties of this type of backfill are often considered to be similar to those of either mill tailings paste fill, depending on the composition of sand (Lidkea and Landriault, 1993). On the other hand, composite fill or "aggregate fill"(Arioglu,1983; Grice, 1989; McKinstry and Laukkanen, 1989; Raffield et al., 1998) properties are assumed to be similar to that of cemented rockfill depending on the content of coarse material.

Boldt et al. (1990), reported on the use of agglomerated pellets containing 10 % cement by weight, as coarse aggregates in laboratory tests. Oven-dried full plant tailings with various amounts of water and cement were added to the aggregate pellets. Cement content for the backfill batch mixes ranged between 3% and 10% by combined weight of tailings and pellets. The proportion of pellets replaced as coarse aggregate in the full plant tailings were 75, 50, and 25 percent by weight. The pulp density of the mixture was 76% by weight of solids.

A maximum 28 days unconfined compressive strength of 2.47 MPa was observed in samples containing 50-50 agglomerates/tailings mixture and 10% cement. This value was to be compared to a set target strength of 8 MPa. It was concluded that even though a target strength of 8 MPa was not achieved, agglomeration could produce a more useful form of fine-grained mine tailings for underground backfill transport and placement. It was also suggested that inactive tailings ponds could be capped with cemented tailings pellets to reduce wind blown dust, and trap native dusts and seed to promote re-vegetation.

2.7.4 Preparation and Placement of Crushed Waste/Classified Tailings Backfill

Raffield et al. (1998) reported on the use of a high quality crushed waste/classified (cw/cct) tailings and crushed aggregates as a special application backfill material. The crushed waste/classified tailings backfill was placed uncemented in a 150m radius around two shafts to permit extraction around the shaft. Porosity of the fill ranged between 28-30%. The mix ratio for cw/cct backfill ranged between 70/30 to 50/50. The success of the stoping was attributed to the high stiffness generated by the cw/cct backfill. It was reported that the fill reduced the back area closure and consequently improved the face and the hanging wall conditions.

Bloss and Greenwood (1998) described a type of "aggregate" fill in use at Mount Isa mine in Australia. The fill is prepared by adding the correct amount of cement to the hydraulic fill to form cemented hydraulic fill. Rock is then placed simultaneously with the cemented hydraulic fill (CHF) at correct placement ratios that ensure good cement distribution in the stope to produce a cemented rockfill (CRF). The report describes an in situ aggregate fill which is placed in a similar manner to cemented rockfill.

2.7.4.1 Summary of "Aggregate" Fill Properties

Arioglu (1983) used a 60 per cent coarse aggregates to 40 per cent tailings ratio in the study on "aggregate" fill. The optimum ratio of composite aggregate/tailings (A/T) fill mixtures has been identified to range between 60-70 per cent coarse and 30-40 per cent fine materials (Arioglu,1983; Grice, 1989; Wingrove, 1993; McKinstry and Laukkanen, 1993; Raffield et al., 1998). Some of this investigations were based on the minus 20mm coarse aggregates similar to that used in concrete technology (Neville, 1987).

There is therefore a need to investigate larger size (minus 152mm) coarse aggregates normally used for preparing cemented rockfill.

2.8 RELATIVE EVALUATION OF HIGH-DENSITY FILL PROPERTIES

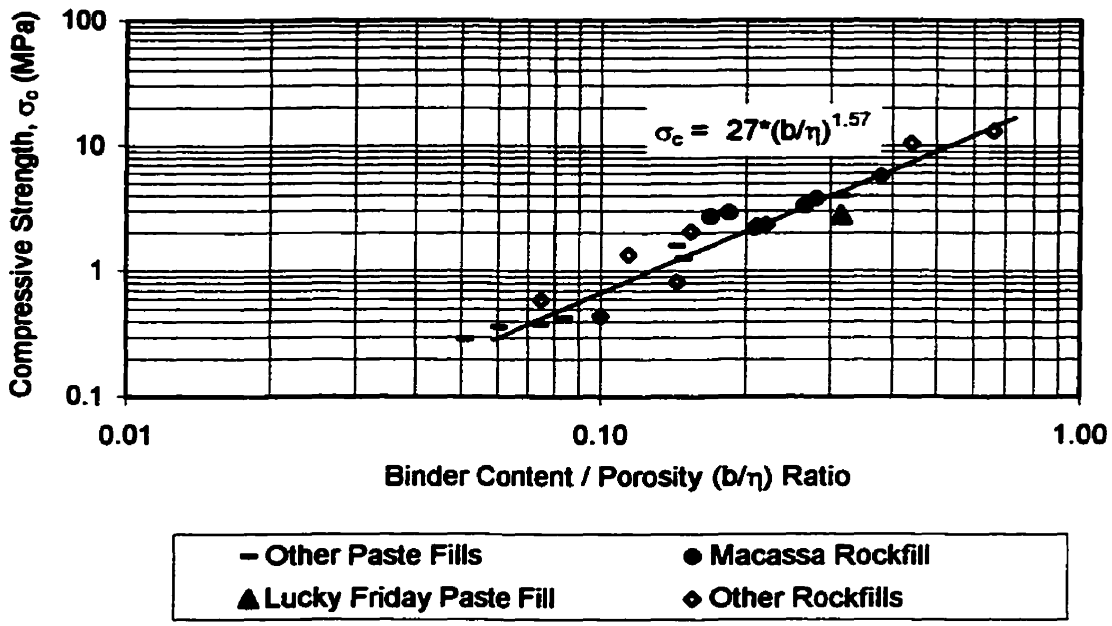
Hedley (1995) conducted a literature review on the physical properties of cemented rockfill and high density paste backfills from mines in North America. Berry (1980) , Wingrove (1993) and Hedley (1995) have concluded that compressive strength and deformation modulus of high density fills are mainly controlled by the cement or binder content, and to a less extent by porosity. Hedley (1995) expressed the compressive strength of paste and cemented rockfill as a function of cement/porosity (c/η) ratio. The deformation modulus of the fill materials were also expressed in terms of uniaxial compressive strength, as shown in Figure 2.2 (a & b). A good agreement was found between compressive strength and cement content-to-porosity ratio by Hedley (1995), as follows:

$$\sigma_c = 26.6\left(\frac{c}{\eta}\right)^{1.58} \tag{2.1}$$

Where:	σ_c	=	compressive strength (MPa)
	c	=	cement or binder content (%)
	η	=	porosity (%)

Hedley also found that there was less scatter in the data than just plotting compressive strength against cement content.

The above findings in effect, suggest that binder composition controls porosity of cemented high-density backfill systems. The findings also indicate the need to find mine backfill systems where void reduction could have significant effects on strength and deformation properties of the fill product.



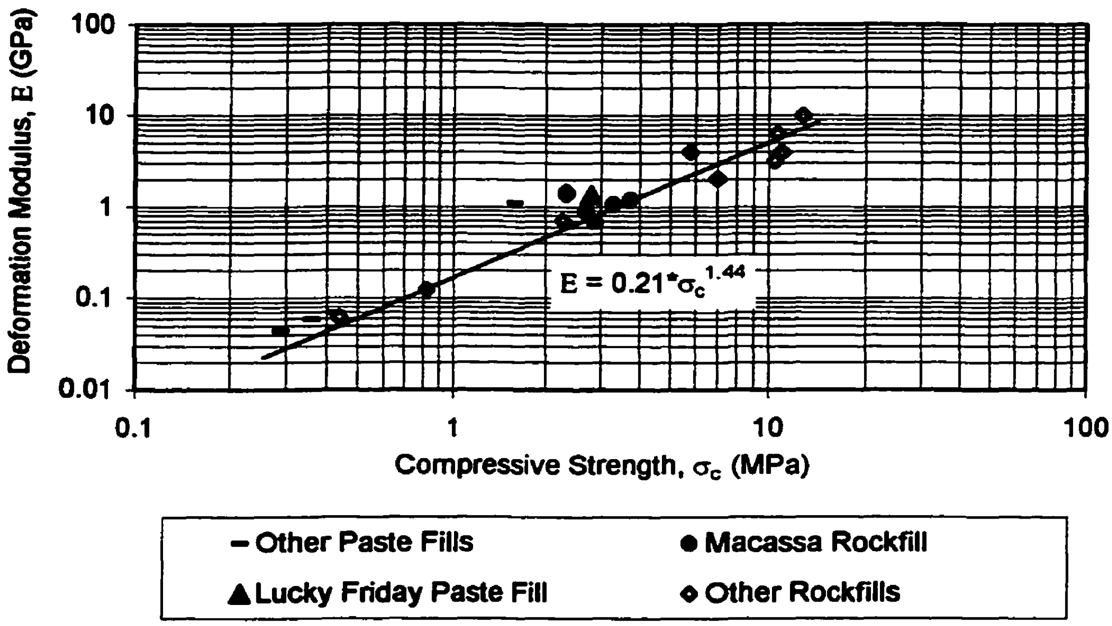


Figure 2.2 (a) Compressive Strength as a function of Cement Contents and Porosity

(b) Deformation Modulus versus Uniaxial Compressive Strength (after Hedley, 1995).

Hedley (1995), compared the laboratory determined properties of cemented rockfill and cemented paste backfills from various sources. It was found that the rate of scatter in the data in (Table 2.2) was much greater for the rockfill than the cemented pastefill. It was therefore, suggested that paste backfill was the better backfill system of the two types of fill.

Table 2.2 Comparison of mechanical properties of pastefill and cemented rockfill samples (after Hedley, 1995)

Backfill Type	Density (Kg/m^3)		Strength (MPa)		Modulus (GPa)	
	Mean	Range	Mean	Range	Mean	Range
Pastefill 10% cement	1920	(1870-2010)	2.74	(2.43-3 .63)	1.37	(0.88-1.71)
Rockfill 5-7% Binder	2000	(1835-2161)	3.24	(2.00-5.63	1.00	(0.48-2.63)

Table 2.2 also suggests the need for further study of Composite-Aggregate Paste (CAP) backfill properties and behaviour in mining. The application of CAP fills could result in the bridging of the wide gap between paste backfill and cemented rockfill properties which has been identified by Hedley (1995).

Hedley (1995), also compared the in situ stiffness of the paste fill and cemented rockfill from various mines in North America. It was found that the stiffness of the cemented paste fill was at the lower end of the range of values as compared to well graded cemented rockfill. The paste fill was found to perform in a superior manner when compared to a poorly graded rockfill. It is interesting to note however, that the comparisons were based on different binder types and compositions. The paste fill samples contained about 10 per cent cement, whereas the binder content for the rockfill samples varied between 5 and 7 per cent. The binder consisted of a mixture of cement and flyash. The particle size distributions and the specific gravities of the fill materials were also omitted from the comparisons.

2.9 OPTIMIZATION METHODS FOR BACKFILL MATERIALS

2.9.1 A Concept of Backfill Material Properties of Optimization

In general terms, the optimization of mine backfill systems refers to the most cost-effective means of improving the operational efficiency of backfill-reliant mines. Optimization is also concerned with the engineering of structurally competent and cost-effective backfill materials and systems including the use of alternative means of fill consolidation (Hassani, 1993). In terms of current practice (Stone, 1993; Reschke, 1993; Landriault, 1995; Farsangi et al., 1996), optimization involves the processes of improving mine backfill performance while reducing production costs. Backfill performance can be measured in terms of improved support potential which is expressed through increases in strength and stiffness of the fill.

Various suggestions have been proposed for improving backfill strength and stiffness. These have included:

a) minimizing fill porosity through aggregates size gradation (Berry, 1980; Thomas, 1981; Arioglu, 1983; Swan, 1985; Hedley, 1995; Lidkea and Landriault, 1993; Swan et al.,1993; Stone, 1993; Wingrove, 1993)

b) moisture control and compaction (Nicholson and Wayment, 1964; Weaver and Luka, 1970, Thomas, 1981; Arioglu, 1983; Arioglu et al., 1986), and through

c) convergence (Hedley, 1995) of the stope walls after placement

Binder composition and curing time can also be considered as backfill optimization parameters because they affect strength development, and therefore influence stope cycle time. The method and condition of fill placement can be considered as a method of backfill optimization because placement can be used to improve backfill properties (Berry, 1980; Corson et al., 1980). The condition of fill placement relates to whether the fill is exposed or confined and it is greatly influenced by the mining method.

2.9.2 <u>Backfill as an Engineered Material</u>

Atchison et al. (1973), Annor and Clarke (1988), Hassani (1993) and others have considered the structural aspects of viewing and designing mine backfill as an engineered product. No specific suggestions have however, been proposed on what nature this design concept should take. With regard to the three high-density backfill systems under study, this concept is recognized as being the most relevant approach for designing and producing composite backfill products from total tailings paste and waste rock aggregate.

Various investigators including Lamos,(1993), Clark (1988), Aref et al. (1989), Vickery and Boldt (1989), Ross-Watt (1989) and Chen and Annor (1995), have reported on the use of total tailings as a suitable material for mine backfilling. The engineering of total tailings as a structural product for backfilling has so far not been fully addressed in the available literature. In general terms, the backfill must be designed to satisfy the structural requirements of the applicable mining method. The design considerations (Chen et al., 1996) should also include the ability of the fill to meet the following essential requirements:

- to be self supporting;
- to apply regional support in convergence control;
- to provide a stable bearing surface for men and equipment and to control dilution control;
- to provide competent stable roof support

The engineering design must also include safety and cost considerations. The fill materials must therefore be optimised to meet the engineering design requirements while minimizing cost. The following is seen as a summary of optimization procedures used in the design of conventional backfill systems based on a review of the available literature:

2.9.3 Particle Size Gradation

The effectiveness of any type of backfill for underground support depends on the inherent material properties, pulp density and conditions of placement. Particle sizes and gradation play important role in the mechanical properties and behaviour of soil-like materials including backfills (Espley et al., 1970). With regard to mine backfill, it has been suggested that the primary objective for optimising particle size distribution is to achieve a well graded aggregate distribution in order to attain optimum porosity (Thomas et al., 1979) and thus, reduce binder consumption and mine operating costs.

Various size distributions have been proposed for backfill materials on the basis of soil mechanics principles (Terzaghi and Peck, 1967; Bowles, 1970), and concrete technology (Neville, 1987). There is no consensus in the available literature on what constitutes optimum size gradation requirements for backfill materials, or how it should be measured. For example, indices such as "The Effective Grain Size (D_{10})", the "Coefficient of Uniformity (C_u)" and the "Coefficient of Curvature (C_c)" are generally used to characterise and quantify particle size distribution of fill materials (Bowles, 1970). These indices are derived as follows:

$$C_u = \frac{D_{60}}{D_{10}} \qquad 2.2$$

and

$$C_c = \frac{(D_{30})^2}{(D_{10})(D_{60})} \qquad 2.3$$

Where

D_{10}	=	grain size at 10% passing (also, the effective grain size)
D_{30}	=	grain size at 30% passing
D_{60}	=	grain size at 60% passing

Well-graded materials usually contain equal representation of all size fractions. For example, C_u values of 4 to 6, and C_c values of 1 to 3 have been proposed (Terzaghi and Peck,1967; Peck et al., 1974; Das,1983; Craig, 1978) as indications of a well graded fill and soil material. Composite backfill and cemented rockfill systems generally contain a broad distribution of fine and coarse aggregates ranging from a few micrometers to several millimetres in size. The particle size gradation of these materials can be expected to be similar to till-soils (Peck et al., 1974). A wide range of coefficient of uniformity ($C_{u)}$ values can therefore be expected for these types of backfill materials.

2.9.3.1 Optimum Particle Size and Distribution for Concrete

In concrete technology, a well established relationship exists between compressive strength and sizes of aggregate materials that are used for concrete mixes (Neville, 1987). The correct choice of particle size distribution results in optimum design of concrete mixes by reducing porosity and thus minimising cement requirements. This approach is widely used as a means of optimizing all types of backfill mix designs. For example, Swan (1985) and others have proposed that the performance of a backfill binder may also be optimized by the correct choice of particle size distribution of the fill material.

It has been suggested by Thomas et al. (1979) that fine particles in a well graded backfill tend to fill the voids between larger particles thus reducing the volume which the cement gel must occupy in order to produce a stronger bonding Figure 2.3. It has also been shown (Clark, 1988; Chen and Annor, 1995) that soon after placement, and before the fill is fully set, an uncemented tailings backfill has a lower porosity than when cemented.

Additionally, it has been shown (Thomas et al., 1979; Vickery and Boldt, 1989; Ross-Watt, 1989) that backfills containing high proportions of fine material, developed higher compressive strengths than fill products containing medium or coarse materials.

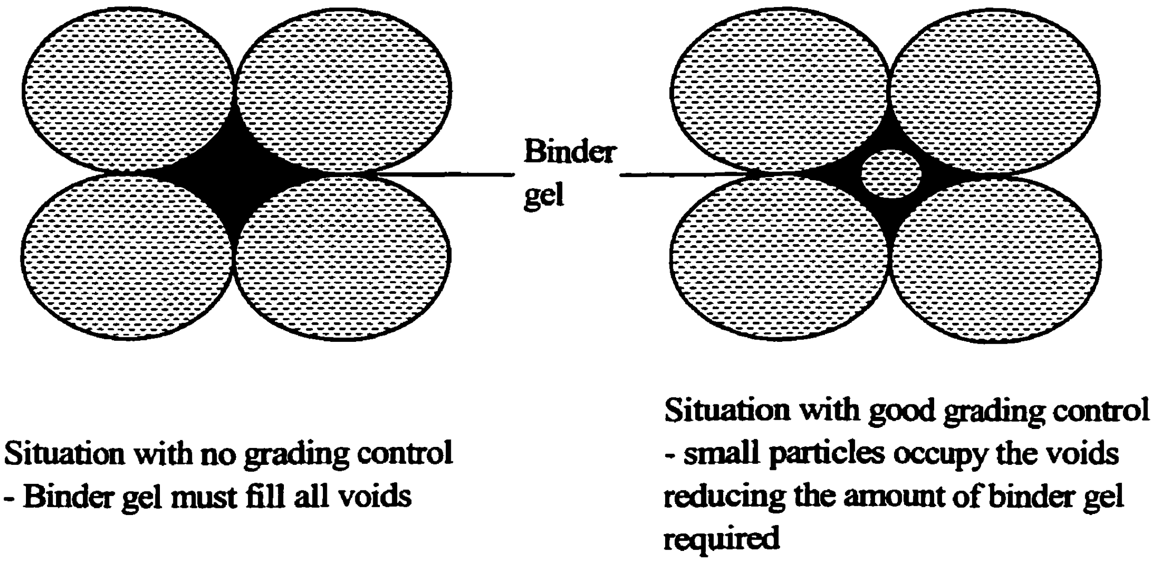


Figure 2.3 Model demonstrating benefits of fine particles in hydraulic fill (after Thomas et al., 1979)

It has been found (Clark, 1988; Chen and Annor, 1995) that tailings fills with higher specific gravities, tend to have higher porosities at the same water to solids ratios. The addition of cement to tailings fills increases the ultra-fine materials content of the fill. These materials, together with the binder, retain more water between the particles resulting in higher placement porosity of the cemented fill when compared with the uncemented fine tailings fill. The application of higher specific gravity aggregates for composite fill preparation merits investigation.

2.9.3.2 Designing The Optimum Particle Size

It is a commonly accepted fact in conventional hydraulic fill technology (Thomas et al., 1979), that the particle size distribution should be chosen to achieve high shear strength parameters (friction angle and cohesion), to increase binder performance and thus, to reduce costs. To help determine the effects of aggregate grading on strength, Swan (1985) applied concepts from concrete technology where the amount of binder consumption and strength development have been related to the size and distribution of the materials. This relationship is based on the Talbot curve. The Talbot curve is often used in the concrete industry to estimate the cement requirements for achieving an acceptable level of strength for poorly graded aggregates.

The Talbot curve is defined by:

$$P_{(u)} = 100(u / u_m)^n \qquad 2.4$$

Where:	$P_{(u)}$	=	percent passing by weight for sieve size u and for particle size u_m
	n	=	experimental constant

Experiments by Talbot and Richart (1923) showed that a specific value for exponent "n" would minimize porosity of a concrete mix, thereby maximising strength. Although this might be true from a theoretical standpoint, it is not considered as a practical solution for optimising a backfill mix. This is because in practice, minimising fill porosity generally results in harsh hydraulic fill mixtures with attendant placement difficulties such as, segregation and poor drainage characteristics (Swan, 1985).

2.9.3.3 Optimization of Backfill Material Size Gradation Using "Binder Number"

Swan (1985) has proposed that a Talbot-based design procedure should be used for designing backfill mixes. The applicable mixes comprised of a mixture of composite materials containing cement. The composite mixtures are described as "aggregate material bound by a matrix consisting of fine aggregates and cement or other binder". The shape effects of the particles are neglected in the proposed theory. It is assumed that all aggregate particles are spherical also, that the aggregate sizes are randomly arranged, according to the proposed mix design model.

According to Swan (1985), there are two important theoretical parameters which are relevant to the mix design. These are: "d_{avg}" which is the mean free distance between aggregate particles in the mix, and "a_p", the aggregate's specific surface area for a constant water cement ratio. Knowledge of the fill aggregate particle size distribution, its porosity and volumetric cement content allows the calculation of a_p and d_{avg} from a series of equations.

A "binder number" which is a dimensionless number equal to $(C_v / (d_{avg} a_p))$ is developed from the series of equations. The "binder number" can be calculated and related to the expected strength of the backfill mix by a relationship that shows that the unconfined compressive strength (known from free-standing height requirements of the fill) is proportional to $(C_v / d_{avg} a_p)^k$ where "k" is a constant.

Swan (1985), plotted the "binder number" against unconfined compressive strength for 68 selected mine backfills and concretes from the available literature. It was reported that a very good correlation was found to exist between binder number and compressive strength. It was also suggested that an increase in strength could be achieved by increasing the binder number. The following empirical relationship between binder number and unconfined compressive strength (UCS) was proposed based on a similarly established relationship by Arioglu (1983) on cemented aggregate fill:

$$UCS = 0.283\left(C_v / d_{avg}\ a_p\right)^{2.36} \qquad 2.5$$

where Cv = volume of cement in a unit dry volume of freshly placed backfill expressed as a weight percent.

Swan (1985) has further suggested that an optimized grading, based on the above design approach, can help reduce the amount of cement needed to achieve a given backfill strength. It has also been suggested that (Swan, 1985) other findings exist in concrete research which may be of interest in backfill mix design. These include the strong dependence of strength on inter particle distance, on the inverse of the aggregate's specific surface area, and on "some power of cement content".

Most of the strength data analyzed by Swan (1985) were developed using concrete samples where the cement content was much higher than that normally used for backfills. Other studies (Boldt et al., 1993) have however, shown that some of the concepts used in concrete technology mix design, do not often apply to the design of backfill mixes. For example, in total tailings pastefills, the porosity is probably largely dependent on moisture content than inter-particle spacing. Also, unlike concrete technology where binder composition can be as high as 35% of the volume of aggregates, in most backfill systems, the amount of binder used per dry weight of fill material is relatively small and often less than 10%.

2.9.4 EXAMPLES OF PRACTICAL APPLICATIONS OF CONCRETE TECHNOLOGY PRINCIPLES TO BACKFILL MIX DESIGN

Boldt et al. (1993) investigated some basic similarities between concrete testing and the testing of cemented mill tailings fills. The effects of "Fineness Modulus" on compressive strengths of total tailings backfill samples containing sand were investigated as part of the study.

Fineness Modulus (FM) is a relationship generally used in concrete technology to express the fineness of aggregate gradation. It is defined as follows:

FM = (Commutative percentage of material retained on US sieve screens 4 to 100)/100

where the screen sizes are Nos 4, 8, 16, 30, 50 and 100.

It was also reported (Boldt et al., 1993) that other previous studies had shown that wide variations in sand grading had no effect on the compressive strengths of mortar or concrete samples. In an alternate reported study (Bureau of Reclamation (1981) in Boldt et al., 1993), mortar is defined as a mixture of cement, sand and water; where as concrete is defined as containing cement, sand, water, gravel, crushed rock or another aggregate.

Two mixes containing 4 and 6 per cent cement were prepared (Boldt et al., 1993) in the reported study. Water was kept constant and the slurry density of the mixture was maintained at 80 percent solids by weight. For the mix containing 4 per cent cement, the compressive strength increased with the amount of tailings when compared to the 100 percent classified sand. For the mixture containing 6 per cent cement, the strength was noted to decrease initially before increasing..

The differences in the compressive strengths of the samples containing 4 and 6 per cent cement were attributed to increases in the amount of fines content. It was suggested that the increased amount of fine material resulted in increased surface area of the particles which required more water for wetting. It was also suggested that the effective consumption of water for particle wetting and cement hydration increased as the amount of fines increased. This resulted in less available free water for hydration because the water was trapped within the material mass. The addition of fines resulted in a better graded aggregate mix. It was found that the compressive strength increased as the particle size of the material became finer and the FM decreased, when slump and water/cement ratio were kept constant.

Boldt et al. (1993) also reported that in tests of structural concrete, when water/cement ratio and slumps were held constant, changes in sand grading was found to have no effect on the compressive strength of the mortar or concrete. It was reported further that there appears to be an optimum water/cement ratio and grain size gradation which favoured strength development in tailings fills. Particle size gradation was found to have little effect on compressive strength development at higher water/cement ratios (7 to 11). Additionally, it was found that the unconfined compressive strength more than doubled when the water/cement ratio was decreased to 4.42.

These findings (Boldt et al., 1993) also suggest that for fine tailings fills, binder content and moisture control seem to have more effect on strength development than aggregate gradation. This approach seems to support the fact that water requirements tend to increase for concrete and similar products (Neville, 1987) including backfills, as the fines content is increased for any given size of coarse aggregates.

2.9.5 EFFECTS OF COARSE AGGREGATES ON STRENGTH PROPERTIES ON TAILINGS FILLS

Arioglu (1983) studied the effects of coarse aggregate addition on the strength properties of cemented tailings fill. This investigation can be considered as a study on the optimization of aggregate size gradation. The material consisted of crushed marble aggregates and tailings. The specific gravities of the tailings and the marble aggregates were 3.01 and 2.65 respectively. The coefficient of uniformity values were 4.7 for the tailings, and 4.89 for the marble aggregates. An ideal grading for the combined material which resulted in maximum density and thus maximum strength was obtained and was expressed by the following equation:

$$P = 124\left(\frac{d}{D_{max}}\right)^{0.47} \qquad 2.6$$

Where: P = percentage of material smaller than size "d"

D_{max} = maximum particle size

The combined material consisted of 40% tailings and 60% marble aggregates with particle sizes ranging between 30mm and 0.15mm. Three mixes were prepared with the following total aggregate/cement ratios: 5:1, 10:1 and 20:1. Water/cement ratios ranged between 0.72 and 2.21.

The test results showed that compared to the cemented tailings fill, the strength properties of the cemented "composite" marble-aggregate and tailings fill increased substantially with increased cement content and decreased water/cement ratio. The strength properties were found to be dependent on cement content and water/cement ratio. It was concluded that the addition of coarse aggregates to the tailings mix resulted in immense increases in the strength properties of the cemented aggregate fill. The test results showing the strength parameters are summarised in Table 2.3

Although the blending of coarse aggregates with the tailings material played some role in the substantial increase in strength properties, the observed increase in strength parameters were mainly attributed to cement content and water/cement ratio. Changes in compressive strength due to the size gradation of the material was considered to be probably minimal.

The reported results by (Mitchell and Wong, 1982; Boldt et al., 1993; and Chen and Annor, 1995) suggest that moisture content and binder composition seem to influence the development of strength properties more than particle size gradation for backfill materials. This suggests that unlike concrete technology where mix design depends on optimal aggregate size distribution, there is no optimum size distribution for backfill materials.

Binder type and composition, moisture content and water/binder ratio seem to control the development of strength properties in backfill mixes.

Table 2.3 Comparison of cemented fill parameters (after Arioglu, 1983)

Total aggregate/ cement ratio	5/1			10/1			20/1		
Fill type	A	B	% increase	A	B	% increase	A	B	%
Cement, kg/m^3	288	352	20	156.5	195	24.6	81.6	103	26
Water/cement by wt.	1.47	0.72	-50	2.72	1.22	-55	5.24	2.21	-57
Compressive Strength, kg/cm^2	43.56	131.77	200	7.98	57.96	626	4.76	23.77	399
Tensile strength, kg/cm^2	6.57	19.77	200	1.57	9.84	526	-	4.11	-
Cohesion, kg/cm^2	8.43	25.52	202	1.77	11.97	576	-	4.95	-
Static elasticity modulus, kg/cm^2	1812	92063	4980	541	39062	7120	-	14705	-

A : Cement fill produced from only tailing
B : Cemented aggregate fill produced from coarse marble aggregate and tailing

2.9.6 MOISTURE CONTROL

Conventional hydraulic sand and mill tailings backfills have always relied on the use of moisture control through drainage, as a means of strength improvement. The available literature on mine backfill technology including the following: (Minefill Conference Proceedings, 1971- 1998; Rawling et al., 1966; Corson, 1970; Thomas et al., 1979; Clark, 1988; Chen and Annor, 1989) are full of references on the subject. For example, the main reason for tailings classification by cycloning is to improve drainage of the fill mass and thus reduce the water/cement ratio of the product after placement in a stope. It is apparent from the available literature on the subject that binder content and moisture control are probably stronger determinants for strength gain in tailings fills than particle size gradation. Figure

2.4 shows the effects of water/cement ratio on backfill strength (Lerche and Renetzeder, 1984).

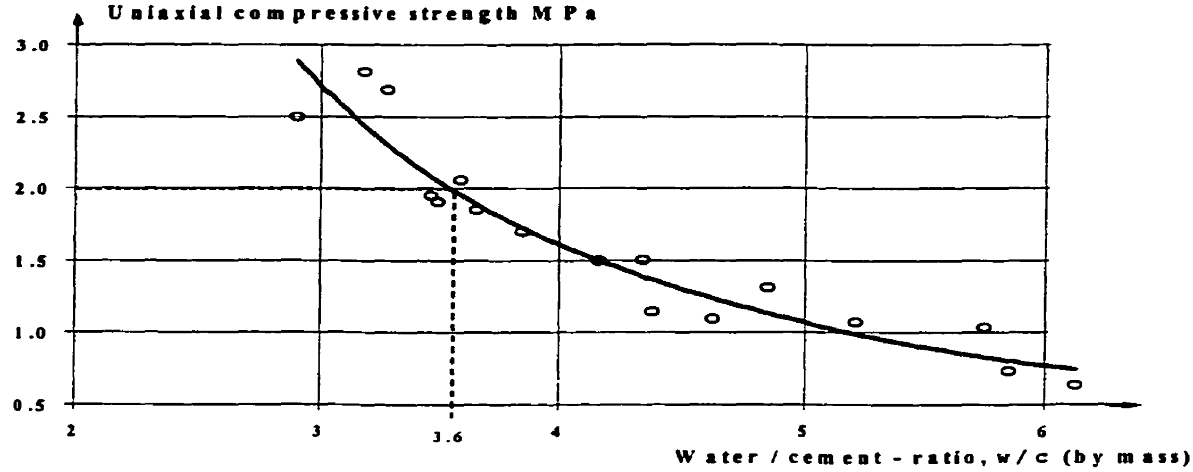


Figure 2.4 Effect of water/cement ratio on backfill strength, (after Lerche and Renetzeder, 1984)

2.9.7 BINDER CONCENTRATION AND CONSOLIDATION TIME AS METHODS OF OPTIMIZATION

Binder and curing time can also be considered as optimization parameters for backfill mix design. The rate of strength development for most backfill binders depend upon their physical and chemical make up and also how they react with the fill material. The physical make up relates to the fineness of grind (ASTM C-204, 1987) which is often expressed in terms of Blaine fineness.

Extensive studies on backfill binder usage have been published in the literature. Several researchers including Thomas (1973a and b), Thomas and Cowling (1978), Manca et al. (1983), Yu and Counter (1986,1988), McGuire (1978), Hassani (1989), Douglas and Malhotra (1989), Hopkins and Beaudry (1989), and others have reported on, or investigated the use of alternative binders such as slags, flyash, anhydrite and other products for consolidating backfills. It has been concluded that the application of binder improves the cohesive properties and stiffness of backfills. Additionally, alternative binders to ordinary Portland cement can provide cost-effective means of consolidating backfills.

The rate of strength development and the ultimate strengths are among the essential requirements for assessing the effectiveness of a binder. Fly ash and slags are known to develop strength at slower rates (Malhotra, 1983) than Portland cement, but they have also been shown (Douglas and Malhotra, 1989; Hopkins and Beaudry, 1989) to achieve higher ultimate strengths than ordinary Portland cement, over long curing periods.

2.9.7.1 Binder Alternatives

The effectiveness of some of the alternative binders for improving backfills have also been investigated and proven. It has been concluded that considerable savings can be achieved by minimising the percentage of Portland cement used (Yu and Counter, 1988; Douglas and Malhotra, 1989; Hopkins and Beaudry, 1989) by partially replacing it with cheaper binder materials. Flyash and slags have been widely used as a partial replacement for up to 50% of ordinary Portland cement in backfill operations. Ground blast furnace slag and ground non-ferrous slags have also been suggested as common replacement binders. Hassani (1989) and others have extensively evaluated the use of Anhydrite as a binder alternative for backfill materials.

Petrolito et al. (1998) studied the strength of backfill stabilized with calcined gypsum. Tailings were obtained from four different sites to simulate materials to widen the scope of the study. It was suggested that calcined gypsum has cementing properties and is cheaper to produce than cement. Unconfined compression strength (UCS) on 50mm diameter by 100 mm long cylinders were used to determined calcined gypsum stabilized backfill strengths. There was significant variations in the strength obtained with different tailings when cement was used as a binder. In terms of the cement stabilized fill samples, it was reported that strength variation was noticeable with samples containing more than 6% cement.

The range of Water/Calcined Gypsum (W/CG) ratios covered in the mix design was 0.83 to 3.0. The range was selected because it was found that the slurry was difficult to mix below W/CG ratio of 0.83. Also, above 3.0, the strength was found to be insignificant. The following empirical relationship was established that can be used to reliably predict the strength of the stabilized backfill in the normal range of strength used in practice.

$$\text{U.C.S. (MPa)} = 2.084\ (\text{W/CG})^{-2.322}$$

The rate of strength development and the ultimate strengths are among the essential requirements for assessing the effectiveness of a binder. Fly ash and slags are known to develop strength at slower rates (Malhotra, 1983) than Portland cement, but they have also been shown (Douglas and Malhotra, 1989; Hopkins and Beaudry, 1989) to achieve higher ultimate strengths than ordinary Portland cement.

2.9.8 CONFINEMENT AS A METHOD OF BACKFILL OPTIMIZATION

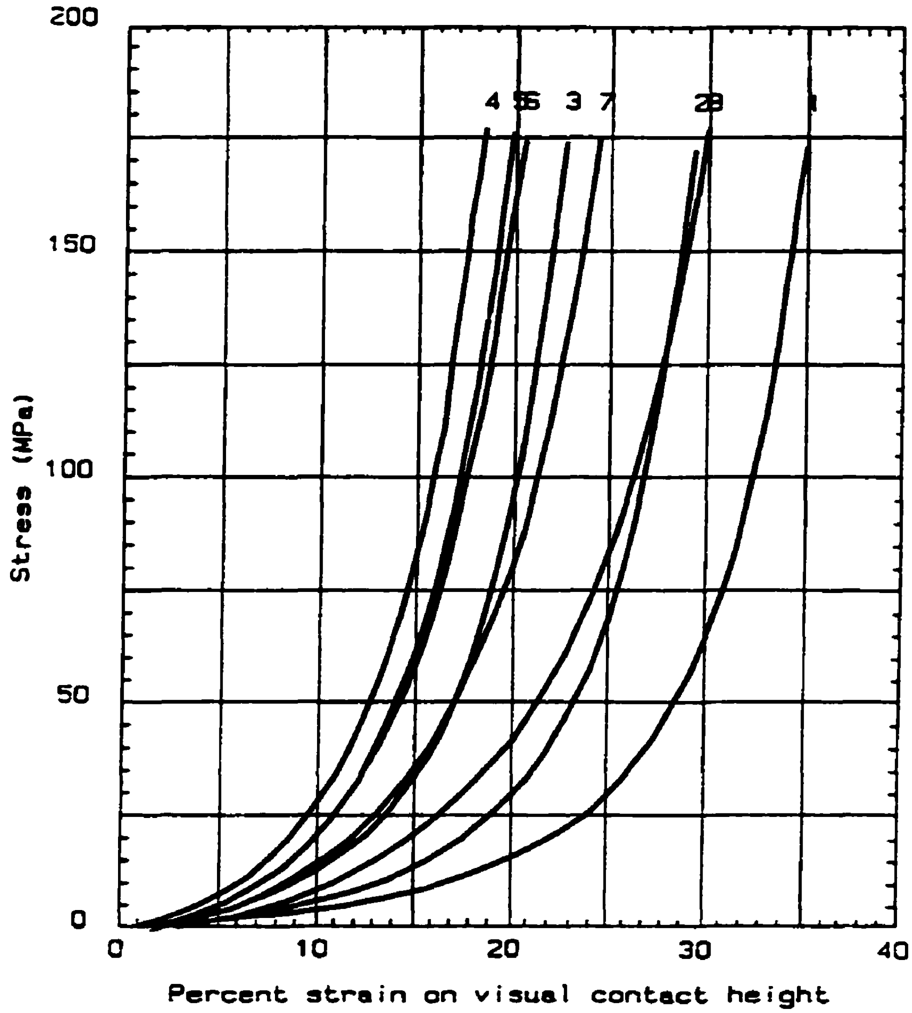
Placed fill is generally subjected to confinement (Moruzi, 1978; Hedley, 1987) due to the convergence of surrounding rock mass and this enhances the stiffness of the fill. Only a slight increase in stiffness often occur under stable ground conditions where there is little or no convergence. Backfill stiffness generally increases under highly stressed conditions, as convergence develops in a stope (Hedley, 1987, 1995).

The effects of confinement on the behaviour of cemented hydraulic backfill samples have been reported by various investigators including Nicholson and Wayment (1964), Moruzi (1978), Mitchell and Wong (1982) and Hunt (1989). Moruzi showed that a confining pressure of only 25 psi (0.17 MPa) increased the strength of fill samples to 125 psi (0.86 MPa). The strength of the fill samples increased further to 300 psi (2.10 MPa) under a confining pressure of 75 psi (0.52 MPa). In comparison, samples containing about 1:30 cement-tailings developed unconfined compressive strength of 30 psi (0.20 MPa) after 14 days of curing; the strength increased by 65% to 50 psi (0.34 MPa) at 90 days.

Hunt (1989) also reported on both uniaxial and triaxial compressive strengths of cemented unclassified (full plant) tailings backfill samples. The results showed a general increase in compressive strength with increasing binder composition and curing period and decreasing moisture content and porosity. Confining pressures of 170 kPa and 340 kPa were used for the triaxial tests. The test results showed increases in shear strength with confinement.

2.9.8.1 Effects of Confinement on Cemented Rockfill Behaviour

The only source of triaxial cemented rockfill data in the published literature is from investigations by Gonano et al. (1978). The maximum confining pressure for the study was 2.0 MPa. The test results showed an increased deformation modulus with confining pressure. Wingrove (1993), Swan et al., (1993) and others have studied the compressibility characteristics of aggregate fills and have concluded that the stiffness of blended fills can be regulated by variation of porosity. The porosity can also be varied by the proportion of fines (Wingrove, 1993) in the backfill as shown in Figure 2.5.



NO.	% CT	POR.	REF. NO.	NO.	% CT	POR.	REF. NO.
1	0	39.6	23/89	5	40	26.7	11/90
2	10	34.1	18/90	6	50	28.3	13/90
3	20	27.3	7/90	7	75	34.3	15/90
4	30	24.4	9/90	8	100	41.5	1/90

Figure 2.5 A comparison of load-bearing properties of eight mixtures in confined compression. (after Wingrove, 1993)

2.10 SCALE EFFECTS ON BACKFILL MATERIAL PROPERTIES

Input data is critical for the accuracy of any numerical modeling simulation program. The reliability of any solution obtained from the numerical modeling predictions can only be as accurate as the input data on which they are based. Ideally, input data for numerical modeling should originate from in-situ monitoring and instrumentation of backfilled stopes. Generally, because of difficulties which can arise from meeting production requirements, coupled with the high cost of conducting field tests, data for numerical modeling are often based on small scale laboratory properties.

The sizes of test samples are known to play an important role in the laboratory assessment of the behaviour of geotechnical materials including backfill. Generally, the observed differences between mechanical properties of laboratory scale samples and in-situ material properties are attributed to the differences in scale. For materials such as soil and rock, these differences have often been the source of exhaustive investigations. Figure 2.6 shows the effects of test sample geometry on the mechanical properties of rock specimens (Hoek and Brown, 1980).

Various scaling factors (Hoek and Brown, 1980) have been proposed for converting laboratory observations to anticipated in-situ conditions for these materials (Figure 2.6). With regard to mine backfill, there are very limited studies which have involved the effects of sample size on material behaviour in the published literature. As a matter of acceptable practice, laboratory test samples generally consist of cylinders with diameter/height ratios of 0.5.

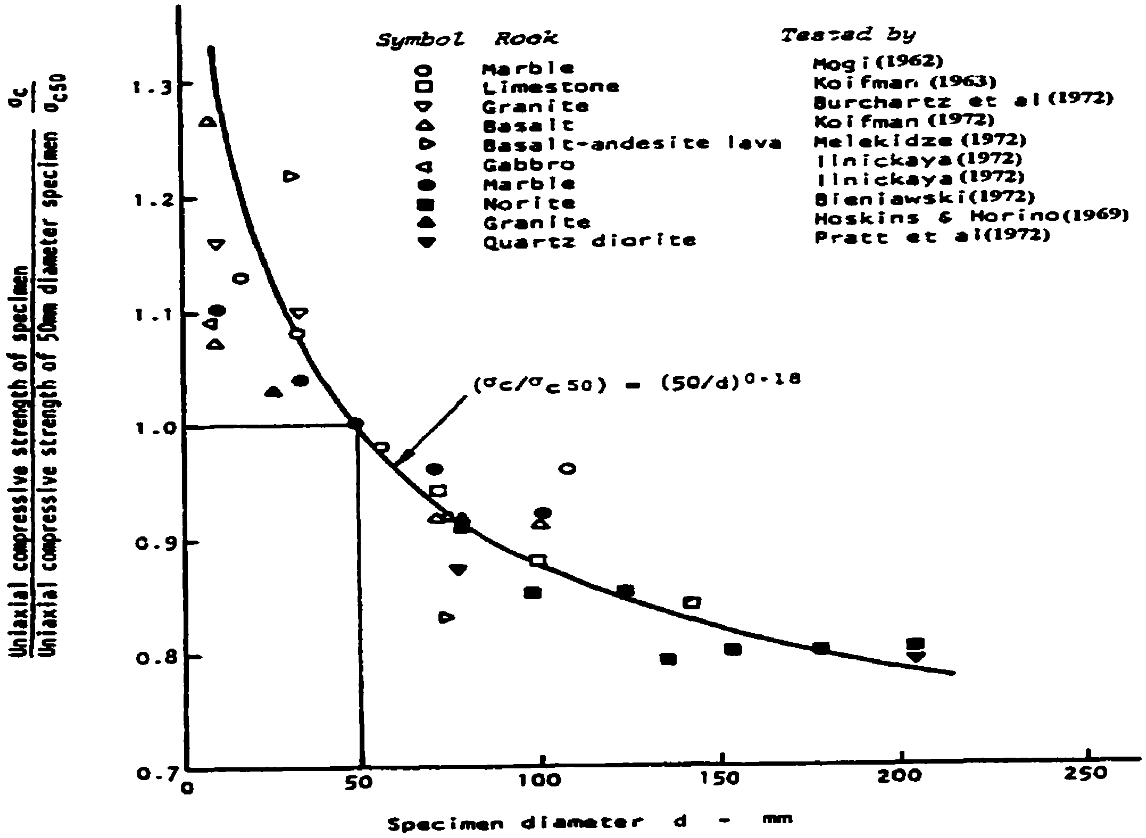


Figure 2.6 Influence of specimen size upon the strength of intact rock (after Hoek and Brown, 1980)

The published studies on scale effects generally relate to rock mechanics (Vutukuri et al., 1974) and civil engineering projects (Marachi et al., 1972). With regard to mine backfill, Gonano and co-workers (1978) conducted triaxial compression tests on large size cemented rockfill samples. Barrett et al. (1983) reported on the scale effects of cemented gravel fills. The strength of the large scale specimens (450 mm in diameter by 900 mm high), containing Portland cement alone was found to be approximately 60% of the laboratory specimen containing scaled down gravel.

Also, Lamos and Clark (1993) investigated the effects of specimen geometry of backfill material properties. They tested specimens with square cross-sections, 48.8mm high and of various widths; width to height ratios ranged between 0.5 and 14.0. Figure 2.7 shows a selection of the stress-strain curves from the study. Lamos and Clark (1993) concluded that the compressive strengths of backfills in the initial elastic response region, is independent of the backfill mass geometry. Also, they found that the compressive strength of cemented backfills at high stresses is dependent on the width/height ratio of the test sample. Other than the above reported studies, no other previous account was found in the available literature, regarding laboratory studies in which the effects of specimen sizes of up to 457mm diameter by 914mm high had previously been made for mine backfill materials.

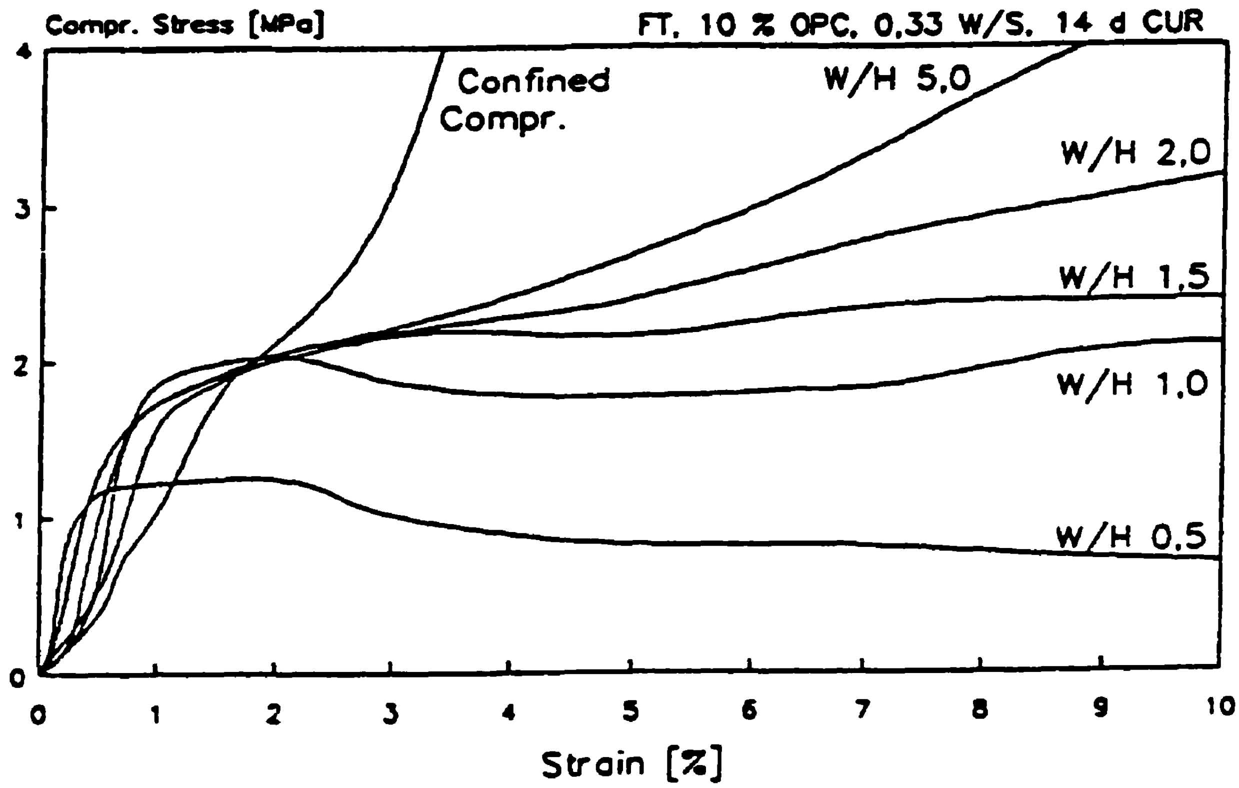


Figure 2.7 Stress/strain responses of cemented backfill samples of increasing width/height ratios (after Lamos and Clark, 1993)

Carefully determined laboratory scale effects test results can play an important role in estimating the potential backfill behaviour in the field especially when developing new backfill systems.

2.10.1 Laboratory Test Specimen Sizes for Backfill Materials

The literature on backfill indicates that reported fill strength data are based on a variety of test sample sizes and geometries. Testing of both cylindrical and cube shaped specimens have been reported. Sample sizes have ranged from 25mm diameter to 150mm diameter cylinders or cubes. Length to diameter ratios have ranged between 1:1 and 1:2.5. There have been instances where no sample sizes were indicated.

The literature (Thomas, 1980; Ross-Watt, 1983; Grice,1989; Boldt et al., 1990; Brackebusch, 1994), also indicates that the specified unconfined compressive strength requirements for cemented stabilized backfills have ranged between 0.5 and 8.3 MPa. These specifications have generally been based on different sample sizes and sometimes even, specimen geometries. For example, test sample sizes of 50mm diameter and lengths ranging between 100mm and 175mm have been cited in the literature (Aref et al.,1989; Hunt (1989); Vickery and Boldt (1989); Hopkins and Beaudry (1989) used 75 mm (3") diameter by 150mm (6") long test cylinders for hydraulic and paste backfill testing. Brechtel et al. (1989); Quesnel et al. (1989) used 150mm by 300mm for testing cemented aggregate fill and rockfill samples. On the other hand, Thomas et al. (1989) tested 100mm by 100mm (cylindrical) fill specimens. Arioglu (1983) determined the compressive strength and elasticity characteristics of fill material by testing 15cm by 30cm cylindrical specimens. Knissel and Helms (1983), investigated the strength of cemented rockfill using 325mm diameter (h/d=1) and 100mm diameter (h/d = 2) test cylinders. On the other hand, Barrett et al. (1983), tested 450mm diameter cemented gravel cylinders for strength determinations.

There have been conflicting observations on the effects of specimen size on fine grain rocks. It is reported (Obert et al., 1946) that the strength of some fine grained rock materials are unaffected by specimen size although, studies on coal samples (Hustrulid, 1976, Bieniawski, 1997) have indicated contrary results.

The cited literature references have shown that there is variability in terms of laboratory test specimen sizes for backfill materials. The effects of this variation on the engineering design of backfill systems are probably offset by the selection of relevant analytical models with the correct safety factors. The laboratory test results may also be used to infer backfill mass behaviour in situ, if different specimen sizes can be tested to develop empirical relationships.

Reschke (1993) also reported on the effect of scale on laboratory tests. It was shown that increasing sample diameter and aggregates sizes resulted in lower compressive strengths (Figure 2.8). The results were found to be consistent with those of Falconbridge's Kidd Creek Mine where the rockfill averaged approximately 66 percent of the laboratory cylinders and about 90 percent for samples of 300mm diameter Reschke (1993).

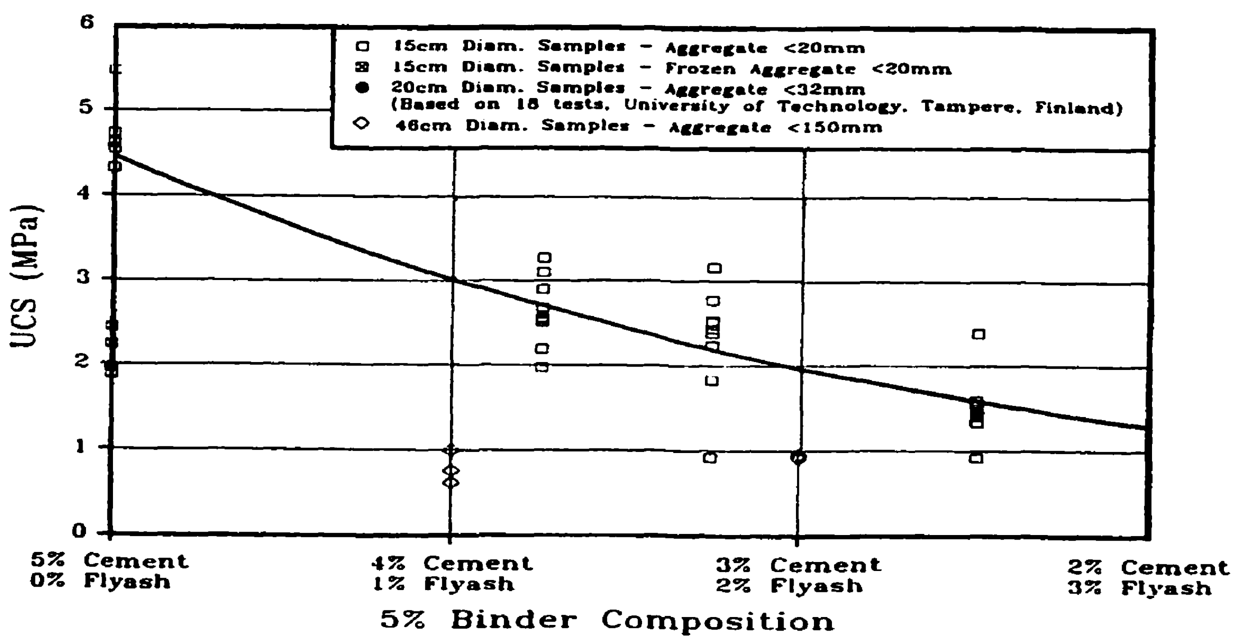


Figure 2.8 Unconfined Compressive Strengths of Rockfill (after Reschke, 1993)

2.11 ENGINEERING DESIGN RATIONALE FOR HIGH-DENSITY BACKFILL SYSTEMS

2.11.1 Approaches to Conventional Backfill Design

Generally, backfill is designed to be capable of sustaining both the gravitational loading of the roof material as well as the dynamic effects applied during blasting. Several investigators have reported on the requirements for strength development with regards to conventional hydraulic backfill systems. Unconfined compressive strength (UCS) has been identified (Thomas et al., 1979) as one of the most important parameters to be considered when dealing with cemented backfill systems. The unconfined compressive strength requirements for cemented backfills are determined by defining the height and width of the most likely fill exposures in a stope. The published reports contains a limited account of UCS data. For example, in order to provide sufficient support, a range of values has been suggested. A design requirement of 8.3MPa UCS has been proposed for a cemented paste backfill (Yu, 1989) operation (Boldt et al., 1990). Similarly, the 28 day design target strength for typical rockfill operations have ranged between 2.3MPa and 7.0 MPa. Yu (1989) and Hassani and Bois (1992) have reported that the compressive strength of Quebec mine backfills ranged between 0.24MPa and 4.30MPa after 28 days curing. The variability between backfill strength requirements reported in the published literature is due in part to the site-specific requirements based on the mining method.

2.11.2 Functions of Backfill in the Mining Cycle

The application of backfill in mining satisfies various essential functions in the mining cycle depending on ground conditions and operational requirements. In terms of "structural" requirements, the following have been identified as some of the roles and purposes of backfill in conventional backfill-reliant mining operations (Thomas et al., 1979, Chen et al., 1996) (Figure 2.9):

1. In pillar recovery operations, the fill is expected to act as a free standing pillar which is unsupported over a significant vertical height, Figure 2.9 (a). The stronger the backfill, the greater the unsupported vertical height. Stabilized backfills develop higher free-standing heights. The use of composite fills could further increase the free-standing height of a fill mass or else, reduce the binder requirements.

2. The fill is used to apply regional support in convergence control. In this role, the fill must have sufficient stiffness to resist the movement of the surrounding rockmass into the mined void. This is generally achieved by using the fill to provide passive support to improve conditions in the void adjacent to an excavation, Figure 2.9 (b). In this regard, tight filling to the back of a stope is an essential requirement for the global stabilization of the mine. A low porosity and stiff fill is required for satisfying this condition.

3. In conventional cut-and-fill operations, the backfill must serve as a bearing surface to support mining activities and assist in controlling the dilution of the mined ore. The bearing capacity of the fill therefore becomes an essential consideration in this role, Figure 2.9 (c).

4. The backfill is used to provide a competent and stable roof support, as in undercut and fill mining methods, Figure 2.9 (d). Tight filling is also an essential requirement in this regard and calls for the use of a low void ratio backfill.

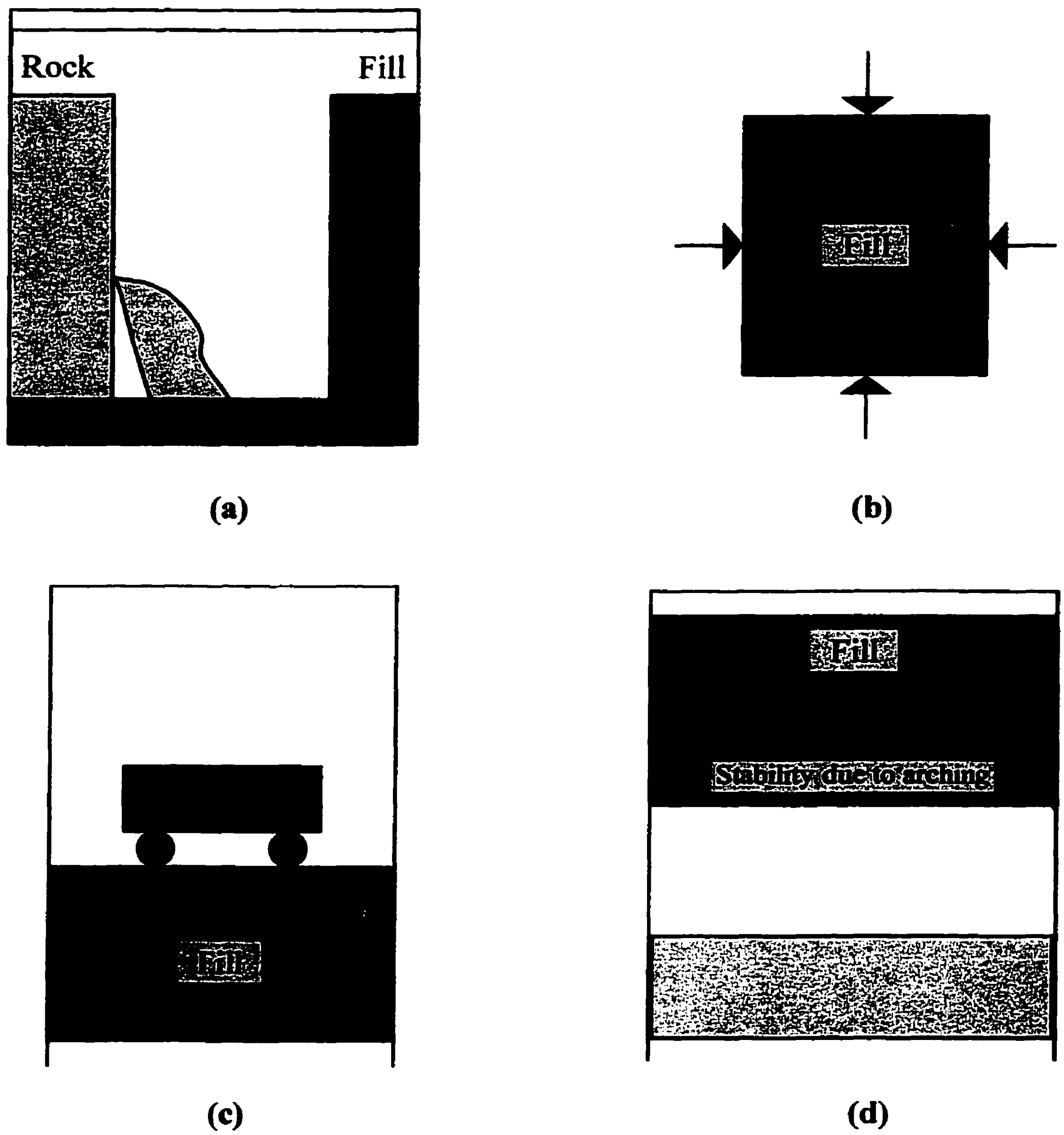


Figure 2.9 Functions of backfill in the mining cycle (after Chen et al., 1996)

In order to meet the preceding design requirements, it is important that any backfill system employed, possess sufficient strength and stiffness. Stope size and geometry, and mining sequence also have significant influence on backfill strength and stiffness requirements. Empirical relationships have been developed, based on failure models, to define various backfill strength requirements for specific mining conditions. Because conditions can vary from one mine site to another, strength and deformation requirements tend to be site specific. For example, in a pillar recovery operation, a compressive strength of 1.0 MPa may be required for a certain free standing height of backfill. It is reported (Yu,1989) that in a pillar recovery operation at Kidd Creek Mines Ltd., that the exposed fill face was designed for a height of 120m and a length of 60m and that this required a cemented rockfill strength of 7.0 MPa. To support the gravity loading alone, the fill required a compressive strength of 2.8 MPa for the above dimensions. A safety factor of 2.5 was applied to allow for such factors as additional blast loading and a reduction in fill strength should inadequate mixing occur (Yu, 1989). On the other hand, undercut and fill mining may require a backfill strength in excess of 5.0 MPa to prevent failure (Yu, 1990). The design of backfill systems requires careful engineering analysis which takes into account potential failure conditions to be encountered underground.

In all of the above cases, the support requirements are proportional to the geology and stability of the rock mass (Hassani and Bois, 1992). The support requirements are also a function of the stope size. There is a need to determine the binder requirements necessary to meet the target strength and deformation requirements. The basic questions are:

i) can the binder requirements be optimized by using a composite materials?
ii) what constitutes suitable mix design parameters for achieving desirable high-density backfill properties?
iii) on what basis should the strength be determined?
iv) what constitutes a suitable test sample size for determining the fill properties?

The consolidation of backfill with a binder has been identified by several investigators to be dependent on several factors of which the major ones include the following:

- Consolidation time - How long does it take the consolidated fill to meet the target strength and stiffness requirements?
- Moisture content - How does the fill strength change with:
  a) pulp density (for tailings fills)?
  b) dewatering for hydraulically placed fills?
  c) water/binder ratios for cemented rockfill and composite aggregate fills?
- Binder composition - What are some of the cost-effective binders to use and in what concentrations; also the effects of the curing environment on binder effectiveness?
- Physical properties - What are the physical characteristics of the fill material (i.e.: particle size and gradation, etc.) and how conducive are they to optimization?
- Condition of fill placement - determines whether the backfill is to be confined or exposed.

2.12 REQUIREMENTS FOR BACKFILL SYSTEMS DESIGN

Failure models have been developed (Mitchell et al., 1982), or adapted from other sources including geotechnical engineering for defining the strength requirements for various mining operational conditions. In mining, the support requirements of a fill of known composition can often be predicted using empirical strength models. The empirical design criteria have been developed based on results of laboratory tests and in-situ investigations. The dynamic tensile strength of a fill mass is an important consideration in the successful design and recovery of pillars. To fulfill this requirement, the fill must be designed to withstand blast vibration damage. Tensile strengths for backfill systems are difficult to determine directly. Indirect estimates by Mitchell and Wong (1982); Arioglu (1983); Yu, (1989) and others (Vickery and Boldt, 1989), have placed the tensile strength of cemented backfills to be approximately 5 to 15 percent of the unconfined compressive strength.

The tensile strength is an important parameter which expresses the ability of a fill mass to resist the tension damage in the form of flexural failure under vertical loading, undercutting and blast vibration.

Mitchell and Wong (1982) determined the tensile strength of a cemented tailings fill to be approximately equal to 12 percent of the unconfined compressive strength. On the other hand, Arioglu (1983) found that tensile strength accounts for up to 15 percent of cemented aggregate fill's unconfined compressive strength. Yu, (1989), determined consolidated rockfill tensile strength to be approximately 5 percent of the unconfined compressive strength.

2.12.1 Backfill Failure Mechanisms and Strength Requirements

Several modes of backfill failure have been identified in the available literature (Yu, 1990). These include the following:

- failure due to insufficient stiffness, leading to excessive deformation under load;
- failure due to saturated fill which is improperly drained and can liquefy under dynamic loading;
- failure by sloughing;
- failure due to inadequate bearing capacity under surface loading from mobile equipment, weight of ore piles and human activity; and
- failure due to insufficient shear strength resulting in slabbing or wedge failure of the exposed vertical fill faces.

These failure modes must be taken into account during the engineering design of backfill reliant stoping systems. There is also a need for fill systems that are best able to meet most of the above mentioned requirements. Cemented rockfill has also been identified as the backfill product which provides the best support in Quebec mines (Hassani and Bois, 1992).

2.12.2 Backfill Design Considerations for Open Stope Mining

The essential requirements for backfill design for open stope mining should include static and dynamic loading from blast vibration. Both static loading and dynamic loading from blast vibration can be responsible for fill stope failures during the recovery of ore pillars. Various analytical methods are used to estimate strength requirements during fill design. Because of difficulties associated with accurate determination of dynamic loading from a blast, the existing methods only consider the static loading of the fill. As a general rule and for safety considerations, a higher safety factor is used to compensate for the effects of dynamic loading induced by the blast vibration.

2.12.3 The Confined Block Failure Model

Among the failure models developed for calculating fill strength, the most applicable in fill design is the confined block with cohesion model. This model refers to an exposed fill where opposite sides of the fill are against stope walls. A wedge failure model as shown in Figure 2.10 is used to analyze the stability of free standing fill by assuming that there exists a shear resistance between the fill and stope walls. The net weight of the block is taken as being equal to the gravity loading, minus the shear component along both walls. For stability considerations, the shear resistance between the fill and the stope walls, and in the failure plane, are expected to exceed the driving force generated from gravity loading. The fill stability therefore is evaluated by using a safety factor, as follows:

$$F_s = \frac{\frac{LCW}{\cos a}}{M_c} \frac{+ M_c \cos a \ \tan\phi}{\cos a} \qquad 2.7$$

where:	L	=	Strike Length of an exposed fill (m)
	W	=	Width of a fill block (m)
	M_c	=	Fill block weight minus the wall shear component (kN)
	M_c	=	$H_e W (L\gamma - 2C)$

H_e = Effective height of the fill block (m)

H_e = $H - 0.5W \tan \alpha$

γ = Unit weight of the fill (kN/m^3)

C = Cohesion of the fill (kN/m^2)

$$\alpha = \left(45^o + \frac{\phi}{2}\right)$$

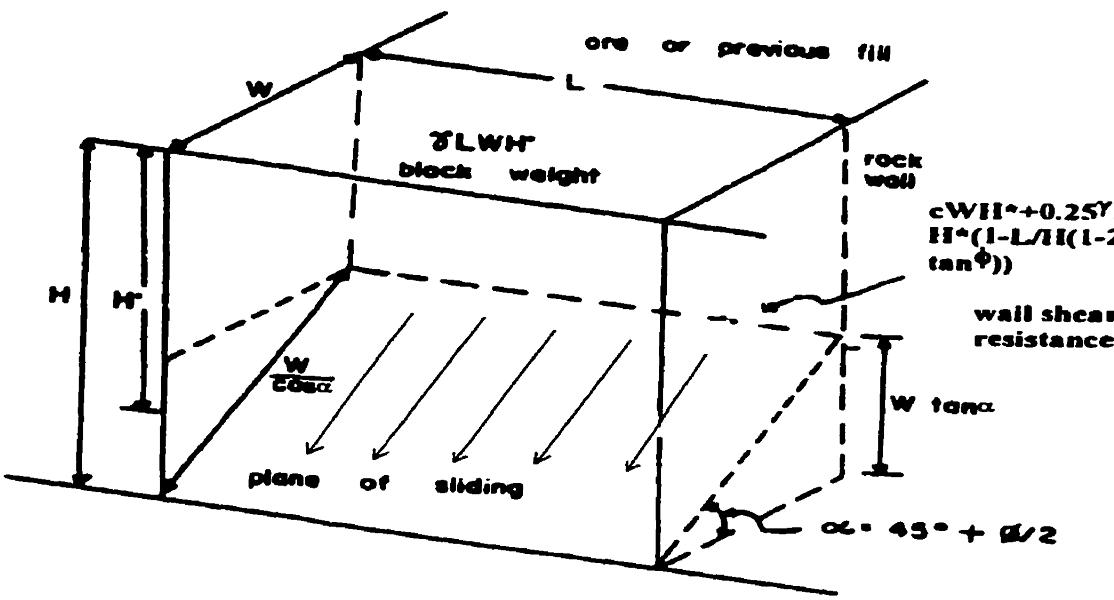


Figure 2.10 Failure Model for a Confined Fill Block (after Mitchell et al., 1982)

This model also has been described in detail by various investigators including Mitchell et al. (1982), Nantel and Lecuyer (1983), Smith et al. (1983) and Arioglu (1984). The model also has applications in cemented hydraulic fill design analysis.

2.12.4 <u>Fill Strength Requirements for Open Stope Mining</u>

Using the failure model described in Section 2.11.4.1 (Figure 2.10), the fill strength required for open stoping is defined by the following relationship:

$$\sigma_c \left\{ \gamma L \left[H - \frac{1}{2} W \tan\alpha \right] X \left(F_s \tan\alpha - \tan\phi \right) \right\} x \left\{ 2M \left[L \tan\alpha + \left(F_s \tan\alpha - \tan\phi \right) x \left(H - \frac{1}{2} W \tan\alpha \right) \sin 2\alpha \right] \right\} \qquad 2.8$$

where:	σ_c	=	compressive strength (in MPA) of fill after 28 days of curing
	ϕ	=	angle of internal friction (usually in the range of 35° - 45°)
	α	=	angle of failure in fill block α=(45° + ϕ/2)
	γ	=	unit weight of fill in tonnes/m^3
	M	=	constant relating to the reaction of cohesion to compressive strength
	M	=	0.18 for cemented rockfill
	M	=	0.35 for cemented tailings and sand fill
	W	=	width of fill block (m)
	L	=	length of exposed fill
	M	=	weight of exposed fill (kN/m^3) (M is a function of density of the fill mass)
	H	=	effective height of fill block (m)
	F_s	=	safety factor for fill stability F_s = 3 to 5

2.12.5 <u>Fill Strength Requirements in Overhand Cut and Fill Mining</u>

A number of models have been developed to analyze the failure mechanisms of fills under various geotechnical conditions. In overhand cut and fill mining, it is assumed that the backfill behaves as a working platform. The failure in the fill under the action of loading by equipment is due to the shearing of a wedge of the fill. It is assumed that the shear strength controls the ability of the fill to support the mining activities on its surface. This is expressed in terms of bearing capacity (Terzaghi and Peck, 1967).

The bearing capacity of a fill is given by:

$$Qc = 1.3c\,N_c + 0.4\gamma\;B\,N\gamma \qquad 2.9$$

In terms of design analysis, the fill is assumed to act as a shallow strip foundation which is in contact with the ground, which in this case is represented by the stope floor. The model used to describe the failure mechanism of the fill is based on Terzaghi's Theory on bearing capacity as indicated by the above equation;

where:	Qc	=	bearing capacity of the fill
	C	=	cohesion of the fill
	γ	=	unit weight of the fill
	B	=	width of bearing contact surface
	N_c	=	bearing capacity factor due to cohesion of the fill
	N_g	=	bearing capacity factor due to unit weight of the fill

The factors N_c and N_g are dependent only on the angle of internal friction (ϕ) of the fill. For mill tailings fills, the friction angle is a function of grain size gradation and the density of the material. Loose fills tend to posses low internal friction angles and therefore poor bearing capacity. High-density fills including composite backfills are expected to develop high bearing capacity values.

The stability of the fill is evaluated using a safety factor which is defined by:

$$F_s = \frac{Q_c}{Q_o} \qquad 2.10$$

where: Q_o = load of the mining equipment

A fill designed for overhand cut and fill must be capable of supporting a maximum load. In this regard, the strength is determined by modifying the analytical failure model as expressed in the above equation. The cohesion (C) is defined by knowing the friction angle of the potential fill.

$$C = \frac{Fs\,Qo - 0.4\gamma\ B\,N\gamma}{1.3Nc} \qquad 2.11$$

The unconfined compressive strength (σ_c) required, is determined by:

$$\sigma_c = \frac{\left(FsQo - 0.4\gamma\ BN\gamma\right)}{1.3Nc} \qquad 2.12$$

where:

	C	=	cohesion of the fill
	Q_o	=	maximum load per unit area
	Q_o	=	$\frac{Q}{nB^2}$
	Q	=	total load of equipment
	n	=	number of tires contacting fill surface
	B	=	width of contact area of a tire on the fill surface

2.12.6 Fill Strength Requirements in Underhand Cut and Fill Mining

In underhand cut and fill mining, the fill acts as a roof span and hence, the flexural failure model is considered as the most likely representation of the failure condition (Figure 2.9d).

A critical operating stage is assumed to occur just after the underlying ore lift is removed. At this stage, the fill is considered to be supporting a non-uniform vertical stress and lateral closure stresses, which build up as the rock deforms. The stress-strain state of the fill is analyzed by idealizing it as a two-dimensional plane strain stability problem. These external stresses are registered by bending and shear stresses in the fill.

Irregularities in the wall rocks due to blasting of the ore, provide interlocking with the fill and prevent fill slippage. With rough wall conditions, the roof fill bends as a uniformly loaded beam with fixed ends on the side walls and it is susceptible to flexural failure under the vertical loading, due to its low tensile strength.

The instability of the fill due to gravity loading can be initiated by tension in the centre part of the span, and failure occurs when the tensile stress exceeds the tensile strength. In underground cut and fill mining, the emphasis has been placed on improving the tensile strength of the backfill. As such, consolidated fills which are generally reinforced with steel screens and bars, have been used to support vertical loads.

The assumption for this condition is that the fill mass, cracks at a high tensile stress which is induced in the roof. For that reason, the fill should be defined on the basis of its tensile strength. In terms of flexural beam analysis, the tensile strength is determined from the following relationship:

$$\sigma_t \geq F_s\left(\frac{\gamma}{h}+\frac{B\gamma r}{2\tan\phi\ h^2}\right)\left(\frac{3}{4}L^2+\frac{1}{5}+h^2\right) \qquad 2.13$$

where:

where:	γ	=	unit weight of the fill
	γ_r	=	unit weight of the rock overlying the fill
	B	=	width of the stope

h	=	height of the cut
ϕ	=	friction angle of the rock
σ_t	=	tensile strength of the fill
F_s	=	factor of safety

To improve fill stability, various factors which relate to both the fill and the mining operations are generally considered. A higher safety factor is usually introduced, in the mathematical model, to compensate for the effects of these factors.

2.13 FILL STRENGTH PREDICTION MODELS

The correlations between the fill strength and the material compositions are used in backfill design. A number of strength models have been proposed by various investigators to help predict the strength and evaluate the potential fill behaviour. The principal prevailing factor in all of these models is the material properties of the fill. These depend on the ability to maximize strength and deformation (stiffness) by reducing porosity or void ratio.

2.13.1 Cemented Fill Strength Models

This model was first introduced by Mitchell and Wong (1982), and relates the unconfined compressive strength to porosity, water-cement ratio, and binder content. The unconfined compressive strength is expressed as:

$$\sigma_x = K_1\, C^a \left(n^{-b}\, W^{-n} + K^2\right) \qquad 2.14$$

where:			
	σ_x	=	unconfined compressive strength (kPa)
	n	=	porosity (%)
	W	=	water content (%)

C	=	binder content (%)
a	=	experimental constant (=1.1 - 1.7)
K_1, K_2	=	constants to be determined experimentally

The following model uses "binder number", a factor relating to the volume of cement in the fill, C_v, to estimate the unconfined compressive strength, σ_x, as reported by Swan (1985). Two parameters, "d_{avg}" which is the mean free distance between aggregate particles in the mix, and "a_p" the aggregate's specific surface area for a constant water-cement ratio, are used in the strength model. Knowing the fill aggregate particle size distribution, its porosity, and volumetric cement content, allows the calculation of a_p and d_{avg}. An empirical relationship is expressed as follows:

$$\sigma_x = 0.283\left(Cv / d_{avg} a_p\right)^{2.36} \qquad 2.15$$

where:

Cv	=	volume of cement in a unit dry volume of freshly placed backfill (%)
d_{avg}	=	mean free distance between aggregate particles in the mix
a_p	=	aggregate surface area for a constant water-cement ratio

The estimation of the above parameters is difficult; hence the Swan (1985) model could be impractical for conventional backfilling applications.

2.13.2 Cemented Hydraulic Fill Strength Models

This model relates fill strength to cement content in the fill, solids concentration of slurry and material size distribution, Chen and Jiao (1991). The fill strength is defined by:

$$\sigma_c = A\, e^{BC^*} + N\,(C_g - 0.65) \qquad 2.16$$

where:

C^*	=	ratio of cement to tailings or sand

C_g	=	solids concentration of fill slurry placed underground
N	=	constant 2.5 for consolidated fill
A, B	=	constant depending on solids grain size

for tailings fill, A = 0.235, B = 14.494

for sand fill (<2.5mm), B = 19.628

for sand fill (<1.2mm), A = 0.255, B = 16.459

Another model relates fill strengths to contents of cementing agents, water content and overall material composition in the fill. Lamos and Clark (1993). The fill strength in MPa is determined by:

$$UCS = \ell^{P1+\left(p2\left(\frac{OPC}{W}+P3\frac{PFA}{W}+P4\frac{PBFC}{W}\right)\right)\left(1+P5\frac{CT}{NCS}+\frac{CW}{NCS}\right)+P7\frac{NCS}{W}} \qquad 2.17$$

where:

P1 = 2.15	P2 = 5.65	P3 = 0.67	P4 = 1.60
P5 = 0.07	P6 = 0.34	P7 = 0.21	

OPC = Ordinary Portland Cement

PFA = Pulverized Fuel Ash

W = Water

CT = Classified Tailings

CW = Comminuted Waste

NCS = Non-Cement Solids (all material masses in same units)

PBFC= Portland Blast Furnace Cement

All material masses in grams

Limiting ranges:	water/cement	2-10
	total solids/water	2-5

2.13.3 Cemented Aggregate Fill and Rockfill Strength Models

Arioglu (1983) investigated the properties of cemented coarse aggregate and tailings fills and established the following relationships between uniaxial compressive strength and water/cement ratio:

$$\sigma_c = A\alpha^{-n} \qquad 2.18$$

where:	σ_c	=	uniaxial compressive strength
	α	=	water/cement ratio by weight
	A, n	=	experimental constants

Also, Arioglu (1983) established a relationship between cohesion strength and uniaxial compressive strength as follows:

$$C = A\sigma_c + B \qquad 2.19$$

where:	C	=	cohesion in kg/cm^2
	σ_c	=	uniaxial compressive strength in kg/cm^2
	A, B	=	experimental constants

Yu (1989) modified Arioglu's model and correlated consolidated rockfill strength with cement content in the fill. The following relationship was also established from laboratory testing of rockfill samples at Kidd Creek Mines:

$$\sigma_c = e^{0.25c} \; for\, 2 \leq c \leq 10 \qquad 2.20$$

where:	σ_c	=	unconfined compressive strength (MPa)
	c	=	cement content by weight % of minus 4 sieve aggregate

2.14 SUMMARY

1. A review of the literature suggests that the placement behaviour of mill tailings fills is influenced by the specific gravity, binder composition as well as particle size distribution of the fill material. It is important to identify the extent of these parameters on composite fill properties.

2. The consistency of paste fill has been noted from the literature review as an essential parameter for composite backfill design because, it controls the requirements for cement stabilization, fill transportation and placement (with regard to segregation). The consistency of tailings/sand paste backfill essentially depends on the moisture content of the fill product.

3. The reported studies on paste backfill have identified high moisture content (at near-saturation levels), as one of the parameters that could adversely affect paste fill stability in situ. Other identified adverse parameters affecting pastefill stability include, reduction in binder effectiveness due to chemical reactivity of the tailings, and the in situ breakdown of the cement bond under high confining pressures.

4. It is apparent from the reported studies that moisture plays a critical role in the stability of backfill materials. The potential effects of moisture on composite fill behaviour need to be investigated. These should include: (i) consistency in terms of slump moisture relationship; (ii) moisture/binder (w/c) ratio and (iii) an identification of a range of optimum moisture contents for the studied fill materials.

5. Moisture affects paste backfill stability by reducing the material's liquefaction resistance.

6. Cemented rockfill placement is affected by aggregate attrition and fills segregation. Aggregate attrition increases the fines content and thus influences both binder and water

demands for effective strength development. Materials segregation increases the void ratio and reduces the strength and deformation properties of the cemented rockfill mass.

7. The literature review has identified the existence of an information gap between the properties of paste backfill and cemented rockfill which could limit the flexibility of backfill systems design. The application of composite fill could reduce this gap; it could also introduce more flexibility in the mine backfill design by increasing the number of available choices of fill types.

8. The application of "aggregate" fills have proven successful in highly stressed areas in deep mines. The majority of the reported work involving "aggregate" fills has consisted of a combination of a maximum size of 20mm diameter coarse materials and classified tailings. This process is similar to procedures used in the concrete industry. The selection of the 20mm maximum aggregate size has been largely due to practical restrictions imposed by transporting the fill materials to the stope in a pipeline. A majority of the "aggregate" fills have also been placed as uncemented fill.

   There is therefore a need to investigate the properties of composite-aggregate paste (CAP) fills that comprise of larger (minus 200mm or, 8 in.) size aggregates and full plant tailings. The use of larger coarse aggregate sizes for CAP fill preparation would be a more practical extension of cemented rockfill preparation methods to composite fills. The use of larger size aggregates may also reduce aggregate crushing costs.

9. Laboratory test results may be used to infer the behaviour of backfill masses in situ. This is possible if different specimen sizes can be tested to develop empirical relationships. Scaling factors ranging between 60 to 90 percent have been proposed for relating laboratory scale test sample data to in situ conditions for cemented rockfill. A review of the backfill literature has also shown that increases in sample diameters and aggregate sizes have resulted in decreased compressive strength of backfill materials. There is a

the backfill literature has also shown that increases in sample diameters and aggregate sizes have resulted in decreased compressive strength of backfill materials. There is a need to investigate the possibility of scale effects on composite-aggregate pastefill (CAP) samples, especially when larger sizes of aggregate (of up to 200mm diameter) are used for fill preparation. The test data could be used to infer potential behaviour of a CAP fill mass in situ.

10. Various Aggregate/Tailings (A/T) combinations have been tried according to the literature survey. An optimal range of 60 to 70% coarse aggregates and 30 to 40% tailings combinations have been identified for the minus 20mm size aggregates and classified tailings. With regard to large size "aggregate" fills, a study comprising of 60% coarse/material and 40% tailings has been reported. This material ratio falls within the identified optimum mix range. Any future studies involving composite fills should therefore, begin with the examination of 70% coarse aggregates and 30% tailings compositions.

11. It has been identified from the literature survey that the porosity of cemented rockfill systems is controlled by the binder content and the water/cement ratio. There is also an established correlation between the compressive strength (σ_c) and the deformation modulus (E) of cemented rockfill properties. There is therefore a need to expand this correlation to other types of mine backfill systems, so that the established relationships can be applied as predictive tools for the design of mine backfill systems.

CHAPTER 3

3. PHYSICAL AND ENGINEERING PROPERTIES INVESTIGATIONS

3.1 INTRODUCTION

This chapter describes the physical and engineering properties of the fill material used in this study.

3.1.1 Objectives and Scope of the Physical and Engineering Properties Investigations

The objectives of the physical property investigations outlined in this chapter were to:

- define the physical and engineering property limits of the studied fill materials;
- evaluate the physical property and characteristics of the fill materials and to use the information as a fundamental requirement for understanding composite fill behaviour;
- develop scientific knowledge regarding the application of the studied fill materials for ground support in mines.

The scope of the investigations included the following:

i) Examination of the physical properties of tailings, alluvial sand and rockfill aggregates as suitable material for the preparation of high-density composite fills.

ii) Investigation of the chemical compositions of the test materials (tailings and sand)

iii) Determination of backfill consistencies for the tailings and sand pastefill in terms of the weight percentage of solids and also in terms of slump and moisture content relationships.

3.1.2 Identification of Fill Materials

The test material for this part of the study consisted of mill tailings, alluvial sands and rock fill aggregates. The mill tailings originated from various sources, including precious metal mines and base metal mining operations in North America. A total of seven (7) different types of tailings were investigated. These consisted of three (3) full plant precious metal ore tailings and three (3) classified and one (1) full plant base metal ore tailings. Three (3) alluvial sands consisting of fine, medium and coarse gradations (Lidkea and Landriault, 1993) were also studied. The sands were typical of the supplementary fill materials normally used for mine backfilling in Northern Ontario.

The type and sources of tailings materials are indicated in this study by the type of ore followed by an assigned number; for example: "Base Metal Tailings (BMT #1)". The alluvial sand sources are represented in a similar manner to the tailings. This method of sample identification was adopted in order to maintain the generic aspects of the study. The fill materials are also identified as Tailings # (1-3) or as Sand # (1-3). Both the type and source representations (eg. "Base Metal Tailings") and generalized identifications (eg. "Tailings #1") are also used through out the text for the tailings materials, depending on the emphasis. The two types of identification for the tailings materials are therefore interchangeable throughout the thesis.

The rockfill aggregates comprised of mine development waste from a precious metal mining operation in Ontario and consisted mainly of minus (6") 152mm size material, Figure 3.1

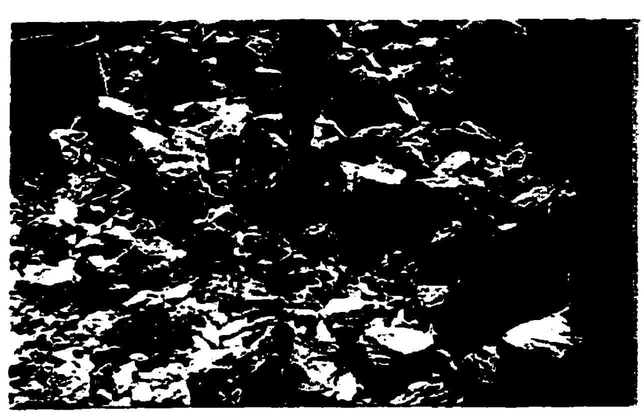


Figure 3.1 Typical Aggregates for Cemented Rockfill and Composite Backfill Studies

The primary materials consisting of mill tailings, alluvial sands, and rockfill aggregates were selected to broaden the scope of the study on composite fill system.

3.1.3 Moisture Content Determination for Rockfill Aggregates

Before performing the particle size analysis, the moisture content of the rockfill aggregates was determined in accordance with ASTM-(D2216) recommended procedures. The moisture content values which ranged between zero (0) and one (1) per cent by weight were required for cemented fill mix batching computations.

3.1.4 Chemical Analysis of Fill Material

The chemical analysis of the fill materials used are reported in Table 3.1

Table 3.1 Chemical Analysis of Test Materials.

Compound	P.M.+ Tailings #1	B.M *. Tailings #2	B.M. Tailings 1a & b	B.M. Tailings #3	P.M. Tailings #3	P.M. Tailings #2	Sand #1	Sands #2 & #3
SiO_4	10.59	52.28	33.56	72.69	27.63	57.56	84.38	77.49
TiO_2	0.09	0.92	0.23	0.16	0.21	0.29	0.88	0.31
Al_2O_3	1.93	12.79	5.97	3.98	3.86	10.20	6.43	8.90
Fe_2O_3	1.71	17.22	35.68	0.65	43.10	7.38	2.91	2.44
MnO	0.821	0.165	0.006	0.122	0.666	0.165	0.147	0.03
CaO	28.47	6.00	0.99	8.32	3.03	7.62	0.17	2.44
MgO	12.80	5.16	0.87	2.77	3.01	4.60	0.57	1.26
Na_2O	0.46	2.02	0.16	0.04	0.42	1.55	0.56	2.61
K_2O	0.35	1.26	1.25	1.29	0.21	0.55	1.19	1.77
P_2O_5	0.06	0.17	0.07	0.04	0.05	0.04	0.06	0.05
LOI	43.58	2.21	19.38	8.99	10.44	9.71	2.33	2.82
S (ppm)	0.31	2.30	28.50	0.23	16.80	0.72	<0.01	70

* B.M. = Base Metal
\+ P.M. = Precious Metal

3.1.5 Particle Size Analysis (Fine Material)

Sizing of the tailings samples was performed using a laser-based optical particle size analyzer (Leblanc and Annor, 1990). The sand samples were analyzed by both sieving (ASTM-C136) and laser-optical analyzer. The ultra fine material was taken to be the minus 20μm size material. Table 3.2 shows particle size analysis results for the tailings.

3.1.6 Particle Size Analysis (Coarse Material)

The sizes and distributions of the coarse and fine rock aggregates and coarse alluvial sand were determined by screening the material through a series of sieves. This was done in accordance with ASTM-(C136) specifications. By definition, the coarse aggregate was taken as the material retained on the 9.5mm (3/8") sieve and not larger than 152mm (6"). Similarly, the fine fraction of the rockfill aggregates was taken as that which passes through the 9.5mm (3/8") sieve.

Table 3.2 Typical Particle Size Distribution for Tailings Material

Particle Size µm	Base Metal Tailings #1a	Base Metal Tailings #1b	Base Metal Tailings #2	Base Metal Tailings #2
12	18.22	29.29	10.67	7.57
16	28.36	44.53	17.29	12.39
20	33.22	52.05	21.37	15.84
24	35.19	57.47	25.23	21.18
32	38.87	61.32	29.31	23.28
40	41.36	62.59	32.26	29.7
60	47.86	68.66	39.06	36.98
80	54.38	69.97	43.30	49.19
100	64.37	75.57	47.61	75.28
200	89.96	90.71	74.57	82.06
300	92.12	94.86	85.10	88.24
400	96.46	100	92.13	96.85
500	96.46	100	95.99	95.99
600	100	100	100	100
Particle Size (µm)		**Precious Metal Tailings #1**	**Precious Metal Tailings #2**	**Precious Metal Tailings #2**
3		3.77	2.78	6.00
4		6.86	5.34	10.56
5		10.39	8.46	15.62
6		16.87	15.02	23.55
7		22.75	22.22	37.84
8		28.51	30.06	42.90
9		33.48	35.42	47.55
10		37.85	39.48	73.54
20		66.38	63.10	84.32
30		79.13	75.69	87.61
40		86.13	78.71	90.14
50		87.30	83.86	91.29
60		89.09	85.38	95.17
70		90.39	86.81	97.53
80		93.50	90.38	100
90		96.01	92.65	100
100		96.01	96.26	100
150		100	100	100
200		100	100	100
300		100	100	100

3.1.7 Particle Size Analysis Results

The mean particle size distribution curves for the various materials are presented in Figure 3.2. The Coefficients of Curvature (Cc) and Uniformity (Cu) were determined as follows:

$$C_c = \frac{(D_{30})^2}{(D_{60})(D_{10})} \qquad 2.2$$

$$C_u = \frac{(D_{60})}{(D_{10})} \qquad 2.3$$

Where: D_{10} = grain size at 10% passing
D_{30} = grain size at 30% passing
D_{60} = grain size at 60% passing

C_c values for the rockfill aggregates ranged between 1.44 and 6.54. Similarly, C_u values varied between 9.15 and 59.7. Figure 3.2 also contains for reference purposes, the general gradation limits for silt (ASTM), medium and coarse tailings, and concrete aggregates. The particle size distribution procedures for the various fill materials are presented in Appendix A-1.

The average particle size distribution curves for the high-density fill materials in Figure 3.2 cover a wide range of particle sizes. The range and size gradation curves for soils, tailings, sands, concrete and cemented rockfill aggregates (Hedley, 1995), have also been presented in Figure 3.3 for purposes of comparison. The range of size gradation parameters for the test materials have also been presented elsewhere in this Chapter (in Table 3.4) in terms of the following: (i) minimum and maximum size ranges; (ii) Coefficient of Curvature (Cc); (iii) Coefficient of Uniformity (Cu); and (iv) Ultra-fine particle size content (%-20μm).

