

Characterisation of Tailings for Paste Backfill System Design

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ABSTRACT

The focus of this work has been to investigate the status of backfill technology, in particular paste backfill, a relatively new technology. A rationale specifically for paste backfill system design has been developed. To date, a limited number of mines have implemented paste backfill systems. An extensive survey of backfill literature and a review of paste backfill operations in practice was undertaken. This has been used to identify target paste backfill design criteria and the critical success factors for paste backfill system design and implementation.

Material characterisation for paste backfill has been identified as one of the key elements of paste backfill design. The criticality of determining the physical, chemical, and mechanical characteristics of tailings, which will affect the paste backfill quality and performance, has been illustrated in a case study of paste characterisation test work conducted by the author at a Canadian base metal mine.

A number of outstanding issues, primarily technical, related to the reliability of paste backfill systems design have been identified. The future advancement of paste backfill technology will depend on the resolution of these key issues.

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1.0

INTRODUCTION

Backfill has long been used to enhance underground stability and for waste disposal. In the ancient Laurion silver mines of Greece pillars of inferior ore were used for support (Shepherd, 1993). Agricola, in the first (1556) edition of his book *De Re Metallica* described the use of waste rock for disposal and to prevent subsidence in ancient mines (Hoover and Hoover, 1950). The most significant innovations in backfill have, however, occurred during the past fifty years. Advances in mineral processing technology facilitated the evolution of backfill technology and its significant influence over the development of new mining methods. Prior to the advent of flotation technology in the latter part of the last century, mine backfills consisted mainly of rock or alluvial sands and gravels. Flotation required finer grinding of ores, resulting in the production of large quantities of process tailings. The coarse fraction of tailings soon became an important source of material for sandfills or hydraulic slurry backfills.

Hydraulic slurry backfills using classified mill tailings were in common use by the 1950's. By the early 1960's cement was being used to increase the strength of hydraulic slurry backfills. Cemented backfills coupled with advances in drilling and mining equipment facilitated the introduction of bulk mining methods. Considerable research and experimentation has been conducted using mixtures of cement and other binding agents with pozzolanic properties including fly ash and smelter slags to improve backfill strengths and to reduce backfill costs.

Rockfill and cemented rockfill also continue to be used as alternatives to classified mill tailings backfills. Cemented rockfills produce high strength fills and provide a means of disposing of waste development rock. Where sufficient waste rock is unavailable, however, rock must be quarried and transported to the mine, adding to the capital and operating costs of rockfill.

More recently, advanced mineral processing and hydrometallurgical technologies have led to still finer grinding of ores. Process tailings often contain a very high percentage of fine particles. Consequently, the volume of coarse tailings sands available for hydraulic slurry backfills is often not adequate for required backfill volumes. It has been common practice to build up conventional tailings dam embankments with the coarse fraction of classified tailings, again leaving only the slimes or fine fractions, which are unsuitable for slurry backfills.

Another significant drawback to the use of hydraulic backfills is the high cost of handling the drainage water from the fill. A typical cemented backfill placed at 65% solids density produces 0.2 tonnes of water for each tonne of solid tailings placed (Cowling, 1998). This drainage water has to be collected underground and pumped to surface for disposal. Classified tailings backfills must have high permeability to ensure that adequate drainage occurs. A standard rule of thumb for percolation rate of slurry backfills is 10 cm/hr (Hassani and Archibald, 1998). However, percolation rate will vary from mine to mine depending on the particle characteristics and binder content. Fill fences and plugs used for slurry backfills must be engineered to retain the fill safely until consolidation of the fill has taken place. Fences must also be designed to freely allow drainage of water from the fill.

Paste backfill technology has emerged as the result of considerable backfill research and development work conducted since the late 1970's. Paste is formed from full plant tailings, making use of the water retention properties of the fine particles to bind water molecules to tailings particles. Consequently, paste backfills will produce a minimal amount of bleed water. Notable advantages of paste fill over hydraulic slurry backfills include reduced mining cycle times, potential equivalent strengths using less binder, and lower costs due to the elimination of elaborate bulkheads, backfill fences, and drainage systems.

In addition there are potential significant environmental advantages to using paste backfill. Paste fill reduces the amount of surface area required for tailings disposal, as typically up to 65% of the total mill tailings can be placed underground. The capital and operating costs associated with conventional tailings dam structures can be reduced. Conventional tailings dams are engineered structures. The costs related to geotechnical aspects of designing, constructing and monitoring tailings dams can be reduced through both the smaller total volumes of tailings disposed on surface and through the decreased volume of water and slimes. Underground paste disposal reduces the environmental impact of potentially acid generating tailings.