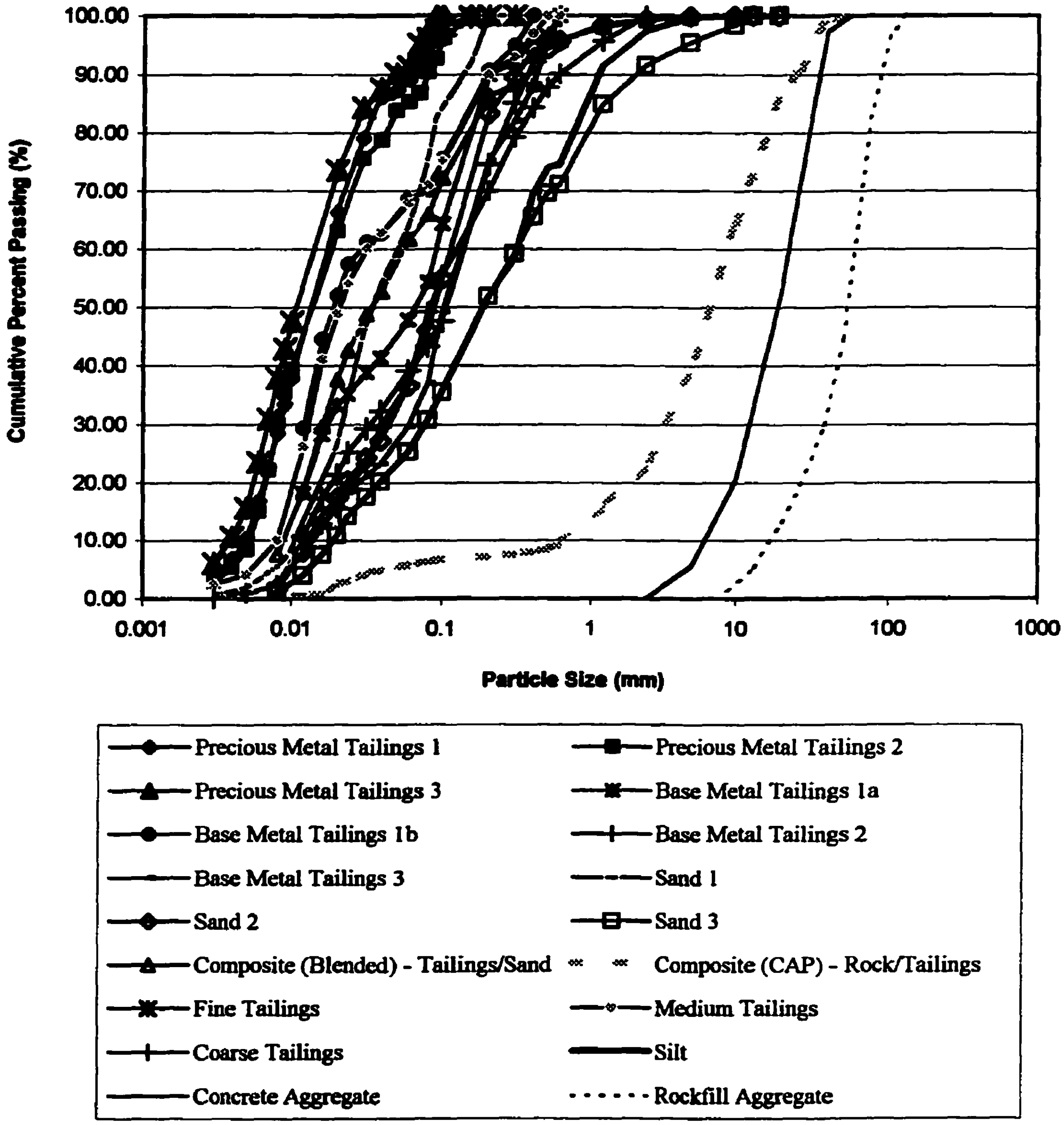


Figure 3.2 Particle size distribution curves for the studied fill materials

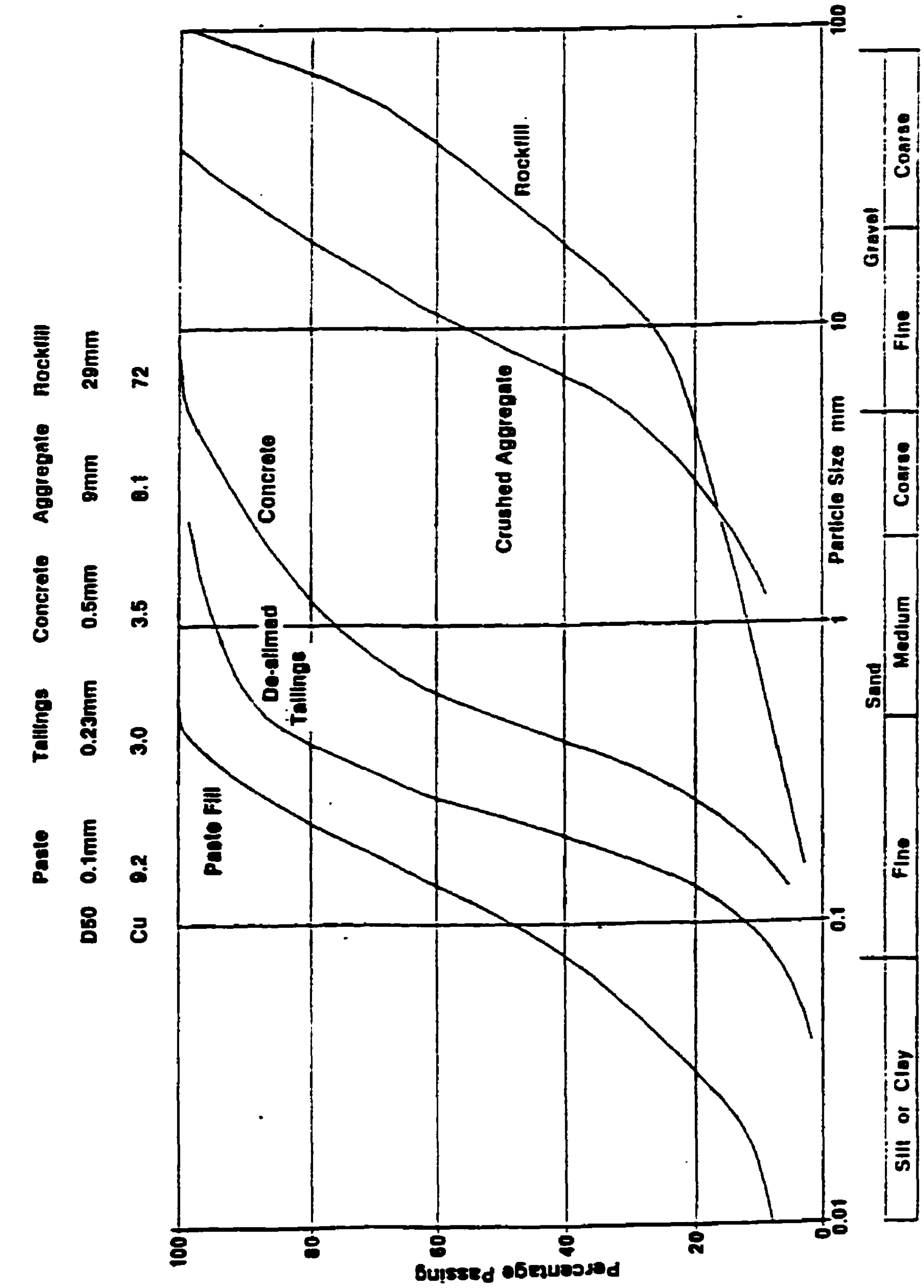


Figure 3.3 Typical size distribution curves for different types of backfill (after Hedley, 1995)

The size gradation curves for the fill material compare favourably to the general shapes of curves shown in Figure 3.4 and are within the range of material gradations found in Canadian Mine sites (Hedley, 1995). For example, Pierce et al., (1998) reported that the Golden Giant Mine tailings had a Cu value equal to 13 also, 27 per cent of the material was below 20μm size. Similarly, Ouellet et al., (1998) found a Cu of 3.1 to be the typical value for the paste backfill materials they studied. Also, they reported that the typical amount of material below the 75μm size ranged between 65 to 80 per cent. The reported values are in general agreement with the tailings materials in this study (Table 3.2).

3.1.8 Relative Densities of Rockfill Aggregates

The maximum and minimum densities of the representative rockfill aggregates were determined using a vibrating table in accordance with ASTM (D4253 & 4254) recommended procedures. The void ratios of the rockfill aggregates in the loosest and densest packing states were also determined. This was done in order to define the minimum achievable void ratio for the rockfill aggregates before mixing with tailings and stabilization with a binder. This information was used as a reference material for comparing changes in the void ratios of the stabilized rockfill and the composite backfill samples.

The minimum and maximum void ratio values for the rockfill aggregates are summarized in Table 3.3

Table 3.3 Minimum and Maximum Void Ratio Values for Rockfill Aggregates

Test #	Moisture Content (%)	Minimum Void Ratio	Maximum Void Ratio
1	0	0.37	0.51
2	0	0.38	0.67
3	0	0.39	0.68
4	0	0.35	0.54
Average		0.37	0.60

3.1.9 Specific Gravity Determinations

The specific gravities of the test materials were carried out in accordance with ASTM (D854) specifications. The specific gravity values for the tailings and sand materials are presented in Table 3.4. The specific gravity values for the rockfill aggregates ranged between 2.54 and 2.90

Table 3.4 Specific Gravity Test Results for the Studied Mill Tailings and Alluvial Sand Samples

	Precious Metal Tailings #1&2	Precious Metal Tailings #3	Base Metal Tailings #1a	Base Metal Tailings #1b	Base Metal Tailings #2	Base Metal Tailings #3	Base Metal Tailings #4	Sand #1	Sands #2&3
Specific Gravity	2.74/ 2.75	3.43	3.93	3.90	2.92	2.64	2.94	2.55	2.67

3.1.10 Summary of Physical Property Limits of the Studied Materials

The observed range of physical property parameters for the studied materials are presented in Table 3.5. The values compare favourably to values in the published literature Hedley (1995); Arioglu (1983); including Aref et al., (1989) Pierce et al., (1998); Farsangi (1996); Ouellet et al., (1998).

Table 3.5 Summary of Physical Properties of Studied Fill Materials

Source Name	S.G.	Size Range		C_u	C_u	% -20µm
Precious Metal Tailings 1	2.75	2 µm	200 µm	0.26	5.3	42.3
Precious Metal Tailings 2	2.74	2 µm	100 µm	0.63	7.5	40.5
Precious Metal Tailings 3	3.43	2 µm	80 µm	0.92	3.8	74.8
Base Metal Tailings 1a	3.93	8 µm	500 µm	1.80	10.9	15.5
Base Metal Tailings 1b	3.90	8 µm	300 µm	0.32	5.9	43.5
Base Metal Tailings 2	2.92	8 µm	500 µm	1.05	10.9	25.2
Base Metal Tailings 3	2.64	8 µm	500 µm	1.72	7.4	18.3
Sand 1	2.60	2 µm	150 µm	1.15	4.2	26.5
Sand 2	2.67	8 µm	4.75 mm	1.45	6.5	13.9
Sand 3	2.67	8 µm	9.50 mm	1.51	11.0	11.9

3.2 ENGINEERING PROPERTIES TESTS

Consistency refers to the degree of firmness or the extent of wetness of a paste fill mixture (ASTM C-143). The main objective of this element of the study was to examine paste backfill consistency from two view-points. These were: the percentage solids composition by weight, and slump and moisture content determinations. The consistency test results contributed to the information base for establishing an optimum range of pulp density and slump values for composite fill mix batching.

3.2.1 <u>Determination of Mill Tailings/Sand Paste Fill Consistency (Pulp Density)</u>

Paste fill has been described as having the consistency of a tooth paste (Hassani and Aref, 1988; Aref et al., 1989). Paste consistency which refers to the thickness or stiffness of a paste mixture cannot be readily measured. Instead, "Pulp Density" which is a reflection of the percentage (%) of solids present in the mixture, can be determined as a function of moisture content based on sedimentation-consolidation tests (Clark, 1988).

Generally, hydraulically placed mill tailings fills can exist at different consistencies depending on the ratio of water to solids (w/s) which is present in the fill. The consistencies are often identified in terms of pulp density or the settled density. Backfill pulp density can also be considered (Clark, 1988) in terms of water-to-solid (w/s) ratio. Slump tests (ASTM C-143; Verkerk and Marcus, 1988) have also been used to determine the pulp density of tailings paste fill. Both methods of determining paste pulp density were investigated as part of this study to assess:

(a) to evaluate the effects of the physical property limits of the fill materials in terms of pulp density and slump determinations.

(b) to establish a range of backfill pulp density values best suited for preparing composite backfill materials.

3.2.2 <u>Sedimentation-Consolidation Tests</u>

The Sedimentation-Consolidation Tests were performed:

- to establish the pulp density range for the tailings and sand pastes,
- to evaluate the effects of specific gravity and particle size gradation on settling rates of the test materials,
- to determine the volume change of the settled particles due to expulsion of water under consolidation.

The sedimentation-consolidation tests were carried out on the tailings and the alluvial sand materials, to establish the settling pulp densities of the particles and the corresponding backfill consistencies. The test method followed was similar to that described by Clark (1988) and Chen and Annor (1995). About 800g dry weight of material were prepared as a slurry at a slurry-density of approximately 30 per cent solids by weight (Millette et al., 1998). This value corresponds to a typical feed pulp density of tailings slurried in most backfill plants. The sedimentation rates of the slurried were studied using a two liter cylinder. The settling rate was measured in terms of the solids interface and the settling time. The observations were made over a 24 hour period and readings were taken at specific time intervals.

With regard to the consolidation tests, about 250g dry weight of material were prepared to a settling pulp density between 70% and 75% by weight of solids depending on the sedimentation test results. The test specimens were tested in a 10mm diameter consolidation cell under a pressure head of approximately 140 kPa (20psi). The consolidation test method was similar to that used in soil mechanics (ASTM D-2435).

Typical sedimentation test results are summarized in Table 3.6. Typical settling-consolidation test graphs are presented in Appendix A-2.

Table 3.6 Materials Properties and Backfill Consistencies Based on Sedimentation-Consolidation Tests **Note HF - Hydraulic Fill, HDSF = High Density Slurry Fill, PF = Pastefill

Source Name	S.G.	Size Range (μm)		Cc	Cu	% -20 μm	Backfill Consistencies	
		Min	Max					
Precious Metal Tailings 1	2.75	2	200	0.26	5.3	42.3	<65% 65% - 72% 72% - 78%	HF HDSF PF
Precious Metal Tailings 2	2.74	2	1000	0.63	7.5	40.5	<65% 65% - 70% 70% - 80%	HF HDSF PF
Precious Metal Tailings 3	3.43	2	80	0.92	3.8	74.8	<65% 65% - 73% 73% - 78%	HF HDSF PF
Base Metal Tailings 1a	3.93	8	500	1.8	10.9	15.5	<70% 70% - 78% 78% - 86%	HF HDSF PF
Base Metal Tailings 1b	3.90	8	300	0.32	5.9	43.5	<70% 70% - 76% 76% - 84%	HF HDSF PF
Base Metal Tailings 2	2.92	8	500	1.05	10.9	25.2	<65% 65% - 70% 68% - 78%	HF HDSF PF
Base Metal Tailings 3	2.64	8	500	1.72	7.4	18.3	<60% 60% - 68% 68% - 80%	HF HDSF PF
Sand 1	2.60	2	150	1.15	4.2	26.5	<65% 65% - 68% 68% - 78%	HF HDSF PF
Sand 2	2.67	8	4.75 mm	1.45	6.5	13.8	<65% 65% - 73% 73% - 84%	HF HDSF PF
Sand 3	2.67	8	9.5 mm	1.51	11	11.9	<65% 65% - 75% 75% - 86%	HF HDSF PF

3.2.3 Discussions of Paste Consistency Determination by Sedimentation-Consolidation Tests

The three common backfill consistencies for hydraulic fills (Chen and Annor, 1995) are: (i) **"Conventional" hydraulic fill**; (ii) **High Density Slurry Fill**; and (iii) **Paste fill.** These consistencies are generally expressed in terms of Pulp Density or Solids content by weight percentage. Pulp Density refers to the weight proportions of solids and water in a given mass of fill expressed in terms of a percentage (Clark, 1988; Brackebusch, 1994). By definition (Millette et al., 1998), the pulp density of a conventional hydraulic fill is lower than the settled density of the tailings material; it usually ranges between 65 and 70% solids by weight. Similarly, high density slurry backfill has pulp densities that are just below the settled density and it generally ranges between 71 and 76% solids by weight. Paste fill on the other hand, has a pulp density that is higher than the settled density of the material. The solids composition of tailings paste backfill generally ranges between 76 and 84% by weight (Lidkea and Landriault, 1993).

The identified values of paste consistency limits are expressed as block diagrams (Figures 3.4 to 3.7) against the following independent variables: (i) Specific Gravity (S.G.), (ii) Ultra-fine particle size content (% -20 μm); (iii) Coefficient of Curvature (Cc); and (iv) Coefficient of Uniformity (Cu). This was done as a method of identifying any general trends in paste formation as a function of the physical properties of the test materials. For the analysis, Cu values of 4 to 6, and Cc values of 1 to 3 were considered (Terzaghi and Peck, 1967; Peck et al., 1974; Das, 1983; Craig, 1978) as indications of a well graded fill material. Additionally, the fine material composition was taken in this study to be tailings or sand containing more than 35% ultra-fine size particles by weight.

An optimum composite mixture is identified in this study as on that is flowable, resilient without segregation and able to minimize porosity of the stabilized fill product.

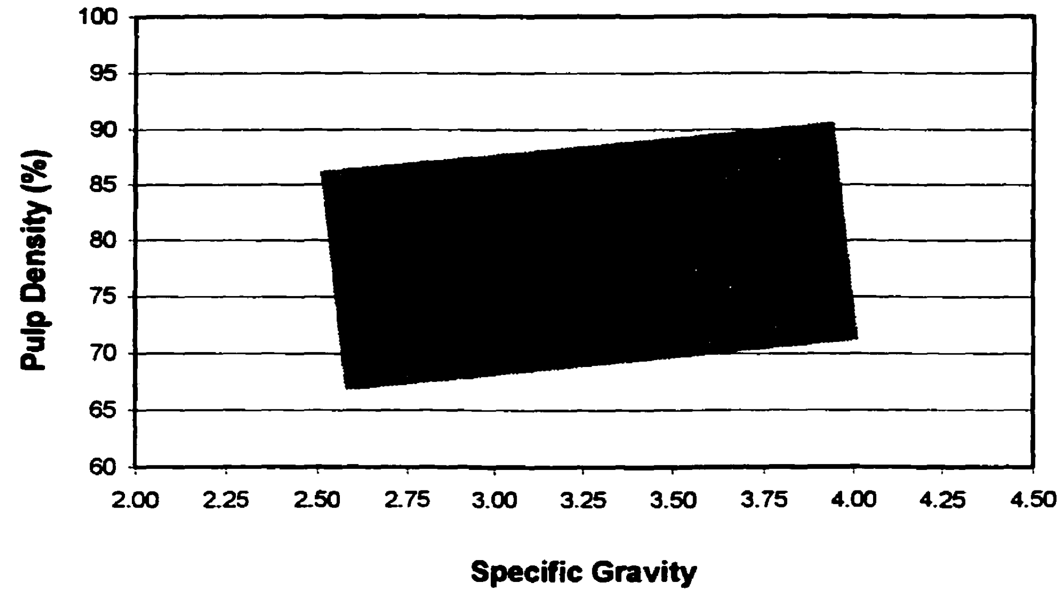


Figure 3.4 Variation of paste pulp density with specific gravity values for fill materials

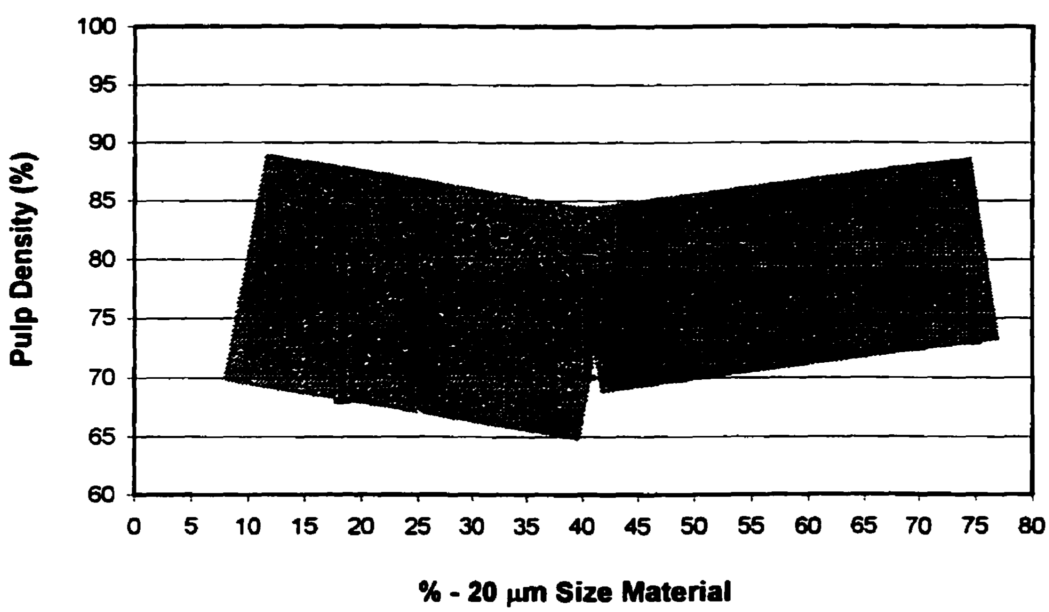


Figure 3.5 Variation of paste pulp density with ultra-fines content (% -20μm) for fill materials

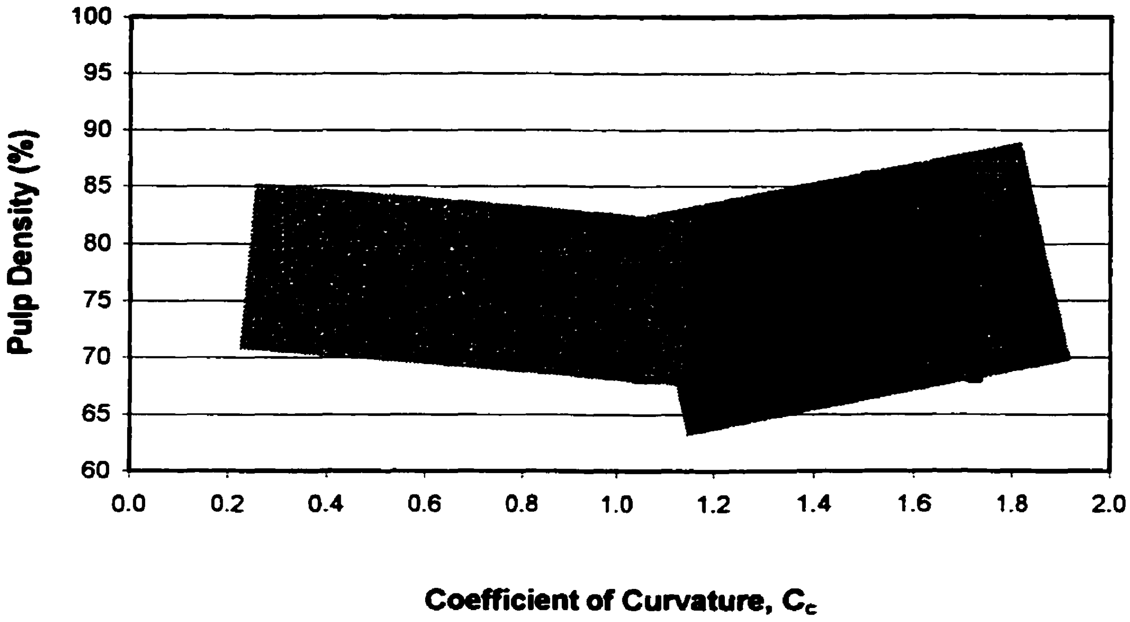


Figure 3.6 Variation of paste pulp density with coefficient of curvature (C_c) values for fill materials

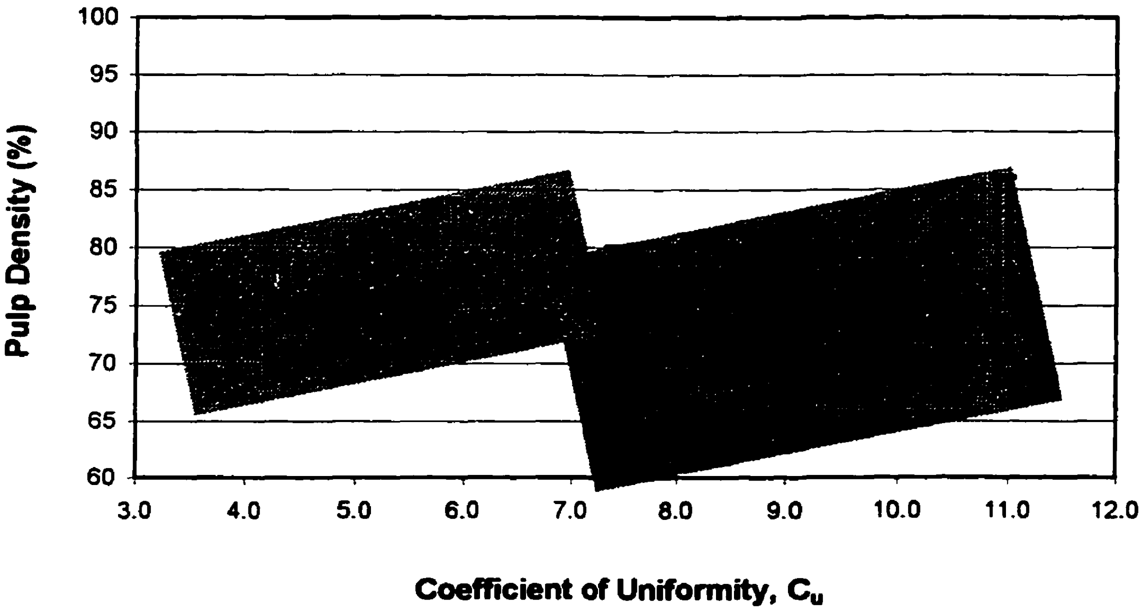


Figure 3.7 Variation of paste pulp density with coefficient of uniformity (C_u) values for fill materials

3.2.4 Slump-Moisture Content Relationships

The standard slump test (ASTM C-143) is generally used in Concrete Technology to indicate the workability of concrete. Slump can also be used to measure consistency of concrete mixtures (Neville, 1987) and is known to change with variables including moisture, air, binder composition and fines content, as well as, the presence of concrete admixtures. In terms of Mine Backfill Technology, the weight per cent of solids is correlated to the moisture content to establish an optimum mix design range for stabilization, transportation and placement (Brackebusch, 1994; Brackebusch and Shillabeer, 1998). The following pulp density limits have been identified for total tailings paste backfill (Landriault, 1995; Landriault and Tenbergen, 1995) based on a 178mm (7-inch) slump:

Tailings Type	wt. solids content
Coarse	79 wt.%
Medium	75 wt.%
Fine	70 wt.%

These values have gained universal acceptance, and are often quoted by consultants when specifying pulp density for paste backfill work although there have been no other published studies to validate them.

In terms of high-density composite fill preparation, the pulp density can be correlated to the moisture content to establish an optimum mix design range for stabilization, transformation and placement.

3.2.4.1 Study Objectives

The objective of this element of the study was to determine the relationships between slump and moisture content for tailings and sand pastefill mixtures. Also, to determine how they relate to paste pulp density for the fill materials in this study when compared with the results of the settling-consolidation measurements (Section 3.2.1). Paste fill can be produced at very solid concentrations so that the material does not bleed water (Brackebusch, 1994; Millette et al., 1995; Brackebusch and Shillabeer, 1998), however effective mixing (Petrolito et al., 1998) and transportation (Brackebusch and Shillabeer, 1998) requirements are often the defining criteria for selecting an optimum pulp density range for the engineering design of backfill systems.

The ultimate goal of this part of the investigation was to determine some favourable mix design limits for the composite backfill materials. In this regard, the ability of the tailings to flow through, penetrate and combine with the coarse aggregates (Yu, 1990) without segregation is a very important requirement for composite fill design. This would ensure effective void reduction, low porosity and improved mechanical properties of the fill product.

3.2.5 Slump Moisture Content Determinations

The slump tests were conducted on both cemented and uncemented material to determine the pulp density range of mixtures for composite fill preparation and stabilization. ASTM (C-143) standard test method was used. Three hundred and ten (310) tests were carried out on the tailings and alluvial sand materials. Figure 3.8 shows a slump test in progress.

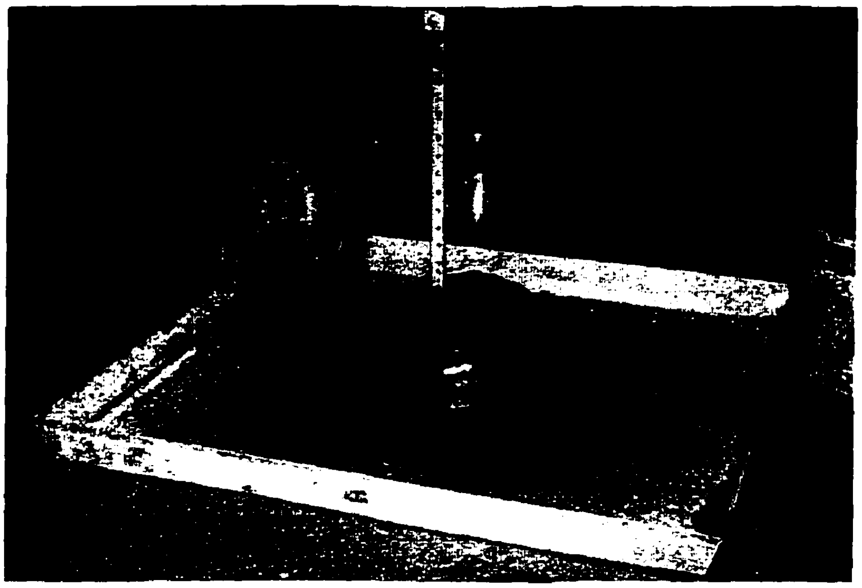


Figure 3.8 Slump Test

The results of the slump test for the tailings and alluvial sand materials are presented in Appendix A-3. Typical tests results are summarized in Table 3.7

Table 3.7 Typical Slump Moisture Content Tests Results for Tailings Paste Fill

Binder Content	Binder Composition (OPC)*	Slump (inches)	Slump (mm)	Moisture Content (%)	Pulp Density
0	**0%**	**3.25**	**82.6**	**17.4**	**82.6**
0	**0%**	**5.50**	**139.7**	**18.8**	**81.2**
0	**0%**	**8.25**	**209.6**	**19.5**	**80.5**
0	**0%**	**10.50**	**266.7**	**21.6**	**78.4**
5	**5%**	**3.25**	**82.6**	**15.8**	**84.2**
5	**5%**	**5.25**	**133.4**	**16.2**	**83.8**
5	**5%**	**8.50**	**215.9**	**17.3**	**82.7**
5	**5%**	**10.50**	**266.7**	**21.9**	**78.1**

*OPC = Ordinary Portland Cement

3.2.6 Discussions-Slump vs Moisture Content Test Results

The slump-moisture content curves for the uncemented tailings and sand paste fills are presented in Figure 3.9. The 178mm (7-inch) to 228mm (9 inch) slump range have also been highlighted on the graphs so that the corresponding consistencies can be readily identified. The shaded zone represents the preferred range for composite tailings/sand paste mix design for stabilization based on the results of this study. Visual observations during testing indicated that paste mixtures remained cohesive, but workable within this zone.

The range of maximum and minimum pulp densities for the paste fill materials, based on the slump tests were also identified and were compared with values obtained from the sedimentation-consolidation tests. For purposes of comparison with the sedimentation-consolidation test results, the maximum and minimum pulp density levels were set respectively at 102mm (4-in) and 254mm (10-in) slump. The range of paste fill pulp density values from both the sedimentation-consolidation and slump tests are summarized in Table 3.8. The values shown in parenthesis in Table 3.8 represent the maximum and minimum ranges of pulp densities from the sedimentation-consolidation tests.

The results indicate a gradual increase in slump values with moisture content for both the uncemented and cemented paste fills (Appendix A-3). The rate of increase seems to vary with the type of material (tailings or sand), the ultra-fines content and the specific gravity of the fill material. The coarser base metal tailings showed a more rapid variation in slump with increasing moisture content compared to the finely ground precious metal tailings. This suggests that the coarser materials have a tendency to bleed water and segregate at high slump values and therefore, could represent the most difficult material to mix and place without segregation as a composite backfill system. The least variation in slump values were observed for the precious metal tailings PMT#3 and the medium and coarse grain sand #2 and sand #3 (Figure 3.9). These materials indicated a high tendency to suddenly segregate into solid and liquid components beyond the 227mm slump during the slump tests.

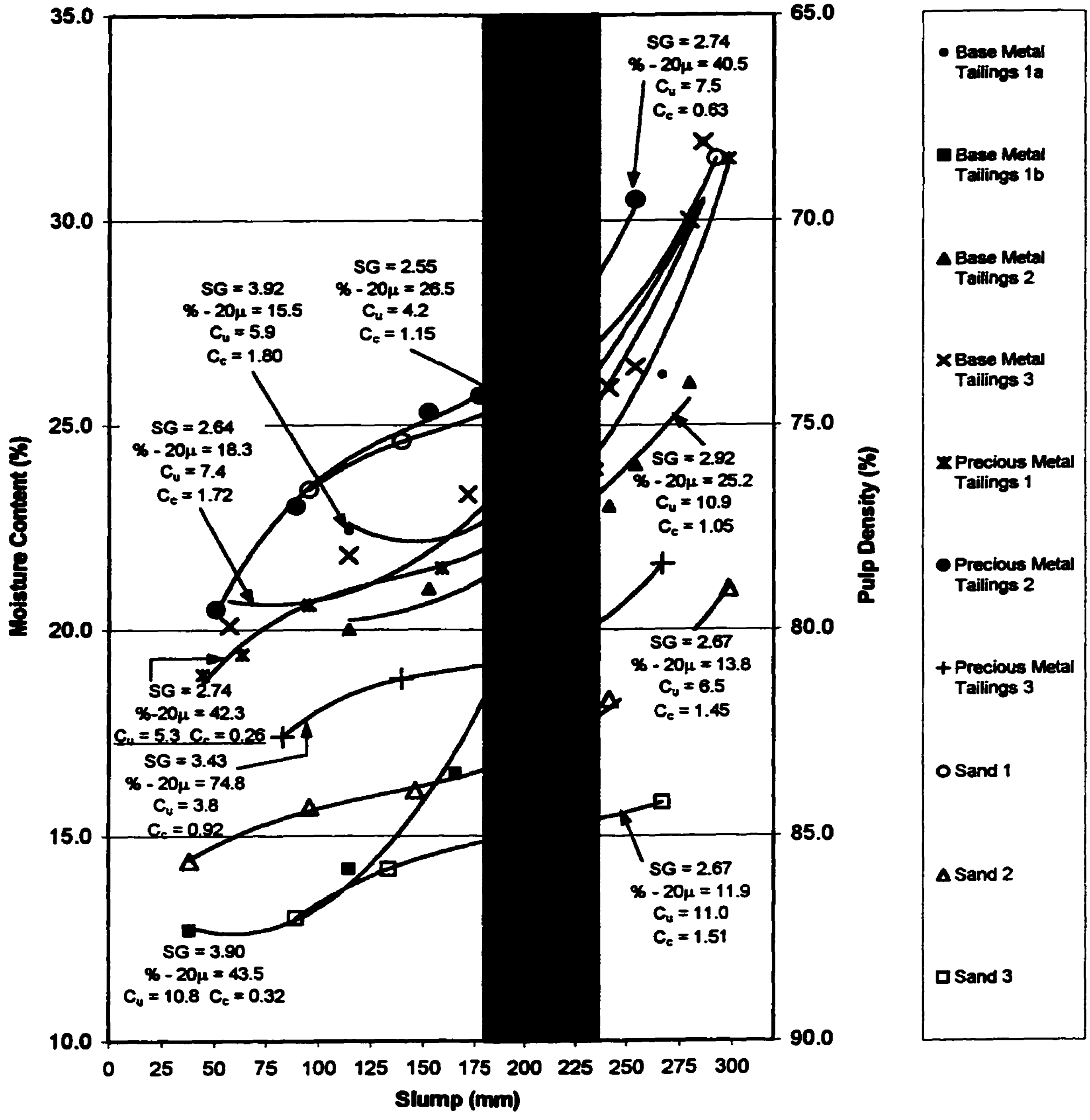


Figure 3.9 Slump - moisture content relationships for uncemented tailings and sand paste fills

Three of the base metal tailings (Base Metal Tailings #1a, 2 and 3) and Precious Metal Tailings #1 produced similar pulp densities at 178mm (7-inch) slump. The estimated paste pulp density for Base Metal #1a (classified tailings) was approximately, 78 wt% solids. The ultra-fine material contents for the three Base Metal Tailings (#1, 2 and 3) were 25.2% and 18.3% respectively. On the other hand, the ultra-fines composition for the precious metal tailings was 42.3%. The differences suggest that size gradation of the tailings alone could not have accounted for the observed similarities in pulp density values. Perhaps the similarities were due to other factors.

There were similarities in the consistency curves within all the tailings material investigated in this study (Figure 3.9). For example, there were close similarities between the pulp densities of Base Metal Tailings #2, and #3, and Precious Metal Tailings #1 over a range of slump values. This occurred between 102mm and 178mm (4-inch to 7-inch) slump values. With reference to 178mm and 227mm slumps, pulp densities of the investigated fill materials ranged between approximately 85%wt. and 72%wt solids for the uncemented products, and 88%wt. to 73%wt. for the cemented products. There seems to be no direct correlation however, between pulp density and particle size gradation for any of the tailings materials in this study. There were only very slight variations in slump values between the cemented and uncemented mixtures for the investigated materials in this study. This was true for each of the fill materials investigated in this study (Appendix A-3).

The alluvial sands however, had similar specific gravity values (2.60 to 2.67) and therefore displayed a specific trend between pulp density and particle size gradation as shown in Figure 3.10. The coarser the sand, the higher the pulp density values that corresponded to paste formation. This trend indicates that specific gravity is an important determining factor in establishing slump-moisture content relationship for paste fill materials.

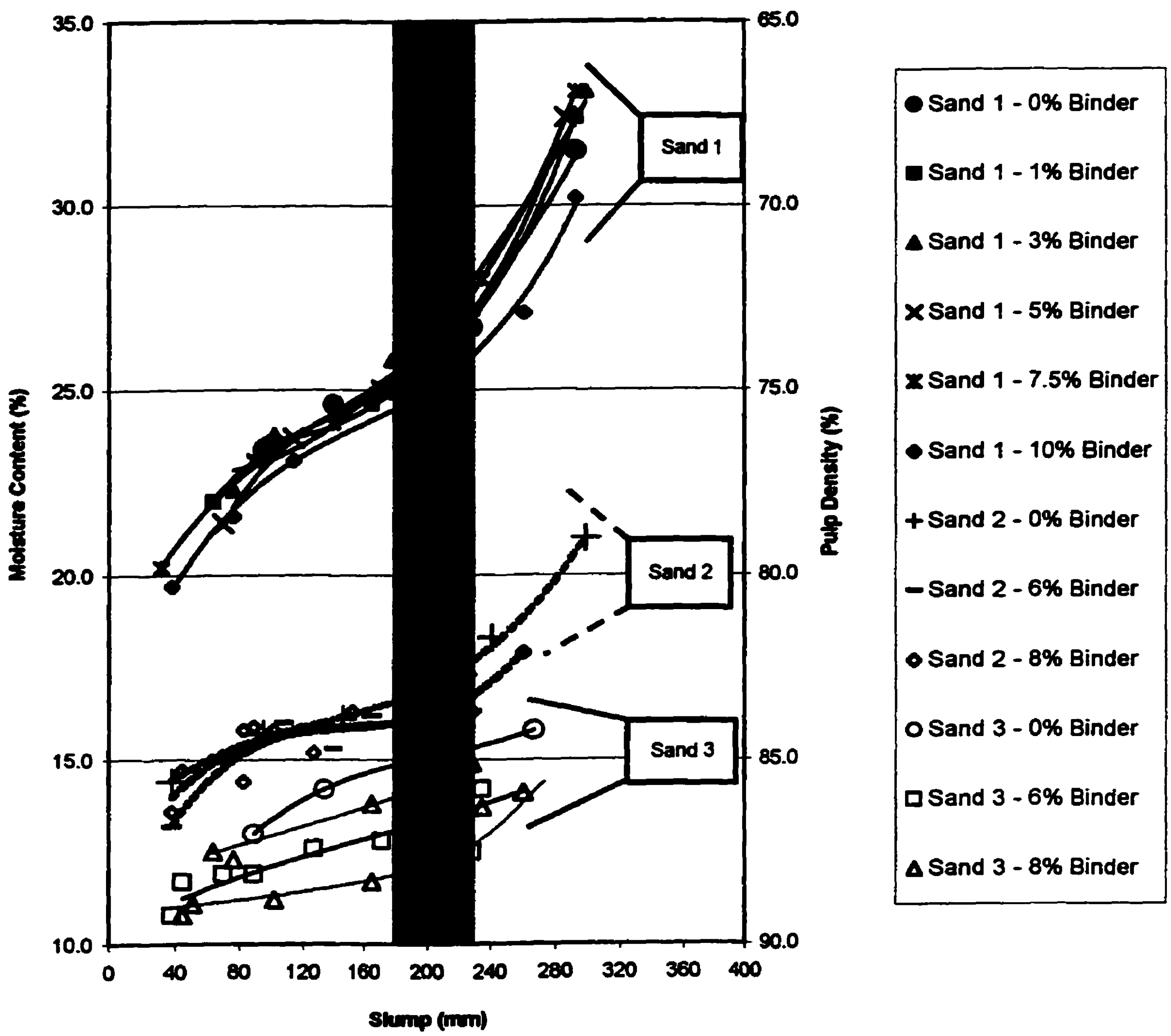


Figure 3.10 Slump - moisture content relationships for cemented sand paste fill (0 - 10% binder)

Table 3.8 Slump vs Pulp Density Relationship for Uncemented Tailings and Sand

Slump		Pulp Density Values (%)									
(Inches	(mm)	Base Metal Tailings 1a	Base Metal Tailings 1b	Base Metal Tailings 2	Base Metal Tailings 3	Precious Metal Tailings 1	Precious Metal Tailings 2	Precious Metal Tailings 3	Sand 1	Sand 2	Sand 3
4	102	(86)	86.6 (84)	79.5 (82)	79.5 (80)	79.4 (80)	76.3 (80)	81.9 (78)	76.4 (78)	84.4 (84)	86.6 (86)
5	127	78.4	85.5	79.2	79.3	79.0	75.4	81.3	75.6	84.2	86.0
6	152	78.3	83.8	78.7	78.8	78.7	74.5	80.8	75.0	84.1	85.7
7	178	78.2	81.6	77.9	78.0	78.3	73.4	80.3	74.4	83.9	85.4
8	203	77.5	78.9	76.8	77.0	77.5	71.8	79.6	73.6	83.6	85.3
9	229	76.3	75.7	75.5	75.7	76.2	69.3	78.5	72.4	83.2	85.3
10	254	74.5 (76)	72.0 (70)	73.9 (70)	74.1 (72)	74.1 (68)	65.8 (70)	76.8 (73)	70.5 (68)	82.5 (73)	85.2 (75)

3.2.7 Summary and Conclusions (Paste Consistency Measurements)

The test results indicate the following:

a) The onset of paste formation for the various materials in this study increased directly as their specific gravity values (Figure 3.4). Paste was formed over a wide range of pulp density values for the lower specific gravity materials, than those with higher specific gravity values.

b) The onset of paste formation decreased with increasing content of uniformly graded ultra-fine material (between 10 and 35%) (Figure 3.5). The pulp density range for paste formation was lower for materials with ultra-fine compositions of greater than 35%.

c) The onset of paste formation decreased for the uniformly graded test materials (those with coefficient of curvature (C_c) values of less than 1.0), and increased as the material became coarser and well graded for (C_c) values of greater than 1.0 (Figure 3.6).

d) Pulp density increased directly as the Coefficient of Uniformity (Cu) values of the test materials. This was true for both the fine and coarse materials (Figure 3.7). Cu values of greater than 4 indicate a well graded soil (Terzaghi and Peck, 1967; Peck et al., 1974; Das, 1983; Craig, 1970).

Almost all of the investigated tailings and sand materials can be considered as well graded, if the Cu value alone is used as the governing criterion for size gradation. However, the results of this investigation suggests that both Cc and Cu values must be taken into consideration in defining the size gradation of the fill materials. The results also suggest that the range of pulp densities over which paste was formed, and sustained for the test materials was influenced by both specific gravity and size gradation. This is in agreement with the findings of Brackebusch (1994), and Brackebusch and Shillabeer (1998). Particle size gradation alone was found not to be the only determinant for paste formation based on the test materials in this study.

The test results suggest that care must be exercised in applying particle size gradation data alone, as a criterion for establishing consistency limits for paste fill materials. The slump-moisture content results in this study was found to be "material specific". Particle size gradation of the tailings did not correlate directly to pulp density values, except in the case of the alluvial sand materials that had very similar specific gravity values.

3.3 SUMMARY

The physical and engineering properties tests on fill materials used for the study were carried out in this Chapter. The fill materials consisted of seven (7) types of base metal and precious metal tailings form six (6) different mine sites, three (3) different sources of alluvial sands, and also rockfill aggregates. The physical property tests consisted of specific gravity, particle size analysis, and moisture content determinations. Chemical analysis of the tailings and sand materials were also carried out.

The engineering property tests consisted of the establishment of maximum and minimum void ratios for the rockfill aggregates, consistency levels of the tailings and alluvial sand materials for paste formation, and the changes in consistency levels with the physical properties of fill materials, (i.e. particle size gradation and specific gravity) were also determined. The results of the physical and engineering properties determination indicate the following:

1. The specific gravity values ranged between 2.60 and 3.93 for the tailings and sands. The rockfill varied between 2.54 and 2.90 in terms of specific gravity.

2. The particle size gradation of the fill materials in this study covered a wide range of sizes ranging form 2μm to 9.5mm for the alluvial sands. Similarly, the rockfill aggregates ranged between 0.02mm to 152mm. The specific gravity and void ratio values are in agreement with results in the published literature.

3. The maximum void ratio which represent the loosest packing state for the rockfill aggregates, ranged between 0.51 and 0.68 with an average value of 0.60. Similarly the minimum void ratio ranged between 0.35 and 0.39 with an average value of 0.37. The efficiency of void reduction by cement stabilization or through the addition of tailings

tailings to the rockfill aggregates (to create a composite CAP fill) will be determined with reference to the average void ratio value of 0.37.

4. Based on the materials in this study, the consistency of pastefill may be determined using either a sedimentation-consolidation approach or else, by the slump moisture content method.

5. There was no direct correlation between particle size gradation and pulp density for any of the materials in this study. Both specific gravity and particle size gradation were found to be determinants of paste formation.

6. The most favourable limits for composite (blended tailings/sand) pastefill mix design and stabilization was found to be between 176mm and 228mm slump. The mixtures in this study remained cohesive but workable within this zone.

CHAPTER 4

4.1 MECHANICAL PROPERTIES TESTS

4.1.1 General

The objectives of the mechanical properties tests were to:

1) examine factors that could affect high-density composite backfill properties including the range of strength development as a function of:

- particle size gradation
- moisture content
- binder type and composition
- curing environment and time and
- condition of loading

2) Also, to investigate the effects of specimen size on the mechanical properties of the composite fill as a means of inferring the potential behaviour of the placed material in situ. The investigations were made relative to cemented rockfill and paste backfill properties.

The following mechanical properties tests were carried out as part of this study:

i) Unconfined compressive strength tests (ASTM C192; C39)
ii) Direct shear tests (ASTM D3080)
iii) Triaxial compressive strength measurements (ASTM D2850)

4.2. UNCONFINED COMPRESSIVE STRENGTH TESTS

4.2.1 General

The main objective of this element of the study was to investigate parameters affecting unconfined compressive strength development in high-density composite backfill samples relative to cemented rockfill and tailings and sand paste backfill samples. The factors of interest included the range of strength development as a function of: Size and gradation of the fill materials, moisture content, binder type and composition, curing environment and time. The unconfined compressive strength tests were conducted in accordance with ASTM (C192 and C39) specifications.

4.2.2 Sample Preparation and Testing

Various batches of material were prepared in a planetary mixer to a uniform consistency. Binder compositions ranged between 0% and 10% dry weight of fill material. Various types of binders were used to consolidate the fill. These included Ordinary Portland cement (Type 10); Blast Furnace Slag and Fly Ash (Type C). Other combinations of Portland cement, slag and fly ash in variable proportions were also tested, with a 50/50 mixture of Portland cement and Fly Ash being the most common. With regard to the precious metal tailings containing a large fraction of minus 20μm material, special blends of ordinary Portland cement with supplementary binders were also tried as cementing agents. These resulted in the formulations of "Product A" and other binders which were used purposely for this study. The other binders included cementing agents containing anhydrite and metallurgical by-products.

4.2.3 Tailings and Sand Pastefill Samples

Tailings, sand and composite or blended tailings and sand paste fill test specimens were prepared at pulp densities ranging between 71% and 78% solids by weight. The test specimens consisted of cylinders with length-to-diameter (L/D) ratios of approximately 2:1. Two types of specimen diameters (102mm and 152mm) were used depending on the quantity of material available for testing. Four hundred and eight (480) test specimens were prepared and tested under this element of the study.

Figure 4.1 shows testing of unconfined compressive samples and a typical mode of failure for the composite tailings/sand samples.

(a) **(b)**

Figure 4.1 (a) and (b) show testing of unconfined compressive strength samples and the specimen failure mode.

4.2.4 Cemented Rockfill and Composite-Aggregate Paste (CAP) Samples

With regard to the cemented rockfill and composite aggregate paste backfill, the test samples were batched on the basis of weight. Ordinary Portland cement (Type 10) and Type C flyash were used as binders. Cement alone was used as a binder in half of the cylinders, while 50% cement/50% flash were used as the binder in the other half of the cylinders. The proportions of binder per dry weight of the fill materials varied between 5% and 7%.

The water and binder were mixed as slurry (pulp) densities ranging between 54% and 56% depending on the moisture content of the test material. The equivalent average water to binder ratio is about 0.8. A known weight of rockfill aggregates was then mixed with the binder slurry to a uniform consistency in a 0.25m^3 (9ft^3) capacity cement mixer. Each test sample was made from a whole mix batch in order to maintain a uniformity.

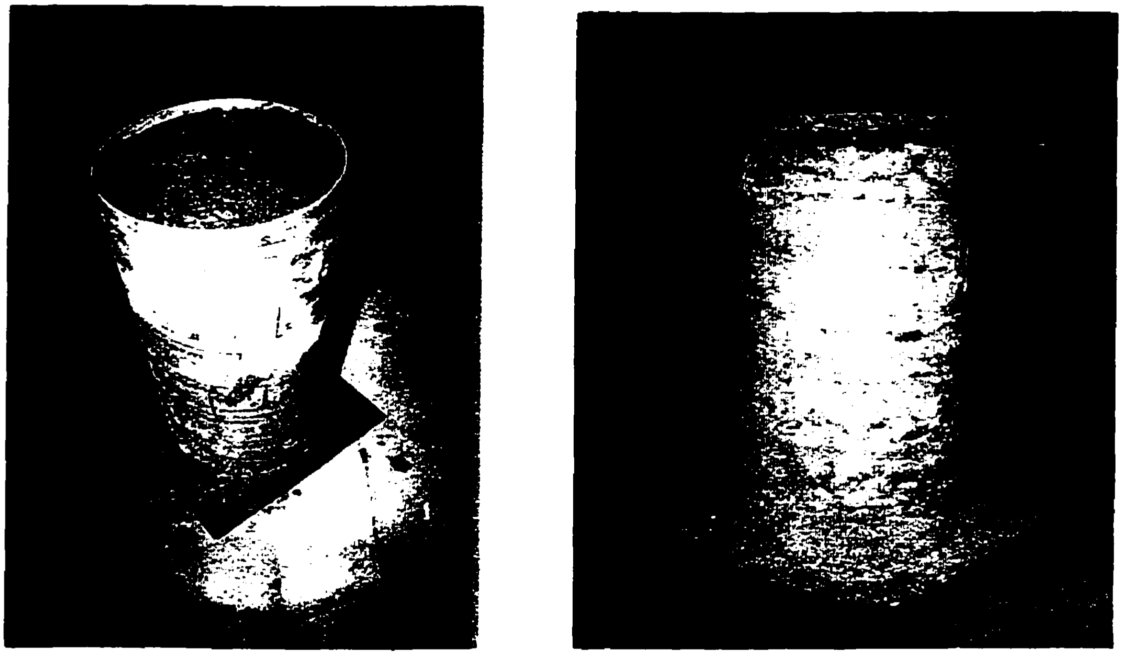
The proportions of coarse and fine aggregate compositions of the mix batches ranged between approximately 64 to 78% and 22 to 36% respectively. For the composite-aggregate paste fill samples, the proportions of coarse and fine aggregates compositions were respectively, 70% waste rock and 30% tailings. The test procedure for the tailings/sand composite were similar to that used for the paste fill specimens.

Two sizes of waxed cardboard cylinders (sono-tubes) were used for casting the test samples to study scale effects on the cemented rockfill properties. These were 152mm and 457mm diameter. Length-to-diameter (L/D) ratio of 2:1 were maintained for the cylinders. Because of the weight and volume of the 457mm diameter samples, special steel pallets were required for support and handling. The pallets also served as base platens for the test cylinders during unconfined compressive strength tests (Figure 4.2)

Prior to the casting of each cylinder, the surface of the pallet was coated with a light film of motor oil. This reduced end friction effect which could result in bonding between the cemented rockfill material and the pallet during testing for compressive strength.

The test batch of cemented rockfill mixtures was placed in the cylinder mould to a depth ranging between approximately 5mm to 30mm below the top of the mould depending on the specimen size as shown in Figure 4.2 (a). The cylinder was then weighed, a plastic bag placed over the top of the cylinder and left to cure in a curing chamber at a temperature range of 20° and 23° and about 95% to 98% Relative Humidity.

Unconfined Compressive strength testing was carried out over 7 to 56 days of sample curing. Two days before the designated testing date of a sample, the cylinder was again weighed. Next, the top of the cylinder mould was capped with a 10mm to 30mm thick mixture of plaster of Paris, and Ordinary Portland Cement with water depending on the specimen size. The mixture was trowelled level and allowed to cure at room temperature (Figure 4.2(b)).



(a) (b)

Figures 4.2 (a) Large Size (457 x 914 mm) Cemented Rockfill Tests Specimen
(b) Cemented Rockfill Tests Specimen showing base platen and cap

The test samples were tested using a 550KN universal testing machine (152mm samples) and a 10MN capacity servo-hydraulic press (457mm samples) (Figure 4.3). The maximum loading rate was 5.5KN/s. Axial displacement of a sample with increasing load was measured with two LVDT transducers positioned on diametrically opposite sides of the test cylinder. The displacements were converted to axial strain by averaging the two axial displacement readings at various load levels and dividing by the length of the cylinder. Figure 4.4 shows a failed tests specimen.

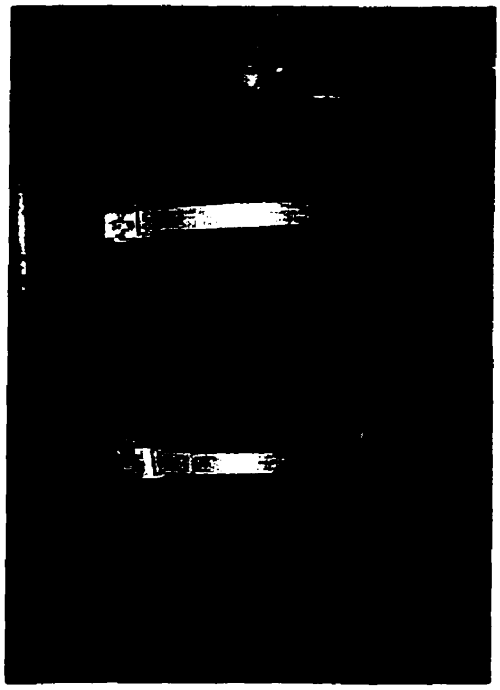


Figure 4.3 457mm x 914mm Composite Aggregate Paste Fill Test Specimen Being Loaded in Unconfined Compression

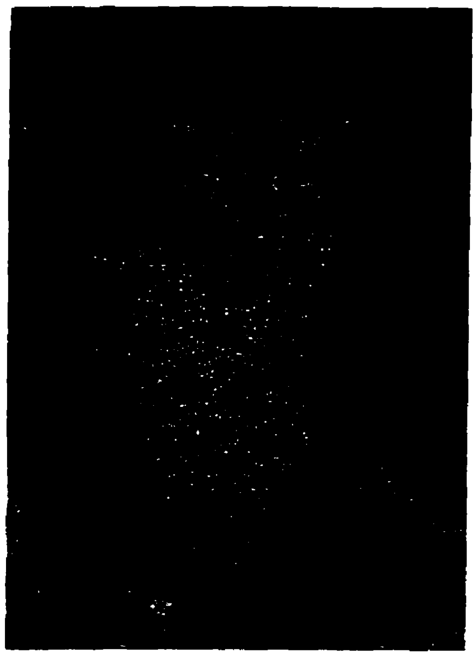


Figure 4.4 Failed 279mm x 558mm Tailings Pastefill Unconfined Compressive Strength Test Specimen

4.2.5 Scale Effects on High-Density Fill Properties

The main objective of this element of the study was to estimate the potential behaviour of composite backfills in situ, with reference to the scale effects observed in high-density tailings paste backfill and cemented rockfill systems (Yu and Counter, 1983; Reschke, 1993). Scale effects are essential for numerical analysis and must be taken into account in determining the required safety factors in stability analysis.

The test procedure was as described under the Unconfined Compressive Strength measurements (Section 4.2). The test specimen sizes for the scale effect determinations for the tailings, sand and tailings/sand composite paste fills consisted of cylinders ranging in sizes form 38mm to 279mm in diameter and cube specimens ranging in sizes from 51mm to 102mm (Figure 4.5). The cemented rockfill and composite-aggregate paste fill samples were limited to 152mm and 457mm diameter cylinders.

4.3 ANALYSIS OF UNCONFINED COMPRESSIVE STRENGTH TEST RESULTS

The unconfined compressive strength test results are presented in Appendix B-1. Typical stress-strain curves for the straight tailings paste fill, the composite tailings/sand paste fill, the cemented rockfill and the composite-aggregate paste (CAP) specimens are provided in Figures 4.6 to 4.8 The deformation modulus for a test sample was calculated as a tangent modulus at 50% the failure strength. The test results are discussed in Section 4.4.

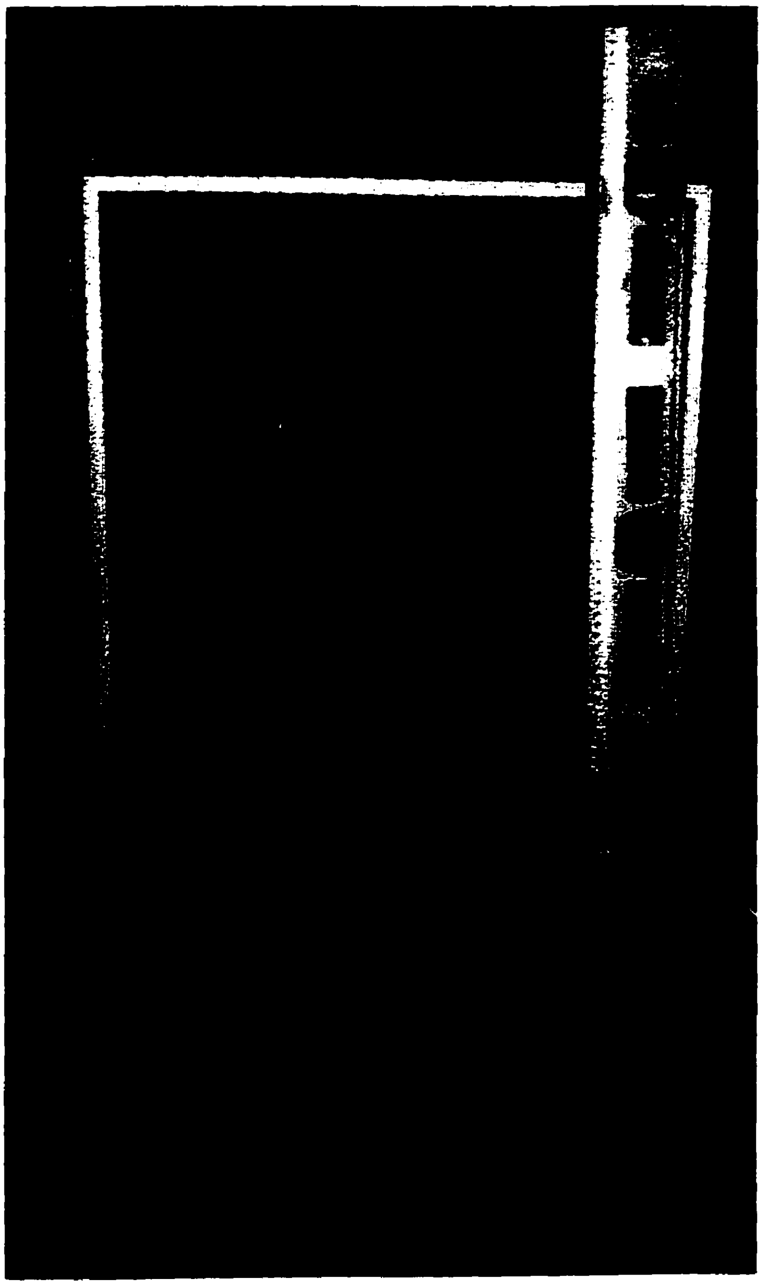


Figure 4.5 Scale Effects Test Specimens for Mill Tailings Samples

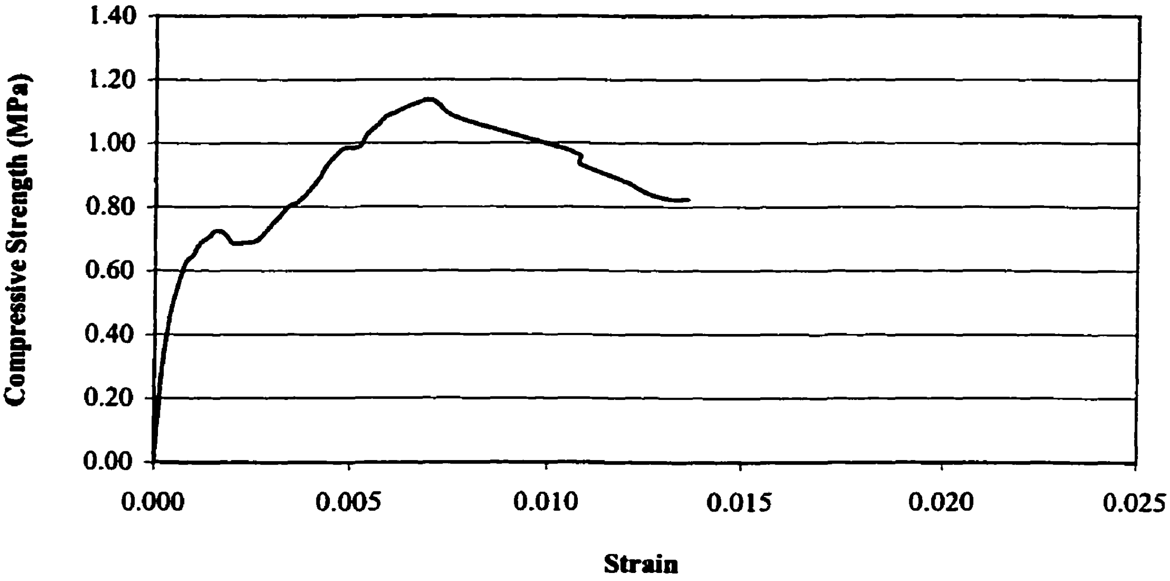


Figure 4.6 (a) Typical Stress vs Strain Curve for Tailings Pastefill Samples in Unconfined Compression Testing (14 Days Curing)

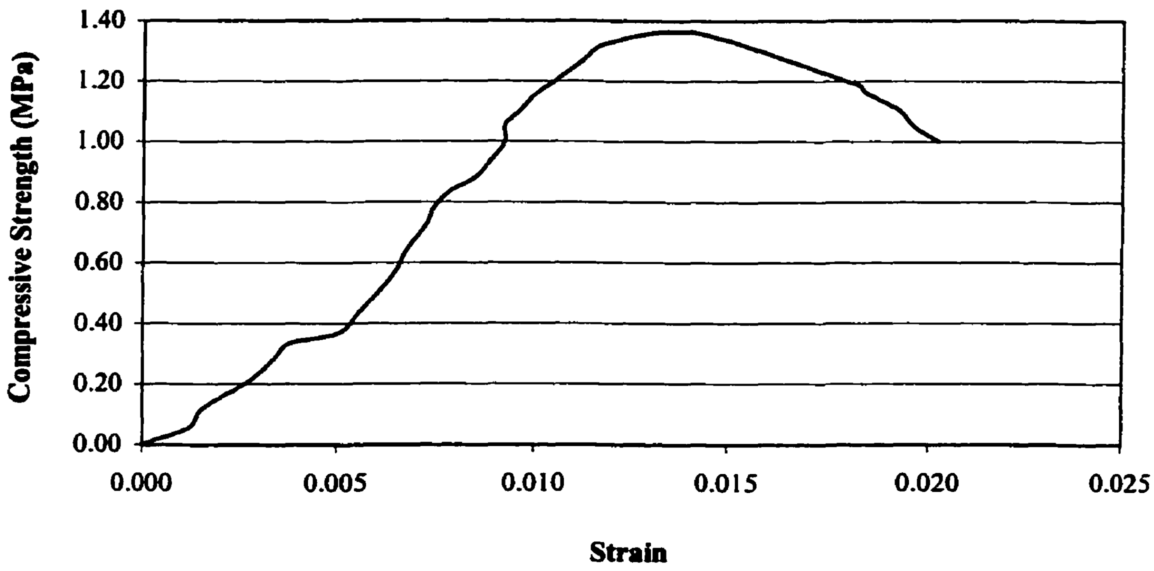


Figure 4.6 (b) Typical Stress vs Strain Curve for Composite-Blended Tailings/ Sand Pastefill Samples (14 Days Curing)

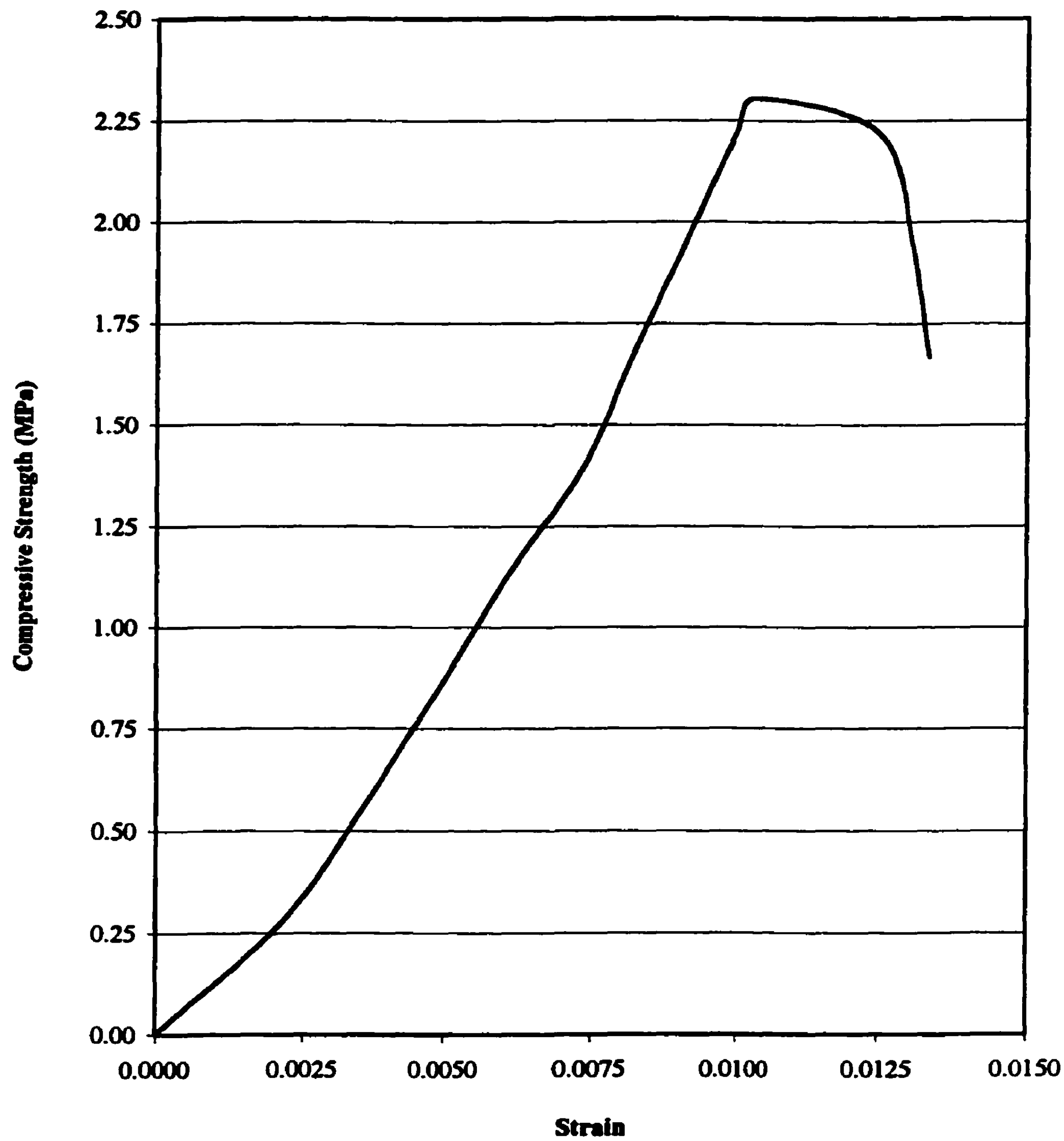


Figure 4.7 Typical Stress vs Strain Curve for Cemented Rockfill Sample in Unconfined Compression Testing (14 Days Curing)

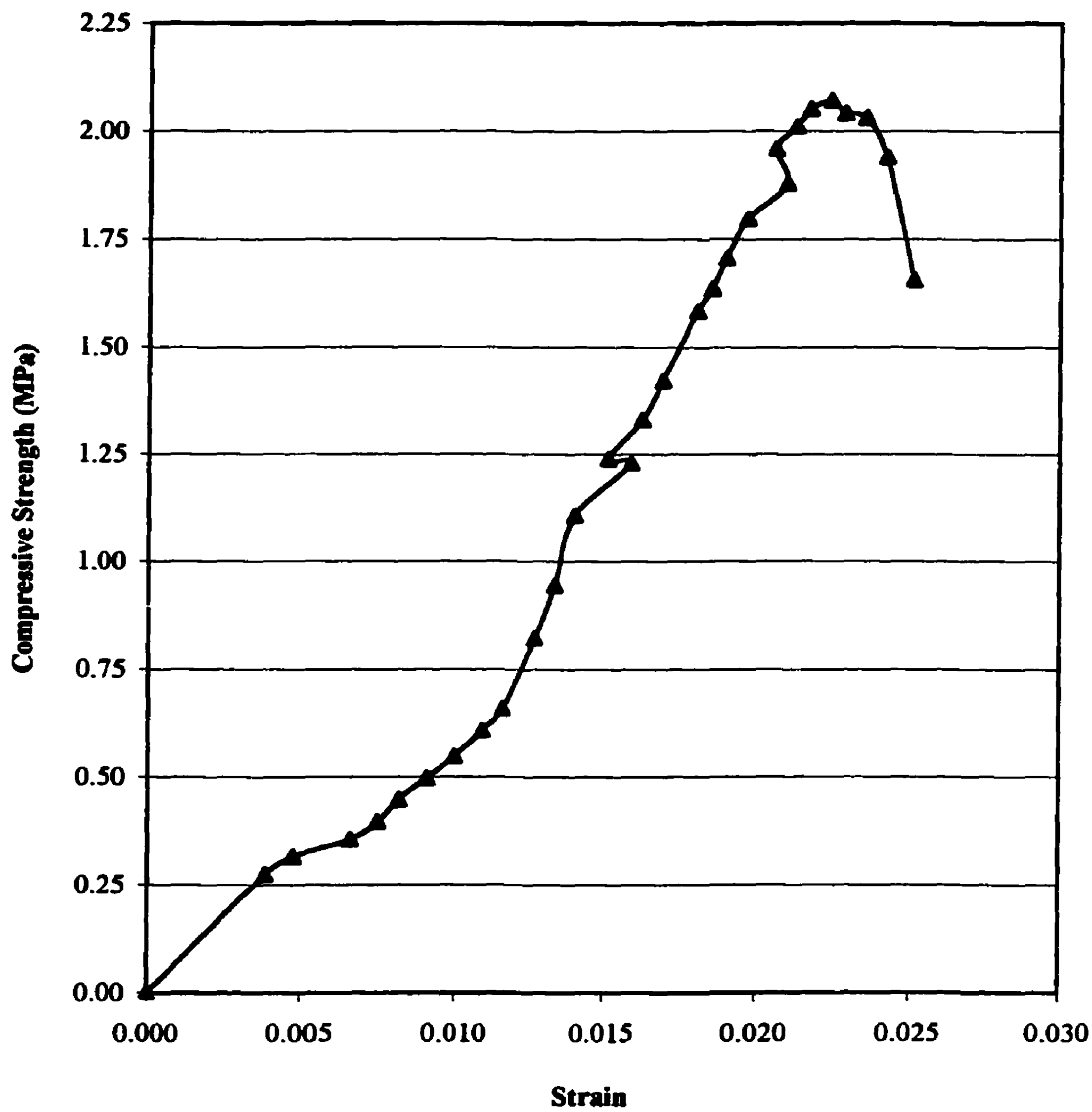


Figure 4.8 Typical Stress vs Strain Curves for Composite-Aggregate Pastefill (CAP) in Unconfined Compression Testing (14 Days Curing)

4.4 DISCUSSIONS-MECHANICAL PROPERTIES AND BEHAVIOUR OF TAILINGS AND SAND PASTE FILL TEST SAMPLES

4.4.1 Summary of Test Results

This section deals with the analyses of the unconfined compressive strength tests results. The mechanical properties of the tailings and sand paste fill samples were determined and evaluated in terms of the type of ore, binder type and composition, moisture content, curing environment and time, as well as, confinement. The scale effects on strength and deformation properties were also investigated for the tailings fill. The bulk of the evaluation consisted of unconfined compressive strength testing; and the tests were conducted using both precious metal and base metal ore tailings. Three types of alluvial sands were also investigated as high-density fill products in this study as a fundamental step towards the understanding of the characteristics and behaviour of high-density composite backfill materials. A total of seven hundred and fifty-nine (759) specimens were tested in uniaxial compression. The results are summarized in terms of the specified test conditions, curing time and binder content in Table 4.4.

In general terms, the results compare favourably with reported results in the published literature, especially those reported by Vickery and Boldt, 1989, Boldt et al., 1993, Ross-Watt (1989) and Hedley (1995) on straight tailings, sand and blended tailings/sand paste fill samples.

The results of the cemented rockfill tests also compare favourably with the range of values reported by Yu and Counter (1983), Yu (1990), Hedley (1995), Farsangi (1996) and Reschke (1993). The results of the uniaxial compressive strength tests are treated in Sections 4.4.2 to 4.4.6 below in terms of the stated objectives of the various test conditions examined in this study.

Table 4.1 Summary of mechanical properties test results for tailings and sand samples

		Physical Properties									Mechanical Properties		Engineering Properties	
Fill Type	**Fill Source**	**Specific Gravity**	**C_u**	**C_c**	**% - 20µm**	**Bulk Density, ρ**	**Curing Period**	**Sample Diameter**	**Total Binder Content**	**Moisture Content at Testing**	**Compressive Strength, σ_c**	**Deformation Modulus, E**	**Void Ratio, e**	**Porosity, η**
						(kg/m^3)	**(Days)**	**(mm)**	**(%)**	**(%)**	**(MPa)**	**(GPa)**		**(%)**
Mill Tailings Paste **Total Tailings**	Base Metal Tailings 1 **N = 27**	3.9	10.8	0.32	43.5	2528 - 2715 ***2618.89***	14	102	2	N/A	0.150 - 0.170 ***0.160***	0.016 - 0.133 ***0.057***	N/A	N/A
									4		0.180 - 0.240 ***0.213***	0.017 - 0.046 ***0.031***		
									6		0.360 - 0.530 ***0.450***	0.080 - 0.136 ***0.102***		
							28	102	2		0.240 - 0.290 ***0.257***	0.011 - 0.020 ***0.015***		
									4		0.340 - 0.410 ***0.373***	0.030 - 0.042 ***0.034***		
									6		0.450 - 0.630 ***0.553***	0.094 - 0.200 ***0.136***		
							56	102	2		0.300 - 0.370 ***0.333***	0.036 - 0.071 ***0.050***		
									4		0.440 - 0.500 ***0.470***	0.043 - 0.067 ***0.053***		
									6		0.750 - 0.830 ***0.800***	0.067 - 0.133 ***0.093***		
							14, 28, 56	102	2, 4, 6	N/A	0.150 - 0.830 ***0.401***	0.011 - 0.200 ***0.063***	N/A	N/A
	Base Metal Tailings 2	2.92	10.9	1.05	25.2	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A
	Base Metal Tailings 3	2.64	7.4	1.72	18.3	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A

Table 4.1 Summary of mechanical properties test results for tailings and sand samples continued.

		Physical Properties									Mechanical Properties		Engineering Properties	
Fill Type	Fill Source	Specific Gravity	C_u	C_c	% - 20m	Bulk Density, ρ	Curing Period	Sample Diameter	Total Binder Content	Moisture Content at Testing	Compressive Strength, σ_c	Deformation Modulus, E	Void Ratio, e	Porosity, η
						(kg/m^3)	(Days)	(mm)	(%)	(%)	(MPa)	(GPa)		(%)
Mill Tailings Paste Total Tailings	Precious Metal Tailings 1	2.74	5.3	0.26	42.3	1870 - 1998 ***1947***			1	22.5 - 24.3 ***23.4***	0.054 - 0.070 ***0.062***	0.001 - 0.001 ***0.001***	0.42 - 0.47 ***0.45***	29.7 - 32.0 ***30.9***
									3	23.1 - 23.7 ***23.4***	0.160 - 0.185 ***0.173***	0.008 - 0.010 ***0.009***	0.39 - 0.41 ***0.4***	28.3 - 29.2 ***28.7***
							3	102	5	20.9 - 21.4 ***21.1***	0.299 - 0.333 ***0.316***	0.02 - 0.024 ***0.022***	0.42 - 0.44 ***0.43***	29.5 - 30.6 ***30.0***
									7.5	19.1 - 19.6 ***19.3***	0.357 - 0.395 ***0.376***	0.055 - 0.079 ***0.067***	0.39 - 0.39 ***0.39***	28.0 - 28.1 ***28.1***
									10	18.4 - 19.4 ***18.9***	0.493 - 0.543 ***0.518***	0.252 - 0.260 ***0.256***	0.38 - 0.38 ***0.38***	27.4 - 27.7 ***27.5***
									1	22.0 - 23.3 ***22.6***	0.271 - 0.296 ***0.284***	0.002 - 0.003 ***0.002***	0.45 - 0.45 ***0.45***	31.0 - 31.2 ***31.1***
									3	20.4 - 22.1 ***21.3***	0.308 - 0.333 ***0.321***	0.009 - 0.030 ***0.02***	0.38 - 0.40 ***0.39***	27.6 - 28.8 ***28.2***
	N = 26						7	102	5	20.4 - 20.8 ***20.6***	0.382 - 0.395 ***0.389***	0.023 - 0.085 ***0.054***	0.39 - 0.43 ***0.41***	28.2 - 30.1 ***29.1***
									7.5	18.9 - 18.9 ***18.9***	0.444 - 0.543 ***0.494***	0.146 - 0.156 ***0.151***	0.39 - 0.40 ***0.4***	28.2 - 28.5 ***28.4***
									10	18.4 - 18.8 ***18.6***	0.703 - 0.789 ***0.746***	0.059 - 0.091 ***0.075***	0.39 - 0.40 ***0.4***	27.9 - 28.7 ***28.3***
							14	102	5	22.2 - 22.5 ***22.4***	0.423 - 0.474 ***0.449***	0.028 - 0.098 ***0.063***	0.44 - 0.46 ***0.45***	30.6 - 31.4 ***31.0***
							28	102	5	22.3 - 23.5 ***22.9***	0.433 - 0.518 ***0.476***	0.018 - 0.078 ***0.048***	0.41 - 0.41 ***0.41***	28.9 - 29.1 ***29.0***
							56	102	5	21.3 - 22.3 ***21.8***	0.581 - 0.581 ***0.581***	0.044 - 0.101 ***0.073***	0.41 - 0.42 ***0.42***	29.2 - 29.5 ***29.3***
							3,7,14, 28, 56	102	1,3,5 7.5,10	18.4 - 24.3 ***21.2***	0.054 - 0.789 ***0.399***	0.001 - 0.260 ***0.065***	0.39 - 0.47 ***0.41***	27.4 - 32.0 ***29.2***

Table 4.1 Summary of mechanical properties test results for tailings and sand samples continued.

Fill Type	Fill Source	Physical Properties: Specific Gravity	Physical Properties: C_u	Physical Properties: C_c	Physical Properties: % - 20m	Physical Properties: Bulk Density, ρ (kg/m³)	Curing Period (Days)	Sample Diameter (mm)	Total Binder Content (%)	Moisture Content at Testing (%)	Mechanical Properties: Compressive Strength, σ_c (MPa)	Mechanical Properties: Deformation Modulus, E (GPa)	Engineering Properties: Void Ratio, e	Engineering Properties: Porosity, η (%)
Mill Tailings Paste Total Tailings	Precious Metal Tailings 2 N = 135	2.74	7.5	0.63	40.5	1734 - 2015 ***1883***	14	102	2	N/A	0.196 - 0.490 ***0.295***	0.017 - 0.026 ***0.023***	N/A	N/A
									3		0.256 - 0.332 ***0.292***	0.019 - 0.029 ***0.025***		
									4		0.578 - 0.623 ***0.605***	0.070 - 0.071 ***0.071***		
									5		0.379 - 0.530 ***0.454***	0.040 - 0.060 ***0.051***		
									6		1.030 - 1.100 ***1.053***	0.110 - 0.143 ***0.130***		
									8		0.598 - 1.504 ***1.014***	0.082 - 0.412 ***0.218***		
							28	102	2		0.231 - 0.289 ***0.253***	0.032 - 0.043 ***0.038***		
									3		0.289 - 0.432 ***0.375***	0.024 - 0.039 ***0.031***		
									4		0.834 - 0.920 ***0.882***	0.092 - 0.111 ***0.103***		
									5		0.395 - 0.650 ***0.524***	0.044 - 0.081 ***0.059***		
									6		1.110 - 1.340 ***1.240***	0.198 - 0.308 ***0.243***		
									8		0.644 - 1.899 ***1.264***	0.081 - 0.519 ***0.287***		
							56	102	2		0.291 - 0.355 ***0.328***	0.036 - 0.094 ***0.060***		
									3		0.290 - 0.530 ***0.409***	0.029 - 0.045 ***0.036***		
									4		0.840 - 1.070 ***0.943***	0.107 - 0.127 ***0.116***		
									5		0.487 - 0.829 ***0.637***	0.057 - 0.106 ***0.078***		
									8		0.730 - 2.515 ***1.524***	0.091 - 0.797 ***0.413***		
							7	152	3	23.8 - 27.0 ***25.5***	0.055 - 0.230 ***0.129***	0.006 - 0.028 ***0.014***	0.44 - 0.55 ***0.46***	30.6 - 35.4 ***31.7***
									5	23.3 - 27.2 ***25.6***	0.120 - 0.430 ***0.286***	0.012 - 0.071 ***0.032***	0.44 - 0.55 ***0.47***	30.3 - 35.4 ***31.7***
							28	152	3	22.3 - 27.4 ***25.2***	0.148 - 0.340 ***0.229***	0.005 - 0.036 ***0.019***	0.45 - 0.47 ***0.46***	31.0 - 32.2 ***31.7***
									5	21.1 - 27.0 ***24.8***	0.247 - 0.660 ***0.408***	0.027 - 0.084 ***0.056***	0.43 - 0.48 ***0.46***	30.3 - 32.5 ***31.4***
							7,14, 28,56	102,152	2,3,4, 5,6,8	17.5 - 27.4 ***22.7***	0.055 - 2.515 ***0.474***	0.005 - 0.797 ***0.069***	0.36 - 0.62 ***0.47***	26.5 - 38.1 ***31.6***

Table 4.1 Summary of mechanical properties test results for tailings and sand samples continued.

Fill Type	Fill Source	Physical Properties: Specific Gravity	C_u	C_c	% - 20m	Bulk Density, ρ (kg/m³)	Curing Period (Days)	Sample Diameter (mm)	Total Binder Content (%)	Moisture Content at Testing (%)	Mechanical Properties: Compressive Strength, σ_c (MPa)	Deformation Modulus, E (GPa)	Engineering Properties: Void Ratio, e	Porosity, η (%)
Mill Tailings Paste Total Tailings	Precious Metal Tailings 3	3.43	3.8	0.92	74.8	1624 - 2694 ***2127***	7	38	6	20.3 - 21.4 ***21.1***	1.050 - 2.810 ***1.730***	0.039 - 0.264 ***0.122***	0.41 - 0.59 ***0.47***	29.2 - 38.9 ***31.7***
							14	38	6	5.5 - 7.7 ***6.9***	1.490 - 3.070 ***2.266***	0.119 - 2.200 ***0.663***	0.46 - 0.59 ***0.51***	31.5 - 36.9 ***33.9***
							28	38	6	1.8 - 3.5 ***2.6***	1.050 - 3.420 ***2.395***	0.034 - 1.706 ***0.336***	0.50 - 0.64 ***0.57***	33.6 - 38.9 ***36.3***
							7	51	6	2.5 - 8.1 ***4.2***	0.271 - 0.691 ***0.462***	0.032 - 0.207 ***0.100***	0.74 - 0.88 ***0.78***	42.6 - 46.9 ***43.7***
							14	51	6	1.0 - 1.8 ***1.3***	2.950 - 4.490 ***4.020***	0.108 - 2.783 ***0.590***	0.74 - 0.79 ***0.76***	42.4 - 44.2 ***43.2***
							28	51	6	0.7 - 1.2 ***1.0***	1.700 - 3.640 ***2.452***	0.032 - 0.157 ***0.082***	0.70 - 0.80 ***0.74***	41.0 - 44.4 ***42.3***
							7	76	6	2.0 - 15.9 ***11.5***	1.270 - 3.460 ***1.898***	0.031 - 0.386 ***0.175***	0.57 - 0.89 ***0.65***	36.3 - 47.1 ***39.1***
							14	76	6	2.3 - 12.3 ***10.0***	1.450 - 2.740 ***1.791***	0.034 - 0.550 ***0.212***	0.63 - 1.11 ***0.71***	38.7 - 52.6 ***41.0***
							28	76	6	1.2 - 8.9 ***7.3***	1.450 - 2.410 ***2.057***	0.040 - 0.330 ***0.199***	0.54 - 0.95 ***0.68***	35.1 - 48.7 ***40.3***
	N = 130						7	102	6	4.5 - 19.5 ***14.3***	0.826 - 2.110 ***1.368***	0.106 - 0.295 ***0.170***	0.53 - 1.04 ***0.66***	34.6 - 51.0 ***39.2***
							14	102	6	1.9 - 17.3 ***13.1***	0.826 - 2.060 ***1.176***	0.062 - 0.180 ***0.132***	0.55 - 0.92 ***0.65***	35.5 - 47.9 ***38.9***
							28	102	6	1.5 - 12.5 ***7.8***	0.962 - 2.190 ***1.365***	0.061 - 0.159 ***0.119***	0.61 - 0.96 ***0.76***	37.9 - 49 ***42.6***
							7	152	6	19.3 - 19.9 ***19.5***	0.445 - 0.757 ***0.631***	0.195 - 0.560 ***0.318***	0.54 - 0.56 ***0.55***	35.1 - 35.9 ***35.5***
							14	152	6	18.9 - 19.7 ***19.4***	1.137 - 1.225 ***1.188***	0.111 - 0.231 ***0.158***	0.54 - 0.55 ***0.55***	35.1 - 35.5 ***35.3***
							28	152	6	15.5 - 17.8 ***16.8***	1.121 - 1.484 ***1.341***	0.173 - 0.650 ***0.408***	0.55 - 0.60 ***0.57***	35.5 - 37.5 ***36.2***
							7	279	6	12.1 ***12.1***	0.342 ***0.342***	0.032 ***0.032***	0.29 ***0.29***	31 ***31.0***
							14	279	6	N/A	0.522 ***0.522***	0.057 ***0.057***	0.54 ***0.54***	29 ***29.0***
							28	279	6	9.2 ***9.2***	0.538 ***0.538***	0.033 ***0.033***	0.27 ***0.27***	29.3 ***29.3***
							7,14,28	38,51,76, 102,152,279	6	0.7 - 21.4 ***9.5***	0.271 - 4.490 ***1.856***	0.031 - 2.783 ***0.260***	0.27 - 1.11 ***0.62***	21.3 - 52.6 ***38.0***

Table 4.1 Summary of mechanical properties test results for tailings and sand samples continued.

Fill Type	Fill Source	Physical Properties: Specific Gravity	Physical Properties: C_u	Physical Properties: C_c	Physical Properties: % - 20m	Physical Properties: Bulk Density, ρ (kg/m³)	Curing Period (Days)	Sample Diameter (mm)	Total Binder Content (%)	Moisture Content at Testing (%)	Mechanical Properties: Compressive Strength, σ_c (MPa)	Mechanical Properties: Deformation Modulus, E (GPa)	Engineering Properties: Void Ratio, e	Engineering Properties: Porosity, η (%)
Mill Tailings Paste Classified Tailings	Base Metal Tailings 1 N = 27	3.92	5.9	1.8	15.5	2624 - 2891 ***2791***	14	102	2	N/A	0.240 - 0.290 ***0.263***	0.014 - 0.040 ***0.026***	N/A	N/A
							14	102	4	N/A	0.250 - 0.330 ***0.283***	0.030 - 0.083 ***0.050***	N/A	N/A
							14	102	6	N/A	0.540 - 0.670 ***0.587***	0.120 - 0.200 ***0.166***	N/A	N/A
							28	102	2	N/A	0.270 - 0.340 ***0.305***	0.017 - 0.031 ***0.024***	N/A	N/A
							28	102	4	N/A	0.300 - 0.510 ***0.407***	0.027 - 0.060 ***0.039***	N/A	N/A
							28	102	6	N/A	0.630 - 0.710 ***0.660***	0.039 - 0.133 ***0.080***	N/A	N/A
							56	102	2	N/A	0.470 - 0.530 ***0.493***	0.050 - 0.067 ***0.060***	N/A	N/A
							56	102	4	N/A	0.440 - 0.630 ***0.540***	0.050 - 0.067 ***0.060***	N/A	N/A
							56	102	6	N/A	0.930 - 1.010 ***0.977***	0.143 - 0.273 ***0.194***	N/A	N/A
							14, 28, 56	102	2, 4, 6	N/A	0.240 - 1.010 ***0.509***	0.014 - 0.273 ***0.080***	N/A	N/A
	Base Metal Tailings 4 N = 84	2.94	6	1.01	9.5	1635 - 2063 ***1879***	14	102	3	N/A	0.195 - 0.605 ***0.397***	0.013 - 0.078 ***0.049***	0.89 - 1.17 ***0.98***	47.1 - 53.8 ***49.3***
							14	102	4	N/A	0.309 - 0.340 ***0.319***	0.041 - 0.047 ***0.044***	0.94 - 0.95 ***0.94***	48.5 - 48.6 ***48.5***
							14	102	5	N/A	0.578 - 1.418 ***0.989***	0.115 - 0.258 ***0.191***	0.74 - 1.04 ***0.87***	42.6 - 51.0 ***46.3***
							14	102	6	N/A	0.961 - 1.100 ***1.030***	0.117 - 0.117 ***0.117***	0.72 - 0.73 ***0.72***	42.0 - 42.1 ***42.0***
							28	102	3	N/A	0.231 - 0.684 ***0.484***	0.029 - 0.095 ***0.059***	0.89 - 1.17 ***0.98***	47.1 - 53.8 ***49.2***
							28	102	4	N/A	0.570 - 0.600 ***0.590***	0.086 - 0.107 ***0.098***	0.94 - 0.95 ***0.95***	48.5 - 48.7 ***48.6***
							28	102	5	N/A	0.834 - 1.554 ***1.186***	0.148 - 0.298 ***0.222***	0.78 - 1.04 ***0.89***	43.7 - 50.9 ***46.8***
							28	102	6	N/A	1.110 - 1.340 ***1.240***	0.167 - 0.176 ***0.171***	0.72 - 0.77 ***0.75***	41.7 - 43.6 ***42.8***
							56	102	3	N/A	0.290 - 0.784 ***0.530***	0.044 - 0.131 ***0.095***	0.89 - 1.12 ***0.99***	47.0 - 52.8 ***49.6***
							56	102	4	N/A	0.620 - 0.830 ***0.703***	0.114 - 0.119 ***0.116***	0.95 - 0.95 ***0.95***	48.6 - 48.6 ***48.6***
							56	102	5	N/A	0.840 - 2.000 ***1.383***	0.197 - 0.857 ***0.381***	0.72 - 1.02 ***0.85***	42.0 - 50.6 ***45.9***
							56	102	6	N/A	0.921 - 1.380 ***1.124***	0.140 - 0.198 ***0.175***	0.76 - 0.82 ***0.78***	43.2 - 45.1 ***43.9***
							14	152	5	N/A	0.578 - 0.623 ***0.605***	0.039 - 0.044 ***0.042***	0.46 - 0.51 ***0.48***	31.3 - 33.9 ***32.3***
							28	152	5	N/A	0.834 - 0.920 ***0.882***	0.055 - 0.086 ***0.073***	0.45 - 0.53 ***0.50***	31.2 - 34.7 ***33.1***
							56	152	5	N/A	0.840 - 1.070 ***0.943***	0.078 - 0.093 ***0.086***	0.46 - 0.47 ***0.46***	31.3 - 31.7 ***31.4***
							14,28,56	102,152	3,4,5,6	N/A	0.195 - 2.000 ***0.811***	0.013 - 0.857 ***0.141***	0.45 - 1.17 ***0.85***	31.2 - 53.8 ***45.2***

Table 4.1 Summary of mechanical properties test results for tailings and sand samples continued.

		Physical Properties									Mechanical Properties		Engineering Properties	
Fill Type	**Fill Source**	**Specific Gravity**	C_u	C_c	**% - 20m**	**Bulk Density, ρ**	**Curing Period**	**Sample Diameter**	**Total Binder Content**	**Moisture Content at Testing**	**Compressive Strength, σ_c**	**Deformation Modulus, E**	**Void Ratio, e**	**Porosity, η**
						(kg/m^3)	(Days)	(mm)	(%)	(%)	(MPa)	(GPa)		(%)
Sand	Sand 1	2.55	4.2	1.15	26.5	1610 - 1845 ***1785***	14	102	1	24.8 - 24.9 ***24.9***	0.038 - 0.086 ***0.062***	0.001 - 0.001 ***0.001***	0.41 - 0.42 ***0.42***	29.0 - 29.8 ***29.4***
									3	29.0 ***29.0***	0.086 ***0.086***	0.002 ***0.002***	0.49 ***0.49***	32.7 ***32.7***
									5	23.4 - 23.6 ***23.5***	0.321 - 0.345 ***0.333***	0.015 - 0.019 ***0.017***	0.43 - 0.43 ***0.43***	30.2 - 30.3 ***30.2***
									7.5	25.0 - 26.8 ***25.9***	0.419 - 0.423 ***0.421***	0.028 - 0.095 ***0.061***	0.46 - 0.47 ***0.47***	31.4 - 32.0 ***31.7***
									10	26.7 - 27.1 ***26.9***	0.432 - 0.481 ***0.457***	0.059 - 0.085 ***0.072***	0.46 - 0.46 ***0.46***	31.5 - 31.5 ***31.5***
							28	102	1	24.8 - 25.1 ***24.9***	0.099 - 0.099 ***0.099***	0.001 - 0.001 ***0.001***	0.42 - 0.43 ***0.43***	29.5 - 30.1 ***29.8***
									3	29.3 ***29.3***	0.099 ***0.099***	0.007 ***0.007***	0.48 ***0.48***	32.2 ***32.2***
	N = 27								5	24.2 - 24.7 ***24.4***	0.370 - 0.419 ***0.395***	0.018 - 0.019 ***0.018***	0.43 - 0.43 ***0.43***	29.9 - 32.7 ***31.8***
									7.5	26.7 - 27.5 ***27.1***	0.469 - 0.493 ***0.481***	0.030 - 0.049 ***0.040***	0.46 - 0.46 ***0.46***	31.4 - 31.5 ***31.4***
									10	24.5 - 25.4 ***24.9***	0.555 - 0.654 ***0.605***	0.026 - 0.077 ***0.051***	0.46 - 0.48 ***0.47***	31.7 - 32.5 ***32.1***
							56	102	1	23.3 - 24.1 ***23.7***	0.123 - 0.123 ***0.123***	0.001 - 0.002 ***0.001***	0.42 - 0.44 ***0.43***	29.7 - 30.6 ***30.1***
									3	29.2 ***29.2***	0.148 ***0.148***	0.009 ***0.009***	0.61 ***0.61***	38.1 ***38.1***
									5	23.8 - 24.2 ***24.0***	0.493 - 0.506 ***0.500***	0.009 - 0.018 ***0.014***	0.43 - 0.43 ***0.43***	30.1 - 30.3 ***30.2***
									7.5	26.0 - 26.5 ***26.2***	0.481 - 0.555 ***0.518***	0.028 - 0.029 ***0.029***	0.48 - 0.48 ***0.48***	32.3 - 32.3 ***32.3***
									10	23.5 - 24.5 ***24.0***	0.629 - 0.641 ***0.635***	0.052 - 0.067 ***0.060***	0.48 - 0.50 ***0.49***	32.6 - 33.2 ***32.9***
							14,28,56	102	1,3,5 7.5,10	23.3 - 29.3 ***25.5***	0.038 - 0.654 ***0.355***	0.001 - 0.095 ***0.028***	0.41 - 0.61 ***0.46***	29.0 - 38.1 ***31.3***
	Sand 2	2.67	6.5	1.45	13.8	1944 - 2077 ***2016***	3	102	8	13.6 - 15.7 ***14.8***	0.370 - 0.875 ***0.607***	0.017 - 0.120 ***0.071***	0.29 - 0.30 ***0.29***	22.2 - 23.1 ***22.5***
	N = 12						7	102	8	10.5 - 13.4 ***11.8***	0.493 - 1.689 ***0.977***	0.060 - 0.158 ***0.089***	0.31 - 0.35 ***0.33***	23.8 - 25.8 ***24.7***
							14	102	8	8.2 - 9.6 ***8.9***	1.147 - 2.207 ***1.606***	0.075 - 0.190 ***0.109***	0.34 - 0.37 ***0.36***	25.4 - 27.2 ***26.3***
							3,7,14	102	6	8.2 - 15.7 ***11.8***	0.370 - 2.207 ***1.063***	0.017 - 0.190 ***0.090***	0.29 - 0.37 ***0.33***	22.2 - 27.2 ***24.5***
	Sand 3	2.67	11.0	1.51	11.9	1944 - 2109 ***2045***	3	102	8	8.8 - 11.0 ***9.9***	0.543 - 0.875 ***0.666***	0.056 - 0.113 ***0.073***	0.27 - 0.32 ***0.30***	21.0 - 24.2 ***22.7***
	N = 12						7	102	8	8.1 - 8.3 ***8.2***	0.875 - 1.085 ***0.946***	0.062 - 0.137 ***0.096***	0.27 - 0.31 ***0.29***	21.4 - 23.6 ***22.3***
							14	102	8	3.3 - 7.3 ***6.1***	1.159 - 2.219 ***1.806***	0.125 - 0.272 ***0.184***	0.29 - 0.37 ***0.34***	22.6 - 27.2 ***25.2***
							3,7,14	102	6	3.3 - 11.0 ***8.1***	0.543 - 2.219 ***1.140***	0.056 - 0.272 ***0.118***	0.27 - 0.37 ***0.31***	21.0 - 27.2 ***23.4***

Table 4.1 Summary of mechanical properties test results for tailings and sand samples continued.

Fill Type	Fill Source	Physical Properties: Specific Gravity	C_u	C_c	% - 20m	Bulk Density, ρ	Curing Period	Sample Diameter	Total Binder Content	Moisture Content at Testing	Mechanical Properties: Compressive Strength, σ_c	Deformation Modulus, E	Engineering Properties: Void Ratio, e	Porosity, η
						(kg/m³)	(Days)	(mm)	(%)	(%)	(MPa)	(GPa)		(%)
Composite Fill (Tailings / Sand)	Precious Metal Tailings 2 77.5% Tailings 22.5% Sand	N/A	N/A	N/A	N/A	1920 - 1967 ***1945***	7	152	3	19.7 - 23.7 ***21.6***	0.100 - 0.430 ***0.240***	0.010 - 0.050 ***0.020***	0.39 - 0.42 ***0.40***	28.2 - 29.3 ***28.8***
									5	18.0 - 23.6 ***21.5***	0.180 - 0.580 ***0.340***	0.020 - 0.070 ***0.040***	0.40 - 0.42 ***0.41***	28.4 - 29.5 ***28.9***
	N = 40						28	152	3	17.5 - 23.9 ***21.5***	0.150 - 0.580 ***0.330***	0.010 - 0.060 ***0.030***	0.40 - 0.43 ***0.42***	28.8 - 29.9 ***29.4***
									5	17.9 - 23.4 ***20.9***	0.140 - 0.990 ***0.570***	0.030 - 0.110 ***0.070***	0.40 - 0.42 ***0.41***	28.5 - 29.8 ***28.9***
							7,28	152	3,5	17.5 - 23.9 ***21.4***	0.100 - 0.990 ***0.368***	0.010 - 0.110 ***0.040***	0.39 - 0.43 ***0.41***	28.2 - 29.9 ***29.0***
	Precious Metal Tailings 2 55% Tailings 45% Sand	N/A	N/A	N/A	N/A	1955 - 2015 ***1989***	7	152	3	20.7 - 21.4 ***20.97***	0.080 - 0.120 ***0.090***	0.010 - 0.010 ***0.010***	0.37 - 0.39 ***0.38***	26.9 - 27.8 ***27.5***
									5	19.7 - 21.4 ***20.5***	0.220 - 0.320 ***0.260***	0.020 - 0.040 ***0.030***	0.36 - 0.38 ***0.37***	26.5 - 27.7 ***27.1***
	N = 23						28	152	3	19.0 - 21.3 ***20.0***	0.120 - 0.440 ***0.280***	0.020 - 0.040 ***0.030***	0.36 - 0.39 ***0.38***	26.7 - 27.9 ***27.3***
									5	19.7 - 21.7 ***20.8***	0.230 - 0.320 ***0.280***	0.010 - 0.060 ***0.030***	0.37 - 0.40 ***0.38***	27.2 - 28.6 ***27.7***
							7,28	152	3,5	19.0 - 21.7 ***20.6***	0.080 - 0.440 ***0.226***	0.010 - 0.060 ***0.023***	0.36 - 0.40 ***0.38***	26.5 - 28.6 ***27.4***
	Precious Metal Tailings 2 90% Tailings 10% Sand	N/A	N/A	N/A	N/A	1852 - 1872 ***1864***	7	152	5		0.450 - 0.490 ***0.477***	0.013 - 0.021 ***0.160***	0.50 - 0.50 ***0.50***	33.1 - 33.5 ***33.2***
							14	152	5	N/A	0.520 - 0.560 ***0.540***	0.031 - 0.036 ***0.033***	0.50 - 0.51 ***0.50***	33.1 - 33.6 ***33.3***
Total N = 99	N = 9						28	152	5		0.580 - 0.640 ***0.607***	0.029 - 0.033 ***0.031***	0.51 - 0.51 ***0.51***	33.6 - 33.9 ***33.7***
							7,14,28	152	5		0.450 - 0.640 ***0.541***	0.013 - 0.036 ***0.027***	0.50 - 0.51 ***0.50***	33.1 - 33.9 ***33.4***
	Precious Metal Tailings 2 75% Tailings 25% Sand	N/A	N/A	N/A	N/A	1907 - 1938 ***1921***	7	152	5		0.490 - 0.490 ***0.490***	0.020 - 0.060 ***0.036***	0.45 - 0.46 ***0.46***	31.1 - 31.6 ***31.4***
							14	152	5	N/A	0.540 - 0.640 ***0.580***	0.037 - 0.068 ***0.053***	0.45 - 0.46 ***0.46***	30.8 - 31.7 ***31.3***
	N = 9						28	152	5		0.620 - 0.720 ***0.667***	0.028 - 0.039 ***0.034***	0.45 - 0.47 ***0.46***	31.2 - 31.9 ***31.5***
							7,14,28	152	5		0.490 - 0.720 ***0.579***	0.020 - 0.068 ***0.041***	0.45 - 0.47 ***0.46***	30.8 - 31.9 ***31.4***
	Precious Metal Tailings 2 60% Tailings 40% Sand	N/A	N/A	N/A	N/A	1948 - 2002 ***1990***	7	152	5		0.580 - 0.620 ***0.600***	0.024 - 0.033 ***0.029***	0.40 - 0.41 ***0.40***	28.5 - 28.9 ***28.7***
							14	152	5	N/A	0.560 - 0.620 ***0.600***	0.060 - 0.076 ***0.066***	0.40 - 0.44 ***0.41***	28.6 - 30.4 ***29.3***
	N = 9						28	152	5		0.720 - 0.850 ***0.793***	0.032 - 0.071 ***0.047***	0.40 - 0.41 ***0.40***	28.5 - 29.0 ***28.7***
							7,14,28	152	5		0.560 - 0.850 ***0.664***	0.024 - 0.076 ***0.047***	0.40 - 0.44 ***0.41***	28.5 - 30.4 ***28.9***
	Precious Metal Tailings 2 45% Tailings 55% Sand	N/A	N/A	N/A	N/A	2016 - 2071 ***2045***	7	152	5		0.660 - 0.700 ***0.680***	0.020 - 0.060 ***0.036***	0.35 - 0.38 ***0.37***	26.0 - 27.4 ***26.7***
							14	152	5	N/A	0.680 - 0.720 ***0.693***	0.038 - 0.068 ***0.053***	0.37 - 0.38 ***0.37***	26.9 - 27.4 ***27.1***
	N = 9						28	152	5		0.810 - 0.910 ***0.877***	0.039 - 0.054 ***0.046***	0.36 - 0.39 ***0.37***	26.3 - 28.0 ***27.1***
							7,14,28	152	5		0.660 - 0.910 ***0.750***	0.020 - 0.068 ***0.045***	0.35 - 0.39 ***0.37***	26.0 - 28.0 ***27.0***

Table 4.1 Summary of mechanical properties test results for tailings and sand samples continued.

Fill Type	Fill Source	Specific Gravity (Physical Properties)	C_u (Physical Properties)	C_c (Physical Properties)	% - 20m (Physical Properties)	Bulk Density, ρ (kg/m³) (Physical Properties)	Curing Period (Days)	Sample Diameter (mm)	Total Binder Content (%)	Moisture Content at Testing (%)	Compressive Strength, σ_c (MPa) (Mechanical Properties)	Deformation Modulus, E (GPa) (Mechanical Properties)	Void Ratio, e (Engineering Properties)	Porosity, η (%) (Engineering Properties)
Composite Fill **(Tailings / Sand)**	Precious Metal Tailings 3 25% Tailings 75% Sand	N/A	N/A	N/A	N/A	2064 - 2213 ***2127***	7	76	6	5.9 - 7.4 ***6.4***	1.075 - 1.140 ***1.097***	0.181 - 2.809 ***1.532***	0.61 - 0.62 ***0.61***	37.9 - 38.3 ***38.0***
							14	76	6	5.4 - 5.9 ***5.7***	0.811 - 1.447 ***1.082***	0.155 - 0.506 ***0.294***	0.61 - 0.63 ***0.62***	37.9 - 38.7 ***38.3***
							28	76	6	4.0 - 5.1 ***4.6***	1.557 - 1.776 ***1.652***	0.115 - 2.565 ***1.305***	0.64 - 0.66 ***0.65***	39.0 - 39.8 ***39.3***
							7	102	6	6.3 - 7.8 ***7.1***	0.764 - 0.789 ***0.777***	0.414 - 0.993 ***0.704***	0.61 - 0.61 ***0.61***	37.9 - 37.9 ***37.9***
							14	102	6	6.9 - 7.5 ***7.2***	1.295 - 1.369 ***1.340***	0.120 - 3.119 ***1.149***	0.62 - 0.64 ***0.63***	38.3 - 39.0 ***38.6***
	N = 23						28	102	6	5.1 - 5.4 ***5.3***	1.640 - 1.874 ***1.718***	0.080 - 0.390 ***0.200***	0.62 - 0.66 ***0.64***	38.3 - 39.8 ***38.9***
							7	152	6	8.2 - 8.8 ***8.5***	1.615 - 1.632 ***1.624***	0.116 - 0.136 ***0.126***	0.55 - 0.56 ***0.56***	35.5 - 35.9 ***35.7***
							14	152	6	8.0 - 8.2 ***8.1***	1.738 - 1.886 ***1.812***	0.138 - 0.175 ***0.157***	0.56 - 0.57 ***0.57***	35.9 - 36.3 ***36.1***
							28	152	6	7.2 - 7.2 ***7.2***	3.059 - 3.081 ***3.070***	0.175 - 0.363 ***0.269***	0.59 - 0.59 ***0.59***	37.1 -37.1 ***37.1***
							7,14,28	76,102 152	6	4.0 - 8.8 ***6.5***	0.764 - 3.081 ***1.532***	0.080 - 3.119 ***0.693***	0.55 - 0.66 ***0.61***	35.5 - 39.8 ***38.0***
Total **N = 53**	Precious Metal Tailings 3 50% Tailings 50% Sand	N/A	N/A	N/A	N/A	2130 - 3059 ***2252***	7	76	6	13.5 - 14.3 ***13.8***	0.285 - 0.768 ***0.585***	0.014 - 0.270 ***0.108***	0.53 - 0.53 ***0.53***	34.5 - 34.7 ***34.6***
							14	76	6	10.3 - 11.6 ***11.0***	0.482- 0.658 ***0.592***	0.066 - 0.115 ***0.086***	0.56 - 0.61 ***0.58***	36.0 - 37.9 ***36.6***
							28	76	6	8.6 - 9.5 ***9.0***	0.482 - 0.702 ***0.586***	0.042 - 0.261 ***0.103***	0.59 - 0.60 ***0.60***	37.2 - 37.6 ***37.3***
							7	102	6	14.0 - 14.7 ***14.4***	0.641 - 0.851 ***0.740***	0.057 - 0.064 ***0.062***	0.54 - 0.55 ***0.54***	34.9 - 35.3 ***35.0***
							14	102	6	10.6 - 14.6 ***12.9***	0.419 - 1.763 ***1.282***	0.161 - 0.358 ***0.244***	0.56 - 0.60 ***0.58***	35.8 - 37.6 ***36.6***
							28	102	6	8.5 - 13.7 ***11.5***	0.493 - 2.306 ***1.130***	0.061 - 0.604 ***0.201***	0.55 - 0.60 ***0.57***	35.3 - 37.4 ***36.3***
	N = 30						7	152	6	14.7 - 15.0 ***14.9***	0.835 - 0.934 ***0.885***	0.119 - 0.153 ***0.136***	0.53 - 0.55 ***0.54***	34.4 - 35.3 ***34.9***
							14	152	6	12.9 - 13.6 ***13.2***	1.363 - 1.418 ***1.391***	0.096 - 0.168 ***0.132***	0.51 - 0.53 ***0.52***	33.8 - 34.6 ***34.2***
							28	152	6	11.4 - 11.6 ***11.5***	1.740 - 1.980 ***1.860***	2.030 - 2.846 ***2.438***	0.54 - 0.57 ***0.56***	35.1 - 36.4 ***35.8***
							7	279	6	12.1 ***12.1***	0.538 ***0.538***	0.06 ***0.060***	0.12 ***0.12***	10.8 ***10.8***
							14	279	6	N/A	0.652 ***0.652***	0.031 ***0.031***	0.4 ***0.40***	28.5 ***28.5***
							28	279	6	11.8 ***11.8***	1.109 ***1.109***	0.091 ***0.091***	0.28 ***0.28***	21.7 ***21.7***
							7,14,28	76,102 152	6	8.5 - 15.0 ***12.2***	0.285 - 2.306 ***0.936***	0.014 - 2.846 ***0.284***	0.12 - 0.61 ***0.53***	10.8 - 37.9 ***34.3***

4.4.2 Effects of Particle Size Gradation on Mechanical Properties of Tailings and Sand Paste Samples

Two methods were used to investigate the effects of size gradation on the mechanical properties and behaviour of the tailings/sand paste fill sample. The first method of investigation involved a direct comparison of the properties of a coarse base metal tailings against those of a fine precious metal total tailings samples.

The second approach involved the alteration of the particle size distribution of the"as-received" fine tailings material through sand addition. These were carried out in three parts in an attempt to identify the most favourable sand addition range for the composite tailings/sand paste fill samples.

4.4.2.1 Method 1: Comparison of Properties of Base Metal and Precious Metal Tailings Pastefill Samples

The effects of particle size gradation on the mechanical properties was evaluated by comparing the properties of two high-density tailings paste fills. The selected materials were Base Metal Tailings (BMT) #3 and Precious Metals Tailings (PMT) #2. The two tailings had different particle size gradations. The test specimens were cast with the same binder composition and cured under similar environmental conditions for 7 and 28 days. The test samples were cast at 76% solids density by weight. The binder composition for this test condition was fixed at 5% per dry weight of solids. The test results are summarized in Table 4.6 and are presented in Figure 4.9.

Observations

The base metal tailings paste fill specimens developed higher compressive strengths at both 7 and 28 days of curing than those of the precious metal tailings. The differences in compressive strength values can be attributed to differences in size gradation of two tailings materials. The observation supports finding by Espley et al., (1970) who concluded that backfill strength is influenced by particle size gradation. On the other hand, Thomas (1981) and Boltd et al., (1993) have determined that backfill samples containing higher amounts of fine tailings developed higher compressive strengths than those containing relatively coarser material. Based on that conclusion, the finer tailings (PMT#2) should have developed higher compressive strength than the coarser tailings (BMT#3). The differences are due to the fact that the BMT#3 samples constituted in relative terms, a better graded material (Cc = 1.74, Cu = 7.4 %-20μm = 18.3%) than that of the PMT#2 samples (Cc = 0.63, Cu 7.5, %-20μm = 45%). Lidkea and Landriault (1993) have suggested that a wider distribution of material sizes is required for strength development in blended tailings/sand pastefill samples.

Table 4.5 Differences in mechanical property test results due to particle size gradation of paste fill specimens

Binder Composition	**Curing Period (Days)**	**Bulk Density (kg/m^3)**	**Unconfined Compressive Strength (MPa)**	**Deformation Modulus (MPa)**	**Void Ratio**	**Porosity (%)**
Precious Metal Tailings (#2)						
5%	7	1909 1903 1907 **1904.7**	0.132 0.200 0.120 **0150.7**	12.0 71.0 14.0 **32**	0.44 0.44 0.44 **0.44**	30.3 30.5 30.4 **30.4**
5%	28	1911 1889 1872 **1890.7**	0.260 0.260 0.250 **0.257**	27.0 84.0 45.0 **52**	0.43 0.45 0.46 **0.45**	30.3 31.1 31.7 **31.0**
Base Metal Tailings (#3)						
5% PC	7 7 7	1976 1964 2037 **1992.3**	0.419 0.370 0.395 **0.395**	38.2 38.5 35.4 **37.3**	0.49 0.50 0.44 **0.48**	32.8 33.2 30.7 **32.2**
5% PC	28 28 28	1964 1960 2007 **1977.0**	0.912 0.888 0.937 **0.912**	60.0 77.8 75.9 **71.21**	0.50 0.50 0.47 **0.50**	33.2 33.3 31.7 **32.8**

(a)

(b)

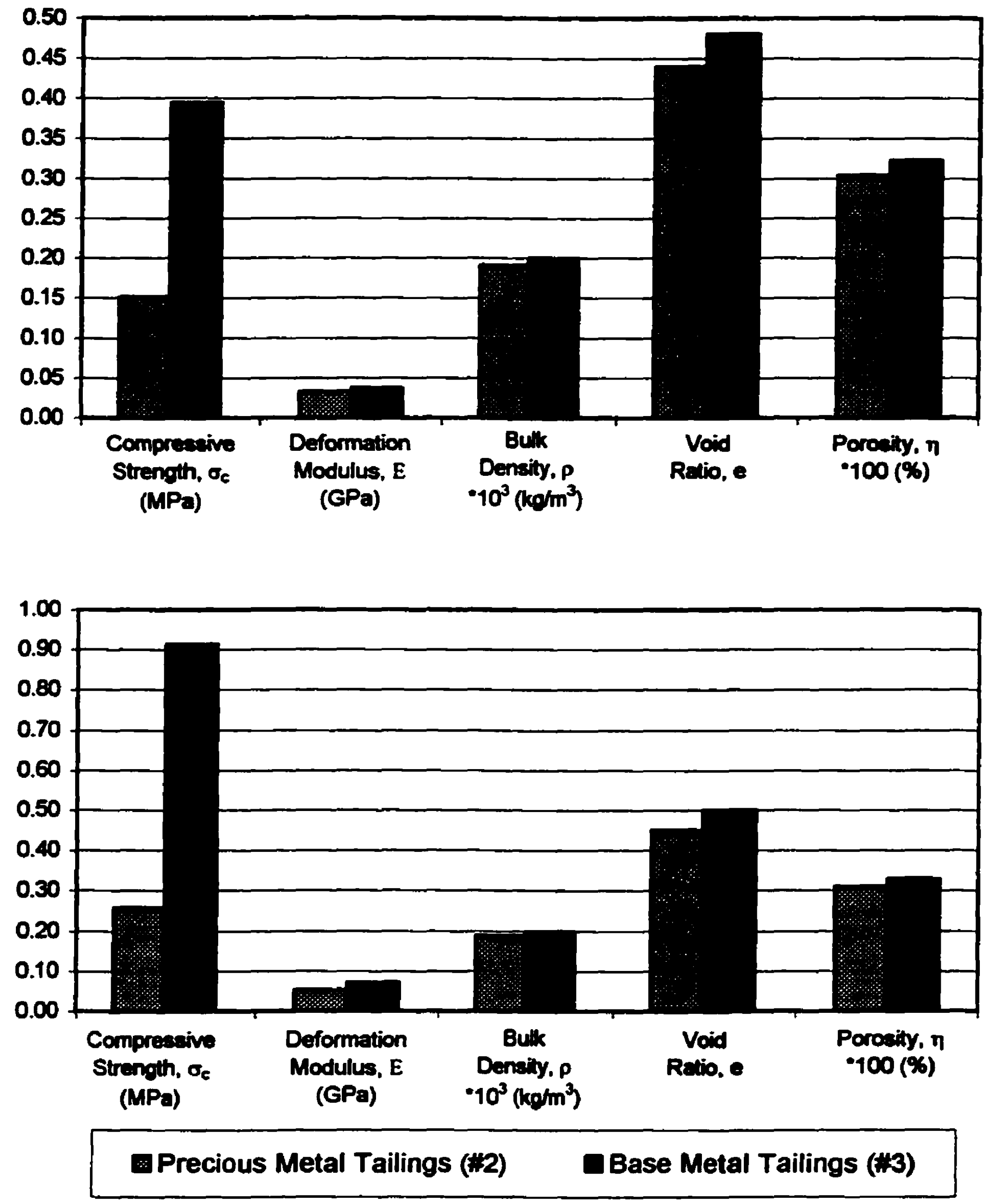


Figure 4.9 (a) & (b) **Differences in mechanical properties test results due to particle size gradation of pastefill specimens for precious metal tailings pastefill (#2) and base metal tailings pastefill (#3) at (a) 7 days curing, (b) 28 days curing (5% binder)**

4.4.2.2 Method 2: Optimization of Particle Size of Tailings Using Sand

Three test conditions involving the addition of sand to the tailings material were investigated in order to determine the suitable condition that would be more conducive to producing strength gain in the composite tailings/sand paste fill samples.

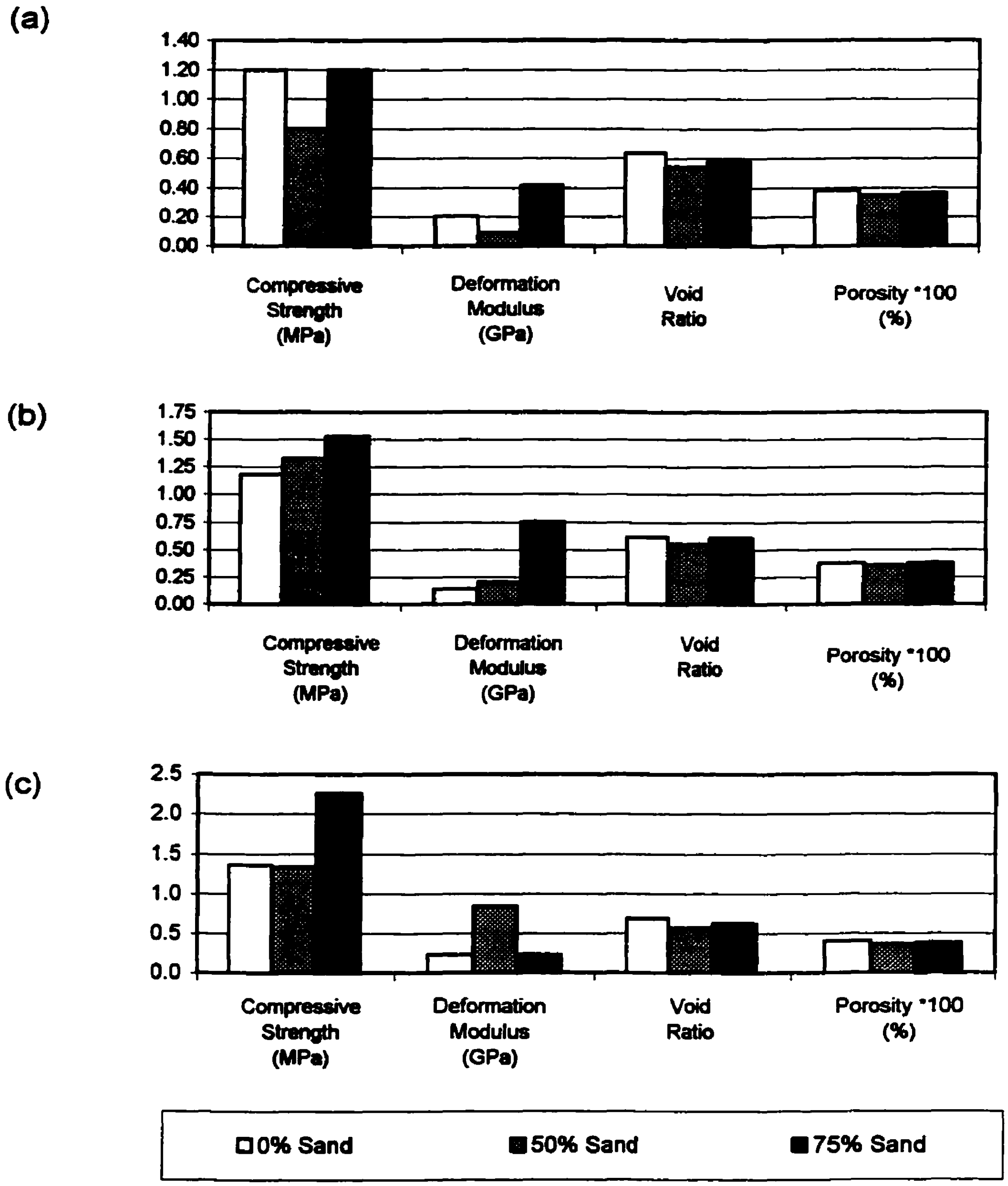
Trial A

The first condition involved the blending of a full plant precious metal tailings (PMT#3) with Sand #2. The following constitutes particle size gradation parameters for the PMT #3 samples were: Cu = 3.8; Cc = 0.92; %-20μm =74.8wt%. Similarly, for the sand, Cu = 6.5; Cc = 1.45; %-20μm = 13.8 wt%. Sand addition rates ranged between 0 and 75 percent by weight. The resulting unconfined compressive strength, deformation modulus, porosity and void ratio of the composite material are presented in terms of curing period in Figures 4.10 and 4.11. The strength gain for the blended tailings/sand composite mixture was erratic in the short terms (7 days), but increased with increasing curing period. The void ratio and porosity values remained very uniform at each curing period however, the general trend seemed to be a reduction in porosity with curing period.

Trial B

The second test condition involved the blending of full plant precious metal tailings (PMT#2) with Sand #2. The particle size gradation parameters for PMT #2 are: Cu = 7.5; Cc = 0.63 and %-20μm = 40.5. The sand addition percentages to the tailings material were 0%, 10%, 25%, 40% and 55%. The revised sand addition rates were made in proportion to the ultra-fine material content of the PMT#2 tailings. OPC was used as the binder; a cement content of 5% instead of 6% was used based on previously calibrated data. The resulting strength and deformation parameters are presented in Figures 4.12 and 4.13

The results show an early increase in compressive strength of the composite product with sand addition. The results also indicate a progressive increases in the compressive strength and deformation modulus values as well as, reductions in void ratio and porosity with increasing sand content and curing period.



4.10 (a),(b)&(c) Effects of sand addition on unconfined compressive strength, modulus of deformation, void ratio and porosity for total tailings fill (PMT#3) at (a) 7 days curing, (b) 14 days curing and (c) 28 days curing (6% binder)

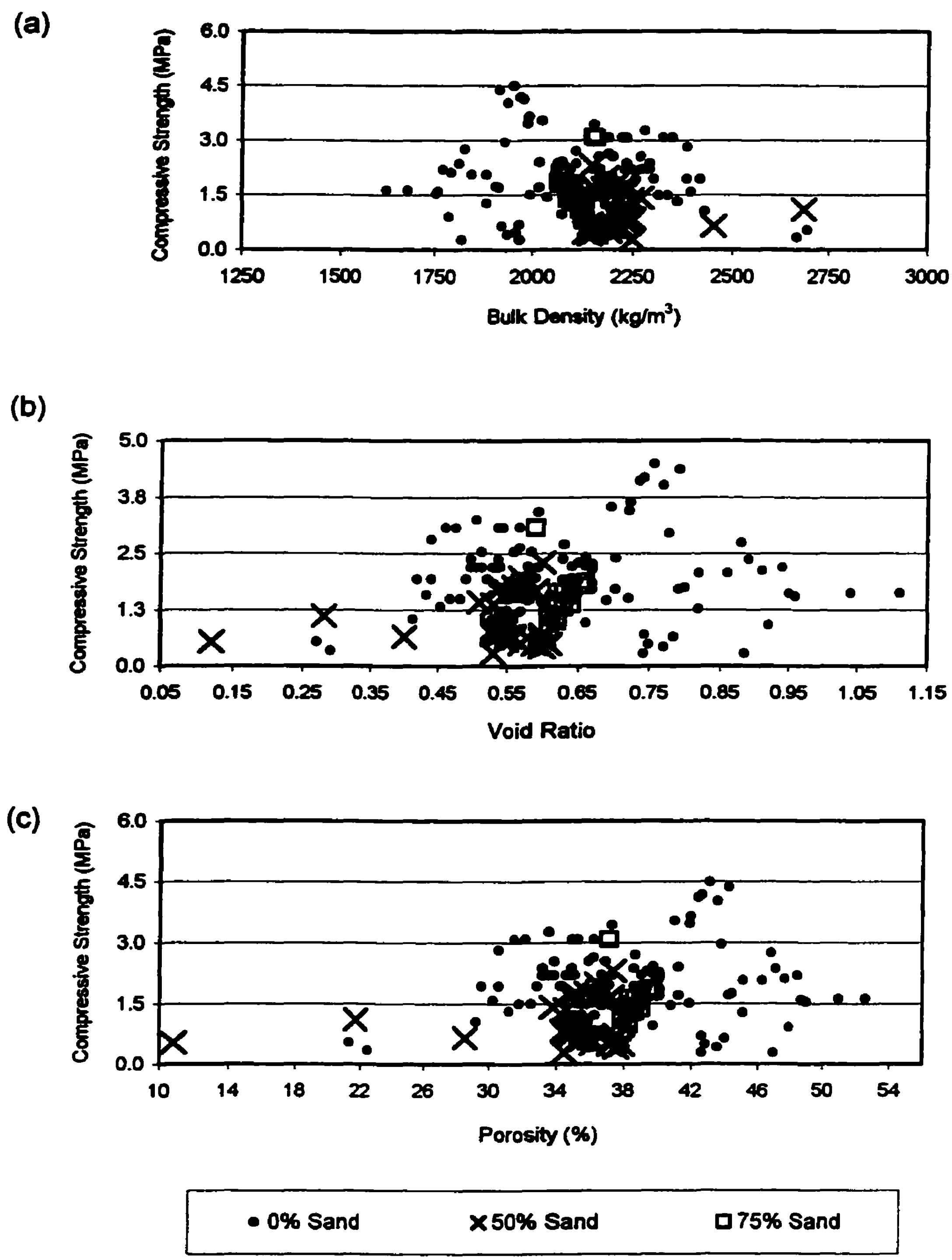


Figure 4.11 (a),(b)&(c) Relationship between compressive strength and (a) bulk density, (b) void ratio, and (c) porosity for tailings (PMT#3) material (6% binder)

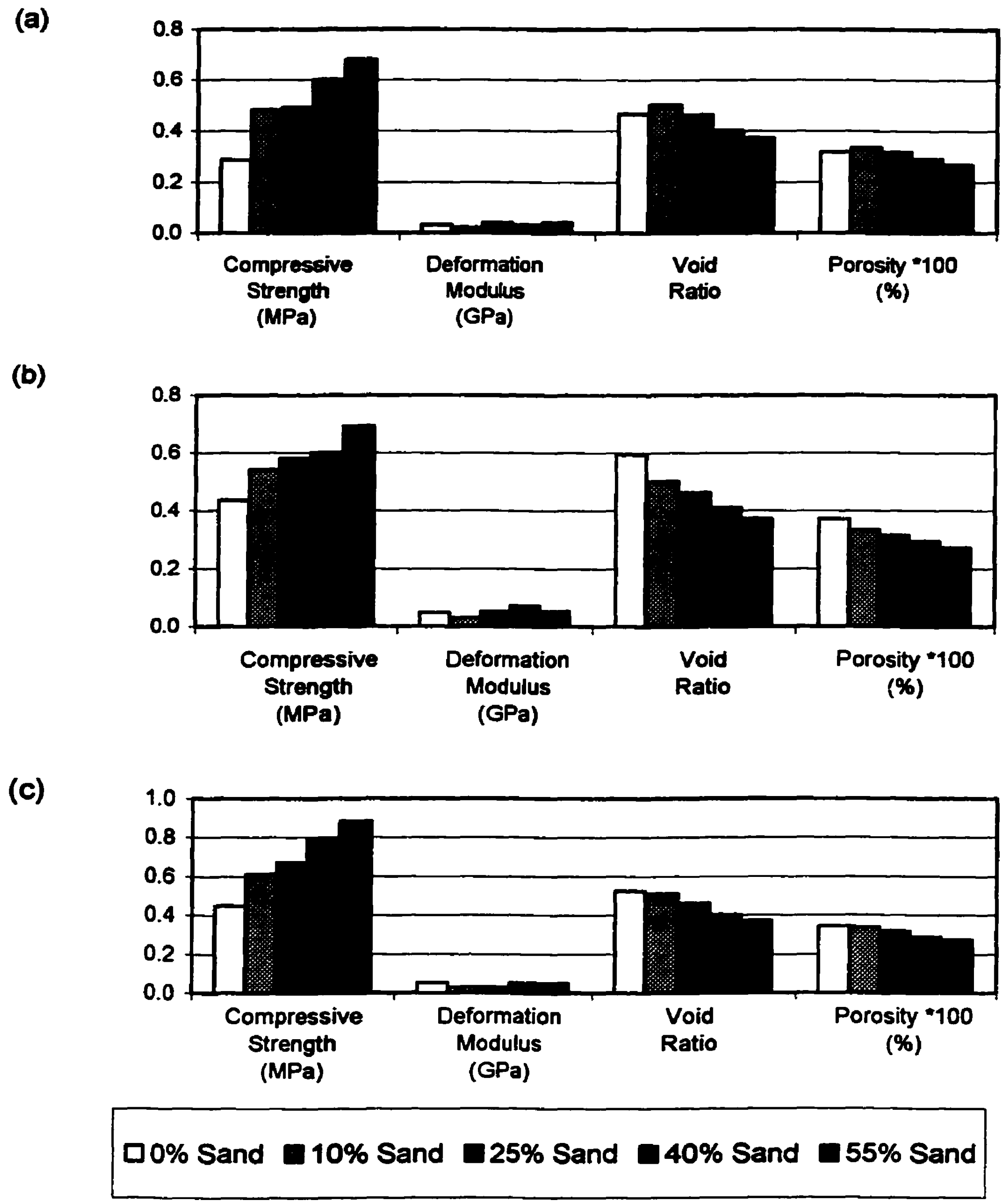


Figure 4.12 (a),(b)&(c) Effects of sand addition on unconfined compressive strength, modulus of deformation, void ratio and porosity for total tailings fill (PMT#2) at (a) 7 days curing, (b) 14 days curing and (c) 28 days curing (5% binder)

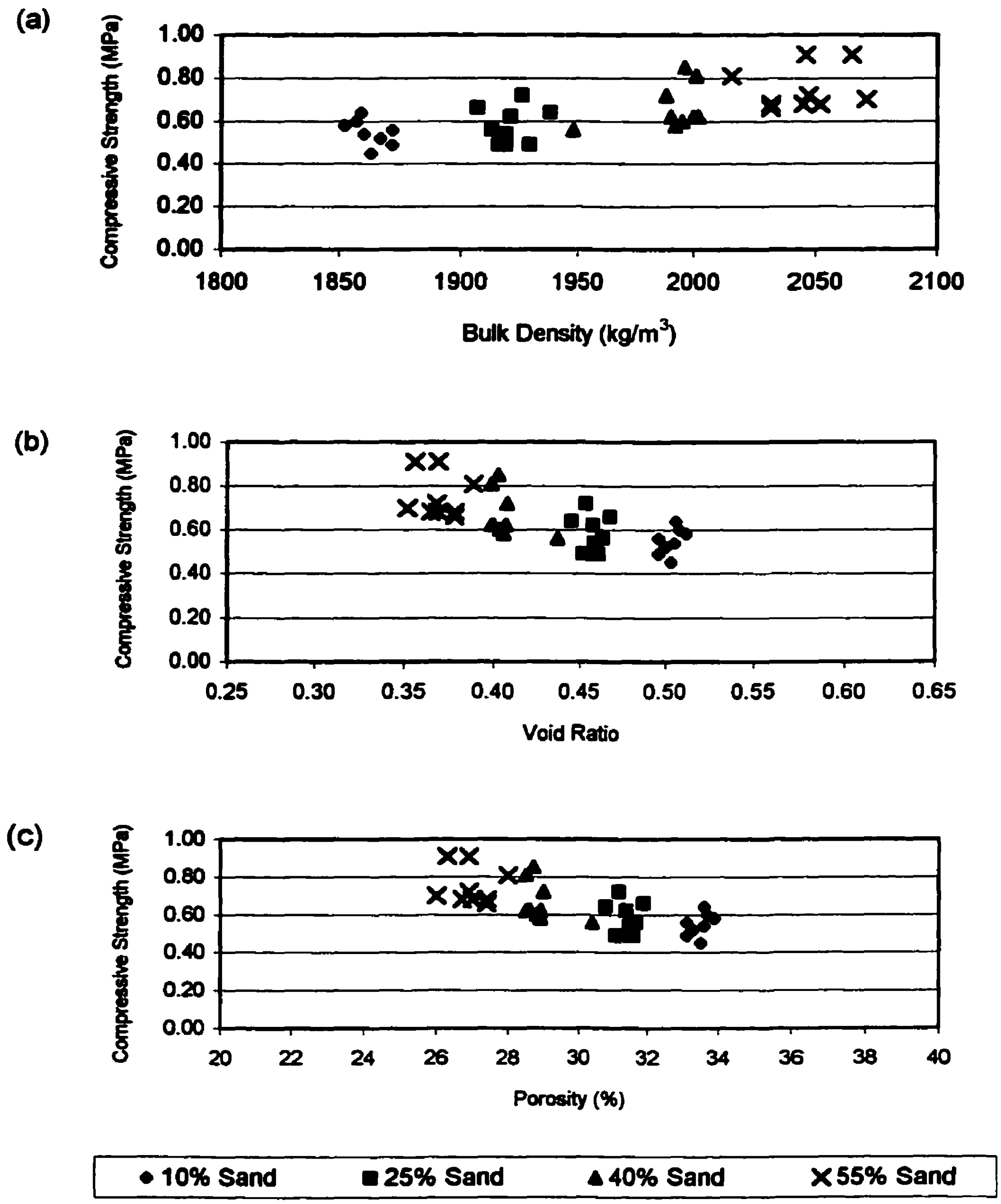


Figure 4.13 (a),(b)&(c) Relationship between compressive strength and (a) bulk density, (b) void ratio and (c) porosity for tailings (PMT#2) material (5% binder)

Trial C

A third test condition involved the blending of PMT#2 tailings with Sand #1 to form a composite product. the particle size gradation parameters for sand #1 are as follows: Cu = 4.2; Cc = 1.15 and %-20μm content = 2.65. The results of this test condition are presented in Figure 4.14. Both Wingrove (1993) and Stone (1993) have suggested that increase in specific density of a fill material can also result in increased stiffness of the product. The test results suggest that the most favourable sand addition level for maximum strength gain for the PMT #2 and Sand #1 composite product is a sand composition of approximately 23%.

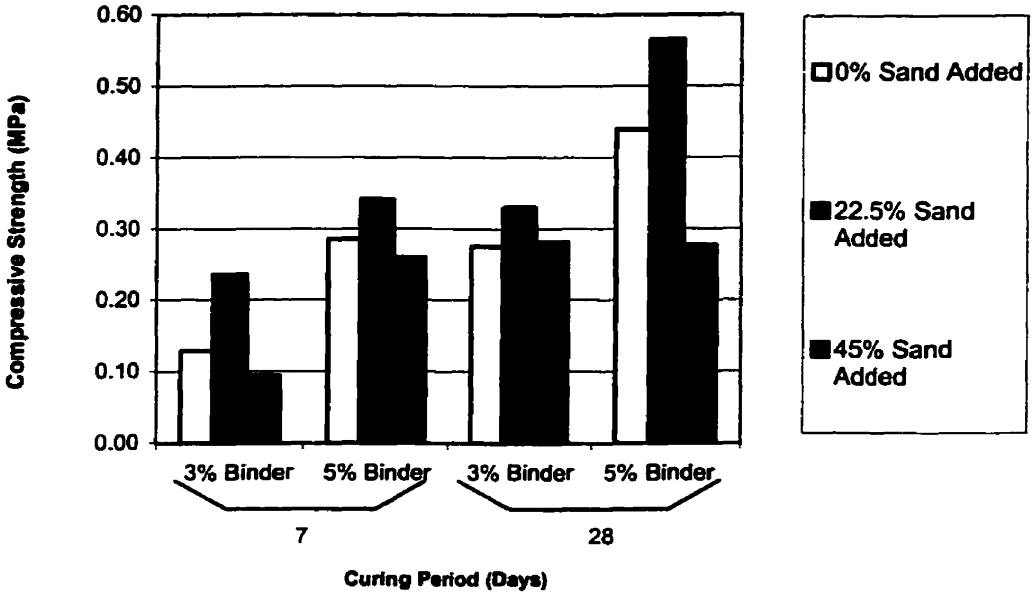
4.4.2.3 Summary of Sand Addition Test Results

The results of the composite tailings and sand trials (Section 4.4.2.2) suggest that, the addition of sand to full plant tailings material as a means of improving particle size gradation, may not always result in a short term strength gain for the composite product. The differences in the results of the sand addition test suggest that the formation of composite blended tailings/sand paste fill does not readily result in an effective strength gain. The properties of the composite product depended largely on the properties of the constituent materials. These finding are supported by previous studies by Boldt et al., (1993) who concluded that sand addition to fine tailings did not readily result in strength gain of the composite product.

A determinant for the effectiveness of sand addition seems to be changes in either bulk density, porosity or void ratio with increasing sand content. Progressively increasing bulk density values or alternatively, decreasing porosity or void ratio values with increasing sand content, could suggest an early strength gain. In the long term however, sand addition to fine total tailings could ultimately result in strength gain at higher sand contents.

It is suggested from the above observations that changes in bulk density, void ratio or porosity with sand content may be used as an indicator for early strength gain for cemented composite tailings/sand fill specimens.

(a)



(b)

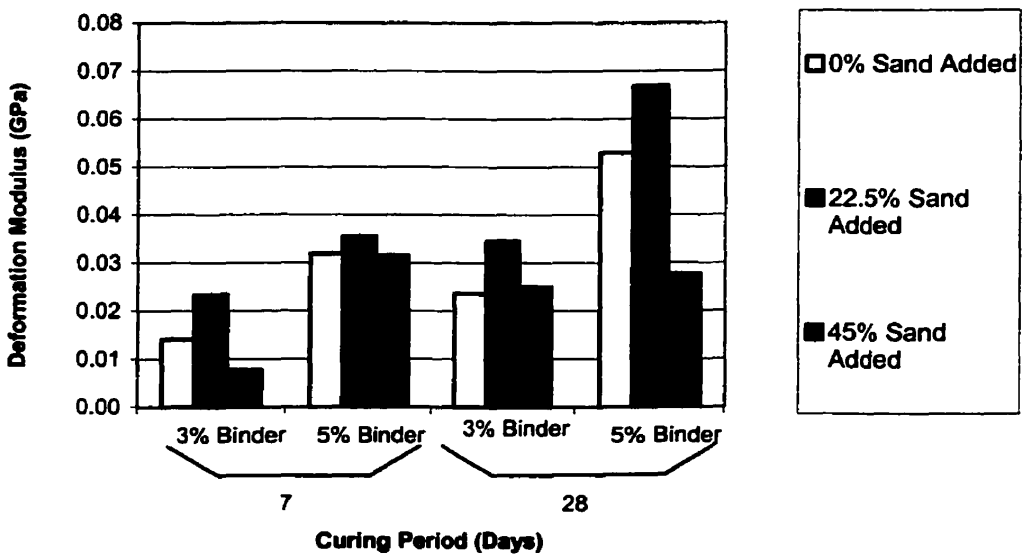


Figure 4.14 (a)&(b) Effects of sand addition on compressive strength and deformation modulus for precious metal tailings paste fill

4.4.2.4 Effects of Binder Type and Composition and Curing Period on Mechanical Properties of High-Density Tailings and Sand Paste Fill Specimens

This element of the study was carried out to determine the effects of different types of binders and their compositions on the mechanical properties of total tailings high-density fill specimens. The primary binder for the study was Ordinary Portland Cement (OPC). Fly Ash and Slag cement were also used in combination with OPC and various amounts of supplementary binders to stabilize the fill samples. Other supplementary binders included Silica Fume, Anhydrite and metallurgical by product which is referred to in this study as PC/X were also used to stabilize the test samples. A special blend of primary and supplementary binders which is designated as "Product A" was produced specifically for this study, and was tested as a potential cementing agent for stabilizing the precious metal total tailings paste fill samples. A blend of OPC and other supplementary binders were also used to stabilize base metal tailings paste fill samples. The curing period for the test samples ranged between 14 and 56 days because of the slow reaction rates of most supplementary binders compared to OPC (Malhotra, 1987).

The test results are presented in (Figures 4.15 & 4.16). The results show that all the alternative binders used in this study, produced unconfined compressive strength values that were similar to, or higher than those obtained using OPC alone. The results suggest that all the investigated products could be used as effective binder alternatives for stabilizing total tailings paste fills.

It is interesting to note that the straight total tailings paste fill samples containing 4% Product "A", and also 8% OPC (Figure 4.15 (a)), developed compressive strengths that were similar to the "blended" (60%) tailings (40%) sand paste fill samples containing 5% OPC at 14 days of curing (Figure 4.12 (b)). Similar to concrete (Neville, 1987), the strength and deformation properties of the stabilized tailings/sand paste fill samples improved with time.

The test results indicate that several alternative binders are available for consolidating backfills. These alternative binders can be prepared from the blending of Ordinary Portland cement with other supplementary binders. The site specific characteristics (and possible variations in tailings material properties, including geochemistry of the ore and milling), would suggest that stabilization trials should be carried out at individual mines site, in order to arrive at suitable binder combinations.

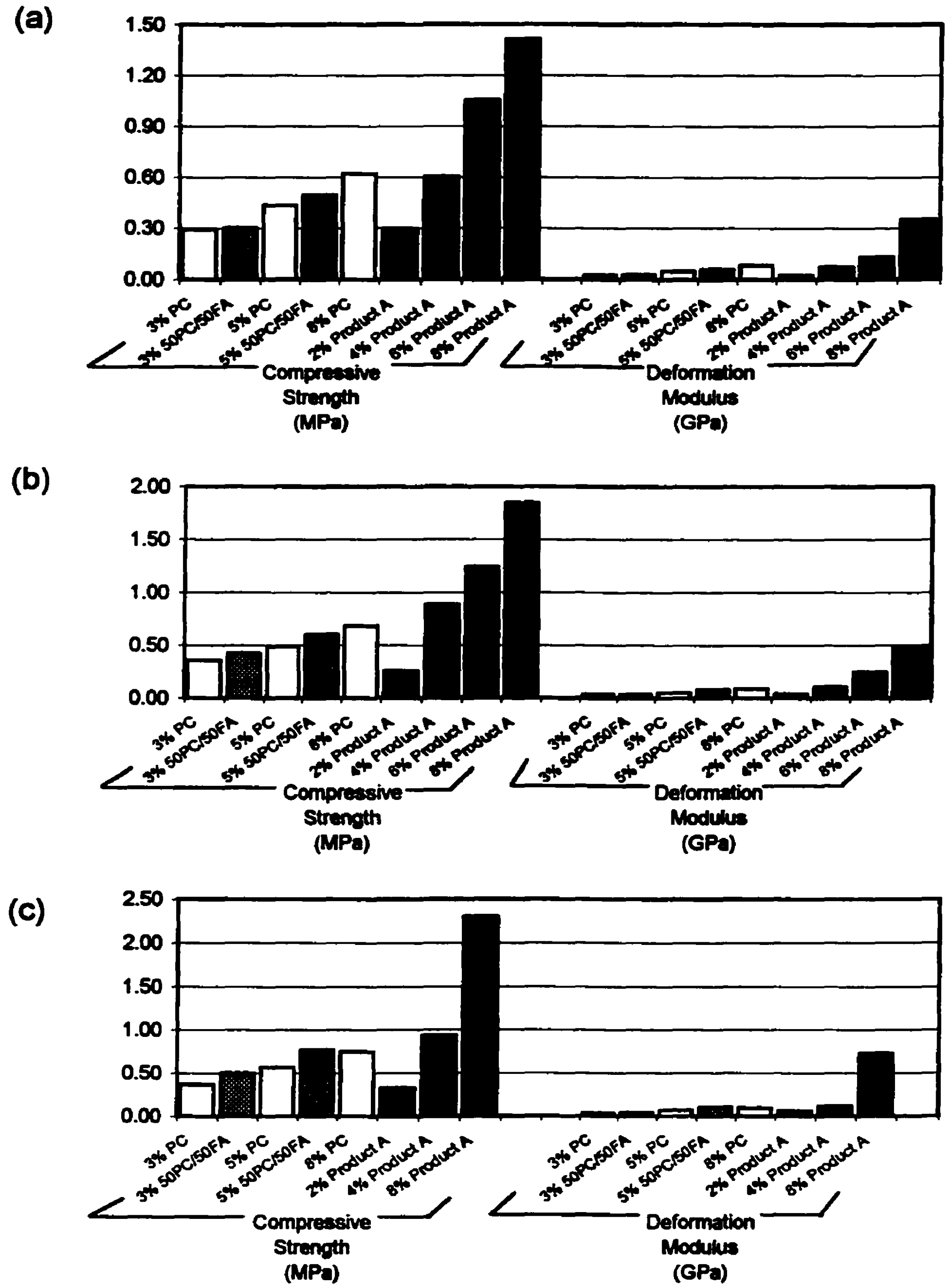


Figure 4.15 (a),(b)&(c) Effects of alternative binder on compressive strength and deformation modulus for precious metal total tailings at (a) 14 days curing, (b) 28 days curing and (c) 56 days curing

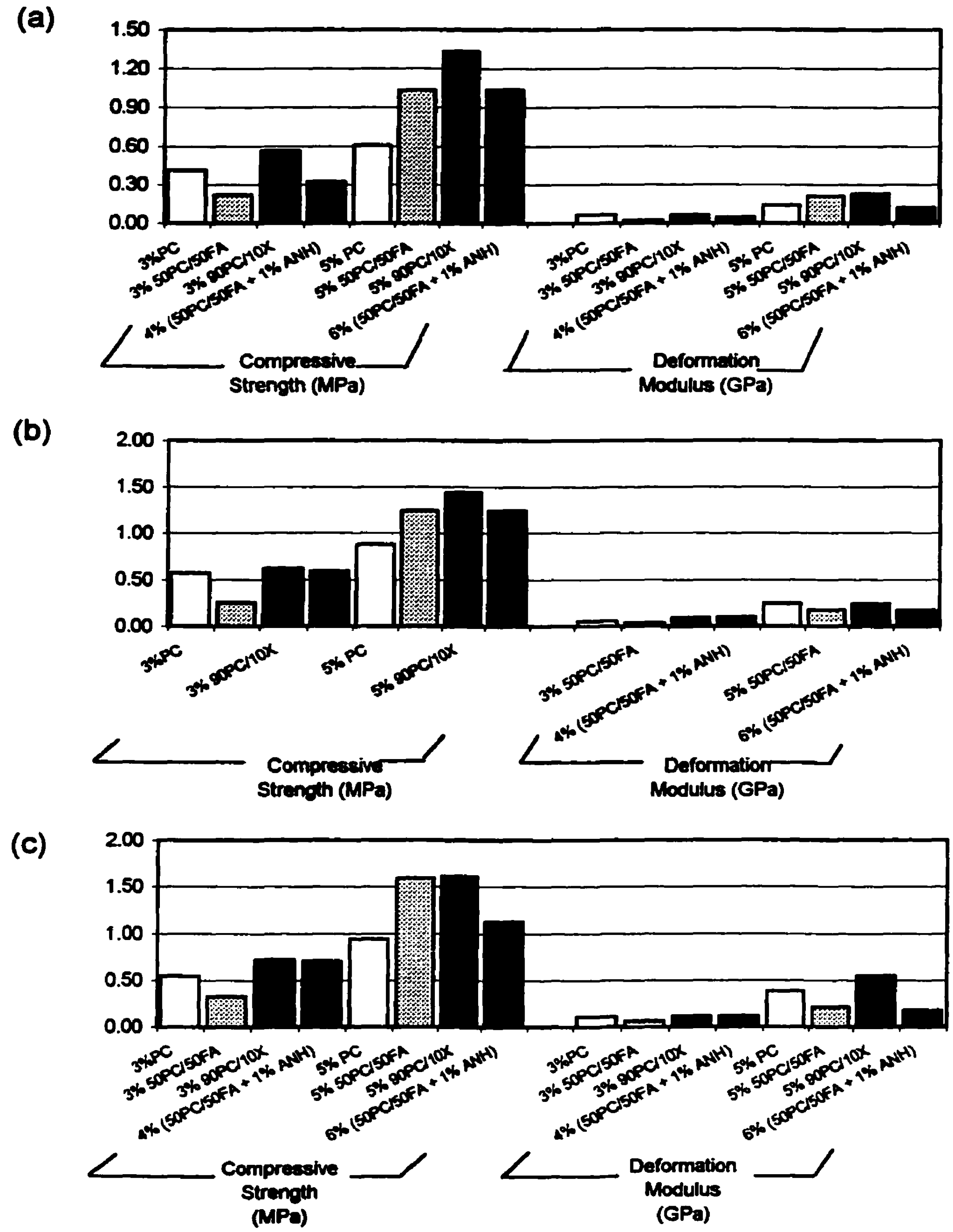


Figure 4.16 (a),(b)&(c) Effects of alternative binder on compressive strength and deformation modulus for base metal total tailings at (a) 14 days curing, (b) 28 days curing and (c) 56 days curing

4.4.2.5 Effects of Curing Period and Environment on the Mechanical Properties of High-Density Fill Systems

As mines go deeper, the ambient temperature of the wall rock and the mine environment can be expected to increase beyond the optimum condition required for effective hydration of the cementing agents. "False setting" (Neville, 1987) can result from rapid curing of concrete materials and stabilized fill products at higher temperatures. False setting often lead to poor strength development in cement stabilized fills (Thomas et al., 1979).

The objective of this element of the investigation was to assess the effects of non-ideal curing conditions on mechanical properties of stabilized high-density tailings paste fills. The information could be useful toward the understanding of the behaviour of high-density backfill systems including composite fills, when mining at greater depths. Two sets of test specimens were prepared for unconfined compressive strength testing. The first set of specimens were cured in a fog room in accordance with ASTM specifications for concrete specimens. The second set of specimens were exposed to the ambient laboratory conditions of 30°C and 18% Relative Humidity.

The test results are presented in Figure 4.17. The results of this study indicate that a curing temperature of 30°C did not affect strength development for the tests samples. This was probably due to the fact that the binder content of the stabilized fill specimens was lower than that normally used in concrete mixtures (Neville, 1987) and therefore, an ideal environment for cement hydration was not a major factor in strength development.

Thomas (1983) investigated fine medium and coarse tailings at curing temperatures of up to 40°C to simulate the anticipated mine stope temperature. Thomas found that, the absence of coarse material in the (75μm to 425μm) range in the fill did not affect strength development. Fine tailings showed the largest improvement at the lowest Portland cement content. A temperature of up to 40°C did not adversely affect strength development.

Thomas (1983) found that strength increased rapidly and then flattened out; he suggested that it showed the typical behviour of elevated temperature curing.

It is suggested however, that care should be exercised in curing cemented tailings pours in situ; the curing should be done in accordance with suggestions made by Thomas et al. (1979).

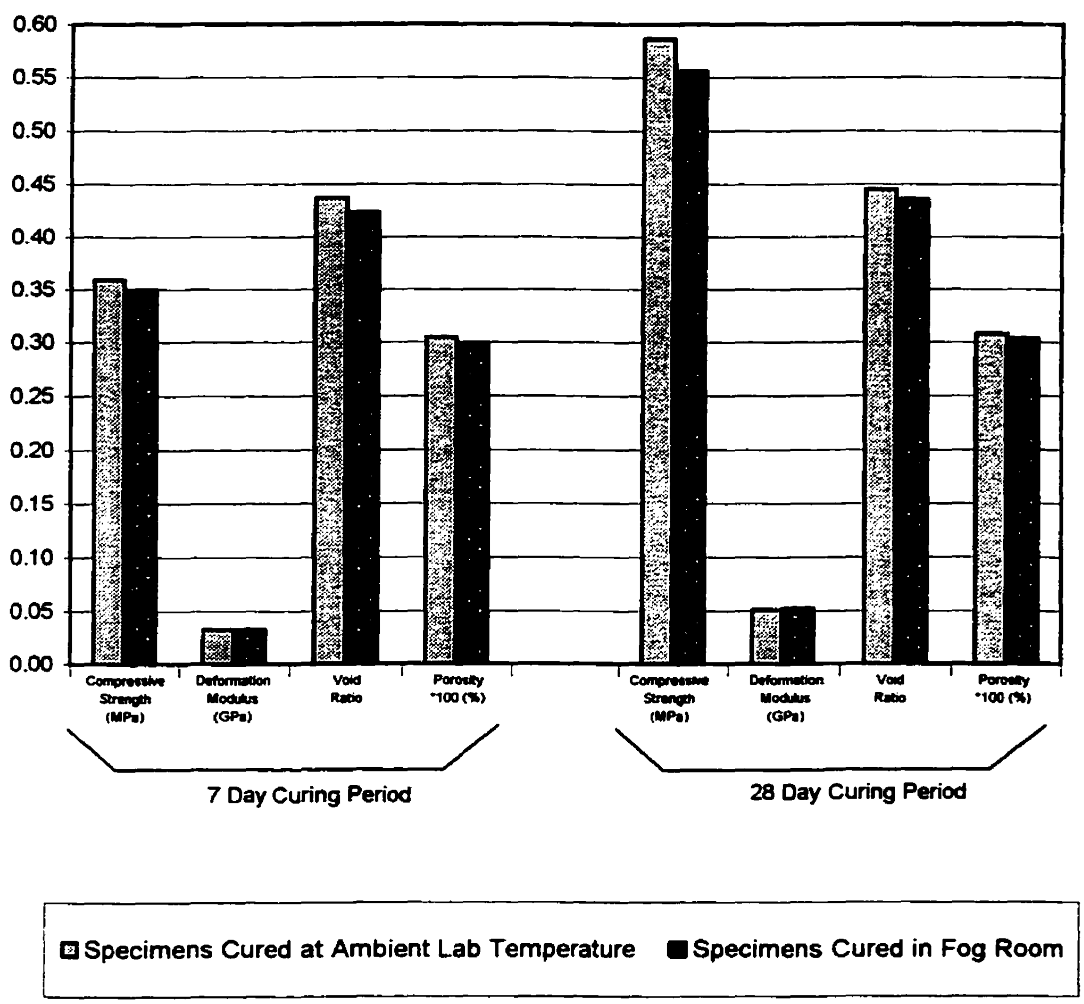


Figure 4.17 Effects of curing condition on mechanical properties of high-density fill specimens

4.4.2.6 Effects of Moisture Content on High-Density Paste Fill Samples

Strength and permeability have been identified (Thomas et al., 1979; Hassani and Bois, 1992) as the most important basic mechanical properties required for designing conventional hydraulic backfill systems. It has also been suggested that drainage is of critical importance for hydraulic fills (Thomas et al., 1979; Lerche and Renetzeder, 1984; Clark, 1988; Chen and Annor, 1995; Yu, 1989; Boldt et al., 1993, Ouellet et al., 1998 and Pierce et al., 1998) have indicated the importance of moisture content on the stability of placed paste backfill in situ.

The objective of this element of the investigation was to evaluate the effects of moisture content on the mechanical properties of the high-density fill samples. Specimens were collected and analyzed for moisture content determinations (ASTM D-2216), during the unconfined compressive strength tests. This was done to establish a relationship between the water/binder ratio for the stabilized paste backfill samples in this study. Also, to determine which of the three pastefill systems is the most efficient material in developing strength in terms of moisture binder ratio.

The test results for the various fill types in terms of water/binder (W/C) ratios are presented in Appendix B-2. The combined tests results is presented in Figure 4.18. The results show a clear dependence of compressive strength on water/binder ratio. The unconfined compressive strength decreased with increasing water-to-binder (W/C) ratio

The established relationships between the unconfined compressive strength (σ_c) and moisture/binder (W/C) ratio for the different types of paste fill samples in this study are as follows:

Tailings paste fill

$$\sigma_c(MPa) = 1.33(W/C)^{-0.69} \qquad 4.1$$

(for $0.5 \leq W/C$)

Sand paste fill

$$\sigma_c(MPa) = 1.10(W/C)^{-0.69} \qquad 4.2$$

(for 0.7 ≤ W/C)

Composite (Blended Tailings/Sand) Paste fill

$$\sigma_c(MPa) = 1.57(W/C)^{-1.04} \qquad 4.3$$

(for 0.9 ≤ W/C)

Based on Figure 4.18, moisture/binder (W/C) ratios of less than 3, produced higher compressive strength values for the composite paste fill samples. The strength development for (W/C) ratios greater than 3, was poor for the sand and tailings fill materials in this study.

The above equations are in general agreement with relationships found in the published literature between unconfined compressive strength and water/cement ratio for backfill materials. For example, Arioglu (1983) found the following relationship between compressive strength (σ_c) and water/cement (α) ratio for "aggregate" fill.

$$\sigma_c = A\alpha^{-n} \qquad 4.4$$

where A and n are experimental constants.

Petrolito et al. (1998) found the following relationship between unconfined compressive strength (U.C.S.) And Water/Calcined gypsum (W/CG) ratio:

$$U.C.S.(MPa) = 2.084(W/CG)^{-2.322} \qquad 4.5$$

Petrolito et al. (1989) reported that the range of (W/CG) ratios covered in the mix design was 0.83 to 3.0. This range was selected because the strength was found to be insignificant above (W/CG) ratios of 3.0. Also, the material was difficult to mix below (W/CG) ratios of

0.83. It was further suggested that, the equation was applicable to U.C.S. values of greater than 1.25 MPa.

The test results are also supported by previous findings involving hydraulic fill systems (Rawlings et al., 1966; Thomas et al., 1979; Boldt et al., 1993) and also, recent studies by Ouellet et al. (1998) and Pierce et al. (1998) regarding paste backfills. The results of this study indicate that moisture control is of critical importance for all types of backfill systems including high-density paste fills. The test results also indicate that the effects of moisture content on strength development was independent of the type of fill material, or binder composition.

Studies by Mitchell and Wong (1982); Lerche and Renetzeder (1984), Boldt et al., (1993), Chen and Annor (1995), Hedley (1995) have also indicated that moisture content and binder composition, rather than size gradation, seem to influence strength development in backfill materials.

The reported studies in the published literature, and the results of this study suggest that there is a need for moisture control in situ for all types of backfill systems including cemented paste fill. A maximum moisture/binder ratio of less than 3.0 seems to be the limiting value for maintaining strength gain in laboratory test samples. Ratios higher than 3 could result in a reduction in fill strength.

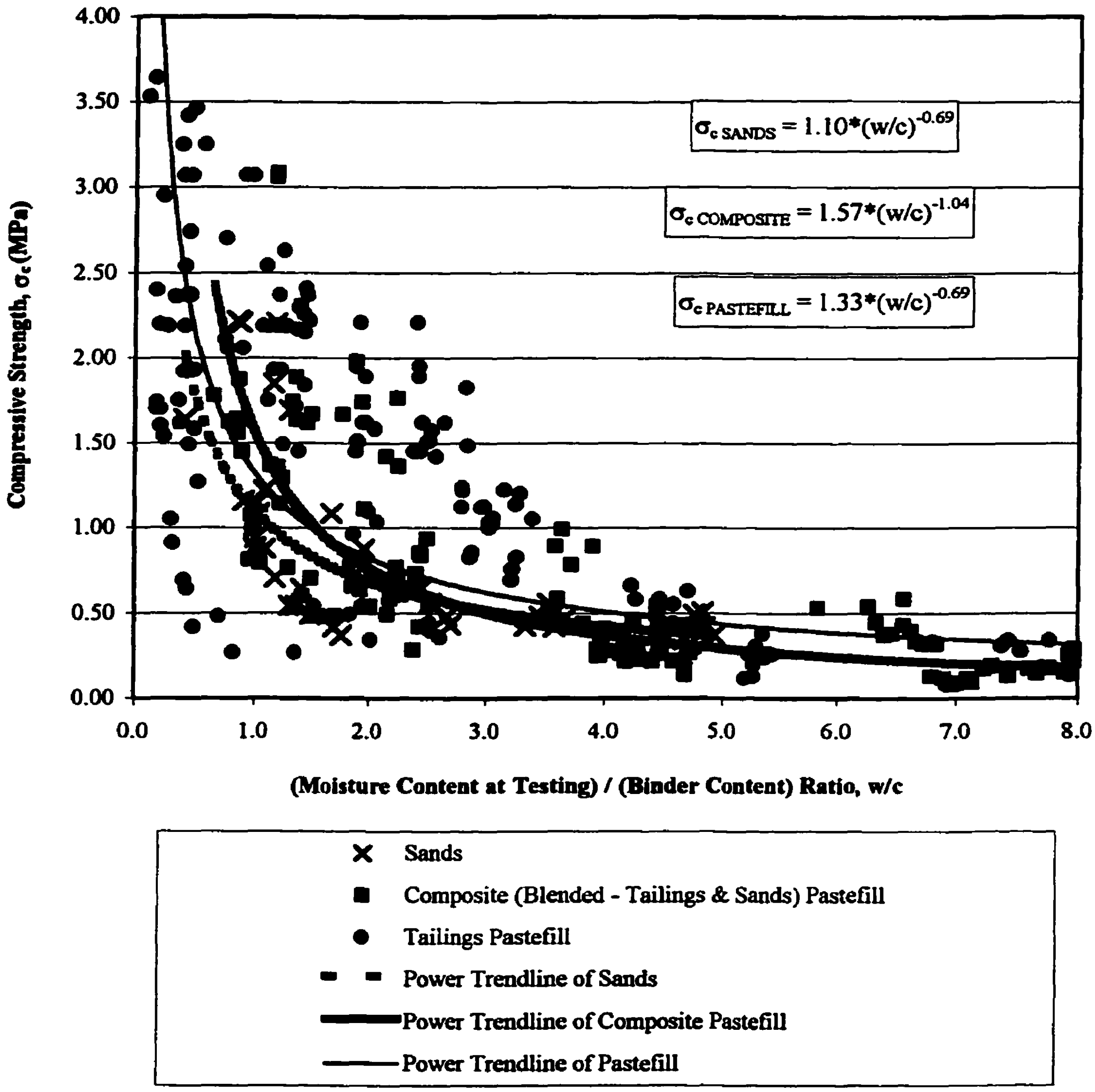


Figure 4.18 Relationship between compressive strength and moisture/binder (W/C) ratio for tailings and sand pastefill, and composite (blended-tailings) pastefill samples

4.4.2.7 Scale Effects on Mechanical Properties of High-Density Straight Tailings Pastefill and Composite Tailings/Sand Pastefill Specimens

There are problems with scaling up laboratory experimental results to in situ mechanical properties. The difficulties often lie in the selection of suitable sample sizes for the laboratory tests. In terms of concrete cylinders the recommendation (ASTM) is to use samples sizes that are at least three times greater than the size of the maximum aggregates with regard to the testing of rock core specimens, it is recommended (Brown, 1981) that the minimum specimen sizes be at least one tenth of the largest grain size of the rock . There is no known size specification or requirement for backfill samples.

The use of 38mm diameter backfill test specimens probably originated from soil mechanics testing. Bishop and Henkel (1962) refers to the use of 1-1/2 in (38mm) diameter triaxial test specimen as generally the accepted standard in Great Britain for testing soils containing no stones. The practice might have also been in existence, as a matter of convenience due to difficulties in obtaining undisturbed field samples for testing. There seems to be no justification for testing small size fill specimens for engineering analysis. The rationale for using small diameter test specimens for strength determinations is probably based on the assumption that similar to fine grained soils, backfill materials are not expected to show scale effects on mechanical properties.

The ultimate objective of this part of the investigation was to estimate the potential behaviour of composite fills in situ based on the scale effects found in high-density tailings paste fills. In this regard, tailings material (Precious Metal Tailings 3, Table 4.1) containing the highest proportion of ultra-fine material (74.8%) among all the materials investigated in this study was selected for the tests. The material was equivalently a clay soil in terms of soil mechanics classification (Terzaghi and Peck, 1967). Both cylindrical and cube specimens were prepared and tested in unconfined compression. The specimen dimensions ranged between 38mm and 279mm diameter for the test cylinders with L/D =2. Similarly, the cube specimens ranged from 38mm to 102mm side lengths.

A total of 138 specimens were tested. Binder composition consisted of 6% ordinary Portland cement. Specimen curing time ranged between 7 days and 28 days.

Stabilized samples of composite (blended tailings/sand) pastefill material were similarly prepared, cured and tested as described in the above.

4.4.2.8 Scale Effects Test Results (Straight Tailings and Composite Tailings/Sand Pastefills)

The test results were summarized previously in Table 4.4, they are presented graphically in Figures 4.19 and 4.20 respectively, for the straight tailings pastefill and the composite (blended) tailings/sand pastefill samples.

The test results clearly indicate the effects of scale on the strength and deformation properties of the samples. The differences are as much as a four fold increase between the largest and smallest straight tailings pastefill specimen sizes tested. The composite (blended) tailings/sand paste fill samples had strength and deformation properties that were generally similar to those of the straight tailings and sand paste backfills. The (blended) composite pastefill samples had void ratio values that were much lower than these of the straight base metal tailings pastefill samples.

The mean void ratio and porosity values for the composite tailings/sand pastefill samples were respectively, 0.38, and 27%. These should be compared with 0.81 and 44% for the straight base metal tailings samples and 0.47 and 31.6% for the precious metal tailings samples.

The observed effects may be due to the presence of more voids in the larger size samples than the smaller ones. The significance of the scale effects data in engineering design of backfill systems is discussed elsewhere in this Chapter (Section 4.4.9).

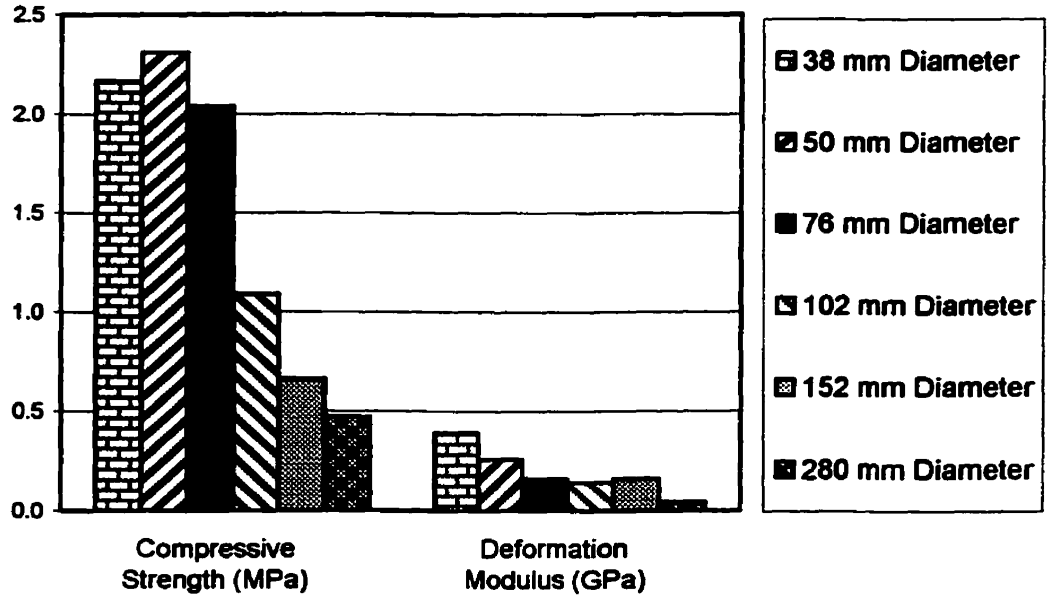


Figure 4.19 Scale effects on high-density tailings fill strength and deformation

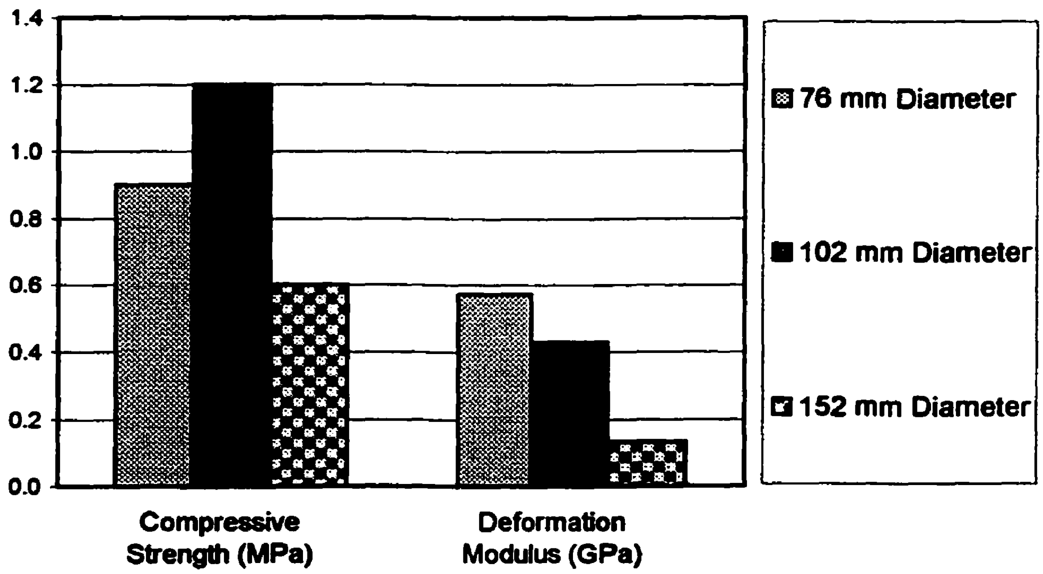


Figure 4.20 Scale effects on high-density composite (blended - tailings and sand) pastefill strength and deformation

4.4.8 STRENGTH AND DEFORMATION PROPERTIES OF CEMENTED ROCKFILL AND COMPOSITE AGGREGATE PASTE (CAP) FILL

4.4.8.1 Properties of Cemented Rockfill

The study on cemented rockfill properties was carried out primarily to determine the characteristics and properties of the material as a reference high-density fill for this study. It has been proposed in the course of this study that the properties of high-density composite backfill are basically a combination of those of cemented rockfill and tailings paste fill.

Two specimens sizes were used for the study. These were 152mm and 457mm diameter cylinders. The 152mm diameter cylinder was selected because it was probably the most commonly used test specimen size for cemented rockfill work at most mine sites (Yu and Counter, 1986). The 457mm diameter cylinder was the largest possible size that could be safely processed and handled by the available laboratory test equipment.

One hundred and one (101) cemented rockfill cylinders were tested at curing periods ranging from 14 days to 56 days. The test results are summarized in Table 4.11.

The bulk density of the test samples ranged between 1790 and 2430 (Kg/m^3) with an average value of 2006 Kg/m^3. Unconfined compressive strength of the material ranged between 0.82 and 10.88 MPa, depending on the test sample size and curing period. The average compressive strength value was 3.88 MPa. The average deformation modulus value was found to be 3.75 GPa; the values ranged between 0.90 and 24.6 GPa. The minimum and maximum void ratios were 0.13 and 0.53 respectively; the average value was 0.36. The averaged minimum void ratio for the rockfill aggregates from the relative density measurements (Section 3.1.8) was 0.37. This value compares favourably to the averaged void ratio for the stabilized rockfill samples.

The results suggests that cement stabilization (within the percentage used in this study) was an effective method for void reduction in the rockfill aggregates. Binder stabilization reduced the average void ratio from 0.60 (for the loosest material) to 0.36 (for the cemented rockfill samples). The maximum and minimum porosities were 34.7 and 11.7% respectively. The average porosity value was 26.5%. The results compare favorably with values in the published literature (Yu and Counter, 1983;Yu, 1989; Stone, 1993; Reschke, 1993; Hedley, 1995).

The properties of the cemented rockfill samples were found to be scale dependent. These are presented in Figure 4.21. Barrett et al. (1983) and Yu and Counter (1983) have proposed scaling factors for estimating in situ cemented rockfill compressive strength based on small scale values. Reschke (1993) has identified the scale effects on laboratory cemented rockfill test samples. Reschke found that unconfined compressive strength of the stabilized rockfill samples decreased with increased diameter and aggregate size. The decrease in sample strength with increasing diameter has been attributed to a decrease in the weight percentage of fine materials. Hedley (1995) has also indicated that the porosity of cemented rockfill samples is controlled by the cement content. This suggests that for cemented rockfill samples, the composition of fine material is basically controlled by the binder. Also, in terms of void reduction, the fines content of the rockfill samples can be improved by adding tailings or sand to the rockfill aggregates to produce a composite fill (Wingrove, 1993).

The results of this study show that on average, the 152mm diameter samples achieved almost twice the strength of 457mm cylinders. In terms of deformation modulus, the ratio between the small and large sample values was almost 10:1. Barrett et al. (1983) have suggested that the in situ compressive strength of cemented rockfill be taken as 60% of the small scale value. Yu and Counter (1983) have suggested that the in situ rockfill varies approximately at about 66 per cent of the laboratory determined value based on 152mm diameter cylinders, and about 90 per cent for sample sizes greater than 300mm diameter. In this regard, the observed compressive strength based on the 457mm diameter samples could be taken as close to the in situ value.

Table 4.3 Summary of mechanical properties test results for cemented rockfill samples

		Physical Properties									Mechanical Properties		Engineering Properties	
Fill Type	Fill Source	Specific Gravity	C_u	C_c	% - 20m	Bulk Density, ρ	Curing Period	Sample Diameter	Total Binder Content	Moisture Content at Testing	Compressive Strength, σ_c	Deformation Modulus, E	Void Ratio, e	Porosity, η
						(kg/m^3)	(Days)	(mm)	(%)	(%)	(MPa)	(GPa)		(%)
Rockfill	Cemented Rockfill	2.54 - 2.90	9.2 - 59.7	1.44 - 6.54	N/A	1790 - 2430 ***2006***	14	150	5	N/A	4.880 - 7.610 ***6.330***	0.970 - 1.410 ***1.170***	0.21 - 0.38 ***0.30***	17.6 - 27.3 ***22.9***
									7		4.570 - 10.880 ***6.440***	0.810 - 24.610 ***7.690***	0.13 - 0.37 ***0.30***	11.7 - 27.2 ***22.9***
							28	150	5		3.610 - 5.270 ***4.370***	4.590 - 17.810 ***12.640***	0.34 - 0.42 ***0.38***	25.5 - 29.7 ***27.4***
									7		3.630 - 8.880 ***6.260***	0.770 - 22.500 ***12.740***	0.27 - 0.36 ***0.32***	21.5 - 26.7 ***23.9***
							56	150	5		2.490 - 7.560 ***5.440***	1.310 - 13.610 ***7.200***	0.31 - 0.53 ***0.41***	23.6 - 34.7 ***28.7***
									7		4.120 - 8.730 ***6.630***	14.280 - 22.030 ***17.910***	0.31 - 0.37 ***0.35***	23.7 - 27.2 ***26.0***
	N = 101						14	457	5	N/A	0.820 - 5.680 ***2.830***	0.090 - 1.890 ***0.740***	0.21 - 0.40 ***0.35***	17.6 - 29.0 ***26.0***
									7		2.440 - 7.800 ***3.690***	0.590 - 2.190 ***1.300***	0.13 - 0.40 ***0.33***	11.7 - 28.5 ***24.7***
							28	457	5		0.820 - 3.390 ***2.540***	0.480 - 1.720 ***0.960***	0.34 - 0.49 ***0.41***	25.5 - 33.1 ***29.1***
									7		2.000 - 5.630 ***3.690***	0.560 - 6.250 ***1.480***	0.27 - 0.49 ***0.35***	21.5 - 32.8 ***25.7***
							56	457	5		1.320 - 5.470 ***2.600***	0.490 - 1.210 ***0.730***	0.31 - 0.53 ***0.42***	23.6 - 34.7 ***29.5***
									7		0.920 - 6.740 ***3.120***	0.260 - 2.490 ***1.020***	0.25 - 0.47 ***0.37***	19.8 - 31.9 ***26.9***
							14, 28, 56	150,457	5,7	N/A	0.820 - 10.880 ***3.880***	0.090 - 24.610 ***3.750***	0.13 - 0.53 ***0.36***	11.7 - 34.7 ***26.5***

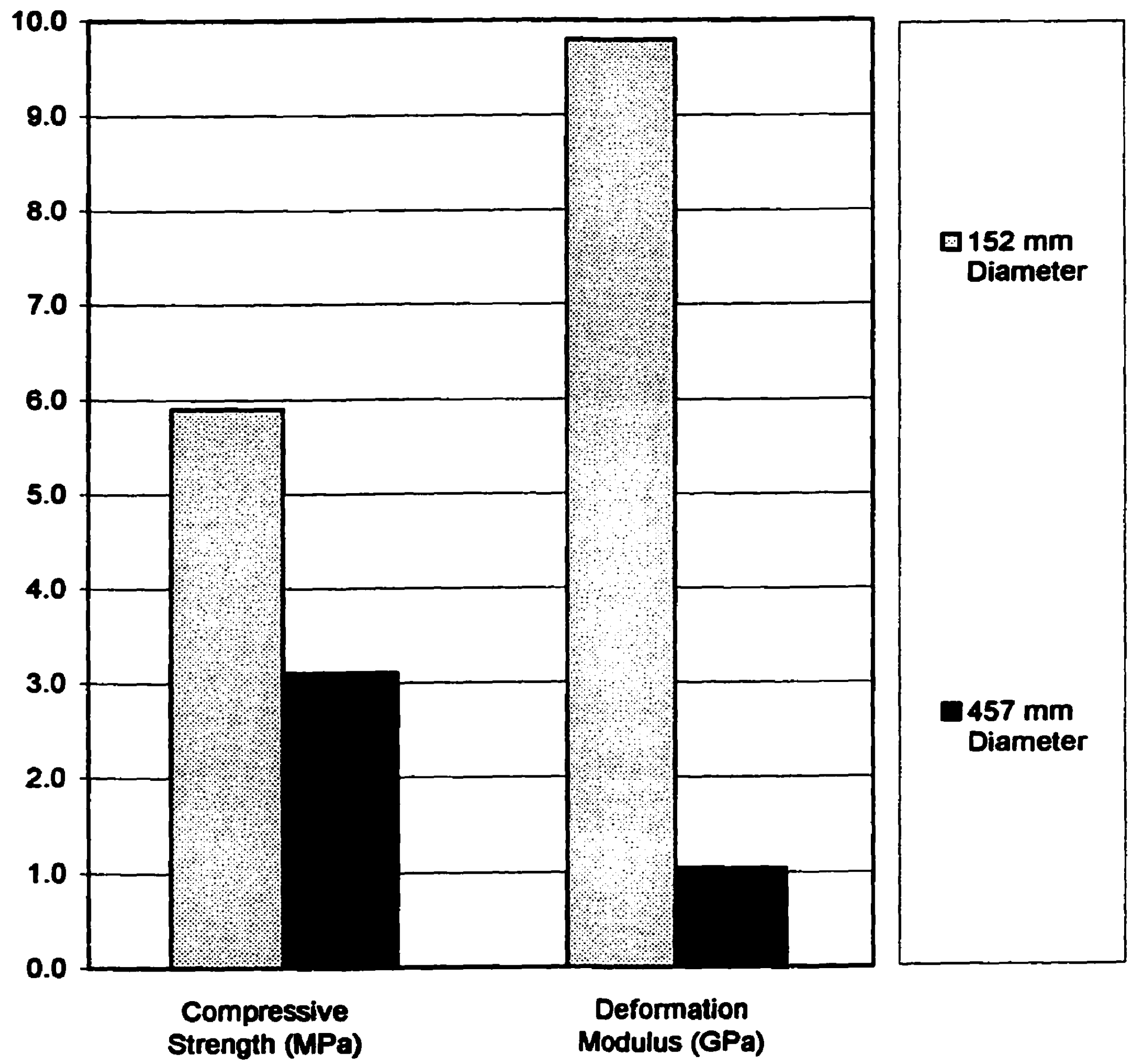


Figure 4.21 Scale effects on cemented rockfill compressive strength and deformation modulus

4.4.8.2 Properties of Composite-Aggregate Paste (CAP) Fill

The objectives of this element of the study were to determine the Composite-Aggregate Paste fill characteristics and properties as a new high-density backfill system, and also to compare the material properties to those of cemented rockfill and tailings paste fill. Additionally, to determine whether CAP fill properties were scale dependent so that the potential in situ properties of CAP fill may be estimated based on cemented rockfill, and paste backfill field data.

The CAP fill samples were prepared and tested as described in (Section 4.2.3). The test results are summarized in Table 4.4. The bulk-density of the test samples ranged between 2049 and 2520 Kg/m^3; the average bulk density value was 2406 Kg/m^3. The void ratio for the material varied between 0.18 and 0.31, with an average value of 0.25. These values are much lower than those obtained from the relative density tests on rockfill aggregates. The test results suggest that the formation of a composite fill enhanced the relative density of the material and reduced the void ratio. The average minimum void ratio of the rockfill aggregates was 0.37 (Table 3.3).

The average porosity value for the CAP fill samples is 19.8%, this value is based on minimum and maximum values of 14.5 and 32% respectively. At the 5% binder content, the compressive strength values varied between 0.43 and 2.12 MPa. Similarly the deformation modulus values ranged between 0.37 and 1.22 GPa. The compressive strength and deformation modulus values were lower than those of the cemented rockfill samples, but higher than the cemented paste backfill samples.

Table 4.4 Summary of mechanical properties test results for composite-aggregate paste (CAP) fill samples

Sample Diameter (mm)	Curing Period (Days)	Binder Content (%)	Sand Content (%)	Void Ratio	Porosity (%)	Compressive Strength (MPa)	Deformation Modulus (GPa)
152	7	5	0	0.23	18.90	1.07	0.83
152	7	5	0	0.26	20.40	1.12	0.83
152	7	5	0	0.24	19.40	1.18	0.83
457	7	5	0	0.24	19.00	1.13	0.72
457	7	5	0	0.24	19.50	0.87	0.48
457	7	5	0	0.25	20.20	1.05	0.67
152	14	5	0	0.21	17.10	1.33	1.10
152	14	5	0	0.24	19.60	1.20	0.53
152	14	5	0	0.20	16.60	1.88	0.68
457	14	5	0	0.31	23.60	0.73	0.73
457	14	5	0	0.24	20.20	2.07	1.22
457	14	5	0	0.27	21.40	1.58	0.88
152	28	5	0	0.20	16.70	1.42	0.60
152	28	5	0	0.21	17.30	1.40	0.70
152	28	5	0	0.22	17.80	1.30	0.82
457	28	5	0	0.28	22.10	2.12	1.18
457	28	5	0	0.27	21.00	1.67	1.18
457	28	5	0	0.30	22.90	1.33	1.13

The low void ratio and porosity values suggest that the composite aggregate fill is a more compact and denser material, compared to either the cemented rockfill and the tailings paste fill. This is also supported by the bulk density values. The CAP fill samples also had lower void ratios and porosities compared to either the cemented rockfill or the straight tailings fill. The characteristics of the stress-strain curves are indicative of a strain-hardening or a resilient material (Lama and Vitukuri, 1974).

In terms of scale effects on the CAP fill samples, there seems to be no apparent differences between the properties of the small scale test samples (152mm diameter cylinders) and those of the large samples (457mm diameter cylinders). These are presented in Figure 4.22 and suggest that CAP fills may not be scale dependent. This suggests that unlike the cemented rockfill samples, the CAP fill achieved maximum packing density with addition of the tailings material. The results are contrary to observations made with the cemented rockfill and the paste backfill samples.

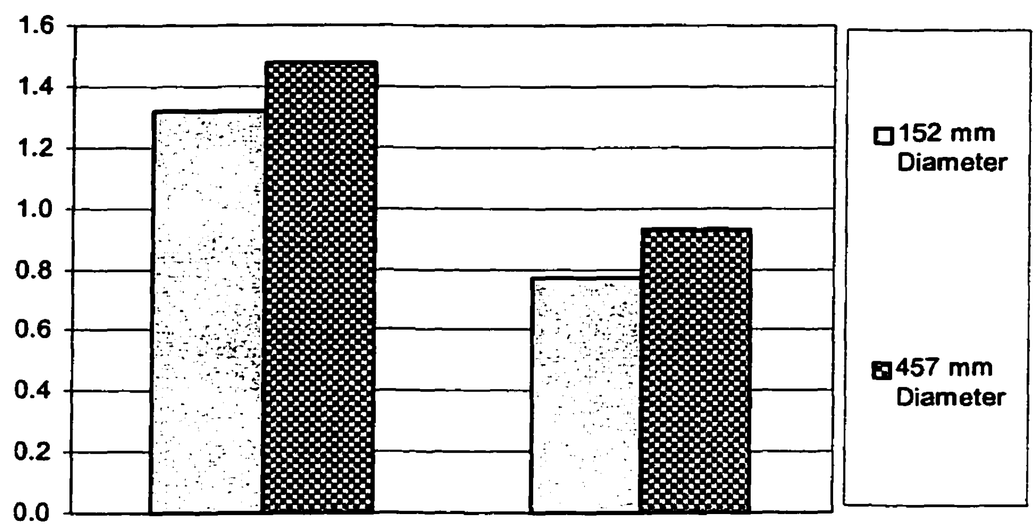


Figure 4.22 Scale Effects on Composite-Aggregate Paste (CAP) Fill, Compressive Strength and Deformation Modulus

4.4.8.3 <u>Application of the Mechanical Properties Tests Results</u>

Farsangi (1996) has indicated some quality control factors that could affect in situ cemented rockfill properties. These have included the use of potable water for mixing, correct batching of aggregates and binder as well as other environmental factors. These factors should also apply to CAP fill preparation and placement. Considering that this study was carried out under well controlled laboratory conditions, the established mechanical properties may not apply to in situ conditions. Care must therefore be exercised in using these results for design.

4.9 SIGNIFICANCE OF SCALE EFFECTS TESTS RESULTS

Barrett et al. (1983), Yu and Counter (1983, 1986), Reschke (1993) and others have shown that laboratory determined backfill properties are scale dependent. For cemented rockfill, it has been suggested that a scaling factor of the field design values be based on, 60 percent to 66 percent of the laboratory determined strength. These scaling factors may also apply to the properties of composite-aggregate paste (CAP) fill, although the results of this study (Section 4.8.3) have suggested that the material is probably not scale dependent.

With regard to the straight tailings pastefill and composite (blended) tailings/sand pastefill samples, there was a four fold difference in compressive strength between the small size (38mm) and large size (279mm) diameter samples. It has been suggested that the unconfined compressive strength can be used for a preliminary design of cement bond strength (Mitchell, 1983). In the simplest case, it has been proposed that low cement bond strength "Cb" can be taken to be equal to the unconfined compressive strength of the material as outlined in the following equation:

$$\sigma_{1f} = 2C_b = \frac{\gamma HF}{N_f} \qquad 4.6$$

where σ_{1f} = limiting confining compressive strength

γ = unit weight

H = free standing height of fill

C_b = low cement bond strength

N_f = stability number (or cohesion)

In terms of the above equation, the estimate for "Cb" can vary up to four times depending on the sample size and if scale effects are considered for the straight tailings pastefll samples in this study.

The stability analysis of tailings fill masses are generally based on the shear strength parameters (Mitchell et al., 1982), and not only on the unconfined compressive strength of the material. Based on Limit Equilibrium Analysis (Mitchell et al.,1982 ; Mitchell, 1983; Pierce et al., 1998) has proposed that the selection of backfill design mixes should be subjected to triaxial testing and effective stress analysis in terms of cohesion and the friction angle. The scale effects on the triaxial compressive strength measurements were not determined as part of this study. Previous preliminary investigations by the author had suggested the possibility of scale effects on the triaxial test results.

It is anticipated that any potential effects of specimen size on engineering design analysis could be offset by the selection of relevant analytical models with applicable safety factors.

The scale effects on triaxial compression test results and the implications on stability analysis of mine backfill systems, require closer examination in future studies.

4.10 SUMMARY

The unconfined compressive strength measurements were carried out on backfill materials consisting of straight tailings pastefill, composite blended tailings/sand pastefill, composite-aggregate pastefill (CAP) and cemented rockfill samples.

The investigations were carried out in terms of the type of tailings, binder composition, moisture content, curing environment and time. The tests were conducted in an attempt to establish the effects of the physical properties of the respective fill materials on strength development, when used as a composite product.

The test results suggest the following:

1. A wider distribution of particle sizes are required for strength development.

2. The addition of sand to fine tailings as a means of improving the size gradation of the composite product did not readily result in strength gain in the short term. Early improvements in compressive strength occurred when the composite tailings/sand mixtures had a wider range of particle size gradation.

3. Changes in bulk density, void ratio or porosity with sand content could be used as an indicator for early strength gain for cemented composite-tailings/sand fill samples.

4. Alternative cementing agents can be prepared from a blend of ordinary portland cement and other supplementary binders including slag, flyash, anhydrite, silica fume and locally available metallurgical by-products. Site specific variations in tailings mineralogy and chemistry, as well as the mine environment, demand that stabilization trials be carried out at each mine site in order to arrive at suitable binder combinations.

5. A water/binder (W/C) ratio of less than 3.0 produced effective strength gain in composite tailings/sand samples based on the results of this study as well as on data in the published literature. The site specific properties of backfill materials demand prior testing at each mine site to determine a suitable W/C ratio.

6. The unconfined compressive strength and deformation modulus of the straight tailings, composite-blended tailings/sand and the cemented rockfill samples were found to be scale dependent. On the other hand, the compressive strength and deformation modulus of the composite-aggregate paste (CAP) fill samples were unaffected by sample size. This suggests that the properties of the composite material in situ could be similar to those determined from laboratory measurements.

7. Further investigation of scale effects on composite fill properties is recommended for future studies.

CHAPTER 5

5.1 EFFECTS OF LOADING CONDITIONS ON HIGH-DENSITY FILL STRENGTH AND DEFORMATION PROPERTIES

The performance of soil-like materials including paste backfill is controlled by shear strength. It has been suggested that similar to soils (Terzaghi and Peck, 1967) the strength of backfill (Singh, 1976; Thomas et al., 1979) materials may be expressed in terms of the Mohr-Coulomb equation:

$$\tau = c + \sigma_n \tan \phi \qquad 5.1$$

where τ= shear strength
c = cohesive strength
σ_n = normal stress
ϕ = friction angle

The above equation basically involves cohesion and friction components.

The strength development depends on either the cohesion component or friction component or both. The cohesion and friction factors are influenced by the physical properties of the fill materials. These include, particle size distribution, moisture content, binder composition, curing environment and time. Condition of loading including confinement (Moruzi, 1978) may also improve the characteristics of the shear strength components. Confinement can also break down the cement bond strength (Mitchell and Wong, 1982; Hassani and Aref, 1988; Aref et al., 1989) and it can adversely affect the stability of the fill mass when exposed (Ouellet, et al., 1998).

Direct shear tests and triaxial compressive strength measurements were carried out to define the shear strength characteristics of the tailings material, and also the cemented and uncemented paste fill samples. The load-response characteristics of the straight tailings and composite paste fill samples were investigated in an attempt to understand the potential behaviour of these materials when used as high-density composite mine backfill.

The direct shear tests were carried out at various moisture contents of the tailings samples. Singh, (1976) has suggested that moisture content can affect the long term behaviour of hydraulic fill masses in-situ. On the other hand, excessive moisture in the uncemented fill can affect the stability of the fill mass in situ by reducing its resistance to liquefaction (Hassani and Aref, 1988; Aref et al., 1989; Ouellet et al., 1998).

5.2 DIRECT SHEAR TESTS

Sample Preparation

The Direct Shear tests were carried out as an initial step in the evaluation of the shear properties of the uncemented tailings fill material. The maximum cohesion and other shear strength properties are known to develop at the optimum moisture content for soil materials (Terzaghi and Peck, 1967). Strong similarities have been established between the behaviour of engineering soils and tailings fills (Singh, 1976, Thomas et al., 1979). In this regard, a multi-step approach was used to evaluate the effects of moisture content on the shear properties of the ultra-fine uncemented total tailings paste fill. This procedure was followed as part of the process for identifying suitable mix design limits for the composite backfill materials in this study.

1) Initially, standard compaction tests (ASTM D-698) were conducted to establish the moisture-density relationships for some of the tailings and composite mixtures of tailings and sands in this study; and also, to identify their optimum moisture contents

and maximum dry densities (Terzaghi and Peck, 1967). Typical results of the Dry-Density-Moisture relationship for mill tailings is presented in Figure 5.1.

2) Having established an optimum moisture content and maximum dry density for a test material, the shear test specimens were prepared and tested, beginning with dry samples and ending with partially saturated samples.

3) Each shear test specimen was prepared at approximately the same density by vibrating the shear box containing the lose material until the desired density had been reach, prior to testing.

5.2.1 Analysis of Direct Shear Testing

The test results are summarized in Tables 5.1a&b. The failure envelopes for the two types of tailings materials are presented in Figure 5.2. The results indicate that the uncemented precious metal total tailings developed similar shear strength properties as the uncemented base metal tailings material. The results also indicate that either tailings material developed some apparent cohesion values without cementation at the optimum moisture contents of the respective tailings materials. The measured optimum moisture contents for the base metal and precious metal tailings were respectively, 14.5% and 16.8% respectively.

Singh (1976) has suggested that 10 to 15% moisture content is essential for maintaining the long term stability of bulk fill in an underground environment. It is interesting to note that the identified optimum moisture levels for the test materials in this study (Table 5.1) are within the identified range of the values suggested by Singh (1976). Also, the maximum cohesive strengths of the moist tailings materials occurred at the maximum densities and the optimum moisture contents of the materials.

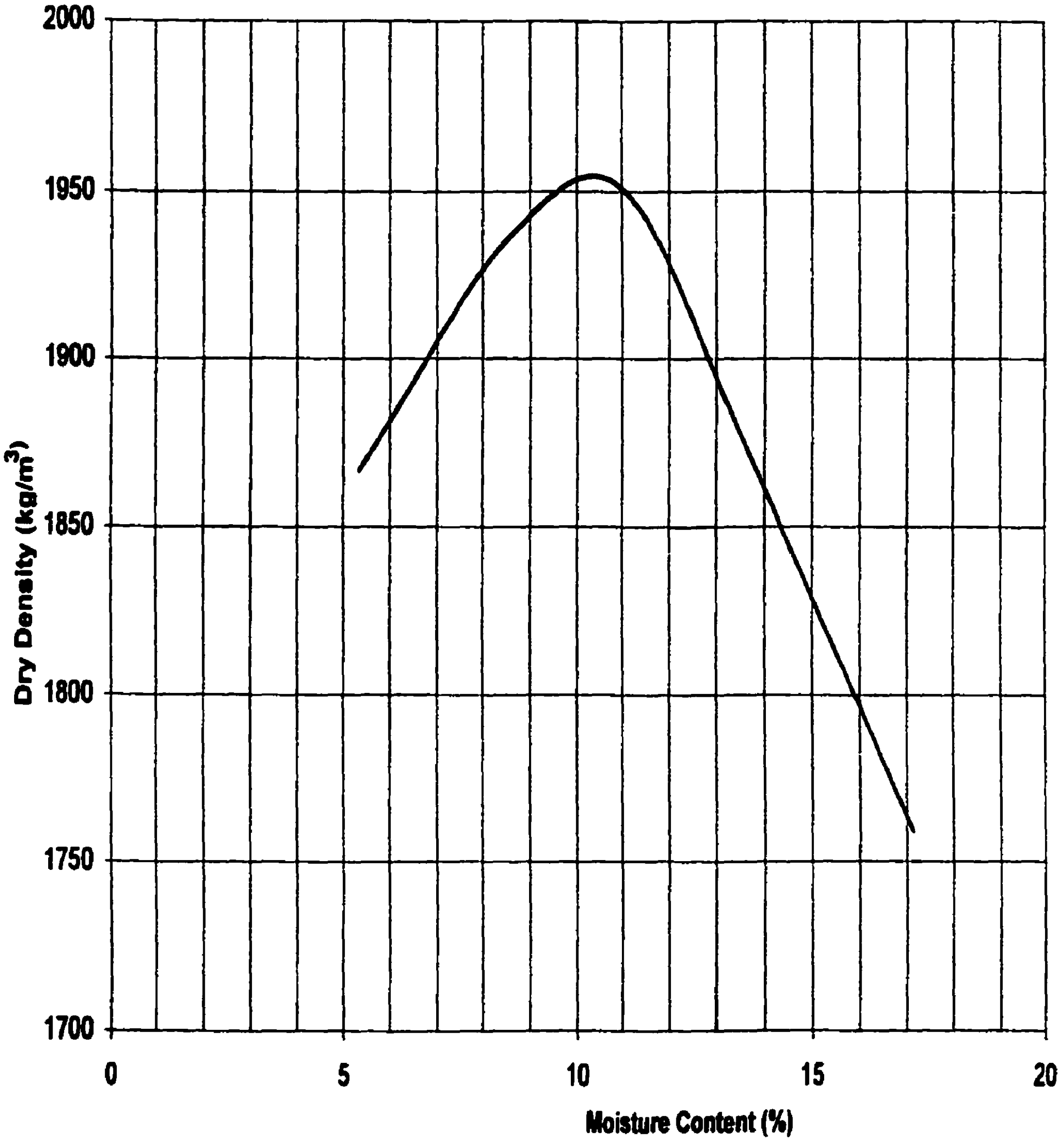


Figure 5.1 Typical Results of Dry Density vs. Moisture Content Relationship for Tailings

Table 5.1 (a) Effects of Moisture Content on Shear Strength Parameters for Base Metal Tailings (Direct Shear Strength Tests)

Sample	Normal Stress (kPa)	Maximum Shear Stress (kPa)	Apparent Cohesion (kPa)	Internal Friction Angle ϕ^o
Dry	24.0 43.6 73.0 102.4	32.1 52.2 86.1 106.2	11.1	42
5% Moisture	24.0 43.6 73.0 102.4	23.4 34.5 50.4 67.5	9.8	30
10% Moisture	24.0 43.6 73.0 102.4	23.7 34.5 55.5 80.4	4.1	37
14.50% Moisture (Optimum)	24.0 43.6 73.0 102.4	32.0 49.4 65.6 78.9	21.1	29
20% Moisture (Partially Saturated)	24.0 43.6 73.0 102.2	37.8 51.6 92.1 138.6	0	53

Table 5.1 (b) Effects of Moisture Content on Shear Strength Parameters for Precious Metal Tailings (Direct Shear Strength Tests)

Sample	Normal Stress (kPa)	Maximum Shear Stress (kPa)	Apparent Cohesion (kPa)	Internal Friction Angle ϕ^{o}
Dry	24.0 43.7 73.0 102.4	22.3 37.1 63.5 9.2	11.3	41
5% Moisture	24.0 43.7 73.0 102.4	23.7 37.9 60.1 83.2	5.0	37
10% Moisture	24.0 43.7 73.0 102.4	23.5 37.8 58.9 84.6	4.4	38
16.80% Moisture (Optimum)	24.0 43.7 73.0 102.4	28.0 39.9 61.10 82.2	10.1	36
19% Moisture (Partially Saturated)	24.0 43.7 73.0 102.2	22.0 40.2 59.9 87.0	3.8	39

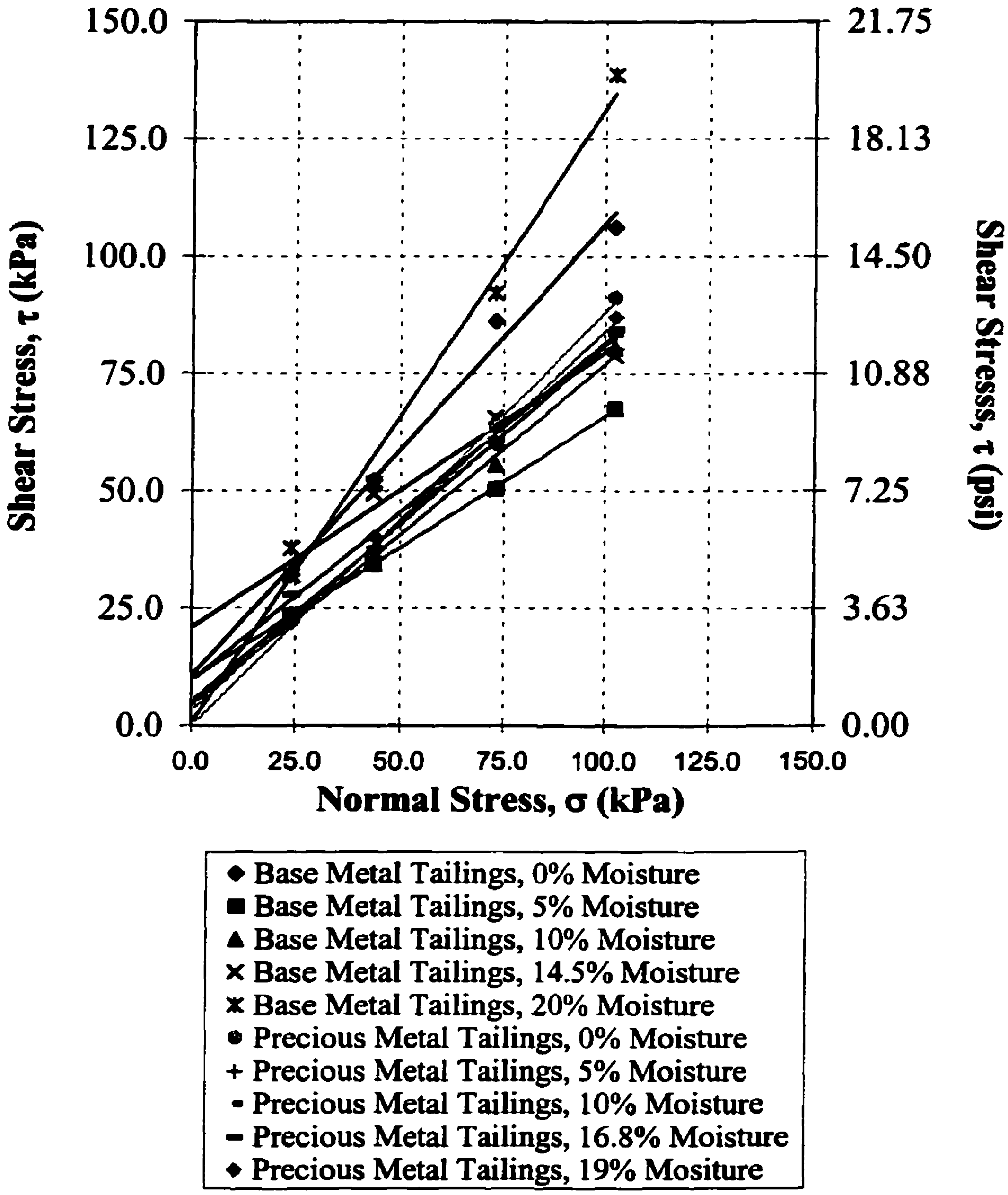


Figure 5.2 Failure Envelope for Base Metal and Precious Metal Tailings as a Function of Moisture Content

5.3 TRIAXIAL COMPRESSION STRENGTH TESTS

5.3.1 Triaxial Compressive Strength Test Procedure and Results

The main objective of the triaxial compressive strength tests was to investigate the relative responses of the various pastefill materials in this study to load.

5.3.2 Sample Preparation and Test Procedures (ASTM C-192; D-2850)

The test samples were prepared and cured in a similar manner to the unconfined compressive strength specimens (Chapter 4). Test sample sizes comprised of 102mm diameter by 204mm long cylinders. Both cemented and uncemented samples were tested. The uncemented samples were prepared at the optimum moisture content of the respective tailings material (Sec. 5.2.1). The cement content was fixed at 5% by dry weight of tailings. The sample mixing and curing procedures were as described in the Section on Unconfined Compressive Strength testing (Section 4.1).

Prior to testing, each triaxial test sample was enclosed in an impermeable rubber membrane. The jacketed sample was placed in a Universal Testing Machine. De-aired, distilled water was injected into the triaxial chamber and the cell was pressurized. A confining pressure (σ_3) was maintained around each sample while an axial stress (σ_1) was applied to the top of the sample. The tests were conducted over a confining pressure range of 207 to 828 kPa (30 to 120 psi) and a uniform axial strain was used. The axial load and sample deformation were continuously monitored throughout the test. The deviator stress (σ_1-σ_3) and axial strain data were used to derive the shear strength parameters of each curing period.

Sixty-nine (69) samples were tested under undrained conditions without pore pressure measurements (Bishop and Henkel, 1962). The samples were partially saturated at the time of testing. Barrett (1973) suggests that triaxial tests should be conducted on samples having the same degree of saturation as the fill in situ. In situ moisture contents of pastefill masses have been reported to be 15% lower than laboratory determined values (Udd and Annor, 1993). Hunt (1989) has also conducted undrained tests on pastefill samples without pore pressure measurements. The triaxial compression tests were conducted at curing periods that ranged between 3 and 28 days. The triaxial test equipment is presented in Figure 5.3a&b.

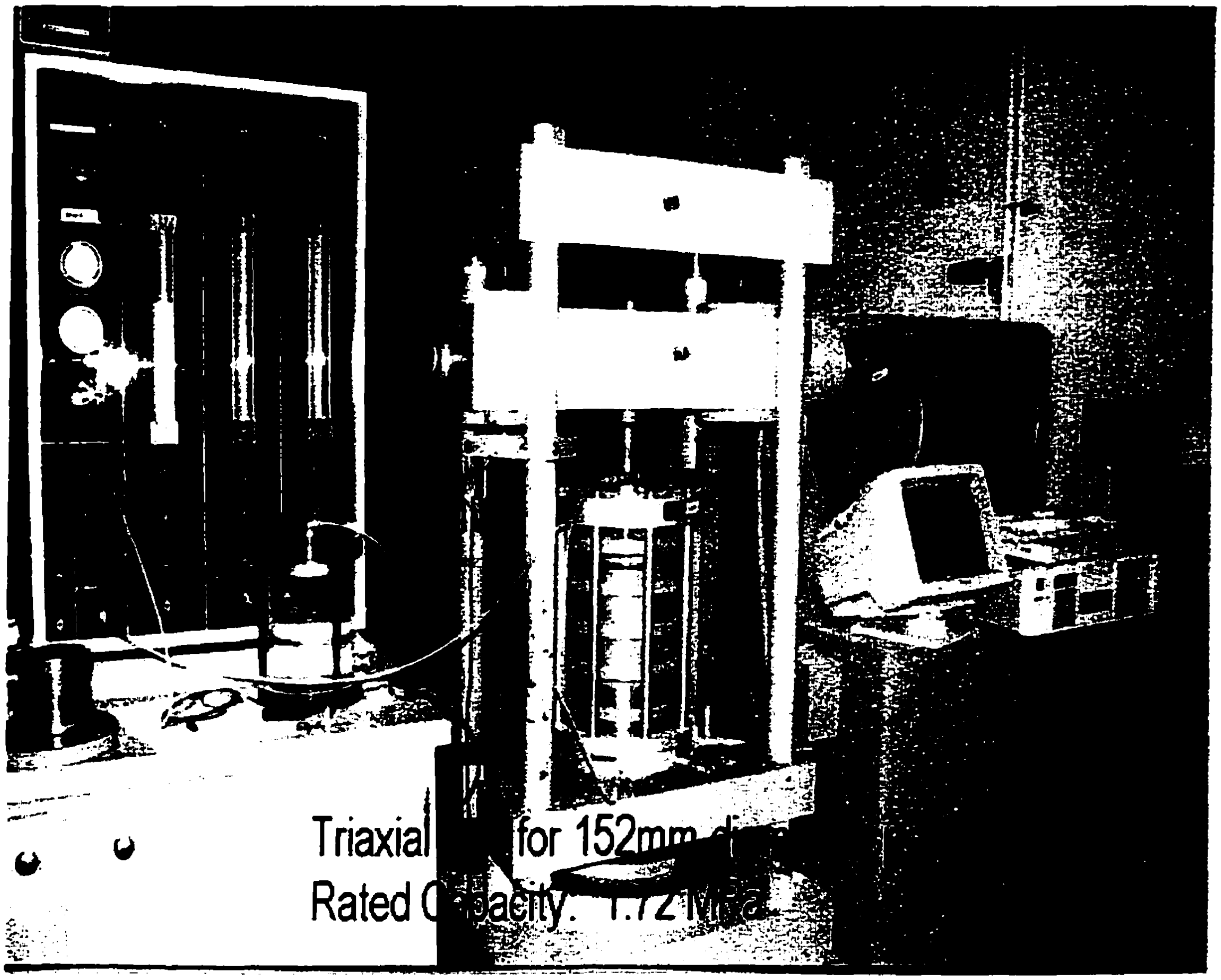


Figure 5.3a Triaxial testing equipment showing cell and control panel.

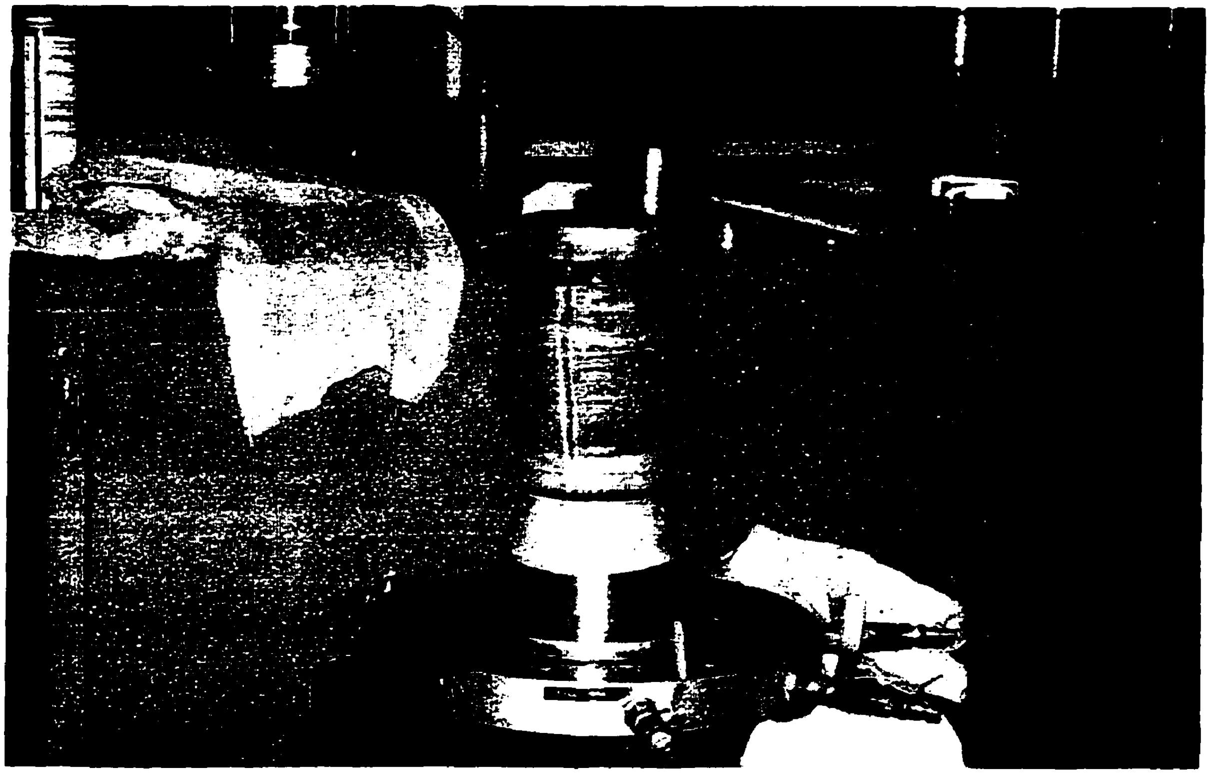


Figure 5.3b A picture of a 102mm diameter by 204mm long test specimen after triaxial compressive strength testing.

5.4 ANALYSIS OF TRIAXIAL COMPRESSIVE STRENGTH TEST RESULTS

The results of the triaxial compression measurements are discussed in this section in terms of the respective pastefill materials response to load.

Typical deviator-stress versus strain curves for the triaxial compression measurements are presented in Figure 5.4. The triaxial compression test results were used to derive the Mohr-Coulomb failure envelope parameters. The failure envelope parameters for the uncemented precious metal and base metal paste pastefill samples are tabulated in Table 5.2. Typical stress-strain curves and the Mohr's circles and the failure envelope parameters for the pastefill samples in this study are presented in Appendix C-1.

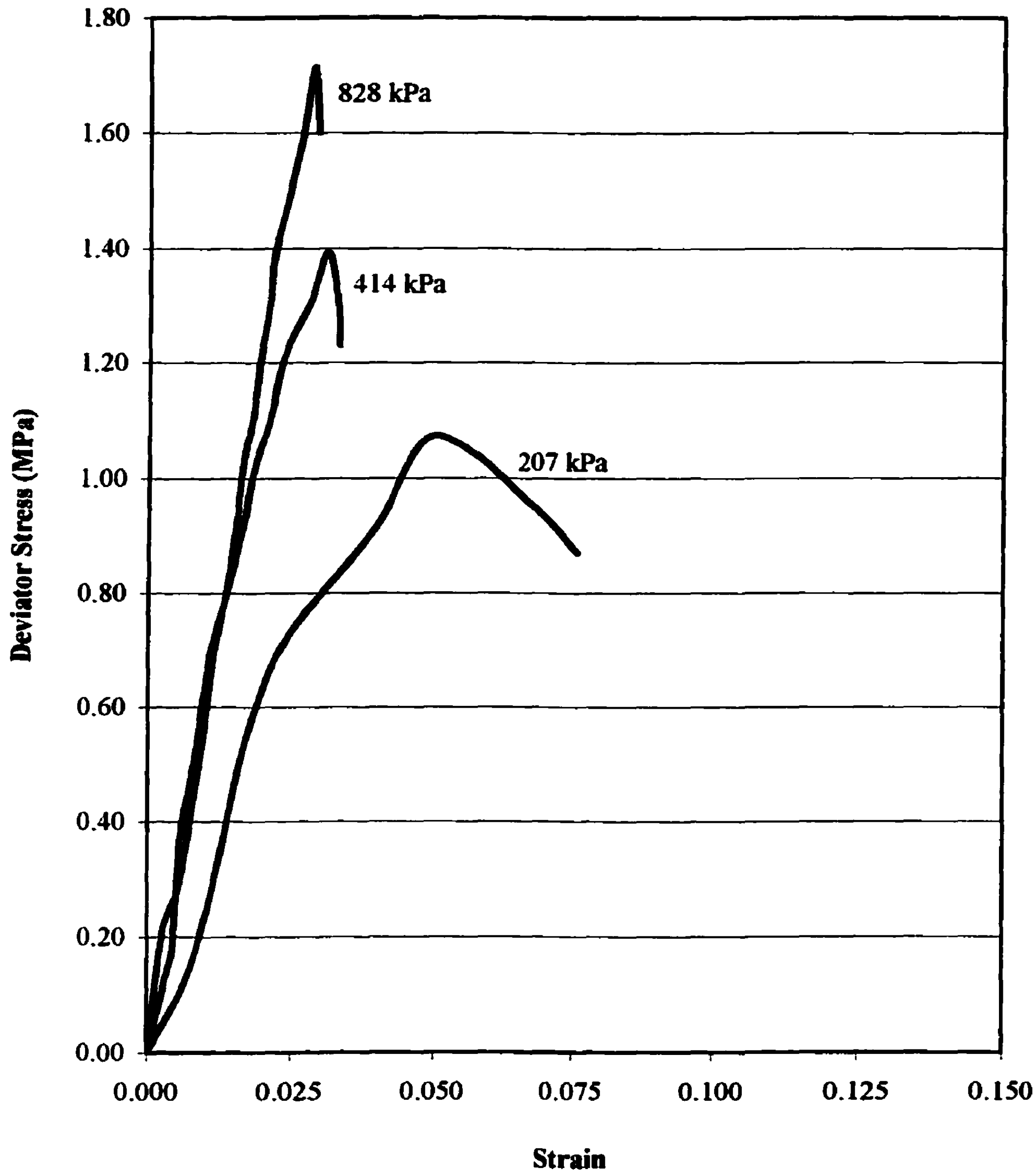


Figure 5.4 Typical Deviator Stress vs Strain Curves for Cemented Sand Paste Fill (5% PC) with Confining Pressures of 207, 414 and 828 kPa (Curing Period = 14 Days)

5.4.1 Generalized Behaviour of Geologic Materials Under Load

The stress strain relationships of most geologic materials are generally considered linear and are governed by Young's modulus, Adams and Williamson (1923), Brace (1965), Walsh (1965) and many others have suggested that the non-linear relationship for rocks is due to cracks and voids in the rock. At low stress levels, the non-linear elastic behaviour of rocks under compression have been attributed to this influence (Annor and Katsube, 1983; Annor, 1985). Lama and Vutukuri (1978) have suggested that the stress-strain curves for geologic materials including rocks and soils can differ depending on the test method, loading-rate, stress level, specimen size, moisture content as well as other factors. Lama and Vutukuri (1978) have also identified three types of idealized stress-strain curves for uniaxial compression tests involving rock samples. These are: (i) Linear-elastic or brittle behaviour; (ii) Strain softening or ductile behaviour; and (iii) strain hardening behaviour. These characteristic curves apply to triaxial compression test results (Mitchell and Wong, 1982) for fill materials in this study and therefore, they can also be used to identify the materials response to applied load.

5.4.2 Load Response Characteristics of the Paste Fill Samples

Typical stress-strain curves from the unconfined compressive strength tests (Chapter 4) are presented in Figures 5.5 to 5.8 for comparison with the triaxial compressive strength test results.

5.4.2.1 Characteristics of the Unconfined Compression-Stress -Strain Curves

Figure 5.5 shows typical stress-strain curves from unconfined compressive strength tests on stabilized tailings pastefill samples, at 7, 14 and 28 days curing periods. The curves show both linearly elastic and stain softening segments, with clearly defined yield points. The characteristics of the stress-strain curves suggest brittle and ductile behaviour under increased load.

Figure 5.6 represents typical stress-strain curves for the cemented rockfill samples. The curves show characteristics of a brittle material at all the curing periods. The curves also show initial strain hardening segments which are attributed to initial settling of the capping material and closure of voids in the bulk sample under load. This behaviour is similar to crack closure in rocks at low stress levels (Annor and Katsube, 1983; Annor, 1985).

Figure 5.7 represents the stress-strain curves for the composite blended tailings/sand pastefill samples. The 28 days cured sample displayed both brittle and ductile behaviour similar to the straight tailings samples (Figure 5.5). The 7 and 14 days curves show strain hardening and ductile behaviour which suggest that the composite samples were resilient during the early stages of curing, but became brittle with longer curing period.

Figure 5.10 shows typical stress-strain curves for the composite-aggregate pastefill (CAP) samples. The curves suggest predominantly resilient behaviour. This is evidenced by the prevalent strain-hardening curves at all the curing periods of testing. The stress-strain curves of the CAP fill samples also have linearly elastic segments which suggest that the material stiffens with increased load.

The characteristics of the unconfined compressive strength curves suggest that the composite materials (Blended tailings/sand pastefill samples and CAP fill samples) in this study displayed more resilient behaviour than either the straight tailings pastefill or the cemented rockfill samples. The resiliency of the composite materials suggest that they could be the most suitable material for tight filling purposes in mine stopes than either the straight tailings pastefill or the cemented rockfill.

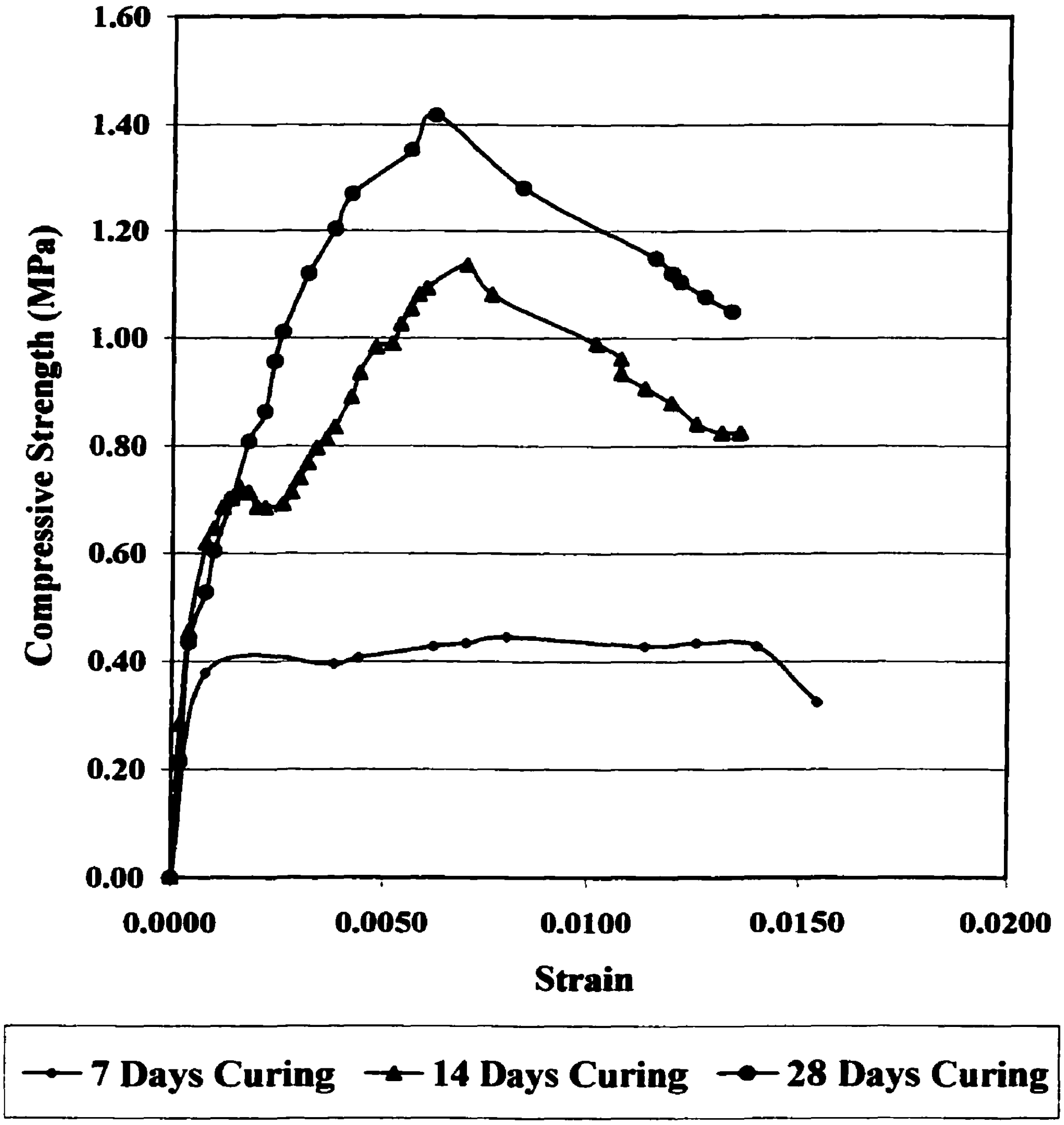


Figure 5.5 Typical Stress vs Strain Curves for Precious Metal Tailings Pastefill in Unconfined Compression Testing

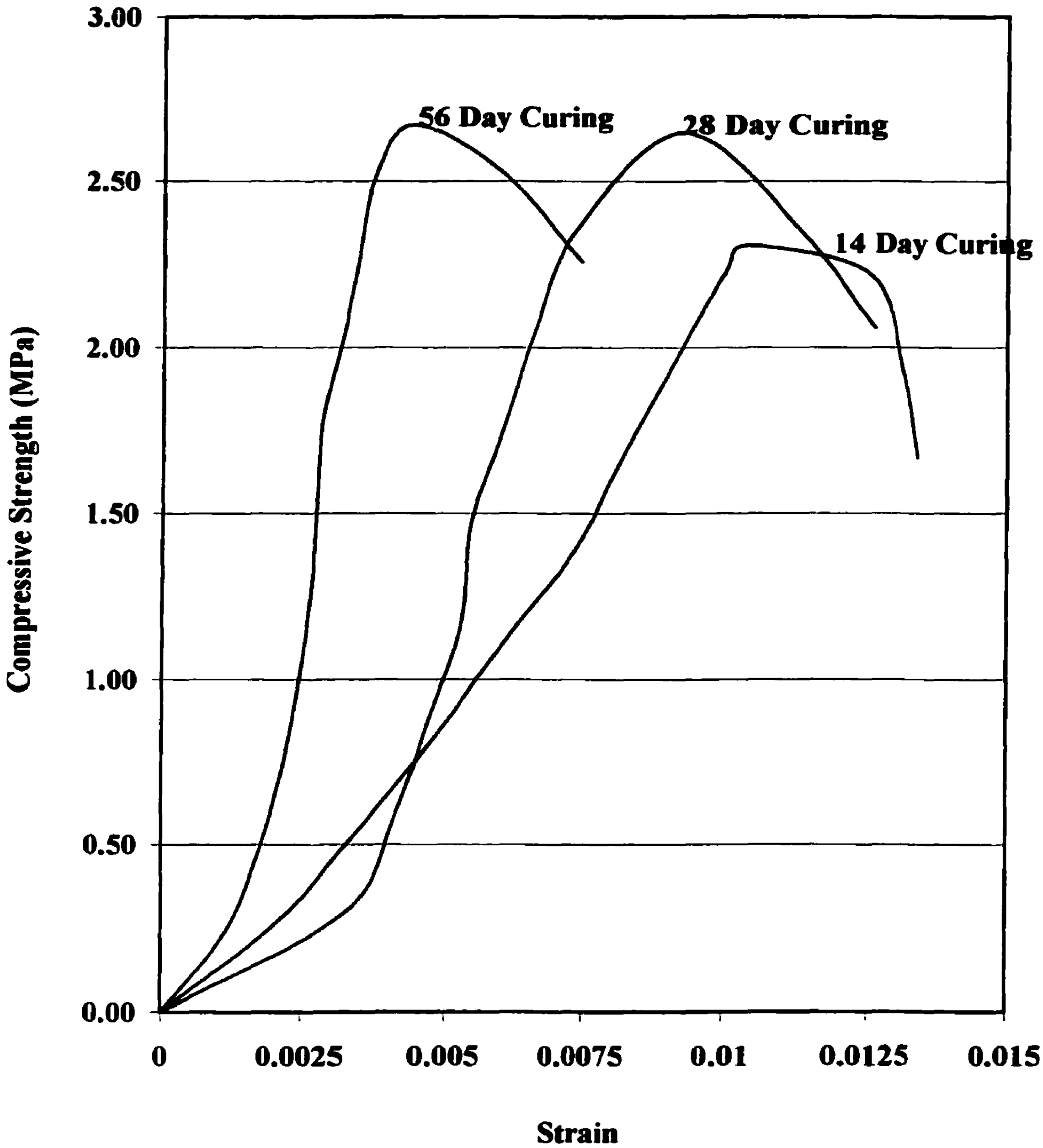


Figure 5.6 Typical Compressive Strength vs. Strain Curves for Cemented Rockfill Specimens - 5% Binder

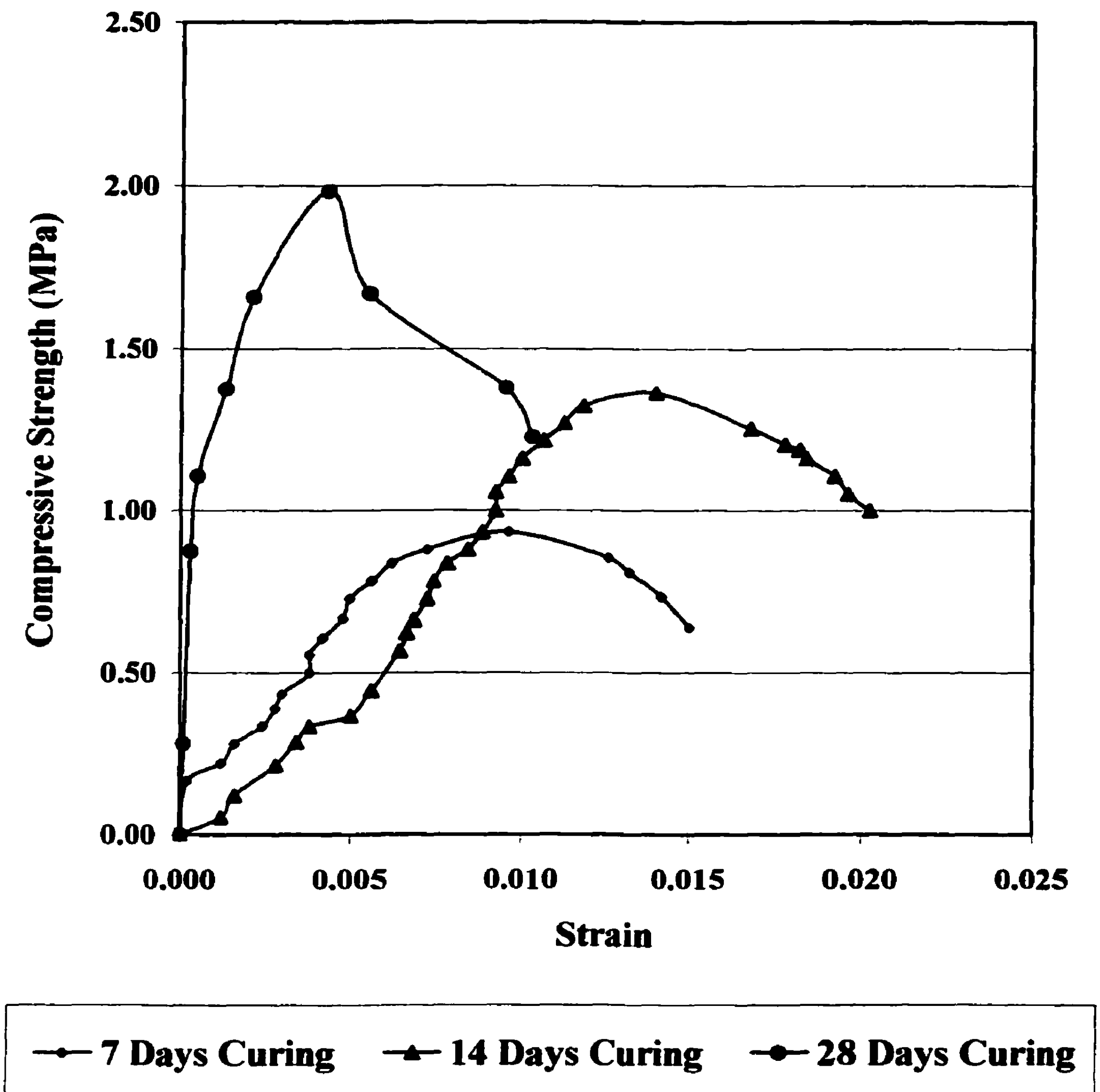


Figure 5.7 Typical Compressive Strength vs. Strain Curves for Composite (Blended Precious Metal Tailings/Sand) Samples

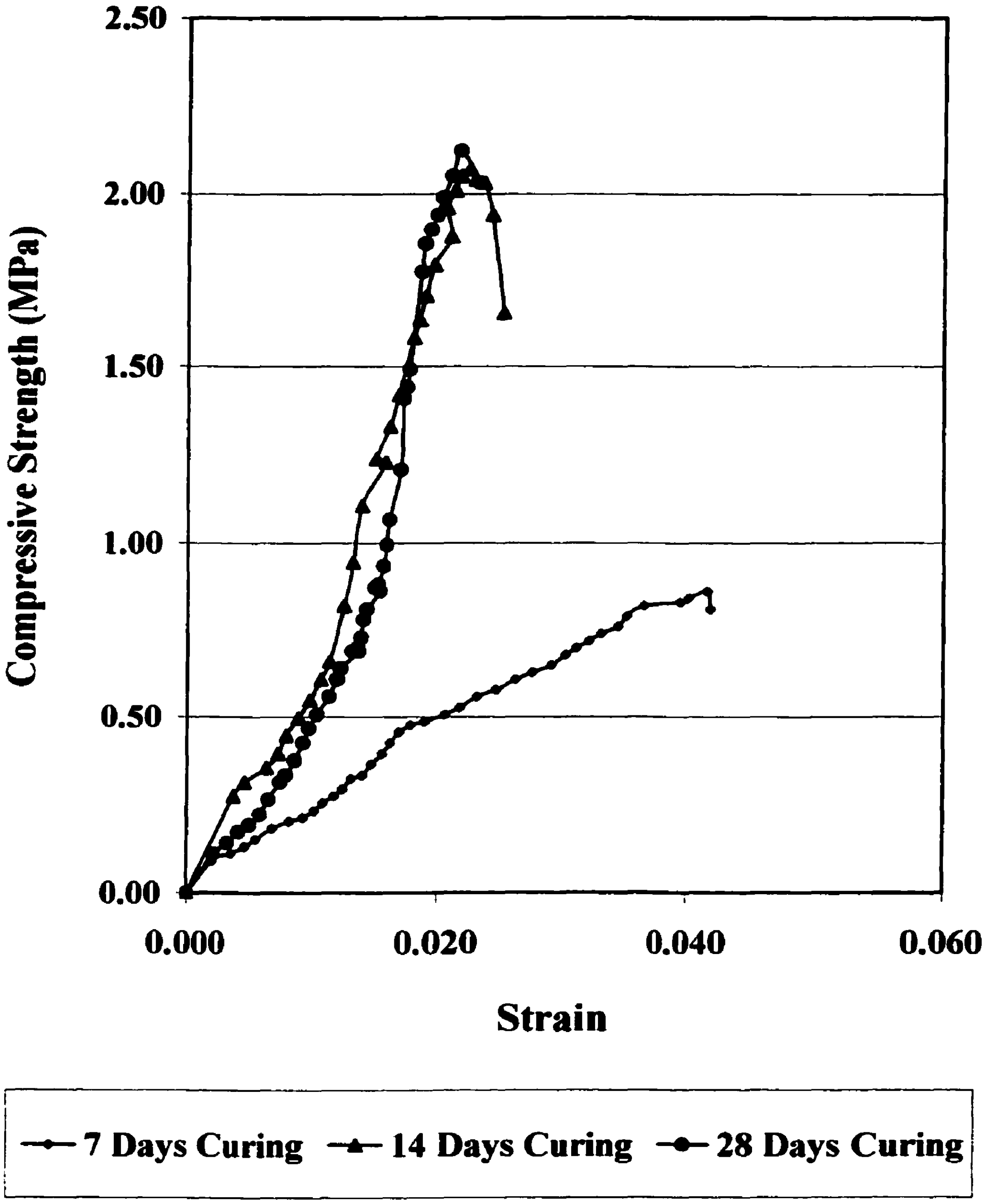


Figure 5.8 Typical Stress vs Strain Curves for Composite-Aggregate Pastefill (CAP) in Unconfined Compression Testing

5.4.2.2 Characteristics of the Deviators Stress vs. Strain Curves

In general terms, the characteristics of the stress-strain curves from the triaxial compression measurements were similar to those of the unconfined compressive strength tests. The deviator-stress versus strain curves were characterized by both brittle and ductile behaviour for the uncemented base metal pastefill samples (Figure 5.9) and the cemented samples (Figure 5.10). There were also some indications of strain-hardening behaviour with regard to the uncemented precious metal tailings fill samples (Figure 5.11).

In general, the uncemented precious metal tailings samples displayed strain-hardening behaviour at most of the confining pressure levels of the investigations. On the other hand, the uncemented base metal tailings samples showed a combination of brittle and ductile behaviour over the confining pressure range of testing. The strain hardening behaviour for the uncemented specimens was generally most pronounced at the confining pressure of 828 kPa (Figure 5.12).

The cemented total tailings samples displayed a combination of brittle and ductile behaviour under confinement. Strain-hardening behaviour was also observed at higher confining pressures (Figure 5.10) for some of the test samples. These observations were independent of curing period. The sand paste fill samples displayed a combination of brittle and ductile behaviour under confinement (Figure 5.13). There were however, marked strain-hardening behaviour at all confining pressure levels for this test material.

5.4.2.3 Summary - Characteristics of the Stress - Strain Curves

In general terms, there was non-uniform response of the paste fill test samples to applied load. The response seemed to be material specific. Both the uncemented and cemented total tailings and sand pastefill samples displayed a variety of behaviours under confinement. The predominant characteristics of the stress-strain curves were initially, linearly elastic which suggest a brittle behaviour. The materials became ductile under increasing load and were characterized by strain softening curves. The cemented sand composite and the uncemented precious metal tailings samples displayed strain-hardening behaviour under confinement. This suggests a resilient behaviour. The test samples were characterized by increasing deformation modulus and internal friction angle, and decreasing cohesive strength values with increasing confining pressure Table 5.2. Tests conducted at a confining pressure of 828 kPa showed strain hardening behaviour for some of the uncemented as well as cemented material.

The characteristics of the deviator stress versus strain curves were similar for both the uncemented (Figure 5.9) and the cemented (Figure 5.10) base metal tailings samples. The curves of the uncemented and cemented base metal tailings samples differ in terms of their respective yield points. The yield points for the uncemented material varied between deviator-stress values of approximately 0.4 MPa to 1.5 MPa. These should be compared with about 0.6 MPa to 1.5 MPa for the stabilized samples containing 5 percent cement.

The yield zone for the uncemented base metal tailings samples seems to be more affected by confining pressure than the cemented material. For example, the transition zones for the uncemented sample ranged between approximately 0.7 MPa and 1.5 MPa for the material tested at a confining pressure of 828 kPa (Figure 5.12). Mitchell and Wong (1982) have suggested that the yield point represent the onset of cement bond breakdown under high confining pressures. The results of this study show that the strength of the uncemented material was rather enhanced beyond the yield point under increased confining pressure probably due to increased density. Head (1980) has indicated that the bulk density of fill materials tend to increase at the optimum moisture contents. Additionally, Head (1980) has suggested that there can be no pore pressure build up at the optimum moisture level of the material. It is also suggested that pore pressure begins to build up after the optimum moisture level has been reached. When this occurs, the bulk density of the fill beings to decrease and the material becomes saturated.

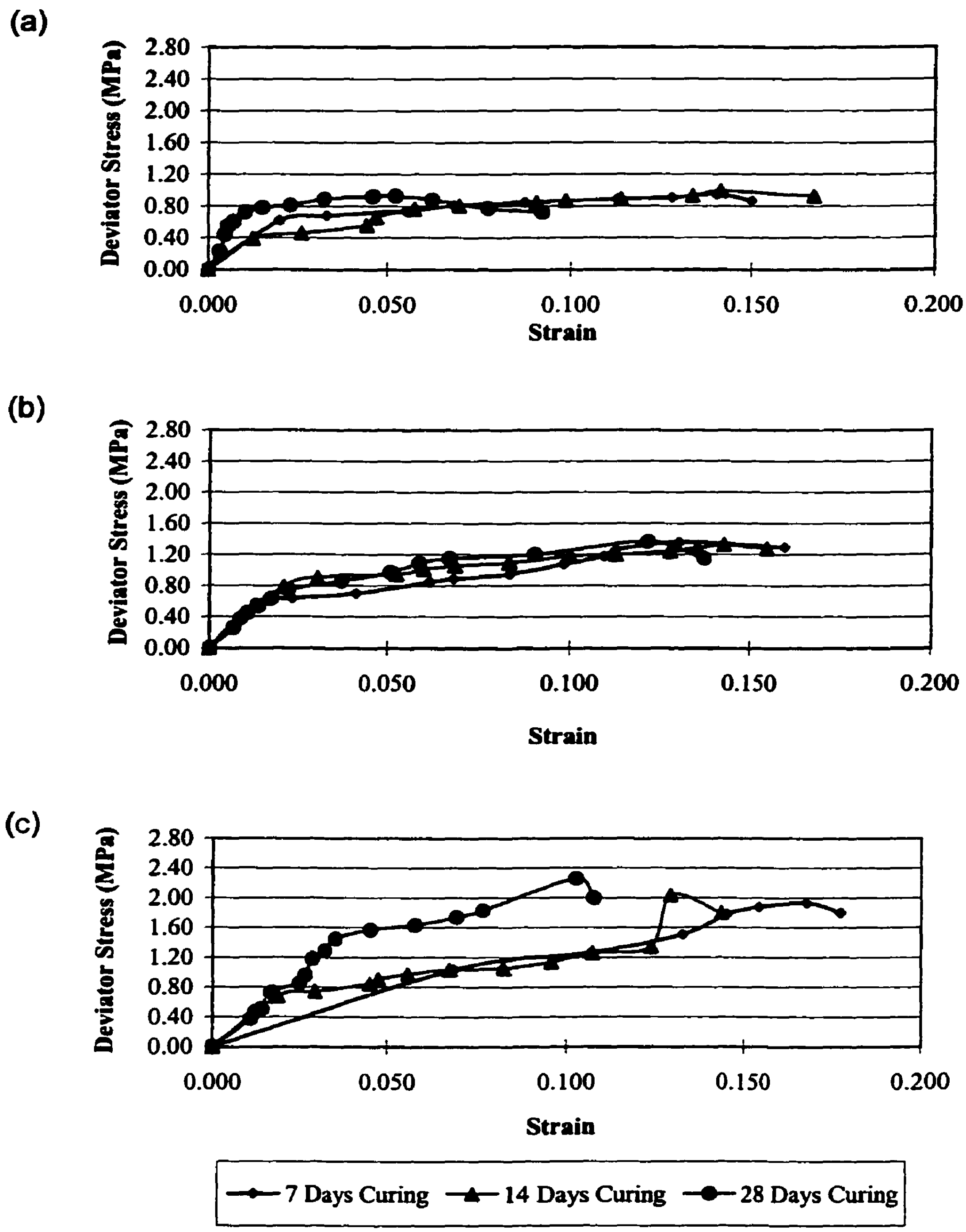


Figure 5.9 Deviator Stress vs. Strain Curves for Base Metal Tailings - Uncemented
(a) 207 kPa, (b) 414 kPa, (c) 828 kPa

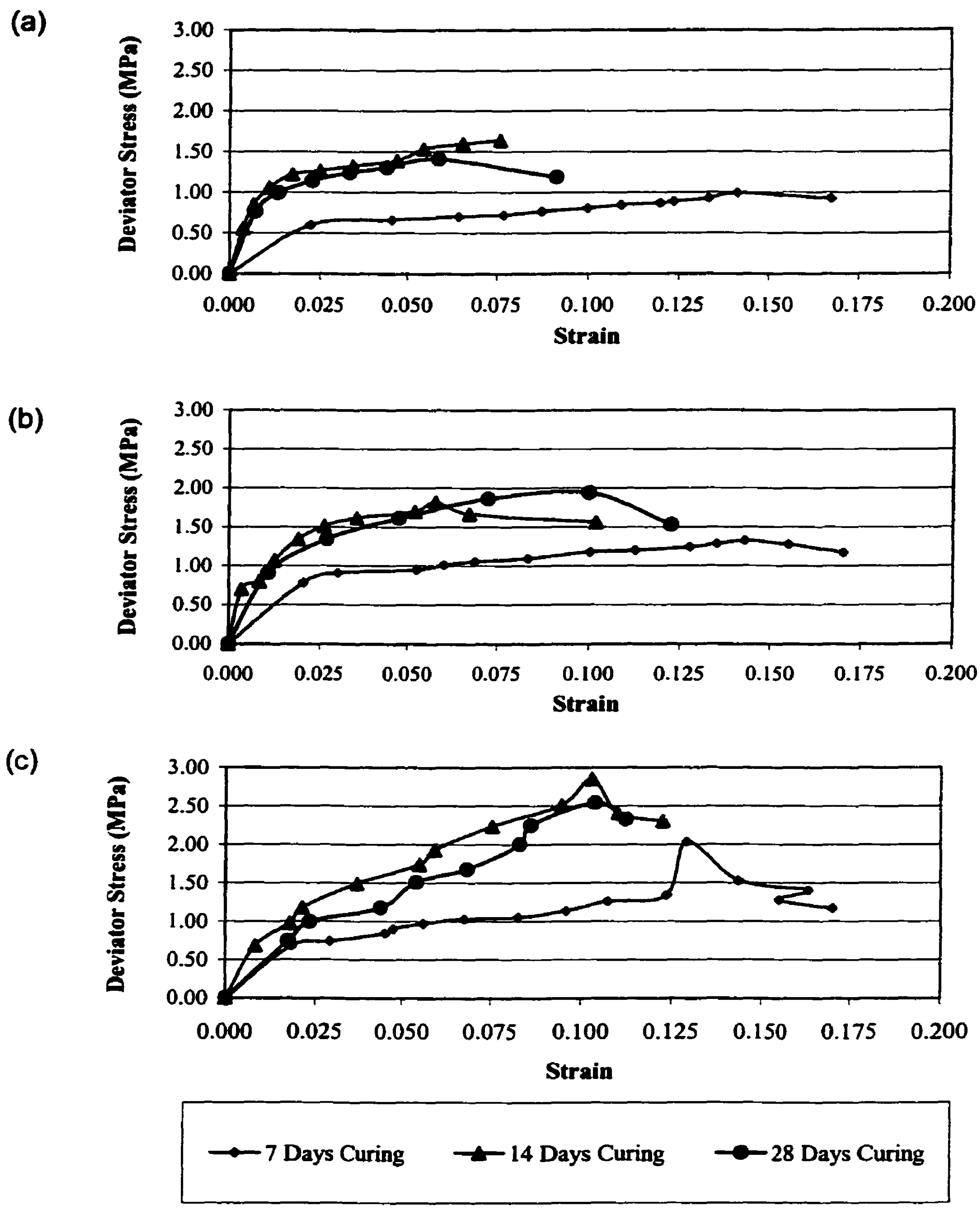


Figure 5.10 Deviator Stress vs. Strain Curves for Base Metal Tailings - 5 % PC
(a) 207 kPa, (b) 414 kPa, (c) 828 kPa

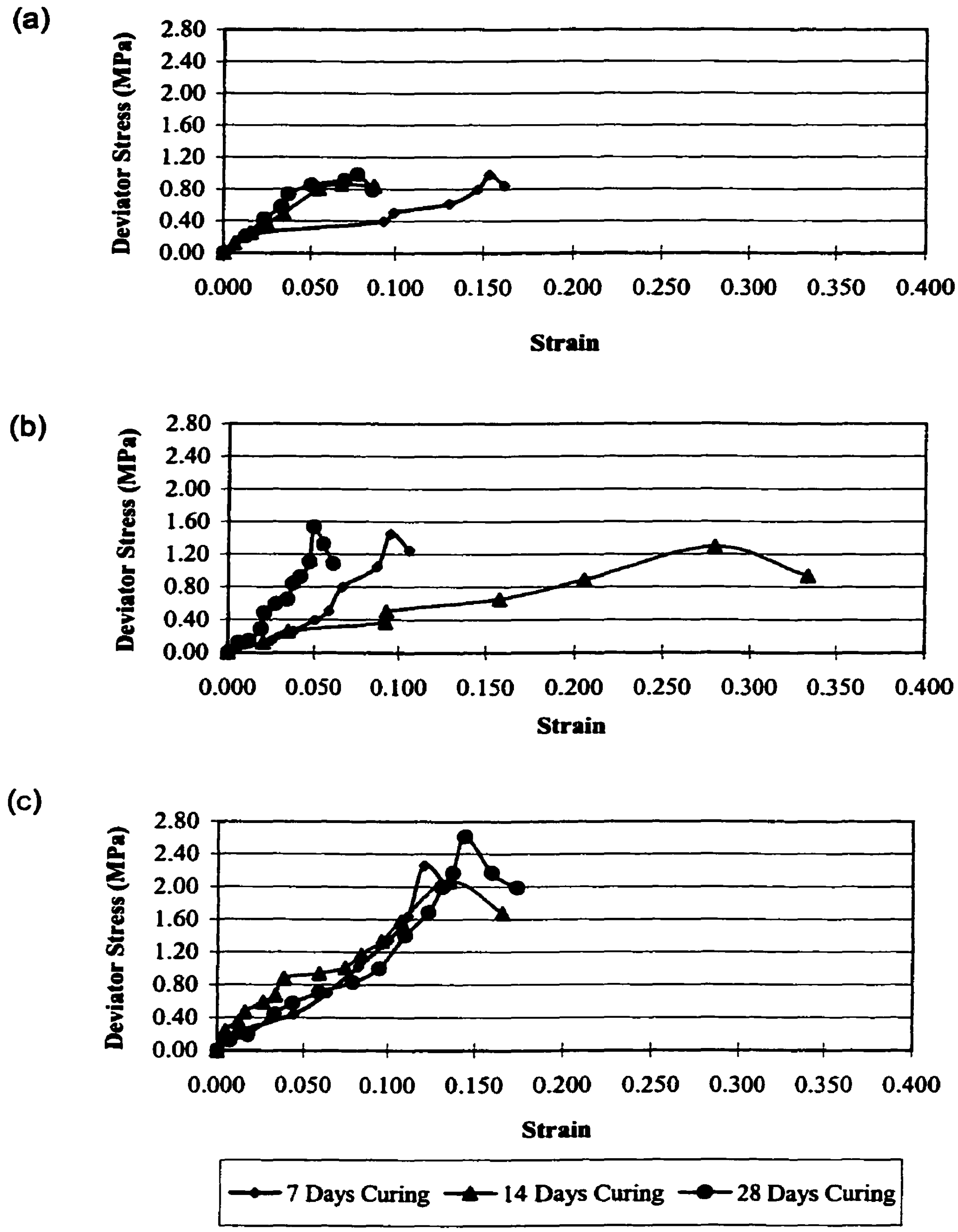


Figure 5.11 Deviator Stress vs. Strain Curves for Precious Metal Tailings - Uncemented

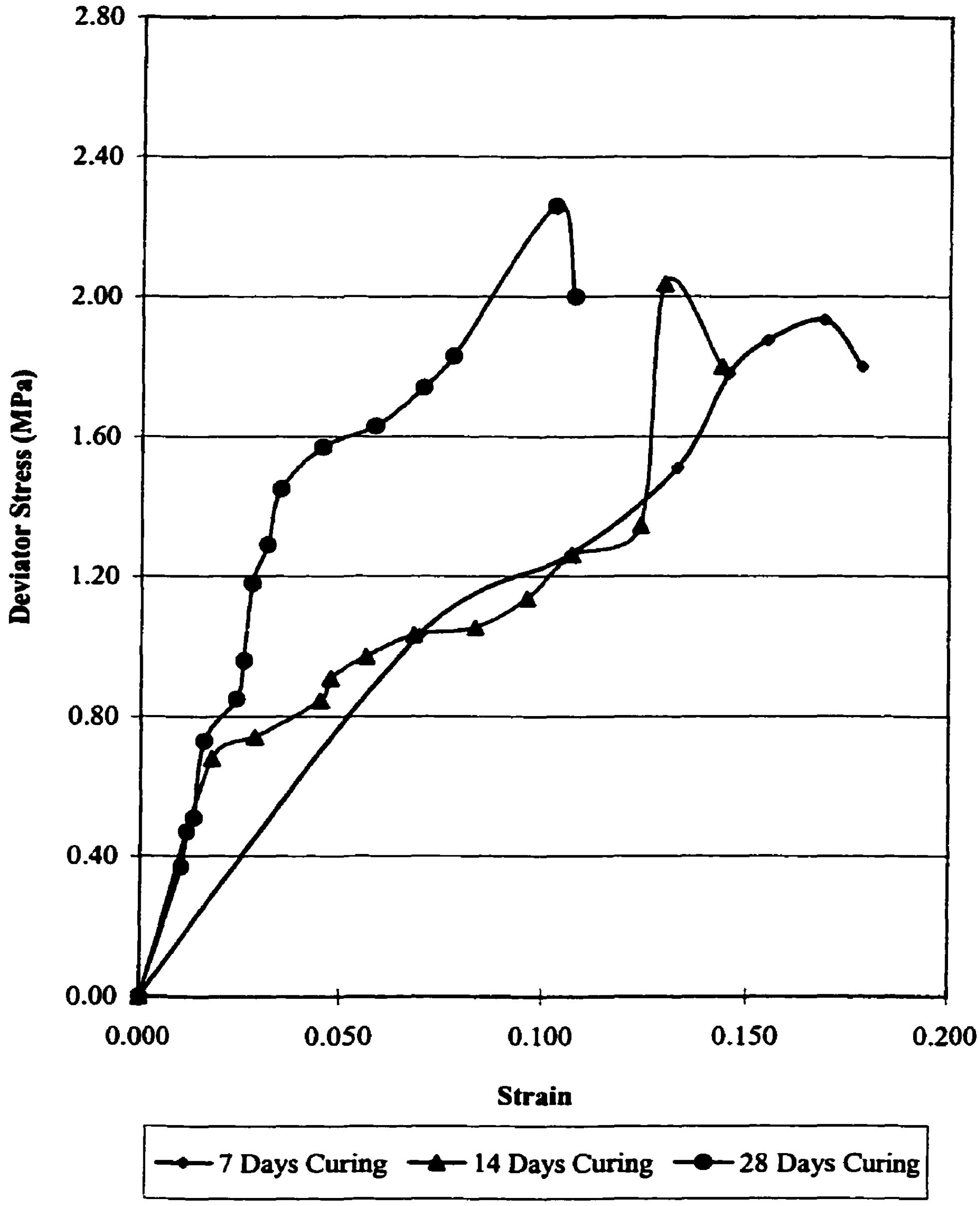


Figure 5.12 Typical Deviator Stress vs Strain Curves for Uncemented Base Metal Tailings with Curing Periods of 7, 14 and 28 Days (Confining Pressure = 828 kPa)

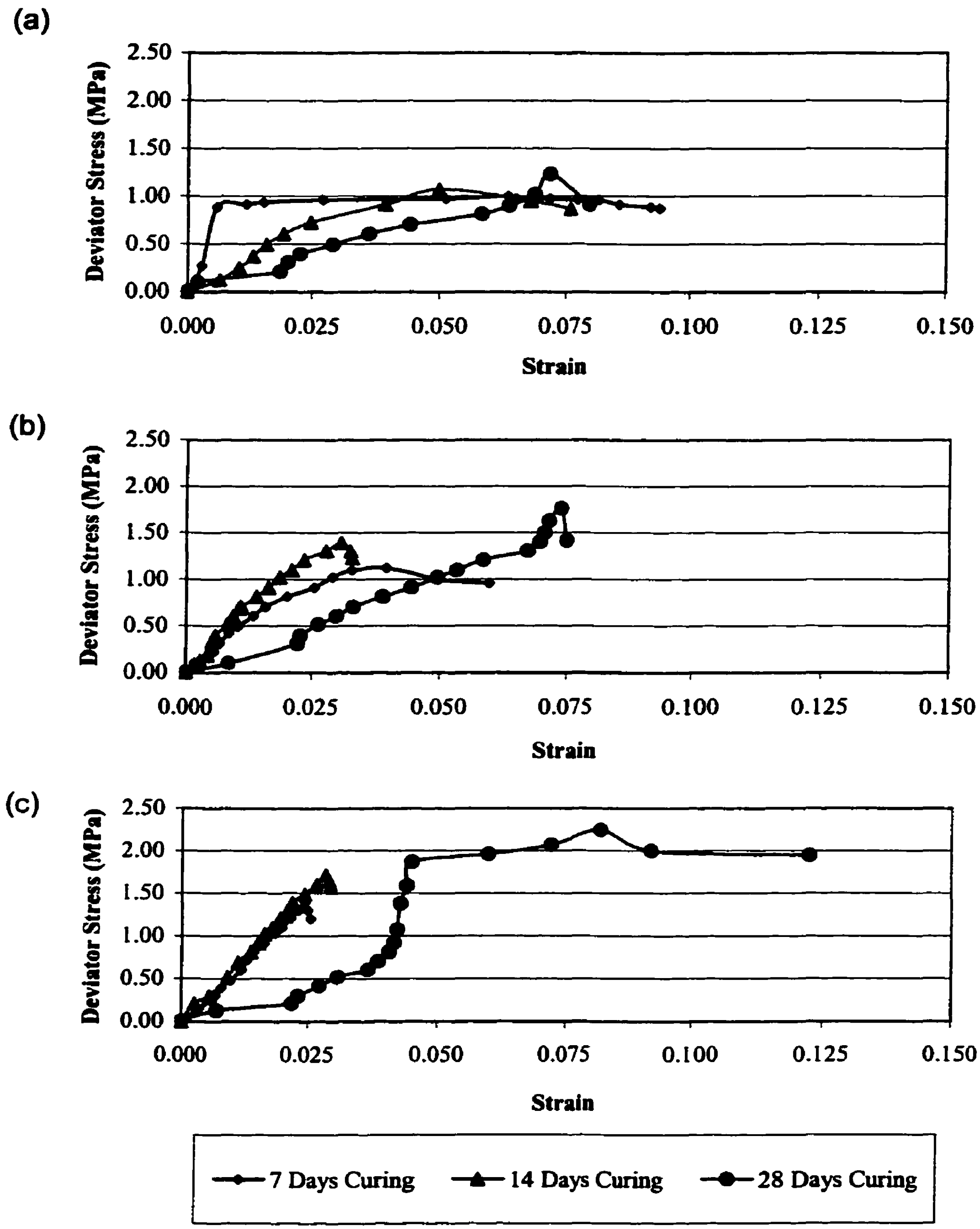


Figure 5.13 Deviator Stress vs. Strain Curves for Sand Paste Fill Samples - 5% PC
(a) 207 kPa, (b) 414 kPa, (c) 828 kPa

Table 5.2 Mohr's Circle Parameters for Uncemented Paste Fill

Sample I.D.	Curing Period	Confining Stress (σ_3) (kPa)	Deviator Stress $(\sigma_1 - \sigma_3)_f$ (kPa)	Avg. Modulus of Deformation (E) (MPa)	Friction Angle φ^0	Cohesion (C) (kPa)
Base Metal Total Tailings @ optimum moisture content 14%	7	207 414 828	951 1359 1933	16.5 29.6 22.2	26	207
	14	207 414 828	991 1324 2037		27	193
	28	207 414 828	930 1366 2058		28	172
Precious Metal Total Tailings @ optimum moisture content 17%	7	207 414 828	978 1453 2259	16.5 22.4 50.8	30	165
	14	207 414 828	862 1292 2072		30	138
	28	207 414 828	984 1536 2613		35	117

Considering that the uncemented triaxial compression test samples were prepared and tested at the optimum moisture contents of the tailings materials, it is conceivable that the observed increases in the yield point strengths were due to the combined effects of the optimum amount of moisture and apparent changes in density of the material with confining pressure (Singh, 1976; Head, 1980).

5.4.2.4 <u>Shear Strength Parameters</u>

The shear strength parameters have been determined in terms of the Mohr-Coulomb failure envelope as well as on the basis of a stress path (q-p) (Bishop and Henkel, 1962, Terzaghi and Peck, 1967) space. The Mohr's circles and the (q-p) plots are presented in Appendix C-2. The shear strength parameters for the uncemented test samples have been established in terms of curing periods. Typical stress path (q-p) plots have been presented in Figures 5.14 to 5.17. The results show that, the uncemented samples that were prepared and tested at the optimum moisture contents of the materials, displayed similar shear strength parameters as the cemented samples containing 5% binder (Table 5.3).

The results show that both the uncemented base metal and precious metal tailings samples developed similar apparent cohesion and friction angle values. As expected, the coarser base metal tailings samples were characterized by higher apparent cohesive strengths than the finely milled precious metal tailings. This is due to the fact that, the base metal tailings had better gradation than the precious metal tailings. Nicholson and Wayment (1967) found that, maximum density is achieved with a well graded material. The results of this study also indicate that the development of apparent cohesion was independent of binder content. This finding is contrary to conclusions reached by Ouellet et al., (1998) regarding paste backfill properties and behaviour. The differences between the two observations could be due to differences in the loading conditions.

The composite blended/tailings and sand samples developed comparable cohesion and frictional properties as the base metal tailings in spite of the high ultra fine (- 20μm) material composition (53%). The test results (Table 5.2), suggest however that the composite samples were the weakest of the investigated materials in the short term; this is also demonstrated by the (q-p) failure envelopes in Figure 5.16.

The stress path plots of the test results also indicate the relative stability of the paste fill materials in this study (Ouellet et al., 1998). In relative terms, the base metal tailings pastefill samples were the strongest.

5.4.2.5 Effects of Sand Addition on the Moisture-Density

The effects of blending tailings and sand mixtures on the density-moisture relationship of the composite products are best illustrated by Table 5.4 and Figure 5.18. The test data for the density-moisture tests is presented in Appendix C-3. The addition of sand to both the base metal tailings and precious metal tailings resulted in reductions in the respective moisture contents of the composite products.

The composite tailings/sand samples had the lowest optimum moisture contents compared to the straight base metal and precious metal tailings samples in this study.

The uncemented composite mixtures also produced lower void ratio and porosity values than the straight tailings materials (Table 5.4). The lower porosities were however, offset by reductions in the optimum moisture levels of the composite products. Pore pressure build up, and the saturation of soil-like materials generally occur beyond the optimum moisture content and after a reduction in the bulk density of the material has taken place (Head, 1980).

The degree of saturation has been adversely identified as an element which affects the liquefaction potential of paste fill masses (Hassani and Aref, 1988; Udd and Annor, 1993; Ouellet et al., 1998). In the simplest case, the degree of saturation for soil (Terzaghi and Peck, 1967) and backfill (Thomas et al; 1979) is a function of the moisture content and the volume of voids in the material. This may be expressed as shown in equation 5.2

$$S = \frac{Vw}{Vv} x100\% \qquad 5.2$$

Where S = degree of saturation

V_w = volume of water

V_v = volume of voids

It is evident from equation 5.2 that any reduction in the volume of voids for a fill mass would result in an increased degree of saturation of the material unless, there is a corresponding decrease in moisture content due to drainage. Figure 5.18 and Table 5.4 seem to support this fact.

The potential implication of the above observations is that, the blending of tailings and sand materials could result in the altering of the water content at which the uncemented composite fill product could become saturated in situ. For example, Ouellet et al. (1998) have reported that, the in situ moisture contents of the investigated paste fills were found to be closer to saturation levels, six months after the fill had been placed. Based on the results of this study, it is conceivable that some of the investigated fills by Ouellet et al. (1998), probably consisted of composite mixtures of tailings and sand. The observed high moisture contents could probably be due to changes in the void ratio and density of the fill masses due to the addition of sand. The reported high moisture levels could also have been due to other causes including, changes in void ratio and ground water infiltration.

Ouellet et al. (1998) have further reported that, uncemented samples were found to display purely frictional behaviour without cohesion. The uncemented samples that were tested at the optimum moisture contents of the respective materials in this study were found to display both cohesion and frictional behaviour (Table 5.3). It is conceivable that the difference in the two observations could be due to differences in test procedures and also, the moisture contents of the fill materials. Both loading history, (Bishop and Henkel, 1962; Terzaghi and Peck, 1967) and moisture content (Singh, 1976), have been shown to play important roles in the properties of geotechnical materials. The observed differences between the results of this study and other reported studies in the literature could be due to these differences.

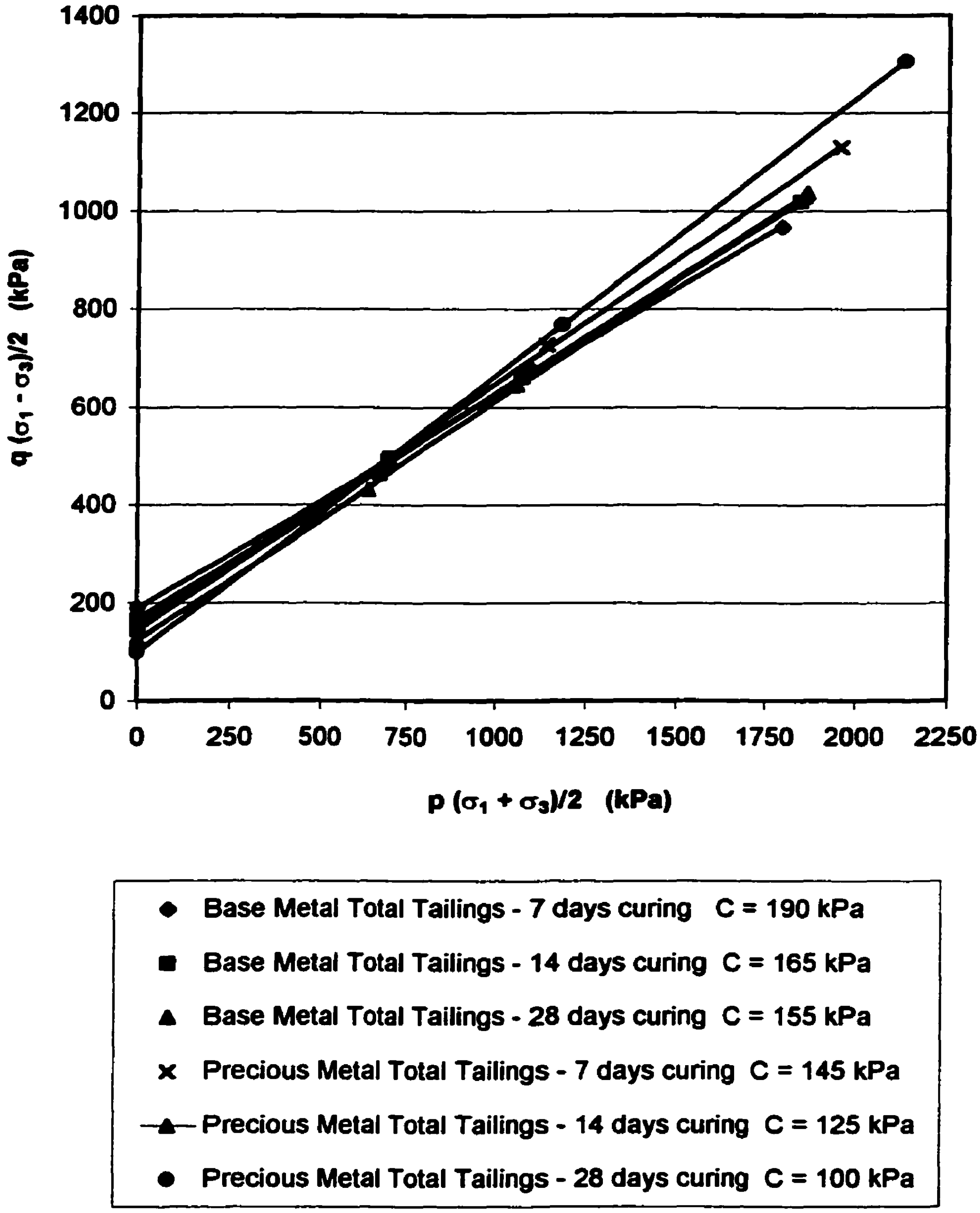


Figure 5.14 Strength Envelopes for Uncemented Pastefill Samples

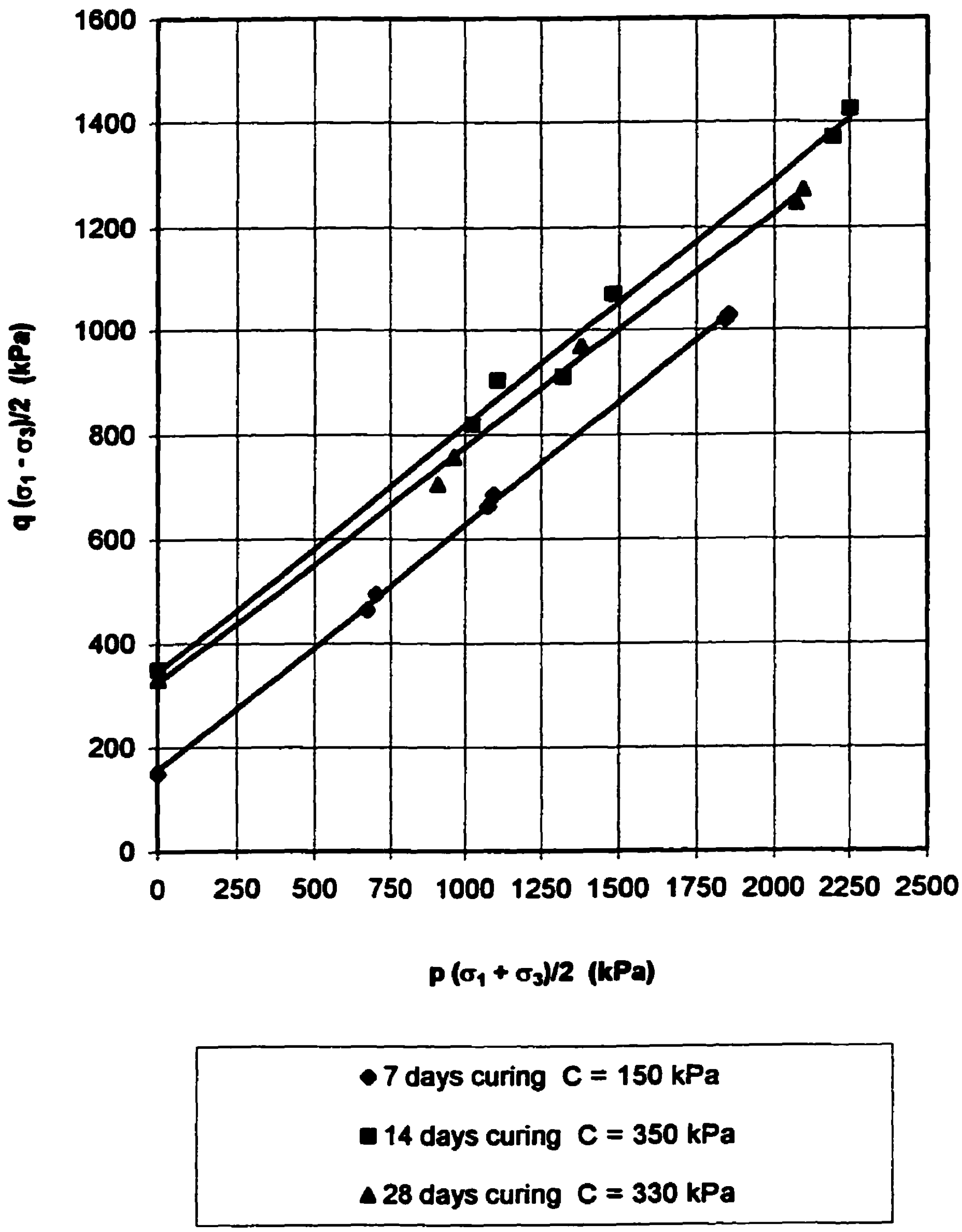


Figure 5.15 Strength Envelopes for Pastefill (Base Metal Tailings) Samples - 5 % Binder

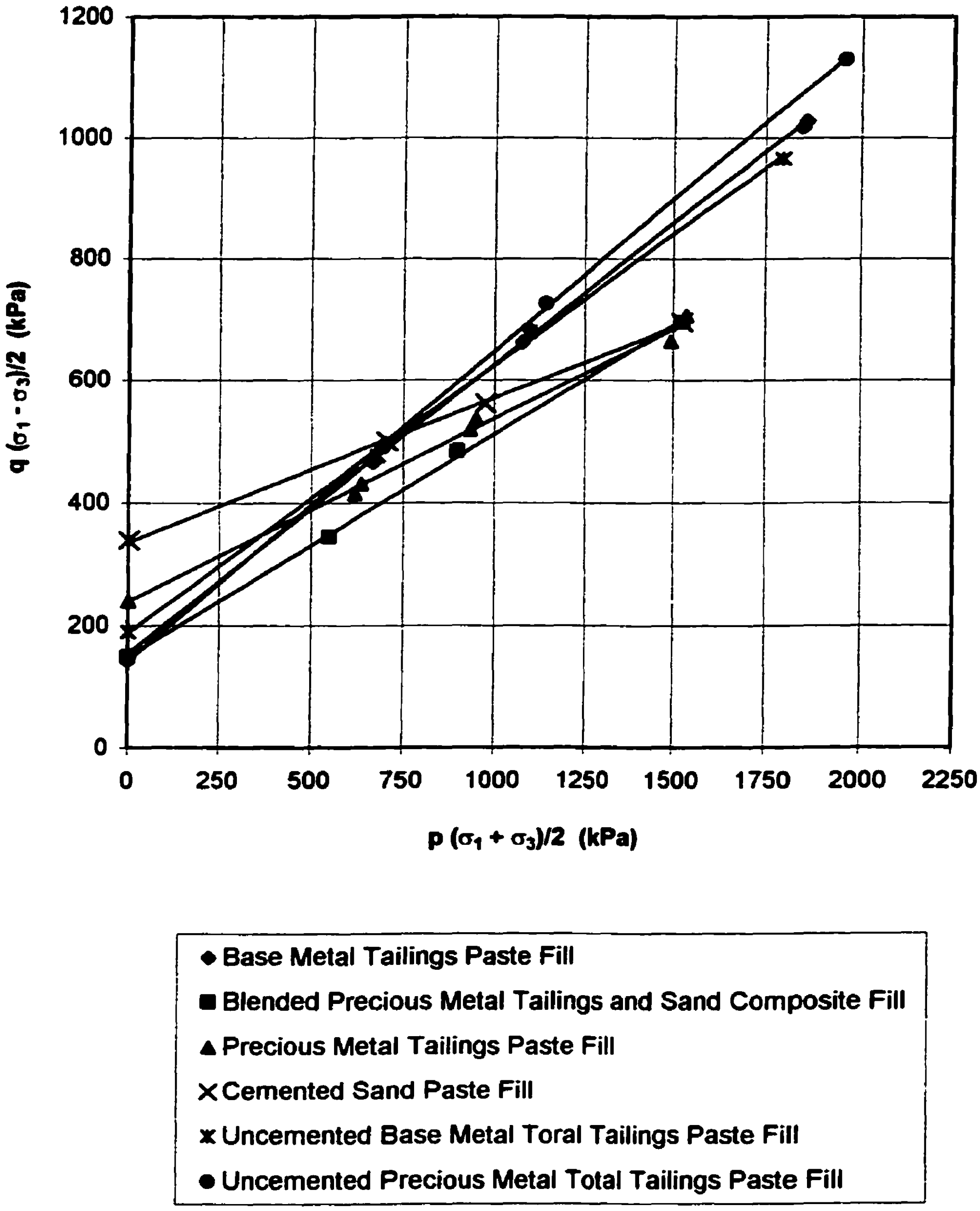


Figure 5.16 Strength Envelopes for Pastefill Samples 7 Days Curing

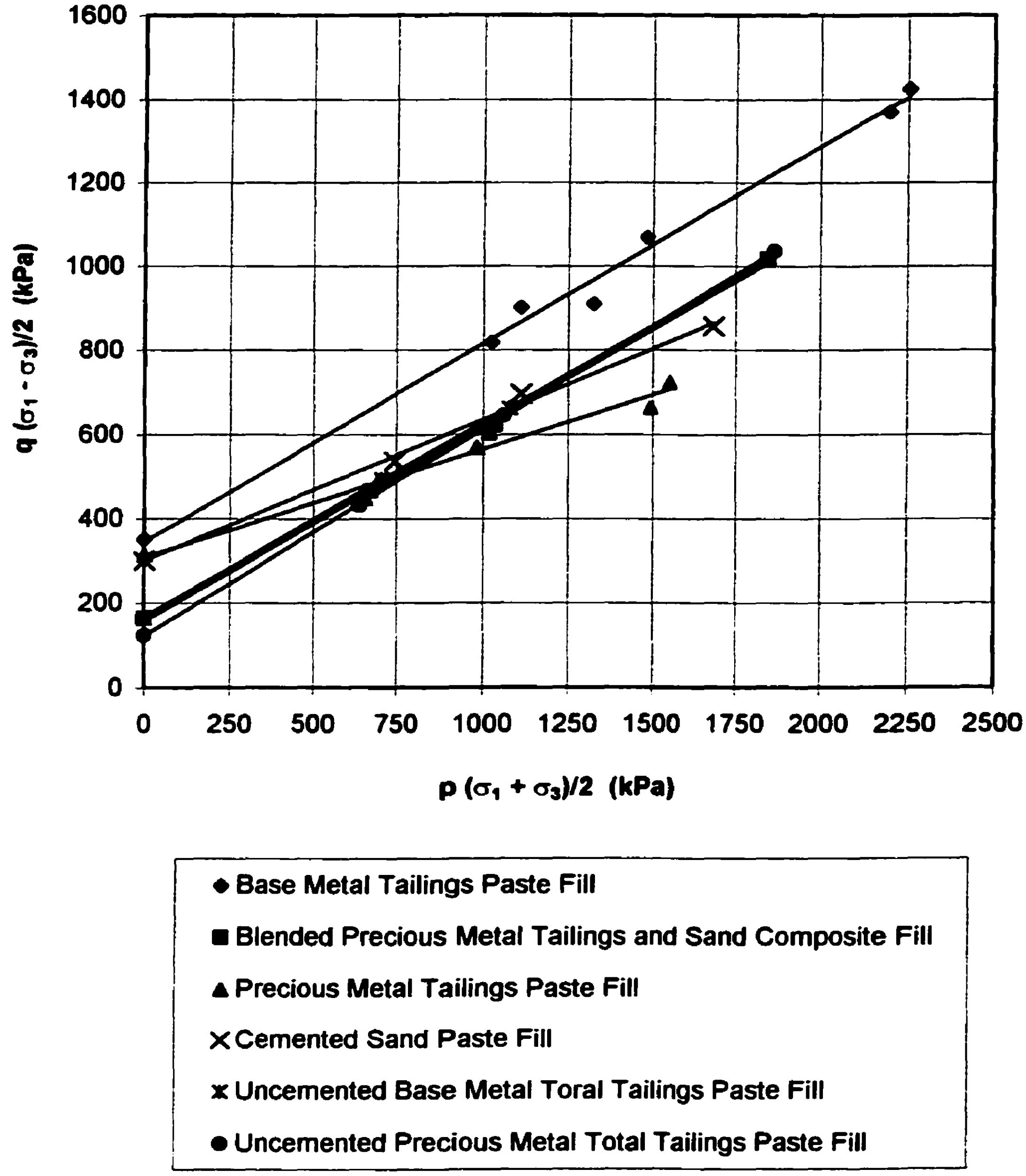


Figure 5.17 Strength Envelopes for Pastefill Samples 14 Days Curing

Table 5.3 Summary of Shear Strength Parameters

Pastefill Material	Gradation Parameters			Optimum Moisture Content (%)	Binder Content (%)	Curing Period (Days)	Apparent Cohesion		Internal Friction Angle $\phi°$	
	Cu	Cc	%-20μm				Mohr's Circle (kPa)	(q-p) (kPa)	Mohr's Circle	(q-p)
Precious Metal Tailings (Cemented)	5.3	0.26	42.3		5	7 14 28	250 302 297	240 313 282	17 16 19	22 18 23
Uncemented Precious Metal Tailings	5.3	0.26	42.3	16.8	0	7 14 28	165 138 117	145 125 100	30 30 35	34 35 38
Cemented Base Metal Total Tailings	5.9	1.80	43.5	-	5	7 14 28	193 310 324	150 350 330	27 31 28	23 22.5 22
Uncemented Base Metal Total Tailings	5.9	1.80	43.5	14.5	0	7 14 28	207 193 172	190 165 155	26 27 28	32 34 32
Alluvial Sand	4.2	1.15	26.5		5	7 14 28	337 317 229	340 300 180	14 20 27	12 17 25
Composite Blended Tailings Sand Paste	5.5	N/A	53.0	9.0	5	7 14 28	166 159 150	150 165 165	21 28 32	18 24 26

Table 5.4 (a) Composite Precious Metal Tailings and Sand

Tailings (%)	Sand (%)	Opt. M.C. (%)	Maximum Density (kg/m3)	Void Ratio	Porosity (%)	S.G.
100	0	10.3	2332	0.468	31.9	3.43
75	25	9.4	2276	0.504	33.5	-
65	35	9	22.86	0.5	33.3	-
25	75	8.5	21.63	0.585	36.9	-
5	95	7.84	2182	0.569	36.3	-
	100	7	-			2.6

Table 5.4 (b) Composite Blended Base Metal Tailings and Sand

Tailings (%)	Sand (%)	Opt. M.C. (%)	Maximum Density (kg/m3)	Void Ratio	Porosity (%)	S.G.
100	0	22	1547	0.91	47.6	2.96
80	20	16.7	1737	0.626	38.5	-
60	40	14.5	1843	0.484	32.6	-
50	50	11.4	1931	0.447	30.9	-
40	60	10.3	1955	0.424	29.8	-
25	75	10.4	1977	0.388	28	-
0	100	9.8	1710	0.541	35.1	2.64

* Opt M.C. = Optimum Moisture Content

The results of this part of the study suggest that, in order to produce a high strength composite-blended tailings/sand pastefill product, the constituent materials must possess similar optimum moisture levels. This could prevent the potential lowering of the moisture content at which the uncemented composite product could become saturated, and thus, improve the overall stability of the material against liquefaction

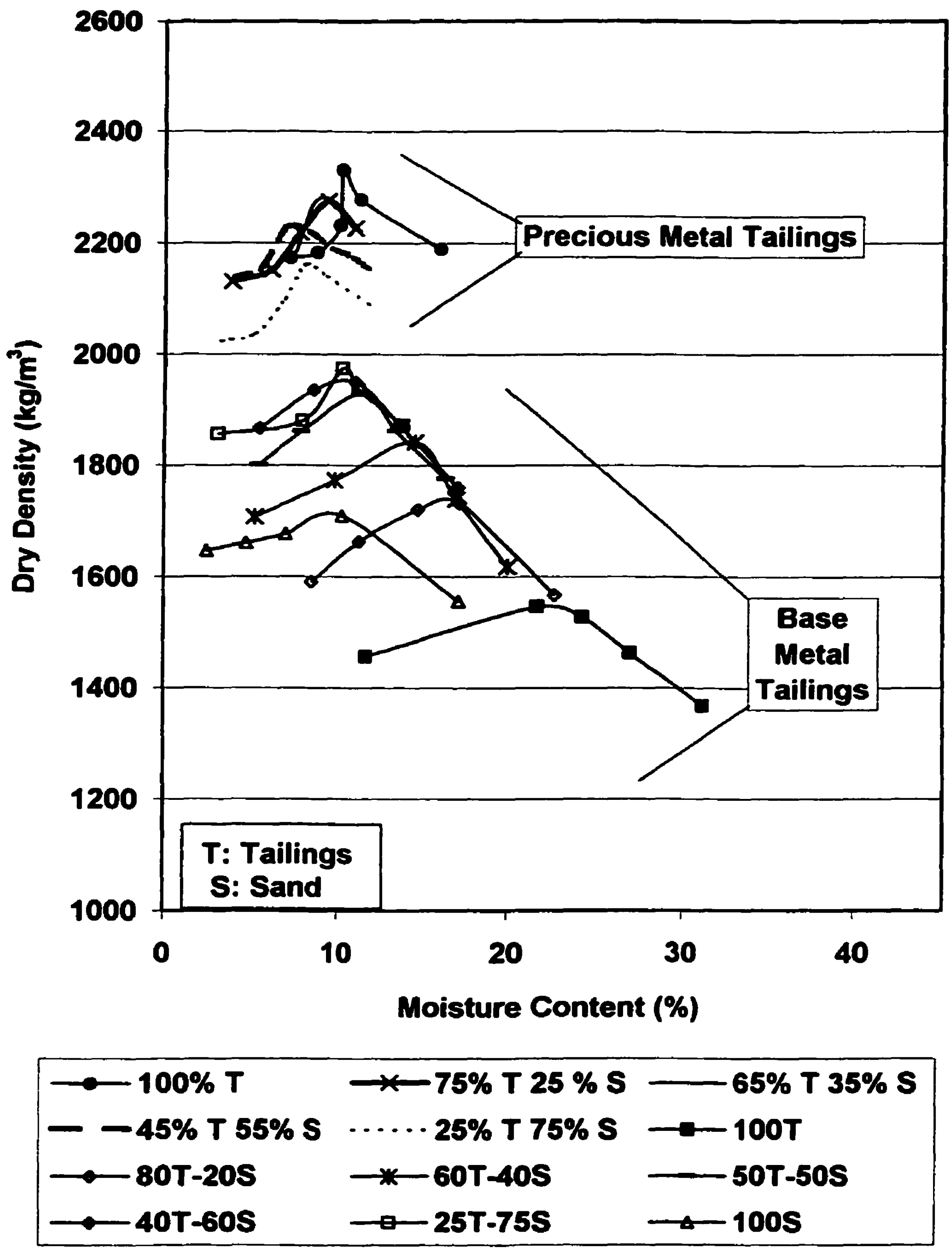


Figure 5.18 Changes in Optimum Moisture Contents for Composite (Blended - Tailings & Sand) Material

5.5 SUMMARY

1. Direct shear strength tests were carried out on uncemented fine and coarse tailings samples as a function of moisture content to investigate the potential effects of moisture content on shear strength parameters. The test results indicate that the shear strength of the uncemented fill samples increased at the optimum moisture content due to the apparent increase in density of the fill material.

2. The uncemented tailings samples developed apparent cohesion and internal friction angles that were similar to those of cemented pastefill samples. The strength gain of the uncemented pastefill samples were attributed to the apparent increases in the densities of the test samples. The strength gain occurred at the optimum moisture contents of the respective materials due to increased bulk density.

3. The results of this study suggest that the shear strength of uncemented pastefill, or low cement content masses could be enhanced, if the in situ moisture content of the fill mass could be maintained at or, near the optimum moisture level of the material.

4. The response of the high-density fill materials to applied load was examined as part of this study. The composite fill samples demonstrated a resilient behaviour. This was characterized by strain-hardening curves. The behaviour of the composite materials in this study suggests that they could be used as special products for tight filling and void reduction in mines.

5. The blending of fine tailings with medium grain sand, produced composite mixtures with reduced porosities and also, optimum moisture levels. This suggests that the saturation of the composite product could occur at relatively low moisture contents, if the bulk density of the fill material begins to decrease due to pore pressure build up. The liquefaction resistance of the fill would diminish if that were to happen.

6. The results of this part of the study suggest that the shear strength and saturation levels of uncemented composite (blended tailings and sand mixtures) could be enhanced through adequate drainage to reduce any pore pressure build up in the materials. Additionally, the shear strength of the composite (blended tailings/sand mixtures) may be improved through the careful selection of the blending (sand) material. The choice of the blending material or sand, should be such that it enhances the properties of the tailings in terms of void ratio and specific gravity. This would increase the density of the material and also, the liquefaction resistance of the composite product, in the event of the loss of cement bond strength in a stabilized fill mass.

CHAPTER 6

6. RELATIVE COMPARISON OF THE HIGH DENSITY BACKFILL PROPERTIES

6.1 General

The strength and deformation properties of geotechnical materials such as rocks, soils and backfills are known to be controlled among other things by void ratio, porosity and cement content (Adams and Williamson, 1923; Brace, 1965; Walsh, 1965; Lama and Vitukuri, 1978; Terzaghi and Peck, 1969; Thomas et al., 1979; Berry, 1980; Hedley, 1995). It has also been suggested by several investigators in the field of geotechnical research, that there is a direct relationship between the deformation modulus and compressive strength for earth materials including backfill (Swan, 1985; Hedley, 1995). In this regard, the relationship between compressive strength and deformation modulus can be used as a basis for comparison.

6.1.1 Relative Comparisons Based on the Mechanical Property Test Results

In this section the high-density fills in this study have been compared in terms of their relative changes in both compressive strength (σ_c) and deformation modulus (E), with void ratio (e), porosity (η) and binder or cement content (b). The properties of the studied high-density fills are summarized in Table 6.1. The mechanical property values for pastefill and cemented rockfill samples reported by Hedley (1995) have also been included in Table 6.1 for the purpose of comparison.

The test results are in general agreement with the published results in the literature. The properties of the straight tailings paste fill samples and the composite (blended tailings/sand) pastefill samples (Table 6.1), are in agreement with the reported results in the literature. There is also agreement particularly with backfill material properties data reported by the following: Vickery and Boldt (1989), Ross-Watt (1989) and Boldt et al., (1993).

The cemented rockfill properties compare favorably with those reported by Yu and Counter (1983, 1996, 1998), Yu (1989), Reschke (1993), Stone (1993), Hedley, (1995) and Farsangi (1996). Stone (1993), has reported on unconfined compressive strength values worldwide for 500mm diameter cemented rockfill samples. The data has been presented in Figure 6.1 and compared with the result of this study. There is a very good agreement between the two sets of data as shown in Figure 6.1. The unconfined compressive strength values for the CAP fill samples also compare favourably with the strength of 5:1 cemented "aggregate" fill samples reported by Arioglu (1983).

6.1.2 Scaling of Test Results to In Situ Conditions

With regard to the scaling of the test results to in situ conditions, Hedley (1995) has reported on the in situ deformation modulus for different types of backfill. The information has been summarized in Table 6.2 for the purpose of comparison with the test data on the large size samples (457mm by 914mm cylinders) from this study. For example, the established deformation modulus for the 457mm by 914mm rockfill cylinders ranged between 0.48 to 6.25 GPa. The mean deformation modulus value was 1.02 GPa. These values compare favourably with the in situ values that are reported by Hedley (1995) in Table 6.2 for various types of rockfill.

The deformation modulus values for the CAP fill samples in this study ranged between 140 and 1,180 MPa. The mean value was 613 MPa. This value compares favourably with the upper range of the in situ stiffness value of 550 MPa for a "pneumatically placed coarse gravel fill" containing 5 to 7% cement (Table 6.2).

The laboratory established values for the large size paste fill samples containing 6% cement ranged between 32 and 57 MPa with a mean value of 33 MPa. Hedley (1995) has reported on in situ stiffness values that ranged between 330 and 380 MPa for pastefill containing 10 per cent cement. The large difference between the results of this study and those reported by Hedley may be due to differences in binder type, and composition (6% for the laboratory tests in this study, compared to 10% for the in situ test reported in Table 6.2), curing conditions and time. Mineralogical compositions and chemistry of the tailings, and also size gradations of the tailings materials are the other factors which can account for the observed differences. McGuire (1978) has determined that, the deformation modulus of backfill mixtures containing slag are higher than mixes without slag. The binder used to stabilize the in situ placed fill was not given in Hedley's (1995) report. A possible explanation for the observed differences could be that it is difficult to simulate the in situ placement of backfill in laboratory scale tests. Additionally, the curing conditions for cement hydration in a mine stope could differ considerably from the conditions in the laboratory (Farsangi, 1996).

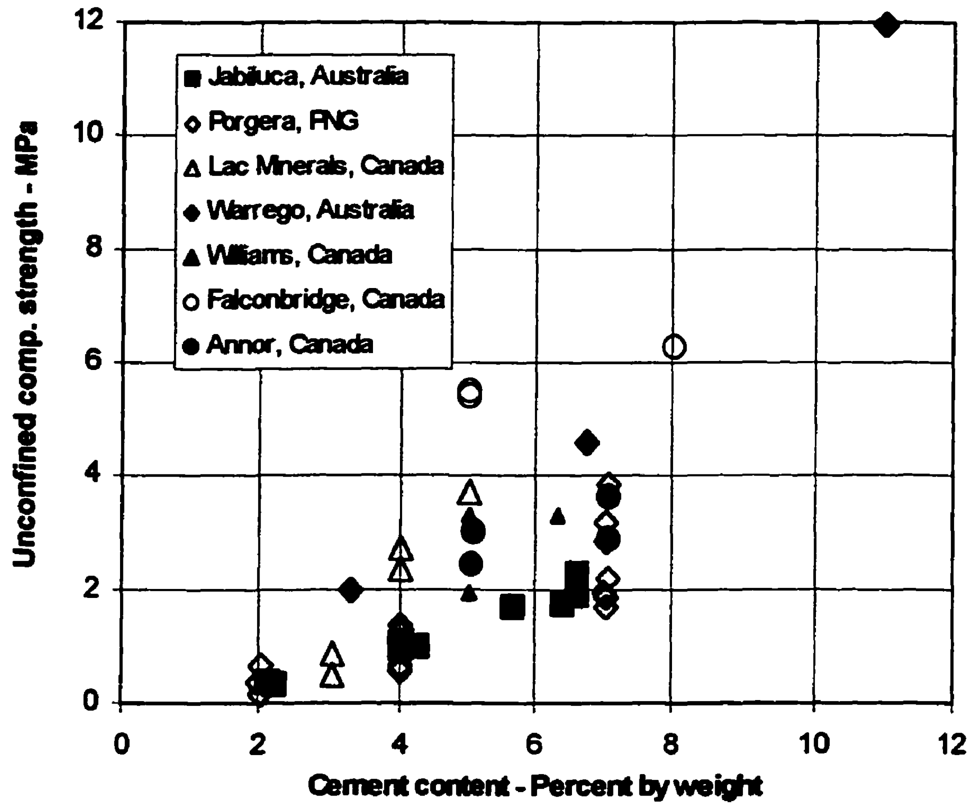


Figure 6.1 Unconfined compression test on cylinders of 500mm diameter for various CRF mixes worldwide (after Stone, 1993), and also data from this study.

Table 6.1 Comparison of Mechanical Properties of High-Density Fill Samples

Backfill Type	Bulk Density (kg/m^3)		Compressive Strength (MPa)		Deformation Modulus (GPa)		Void Ratio		Porosity (%)		Binder Content (%)
	Mean	Range	Mean	Range	Mean	Range	Mean	Range	Mean	Range	
Paste Fill (Base Metal Tailings)	2619	(2528 - 2715)	0.401	(0.510 - 0.830)	0.063	(0.011 - 0.200)	0.81	(0.45 - 1.17)	44.0	(31.2 - 53.8)	2-6
Paste Fill (Precious Metal Tailings)	1915	(1624 - 2694)	0.437	(0.055 - 2.515)	0.067	(0.001 - 0.797)	0.47	(0.36 - 0.67)	31.6	(25.6 - 31.6)	1-10
Sand	1957	(1610 - 2077)	1.226	(0.038 - 2.219)	0.055	(0.001 - 0.272)	0.40	(0.27 - 0.61)	28.2	(21.0 - 38.1)	1-10
Cemented Rockfill	2006	(1790 - 2430)	3.880	(0.820 - 10.880)	3.750	(0.090 - 24.610)	0.36	(0.13 - 0.53)	26.5	(11.7 - 34.7)	5-7
Composite (Blended - Tailings/Sand)	2017	(1852 - 3059)	0.741	(0.100 - 2.306)	0.150	(0.027 - 3.119)	0.38	(0.12 - 0.66)	27.0	(10.8 - 39.8)	3-6
Composite (CAP - Rockfill/ Tailings)	2406	(2049 - 2520)	0.916	(0.230 - 2.120)	0.613	(0.140 - 1.180)	0.25	(0.18 - 0.47)	19.8	(15.5 - 32.0)	3-5
Paste Fill*	1920	(1870 - 2010)	2.740	(2.430 - 3.630)	1.370	(0.880 - 1.710)	n/a	n/a	n/a	n/a	10
Cemented Rockfill*	2000	(1835 - 2161)	3.240	(2.000 - 5.630)	1.000	(0.480 - 2.630)	n/a	n/a	n/a	n/a	5-7

*(After Hedley, 1995)

Table 6.2 In-Situ Stiffness of Different Types of Backfill (after Hedley, 1995)

Type of fill	Cement Contents %	Reported No. Of Mines	Modulus MPa	Comments
Smelter Slag	0	1	2	
Tailings	0	3	8-48	
Alluvial Sand	0	1	55	
Tailings	3	2	13-227	
Tailings	5	1	13-227	
Tailings	6	1	60-530	
Alluvial Sand	10	1	245	
Tailings	15	1	90-570	
Coarse Gravel	5-7	1	100-550	pneumatically placed
Rockfill	5	1	6-140	cement grout percolation
Rockfill	5	1	650-1200	mechanical mixing, placement
Rockfill	5-7	1	100-400	poorly graded
Rockfill	5-7	1	400-1500	well graded
Rockfill	5-7	1	10-50	coarse aggregate, little cement
Rockfill	5-7	1	96-200	med. aggregate, some cement
Rockfill	5-7	1	1500-4000	fine aggregate, high cement

6.1.3 Relative Comparisons of the Studied High-Density Fill Strength and Deformation Modulus

Figure 6.2 shows the relative changes in the unconfined compressive strength and deformation modulus for the fill materials in this study. The comparison is based on the 152mm diameter by 300mm high size test sample. The results of this study indicate that the cemented rockfill achieved the highest strength and deformation modulus values when compared with the pastefill and the composite fill samples. The straight tailings pastefill and the composite (blended tailings and sand pastefill) samples had similar strength and deformation property values, based on the test conditions of this study. The composite-aggregate pastefill samples displayed strength and deformation properties that were intermediate between those of the pastefill material and the cemented rockfill. This behaviour was as expected because the composite fill is considered to be a derivative of cemented rockfill and paste backfill.

Caution must be exercised however, in using any of the data from this study for actual design of backfill systems. This is because of possible differences in quality control measures (Farsangi, 1996) and operating conditions that may exist at individual mine sites, and the laboratory conditions under which the reported data in this study were obtained.

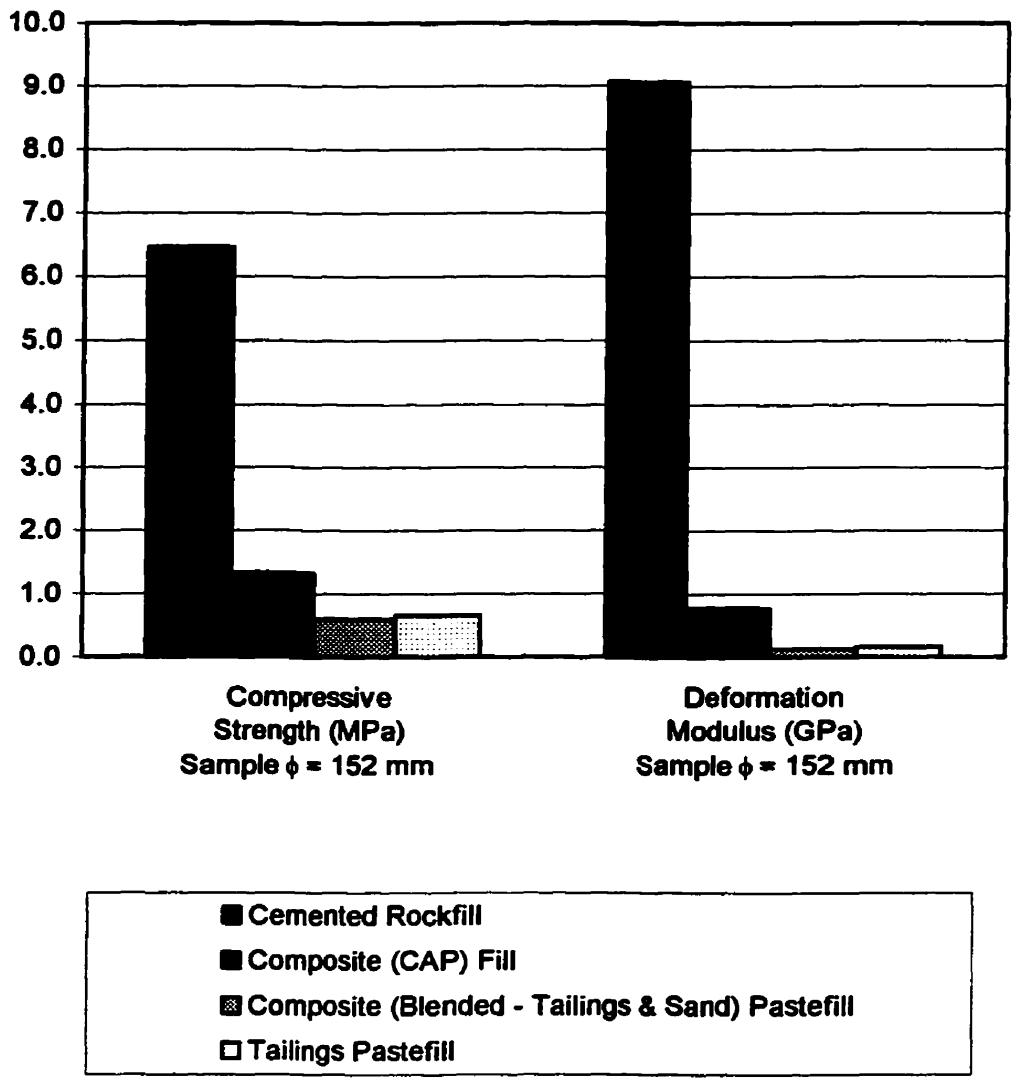


Figure 6.2 Relative Comparison of High-Density Fill Systems

6.2 RELATIONSHIPS BETWEEN COMPRESSIVE STRENGTH, DEFORMATION MODULUS AND VOID RATIO, POROSITY AND BINDER/POROSITY RATIO

An alternative method of comparing the high-density fill properties involved the examination of relative changes in the compressive strength (σ_c) and deformation modulus (E) with the following parameters; void ratio (e), porosity (η) and binder/porosity (b/η) ratio. The established relationships between compressive strength and void ratio, and deformation modulus and void ratio are provided in Figures 6.3 and 6.4, respectively. Similar relationships between compressive strength and porosity, and deformation modulus and porosity are also provided in Appendix D.

The results show that, in terms of void ratio and/or porosity, the composite fills are better fill systems than either the cemented rockfill or straight tailings paste fills. The results show that both the unconfined compressive strength and deformation modulus values increased with decreased void ratio, and porosity values for all the high-density fill systems in this study. The composite fills developed the lowest porosity and void ratio values, than either the straight tailings pastefill or the cemented rockfill samples.

The observed low porosity and void ratio values characteristics of the composite (blended tailings/sand pastefill and CAP fills), suggest that their application for ground support would permit a higher volume of fill material to be used for backfill preparation, compared to either the paste fill or the cemented rockfill. In this regard, a larger percentage of mining and milling wastes could be used for backfilling, when a composite fill system is selected at a mine site. This could also benefit the underground and the surface mine environments in terms of effective utilization of mining and mineral waste products for ground support.

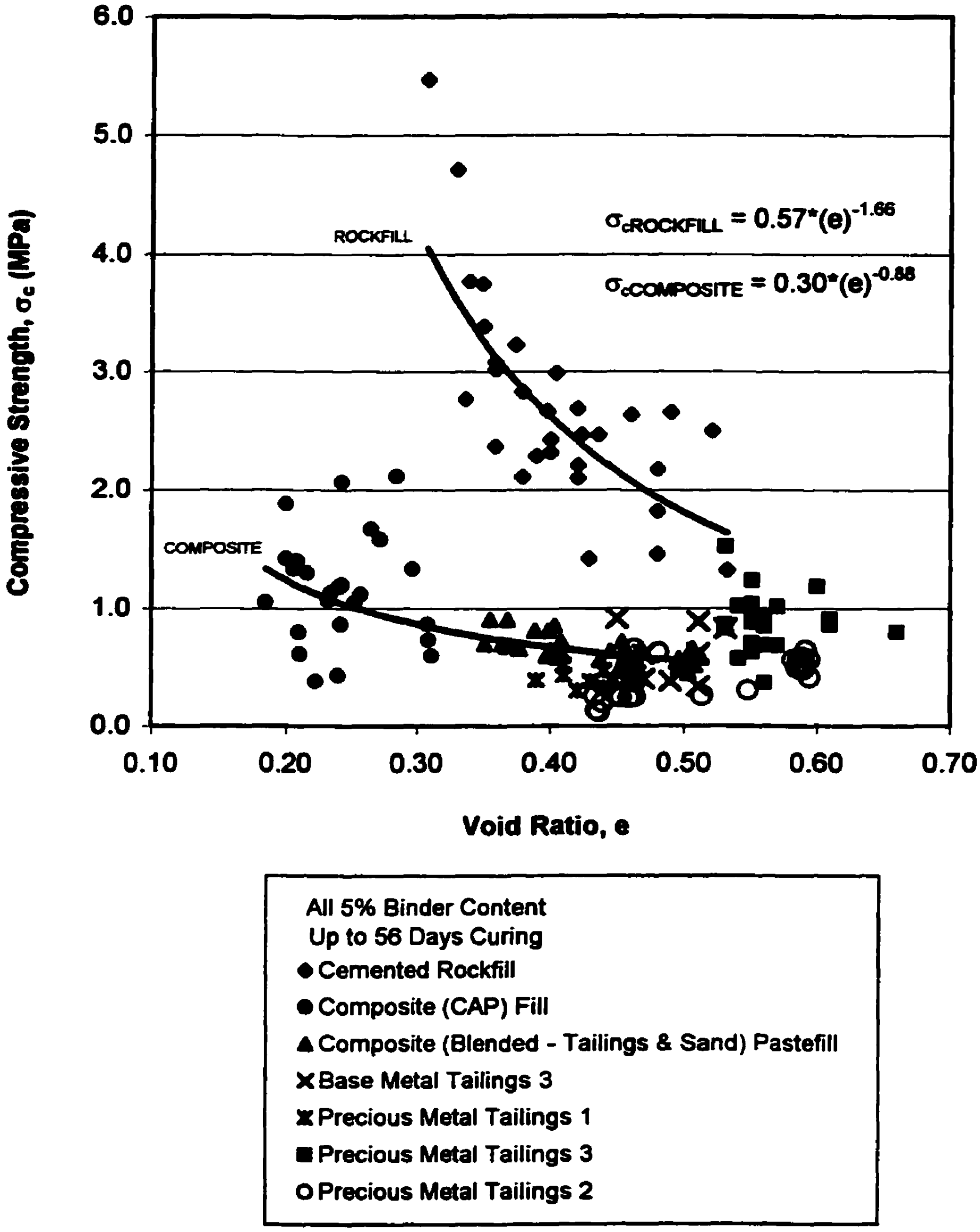


Figure 6.3 Relationship between Compressive Strength and Void Ratio for High-Density Fill Systems

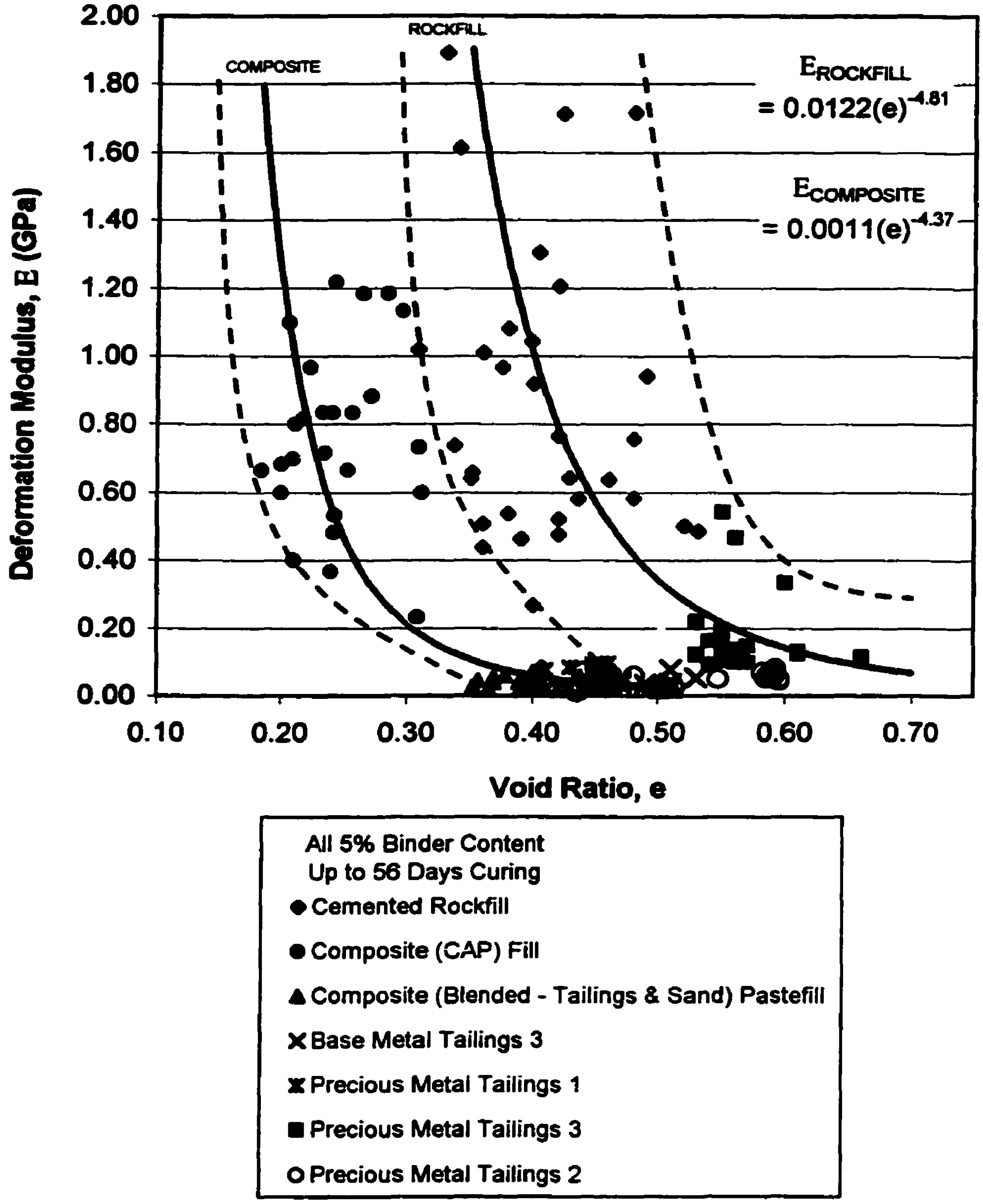


Figure 6.4 Relationship between Deformation Modulus and Void Ratio for High-Density Fill Systems

6.3 SOME PRACTICAL APPLICATIONS OF THE RESULTS OF THIS STUDY

Berry (1980) and Hedley (1995) have proposed that the strength of a backfill material is directly proportional to cement or binder content, and inversely proportion to porosity. Hedley has determined that the following relationship exists between compressive strength and cement/porosity ratio (b/η) based on a selected cemented rockfill and paste backfill data.

$$\sigma_c = 27\ (c/\eta)^{1.57} \qquad 6.1$$

Hedley (1995) has suggested further that, the application of the above relationship would allow the selection of the binder (("b") replaces "c" cement, in equation 6.1) or, porosity (η) required for achieving a certain level of stiffness to be estimated for various fill materials. The estimates can be made if relationships can be established between the compressive strength (σ_c), and the deformation modulus or the "stiffness" (E) of the materials.

Swan (1985) has also determined that the following relationship exists between the deformation modulus and the compressive strength of cemented rockfill.

$$E = 0.21(\sigma_c)^{1.44} \qquad 6.2$$

Similarly, Hedley (1995) has shown that the established relationship by Swan (1985) can be applied to both cemented rockfill and paste backfill properties. The above two relationships, (Equation 6.1 and 6.2) were established based on a limited laboratory test data.

The suggested method by Hedley (1995) has been used to develop relationships between compressive strength (σ_c) and binder/porosity (b/η) ratio for the cemented rockfill, composite fills and the straight tailings pastefill in this study. Similar relationships have also been developed between the compressive strength (σ_c), and deformation modulus (E) for each of the three types of the high-density fills in this study. The relationships between compressive

compressive strength and binder porosity ratio were established by plotting the respective parameters as indicated in Figures 6.5 to 6.7 using the laboratory test results from this study. The established relationships between compressive strength and binder/porosity ratio for the various fills are as follows:

$$\sigma_{c\ pastefill} = 10.3\ (b/\eta)^{1.30} \qquad 6.3$$

$$\sigma_{c\ rockfill} = 13.2\ (b/\eta)^{0.90} \qquad 6.4$$

$$\sigma_{c\ composite\ fills} = 7.2\ (b/\eta)^{1.39} \qquad 6.5$$

The information from Figures 6.5 to 6.7 has also been presented collectively in Figure 6.8 which shows that in relative terms, the composite fills were the most efficient backfill systems in this study in terms of binder utilization; this is because of the lower porosity values. The relationship between (σ_c) and (b/η) for the combined high-density fill properties in this study has been provided in Figure 6.9 in order to permit comparison with other published data. The established relationship between (σ_c) and (b/η) for the combined fills is as follows:

$$\sigma_{c\ combined\ fills} = 27.5\ (b/\eta)^{1.83} \qquad 6.6$$

Similarly, relationships have been found between compressive strength (σ_c) and deformation modulus (E) for the individual high-density fills in this study as shown in Figures 6.10 to 6.12 as follows:

$$E_{paste\ fill} = 0.11\ (\sigma_c)^{0.99} \qquad 6.7$$

$$E_{rockfill} = 0.35\ (\sigma_c)^{1.16} \qquad 6.8$$

$$E_{composite\ fills} = 0.21\ (\sigma_c)^{1.28} \qquad 6.9$$

Figure 6.13 also shows that in relative terms, the cemented rockfill was the strongest fill system among the three high density fills in this study. It is interesting to note that most of the composite (CAP) fill data in Figure 6.13 is closely identified with those of the cemented rockfill. This suggests that the two materials have similar properties. The trend line of the composite-aggregate fill lies between those of the cemented rockfill and the straight tailings paste fill. Figure (6.13) also supports the initial assertion of this study that fundamentally, composite fill properties can be derived from those of cemented rockfill and paste backfill.

A relationship for the combined high-density fill properties is also shown in Figure 6.14. The established relationship for the combined high-density fills was as follows:

$$E_{\text{high-density fills}} = 0.162\,(\sigma_c)^{1.29} \qquad 6.10$$

A plot of the data reported by Hedley (1995) and the data from this study have also been provided in Figures 6.15 and 6.16 respectively in terms of (σ_c) vs. (b/η) and (E) vs. (σ_c) plots. Both of figures 6.15 and 6.16 show that there is a very good agreement between the two sets of data.

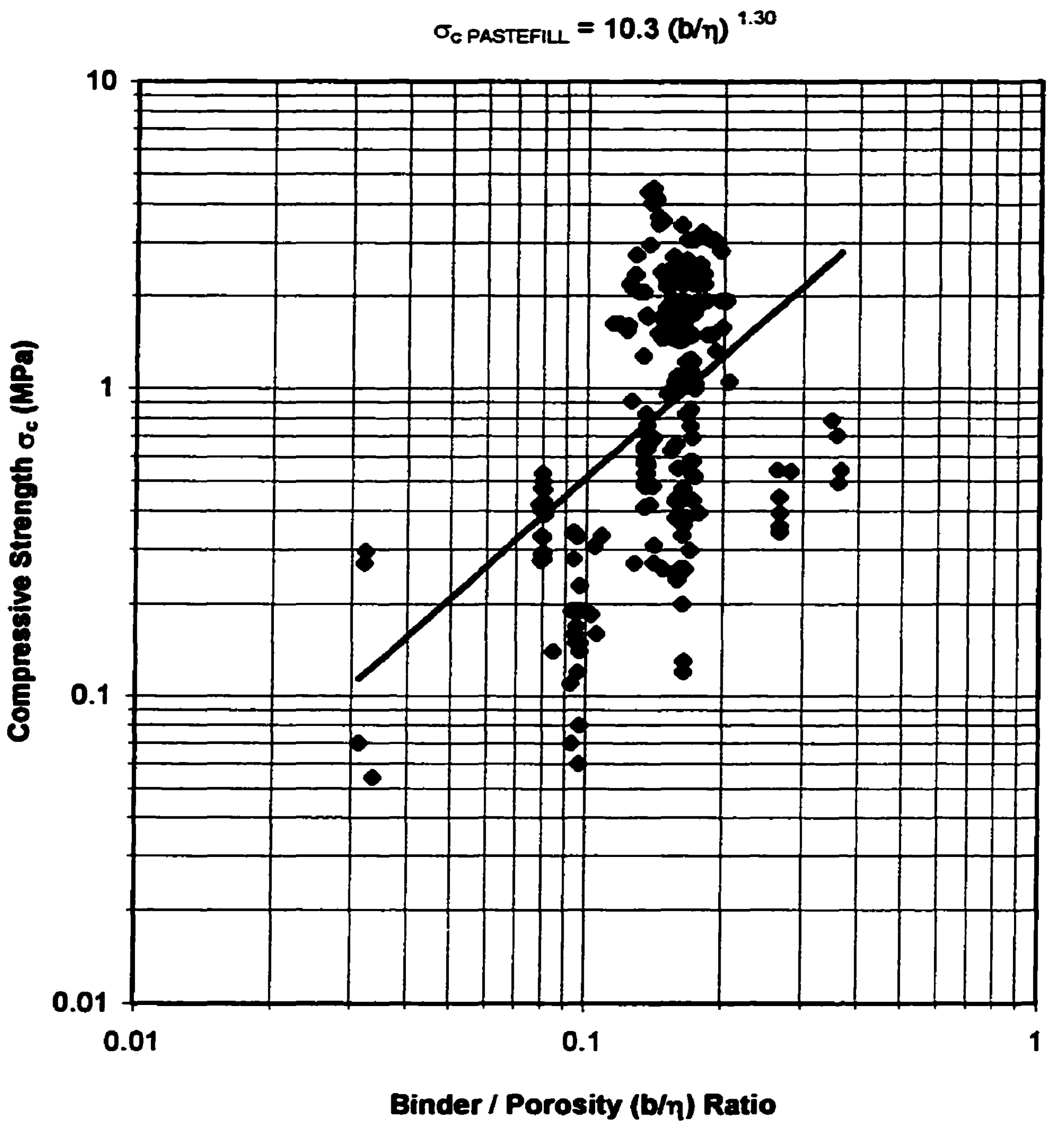


Figure 6.5 Compressive Strength as a Function of Binder/Porosity (b/η) Ratio for Tailings Pastefill

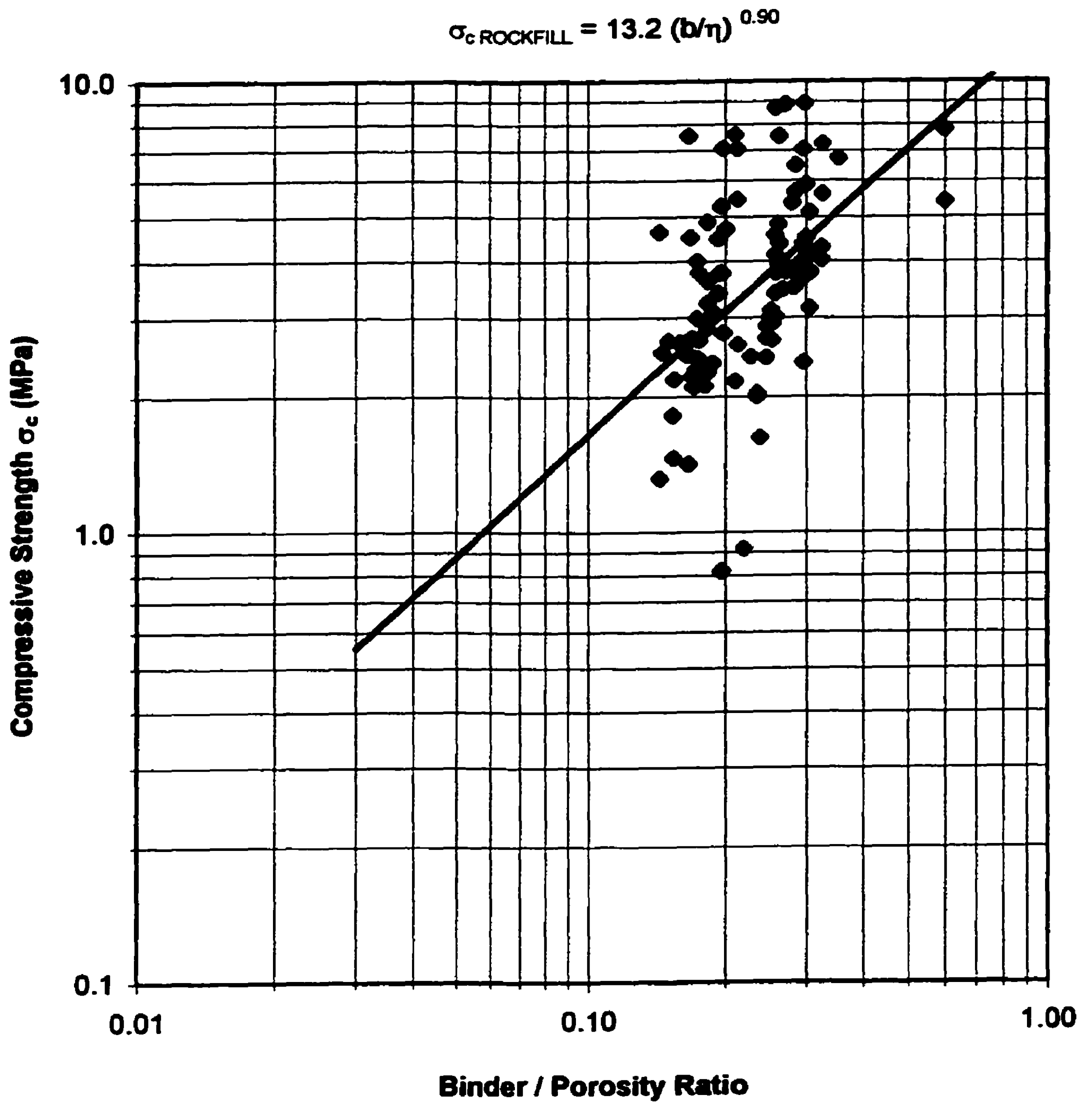


Figure 6.6 Compressive Strength as a Function of Binder/Porosity (b/η) Ratio for Cemented Rockfill

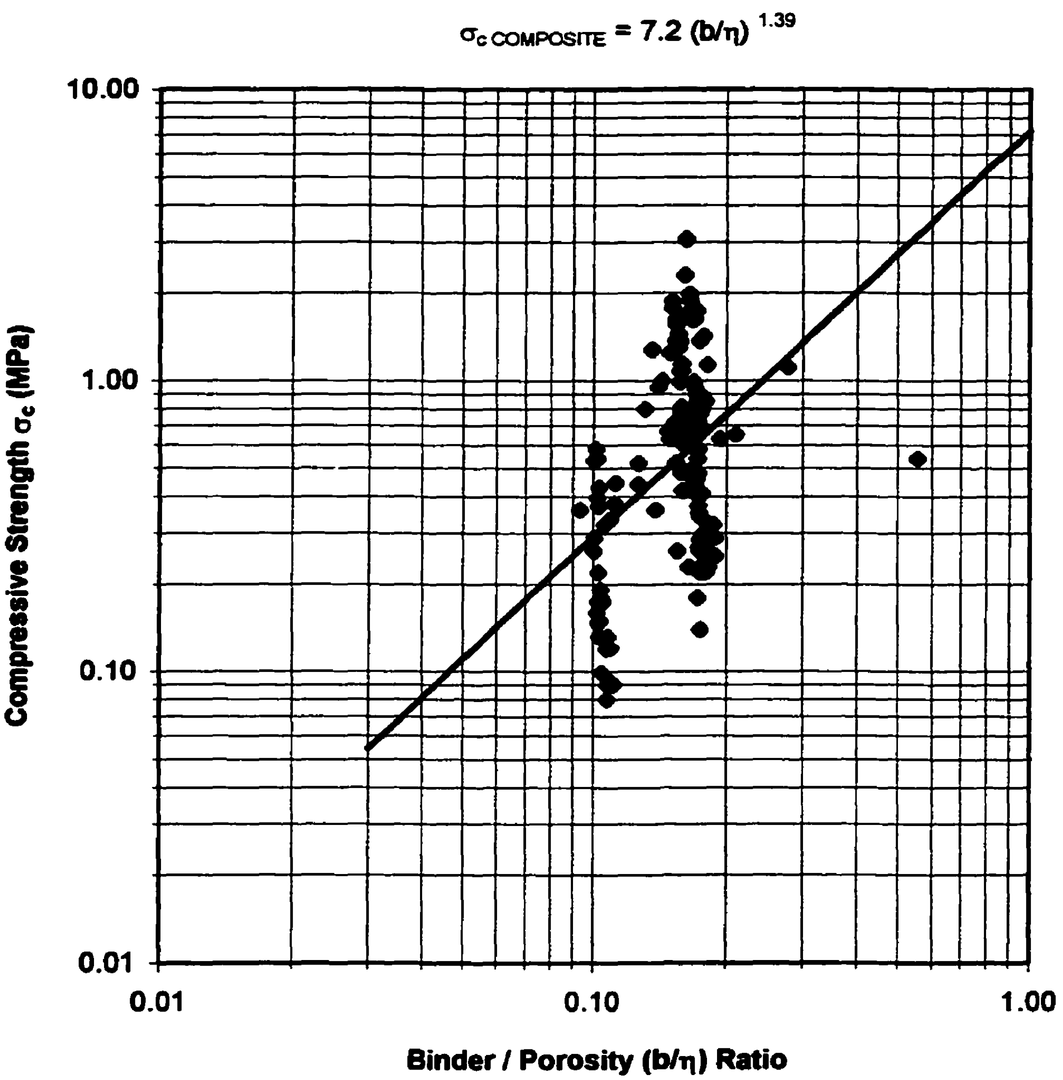


Figure 6.7 Compressive Strength as a Function of Binder/Porosity (b/η) Ratio for Composite Fill

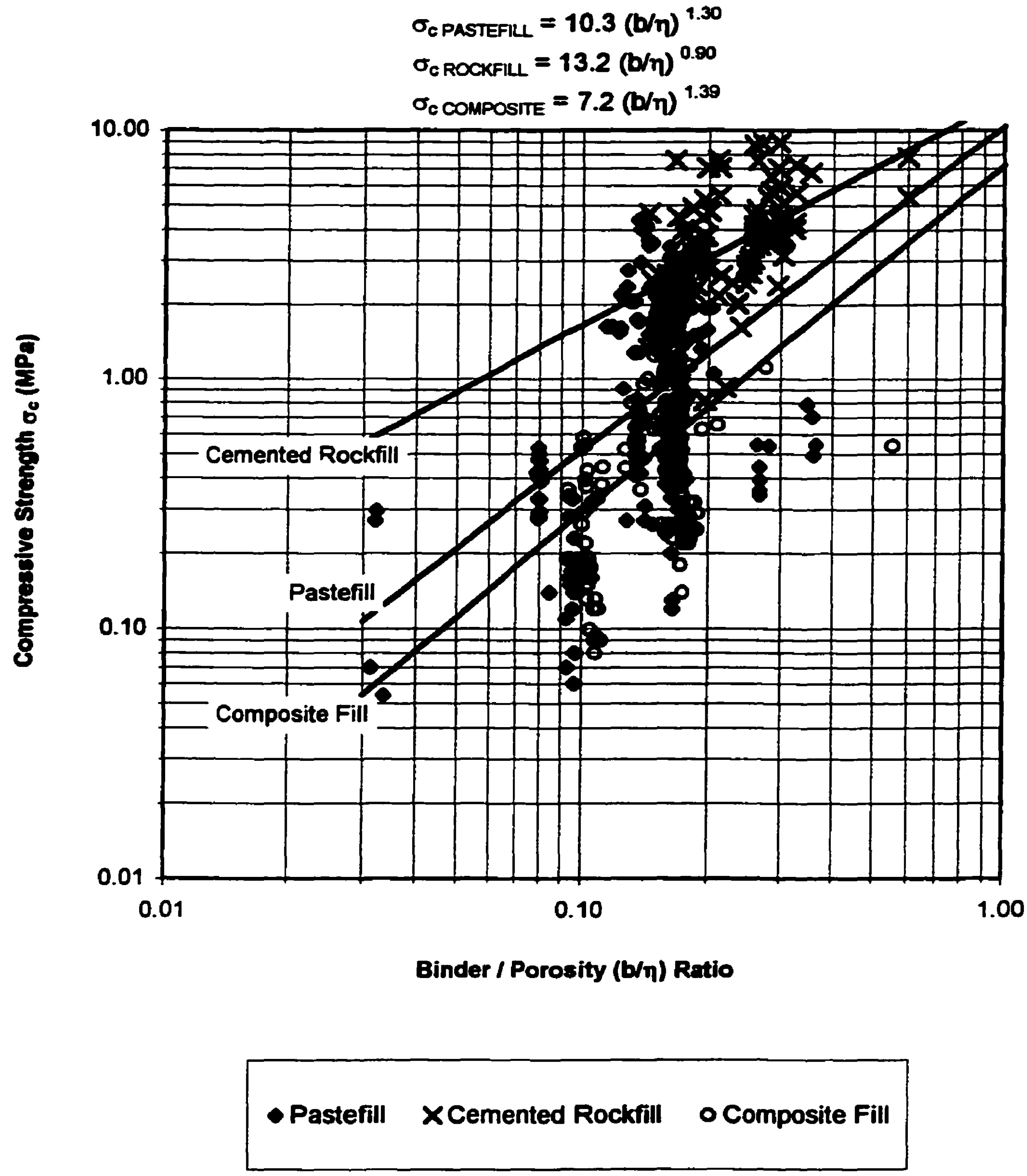


Figure 6.8 Compressive Strength as a Function of Binder/Porosity (b/η) Ratio

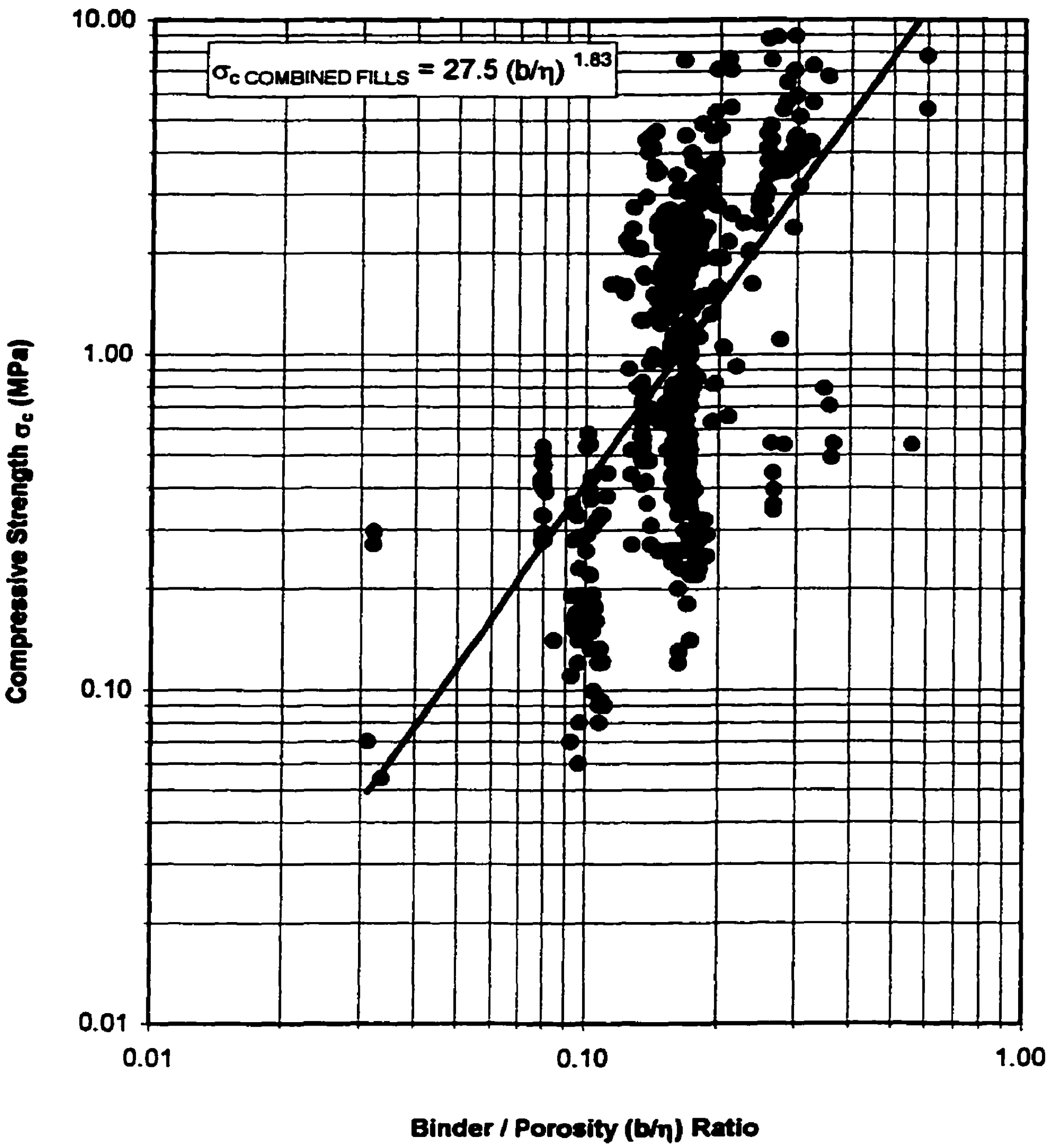


Figure 6.9 Compressive Strength as a Function of Binder/Porosity Ratio for Combined High-Density Fills

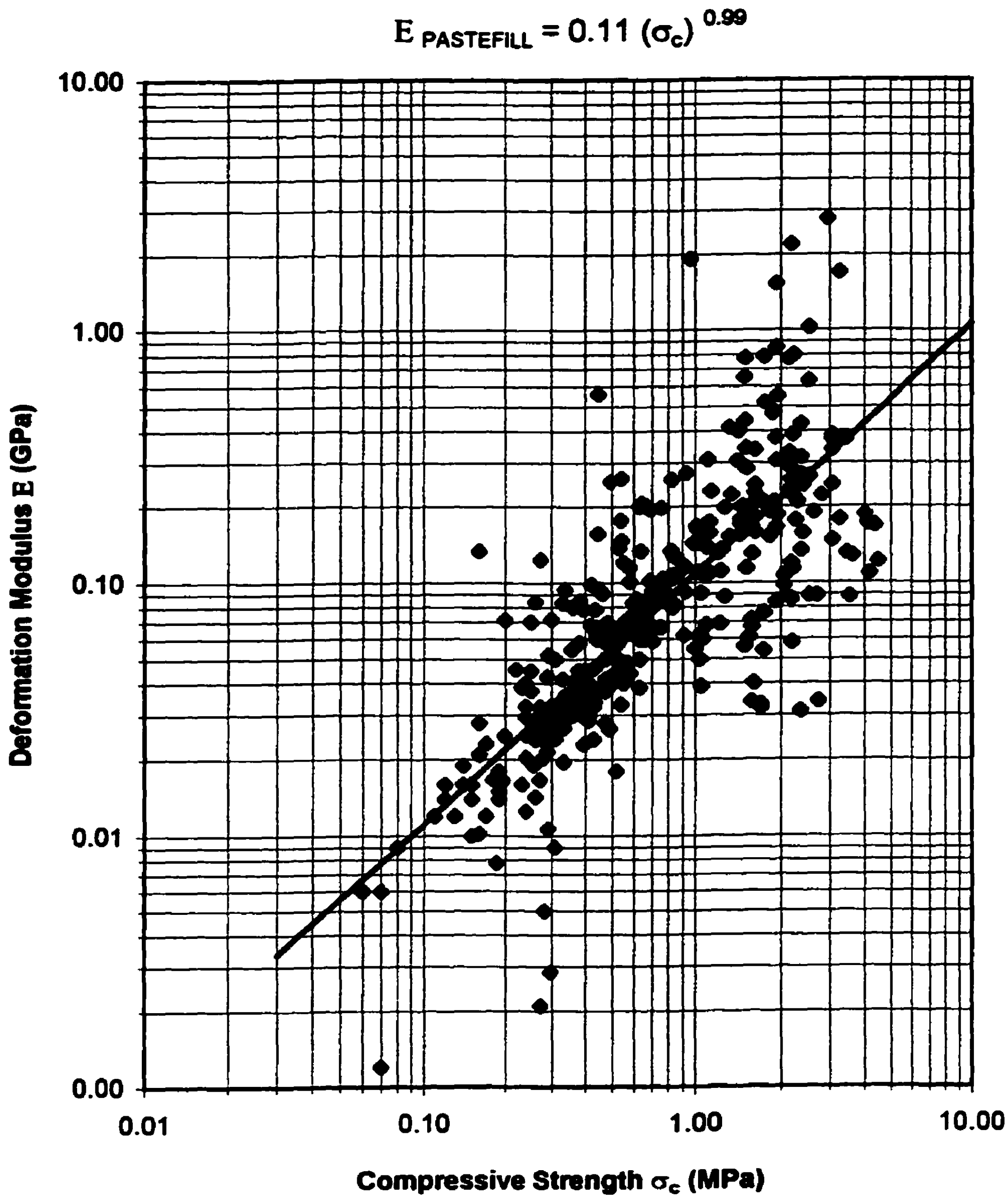


Figure 6.10 Relationship Between Deformation Modulus and Compressive Strength for Tailings Pastefill

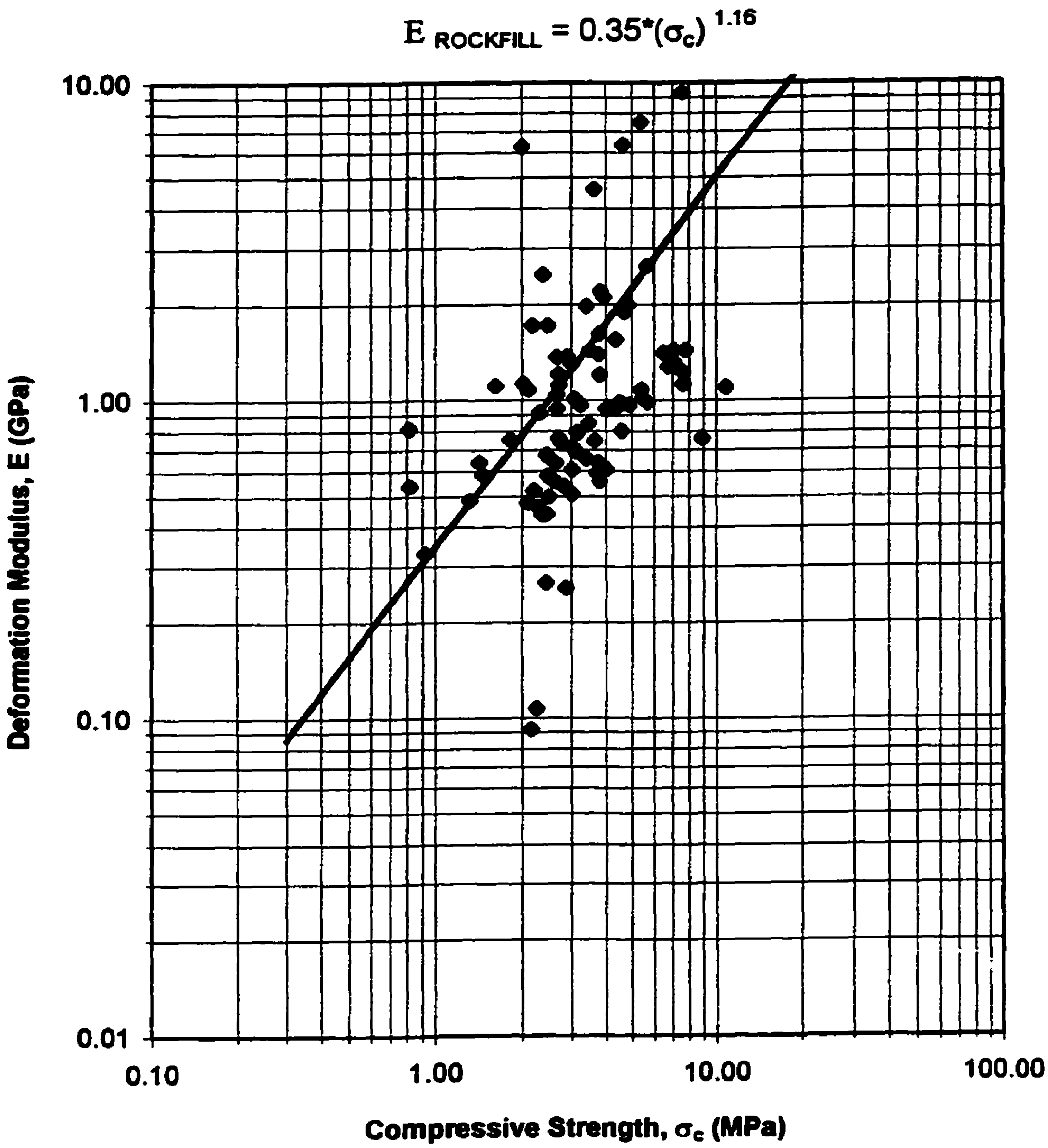


Figure 6.11 Relationship Between Deformation Modulus and Compressive Strength for Cemented Rockfill

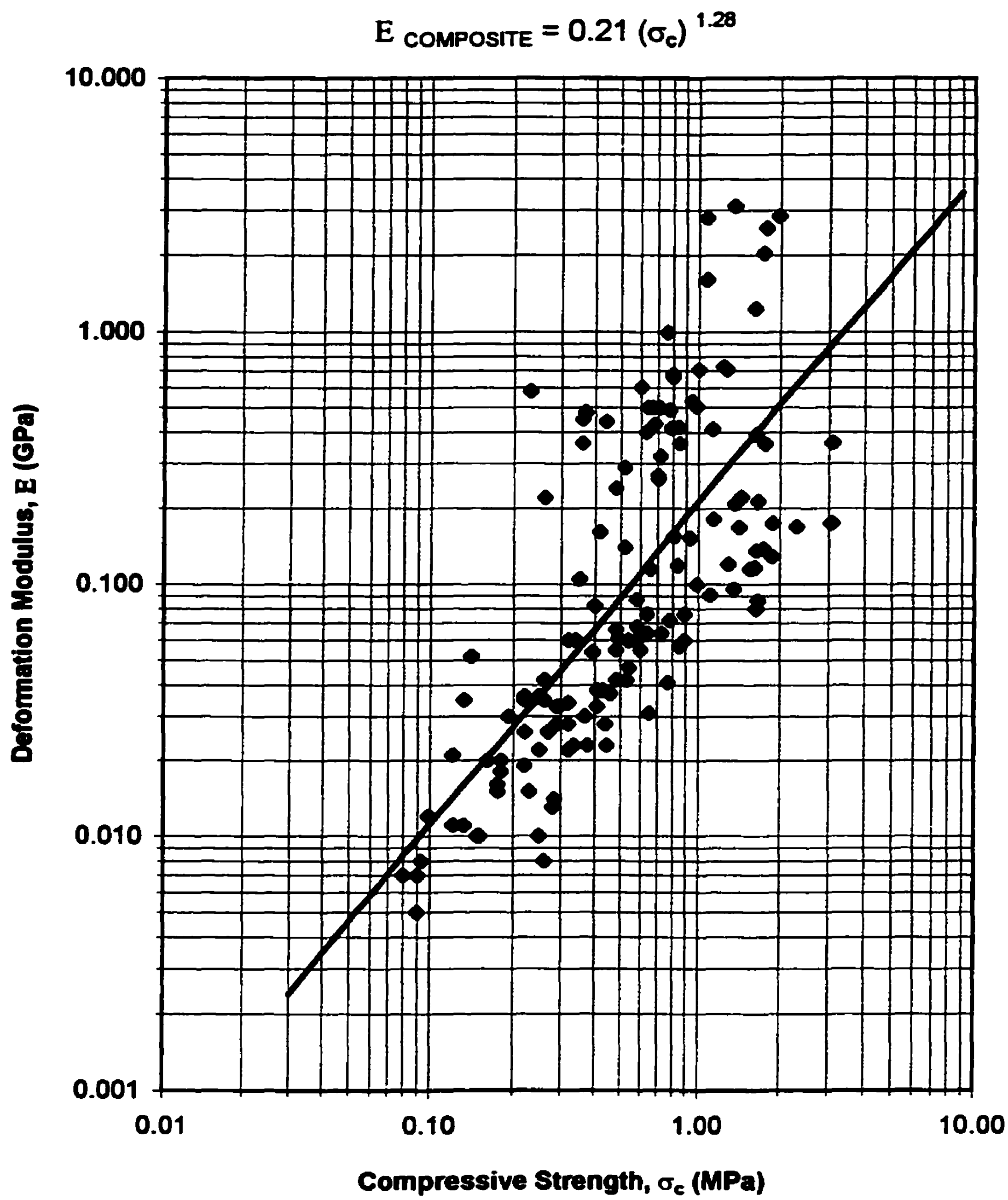


Figure 6.12 Relationship Between Deformation Modulus and Compressive Strength for Composite Fill

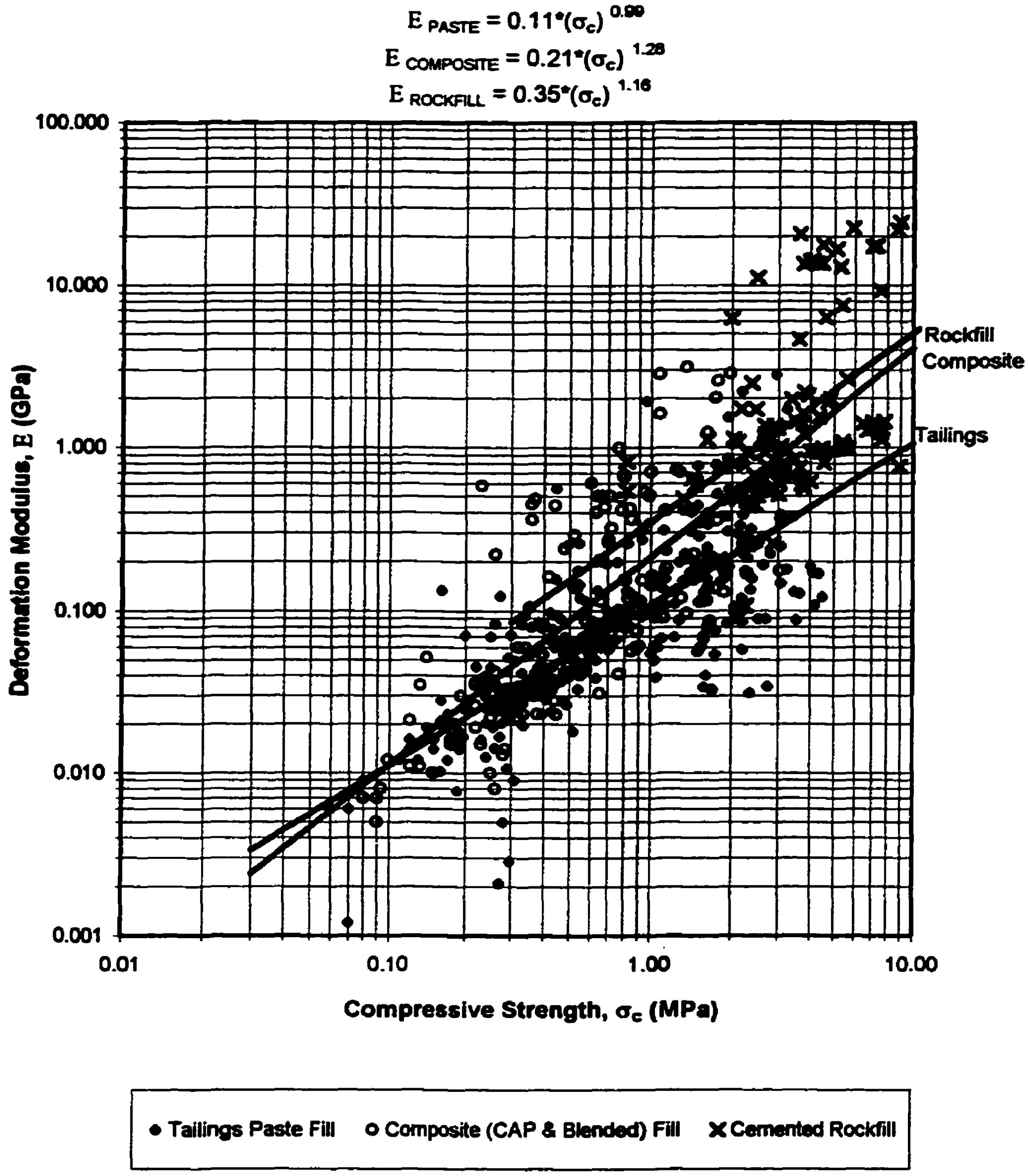


Figure 6.13 Relationship between Compressive Strength and Deformation Modulus for Individual High-Density Fill Systems

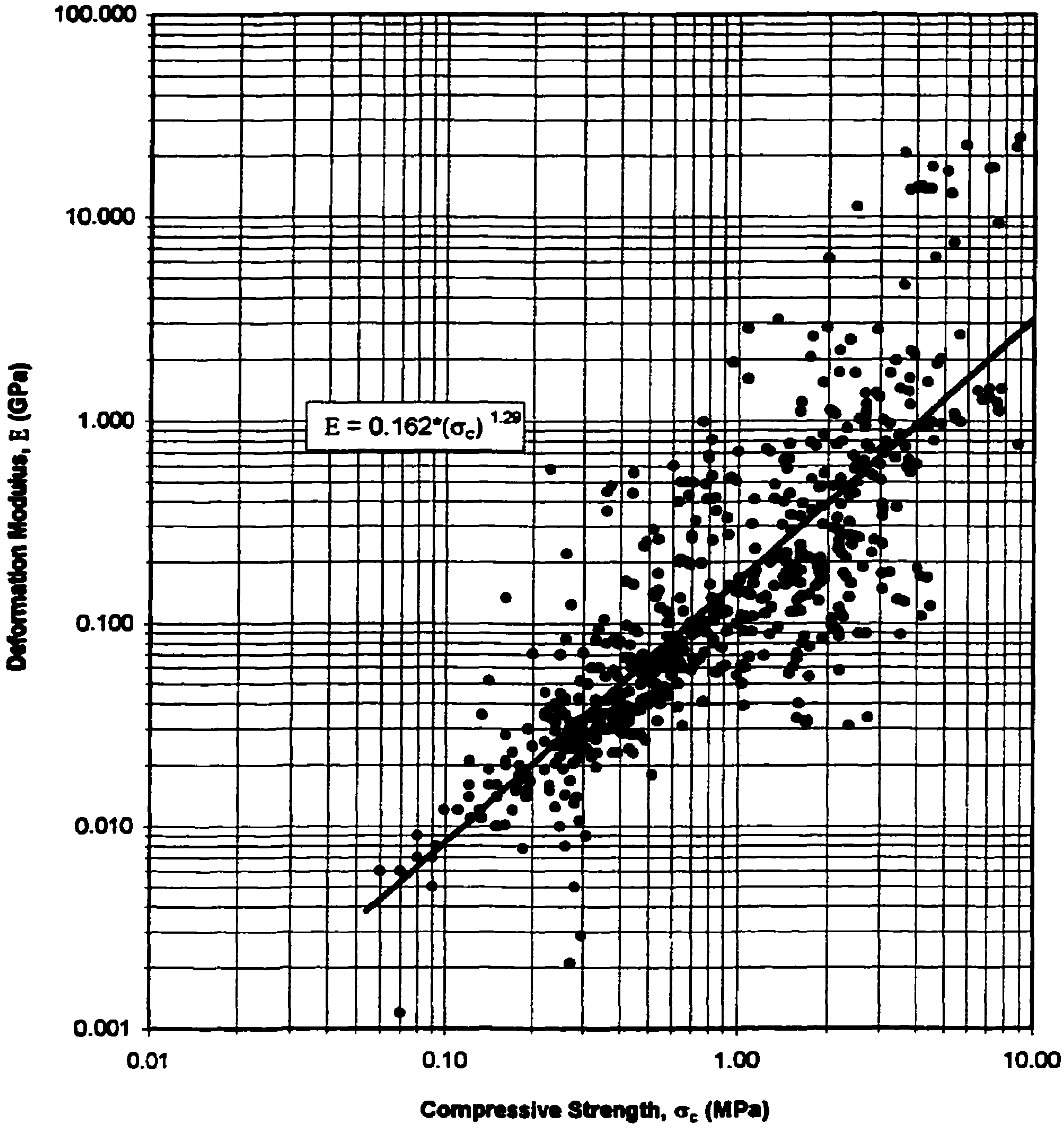


Figure 6.14 Relationship between Compressive Strength and Deformation Modulus for Combined High-Density Fill Systems

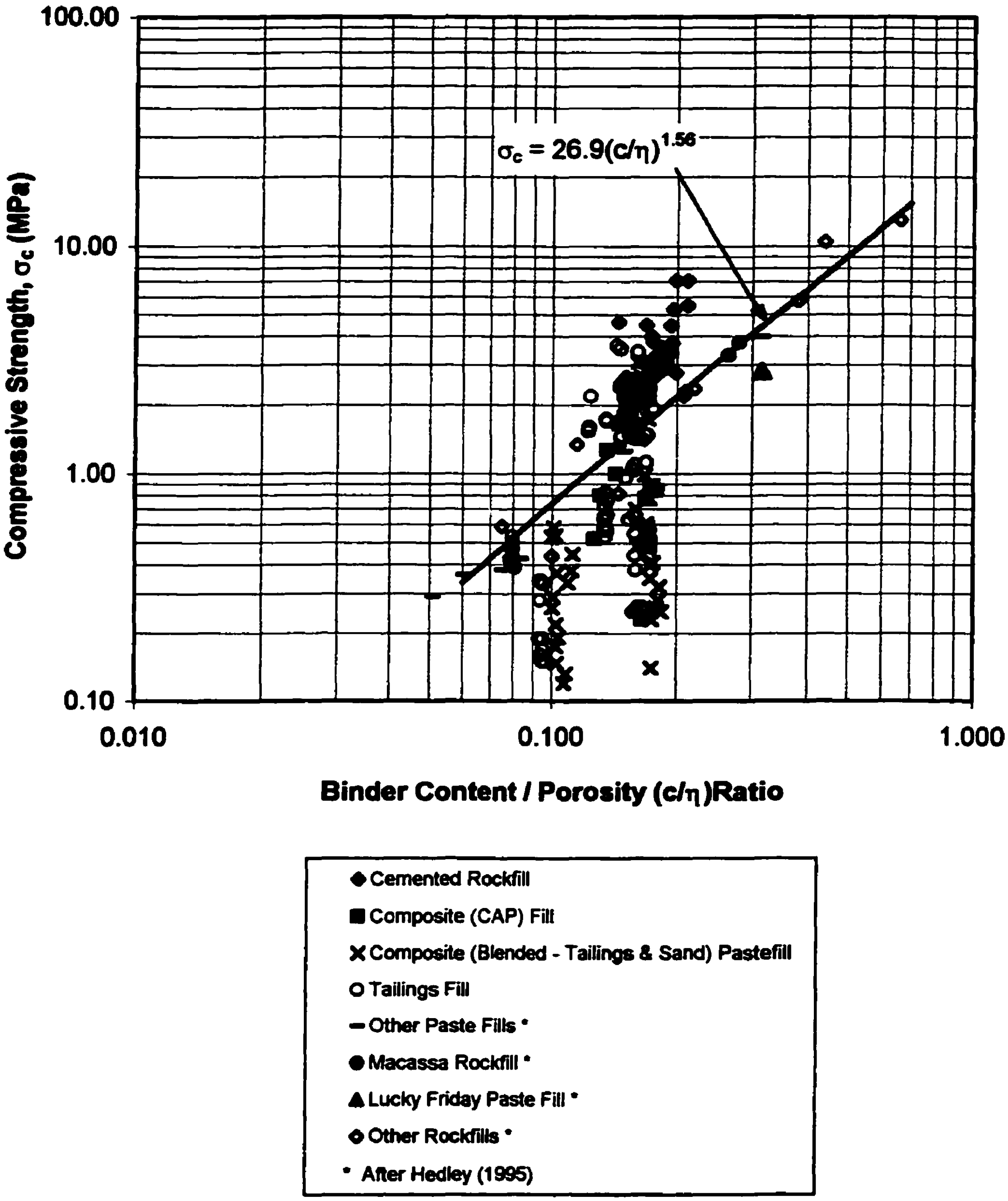


Figure 6.15 Relationship between Compressive Strength and Binder Content/Porosity Ratio for Cemented Rockfill and Paste Backfills from some Mines

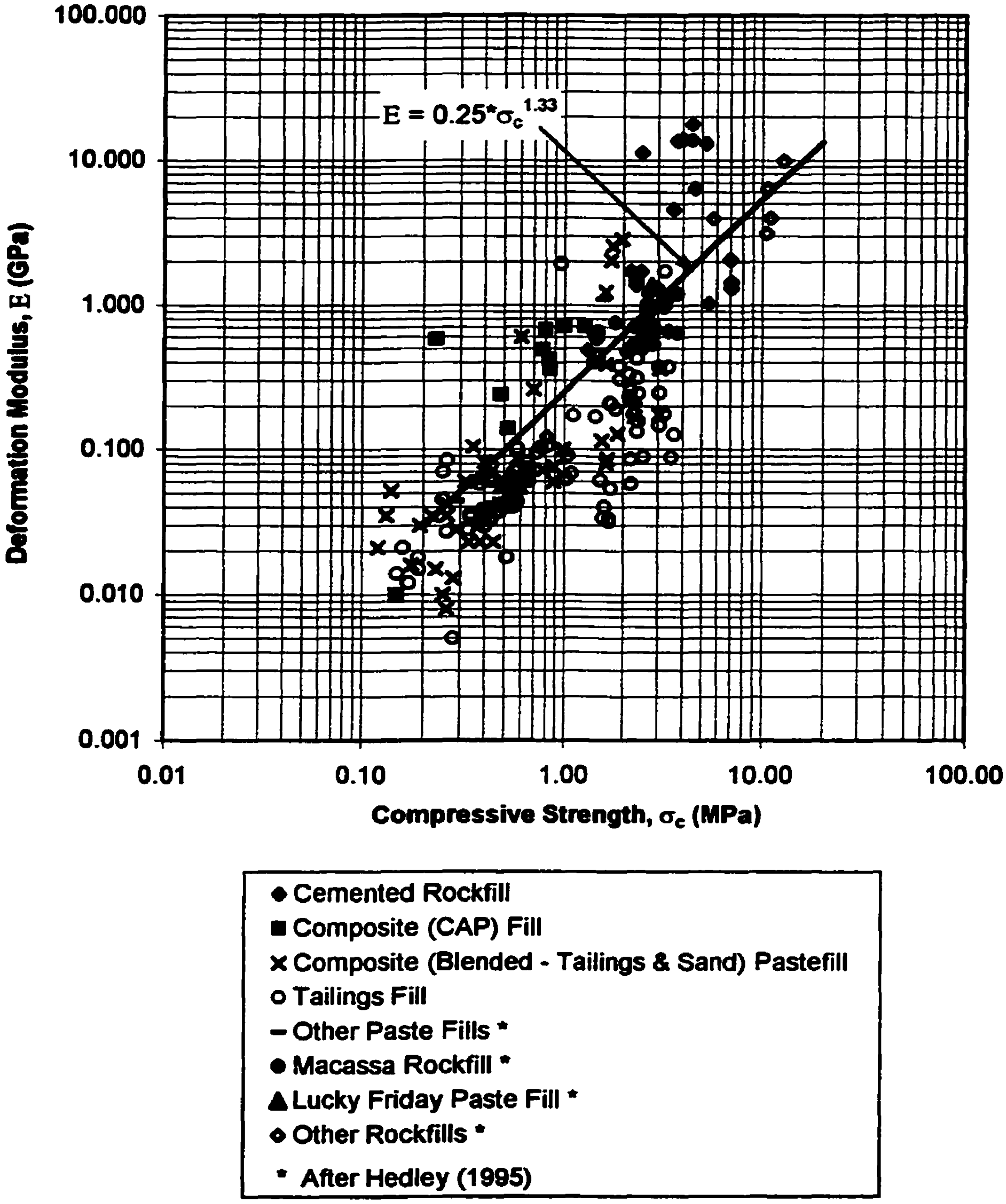


Figure 6.16 Relationship between Compressive Strength and Deformation Modulus for Cemented Rockfill and Paste Backfills from some Mines

6.3.1 <u>Potential Applications of the Test Results</u>

The established relationships between compressive strength (σ_c), moisture/binder (W/C) ratio, binder/porosity (b/η) ratio and deformation modulus (E) for the straight tailings pastefill, composite fills and the cemented rockfill samples, as well as the information provided in Table 6.1, may be used as a planning tool for fill selection toward the engineering design of backfill systems.

For example, the established relationships represented by equations 6.1 to 6.10 may be used in combination with the applicable graphs (Figures 6.5 to 6.16), to estimate the requirements for binder/porosity ratio, compressive strength, and the deformation modulus. The range and mean porosity values for the various fill types are available from Table 6.1. These values can be used for estimating the potential range of binder/cement requirements for achieving a specific deformation modulus value, for any of the high density fills in this study.

A computer program may be written to estimate for each of the fill types in this study, the potential changes in deformation modulus given the binder/porosity (b/η) ratio and vise versa.

An output of one such computer program is provided below in Table 6.3 for the three high-density fill types in this study.

Table 6.3 An Example of Estimated Deformation Modulus values, for a given Binder/Porosity (b/η) Ratio

Binder Porosity (b/η) Ratio	Estimated Deformation Modulus (E) Values		
	Tailings Pastefill (MPa)	Composite Fill (MPa)	Cemented Rockfill (MPa)
0.08	43	29	501
0.10 0.12	57 72	44 61	632 764
0.15	97	90	965
0.20	140	150	1,303
0.25	186	223	1,645
0. 30	236	309	1,990
0.35 0.40	287 341	407 516	2,337 2,687
0.45	397	636	3,038
0.50	455	767	3,392
0.55	514	909	3,747
0.60	575	1,061	4,103
0.65	638	1,223	4,460
0.70	701	1,396	4,819
0.75	767	1,578	5,179

The mean porosity values for the various fill types from Table 6.1 are:

<u>Tailings pastefills:</u>

Base Metal Tailings = 44%

Precious Metal Tailings = 32%

<u>Composite Fills:</u>

Blended Tailings/Sand Paste = 27%

CAP = 20%

Cemented Rockfill = 27%

The binder requirements for developing a specific deformation modulus value may be estimated by using the respective mean porosity values. For example, the designer has the option of selecting a binder content using the mean porosity values as initial estimates. Adjustments can be made in terms of a choice of backfill systems and the binder/porosity (b/η) ratio.

It is suggested that care should be exercised in using the results of this study for purposes of engineering design. The selection of a any type of backfill for practical applications in mining should be based on other site specific requirements (Thomas et al., 1979; Stone, 1993) and not necessarily on the properties of the backfill materials alone.

6.4 SUMMARY

1. The strength and deformation properties of the investigated backfill materials compare favourably with data in the published literature.

2. The composite fill samples developed the lowest void ratios and porosity values when compared to the straight tailings and the cemented rockfill samples. This suggests that the application of composite fill systems will enable more mining and mineral waste products to be used for backfilling. This could benefit the underground mine and surface environments through effective waste utilization.

3. The results of this study also supports the assertion that the properties of composite backfill are essentially a combination of properties of the tailings pastefill and the cemented rockfill properties.

The cemented rockfill was found to be the strongest fill system in this study. The composite fill properties were intermediate between that of the tailing fills and the properties of the cement rockfill.

4. A series of relationships have been developed for the various high-density fill types based on the large data base from this study. The established relationships may permit the selection of binder requirements for achieving a specific backfill stiffness, given the binder porosity ratio and the type of backfill. The information may be used as a preliminary engineering design tool for mine backfill mix design. It is suggested that caution should be exercised in using the test data in this study for design applications in the field. This is because the test data was developed under carefully controlled laboratory conditions and therefore, the information may not apply to field conditions.

CHAPTER 7

7.1 SUMMARY AND CONCLUSIONS

Composite backfill is a derivative of tailings/sand paste backfill and cemented rockfill and it is increasingly gaining recognition as an effective backfill for use in highly stressed zones (Raffield et al., 1998; McKinsly and Hakkanen, 1993; Wingrove, 1993). There is limited information on the properties and behaviour of this new backfill system. In this regard, as mines go deeper, and in situ stresses increase, this new fill system could become the future direction of mine backfill technology.

A review of the mine backfill literature shows that most of the reported work on composite fill was conducted on minus 20 mm coarse aggregate material and classified tailings. The material was placed uncemented, and concrete placement methods were used. This study was undertaken to determine the characteristics and properties of composite fill as a new high-density backfill product. Two types of composite fills were studied. These were: a mixture of tailings and sand; and composite-aggregate paste or "CAP" fill which consisted of a blend of minus 152 mm rockfill aggregates and full plant tailings. The properties of tailings/sand paste backfill and cemented rockfill were also studied as a fundamental step towards the understanding of composite fill behaviour. It was initially proposed that the properties of composite fills could be derived from the combined properties of cemented rockfill and tailings/sand paste backfills.

A comprehensive literature survey was carried on the properties of high-density backfills. Detailed laboratory investigations were conducted to determine the range of tailings and sand paste fill pulp densities that would be conducive to composite backfill mix design. The information of interest to composite fill mix design was the range of paste fill pulp densities that would minimize product segregation, reduce void ratio and porosity of the composite mixture, and thus enhance stiffness and strength development.

The effects of physical properties of the various tailings and sand test materials on paste formation were also determined as part of the investigation. This contributed towards identifying the effective consistency range for mixing composite fill materials. The following physical properties were determined: specific gravity (S.G.) and the particle size gradation parameters including, Coefficient of Curvature (Cc), Coefficient of Uniformity (Cu), and the ultra-fine materials composition which is expressed in terms of the (% -20μm) size particles present in the fill material.

The effects of sand addition on strength development involving "blended" tailings/sand composite fills were also investigated. Sand content ranged between 0 and 75%.

A concept of Composite-Aggregate Paste (CAP) fill was introduced and the properties of this material were examined relative to those of the cemented tailings paste backfill and rockfill samples. An identified optimal mix proportions for coarse aggregates and classified tailings ranged between 60 to 70 percent aggregates and 30 to 40 percent tailings material by weight. The effect of the following variables on the strength and deformation properties of the fill materials in this study were determined: i) binder type and composition; ii) water/binder (w/c) ratio; iii) binder/porosity (b/η) ratio; iv) void ratio (e); v) curing environment and time.

Seven hundred and eighty nine (789) unconfined compressive strength tests were conducted to define the above mentioned parameters. One hundred (100) direct shear box tests and triaxial compression measurements were conducted to define the shear strength parameters for the test materials. The effects of optimum moisture content on the shear strength parameters of the pastefill samples as well as composite-blended tailing/sand pastefill mix design and stability were also investigated. Additionally, the response of the fill materials to load was examined.

The scale effects on the strength and deformation properties of all three types of high-density fill samples were determined in order to infer the potential behaviour of the composite backfill in situ. Test sample sizes for the study ranged between 38mm and 457mm diameter cylinders with length to diameter (l/d) ratio of 2:1. A few cube samples were also tested.

The following conclusions were reached based on the comprehensive investigation conducted as part of this study:

1. The results of the study show that paste fill formation depends on several factors and therefore, particle size distribution alone, cannot be used as a criterion for determining tailings or sand paste fill pulp density. Other material properties such as specific gravity, and the composition of fine materials were found to influence water retention and the onset of paste formation. Paste formation occurred over a wide range of pulp densities for the fill materials in this study. The pulp density range for paste formation was also influenced by particle size gradation parameters of the fill materials. These included: the coefficient of curvature (Cc), the coefficient of uniformity (Cu) and the ultra-fine particles content which is represented by (% - 20μm) material in the fill. The pulp density limits for paste formation was found to be in general terms, a "material specific" parameter which can only be determined through testing.

2. In terms of setting mix-design limits for composite fills, the composite materials that were prepared at high pulp densities (low slumps or high solids concentrations) developed relatively higher compressive strengths compared to those prepared at lower solids concentration. An effective mixing range for the composite tailings/sand fills in this study was found to be between 178mm (7 in) and 228mm (9 in) slump. Slump values of less than 178 mm were found to be too stiff for mixing based on the available laboratory equipment. On the other hand, slump values of higher than 228 mm proved to be too "soupy" and resulted in the segregation of the materials in this study during mixing.

3. The addition of sand to fine tailings as a means of improving the size gradation of the composite product did not readily result in strength gain in the short term. Early improvements in compressive strength occurred when the composite tailings/sand mixtures had a wider range of particle size gradation. Both Cc and Cu values were required in order to accurately define the size gradation of the fill materials in this study. For example, Cc values greater than one (1), and (Cu) values greater than four (4) have been identified in the literature as indicative of a well distributed material. These identified size gradation limits were applied to the fill materials in this study. In this regard, the precious metal tailings failed the criterion for being well distributed materials.

4. Bulk density has been identified from this study as an effective indicator of early strength gain in "blended" tailings/sand composite fill mixtures. Direct increases in the bulk density of a composite tailings/sand mixture with increasing sand content was found to be indicative of an early strength gain for the fill samples in the study.

5. Moisture content had a negative effect on the development of compressive strength and deformation modulus for the studied materials. On the other hand, the shear strength parameters improved at the optimum moisture contents of the respective test materials. Moisture/binder (w/c) ratio was also found to be an effective parameter for selecting composite fill mix design limits. Moisture/binder (w/c) ratios of less than three (3) produced higher compressive strengths in composite tailings/sand paste fill samples in this study. The shear strength parameters were also enhanced at the optimum moisture contents of the studied fill materials. This observation suggests that the stability of an uncemented fill mass could be improved if the moisture content could be kept within the optimum range of the specific material.

6. The blending of tailings and sand materials repeatedly produced lower porosity composite products. The low porosities were however, offset by reductions in the

optimum moisture contents and consequently, the lowering of the moisture contents at which the respective composite pastefill materials reach saturation. A high degree of saturation implies an increased risk of the uncemented fill material stability due to liquefaction. The potential risk of saturation may be reduced through a careful selection of the blending (sand) material by prior testing. Fill stabilization with binder has also been reported to improve the liquefaction resistance of paste backfill.

7. Test samples consisting of composite "blended" tailings/sand pastefill developed strength and deformation properties that were similar to, or higher than those of the straight base metal tailings pastefill. When considered in terms of void ratio or porosity, the composite material was found to be a better product than the straight tailings pastefill. The composite fill samples had lower void ratios and porosities, when compared with the cemented rockfill or the straight tailings paste fill samples. The composite material also showed more resiliency than the straight cemented tailings pastefill which seemed to be brittle at higher binder contents and curing periods of more than 14 days. The low void ratios of the composite fill materials in this study suggest that, more mining and mineral wastes can be used for fill preparation at mine sties, if a composite fill system is used. This could benefit the mine environment by effectively reducing the size of waste disposal areas.

8. The combined features of low void ratio and resiliency, which were the main characteristics of the composite fill materials in this study, suggest that these materials could be used effectively for void reduction and tight filling in mine stopes.

9. The response of the high-density straight tailings/sand pastefill and the composite-aggregate paste fill under load was found to be material specific. Both uncemented and cemented total tailings paste and sand composite fill samples displayed a variety of behaviours under confinement. The predominant characteristics of the stress-strain curves were initially linear elastic, which suggest a brittle behaviour. The

materials became ductile under increasing load and were characterized by strain softening curves. The cemented sand/composite and the uncemented precious metal total tailings pastefill samples displayed strain-hardening behaviour under confinement. They were characterized by increasing deformation modulus and internal friction angles, and decreasing cohesive strength values with increasing confining pressure. Tests conducted at higher confining pressures indicated strain hardening behaviour for some of the cemented fill samples. Deformation modulus increased with both curing period and confining pressure for the sand paste/composite fill. This material also showed some strain-hardening behaviour with increasing confining pressure which suggests the material stiffened under increased load. This behaviour suggests that confining pressures of up to 800 kPa could enhance the in situ strength and deformation properties of composite fill materials with similar compositions as those in this study.

10. With regards to scale effects, the properties of the straight tailings paste fill, the composite blended paste fill and the cemented rockfill samples were found to be strongly scale dependent when compared to that of the Composite-Aggregate Paste (CAP) fill. The CAP fill properties did not indicate any scale dependancy based on the specimen sizes tested in this study. The results suggest that the composite-aggregate fill may not be scale dependent in situ. Because of the differences between the laboratory conditions and the underground mine environment, the in situ properties of the fill could vary from those determined in the laboratory.

11. Based on the results of this study, it can be concluded that the composite fills have some unique features that merit closer examination and comparison with straight tailings or sand pastefills and cemented rockfill systems. In terms of engineering analysis, the application of composite fills could introduce more flexibility into backfill mix design. The use of composite fills would permit more competent fills to be produced and targeted for special applications, including tight filling of the stope

back and void reduction. In terms of engineering analysis, composite fills are stronger materials and will permit backfill systems to be designed for higher free standing heights when compared to using straight tailings pastefill. The composite fills in this study displayed strain hardening characteristics which suggests that they do develop higher stiffness when compressed. Their application in stressed zones of a mine could have the potential for achieving global stability.

7.2 FUTURE WORK

This study has concentrated on fundamental work required to develop concepts and understanding regarding composite backfill preparation and material properties. Future work should take into consideration composite backfill distribution systems, transportation and in-situ placement. For example:

1. The state-of-the-art technique for transporting and placing 20mm "aggregate" fill involves the application of Concrete Technology principles. The aggregates are transported in pipelines at a pipe diameter-to-maximum aggregate size ratio of 5:1 (Wingrove, 1993). New methods are required for placing composite mixtures of full plant tailings and coarser material of up to minus 200mm diameter aggregates in mine stopes. These need to be developed for the composite-aggregate paste (CAP) fill.

2. Various options should be investigated for placing CAP fill in situ. This should include approaches similar to the placement of Consolidated Sand-Rockfill, Consolidated Sand Waste Fill and Consolidated Sand fill (Yu, 1990). The method involves the use of conventional cemented rockfill placement methods with varying amounts of added sand. The fine material composition of between 5 - 10% of the rockfill aggregates by weight, must be adjusted to satisfy the 20 - 40% fine materials composition of the CAP fill. Additionally, a review of Special applications concrete

concrete placement techniques and large aggregate concrete placement methods should be investigated and the suitable methods adapted to the mining industry.

3. Alternative mix ratios of coarse and fine materials should be considered in combination with some of the supplementary binders and binder addition ratios.

4. There is a need to investigate further the feasibility of advantageously applying the optimal moisture contents of fill materials and a minimum amount of binder for composite pastefill stabilization in situ. The results of this study suggests that possibly, the potential exists for using effective moisture control and a minimum amount of binder for fill stabilization.

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APPENDICES

APPENDIX A

APPENDIX A-1

APPENDIX A-1

Equipment Description and Test Procedures for Particle Size Analysis

Three pieces of equipment were used for the particle size analysis of the materials in this study. This was done in order to provide a full range of size data and therefore to produce a better representation of particle sizes and gradation.

Two pieces of sieving equipment were used for analysing the coarse materials which consisted of rockfill aggregates and coarser alluvial sand. The particle size analysis of the coarse material was performed in accordance with ASTM (C-136) standard procedure.

For the rockfill aggregates initial sieving was done using 457mm x 660mm (18 " x 26") rectangular screens in a hydraulic clamping screen shaker. The screen sizes ranged from a maximum of about 102mm (4") opening to a minimum of 9.5mm (3/ 8) opening sizes. For the finer rockfill material left after sieving with the equipment described above as well as for the coarse sands, a sieve shaker with 203mm (8") diameter sieves was used. These sieve openings range from a maximum of 4.76mm (3/16 or the minimum size used in the larger machine) down to a minimum opening size of 53 micrometers.

Sizing of the tailings and the finer sand samples were performed using a laser-based optical particle size analyzer (Leblanc and Annor, 1990). The sizing range of the analyzer is 0.5 to 1200 um.

Typical results of the particle size analysis for the composite materials are presented in the following figures:

APPENDIX A-2

Sedimentation and Consolidation Tests

Sedimentation Tests

The settling of tailings and sand particles in a slurry suspension is influenced by number of factors including, particle size, specific gravity, the viscosity and pulp density of the suspension. The sedimentation tests were performed to assess the settling pulp densities of the sand and tailings materials in this study.

Tests Method

In general terms, tailings or sand samples of known dry mass (approximately 800g) was prepared to an initial pulp density of 30% (by weight of solids) and was placed into a 200ml graduated cylinder. The interface height of the solids suspension was measured consecutively within time intervals. The settling pulp densities at each settling level was then determined.

The pulp density of a settled suspension is calculated as follows:

$$C_w = \{G_s / [G - (V_0 - (H_1 / H_0) \times V_0) \times \gamma_0]\} \times 100$$

Where C_w = pulp density % by weight

G_s = solid weight in the sample in the cylinder

G = initial weight of the suspension in the cylinder

V_0 = initial volume of the suspension in the cylinder

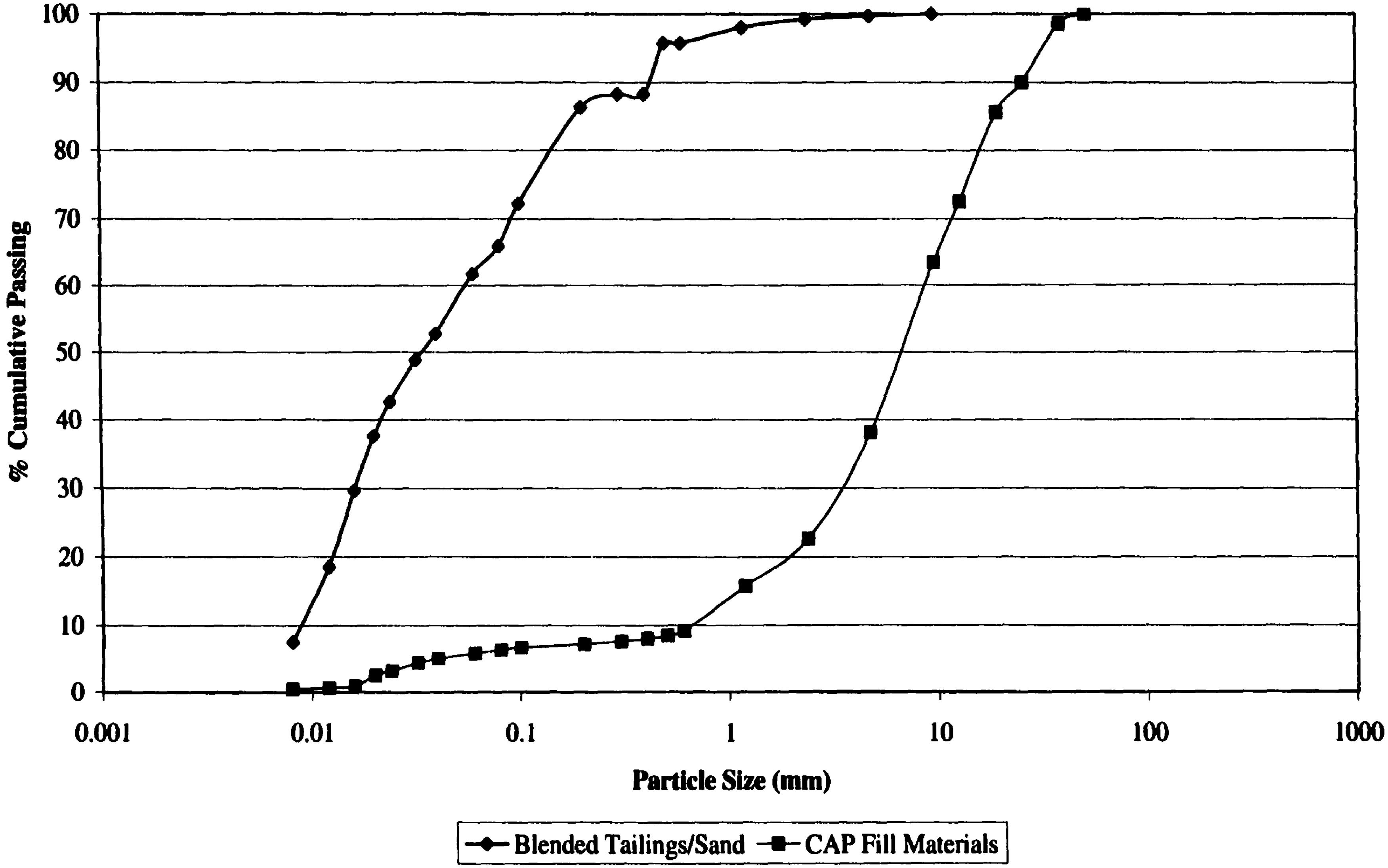
H_1 = initial height of the interface

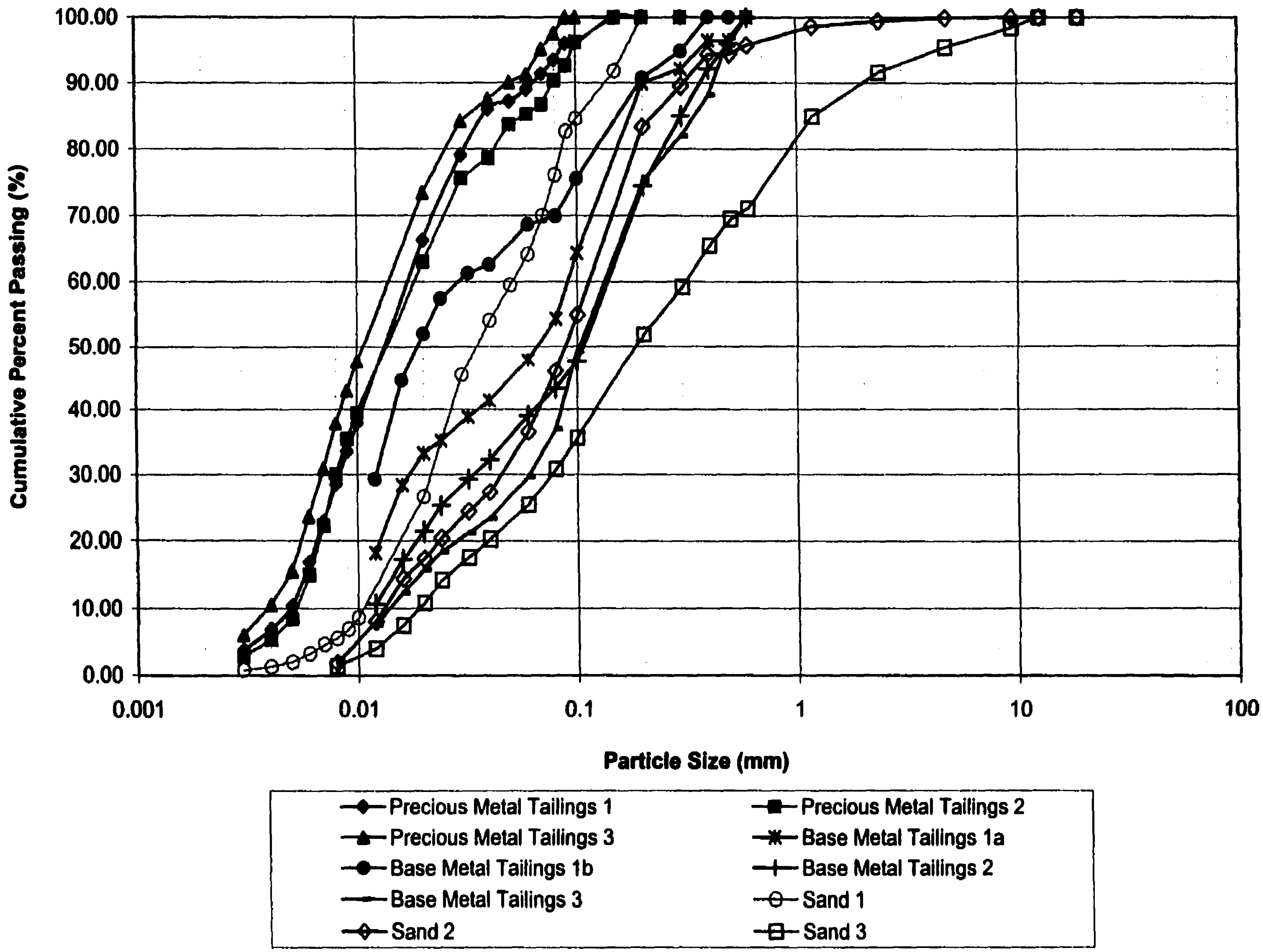
H_0 = height of the interface measured

Particle Size Distribution Data for Composite Fill Materials

Particle size (mm)	CAP Fill Materials	Blended Tailings/Sand
0.008	0.414	7.5
0.012	0.606	18.5
0.016	0.886	29.6
0.020	2.49	37.6
0.024	3.14	42.6
0.032	4.35	48.8
0.040	5.0	52.7
0.060	5.8	61.7
0.080	6.3	65.9
0.100	6.7	72.2
0.200	7.2	86.3
0.300	7.6	88.2
0.400	8.0	88.2
0.500	8.5	95.7
0.600	9.2	95.7
1.18	15.7	98.0
2.36	22.6	99.2
4.76	38.1	99.7
9.50	63.5	100
12.7	72.5	100
19.1	85.6	100
25.4	90.0	100
38.1	98.5	100
50.8	100	100

Particle Size Distribution Graphs for Composite Fill Materials





100.00
90.00
80.00
70.00
60.00
50.00
40.00
30.00
20.00
10.00
0.00
Cumulative Percent Passing (%)
0.001
0.01
0.1
1
10
100
Particle Size (mm)
Precious Metal Tailings 1
Precious Metal Tailings 2
Precious Metal Tailings 3
Base Metal Tailings 1a
Base Metal Tailings 1b
Base Metal Tailings 2
Base Metal Tailings 3
Sand 1
Sand 2
Sand 3

APPENDIX A-2

APPENDIX A-2

Sedimentation and Consolidation Tests

Sedimentation Tests

The settling of tailings and sand particles in a slurry suspension is influenced by number of factors including, particle size, specific gravity, the viscosity and pulp density of the suspension. The sedimentation tests were performed to assess the settling pulp densities of the sand and tailings materials in this study.

Tests Method

In general terms, tailings or sand samples of known dry mass (approximately 800g) was prepared to an initial pulp density of 30% (by weight of solids) and was placed into a 200ml graduated cylinder. The interface height of the solids suspension was measured consecutively within time intervals. The settling pulp densities at each settling level was then determined.

The pulp density of a settled suspension is calculated as follows:

$$C_w = \{G_s / [G - (V_0 - (H_1 / H_0) \times V_0) \times \gamma_0]\} \times 100$$

Where C_w = pulp density % by weight

G_s = solid weight in the sample in the cylinder

G = initial weight of the suspension in the cylinder

V_0 = initial volume of the suspension in the cylinder

H_1 = initial height of the interface

H_0 = height of the interface measured

γ_0 = specific gravity of water

Consolidation Tests

When backfill is placed underground, it becomes consolidated due to moisture loss and successive mining and backfilling activities. The consolidation tests may be used to assess the extent of changes in fill density due to reduction in porosity under loading from successive mining and backfilling operations. The procedure may also be used to estimated the maximum pulp density of tailings or sand fill in a silo (Millette et al., 1998).

Test Method

A standardized consolidation test procedure used in soil mechanics (ASTM D-2435) was followed. A sample of dry material of 250g was prepared to a settling pulp density ranging between (70-75% by weight) and was placed into a 6.4mm diameter consolidation cell. Pressure ranging in 70-210 kPa was applied to compress the sample. The pressure corresponds to a silo height of 5 to 10m. The changes in volume of a sample were measured over a period of 24 hours.

Data Presentation

The combined sedimentation and consolidation tests results and graphs for the studied material are presented in the following pages:

SEDIMENTATION TEST
Precious Metal Tailings 3

Wt of tare =	0.927	kg
Wt of tare + slurry =	3.506	kg
Wt of slurry =	2.579	kg
Initial slurry pulp =	30	%
Vol. Of initial slurry	2000	ml
Ht. Of initial slurry =	41	cm
Wt of solid in slurry	773.7	g

Time (Hour)	Height (cm)	Coresp. Volume (ml)		Wt of H_2O (g)	Wt of new slurry (g)	Pulp density (%)
0.000	41.00	2000.00		0.00	2579.00	30.00
0.008	40.60	1980.49		19.51	2559.49	30.23
0.017	40.50	1975.61		24.39	2554.61	30.29
0.025	39.70	1936.59		63.41	2515.59	30.76
0.033	39.10	1907.32		92.68	2486.32	31.12
0.050	38.60	1882.93		117.07	2461.93	31.43
0.067	38.00	1853.66		146.34	2432.66	31.80
0.083	37.30	1819.51		180.49	2398.51	32.26
0.100	36.70	1790.24		209.76	2369.24	32.66
0.117	36.10	1760.98		239.02	2339.98	33.06
0.133	35.60	1736.59		263.41	2315.59	33.41
0.150	35.00	1707.32		292.68	2286.32	33.84
0.167	32.10	1565.85		434.15	2144.85	36.07
0.250	29.00	1414.63		585.37	1993.63	38.81
0.333	26.10	1273.17		726.83	1852.17	41.77
0.417	23.50	1146.34		853.66	1725.34	44.84
0.500	14.10	687.80		1312.20	1266.80	61.07
1.000	12.40	604.88		1395.12	1183.88	65.35
2.000	11.90	580.49		1419.51	1159.49	66.73
3.000	11.70	570.73		1429.27	1149.73	67.29
4.000	11.50	560.98		1439.02	1139.98	67.87
5.000	11.40	556.10		1443.90	1135.10	68.16
24.000	11.40	556.10		1443.90	1135.10	68.16

SEDIMENTATION TEST

Wt of tare =	0.9413	**kg**
Wt of tare + slurry =	3.556	**kg**
Wt of slurry =	2.6147	**kg**
Initial slurry pulp =	30	**%**
Vol. Of initial slurry =	2000	**ml**
Ht. Of initial slurry =	41	**cm**
Wt of solid in slurry =	784.41	**g**

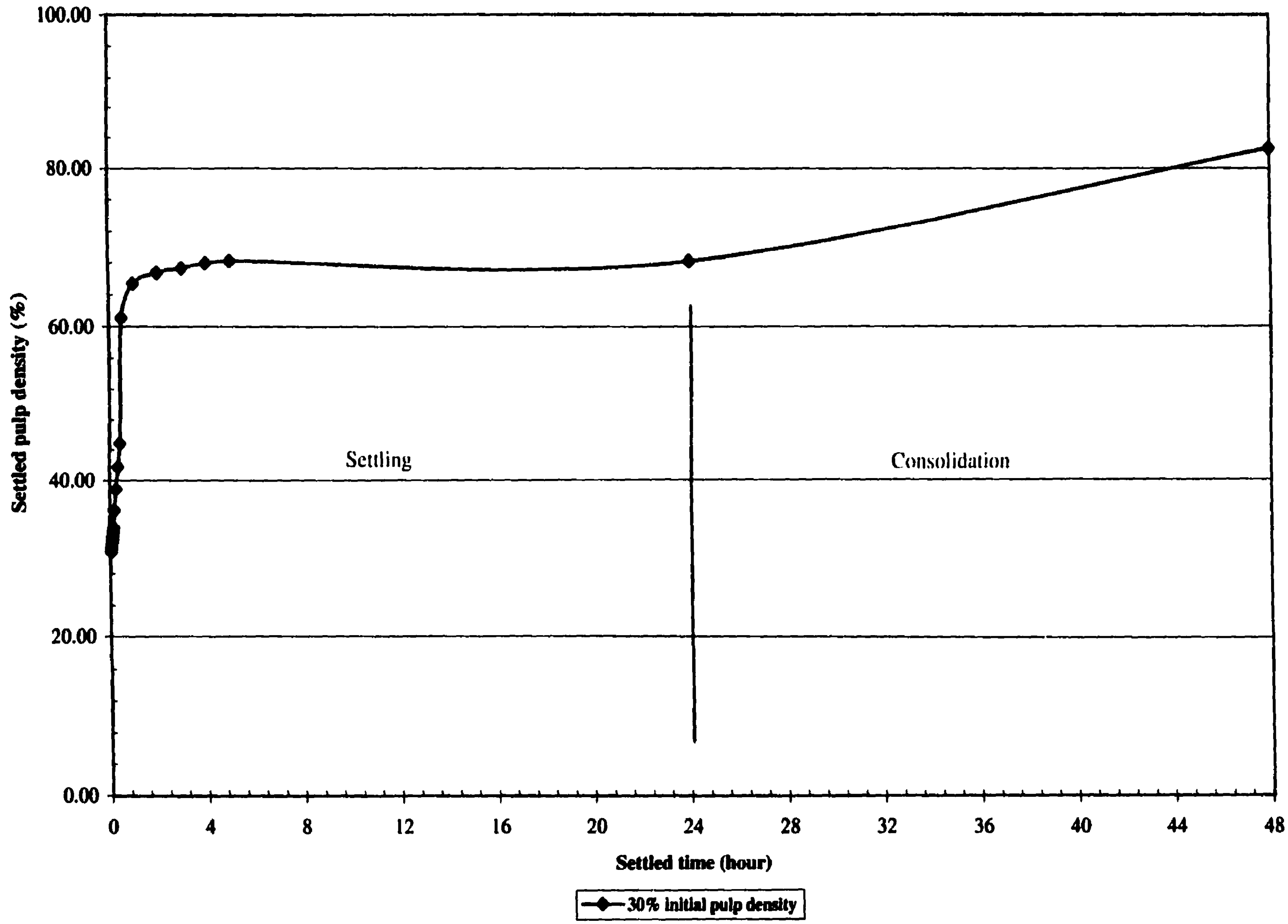
Time (Hour)	**Height (cm)**	**Coresp. Volume (ml)**		**Wt of H_2O (g)**	**Wt of new slurry (g)**	**Pulp density (%)**
0.000	41.00	2000.00		0.00	2614.70	30.00
0.008	40.70	1985.37		14.63	2600.07	30.17
0.017	40.50	1975.61		24.39	2590.31	30.28
0.025	40.10	1956.10		43.90	2570.80	30.51
0.033	39.90	1946.34		53.66	2561.04	30.63
0.050	39.50	1926.83		73.17	2541.53	30.86
0.067	39.00	1902.44		97.56	2517.14	31.16
0.083	38.50	1878.05		121.95	2492.75	31.47
0.100	38.20	1863.42		136.58	2478.12	31.65
0.117	37.60	1834.15		165.85	2448.85	32.03
0.133	37.30	1819.51		180.49	2434.21	32.22
0.150	37.00	1804.88		195.12	2419.58	32.42
0.167	36.50	1780.49		219.51	2395.19	32.75
0.250	34.20	1668.29		331.71	2282.99	34.36
0.333	31.90	1556.10		443.90	2170.80	36.13
0.417	29.90	1458.54		541.46	2073.24	37.84
0.500	27.90	1360.98		639.02	1975.68	39.70
1.000	18.50	902.44		1097.56	1517.14	51.70
2.000	16.40	800.00		1200.00	1414.70	55.45
3.000	15.00	731.71		1268.29	1346.41	58.26
4.000	14.50	707.32		1292.68	1322.02	59.33
5.000	14.00	682.93		1317.07	1297.63	60.45
24.000	12.50	609.76		1390.24	1224.46	64.06

SEDIMENTATION TEST
Sand 1

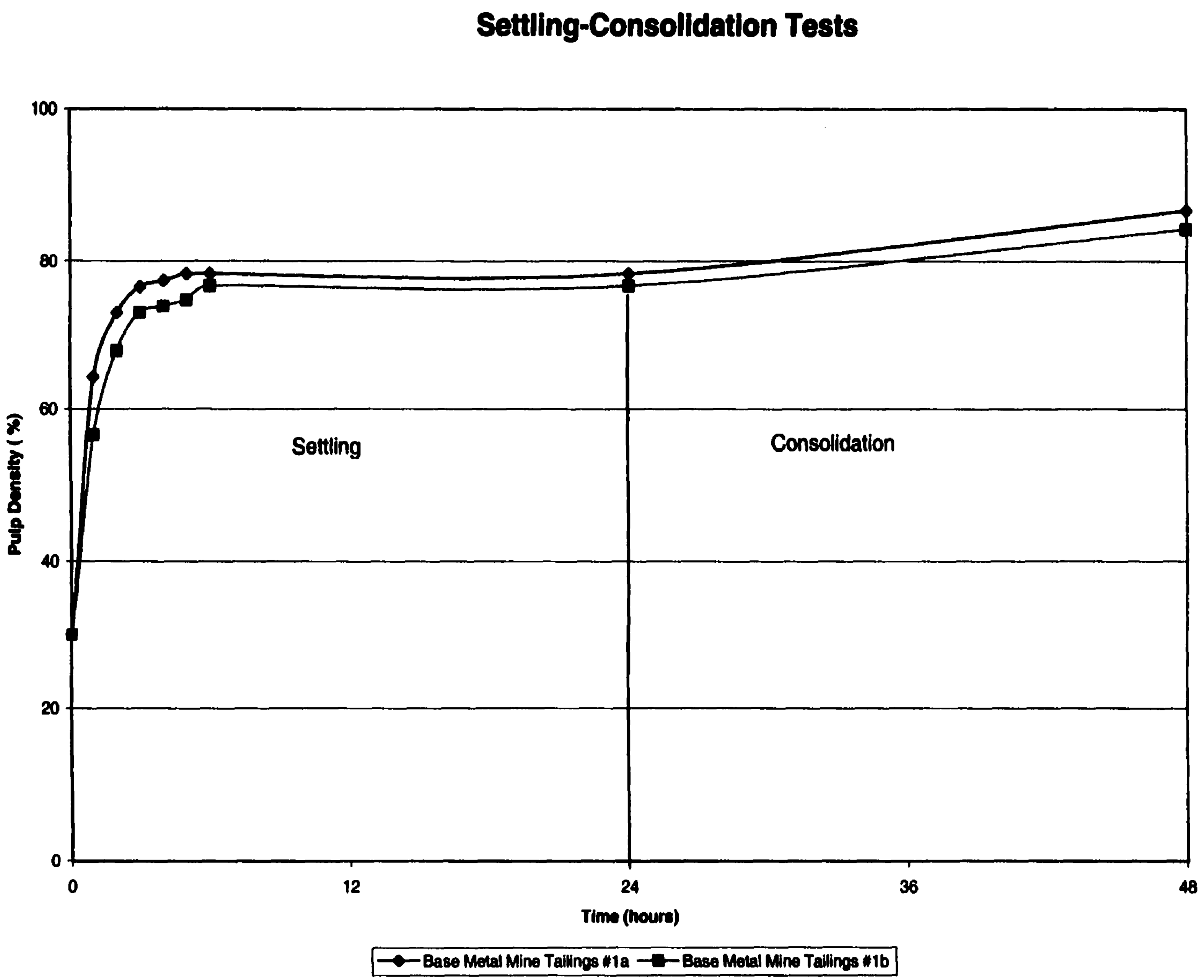
Wt of tare =	0.928	kg
Wt of tare + slurry =	3.572	kg
Wt of slurry =	2.644	kg
Initial slurry pulp =	30	%
Vol. Of initial slurry	2000	ml
Ht. Of initial slurry =	41	cm
Wt of solid in slurry	793.2	g

Time (Hour)	Height (cm)	Coresp. Volume (ml)		Wt of H_2O (g)	Wt of new slurry (g)	Pulp density (%)
0.000	41.00	2000.00		0.00	2644.00	30.00
0.008	40.75	1987.80		12.20	2631.80	30.14
0.017	40.45	1973.17		26.83	2617.17	30.31
0.025	38.25	1865.85		134.15	2509.85	31.60
0.033	37.25	1817.07		182.93	2461.07	32.23
0.050	36.15	1763.41		236.59	2407.41	32.95
0.067	34.75	1695.12		304.88	2339.12	33.91
0.083	32.85	1602.44		397.56	2246.44	35.31
0.100	31.35	1529.27		470.73	2173.27	36.50
0.117	29.95	1460.98		539.02	2104.98	37.68
0.133	28.25	1378.05		621.95	2022.05	39.23
0.150	26.75	1304.88		695.12	1948.88	40.70
0.167	25.25	1231.71		768.29	1875.71	42.29
0.250	17.75	865.85		1134.15	1509.85	52.53
0.333	14.75	719.51		1280.49	1363.51	58.17
0.417	14.15	690.24		1309.76	1334.24	59.45
0.500	14.00	682.93		1317.07	1326.93	59.78
1.000	13.65	665.85		1334.15	1309.85	60.56
2.000	13.55	660.98		1339.02	1304.98	60.78
3.000	13.45	656.10		1343.90	1300.10	61.01
4.000	13.45	656.10		1343.90	1300.10	61.01
5.000	13.45	656.10		1343.90	1300.10	61.01
24.000	13.45	656.10		1343.90	1300.10	61.01

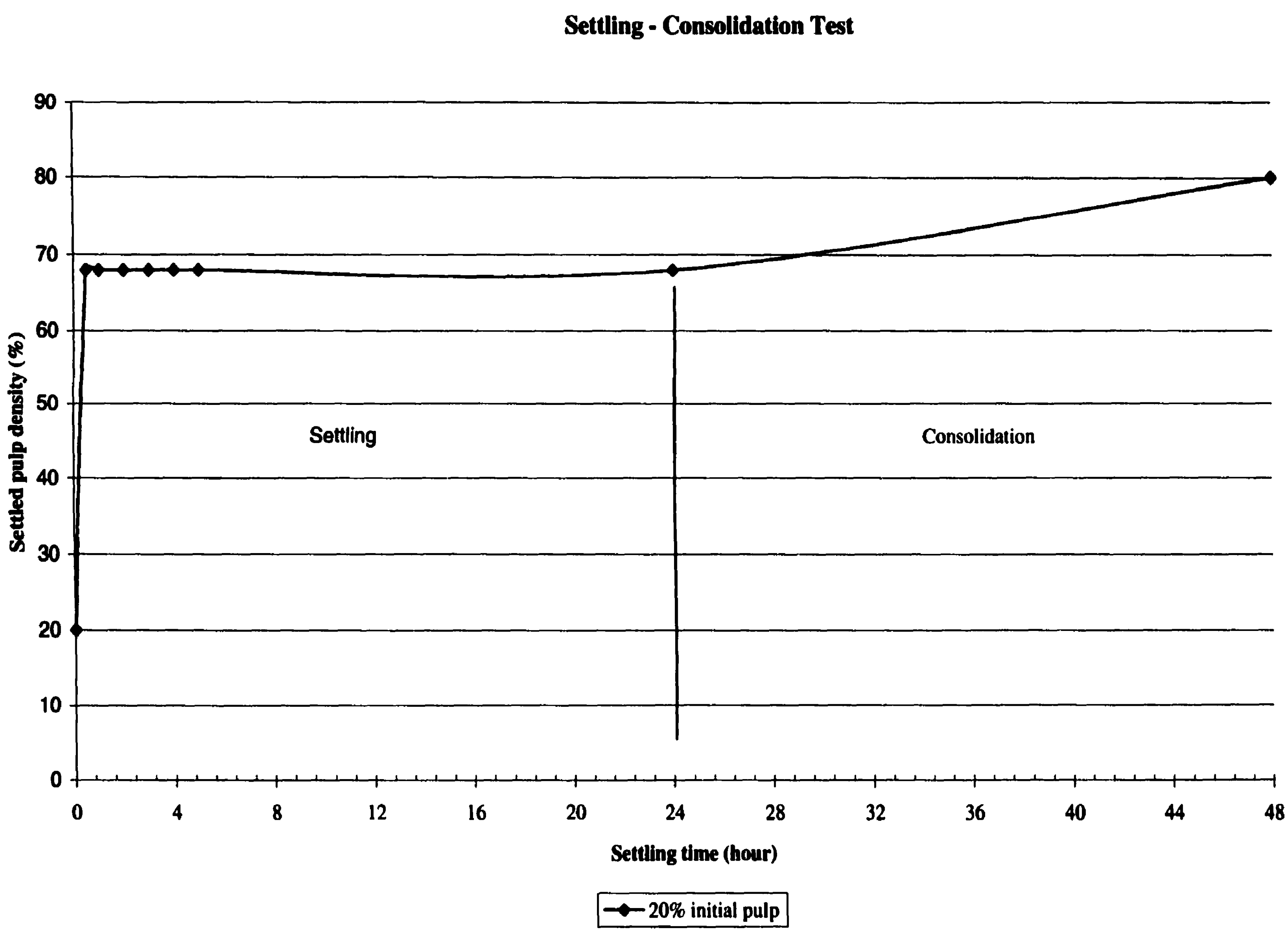
Settling-Consolidation Tests



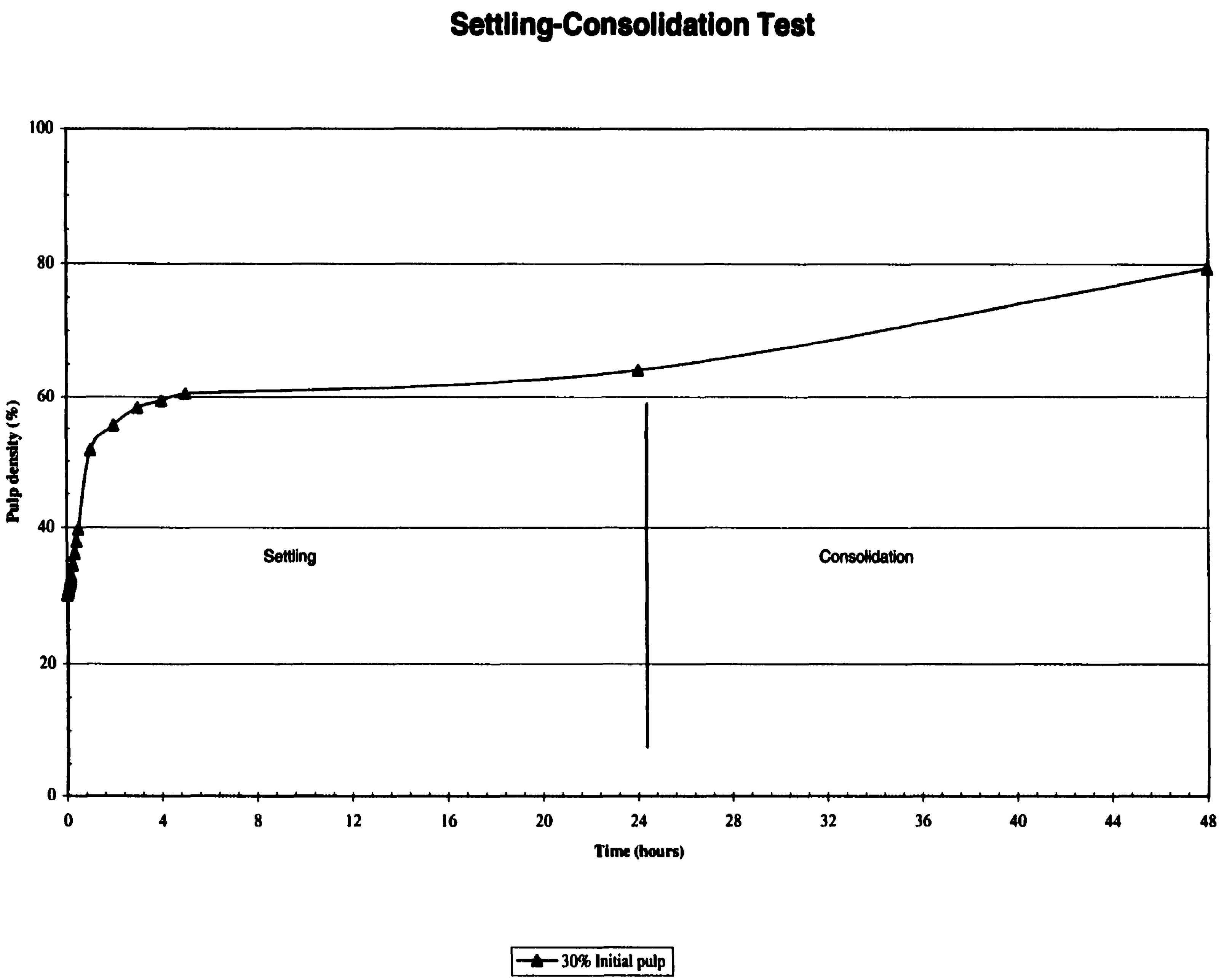
Settling-Consolidation Test for Precious Metal Tailings #3



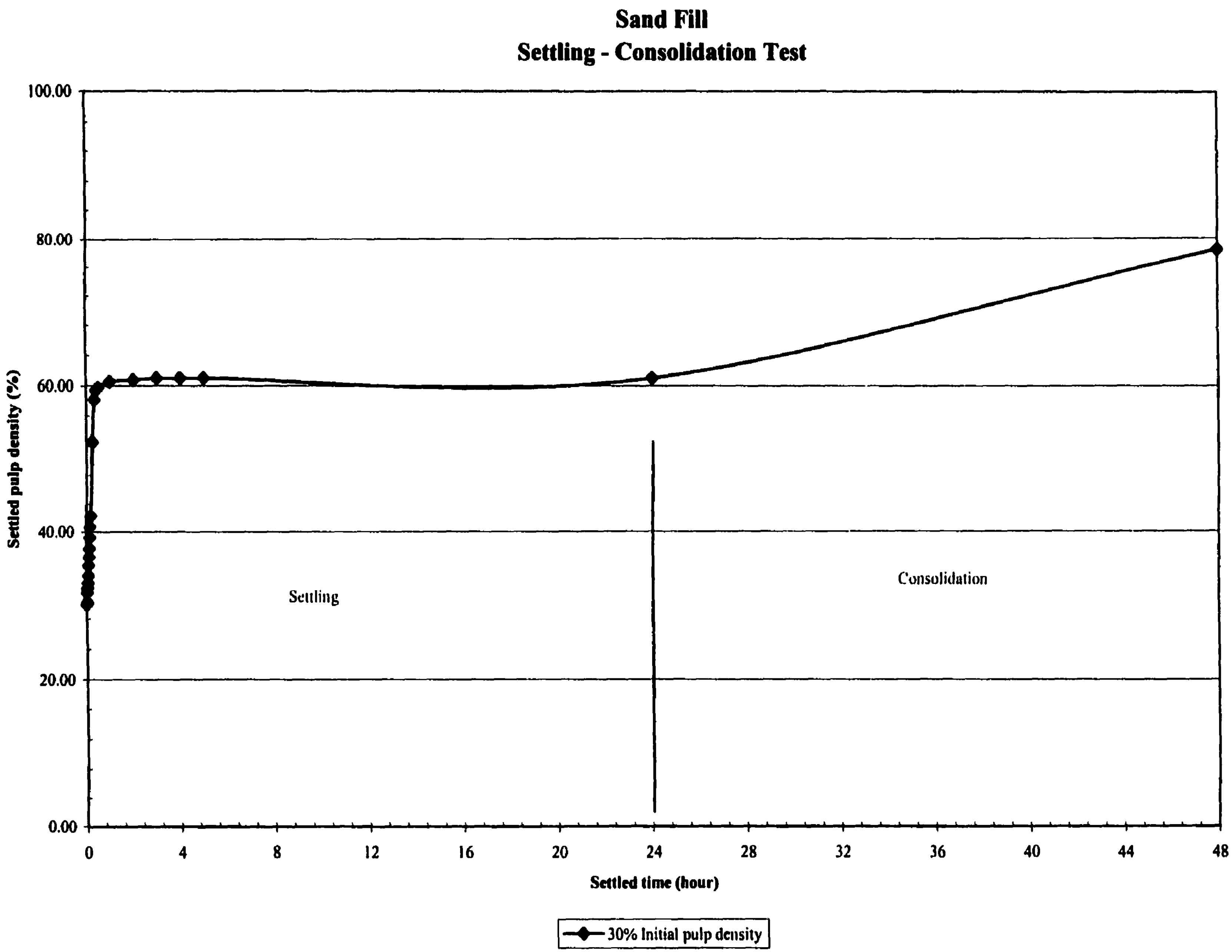
Settling-Consolidation Test for Base Metal Tailings #1a and #1b



Settling-Consolidation tests for Base Metal Tailings #2



Settling-Consolidation tests for Base Metal Tailings #3



Sand Fill
Settling - Consolidation Test
100.00
80.00
60.00
40.00
20.00
0.00
Settled pulp density (%)
0
4
8
12
16
20
24
28
32
36
40
44
48
Settled time (hour)
Settling
Consolidation
30% Initial pulp density

APPENDIX A-3

APPENDIX A-3

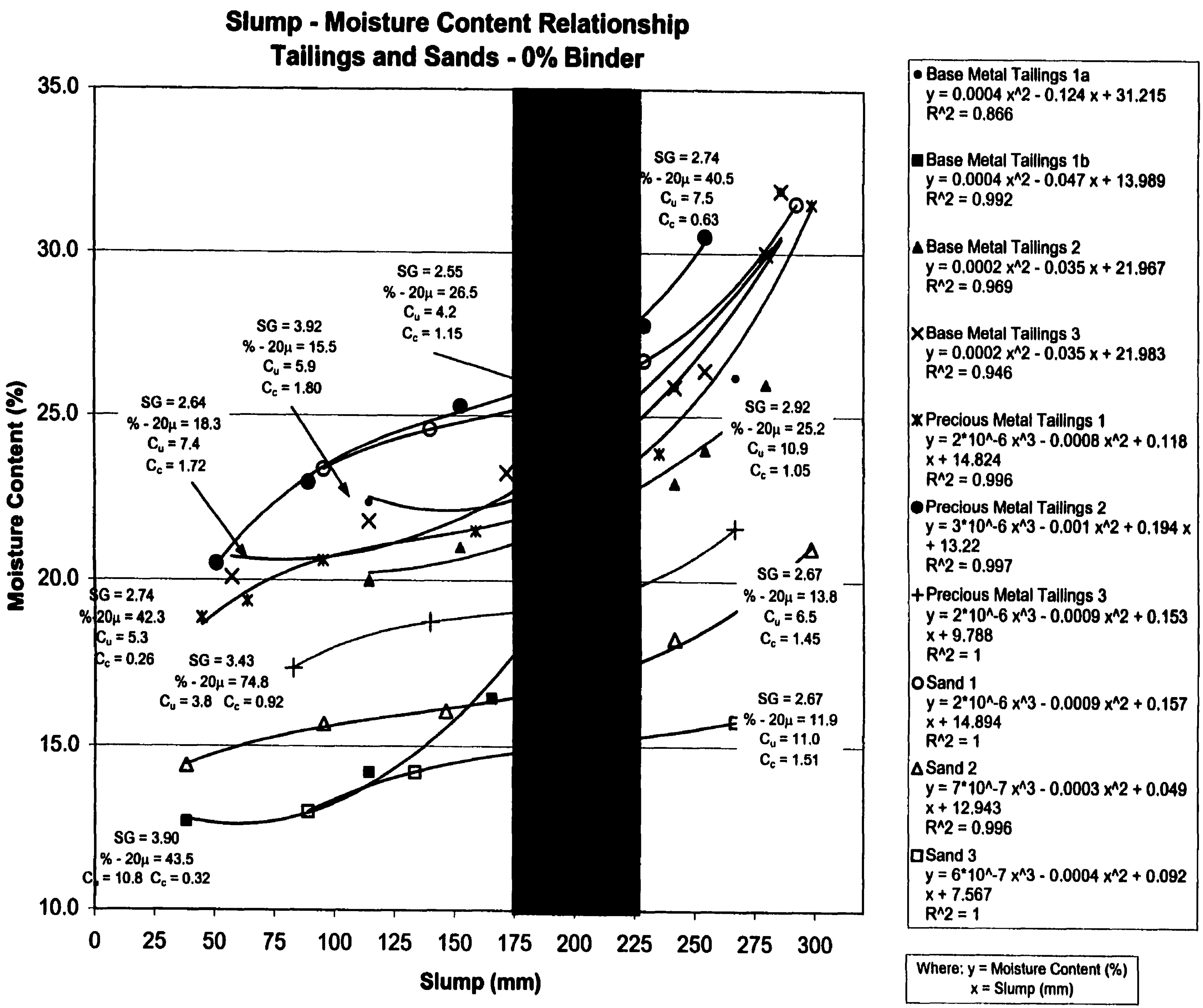
Slump- Tests

The slump tests were carried out to determine the relationships between slump and moisture content for the tailings and sand pastefill mixtures in this study.

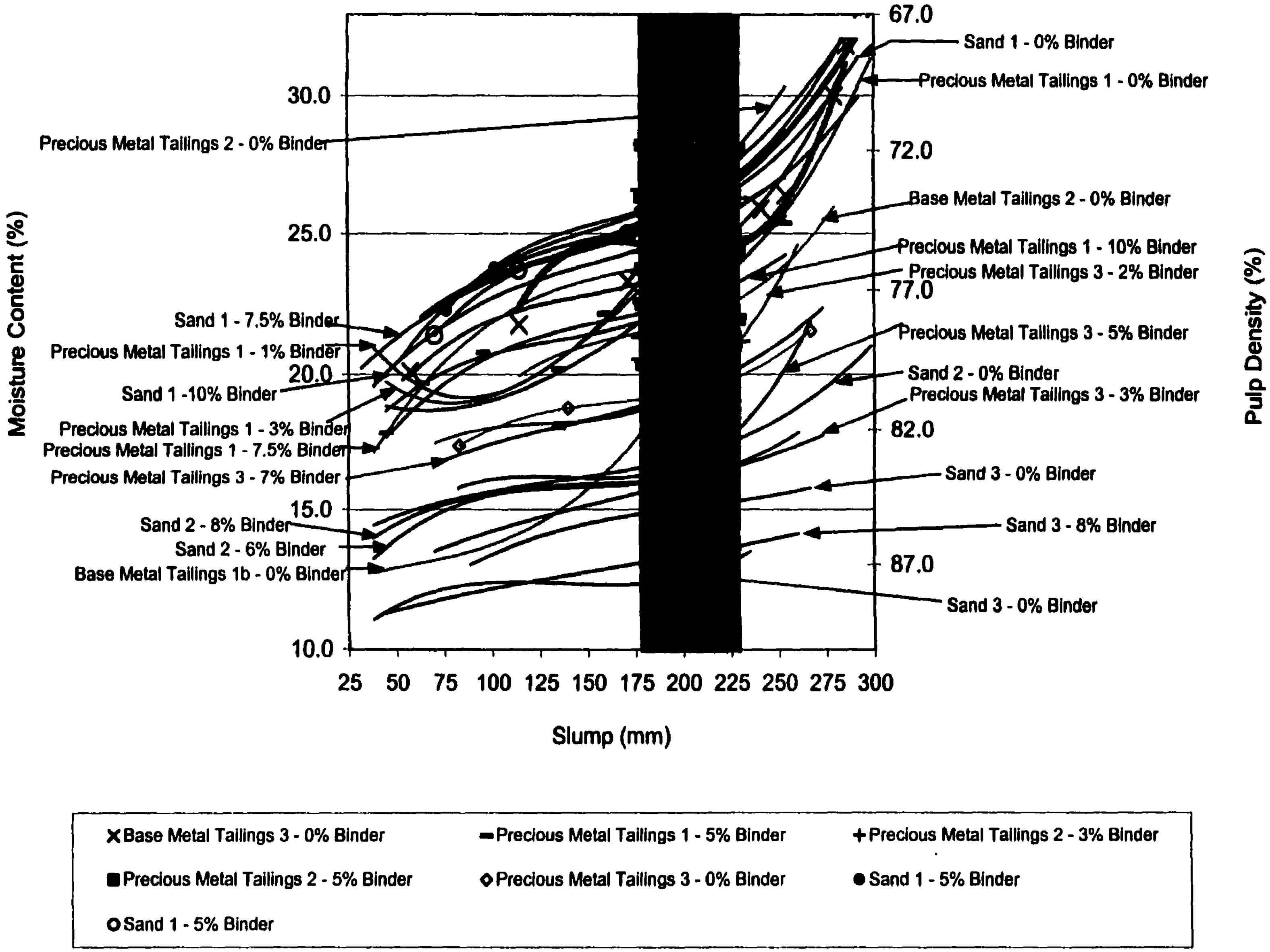
Test Method and Results

The standard slump test (ASTM C-143) procedure used in concrete technology was followed.

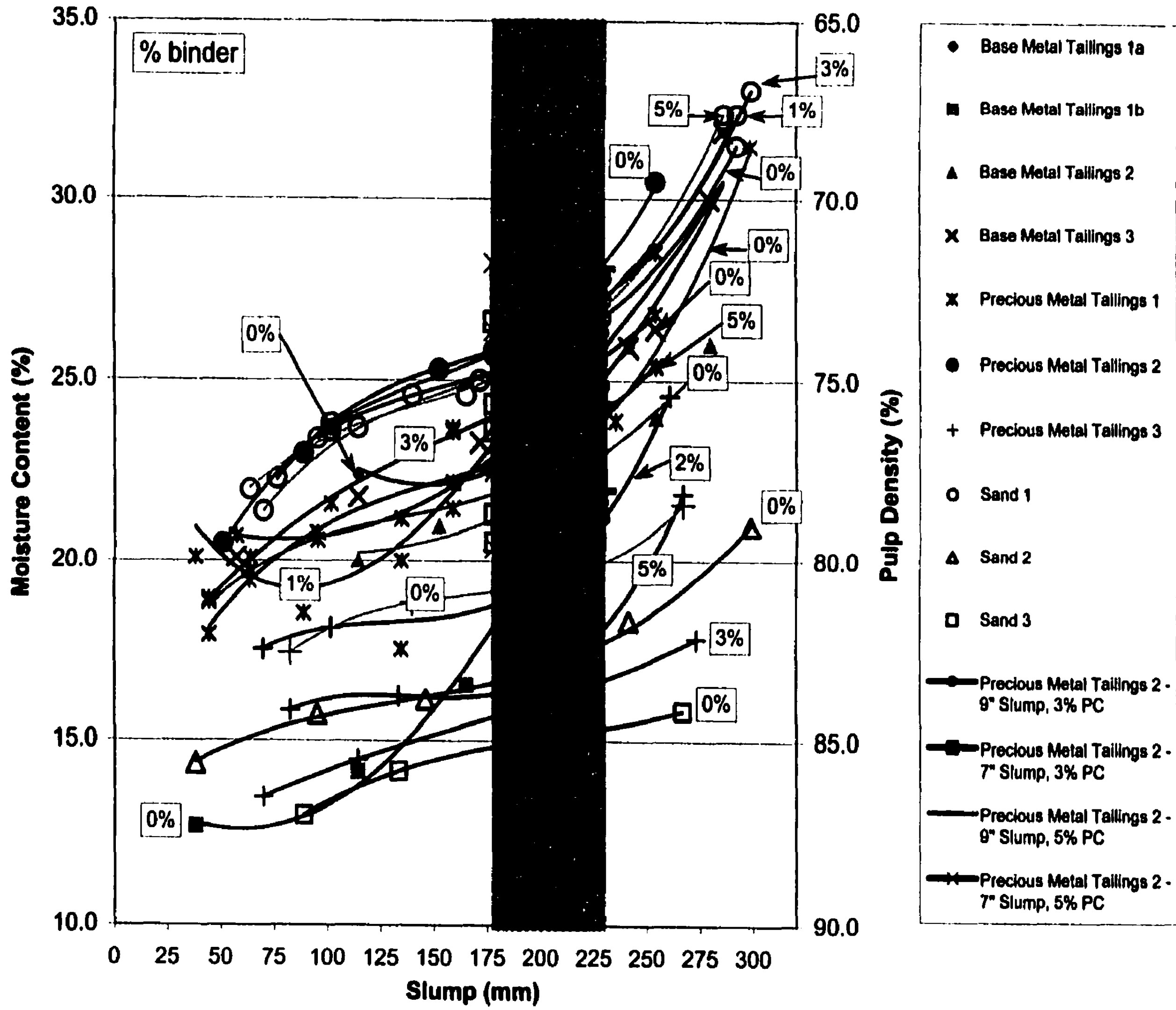
The test results in the form of slump-vs-moisture content graphs for the studied uncemented and cemented pastefill samples are presented in the following pages:



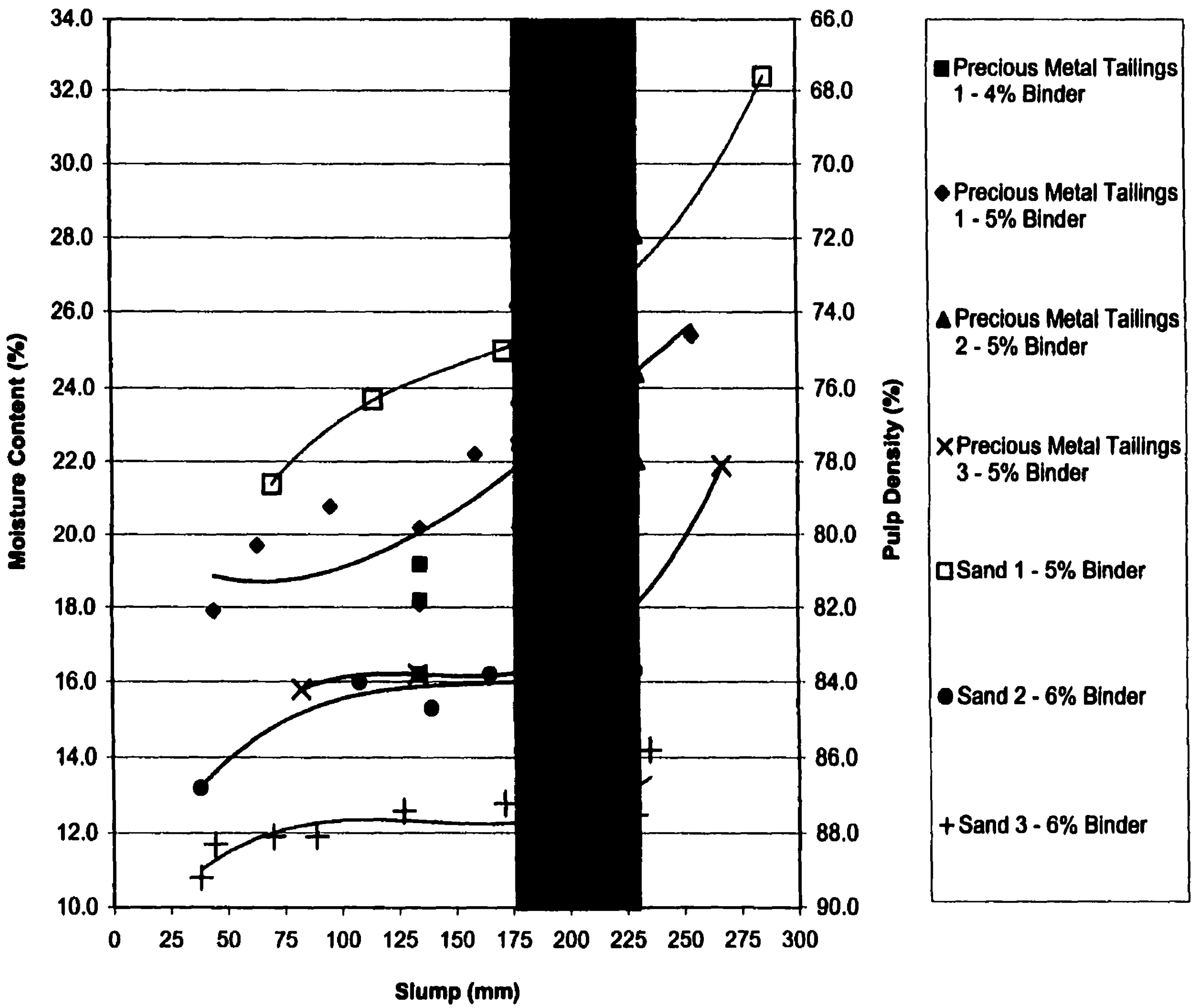
Slump-Moisture Content Relationships for Uncemented Tailings and Sand Paste Fills



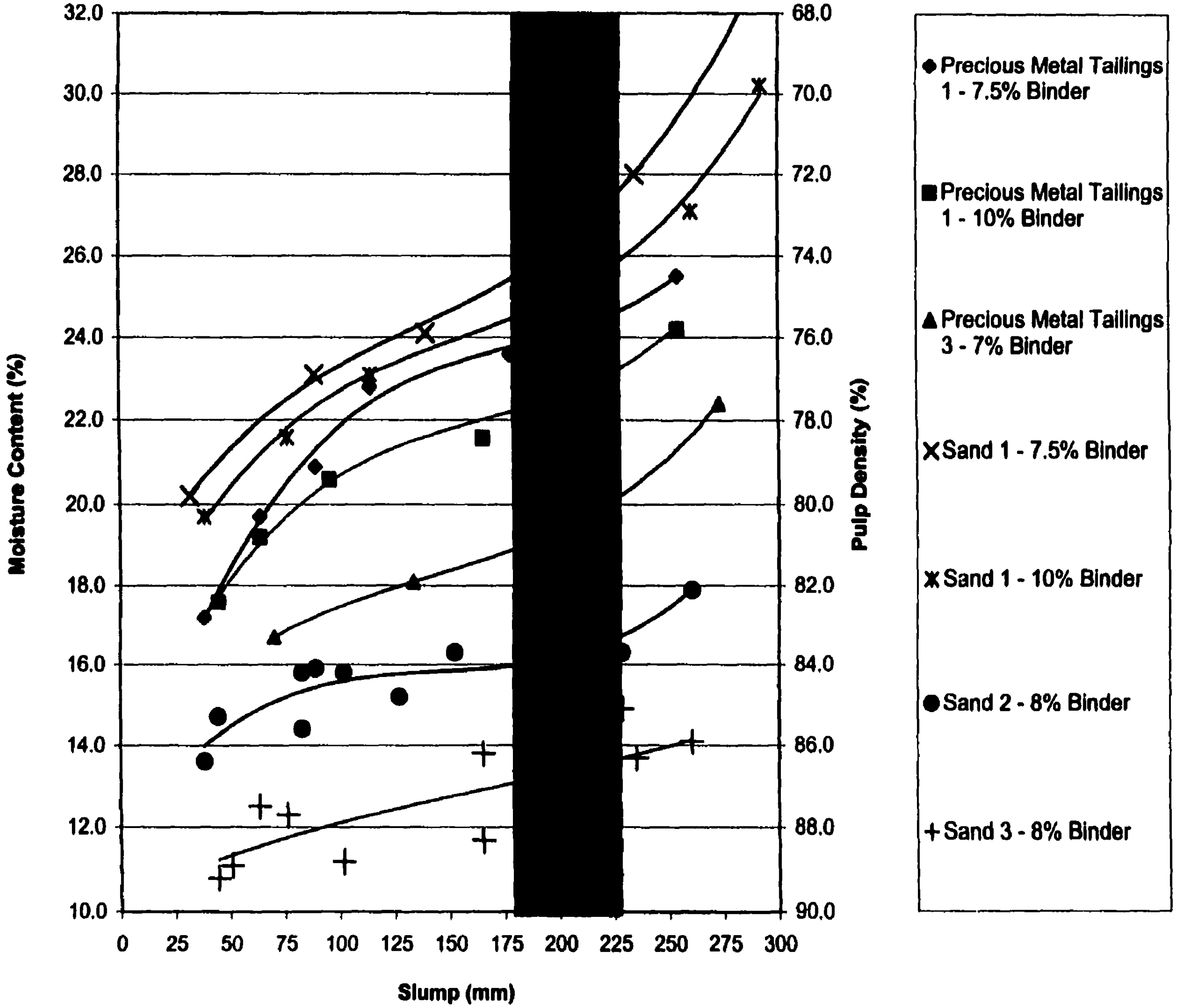
Moisture Content (%)
Pulp Density (%)
Slump (mm)
10.0
15.0
20.0
25.0
30.0
67.0
72.0
77.0
82.0
87.0
25
50
75
100
125
150
175
200
225
250
275
300
Sand 1 - 0% Binder
Precious Metal Tailings 1 - 0% Binder
Precious Metal Tailings 2 - 0% Binder
Base Metal Tailings 2 - 0% Binder
Precious Metal Tailings 1 - 10% Binder
Precious Metal Tailings 3 - 2% Binder
Sand 1 - 7.5% Binder
Precious Metal Tailings 1 - 1% Binder
Precious Metal Tailings 3 - 5% Binder
Sand 1 -10% Binder
Sand 2 - 0% Binder
Precious Metal Tailings 3 - 3% Binder
Precious Metal Tailings 1 - 3% Binder
Precious Metal Tailings 1 - 7.5% Binder
Precious Metal Tailings 3 - 7% Binder
Sand 3 - 0% Binder
Sand 2 - 8% Binder
Sand 3 - 8% Binder
Sand 2 - 6% Binder
Base Metal Tailings 1b - 0% Binder
Sand 3 - 0% Binder
Base Metal Tailings 3 - 0% Binder
Precious Metal Tailings 1 - 5% Binder
Precious Metal Tailings 2 - 3% Binder
Precious Metal Tailings 2 - 5% Binder
Precious Metal Tailings 3 - 0% Binder
Sand 1 - 5% Binder
Sand 1 - 5% Binder



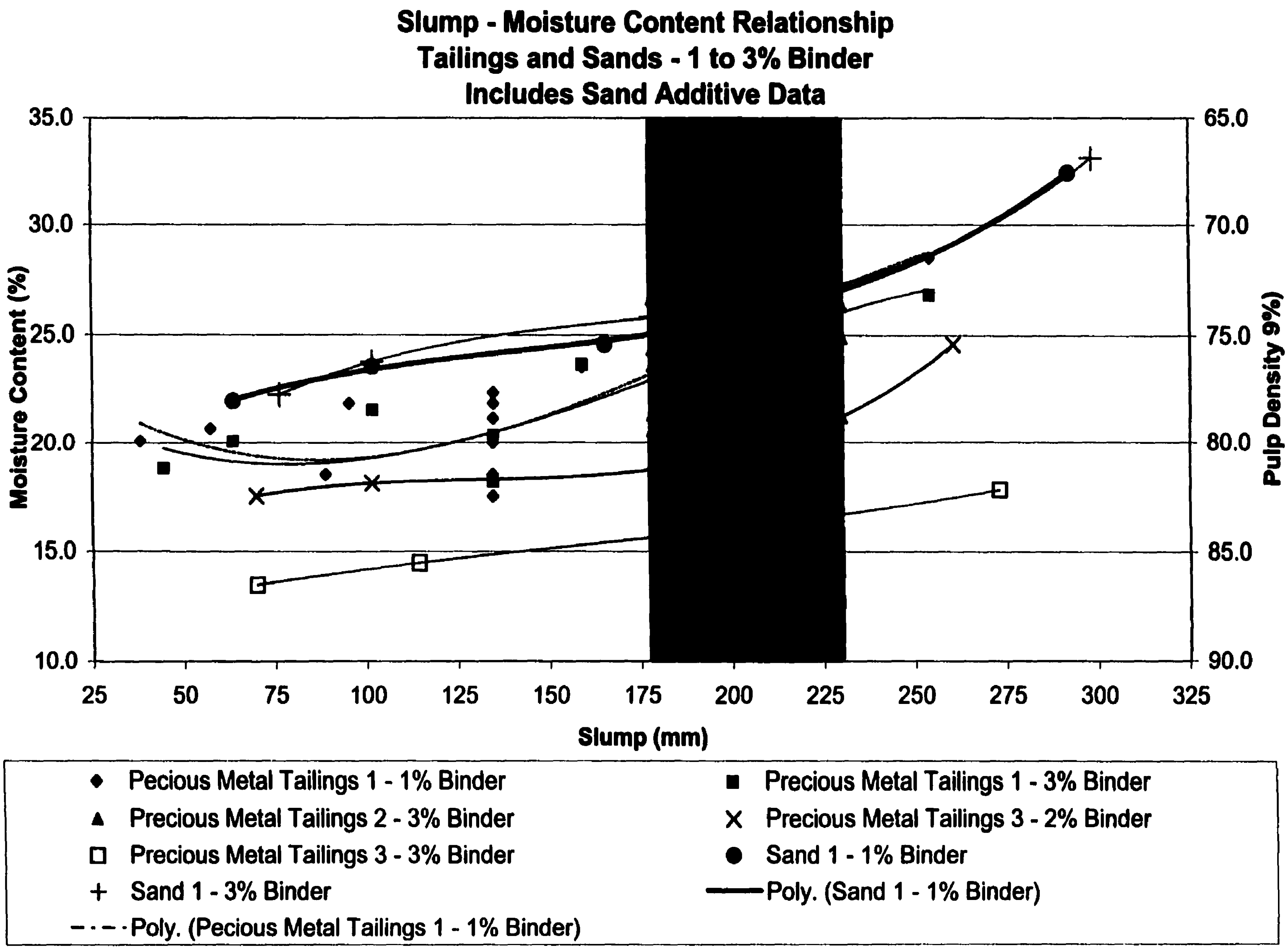
Slump-Moisture Content Relationships for Cemented Tailings and Sand Paste Fill (0 - 5% binder)



Slump-Moisture Content Relationship for Cemented Tailings and Sand Paste Fill (4 - 6% binder)



Slump-Moisture Content Relationship for Cemented Tailings and Sand Paste Fills (7 - 10% binder)



Slump-Moisture Content Relationships for Cemented Tailings and Sand Paste Fill (1 - 3% binder)

APPENDIX B

APPENDIX B-1

APPENDIX B-1

Mechanical Properties

Unconfined Compressive Strength Test Results

The unconfined compressive strength tests were conducted in accordance with ASTM (C-192 and C-39) specifications.

The results for the various test conditions are presented in the following pages:

Fill Source Name	Bulk Density (kg/m3)	Sample Diameter (mm)	Total Binder Content (%)	Binder Composition	Curing Period (Days)	Compressive Strength (MPa)	Deformation Modulus (GPa)	Void Ratio	Porosity (%)
Precious Metal Tailings	1734 - 1772 ***1755***	102	3	3% PC	14	0.295	0.028	0.60	37.3
			3	3% PC	14	0.280	0.029	0.60	37.4
			3	3% PC	14	0.330	0.029	0.60	37.3
			3	3% PC	14	0.289	0.022		
			3	3% PC	14	0.256	0.019		
			3	3% PC	14	0.277	0.020		
			3	1.5% PC+1.5% FA	14	0.290	0.027	0.61	37.7
			3	1.5% PC+1.5% FA	14	0.276	0.024	0.61	37.9
			3	1.5% PC+1.5% FA	14	0.332	0.027	0.61	37.8
			5	5% PC	14	0.411	0.045	0.60	37.3
			5	5% PC	14	0.501	0.052	0.59	36.9
			5	5% PC	14	0.530	0.059	0.59	36.9
			5	5% PC	14	0.400	0.040		
			5	5% PC	14	0.387	0.043		
			5	5% PC	14	0.379	0.045		
			5	2.5% PC+2.5% FA	14	0.482	0.060	0.59	37.1
			5	2.5% PC+2.5% FA	14	0.501	0.058	0.59	37.0
			5	2.5% PC+2.5% FA	14	0.493	0.056	0.59	37.2
			8	8% PC	14	0.598	0.083		
			8	8% PC	14	0.640	0.082		
			8	8% PC	14	0.619	0.086		
			2	2% Product A	14	0.490	0.026		
			2	2% Product A	14	0.199	0.025		
			2	2% Product A	14	0.196	0.017		
			4	4% Product A	14	0.614	0.071		
			4	4% Product A	14	0.578	0.070		
			4	4% Product A	14	0.623	0.071		
			6	6% Product A	14	1.030	0.143		
			6	6% Product A	14	1.100	0.138		
			6	6% Product A	14	1.030	0.110		
			8	8% Product A	14	1.406	0.306		
			8	8% Product A	14	1.504	0.342		
			8	8% Product A	14	1.319	0.412		
			3	3% PC	28	0.411	0.035	0.61	37.9
			3	3% PC	28	0.422	0.039	0.62	38.1
			3	3% PC	28	0.390	0.038	0.59	37.1
			3	3% PC	28	0.309	0.024		
			3	3% PC	28	0.289	0.024		
			3	3% PC	28	0.296	0.026		
			3	1.5% PC+1.5% FA	28	0.432	0.034	0.60	37.4
			3	1.5% PC+1.5% FA	28	0.428	0.032	0.61	37.8
			3	1.5% PC+1.5% FA	28	0.400	0.029	0.60	37.6
			5	5% PC	28	0.570	0.048	0.60	37.3
			5	5% PC	28	0.530	0.059	0.59	37.1
			5	5% PC	28	0.582	0.063	0.59	37.0
			5	5% PC	28	0.404	0.044		
			5	5% PC	28	0.395	0.045		
			5	5% PC	28	0.432	0.046		
			5	2.5% PC+2.5% FA	28	0.564	0.071	0.58	36.8
			5	2.5% PC+2.5% FA	28	0.587	0.073	0.59	37.1
			5	2.5% PC+2.5% FA	28	0.650	0.081	0.59	37.2
			8	8% PC	28	0.703	0.088		
			8	8% PC	28	0.644	0.081		
			8	8% PC	28	0.703	0.093		
			2	2% Product A	28	0.289	0.043		
			2	2% Product A	28	0.231	0.039		
			2	2% Product A	28	0.240	0.032		
			4	4% Product A	28	0.920	0.092		
			4	4% Product A	28	0.834	0.104		
			4	4% Product A	28	0.891	0.111		
			6	6% Product A	28	1.340	0.223		
			6	6% Product A	28	1.270	0.198		
			6	6% Product A	28	1.110	0.308		
			8	8% Product A	28	1.874	0.469		
			8	8% Product A	28	1.899	0.475		
			8	8% Product A	28	1.763	0.519		

Effect of Alternative Binders
Precious Metal Tailings

Fill Source Name	Bulk Density (kg/m3)	Sample Diameter (mm)	Total Binder Content (%)	Binder Composition	Curing Period (Days)	Compressive Strength (MPa)	Deformation Modulus (GPa)	Void Ratio	Porosity (%)
Precious Metal Tailings	1734 - 1772 *1755*	102	3	3% PC	56	0.476	0.042	0.61	37.8
			3	3% PC	56	0.411	0.036	0.60	37.3
			3	3% PC	56	0.400	0.036	0.60	37.3
			3	3% PC	56	0.303	0.029		
			3	3% PC	56	0.298	0.032		
			3	3% PC	56	0.290	0.031		
			3	1.5% PC+1.5% FA	56	0.470	0.037	0.60	37.4
			3	1.5% PC+1.5% FA	56	0.501	0.040	0.60	37.5
			3	1.5% PC+1.5% FA	56	0.530	0.045	0.60	37.6
			5	5% PC	56	0.662	0.074	0.58	36.8
			5	5% PC	56	0.631	0.066	0.59	37.2
			5	5% PC	56	0.649	0.074	0.59	37.1
			5	5% PC	56	0.487	0.057		
			5	5% PC	56	0.491	0.061		
			5	5% PC	56	0.501	0.066		
			5	2.5% PC+2.5% FA	56	0.723	0.095	0.59	37.2
			5	2.5% PC+2.5% FA	56	0.763	0.103	0.58	36.7
			5	2.5% PC+2.5% FA	56	0.829	0.106	0.59	37.0
			8	8% PC	56	0.783	0.100		
			8	8% PC	56	0.739	0.092		
			8	8% PC	56	0.730	0.091		
			2	2% Product A	56	0.355	0.036		
			2	2% Product A	56	0.337	0.094		
			2	2% Product A	56	0.291	0.052		
			4	4% Product A	56	1.070	0.107		
			4	4% Product A	56	0.920	0.115		
			4	4% Product A	56	0.840	0.127		
			8	8% Product A	56	2.515	0.629		
			8	8% Product A	56	2.145	0.766		
			8	8% Product A	56	2.232	0.797		

Effect of Alternative Binders
Precious Metal Tailings

Fill Source Name	Bulk Density (kg/m3)	Sample Diameter (mm)	Total Binder Content (%)	Binder Composition	Curing Period (Days)	Compressive Strength (MPa)	Deformation Modulus (GPa)	Void Ratio	Porosity (%)
Base Metal Tailings	1635 - 2063 *1864*	102	3	3% PC	14	0.406	0.078	0.89	47.1
			3	3% PC	14	0.506	0.074	0.89	47.1
			3	3% PC	14	0.323	0.044	0.90	47.2
			3	1.5% PC+1.5% FA	14	0.258	0.031	0.95	48.6
			3	1.5% PC+1.5% FA	14	0.198	0.017	0.95	48.6
			3	1.5% PC+1.5% FA	14	0.195	0.013	0.95	48.8
			3	2.7% PC+0.3% X	14	0.605	0.070	1.03	50.8
			3	2.7% PC+0.3% X	14	0.554	0.054	1.17	53.8
			3	2.7% PC+0.3% X	14	0.527	0.061	1.08	52.0
			4	1.5% PC+1.5% FA+1% Anhydrite	14	0.340	0.047	0.95	48.6
			4	1.5% PC+1.5% FA+1% Anhydrite	14	0.309	0.041	0.94	48.5
			4	1.5% PC+1.5% FA+1% Anhydrite	14	0.309	0.045	0.94	48.5
			5	5% PC	14	0.614	0.140	0.81	44.7
			5	5% PC	14	0.578	0.161	0.74	42.6
			5	5% PC	14	0.623	0.115	0.79	44.2
			5	2.5% PC+2.5% FA	14	1.030	0.258	0.78	43.7
			5	2.5% PC+2.5% FA	14	1.099	0.229	0.80	44.4
			5	2.5% PC+2.5% FA	14	0.961	0.141	0.80	44.4
			5	4.5% PC+0.5% X	14	1.282	0.221	1.03	50.8
			5	4.5% PC+0.5% X	14	1.295	0.208	1.04	51
			5	4.5% PC+0.5% X	14	1.418	0.244	1.03	50.6
			6	2.5% PC+2.5% FA+1% Anhydrite	14	1.030	0.117	0.72	42.0
			6	2.5% PC+2.5% FA+1% Anhydrite	14	1.100	0.117	0.73	42.1
			6	2.5% PC+2.5% FA+1% Anhydrite	14	0.961	0.117	0.72	42.0
			3	3% PC	28	0.527	0.061	0.89	47.1
			3	3% PC	28	0.582	0.050	0.89	47.1
			3	3% PC	28	0.612	0.048	0.90	47.2
			3	1.5% PC+1.5% FA	28	0.289	0.029	0.94	48.4
			3	1.5% PC+1.5% FA	28	0.231	0.039	0.89	47.1
			3	1.5% PC+1.5% FA	28	0.242	0.040	0.91	47.6
			3	2.7% PC+0.3% X	28	0.684	0.095	1.09	52.1
			3	2.7% PC+0.3% X	28	0.580	0.085	1.17	53.8
			3	2.7% PC+0.3% X	28	0.605	0.086	1.11	52.7
			4	1.5% PC+1.5% FA+1% Anhydrite	28	0.570	0.086	0.94	48.5
			4	1.5% PC+1.5% FA+1% Anhydrite	28	0.600	0.107	0.95	48.6
			4	1.5% PC+1.5% FA+1% Anhydrite	28	0.600	0.100	0.95	48.7
			5	5% PC	28	0.920	0.148	0.80	44.4
			5	5% PC	28	0.834	0.298	0.81	44.6
			5	5% PC	28	0.890	0.297	0.81	44.7
			5	2.5% PC+2.5% FA	28	1.340	0.186	0.78	43.7
			5	2.5% PC+2.5% FA	28	1.264	0.192	0.82	45.2
			5	2.5% PC+2.5% FA	28	1.110	0.154	0.84	45.5
			5	4.5% PC+0.5% X	28	1.418	0.253	1.04	50.9
			5	4.5% PC+0.5% X	28	1.554	0.235	1.04	50.9
			5	4.5% PC+0.5% X	28	1.344	0.232	1.04	50.9
			6	2.5% PC+2.5% FA+1% Anhydrite	28	1.340	0.176	0.76	43.2
			6	2.5% PC+2.5% FA+1% Anhydrite	28	1.270	0.167	0.77	43.6
			6	2.5% PC+2.5% FA+1% Anhydrite	28	1.110	0.168	0.72	41.7

Effect of Alternative Binders
Base Metal Tailings

Fill Source Name	Bulk Density (kg/m3)	Sample Diameter (mm)	Total Binder Content (%)	Binder Composition	Curing Period (Days)	Compressive Strength (MPa)	Deformation Modulus (GPa)	Void Ratio	Porosity (%)
Base Metal Tailings	1635 - 2063 ***1864***	102	3	3% PC	56	0.513	0.095	0.89	47.1
			3	3% PC	56	0.522	0.113	0.89	47.0
			3	3% PC	56	0.596	0.124	0.96	48.9
			3	1.5% PC+1.5% FA	56	0.355	0.077	0.93	48.3
			3	1.5% PC+1.5% FA	56	0.337	0.062	0.94	48.4
			3	1.5% PC+1.5% FA	56	0.290	0.044	0.95	48.8
			3	2.7% PC+0.3% X	56	0.737	0.102	1.12	52.8
			3	2.7% PC+0.3% X	56	0.633	0.106	1.12	52.8
			3	2.7% PC+0.3% X	56	0.784	0.131	1.11	52.7
			4	1.5% PC+1.5% FA+1% Anhydrite	56	0.620	0.115	0.95	48.6
			4	1.5% PC+1.5% FA+1% Anhydrite	56	0.660	0.114	0.95	48.6
			4	1.5% PC+1.5% FA+1% Anhydrite	56	0.830	0.119	0.95	48.6
			5	5% PC	56	1.070	0.382	0.80	44.4
			5	5% PC	56	0.920	0.354	0.79	44.2
			5	5% PC	56	0.840	0.420	0.79	44.3
			5	2.5% PC+2.5% FA	56	1.380	0.197	0.79	44.2
			5	2.5% PC+2.5% FA	56	2.000	0.233	0.78	43.7
			5	2.5% PC+2.5% FA	56	1.400	0.206	0.72	42.0
			5	4.5% PC+0.5% X	56	1.529	0.364	1.02	50.6
			5	4.5% PC+0.5% X	56	1.591	0.419	0.99	49.8
			5	4.5% PC+0.5% X	56	1.714	0.857	0.99	49.8
			6	2.5% PC+2.5% FA+1% Anhydrite	56	0.921	0.139	0.76	43.2
			6	2.5% PC+2.5% FA+1% Anhydrite	56	1.38	0.186	0.77	43.5
			6	2.5% PC+2.5% FA+1% Anhydrite	56	1.07	0.198	0.82	45.1

Effect of Alternative Binders
Base Metal Tailings

Curing Period *(Days)*	Sample Diameter *(mm)*	Bulk Density *(kg/m3)*	Total Binder Content *(%)*	Fill Composition	Compressive Strength *(MPa)*	Deformation Modulus *(GPa)*
7	76	2115	6	25% Tailings / 75% Sand	1.075	2.809
7	76	2130	6	25% Tailings / 75% Sand	1.140	0.181
7	76	2124	6	25% Tailings / 75% Sand	1.075	1.605
14	76	2108	6	25% Tailings / 75% Sand	1.447	0.221
14	76	2124	6	25% Tailings / 75% Sand	0.811	0.155
14	76	2123	6	25% Tailings / 75% Sand	0.987	0.506
28	76	2094	6	25% Tailings / 75% Sand	1.557	0.115
28	76	2066	6	25% Tailings / 75% Sand	1.776	2.565
28	76	2088	6	25% Tailings / 75% Sand	1.623	1.235
7	102	2132	6	25% Tailings / 75% Sand	0.789	0.414
7	102	2128	6	25% Tailings / 75% Sand	0.764	0.993
14	102	2114	6	25% Tailings / 75% Sand	1.356	3.119
14	102	2099	6	25% Tailings / 75% Sand	1.295	0.120
14	102	2087	6	25% Tailings / 75% Sand	1.369	0.209
28	102	2119	6	25% Tailings / 75% Sand	1.640	0.390
28	102	2064	6	25% Tailings / 75% Sand	1.874	0.129
28	102	2101	6	25% Tailings / 75% Sand	1.640	0.080
7	152	2213	6	25% Tailings / 75% Sand	1.632	0.136
7	152	2198	6	25% Tailings / 75% Sand	1.615	0.116
14	152	2184	6	25% Tailings / 75% Sand	1.886	0.175
14	152	2200	6	25% Tailings / 75% Sand	1.738	0.138
28	152	2151	6	25% Tailings / 75% Sand	3.081	0.363
28	152	2157	6	25% Tailings / 75% Sand	3.059	0.175
7	76	2239	6	50% Tailings / 50% Sand	0.702	0.270
7	76	2245	6	50% Tailings / 50% Sand	0.768	0.041
7	76	2248	6	50% Tailings / 50% Sand	0.285	0.014
14	76	2196	6	50% Tailings / 50% Sand	0.636	0.076
14	76	2130	6	50% Tailings / 50% Sand	0.482	0.066
14	76	2196	6	50% Tailings / 50% Sand	0.658	0.115
28	76	2155	6	50% Tailings / 50% Sand	0.482	0.055
28	76	2155	6	50% Tailings / 50% Sand	0.482	0.042
28	76	2142	6	50% Tailings / 50% Sand	0.702	0.261
28	76	2148	6	50% Tailings / 50% Sand	0.598	0.055
7	102	2234	6	50% Tailings / 50% Sand	0.727	0.064
7	102	2218	6	50% Tailings / 50% Sand	0.641	0.064
7	102	2233	6	50% Tailings / 50% Sand	0.851	0.057
14	102	2178	6	50% Tailings / 50% Sand	1.665	0.213
14	102	2203	6	50% Tailings / 50% Sand	1.763	0.358
14	102	2142	6	50% Tailings / 50% Sand	0.419	0.161
28	102	2154	6	50% Tailings / 50% Sand	1.665	0.086
28	102	2148	6	50% Tailings / 50% Sand	2.306	0.169
28	102	2213	6	50% Tailings / 50% Sand	0.604	0.604
28	102	2220	6	50% Tailings / 50% Sand	0.580	0.087
28	102	2191	6	50% Tailings / 50% Sand	0.493	0.061
7	152	2218	6	50% Tailings / 50% Sand	0.934	0.153
7	152	2249	6	50% Tailings / 50% Sand	0.835	0.119
14	152	2270	6	50% Tailings / 50% Sand	1.418	0.168
14	152	2244	6	50% Tailings / 50% Sand	1.363	0.096
28	152	2181	6	50% Tailings / 50% Sand	1.980	2.846
28	152	2227	6	50% Tailings / 50% Sand	1.740	2.030
7	279	3059	6	50% Tailings / 50% Sand	0.538	0.060
14	279	2452	6	50% Tailings / 50% Sand	0.652	0.031
28	279	2685	6	50% Tailings / 50% Sand	1.109	0.091

Composite Fill
Precious Metal Tailings

Curing Period *(Days)*	Sample Diameter *(mm)*	Total Binder Content *(%)*	Compressive Strength *(MPa)*	Deformation Modulus *(GPa)*
14	150	5	4.880	0.968
14	150	5	7.610	1.122
14	150	5	6.490	1.405
14	150	7	4.820	1.990
14	150	7	4.570	0.807
14	150	7	10.880	1.102
14	150	7	5.410	1.082
14	150	7	8.920	24.610
14	150	7	5.120	16.780
14	150	7	5.370	7.482
28	150	5	3.610	4.592
28	150	5	5.270	13.013
28	150	5	4.480	13.755
28	150	5	4.500	17.806
28	150	5	4.000	14.021
28	150	7	7.560	1.229
28	150	7	8.880	0.765
28	150	7	4.290	13.756
28	150	7	7.270	17.529
28	150	7	5.900	22.497
28	150	7	3.630	20.647
56	150	5	7.080	1.311
56	150	5	7.080	1.436
56	150	5	2.490	11.203
56	150	5	3.770	13.605
56	150	5	4.640	6.351
56	150	5	7.560	9.309
56	150	7	8.730	22.030
56	150	7	4.120	14.280
56	150	7	7.050	17.418
14	457	5	3.020	0.509
14	457	5	2.320	0.918
14	457	5	0.820	0.539
14	457	5	2.430	0.267
14	457	5	2.110	1.082
14	457	5	2.290	0.464
14	457	5	4.710	1.892
14	457	5	3.770	1.614
14	457	5	2.370	0.438
14	457	5	2.260	0.107
14	457	5	2.160	0.092

14	457	5	5.680	0.986
14	457	7	2.670	1.360
14	457	7	2.440	0.676
14	457	7	2.910	1.365
14	457	7	3.010	0.609
14	457	7	3.940	2.102
14	457	7	3.390	1.966
14	457	7	3.790	1.204
14	457	7	7.800	1.438
14	457	7	3.700	0.591
14	457	7	3.150	0.796
14	457	7	3.810	2.189
28	457	5	3.080	1.011
28	457	5	2.210	0.522
28	457	5	2.100	0.478
28	457	5	2.180	1.717
28	457	5	2.660	0.941
28	457	5	2.640	0.638
28	457	5	2.690	0.765
28	457	5	3.230	0.967
28	457	5	0.816	0.816
28	457	5	3.390	0.659
28	457	5	2.470	1.714
28	457	5	2.990	1.304
28	457	7	4.010	0.943
28	457	7	3.470	0.848
28	457	7	2.000	6.253
28	457	7	2.610	0.559
28	457	7	2.030	1.127
28	457	7	4.200	0.947
28	457	7	4.370	1.547
28	457	7	3.790	0.558
28	457	7	4.030	0.612
28	457	7	5.630	2.627
28	457	7	4.510	0.987
28	457	7	3.640	0.750
56	457	5	2.500	0.501
56	457	5	1.460	0.584
56	457	5	2.690	1.207
56	457	5	2.830	0.539
56	457	5	3.750	0.642
56	457	5	1.820	0.757
56	457	5	5.470	1.020
56	457	5	2.770	0.737

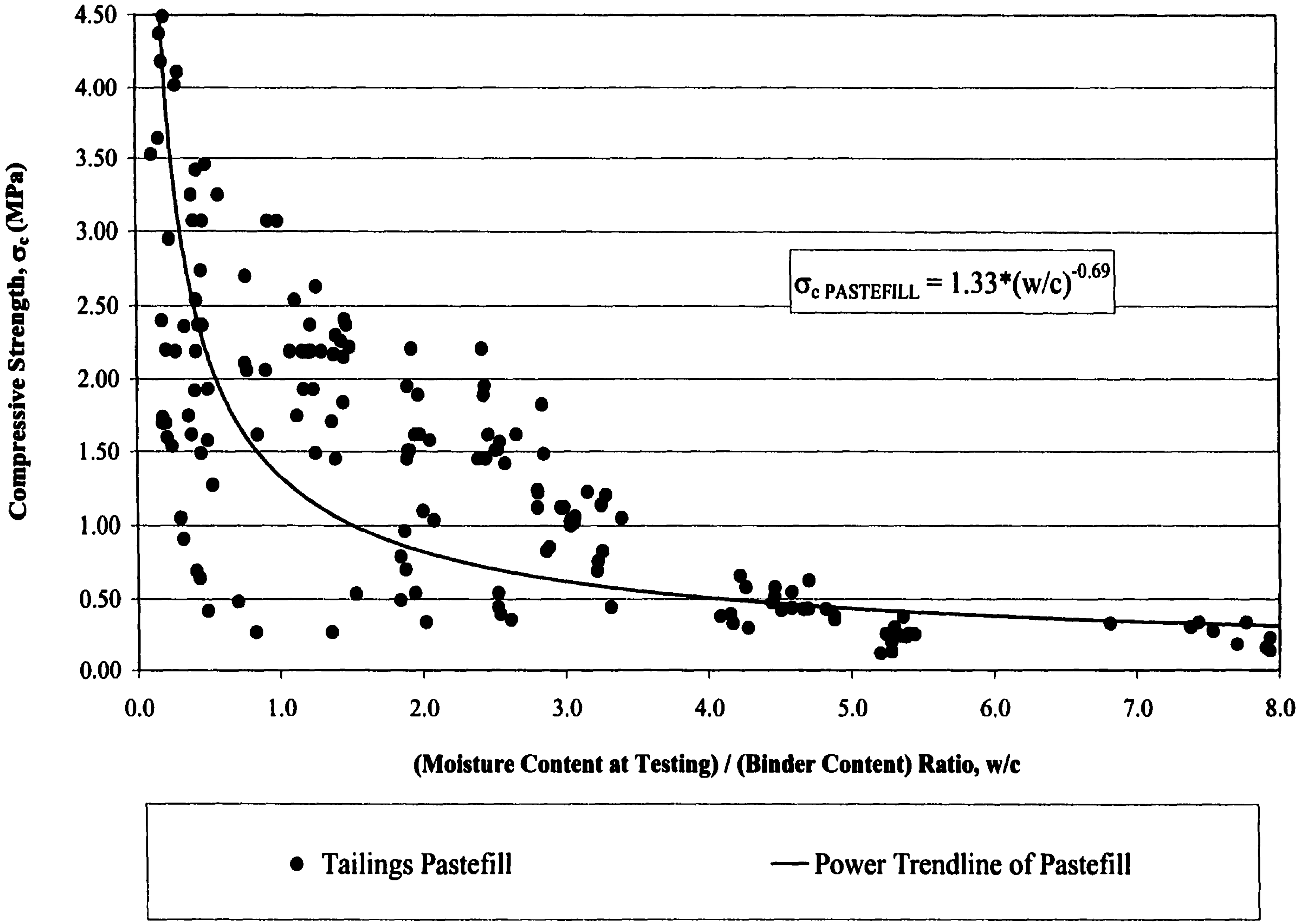
56	457	5	2.470	0.583
56	457	5	2.670	1.044
56	457	5	1.320	0.486
56	457	5	1.420	0.643
56	457	7	3.120	0.696
56	457	7	2.700	1.113
56	457	7	2.870	0.257
56	457	7	2.450	0.440
56	457	7	3.510	1.436
56	457	7	2.380	2.485
56	457	7	1.620	1.109
56	457	7	0.920	0.329
56	457	7	6.740	1.273
56	457	7	3.760	1.395
56	457	7	3.030	0.715
56	457	7	4.370	0.946

APPENDIX B-2

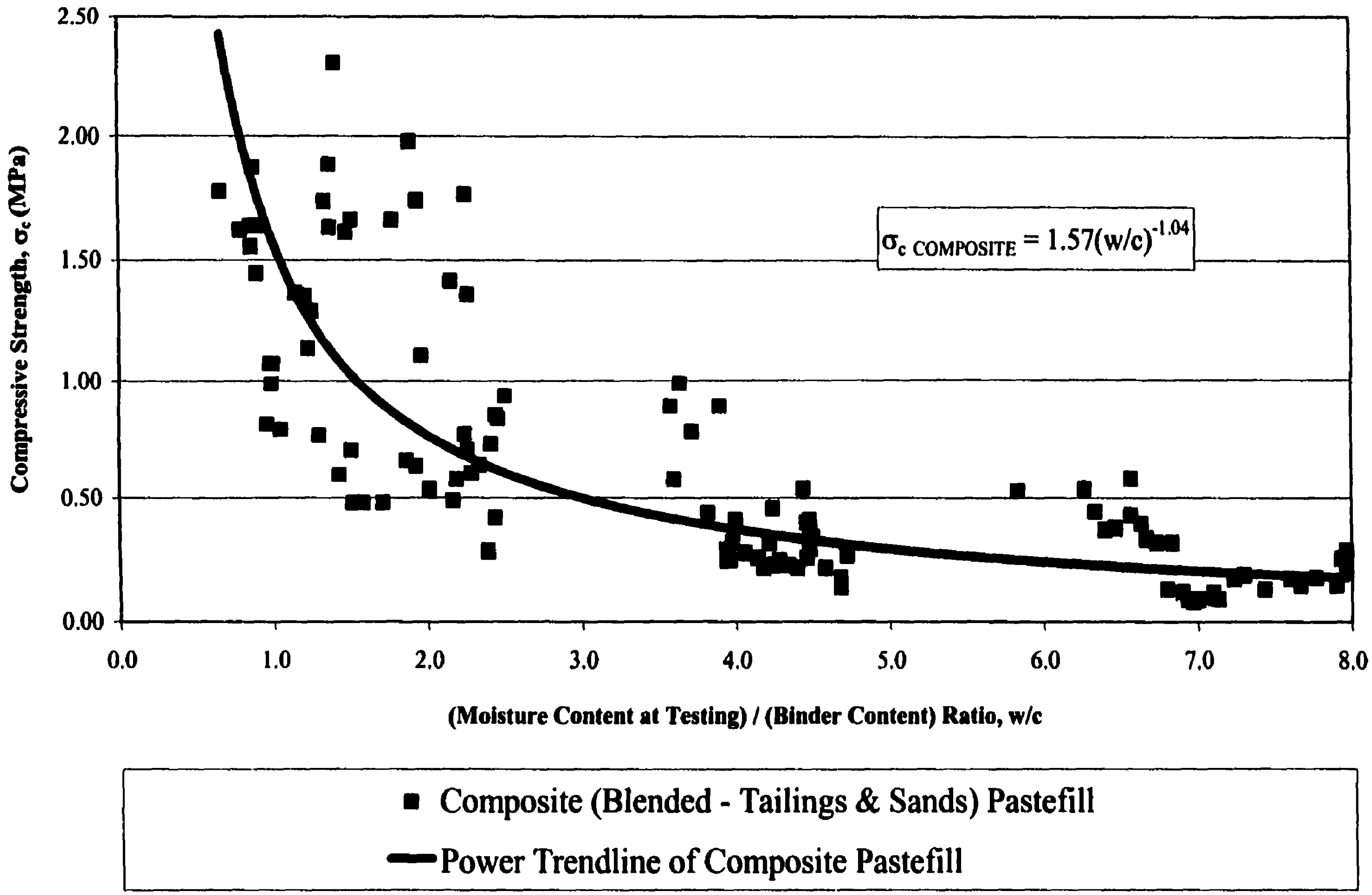
APPENDIX B-2

Relationship between unconfined compressive strength and water cement (W/C) ratio for the studied materials.

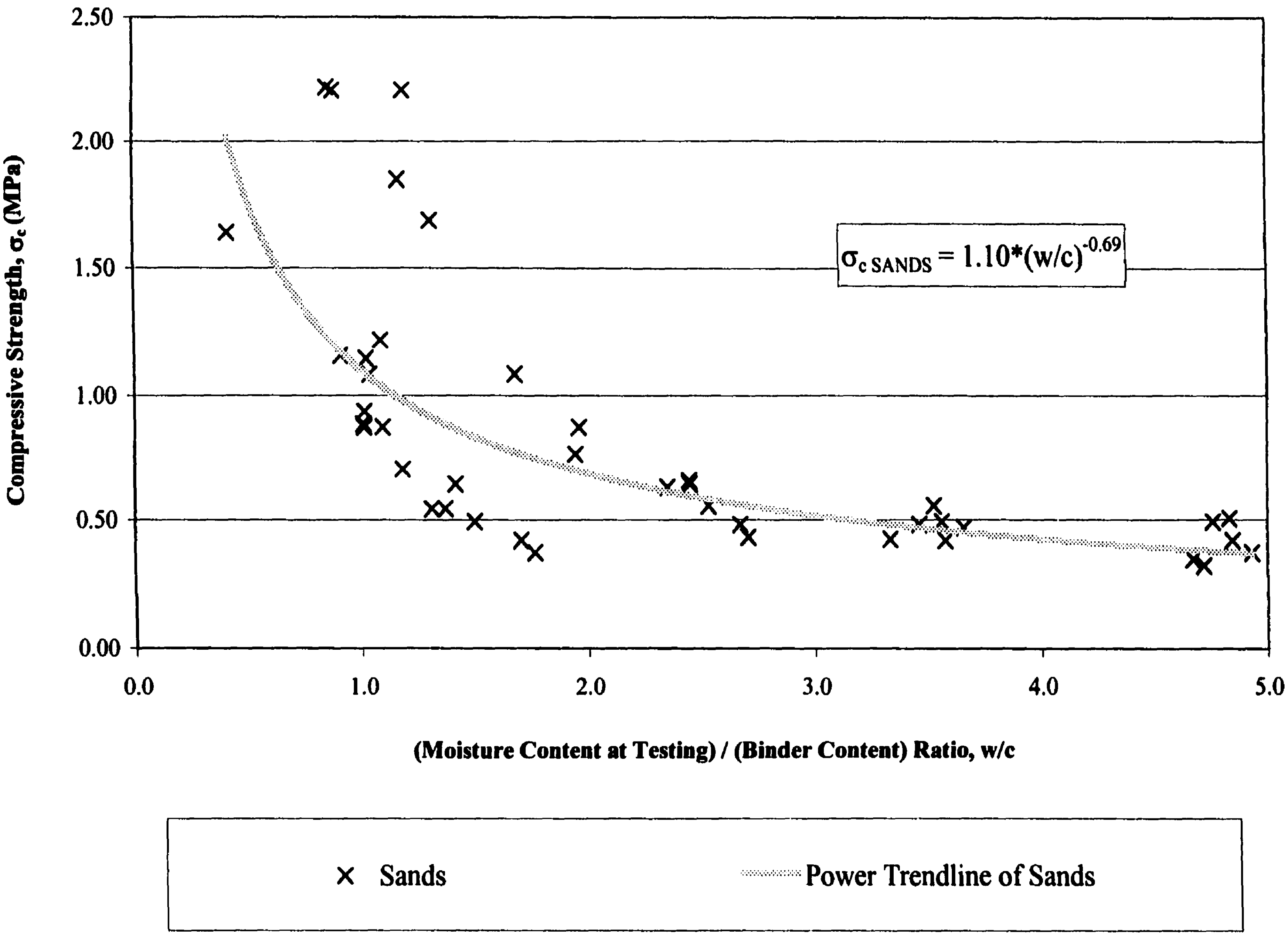
This appendix contains the graphs of unconfined compressive strength (σ_c)versus water/cement (w/c) ratio for the straight tailings, composite and sand pastefill samples.



Relationship between Compressive Strength and Moisture/Binder Content Ratio for Tailings Pastefill



Relationship between Compressive Strength and Moisture/Binder Content Ratio
for Composite (Blended - Tailings and Sand) Pastefill



Relationship between Compressive Strength and Moisture/Binder Content Ratio for Sand Fill

APPENDIX C

APPENDIX C-1

APPENDIX C-1

Triaxial Compressive Strength Tests

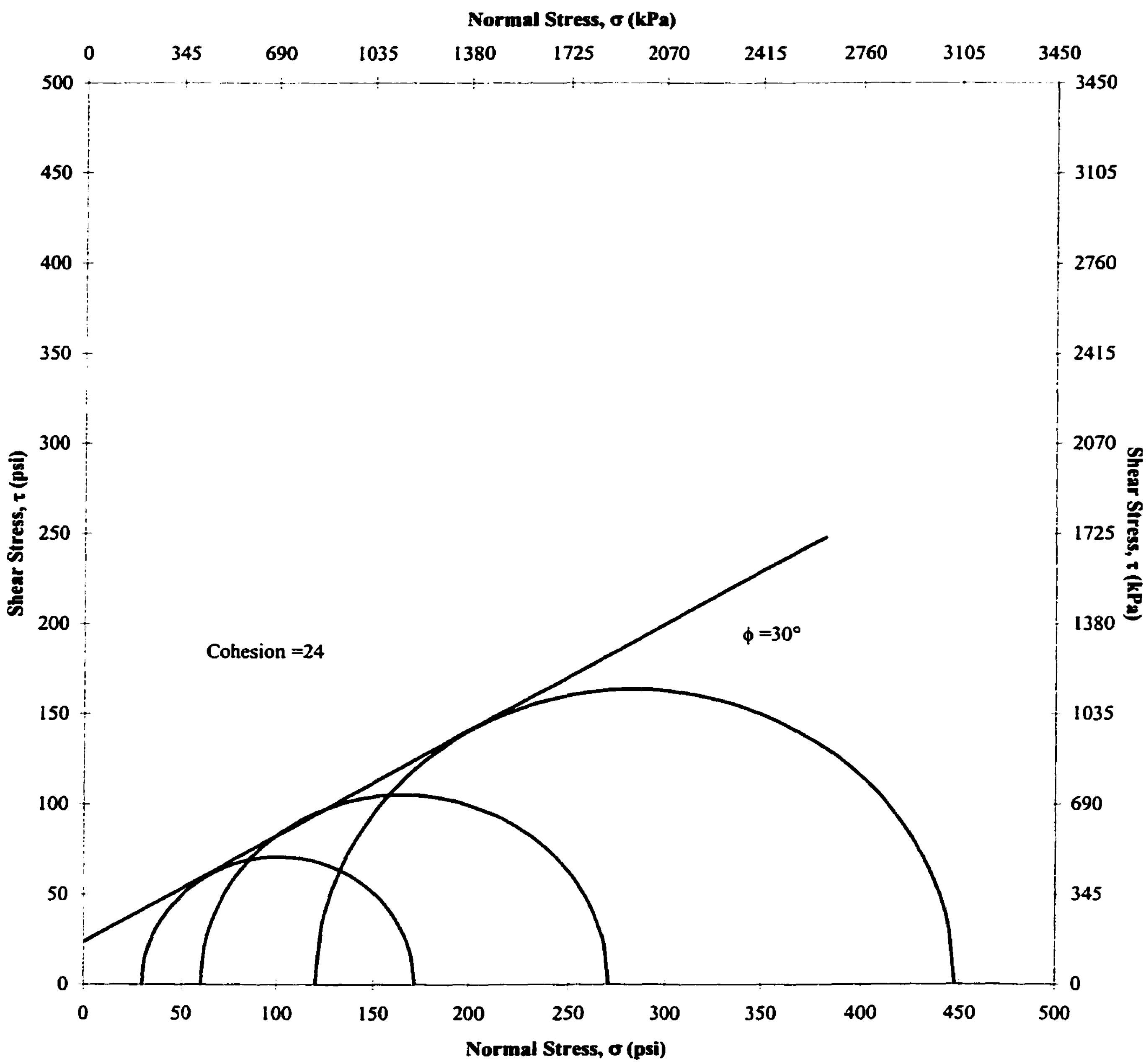
Shear Strength Parameters

Mohr's Circles and Failure Envelope Parameters for the pastefill samples are summarized in this section:

Mohr's Circle Parameters for Uncemented Paste Fill Samples

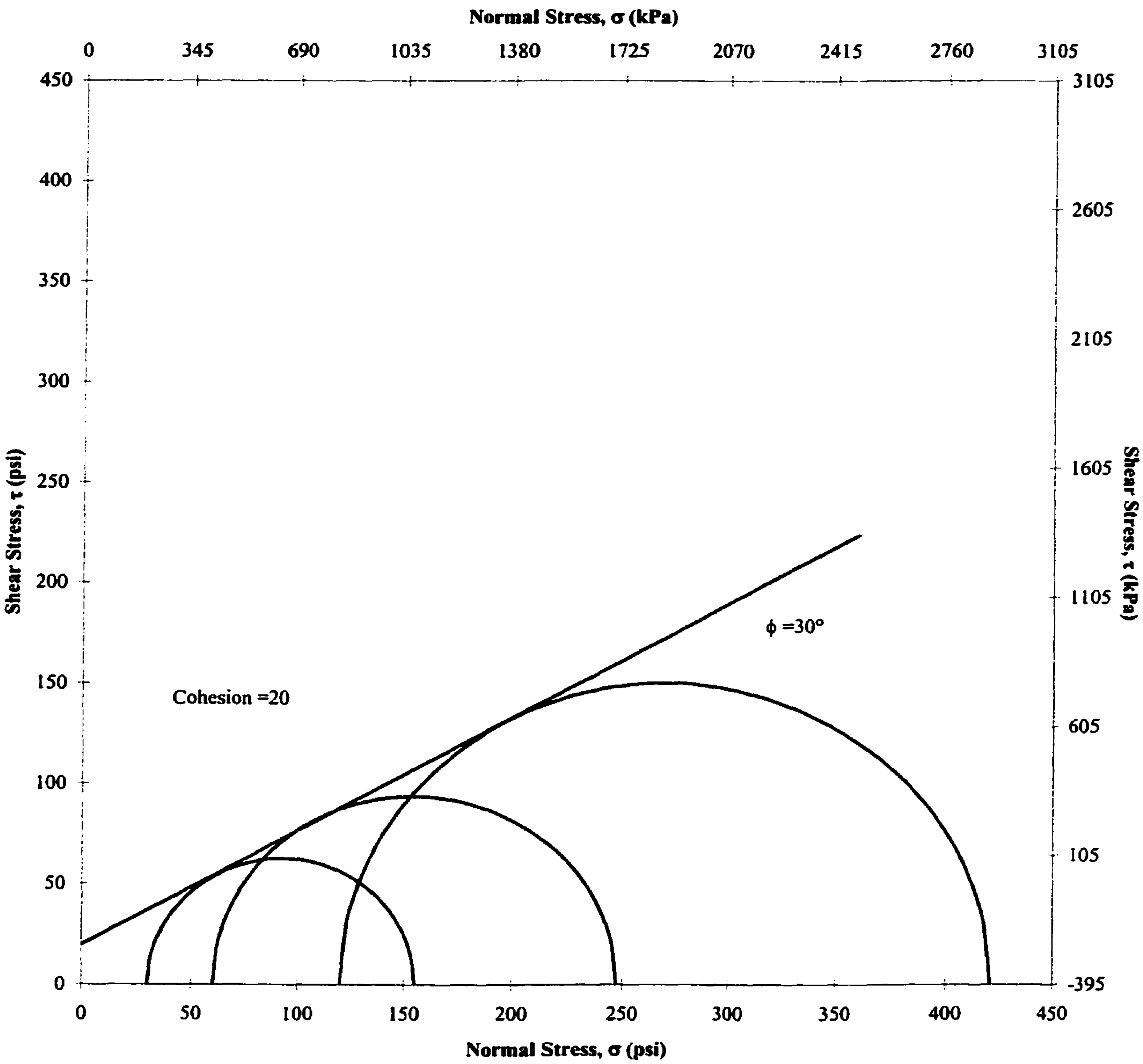
Sample I.D.	Curing Period	Confining Pressure (σ_3)	Deviator Stress (σ_3 - σ_3)	Avg. Modulus of Deformation (E)	Friction Angle ϕ^0	Cohesion (C)
		(kPa)	(kPa)	(MPa)		(kPa)
Total Tailings Base Metal	7	207 414 828	951 1359 1933	16.5 29.6 22.2	26	207
	14	207 414 828	991 1324 2037		27	193
	28	207 414 828	930 1366 2058		28	172
Total Tailings Precious Metal	7	207 414 828	978 1453 2259	16.5 22.4 50.8	30	165
	14	207 414 828	862 1292 2072		30	138
	28	207 414 828	984 1536 2613		35	117

Mohr-Coloumb Failure Envelope
Precious Metal Tailings - Uncemented

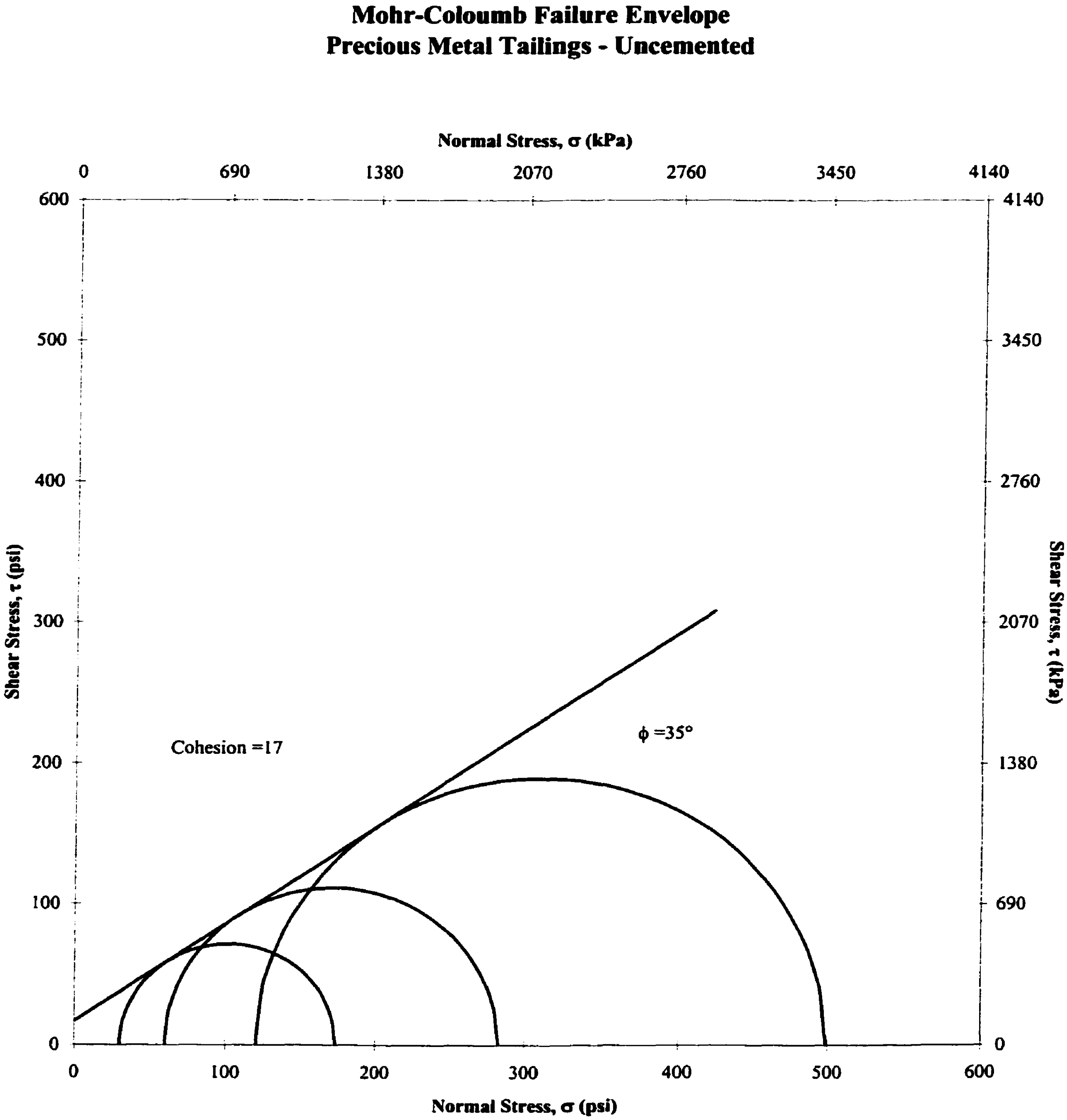


Typical Mohr's Circle Failure Envelope for Uncemented Precious Metal Tailings
(7 days curing)

Mohr-Coloumb Failure Envelope
Precious Metal Tailings - Uncemented

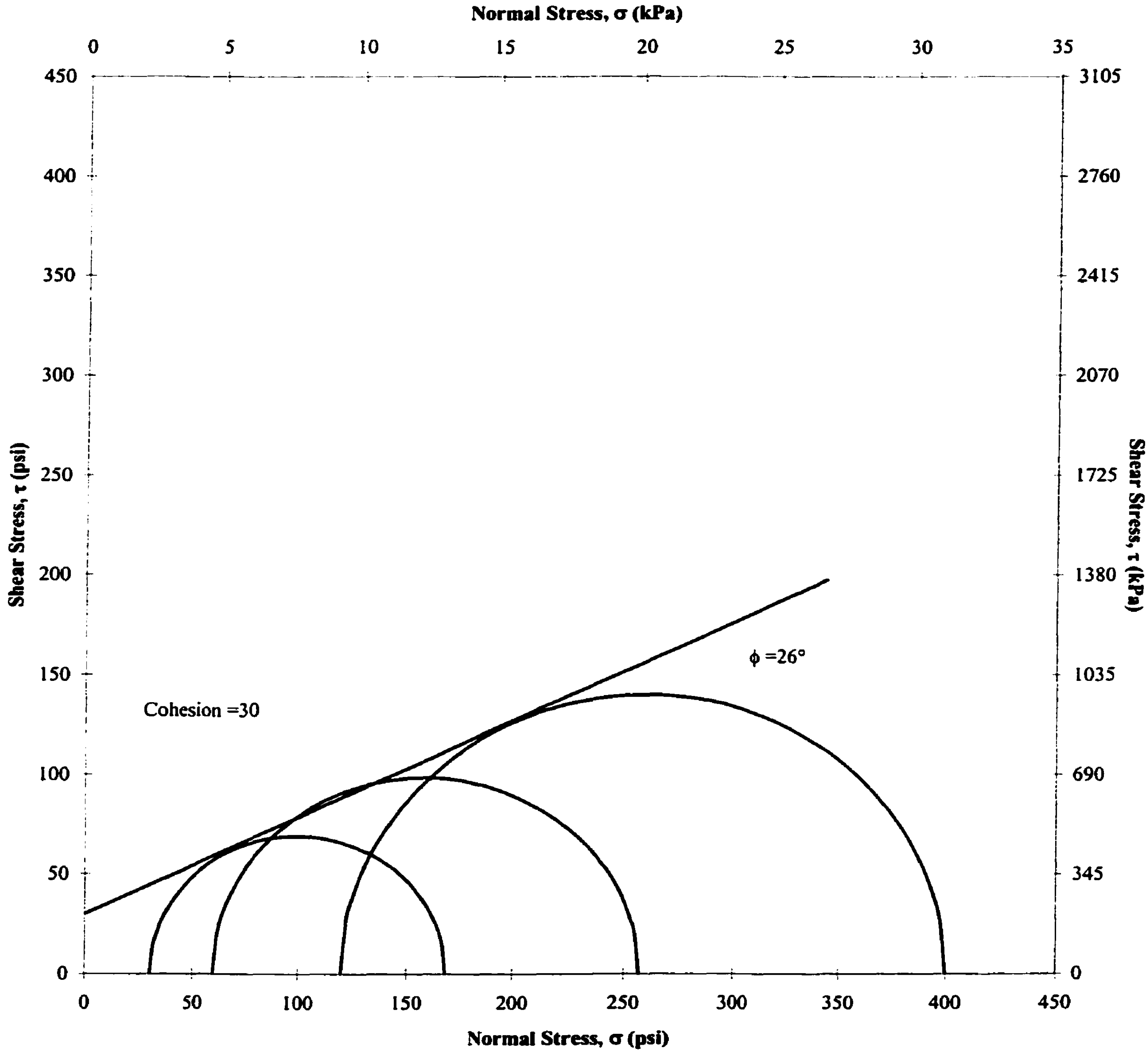


Typical Mohr's Circle Failure Envelope for Uncemented Precious Metal Tailings
(14 days curing)



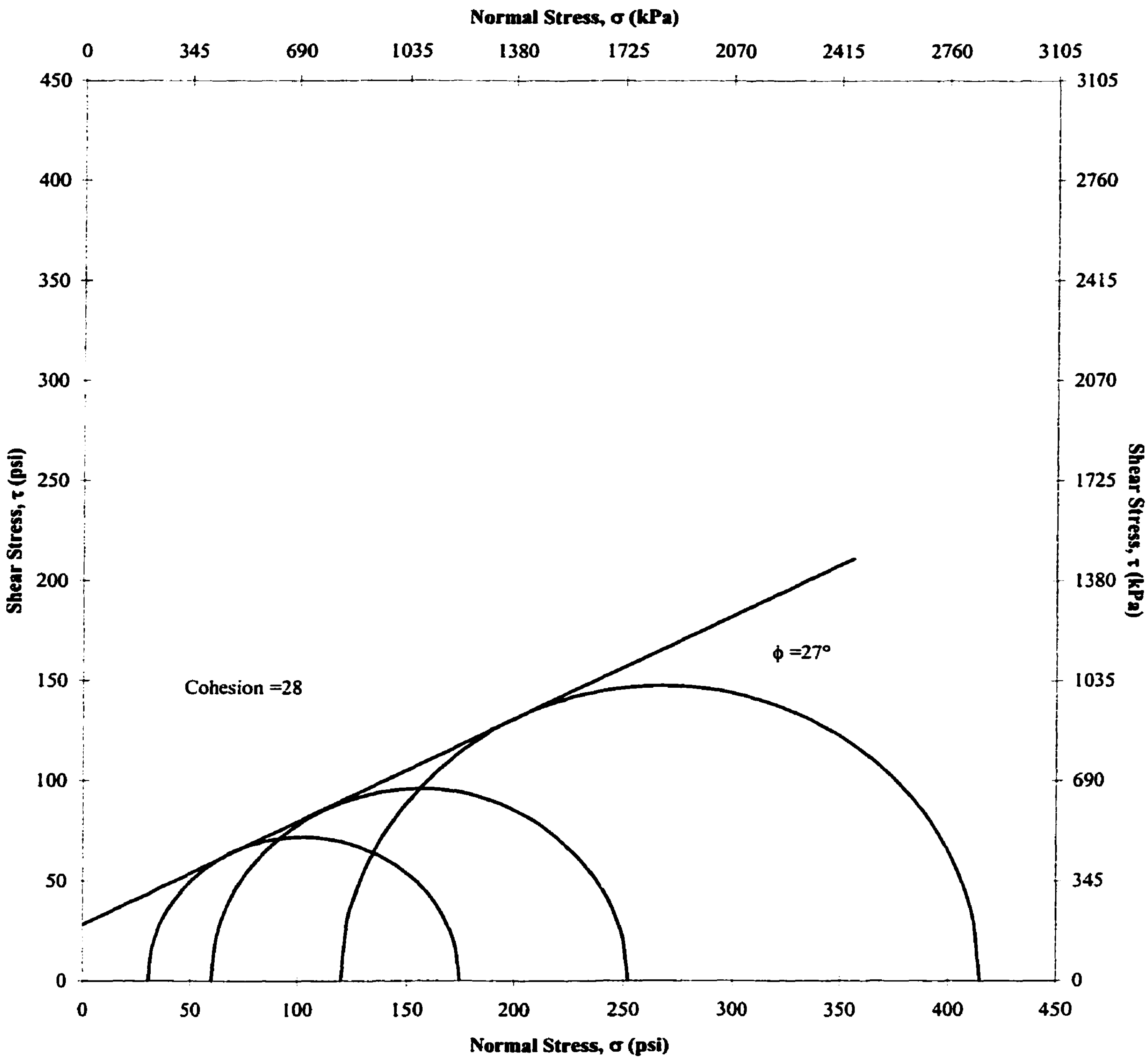
Typical Mohr's Circle Failure Envelope for Uncemented Precious Metal Tailings
(28 days curing)

Mohr-Coloumb Failure Envelope
Base Metal Tailings - Uncemented



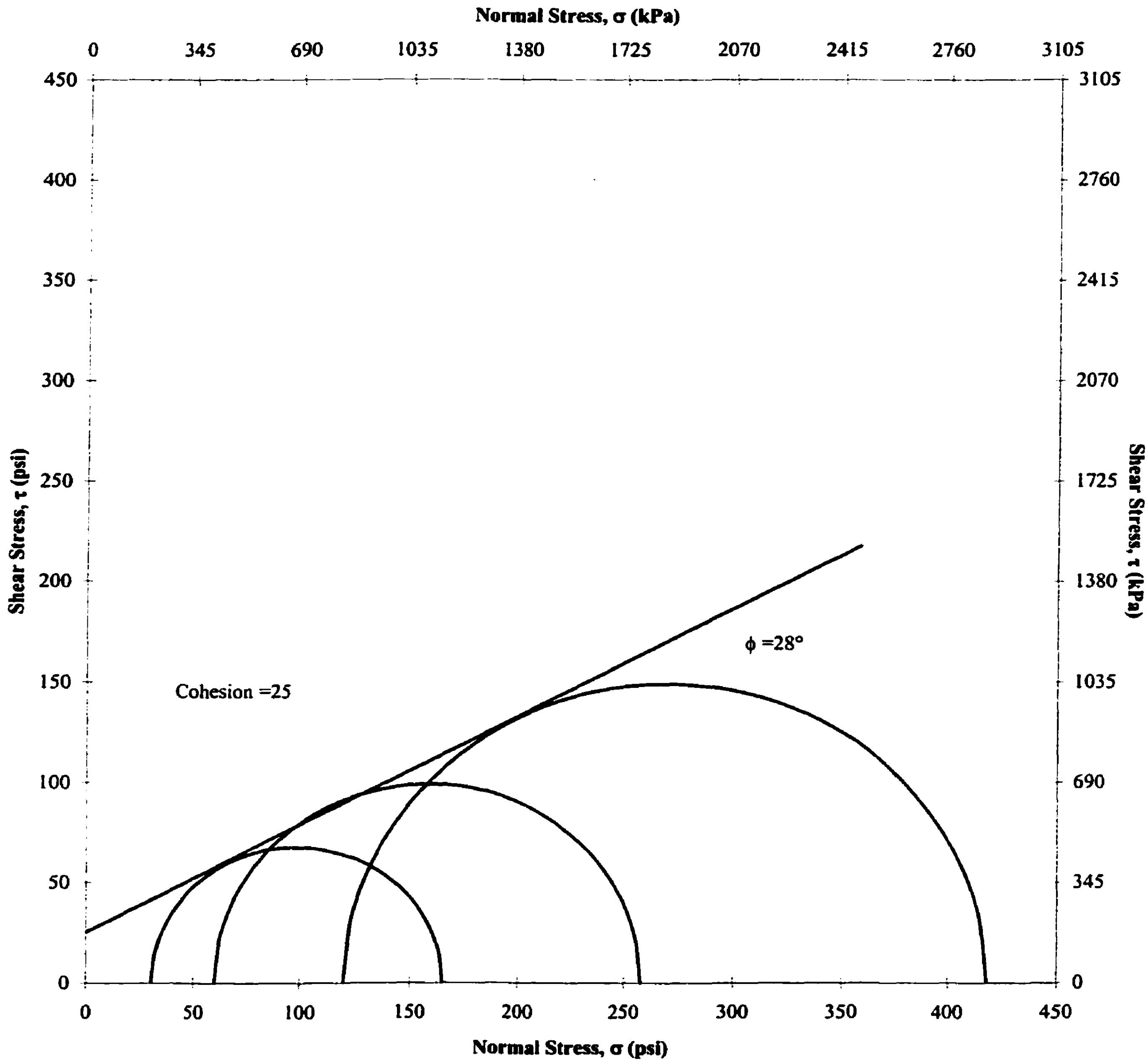
Typical Mohr's Circle Failure Envelope for Uncemented Base Metal Tailings
(7 days curing)

Mohr-Coloumb Failure Envelope
Base Metal Tailings - Uncemented



Typical Mohr's Circle Failure Envelope for Uncemented Base Metal Tailings
(14 days curing)

Mohr-Coloumb Failure Envelope
Base Metal Tailings - Uncemented

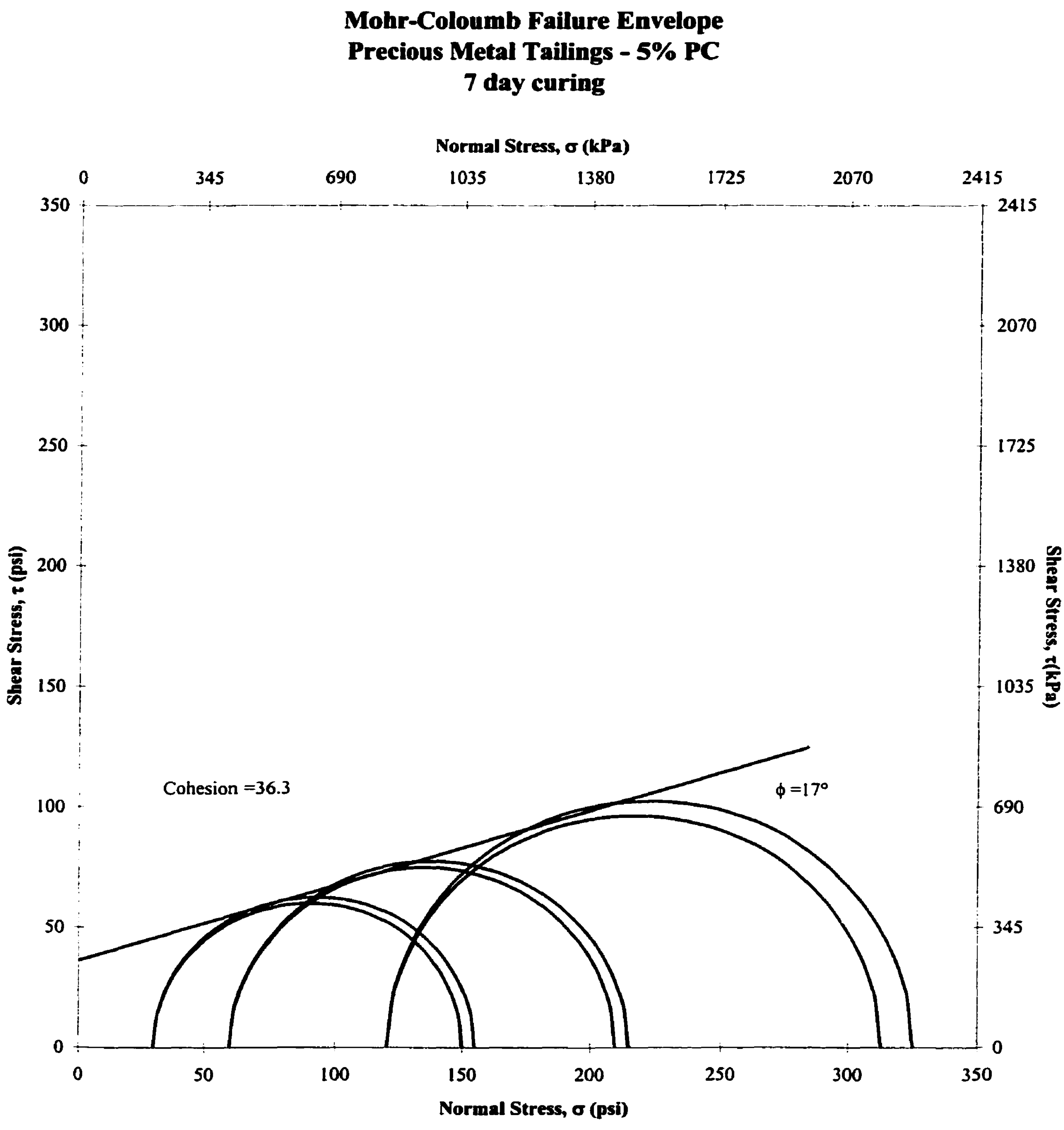


Typical Mohr's Circle Failure Envelope for Uncemented Base Metal Tailings
(28 days curing)

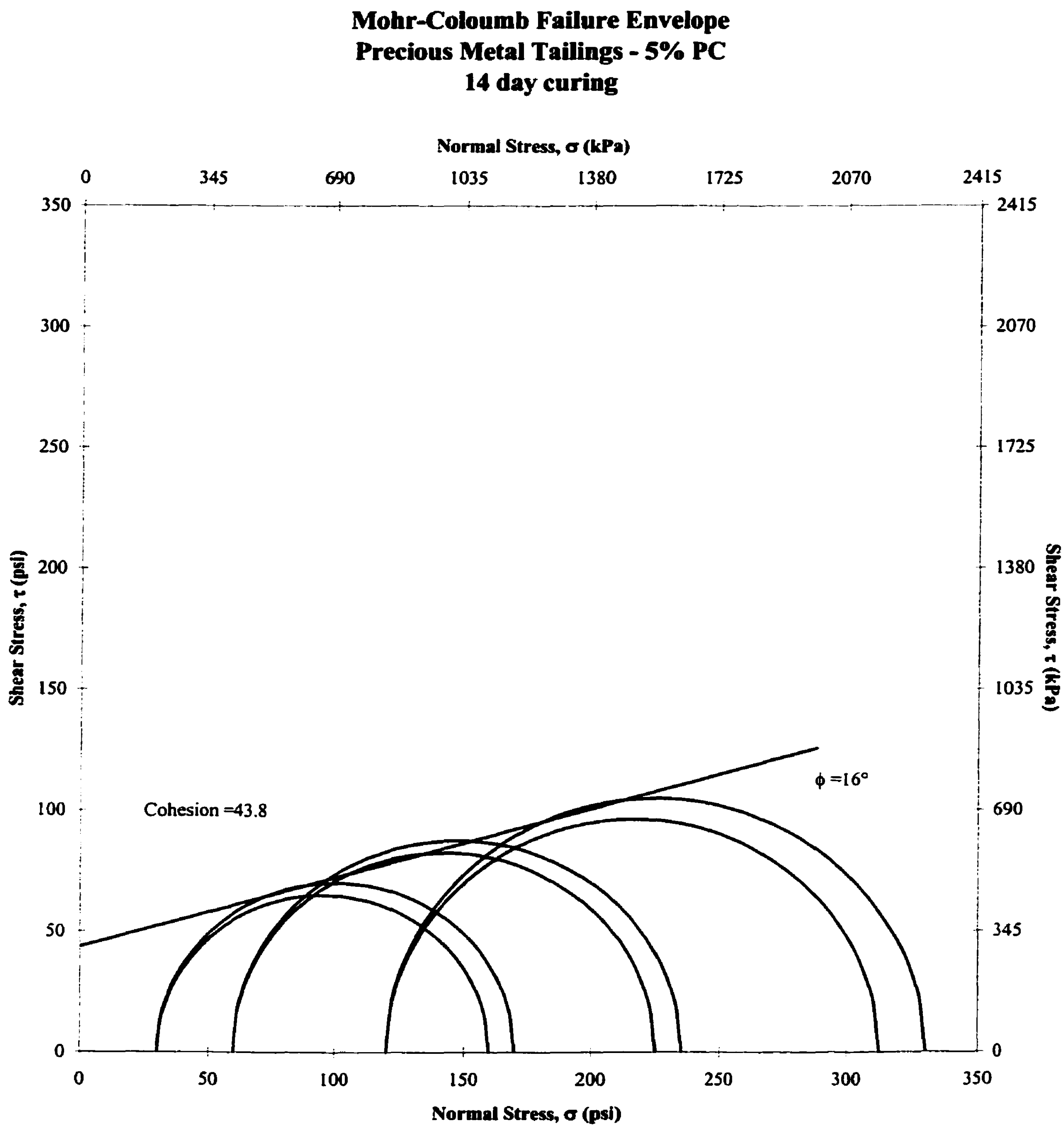
Mohr's Circle Parameters for Cemented Paste Fill Samples

Precious Metal Tailings (C_u = 5.3, % - 20μm = 42.3%)

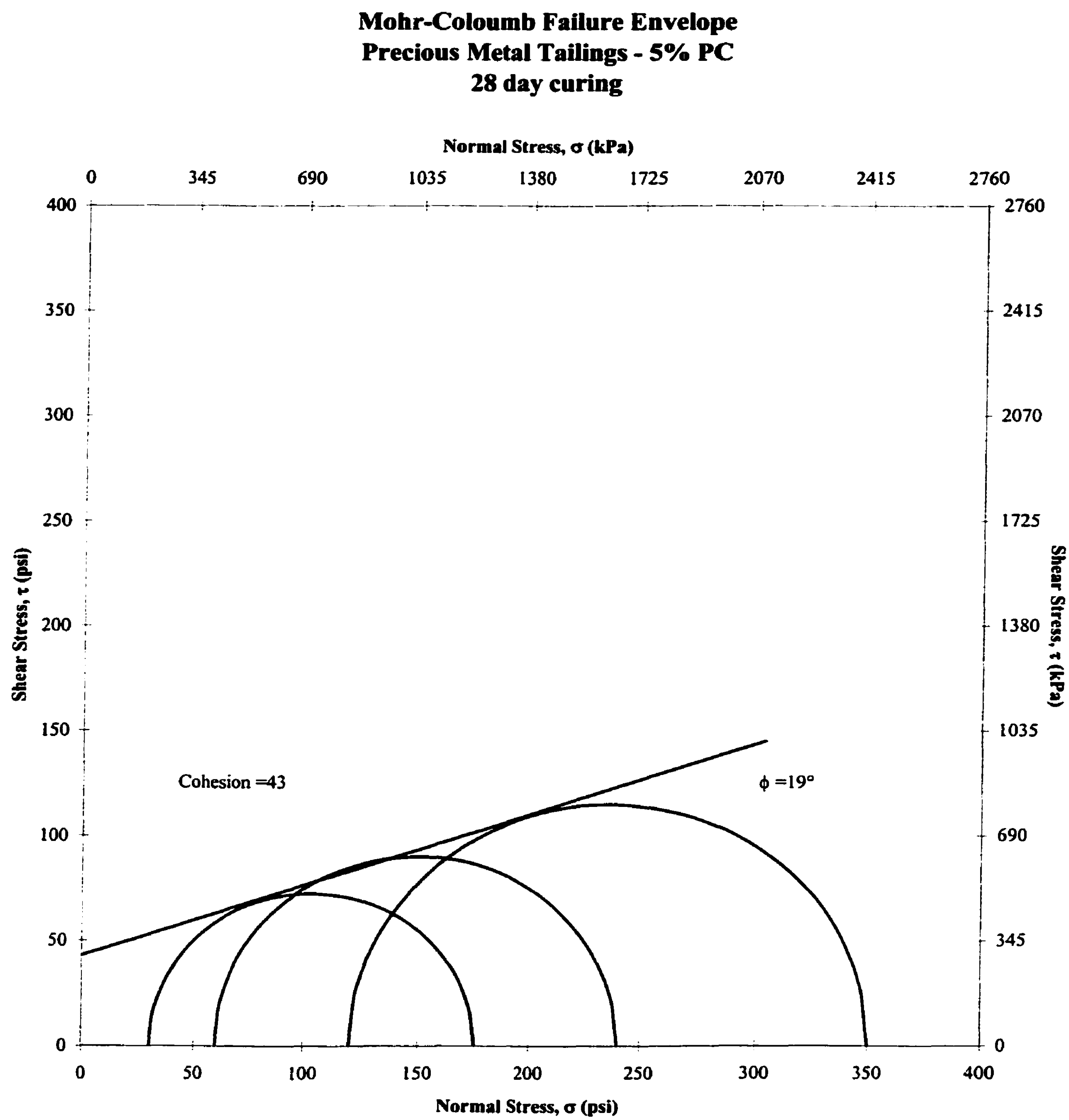
Cement Content (%)	Curing Period (days)	Confining Pressure (σ_3) (kPa)	Deviator Stress ($\sigma_1 - \sigma_3$) (kPa)	Deformation Modulus E (MPa)	Friction Angle ϕ°	Cohesion Range C (kPa)
5	7	207 414 828 207 414 828	828 1034 1329 862 1069 1414	12.6 20.5 17.5 12.8 17.5 19.5	17	250
	14	207 414 828 207 414 828	897 1207 1329 966 1138 1448	19.7 41.0 12.8 14.9 16.4 12.1	16	302
	28	207 414 828	1000 1241 1586	42.0 65.0 68.0	19	297



Typical Mohr's Circle Failure Envelope for Cemented Precious Metal Tailings Paste Fill
(7 days curing)



Typical Mohr's Circle Failure Envelope for Cemented Precious Metal Tailings Paste Fill
(14 days curing)



Typical Mohr's Circle Failure Envelope for Cemented Precious Metal Tailings Paste Fill (28 days curing)

Mohr's Circle Parameters for Cemented Paste Fill Samples

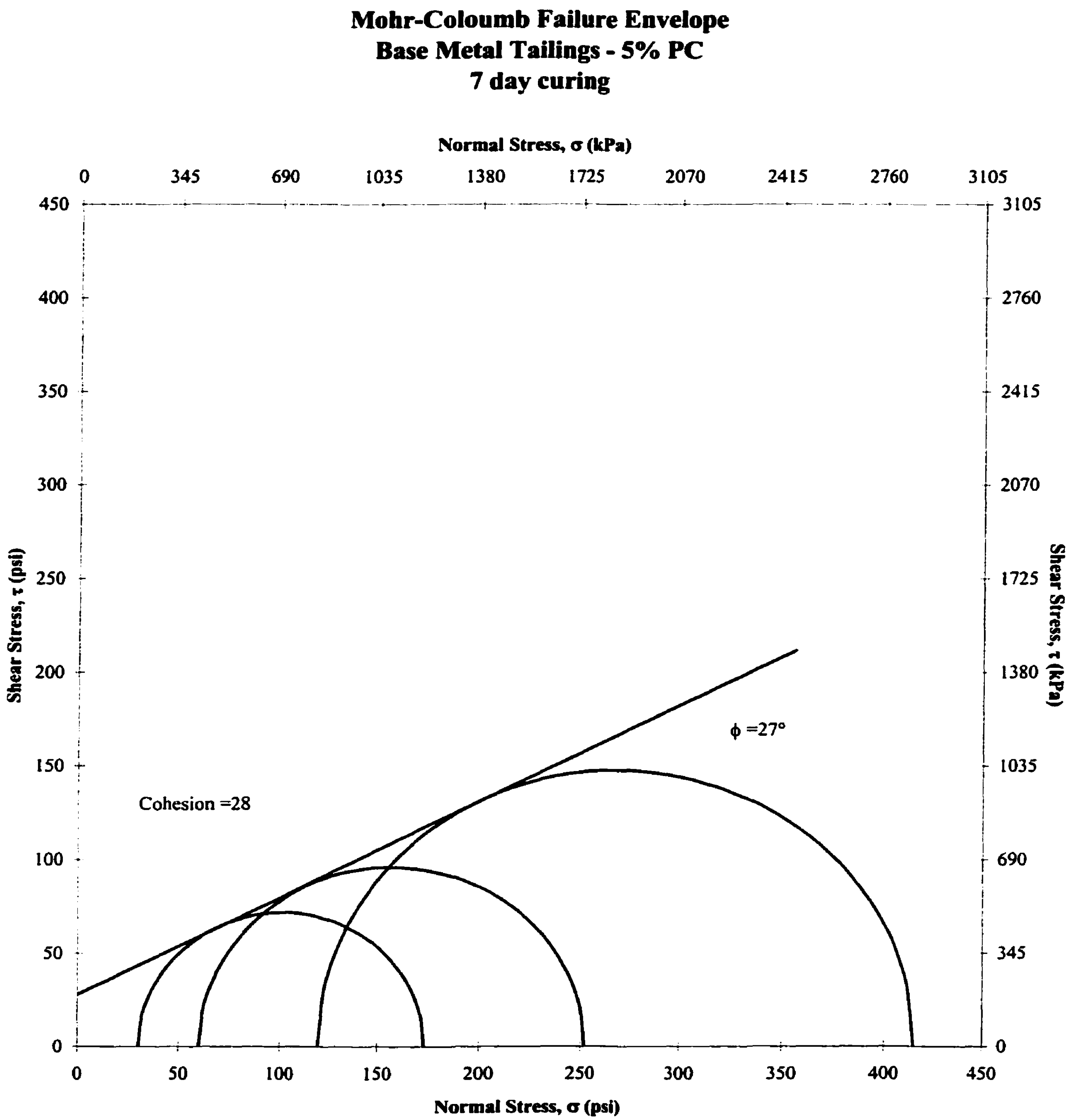
Sand ($C_u = 4.2$, % - 20μm = 26.5%)

Cement Content (%)	Curing Period (days)	Confining Pressure (σ_3) (kPa)	Deviator Stress ($\sigma_1 - \sigma_3$) (kPa)	Deformation Modulus E (MPa)	Friction Angle $\phi°$	Cohesion Range C (kPa)
5	7	207 414 828	999 1122 1393	12.0 17.0 10.0	14	337
	14	207 414 828	1073 1393 1714	25.6 18.7 35.0	20	317
	28	207 414 828	1233 1763 2244	27.0 19.1 70.0	27	299

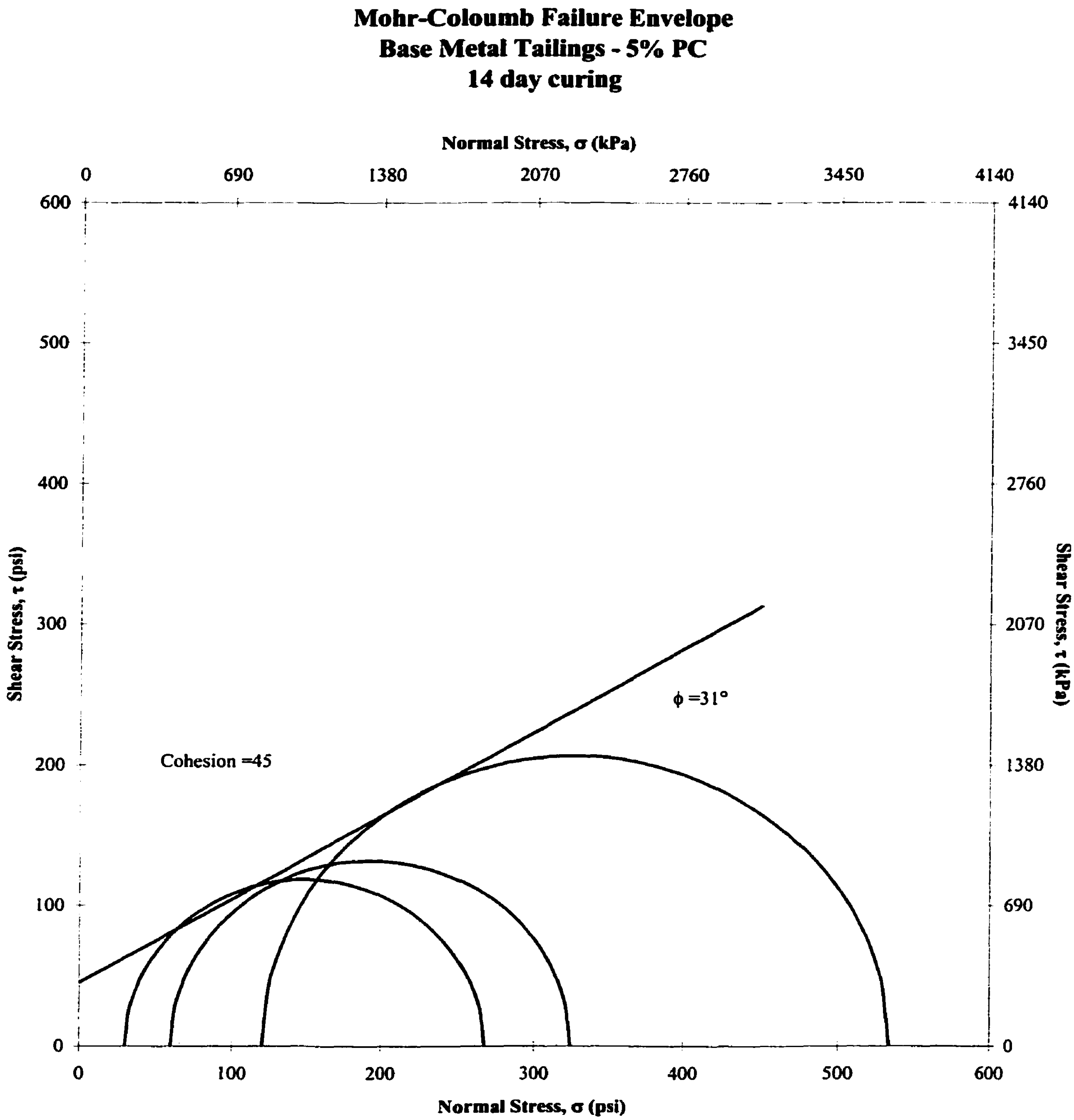
Mohr's Circle Parameters for Cemented Paste Fill Samples

Base Metal Tailings ($C_u = 5.9$, % - 20μm = 43.5%)

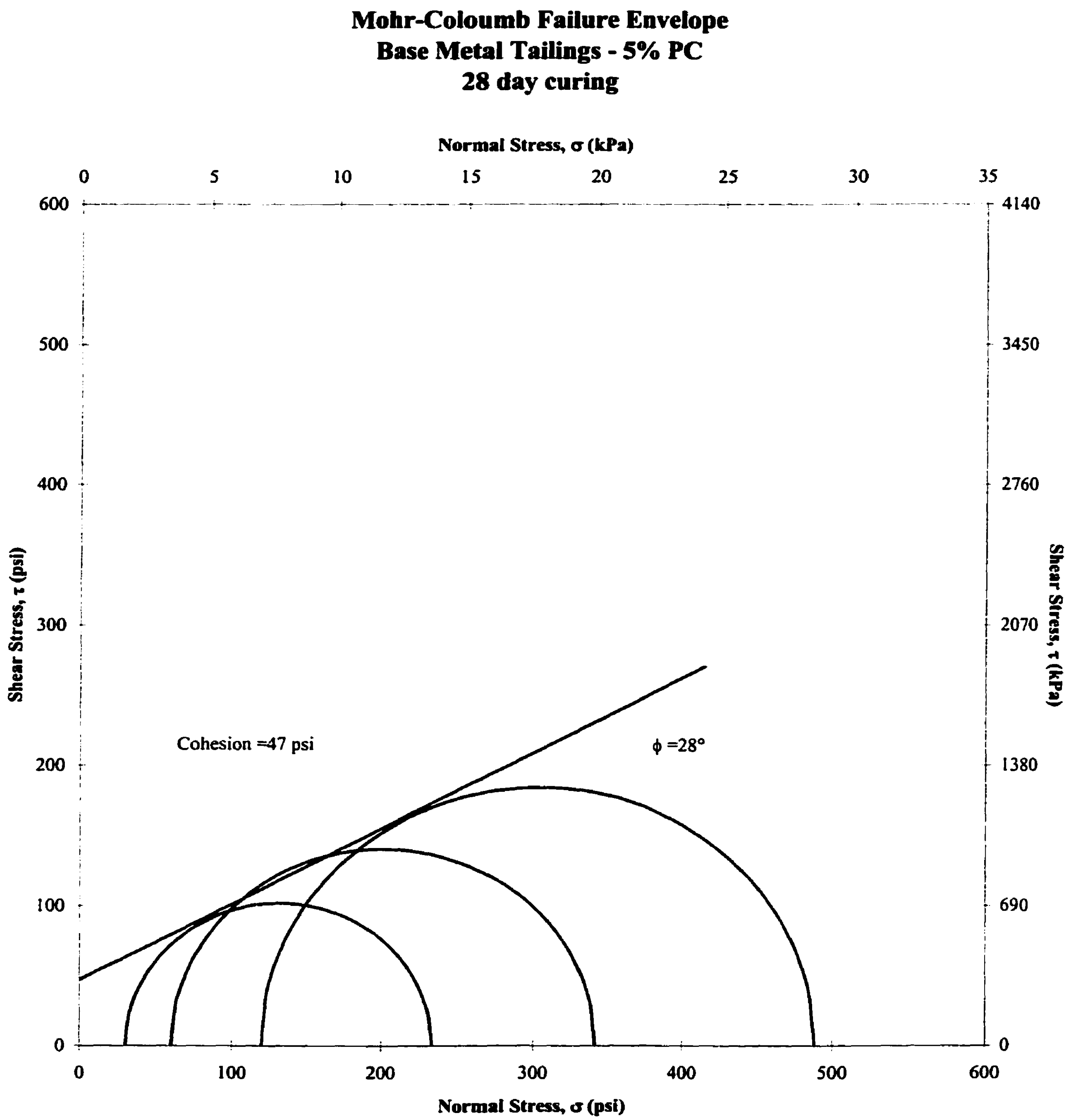
Cement Content (%)	Curing Period (days)	Confining Pressure (σ_3) (kPa)	Deviator Stress ($\sigma_1 - \sigma_3$) (kPa)	Deformation Modulus E (MPa)	Friction Angle ϕ°	Cohesion Range C (kPa)
5	7	207	991	49.2	27	193
		414	1324	41.0		
		828	2036	35.1		
		207	930	35.0		
		414	1366	42.5		
		828	2057	61.5		
	14	207	1637	117.1	31	310
		414	1820	76.5		
		828	2850	41.0		
		207	1804	170.0		
		414	2137	102.5		
		828	2740	102.5		
	28	207	1408	102.8	28	324
		414	1938	77.5		
		828	2541	41.0		
		207	1512	63.8		
		414	1938	42.5		
		828	2492	51.0		



Typical Mohr's Circle Failure Envelope for Cemented Base Metal Tailigns Paste Fill
(7 days curing)

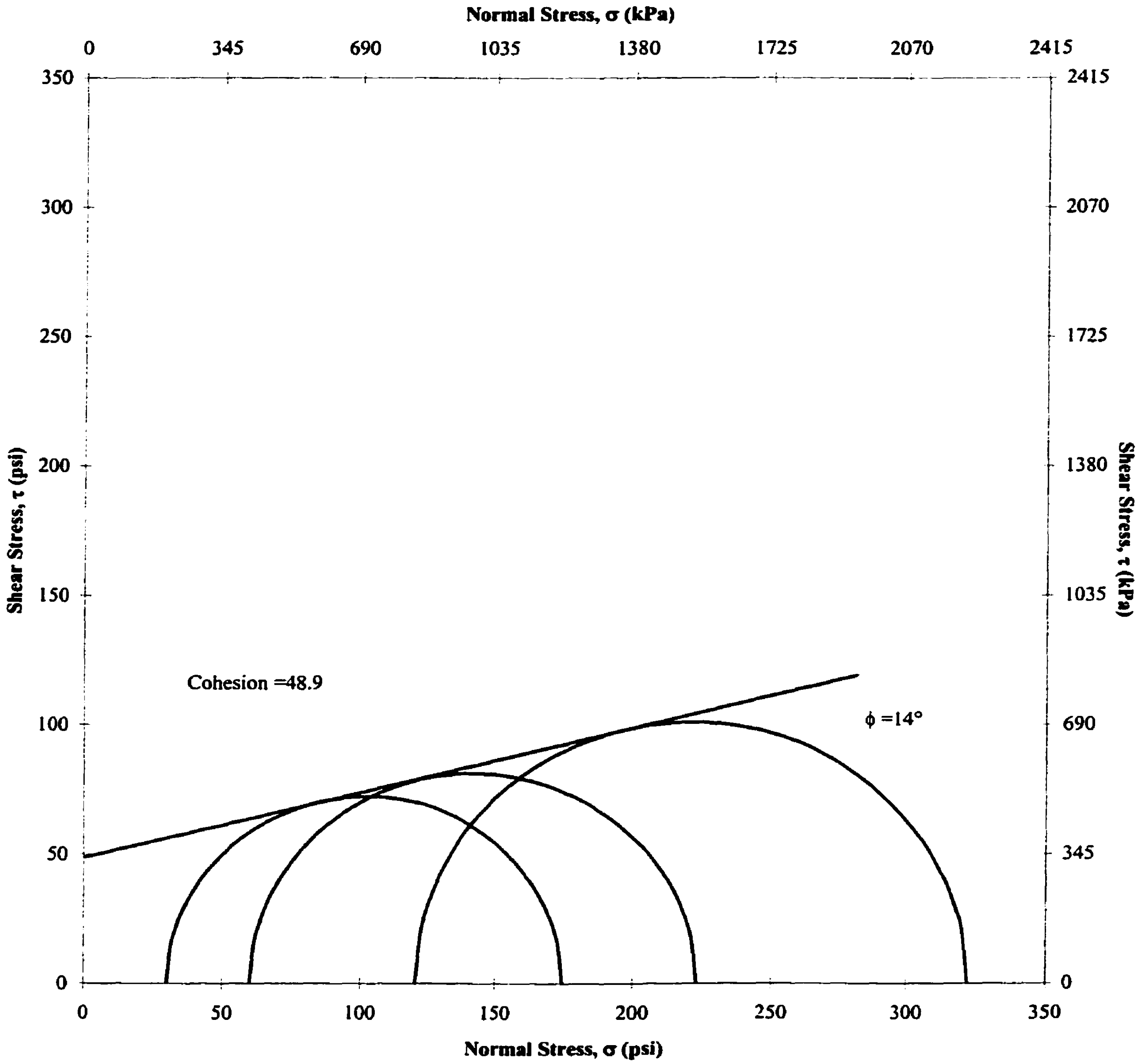


Typical Mohr's Circle Failure Envelope for Cemented Base Metal Tailings Paste Fill
(14 days curing)

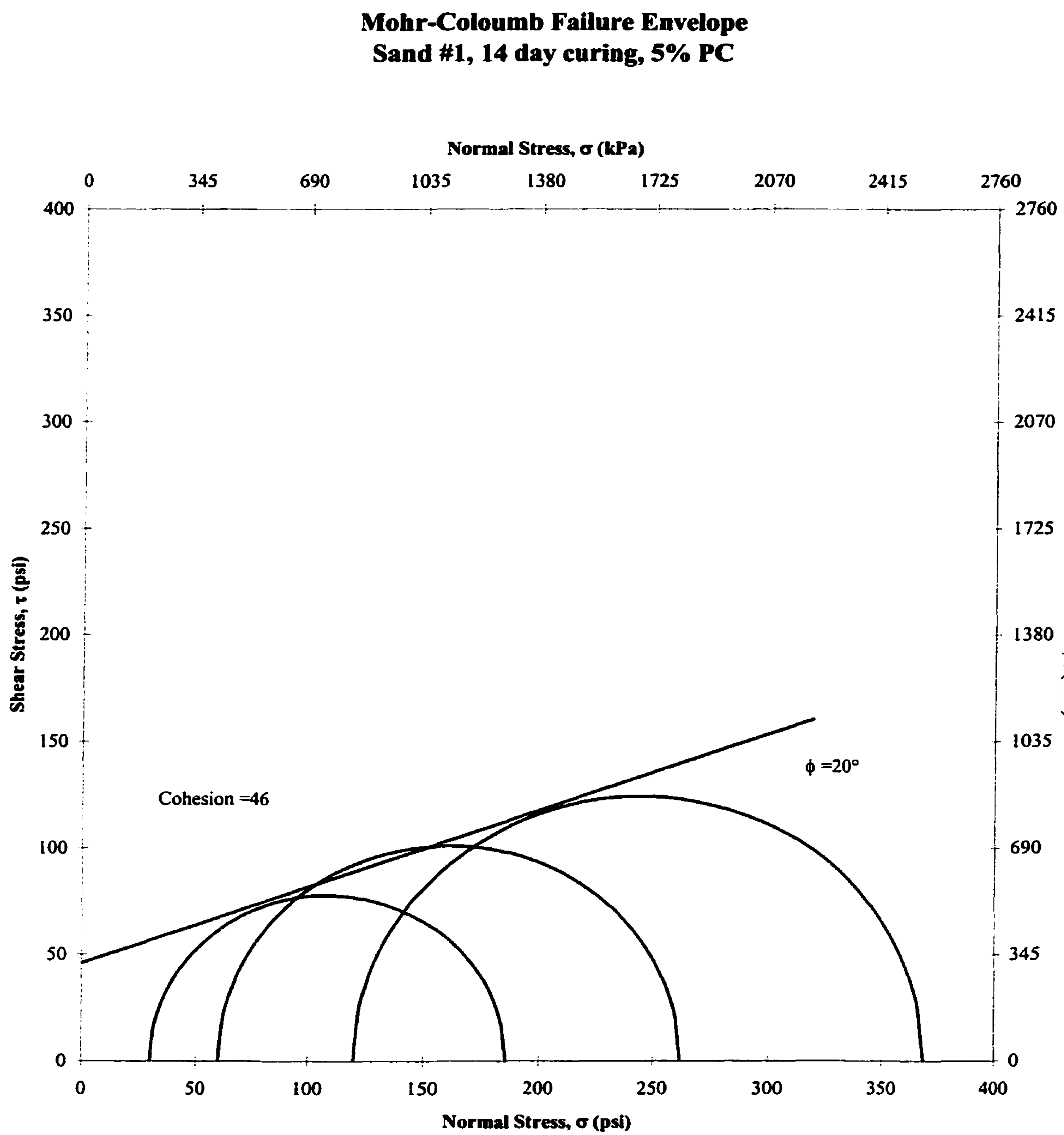


Typical Mohr's Circle Failure Envelope for Cemented Base Metal Tailings Paste Fill
(28 days curing)

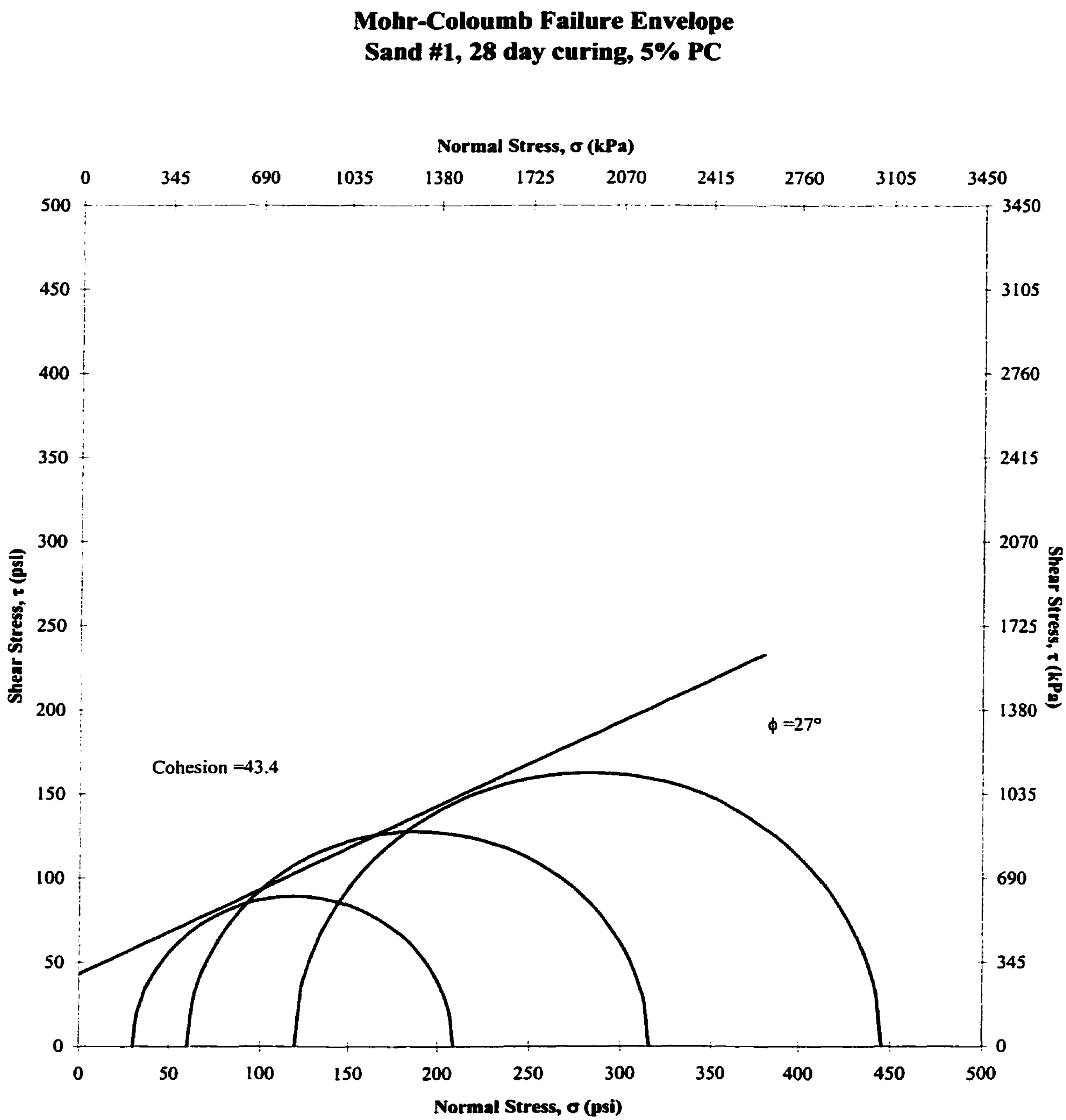
Mohr-Coloumb Failure Envelope
Sand #1, 7 day curing, 5% PC



Typical Mohr's Circle Failure Envelope for Cemented Sand Paste Fill
(7 days curing)



Typical Mohr's Circle Failure Envelope for Cemented Sand Paste Fill
(14 days curing)



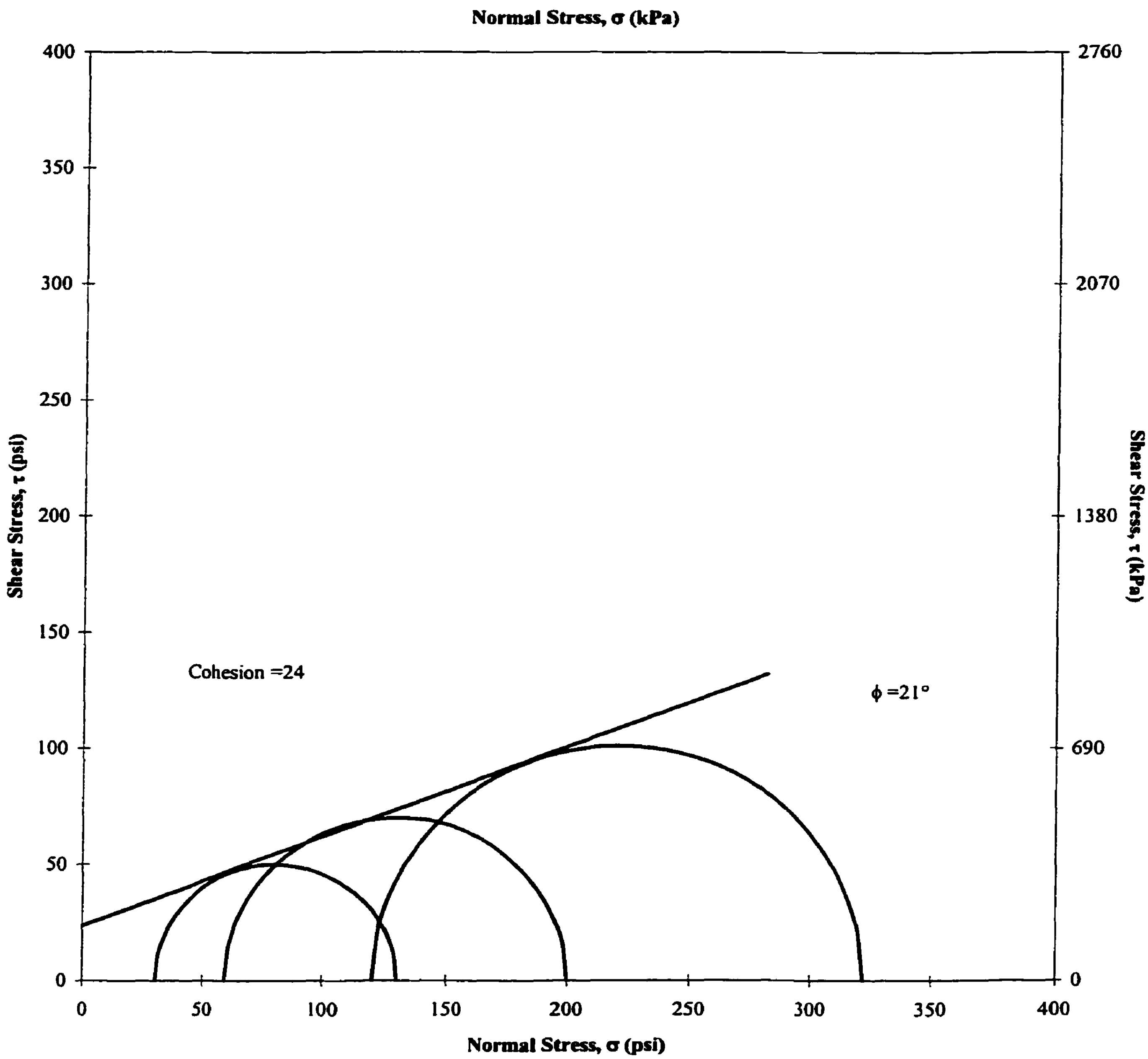
Typical Mohr's Circle Failure Envelope for Cemented Sand Paste Fill
(28 days curing)

Mohr's Circle Parameters for Cemented Paste Fill Samples

Composite Blended Precious Metal Tailings & Sand (C_u = 5.0, % - 20μm = 55%)

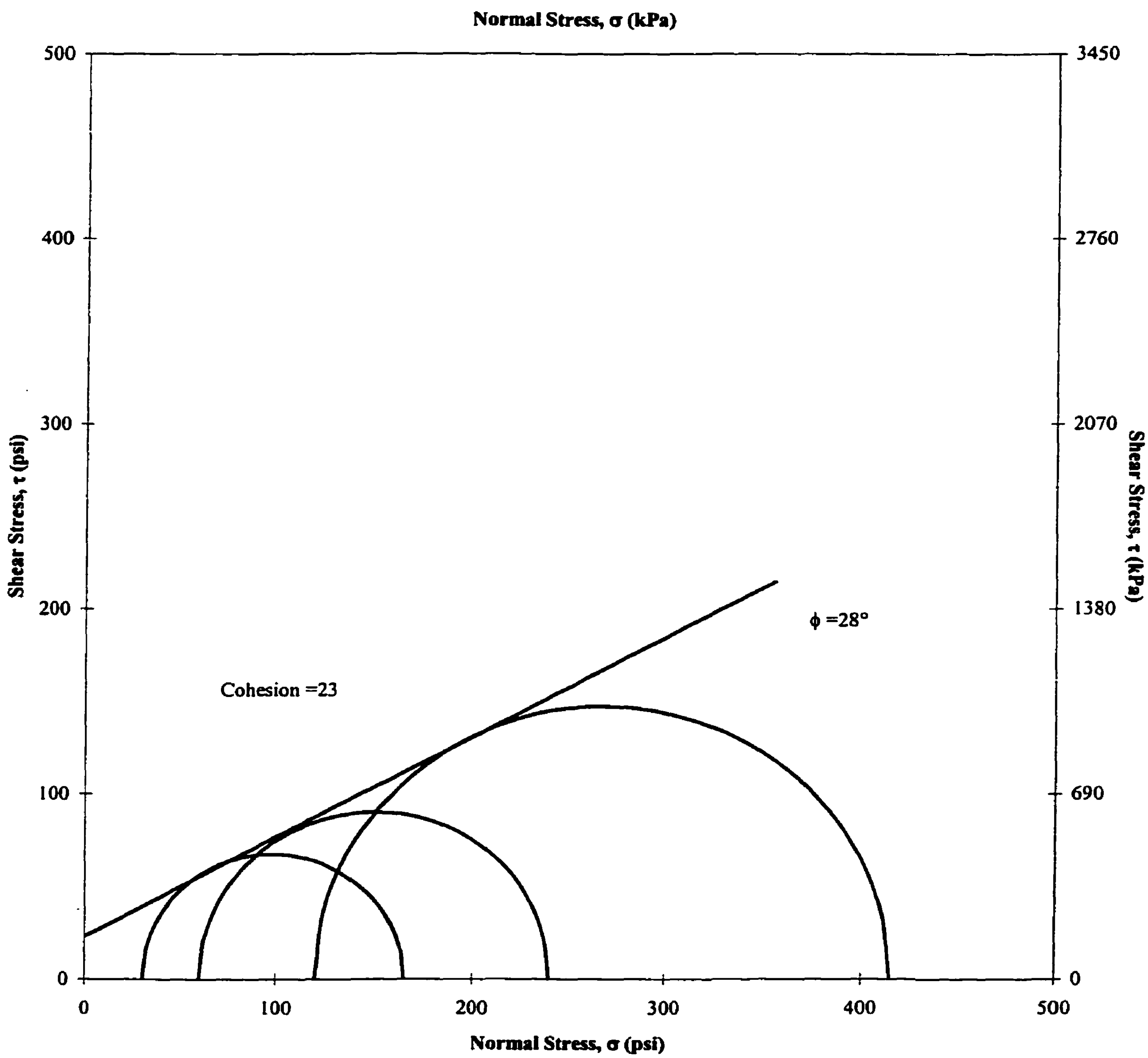
Cement Content (%)	Curing Period (days)	Confining Pressure (σ_3) (kPa)	Deviator Stress (σ_1 - σ_3) (kPa)	Deformation Modulus E (MPa)	Friction Angle ϕ°	Cohesion Range (kPa)
5	7	207 414 828	690 965 1393	7.1 8.2 6.1	21	166
	14	207 414 828	931 1241 2034	14.3 4.3 14.1	28	159
	28	207 414 828	1023 1358 2367	24.7 21.1 11.0	32	150

Mohr-Coloumb Failure Envelope
Composte Blended Tailings/Sand



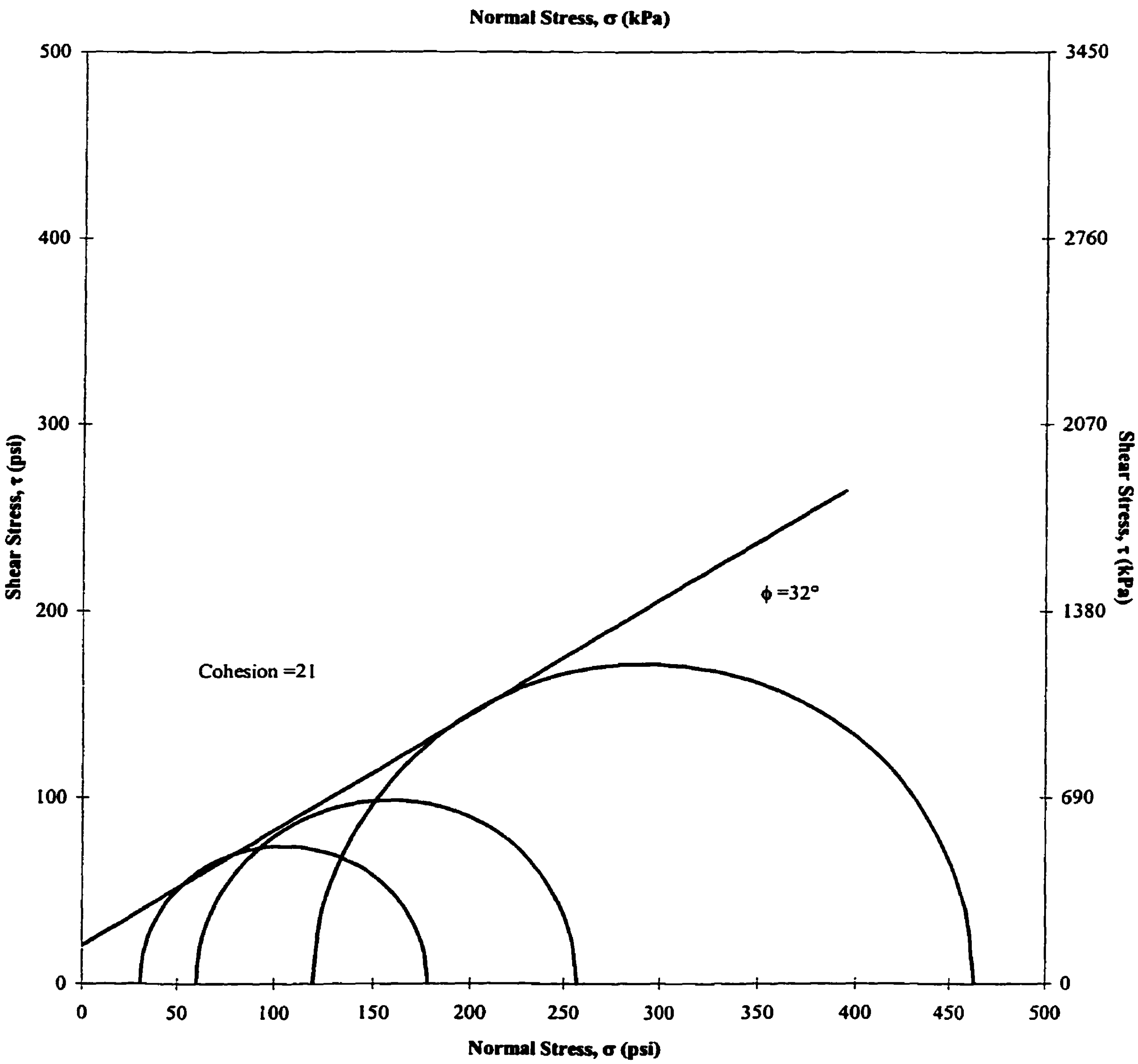
Typical Mhor's Circle Failure Envelope for Composite Blended Tailings/Sand
(7 days curing)

Mohr-Coloumb Failure Envelope
Composte Blended Tailings/Sand



Typical Mohr's Circle Failure Envelope for Composite Blended Tailings/Sand
(14 days curing)

Mohr-Coloumb Failure Envelope
Composte Blended Tailings/Sand



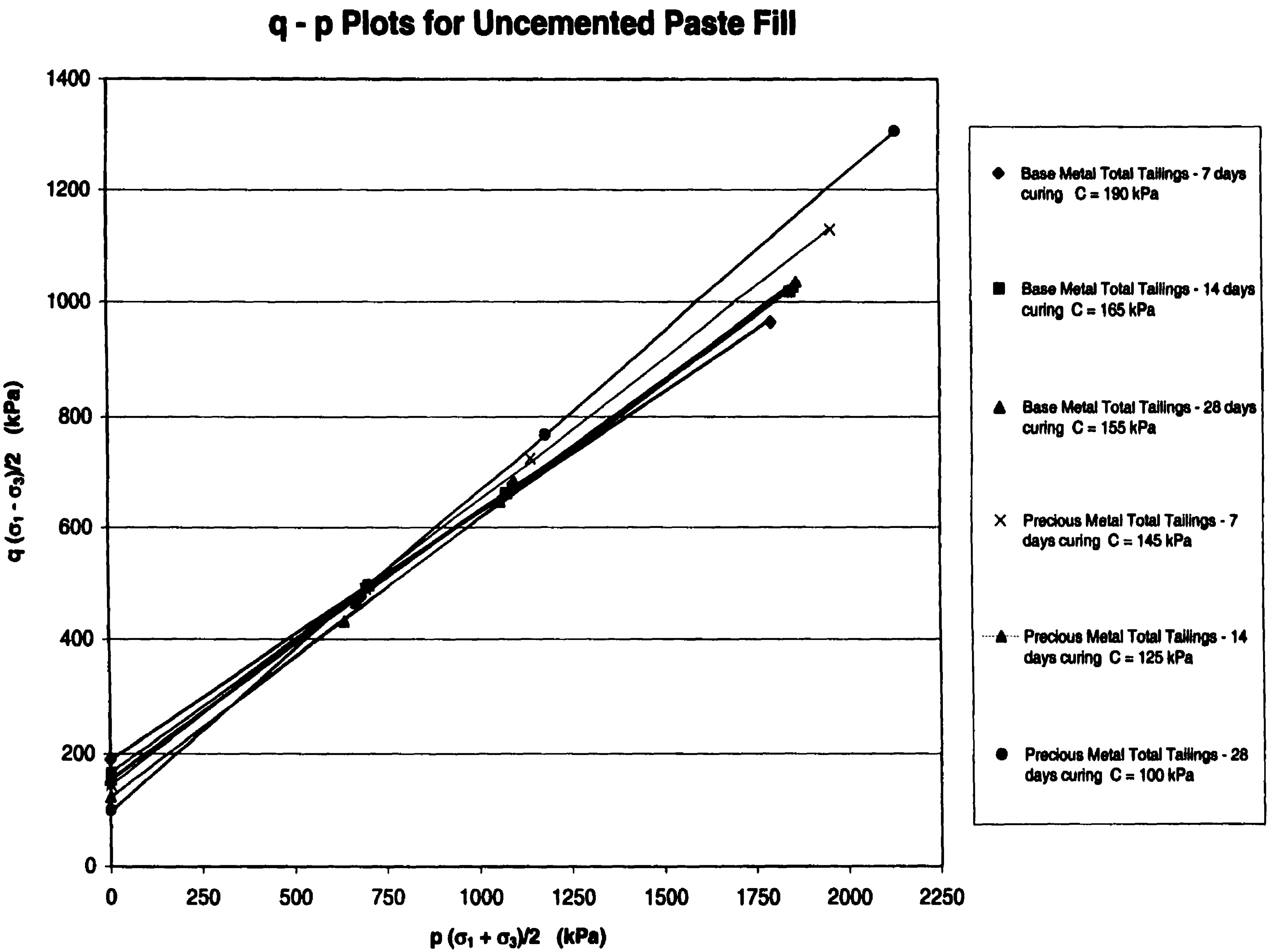
Typical Mohr's Circle Failure Envelope for Composite Blended Tailings/Sand
(28 days curing)

APPENDIX C-2

APPENDIX C-2

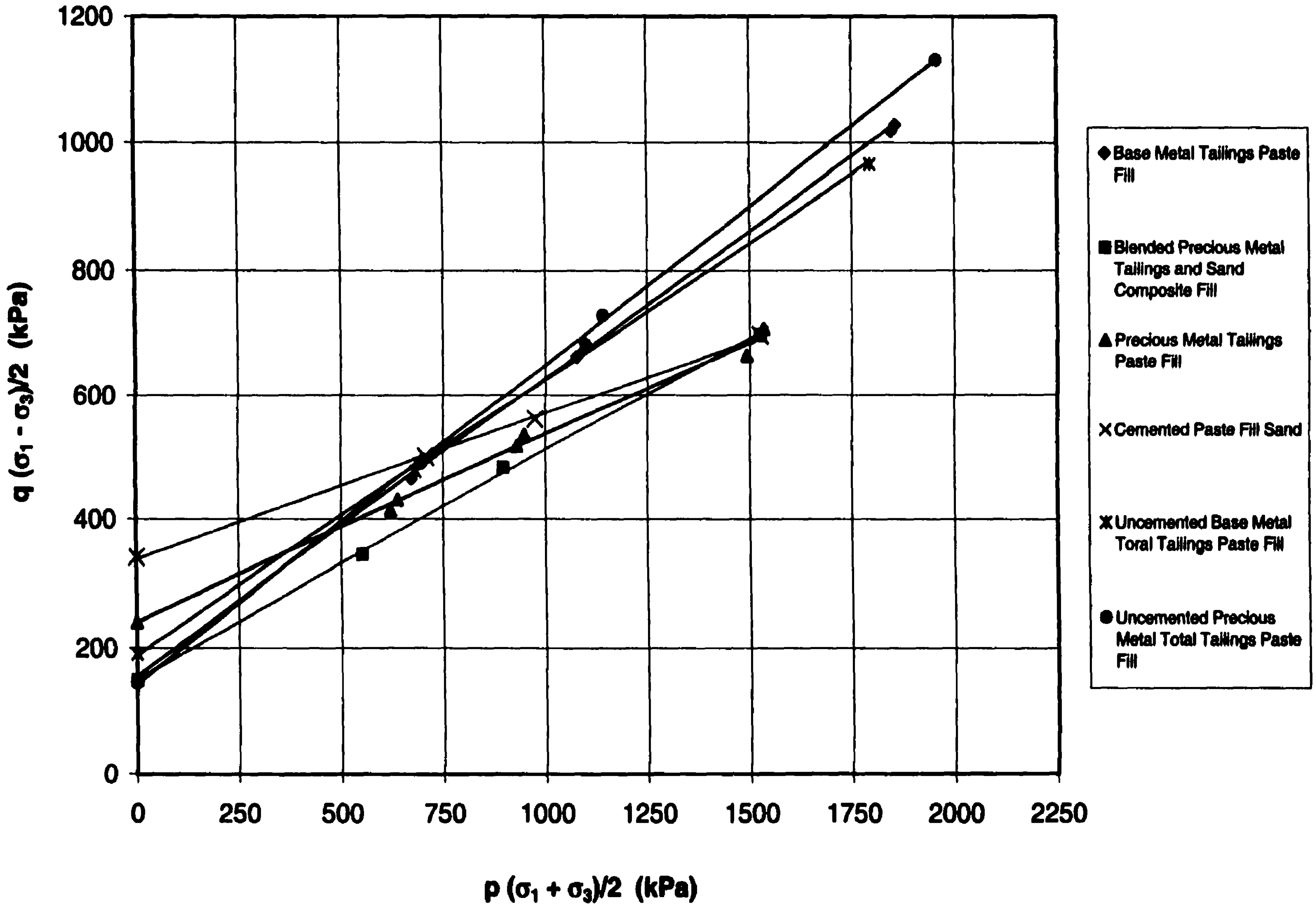
Triaxial Compressive Strength Test Results

Failure envelopes based on (q-p) stress path plots for the pastefill samples are presented in this section.

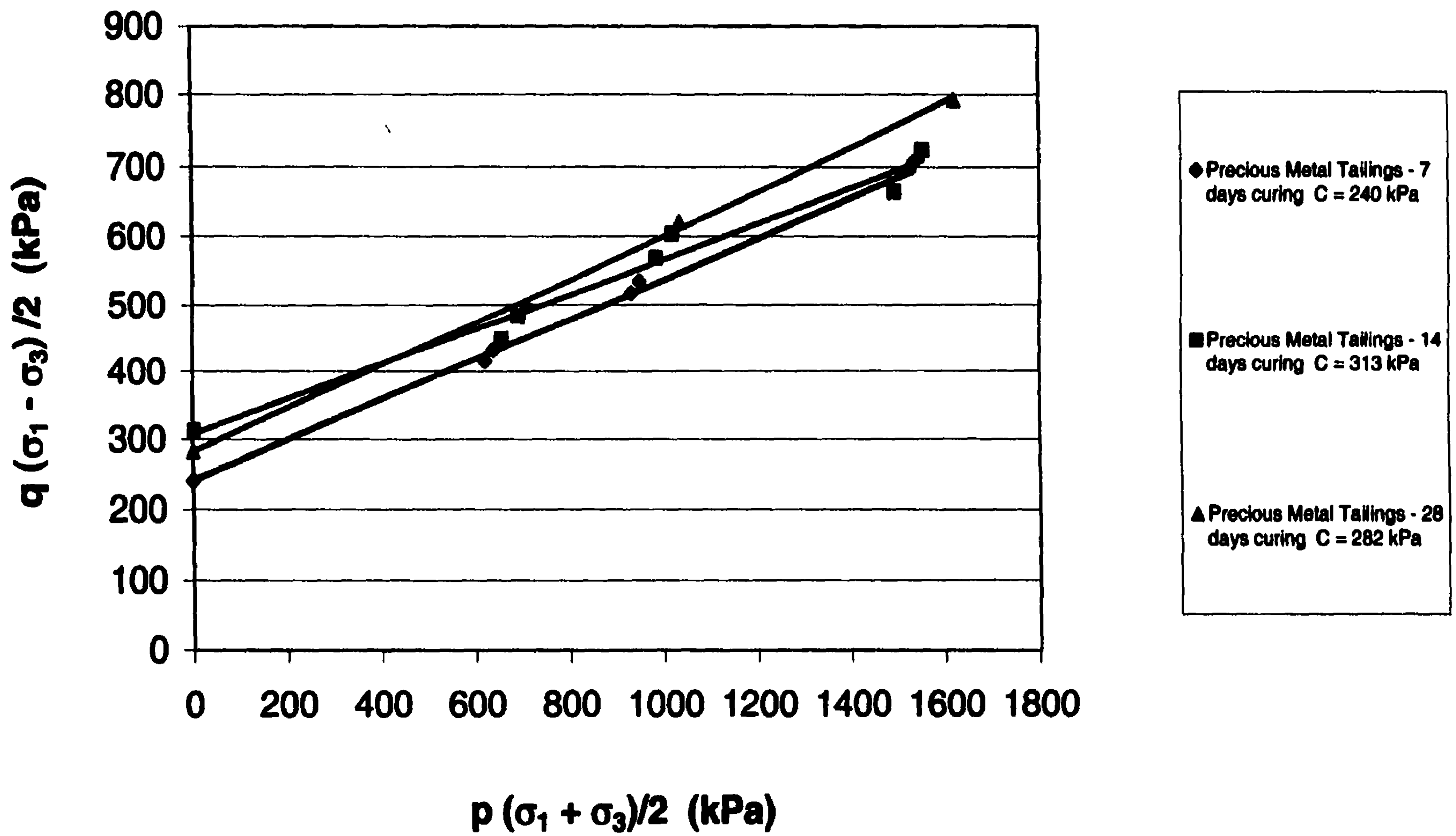


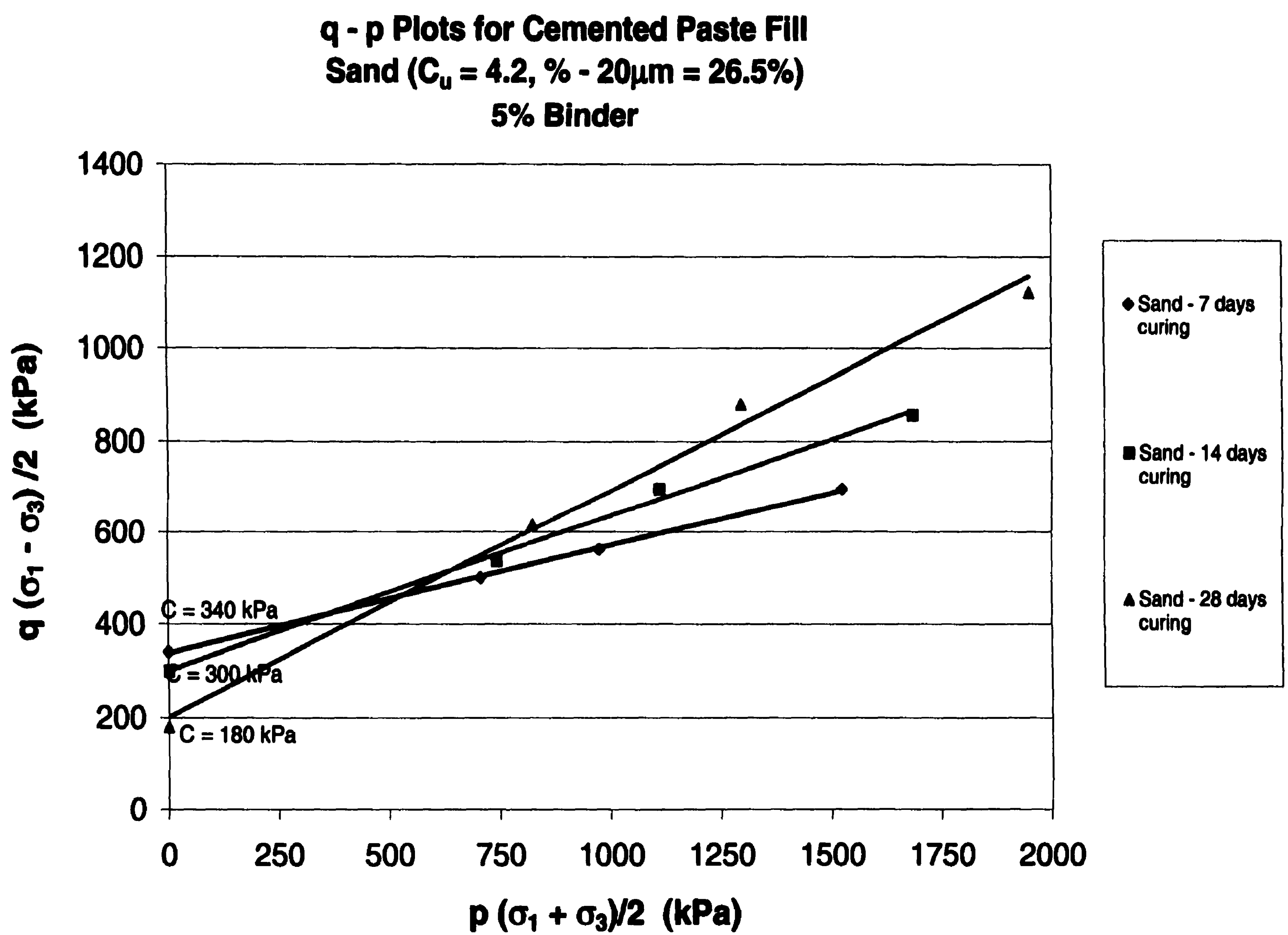
q - p Plots for Uncemented Paste Fill
1400
1200
1000
800
600
400
200
0
q (σ1 - σ3)/2 (kPa)
0
250
500
750
1000
1250
1500
1750
2000
2250
p (σ1 + σ3)/2 (kPa)
Base Metal Total Tailings - 7 days curing C = 190 kPa
Base Metal Total Tailings - 14 days curing C = 165 kPa
Base Metal Total Tailings - 28 days curing C = 155 kPa
Precious Metal Total Tailings - 7 days curing C = 145 kPa
Precious Metal Total Tailings - 14 days curing C = 125 kPa
Precious Metal Total Tailings - 28 days curing C = 100 kPa

q - p Plots for Paste Fills after 7 Days Curing

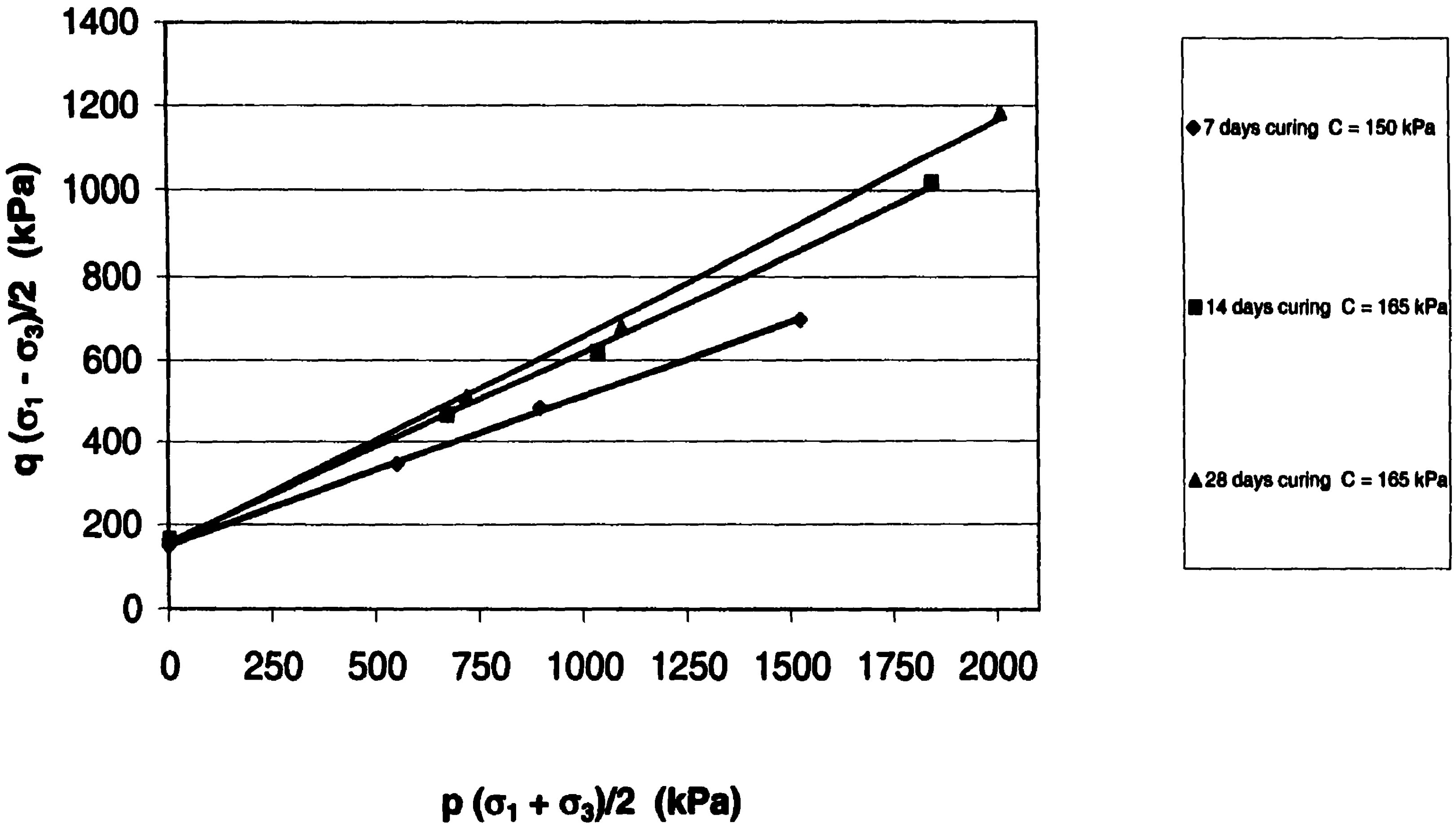


q - p Plots for Precious Metal Tailings
(C_u = 5.3, % - 20μm = 42.3%)
5 % Binder




q - p Plots for Cemented Paste Fill
Sand (C_u = 4.2, % - 20μm = 26.5%)
5% Binder
q (σ_1 - σ_3) /2 (kPa)
p (σ_1 + σ_3)/2 (kPa)
1400
1200
1000
800
600
400
200
0
0
250
500
750
1000
1250
1500
1750
2000
C = 340 kPa
C = 300 kPa
C = 180 kPa
Sand - 7 days curing
Sand - 14 days curing
Sand - 28 days curing

q - p Plots for Blended Precious Metal Tailings and Sand Composite Fill
(C_u = 5.5, % - 20μm = 53%)
5% Binder



APPENDIX C-3

APPENDIX C-3

Dry Density vs. Moisture Relationships (Compaction Tests)

The compaction tests were conducted to evaluate the variation in density and optimum moisture for the composite (blended tailing and sand) materials in this study. The tests were performed in accordance with ASTM (D-698) specifications

The tests results are presented in this section.

Moisture-Density Relationship Testing

Description of Soil: 100% Base Metal Tailings
Date of Test: June 30, 97
Performed By: Stefan Furey/Soutsay Boualavong

Weight of Mold: 4.293 kg
Volume of Mold: 944 cm^3
Specific Gravity: 2.96

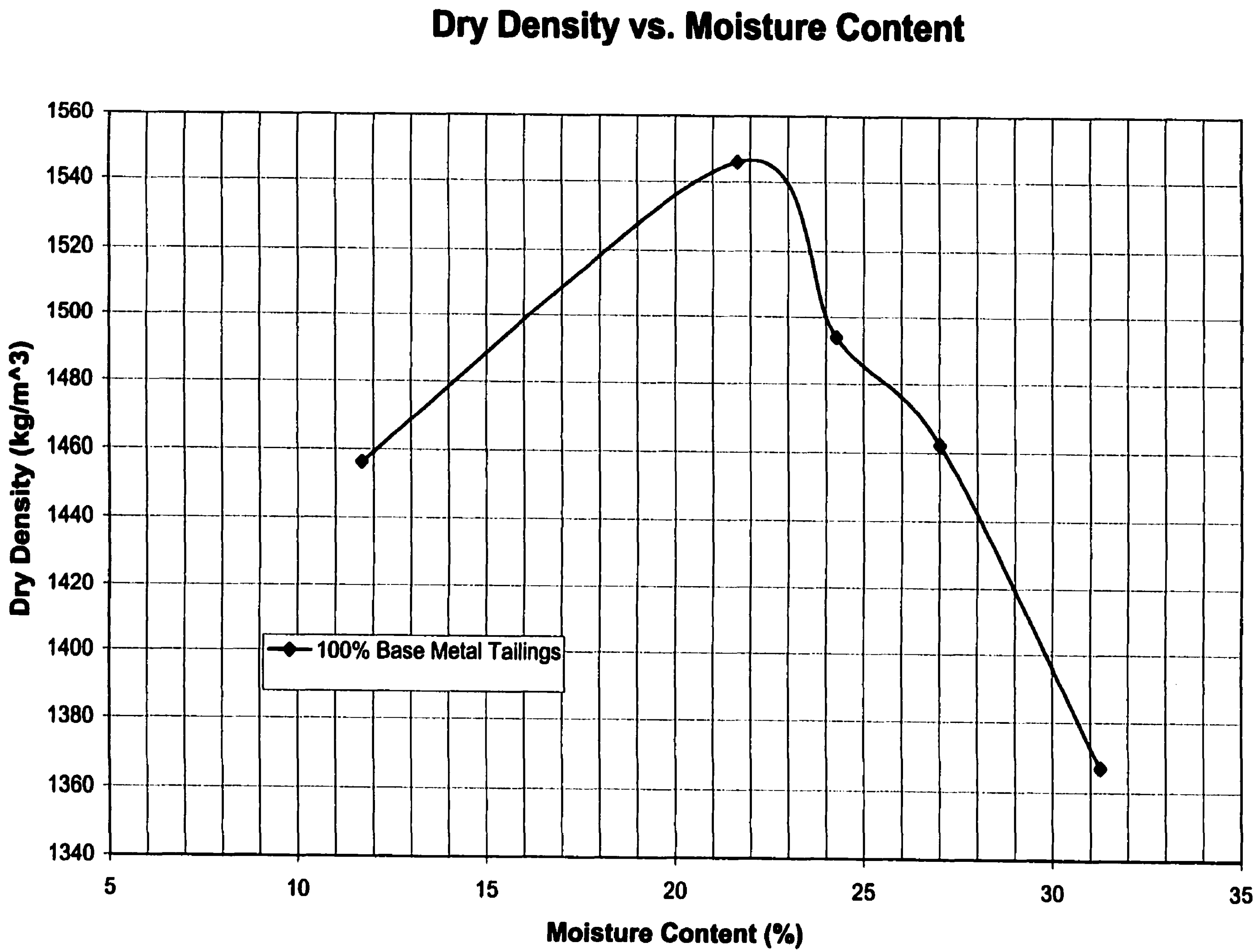
Dry Unit Weight Determination

Trial No.	1	2	3	4	5
Wt. Mold + Compacted Soil (kg)	5.831	6.072	6.044	6.041	5.985
Wt. Mold (kg)	4.293	4.293	4.293	4.293	4.293
Wt. Compacted Soil (kg)	1.538	1.779	1.751	1.748	1.692
Wet Unit Weight (kg/m^3)	1629.2	1884.5	1854.9	1851.7	1792.4
Dry Unit Weight (kg/m^3)	1456.3	1545.9	1493.9	1462.2	1366.9
Void Ratio	1.028	0.911	0.977	1.020	1.161
Porosity (%)	50.7	47.7	49.4	50.5	53.7

Moisture Content Determination

Trial No.		1	2	3	4	5
Wt. Tare (g)	1	21.36	21.96	21.84	22.24	22.08
	2	22.46	21.94	21.09	21.92	21.07
Wt. Wet Soil + Tare (g)	1	50.75	53.91	59.95	87.78	81.45
	2	57.61	55.32	61.69	58.04	69.83
Wt. Dry Soil + Tare (g)	1	47.26	46.91	50.74	70.32	62.97
	2	53.56	48.17	51.79	48.15	54.52
Moisture Content (%)	1	11.87	21.91	24.17	26.64	31.13
	2	11.52	21.42	24.38	27.38	31.40
Avg. Moisture Content		11.70	21.66	24.28	27.01	31.26

Optimum Moisture Content: 22.0 %
Maximum Dry Density: 1547 kg/m^3


Dry Density vs. Moisture Content
Dry Density (kg/m^3)
1560
1540
1520
1500
1480
1460
1440
1420
1400
1380
1360
1340
5
10
15
20
25
30
35
Moisture Content (%)
100% Base Metal Tailings

Moisture-Density Relationship Testing

Description of Soil: 80% Base Metal Tailings/20% Sand
Date of Test: July 02, 97
Performed By: Stefan Furey/Soutsay Boualavong

Weight of Mold: 4.292 kg
Volume of Mold: 944 cm^3
Specific Gravity: 2.83

Dry Unit Weight Determination

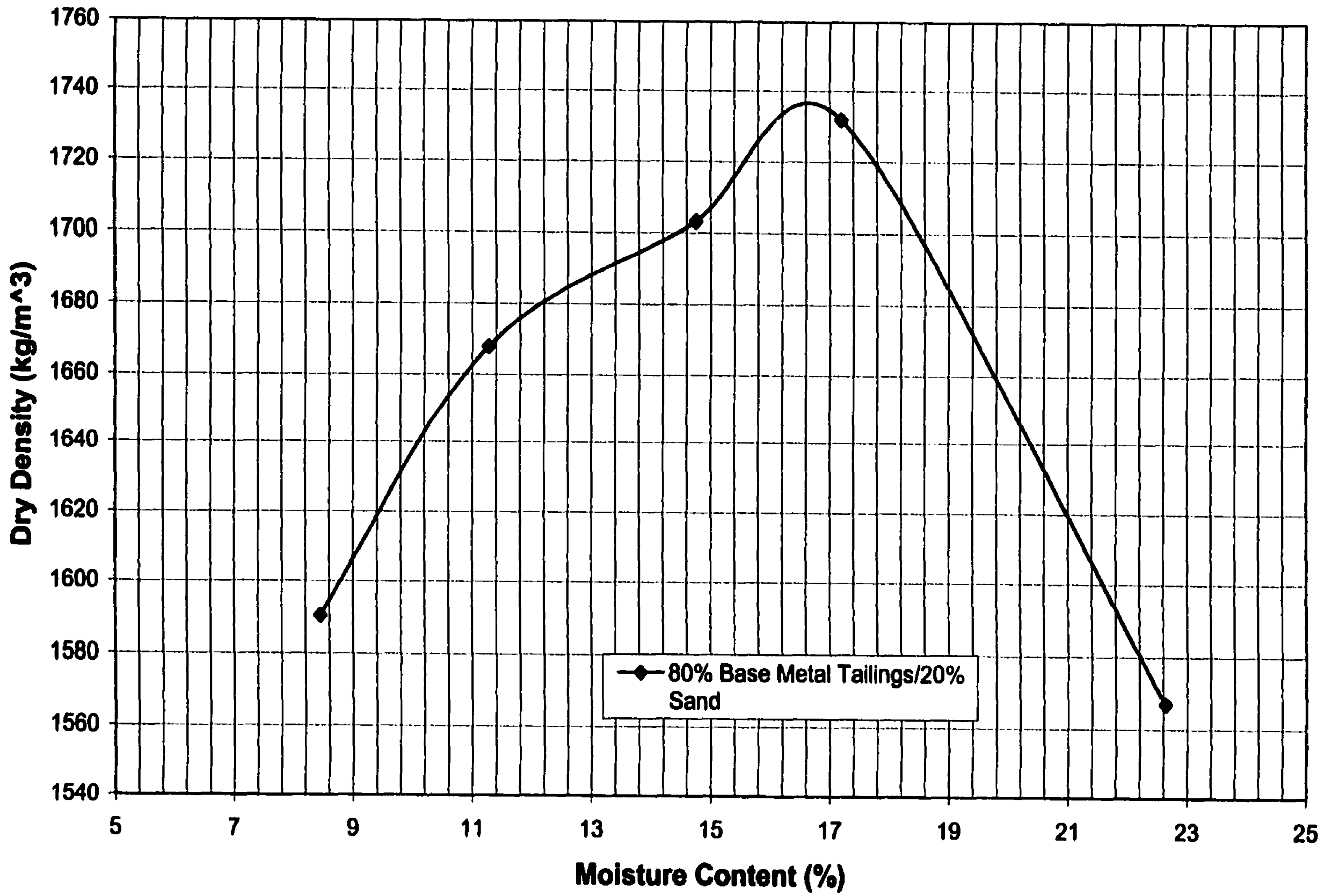
Trial No.	1	2	3	4	5
Wt. Mold + Compacted Soil (kg)	5.920	6.044	6.137	6.208	6.106
Wt. Mold (kg)	4.292	4.292	4.292	4.292	4.292
Wt. Compacted Soil (kg)	1.628	1.752	1.845	1.916	1.814
Wet Unit Weight (kg/m^3)	1724.6	1855.9	1954.4	2029.7	1921.6
Dry Unit Weight (kg/m^3)	1590.2	1667.9	1703.1	1731.9	1567.0
Void Ratio	0.776	0.693	0.658	0.631	0.802
Porosity (%)	43.7	40.9	39.7	38.7	44.5

Moisture Content Determination

Trial No.		1	2	3	4	5
Wt. Tare (g)	1	22.18	22.35	21.86	21.42	22.54
	2	21.27	21.91	21.08	22.01	21.96
Wt. Wet Soil + Tare (g)	1	47.60	49.05	49.99	57.06	75.48
	2	33.25	57.86	41.57	75.86	76.65
Wt. Dry Soil + Tare (g)	1	45.34	45.80	45.75	50.80	63.48
	2	32.29	54.13	38.61	66.80	64.29
Moisture Content (%)	1	8.89	12.17	15.07	17.56	22.67
	2	8.01	10.38	14.45	16.82	22.60
Avg. Moisture Content		8.45	11.27	14.76	17.19	22.63

Optimum Moisture Content: 16.7 %
Maximum Dry Density: 1737 kg/m3

Dry Density vs. Moisture Content



Moisture-Density Relationship Testing

Description of Soil: 60% Base Metal Tailings/40% Sand
Date of Test: July 02, 97
Performed By: Stefan Furey/Soutsay Boualavong

Weight of Mold: 4.291 kg
Volume of Mold: 944 cm^3
Specific Gravity: 2.74

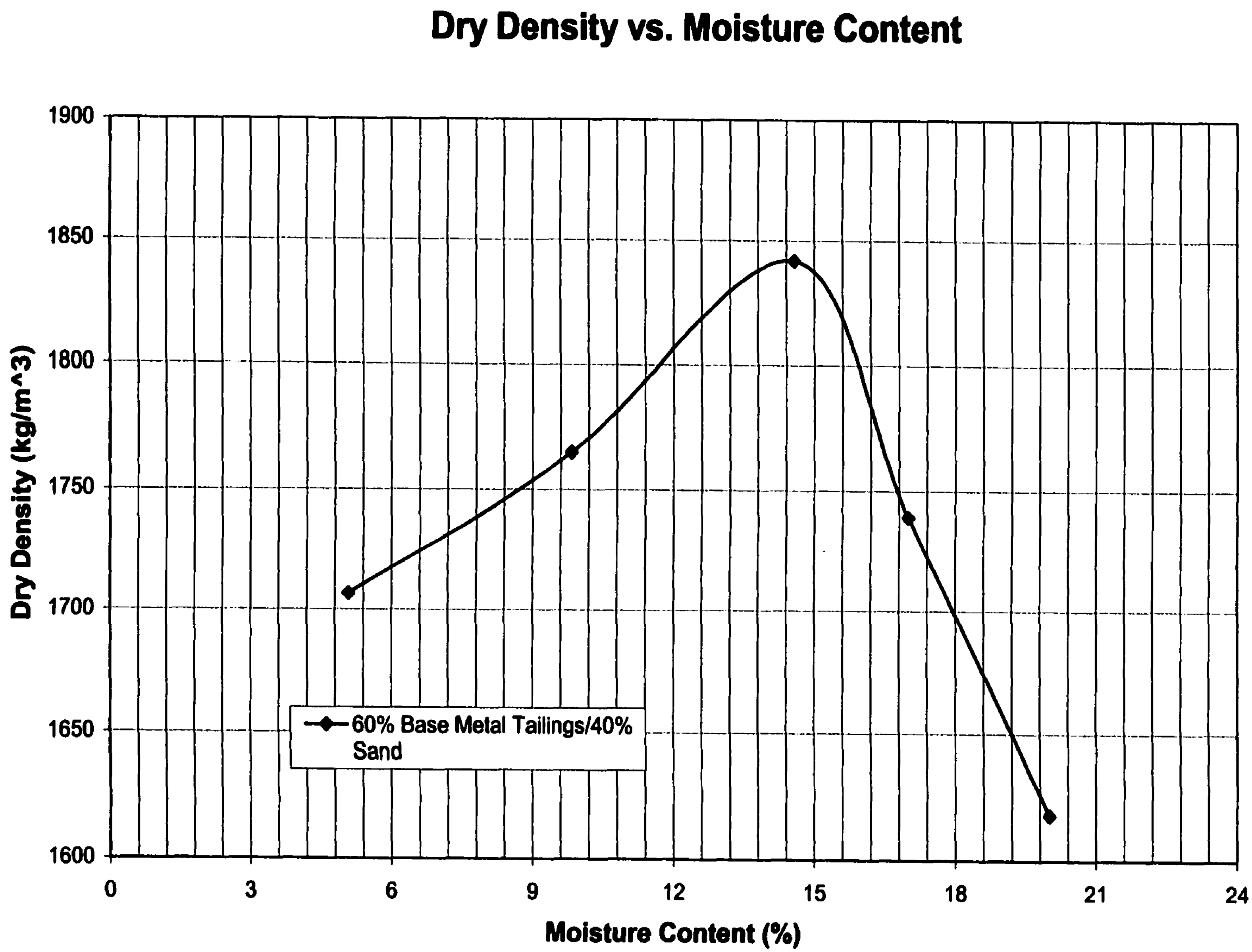
Dry Unit Weight Determination

Trial No.	1	2	3	4	5
Wt. Mold + Compacted Soil (kg)	5.984	6.121	6.283	6.212	6.124
Wt. Mold (kg)	4.291	4.291	4.291	4.291	4.291
Wt. Compacted Soil (kg)	1.693	1.83	1.992	1.921	1.833
Wet Unit Weight (kg/m^3)	1793.4	1938.6	2110.2	2035.0	1941.7
Dry Unit Weight (kg/m^3)	1706.6	1764.9	1841.7	1739.3	1618.0
Void Ratio	0.602	0.549	0.485	0.572	0.690
Porosity (%)	37.6	35.5	32.6	36.4	40.8

Moisture Content Determination

Trial No.		1	2	3	4	5
Wt. Tare (g)	1	19.93	20.25	23.52	22.34	22.57
	2	20.88	22.85	22.76	21.14	21.56
Wt. Wet Soil + Tare (g)	1	61.70	58.78	77.53	77.85	76.65
	2	60.48	58.29	81.44	71.60	85.67
Wt. Dry Soil + Tare (g)	1	59.78	55.07	69.72	68.33	65.91
	2	58.27	54.73	72.82	63.10	72.75
Moisture Content (%)	1	4.60	9.63	14.46	17.15	19.86
	2	5.58	10.05	14.69	16.85	20.15
Avg. Moisture Content		5.09	9.84	14.58	17.00	20.01

Optimum Moisture Content: 14.5 %
Maximum Dry Density: 1843 kg/m^3


Dry Density vs. Moisture Content
Dry Density (kg/m^3)
1900
1850
1800
1750
1700
1650
1600
0
3
6
9
12
15
18
21
24
Moisture Content (%)
60% Base Metal Tailings/40% Sand

Moisture-Density Relationship Testing

Description of Soil: 40% Base Metal Tailings/60% Sand
Date of Test: July 07, 97
Performed By: Stefan Furey

Weight of Mold: 4.293 kg
Volume of Mold: 944 cm^3
Specific Gravity: 2.79

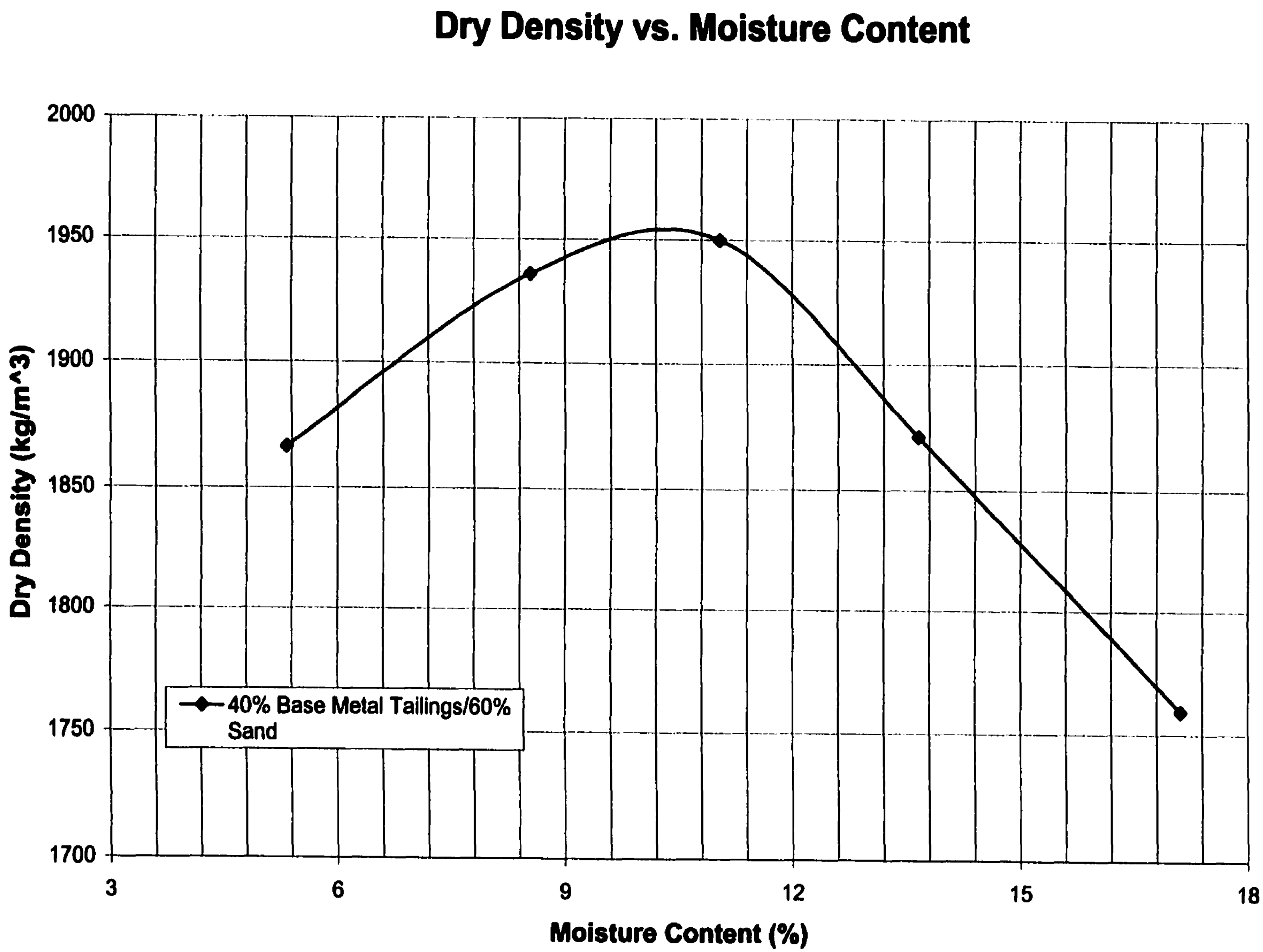
Dry Unit Weight Determination

Trial No.	1	2	3	4	5
Wt. Mold + Compacted Soil (kg)	6.149	6.277	6.337	6.301	6.238
Wt. Mold (kg)	4.293	4.293	4.293	4.293	4.293
Wt. Compacted Soil (kg)	1.856	1.984	2.044	2.008	1.945
Wet Unit Weight (kg/m^3)	1966.1	2101.7	2165.3	2127.1	2060.4
Dry Unit Weight (kg/m^3)	1866.6	1936.1	1949.9	1871.6	1759.3
Void Ratio	0.492	0.438	0.428	0.488	0.583
Porosity	33.0	30.5	30.0	32.8	36.8

Moisture Content Determination

Trial No.		1	2	3	4	5
Wt. Tare (g)	1	22.12	20.24	21.85	19.94	22.33
	2	21.26	22.33	21.40	21.00	21.87
Wt. Wet Soil + Tare (g)	1	60.53	53.62	69.56	77.63	83.88
	2	56.73	55.84	58.81	73.29	84.90
Wt. Dry Soil + Tare (g)	1	58.47	50.78	64.34	69.71	73.30
	2	54.85	52.96	54.64	66.19	74.16
Moisture Content (%)	1	5.36	8.51	10.94	13.73	17.19
	2	5.30	8.59	11.15	13.58	17.04
Avg. Moisture Content		5.33	8.55	11.04	13.65	17.11

Optimum Moisture Content: 10.3 %
Maximum Dry Density: 1955 kg/m^3


Dry Density vs. Moisture Content
2000
1950
1900
1850
1800
1750
1700
Dry Density (kg/m^3)
3
6
9
12
15
18
Moisture Content (%)
40% Base Metal Tailings/60% Sand

Moisture-Density Relationship Testing

Description of Soil: 50% Base Metal Tailings/50% Sand
Date of Test: June 30, 97
Performed By: Stefan Furey/Soutsay Boualavong

Weight of Mold: 4.293 kg
Volume of Mold: 944 cm^3
Specific Gravity: 2.8

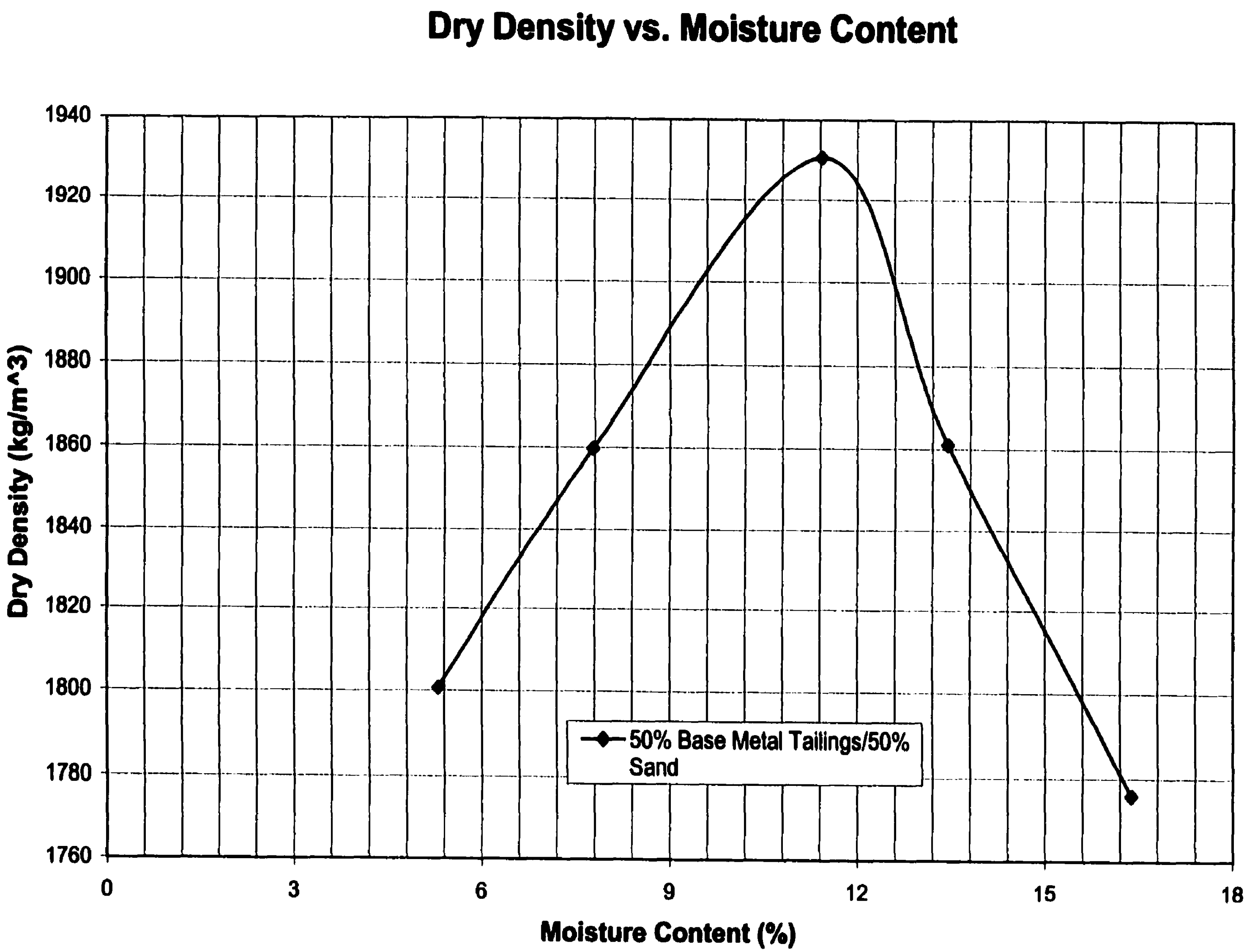
Dry Unit Weight Determination

Trial No.	1	2	3	4	5
Wt. Mold + Compacted Soil (kg)	6.083	6.185	6.324	6.286	6.244
Wt. Mold (kg)	4.293	4.293	4.293	4.293	4.293
Wt. Compacted Soil (kg)	1.79	1.892	2.031	1.993	1.951
Wet Unit Weight (kg/m^3)	1896.2	2004.2	2151.5	2111.2	2066.7
Dry Unit Weight (kg/m^3)	1800.7	1859.7	1930.8	1861.1	1775.8
Void Ratio	0.552	0.503	0.447	0.501	0.574
Porosity	35.6	33.4	30.9	33.4	36.5

Moisture Content Determination

Trial No.		1	2	3	4	5
Wt. Tare (g)	1	22.75	23.04	22.56	21.62	21.57
	2	22.00	22.91	21.99	20.78	22.76
Wt. Wet Soil + Tare (g)	1	55.85	51.13	65.01	60.32	55.75
	2	54.69	49.61	61.56	61.37	62.29
Wt. Dry Soil + Tare (g)	1	54.14	48.92	60.24	55.18	50.23
	2	52.91	47.56	56.96	55.85	55.72
Moisture Content (%)	1	5.17	7.87	11.24	13.28	16.15
	2	5.45	7.68	11.62	13.60	16.62
Avg. Moisture Content		5.31	7.77	11.43	13.44	16.39

Optimum Moisture Content: 11.4 %
Maximum Dry Density: 1931 kg/m3


Dry Density vs. Moisture Content
Dry Density (kg/m^3)
1940
1920
1900
1880
1860
1840
1820
1800
1780
1760
0
3
6
9
12
15
18
Moisture Content (%)
50% Base Metal Tailings/50% Sand

Moisture-Density Relationship Testing

Description of Soil: 25% Base Metal Tailings/75% Sand
Date of Test: June 30, 97
Performed By: Stefan Furey/Soutsay Boualavong

Weight of Mold: 4.293 kg
Volume of Mold: 944 cm^3
Specific Gravity: 2.75

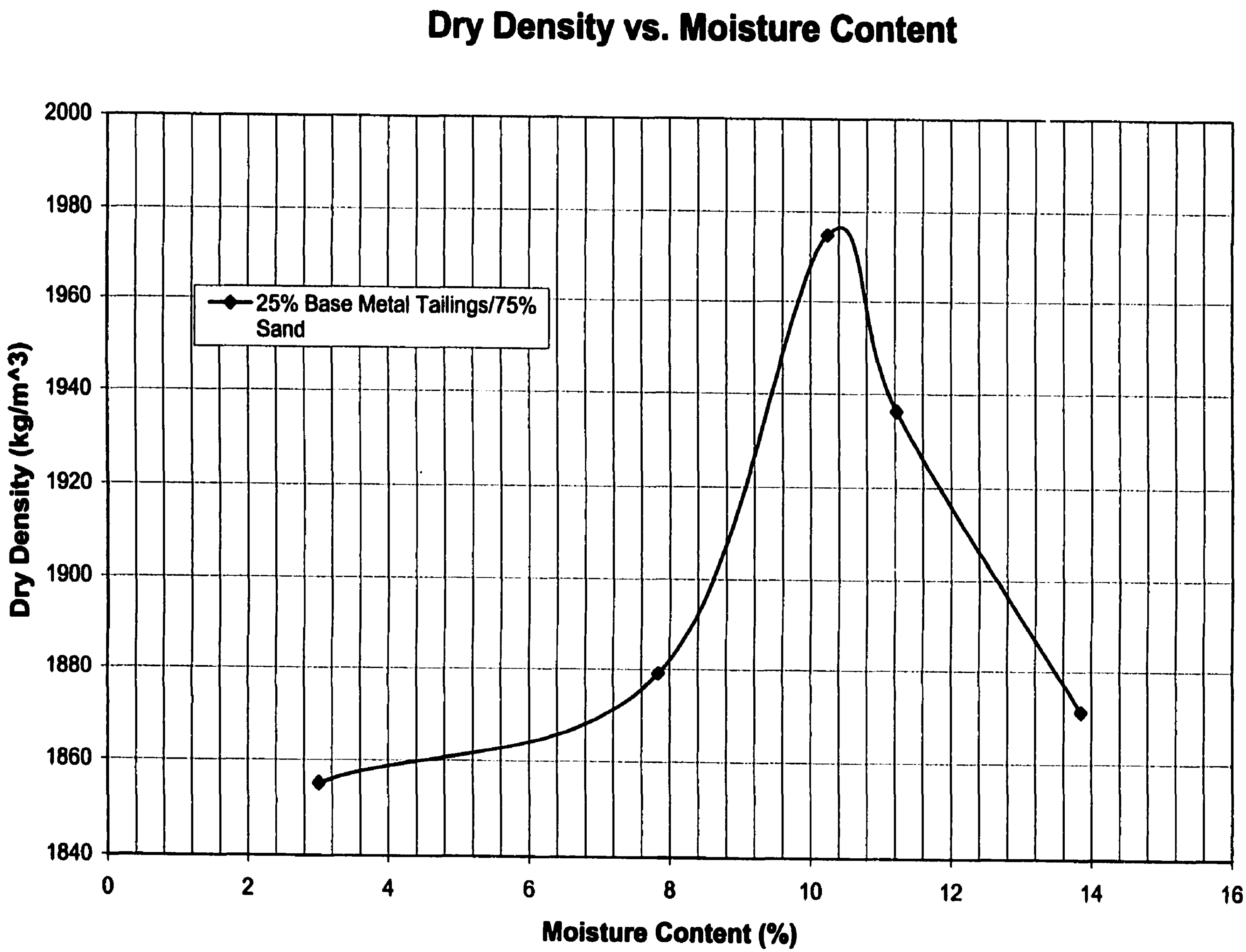
Dry Unit Weight Determination

Trial No.	1	2	3	4	5
Wt. Mold + Compacted Soil (kg)	6.097	6.206	6.348	6.326	6.304
Wt. Mold (kg)	4.293	4.293	4.293	4.293	4.293
Wt. Compacted Soil (kg)	1.804	1.913	2.055	2.033	2.011
Wet Unit Weight (kg/m^3)	1911.0	2026.5	2176.9	2153.6	2130.3
Dry Unit Weight (kg/m^3)	1855.2	1879.1	1974.7	1936.2	1871.2
Void Ratio	0.479	0.461	0.390	0.417	0.467
Porosity (%)	32.4	31.5	28.0	29.5	31.8

Moisture Content Determination

Trial No.		1	2	3	4	5
Wt. Tare (g)	1	22.13	21.86	21.08	22.03	22.50
	2	21.18	22.35	21.86	21.95	21.39
Wt. Wet Soil + Tare (g)	1	51.72	62.00	57.91	68.42	78.38
	2	43.60	58.04	67.43	60.12	92.19
Wt. Dry Soil + Tare (g)	1	50.81	58.83	54.19	63.12	70.63
	2	42.94	55.26	62.70	55.91	82.40
Moisture Content (%)	1	3.08	7.90	10.10	11.42	13.87
	2	2.94	7.79	10.38	11.03	13.83
Avg. Moisture Content		3.01	7.84	10.24	11.23	13.85

Optimum Moisture Content: 10.4 %
Maximum Dry Density: 1977 kg/m^3


Dry Density vs. Moisture Content
25% Base Metal Tailings/75% Sand
Dry Density (kg/m^3)
2000
1980
1960
1940
1920
1900
1880
1860
1840
0
2
4
6
8
10
12
14
16
Moisture Content (%)

Moisture-Density Relationship Testing

Description of Soil: 100% Sand
Date of Test: June 30, 97
Performed By: Stefan Furey/Soutsay Boualavong

Weight of Mold: 4.293 kg
Volume of Mold: 944 cm^3
Specific Gravity: 2.64

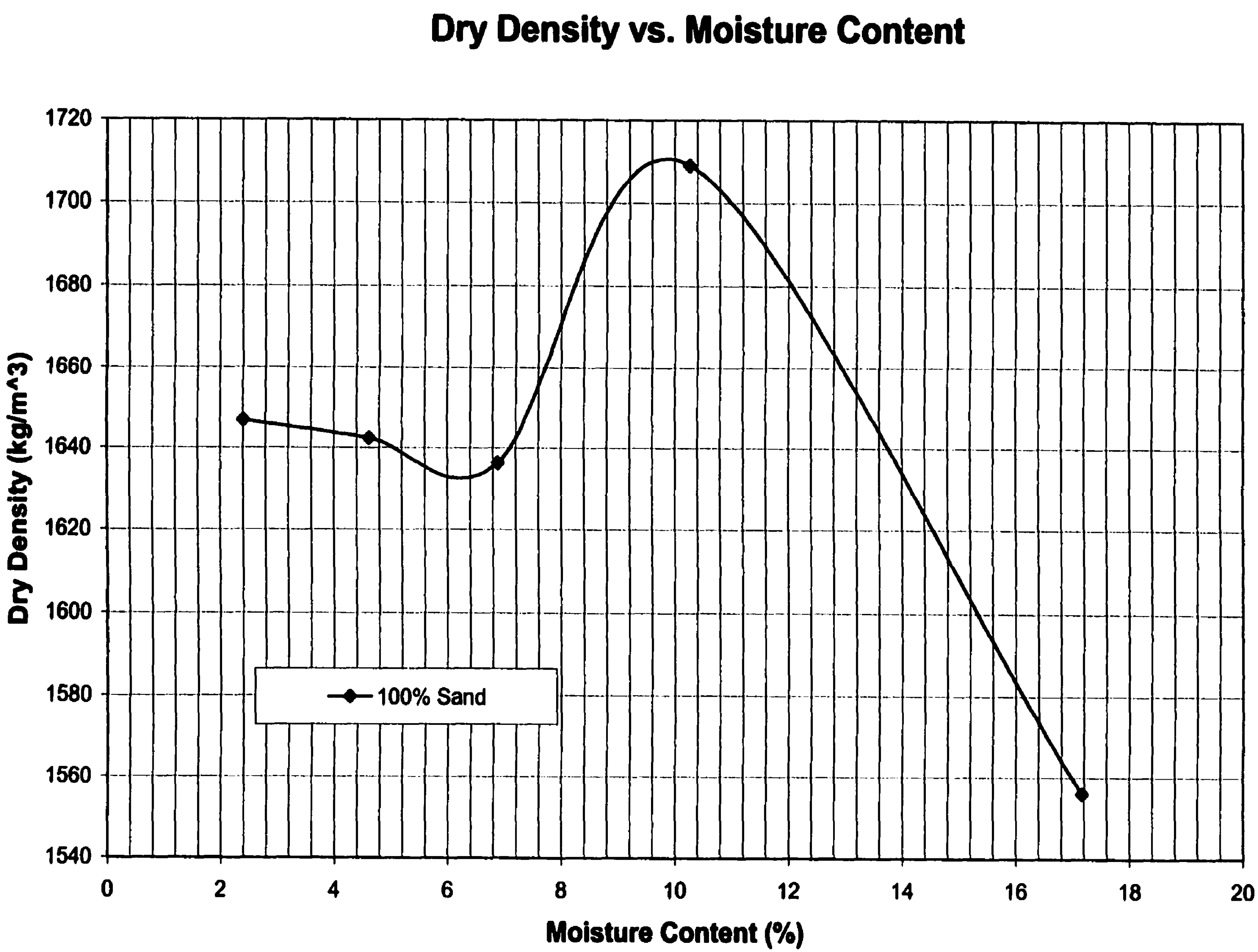
Dry Unit Weight Determination

Trial No.	1	2	3	4	5
Wt. Mold + Compacted Soil (kg)	5.885	5.915	5.944	6.072	6.014
Wt. Mold (kg)	4.293	4.293	4.293	4.293	4.293
Wt. Compacted Soil (kg)	1.592	1.622	1.651	1.779	1.721
Wet Unit Weight (kg/m^3)	1686.4	1718.2	1748.9	1884.5	1823.1
Dry Unit Weight (kg/m^3)	1646.8	1642.3	1636.3	1709.1	1556.0
Void Ratio	0.600	0.604	0.610	0.542	0.693
Porosity (%)	37.5	37.7	37.9	35.1	40.9

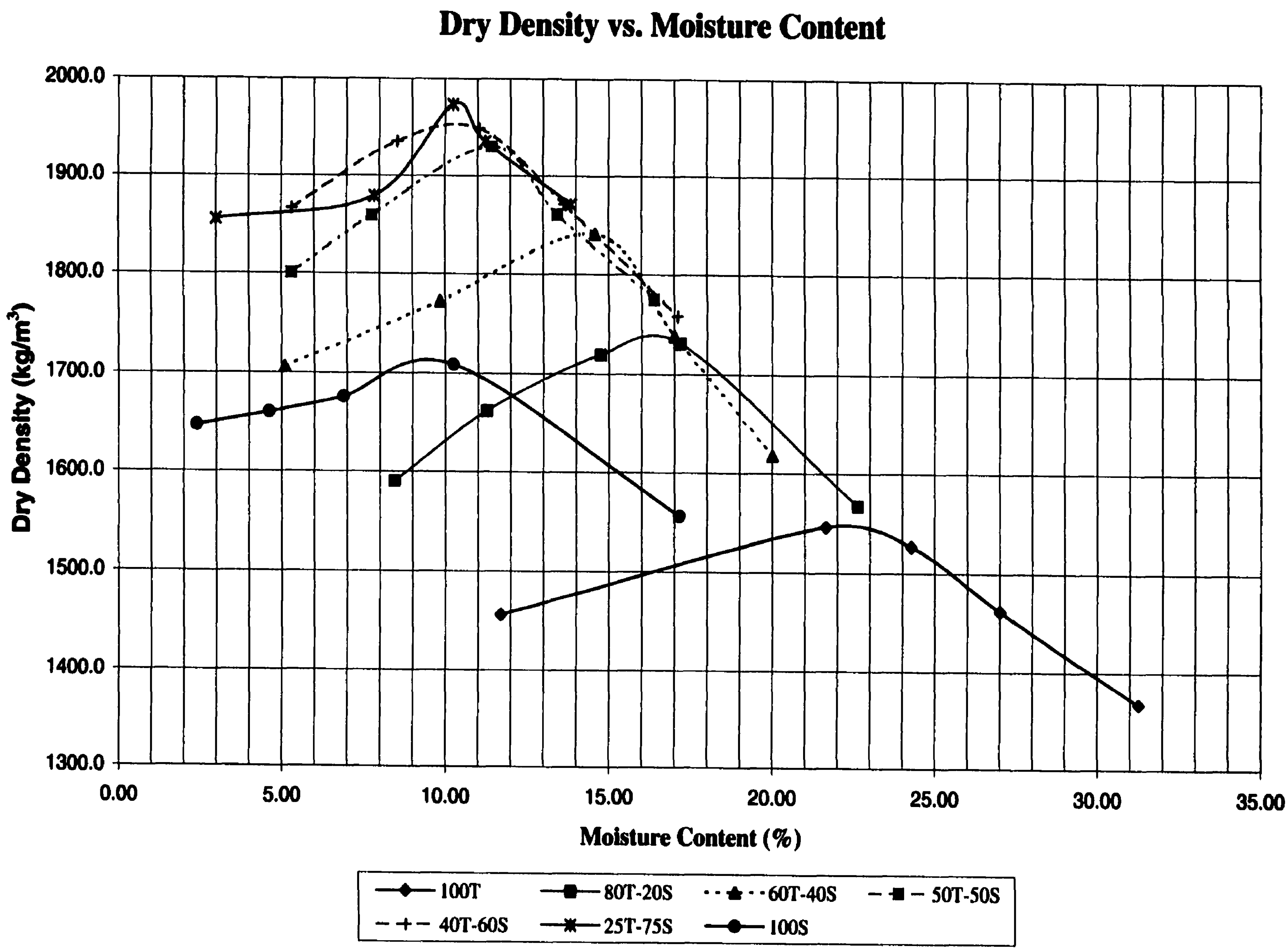
Moisture Content Determination

Trial No.		1	2	3	4	5
Wt. Tare (g)	1	22.36	20.24	22.88	22.74	21.05
	2	19.91	20.87	21.14	22.60	20.97
Wt. Wet Soil + Tare (g)	1	50.76	47.10	46.56	51.27	70.17
	2	53.32	52.04	41.77	51.85	58.41
Wt. Dry Soil + Tare (g)	1	50.08	45.85	44.94	48.35	61.73
	2	52.51	50.61	40.34	48.84	51.99
Moisture Content (%)	1	2.39	4.65	6.84	10.23	17.18
	2	2.42	4.59	6.93	10.29	17.15
Avg. Moisture Content		2.41	4.62	6.89	10.26	17.16

Optimum Moisture Content: 9.8 %
Maximum Dry Density: 1710 kg/m3



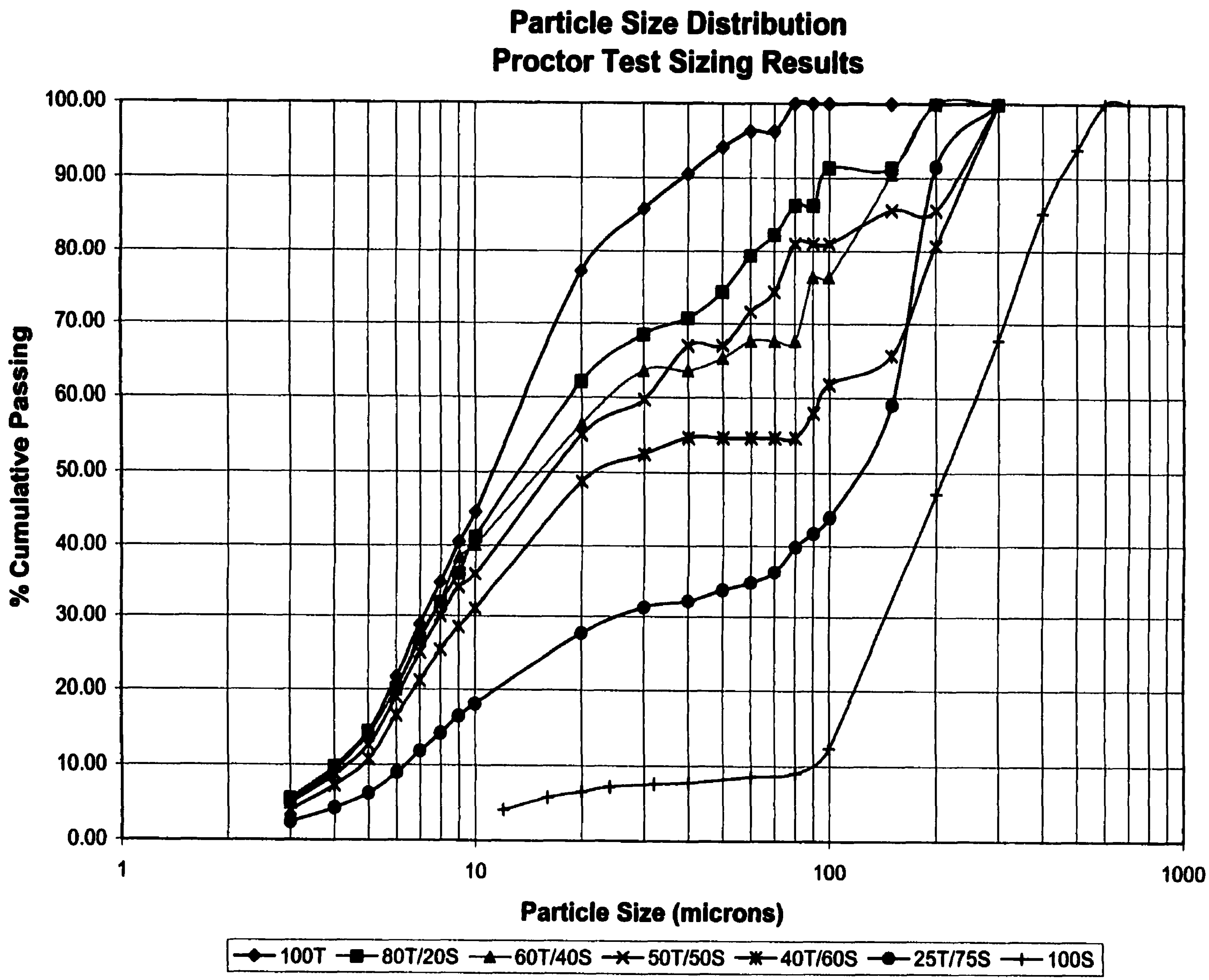
Dry Density vs. Moisture Content
1720
1700
1680
1660
1640
1620
1600
1580
1560
1540
Dry Density (kg/m^3)
100% Sand
0
2
4
6
8
10
12
14
16
18
20
Moisture Content (%)


Dry Density vs. Moisture Content
Dry Density (kg/m3)
2000.0
1900.0
1800.0
1700.0
1600.0
1500.0
1400.0
1300.0
0.00
5.00
10.00
15.00
20.00
25.00
30.00
35.00
Moisture Content (%)
100T
80T-20S
60T-40S
50T-50S
40T-60S
25T-75S
100S

Particle Size (µm)	100T	80T/20S	60T/40S	50T/50S	40T/60S	25T/75S	Particle Size (µm)	100S
3	5.11	5.50	5.51	4.95	3.99	2.36	12	4.08
4	9.38	9.76	9.79	8.70	7.27	4.25	16	5.77
5	14.27	14.40	13.95	12.68	10.77	6.24	20	6.56
6	21.70	20.13	20.34	19.00	16.55	8.99	24	7.18
7	28.84	26.82	27.34	25.06	21.23	11.88	32	7.56
8	34.75	31.91	32.09	30.04	25.43	14.21	40	7.77
9	40.49	36.07	38.41	34.07	28.55	16.47	60	8.56
10	44.59	41.20	40.10	36.00	31.08	18.11	80	9.06
20	77.19	62.16	56.55	55.05	48.76	27.71	100	12.40
30	85.83	68.52	63.57	59.76	52.45	31.29	200	47.11
40	90.54	70.77	63.57	67.04	54.66	32.19	300	67.83
50	94.13	74.38	65.33	67.05	54.66	33.78	400	85.31
60	96.27	79.37	67.71	71.64	54.66	34.81	500	93.86
70	96.27	82.30	67.71	74.38	54.66	36.32	600	100.00
80	100.00	86.35	67.71	81.05	54.66	39.87	700	100.00
90	100.00	86.35	76.41	81.05	58.00	41.77		
100	100.00	91.41	76.41	81.05	61.79	43.92		
150	100.00	91.41	90.55	85.71	65.71	59.11		
200	100.00	100.00	100.00	85.71	80.79	91.56		
300	100.00	100.00	100.00	100.00	100.00	100.00		

Summary

	100T	80T/20S	60T/40S	50T/50S	40T/60S	25T/75S	100S
- 20 µm	77.19	62.16	56.55	55.05	48.76	27.71	6.56
- 75 µm	98.14	84.33	67.71	77.72	54.66	38.10	8.94
Coefficient uniformity ($C_u = d_{60}/d_{10}$)	3	4.5	5.5	7.3	19.6	24.2	2.9



Particle Size Distribution
Proctor Test Sizing Results
% Cumulative Passing
100.00
90.00
80.00
70.00
60.00
50.00
40.00
30.00
20.00
10.00
0.00
1
10
100
1000
Particle Size (microns)
100T
80T/20S
60T/40S
50T/50S
40T/60S
25T/75S
100S

Moisture-Density Relationship Testing

Description of Soil: 100% Precious Metal Tailings
Date of Test: 1/19/1998
Performed By: Andrew Breckon

Weight of Mold: 4.292 kg
Volume of Mold: 937 cm^3
Specific Gravity: 3.43

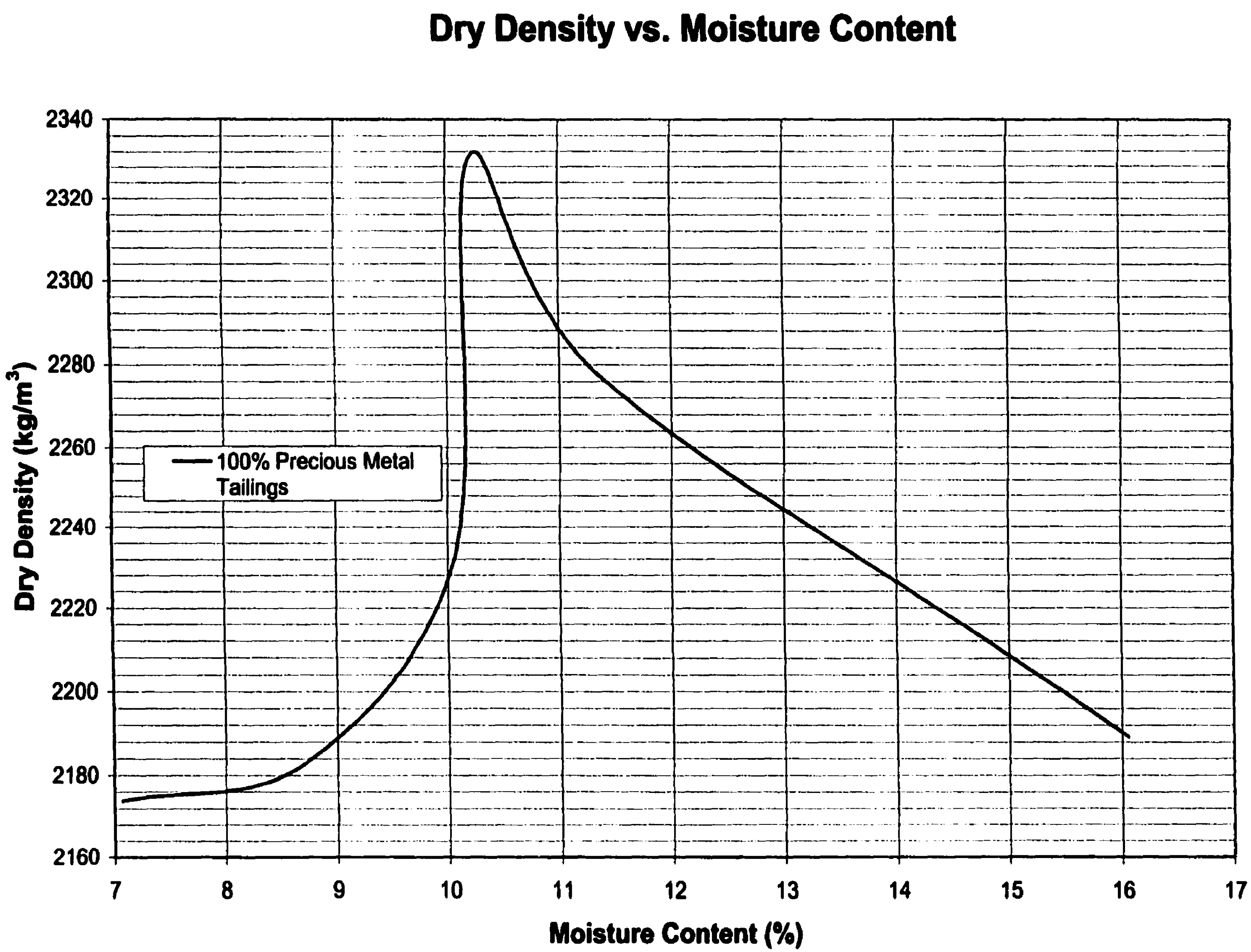
Dry Unit Weight Determination

Trial No.	1	2	3	4	5	6
Wt. Mold + Compacted Soil (kg)	6.473	6.524	6.595	6.691	6.673	6.673
Wt. Mold (kg)	4.292	4.292	4.292	4.292	4.292	4.292
Wt. Compacted Soil (kg)	2.181	2.232	2.303	2.399	2.381	2.381
Wet Unit Weight (kg/m^3)	2327.6	2382.1	2457.8	2560.3	2541.1	2541.1
Dry Unit Weight (kg/m^3)	2173.8	2182.9	2231.6	2331.2	2278.0	2189.3
Void Ratio	0.575	0.568	0.534	0.468	0.503	0.564
Porosity (%)	36.5	36.2	34.8	31.9	33.5	36.0

Moisture Content Determination

Trial No.		1	2	3	4	5	6
Wt. Tare (g)	1	21.29	20.34	20.30	19.81	20.80	21.92
	2	21.06	20.87	22.18	20.97	21.13	20.40
Wt. Wet Soil + Tare (g)	1	35.84	37.55	35.69	29.68	53.18	45.94
	2	37.20	43.15	47.05	43.84	40.11	39.47
Wt. Dry Soil + Tare (g)	1	34.81	35.98	34.13	28.71	49.44	42.08
	2	36.06	41.30	44.57	41.41	38.00	36.41
Moisture Content (%)	1	7.08	9.12	10.14	9.83	11.55	16.07
	2	7.06	8.30	9.97	10.63	11.12	16.05
Avg. Moisture Content		7.07	8.71	10.05	10.23	11.33	16.06

Optimum Moisture Content: 10.3 %
Maximum Dry Density: 2332 kg/m^3



Dry Density vs. Moisture Content
Dry Density (kg/m³)
2340
2320
2300
2280
2260
2240
2220
2200
2180
2160
100% Precious Metal Tailings
7
8
9
10
11
12
13
14
15
16
17
Moisture Content (%)

Moisture-Density Relationship Testing

Description of Soil: 45% Precious Metal Tailings: 55%Sand
Date of Test: 1/19/1998
Performed By: Andrew Breckon

Weight of Mold: 4.292 kg
Volume of Mold: 937 cm^3
Specific Gravity: 3.43

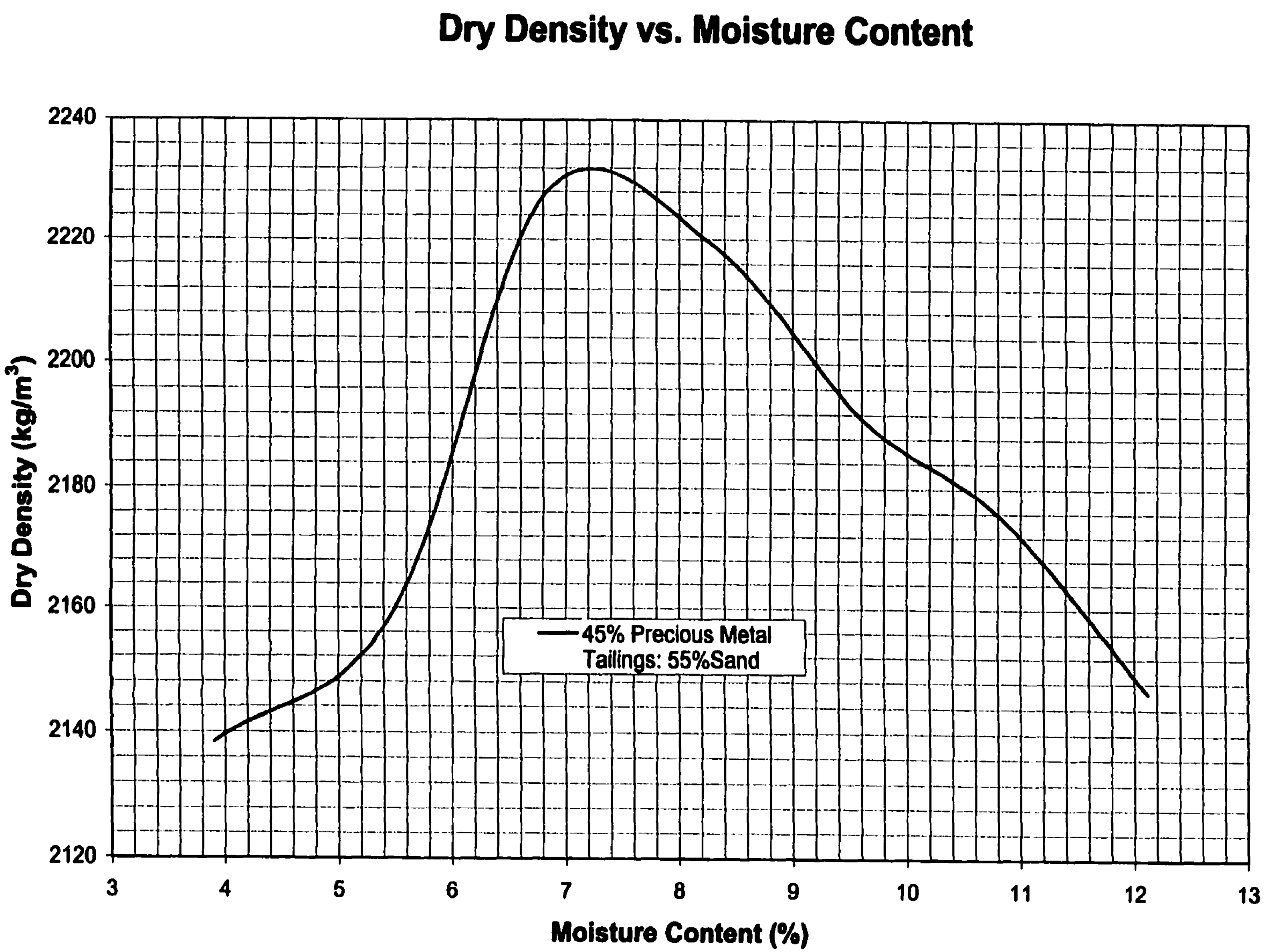
Dry Unit Weight Determination

Trial No.	1	2	3	4	5	6	7
Wt. Mold + Compacted Soil (kg)	6.367	6.426	6.521	6.537	6.544	6.551	6.548
Wt. Mold (kg)	4.292	4.292	4.292	4.292	4.292	4.292	4.292
Wt. Compacted Soil (kg)	2.075	2.134	2.229	2.245	2.252	2.259	2.256
Wet Unit Weight (kg/m^3)	2214.5	2277.5	2378.9	2395.9	2403.4	2410.9	2407.7
Dry Unit Weight (kg/m^3)	2138.4	2158.4	2228.2	2219.1	2191.0	2175.2	2146.4
Void Ratio	0.601	0.586	0.536	0.543	0.562	0.574	0.595
Porosity (%)	37.5	36.9	34.9	35.2	36.0	36.5	37.3

Moisture Content Determination

Trial No.		1	2	3	4	5	6	7
Wt. Tare (g)	1	21.73	20.93	21.00	21.22	20.41	21.97	22.05
	2	21.91	20.99	21.86	20.85	20.34	20.34	21.04
Wt. Wet Soil + Tare (g)	1	38.03	31.62	41.11	32.89	57.33	61.48	59.68
	2	32.99	36.77	41.07	41.71	39.83	48.50	47.98
Wt. Dry Soil + Tare (g)	1	37.45	31.03	39.75	31.96	53.75	57.20	55.10
	2	32.52	35.92	39.74	39.91	37.97	45.45	44.73
Moisture Content (%)	1	3.56	5.52	6.76	7.97	9.70	10.83	12.17
	2	4.24	5.39	6.92	8.63	9.54	10.83	12.06
Avg. Moisture Content		3.90	5.45	6.84	8.30	9.62	10.83	12.12

Optimum Moisture Content: 7.20 %
Maximum Dry Density: 2232 kg/m^3


Dry Density vs. Moisture Content
Dry Density (kg/m3)
2240
2220
2200
2180
2160
2140
2120
3
4
5
6
7
8
9
10
11
12
13
Moisture Content (%)
45% Precious Metal Tailings: 55%Sand

Moisture-Density Relationship Testing

Description of Soil: 65% Precious Metal Tailings: 35% Sand
Date of Test: 19 Jan 1998
Performed By: Andrew Breckon

Weight of Mold: 4.292 kg
Volume of Mold: 937 cm^3
Specific Gravity: 3.43

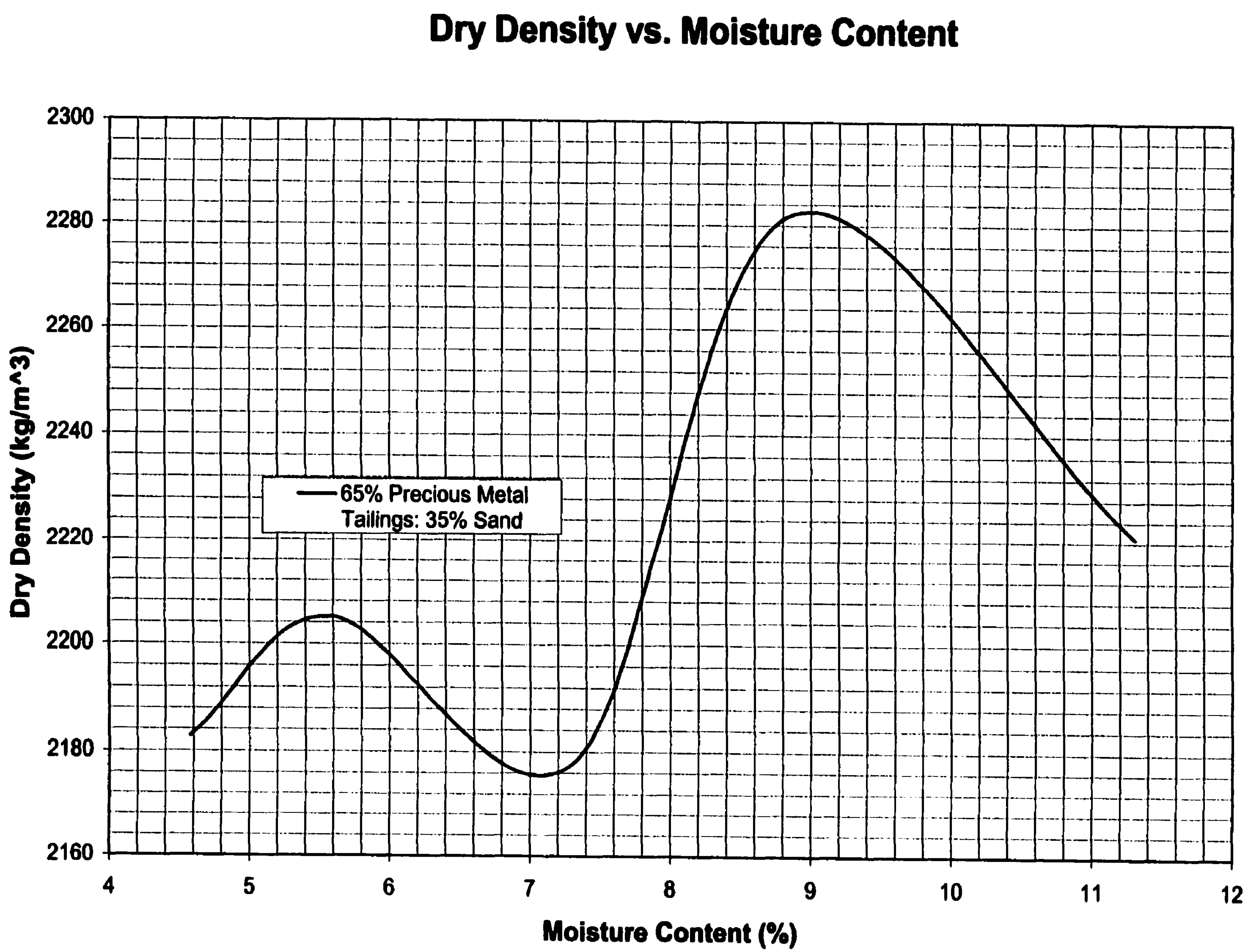
Dry Unit Weight Determination

Trial No.	1	2	3	4	5
Wt. Mold + Compacted Soil (kg)	6.429	6.475	6.483	6.616	6.608
Wt. Mold (kg)	4.292	4.292	4.292	4.292	4.292
Wt. Compacted Soil (kg)	2.137	2.183	2.191	2.324	2.316
Wet Unit Weight (kg/m^3)	2280.7	2329.8	2338.3	2480.3	2471.7
Dry Unit Weight (kg/m^3)	2182.7	2205.0	2178.0	2281.9	2220.7
Void Ratio	0.568	0.552	0.572	0.500	0.541
Porosity (%)	36.2	35.6	36.4	33.3	35.1

Moisture Content Determination

Trial No.		1	2	3	4	5
Wt. Tare (g)	1	21.79	22.12	21.94	21.98	19.82
	2	21.98	21.98	21.02	21.75	20.80
Wt. Wet Soil + Tare (g)	1	35.38	30.78	35.12	36.59	54.24
	2	28.19	32.31	36.57	42.94	51.38
Wt. Dry Soil + Tare (g)	1	34.77	30.29	34.15	35.32	50.35
	2	27.90	31.74	35.44	41.03	47.92
Moisture Content (%)	1	4.49	5.66	7.36	8.69	11.30
	2	4.67	5.52	7.27	9.01	11.31
Avg. Moisture Content		4.58	5.59	7.31	8.85	11.31

Optimum Moisture Content: 9.0 %
Maximum Dry Density: 2283 kg/m^3


Dry Density vs. Moisture Content
2300
2280
2260
2240
2220
2200
2180
2160
Dry Density (kg/m^3)
4
5
6
7
8
9
10
11
12
Moisture Content (%)
65% Precious Metal Tailings: 35% Sand

Moisture-Density Relationship Testing

Description of Soil: 75% Precious Metal Tailings: 25% Sand
Date of Test: 19 Jan 1998
Performed By: Andrew Breckon

Weight of Mold: 4.292 kg
Volume of Mold: 937 cm^3
Specific Gravity: 3.43

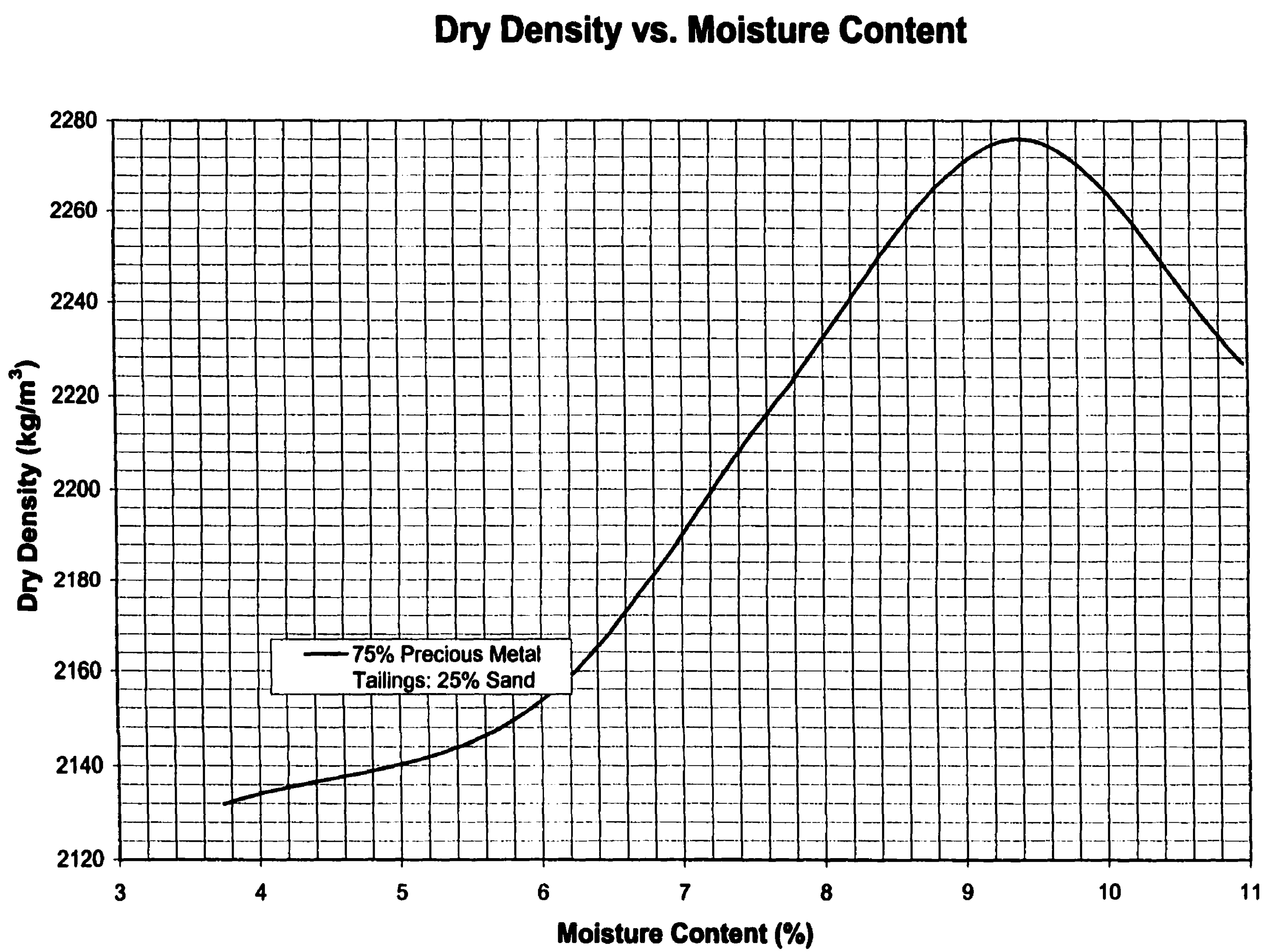
Dry Unit Weight Determination

Trial No.	1	2	3	4	5
Wt. Mold + Compacted Soil (kg)	6.366	6.425	6.525	6.628	6.609
Wt. Mold (kg)	4.292	4.292	4.292	4.292	4.292
Wt. Compacted Soil (kg)	2.074	2.133	2.233	2.336	2.317
Wet Unit Weight (kg/m^3)	2213.4	2276.4	2383.1	2493.1	2472.8
Dry Unit Weight (kg/m^3)	2131.9	2152.8	2218.9	2275.9	2226.7
Void Ratio	0.606	0.590	0.543	0.504	0.537
Porosity (%)	37.7	37.1	35.2	33.5	35.0

Moisture Content Determination

Trial No.		1	2	3	4	5
Wt. Tare (g)	1	21.91	21.86	20.86	20.41	21.10
	2	21.73	21.00	20.36	21.22	20.93
Wt. Wet Soil + Tare (g)	1	35.77	40.15	41.94	47.35	54.67
	2	36.21	37.44	39.49	45.61	40.95
Wt. Dry Soil + Tare (g)	1	35.24	39.10	40.38	44.78	50.96
	2	35.68	36.42	37.97	43.37	38.77
Moisture Content (%)	1	3.82	5.74	7.40	9.54	11.05
	2	3.66	6.20	7.95	9.18	10.89
Avg. Moisture Content		3.74	5.97	7.67	9.36	10.97

Optimum Moisture Content: 9.4 %
Maximum Dry Density: 2276 kg/m^3



Dry Density vs. Moisture Content
2280
2260
2240
2220
2200
2180
2160
2140
2120
Dry Density (kg/m3)
3
4
5
6
7
8
9
10
11
Moisture Content (%)
75% Precious Metal Tailings: 25% Sand

Moisture-Density Relationship Testing

Description of Soil: 25% Precious Metal Tailings: 75% Sand
Date of Test: 19 Jan 1998
Performed By: Andrew Breckon

Weight of Mold: 4.292 kg
Volume of Mold: 937 cm^3
Specific Gravity: 3.43

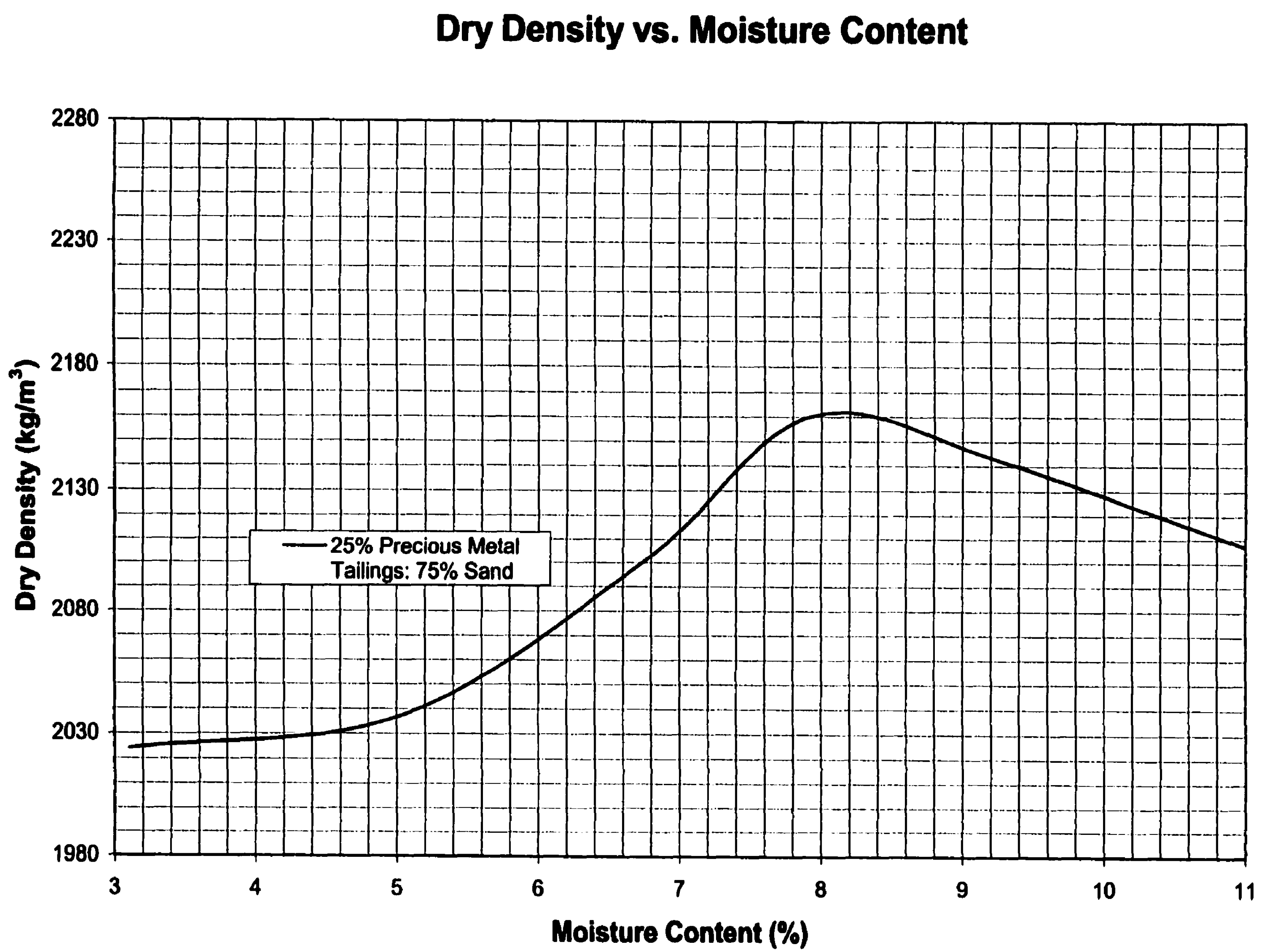
Dry Unit Weight Determination

Trial No.	1	2	3	4	5	6
Wt. Mold + Compacted Soil (kg)	6.25	6.301	6.398	6.478	6.488	6.484
Wt. Mold (kg)	4.292	4.292	4.292	4.292	4.292	4.292
Wt. Compacted Soil (kg)	1.958	2.009	2.106	2.186	2.196	2.192
Wet Unit Weight (kg/m^3)	2089.6	2144.1	2247.6	2333.0	2343.6	2339.4
Dry Unit Weight (kg/m^3)	2024.2	2037.5	2100.0	2160.0	2141.8	2090.5
Void Ratio	0.691	0.680	0.630	0.585	0.598	0.637
Porosity (%)	40.9	40.5	38.7	36.9	37.4	38.9

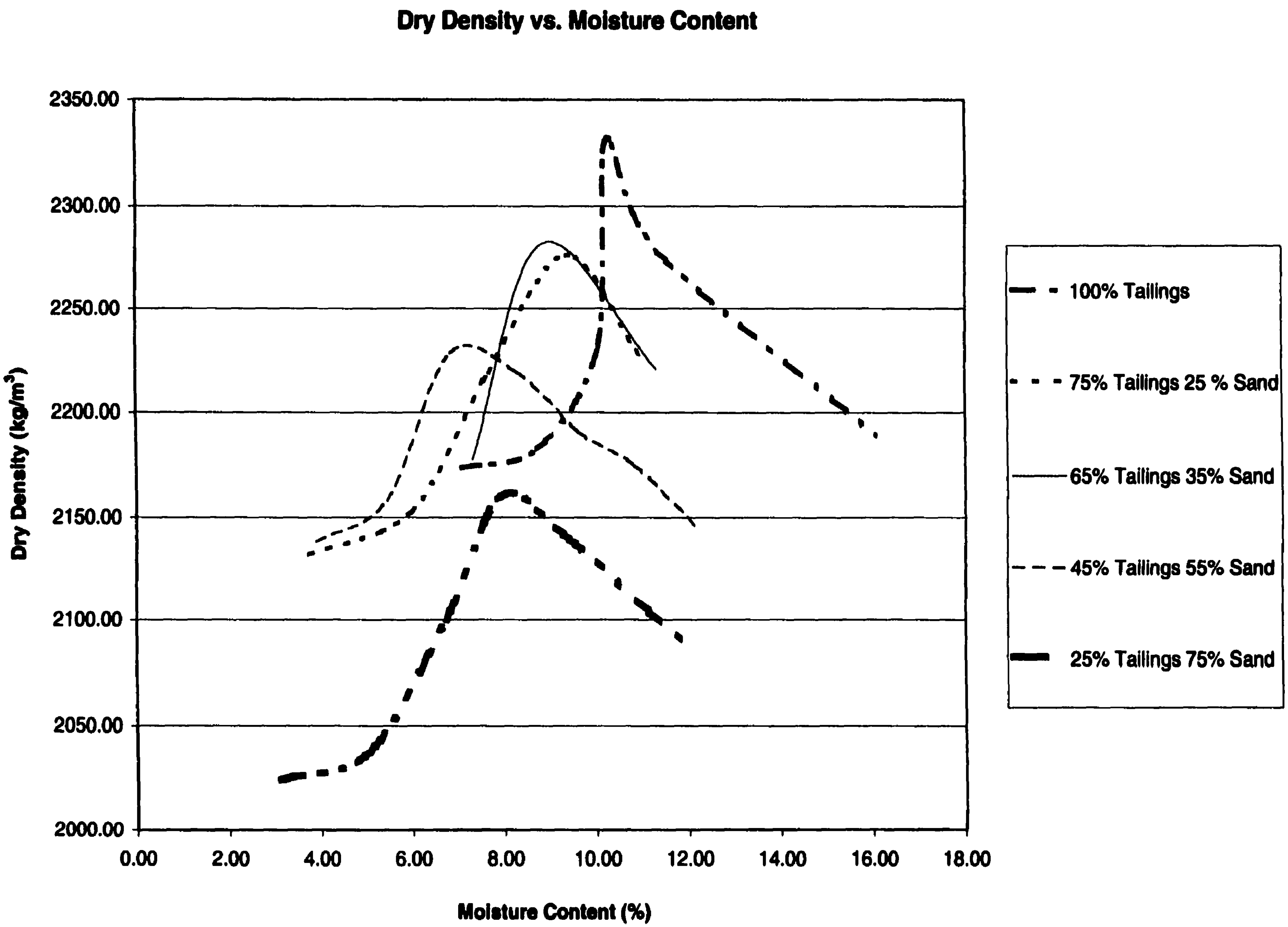
Moisture Content Determination

Trial No.		1	2	3	4	5	6
Wt. Tare (g)	1	20.97	21.77	21.82	21.09	22.06	20.43
	2	21.93	21.03	21.28	21.20	20.50	22.18
Wt. Wet Soil + Tare (g)	1	28.39	31.14	29.79	36.45	56.13	46.97
	2	34.70	30.47	42.68	53.88	43.45	52.58
Wt. Dry Soil + Tare (g)	1	28.15	30.65	29.23	35.22	52.92	43.81
	2	34.32	30.01	41.30	51.32	41.35	49.03
Moisture Content (%)	1	3.23	5.23	7.03	8.01	9.42	11.91
	2	2.98	4.87	6.45	7.83	9.15	11.68
Avg. Moisture Content		3.11	5.05	6.74	7.92	9.29	11.79

Optimum Moisture Content: 8.15 %
Maximum Dry Density: 2163 kg/m^3



Dry Density vs. Moisture Content
Dry Density (kg/m3)
2280
2230
2180
2130
2080
2030
1980
3
4
5
6
7
8
9
10
11
Moisture Content (%)
25% Precious Metal Tailings: 75% Sand

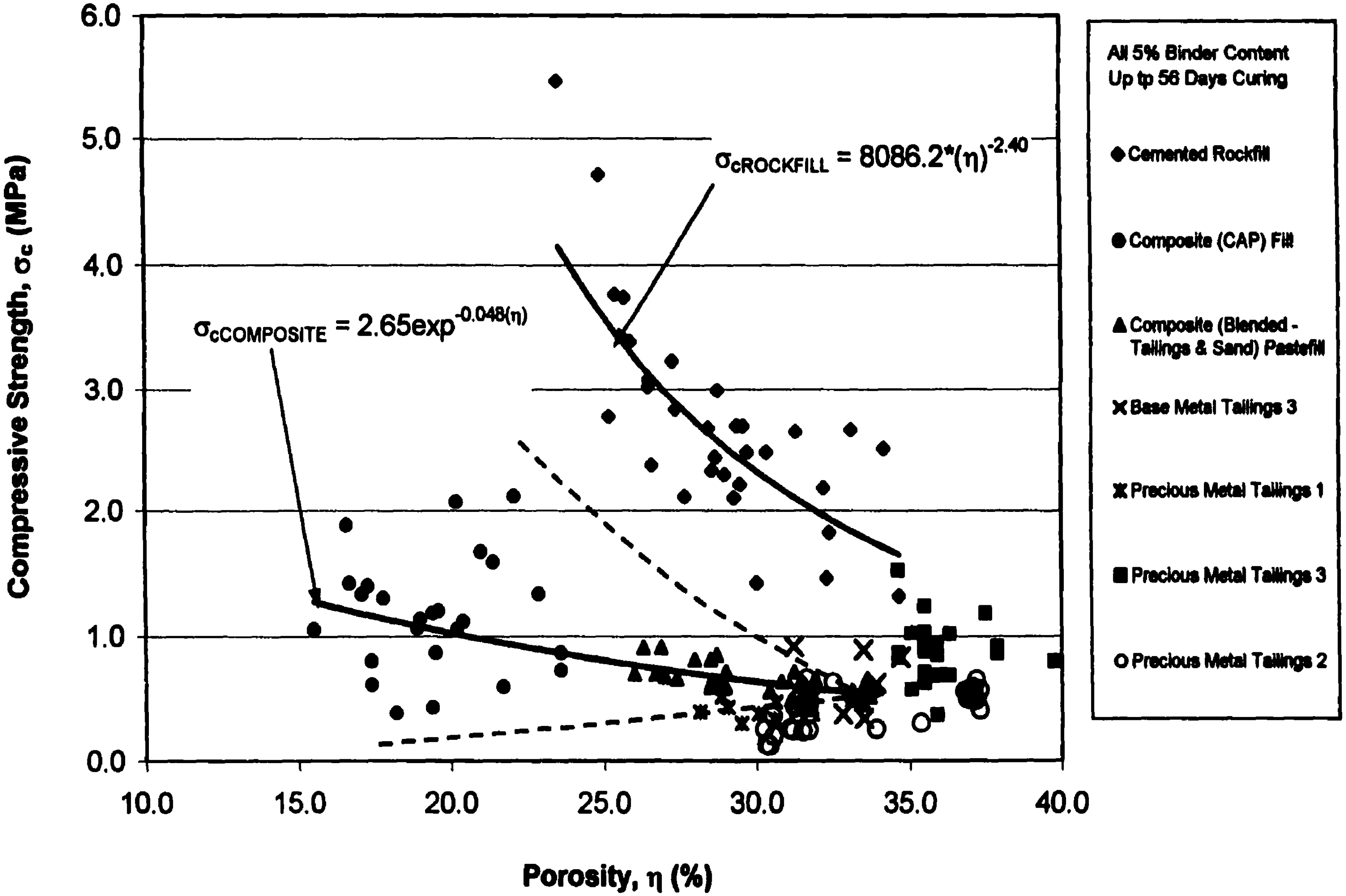

Dry Density vs. Moisture Content
2350.00
2300.00
2250.00
2200.00
2150.00
2100.00
2050.00
2000.00
Dry Density (kg/m³)
0.00
2.00
4.00
6.00
8.00
10.00
12.00
14.00
16.00
18.00
Moisture Content (%)
100% Tailings
75% Tailings 25 % Sand
65% Tailings 35% Sand
45% Tailings 55% Sand
25% Tailings 75% Sand

APPENDIX D

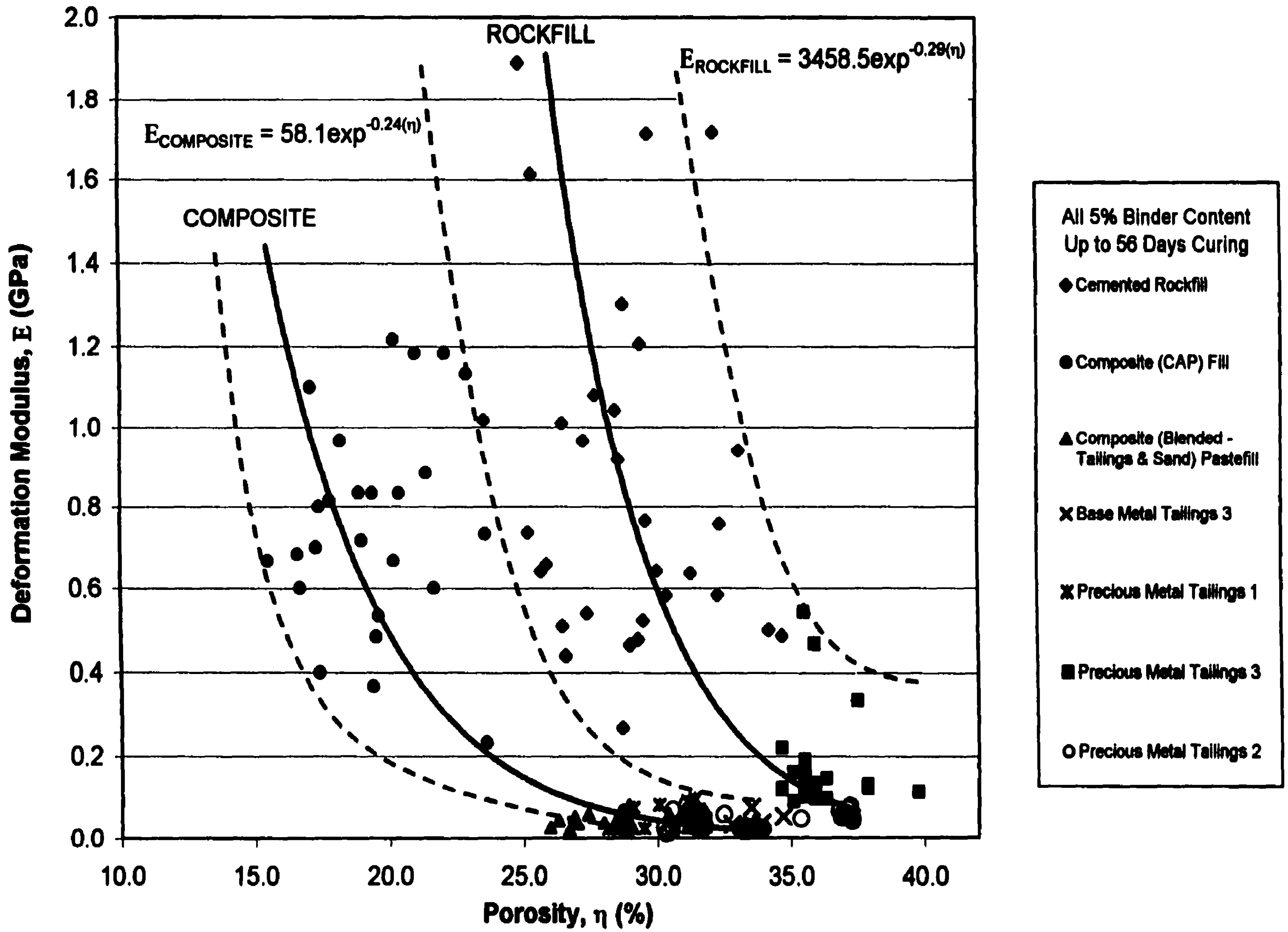
APPENDIX D

Comparisons of Mechanical Properties of the Studied Fill Materials

The established relationships between Compressive Strength, Deformation Modulus and Porosity for the studied materials are presented in this section.



Relationship between Compressive Strength and Porosity for High-Density Fill Systems



Relationship between Deformation Modulus and Porosity for High-Density Fill Systems