Particularly in areas where water is a scarce commodity, the de-watering of full plant tailings to produce pastes maximizes the amount of re-cycled water available for processing. Surface disposal of paste tailings also offers environmental advantages including a reduction in the footprint compared to conventional tailings impoundments. Surficial deposition of pastes potentially reduces the amount of water that must be collected and treated prior to discharge. Another environmental advantage is that pastes deposited on surface can be contoured and reclaimed almost immediately instead of waiting until mine closure.

While the fine particles in tailings are the basis for producing paste, they also contribute to technological challenges in the preparation, transportation, and placement of paste fill. The fines exhibit unique rheological behavior and geomechanical properties that have a significant effect on the de-watering and the transportation of paste fills. The mineralogical/chemical composition of tailings can also have a profound effect on the strength, stability, and environmental performance of paste. A complete characterisation of the tailings material must be conducted in order to ensure that the required target paste backfill design criteria can be achieved.

Initially, the high cost of producing and transporting paste fills was prohibitive and paste backfill quality and performance was inconsistent, resulting in limited use of paste backfill. However, research and operating experience has increased the understanding of the rheological and mechanical behavior of pastes. Advances in de-watering and transportation technology, combined with the growing environmental pressures concerning the disposal of mine waste have improved the economics of using paste fill.

During the past decade paste backfill systems have been implemented in some underground mines. The technology, however, has failed to gain widespread acceptance and use. Paste backfill systems are relatively complex and costly to design and operate. It is clear that a need remains for both theoretical and applied research to improve the efficiency and reliability of paste systems in practice. There are a number of technological, environmental, and economic issues concerning paste backfill, which still need to be addressed. More research and development is required to overcome the technological challenges, improve paste backfill system design, enhance paste backfill quality and performance, and facilitate the dissemination of paste technology.

1.1 Thesis Overview

This thesis focuses on the elements of design for paste backfill systems. It has assembled and analysed the design and implementation of paste backfill systems at several mines and has attempted to analyse the effectiveness of procedures and methodology for design.

Paste backfill is a relatively new technology and will continue to gain acceptance by the mining industry as the technological "kinks" are ironed out and as the database of knowledge and practical experience with paste system design grows. This thesis aims to contribute to the advancement of knowledge in this process and to show the benefit of building on the design and practical operating

experience gained by existing paste backfill operators to enhance the design of future systems.

A design flowchart is presented in this thesis, which is founded on backfill engineering design rationales previously proposed by Scoble and Piciacchia (1986) and by Hassani and Bois (1992). Paste backfill system design criteria are established, which are based on target backfill properties, derived from a set of proposed backfill functions. An extensive literature review, supplemented by personal communication with engineering personnel at mines using paste backfill, as well as with paste consultants and suppliers, has been used to identify these backfill functions and target paste backfill design criteria.

Material characterisation has been identified as one of the key elements of paste backfill design. The physical, chemical, and mechanical characteristics of tailings material are critical in governing the performance of paste backfill. In this thesis, particular attention is therefore given to the physical, chemical, and mechanical characterisation procedures and methodologies. The manner in which these characteristics can affect backfill quality and performance is then reviewed.

The thesis concludes by reviewing a program of characterisation test work undertaken by the author over the summer of 1998, in conjunction with paste consultants and mine personnel, at Boliden-Westmin's Myra Falls operation, as part of its investigation of the feasibility of converting from an existing hydraulic backfill system to paste backfill.

1.1.1 Contribution

Previous research on backfill design rationale has tended to consider backfill types in general. This thesis focuses on paste backfill.

Research and operating experience have advanced paste technology considerably in the past two decades. Despite the significant amount of time and effort that has been invested in paste research, however, there are still relatively few totally successful paste backfill systems in operation today. Today's high productivity bulk mining methods fundamentally depend on reliable backfill systems. Mining companies are understandably reluctant to embrace "unproven" technologies, especially when their entire production schedule is riding on the reliability of the new technology.

Reliability notwithstanding, there is often a mindset against the acceptance of new technology. The "if it isn't broken, don't fix it" attitude is very difficult to overcome. For a new technology such as paste backfill, which is relatively complex and is still evolving, acceptance is slow. Also, experience and expertise in paste technology tends to be concentrated in relatively few firms and institutions. Dissemination of the technology is slowed due to the retention of proprietary knowledge associated with equipment, processes, reagents, and operating data. It is therefore, important to identify, examine, and resolve the issues that are critical to the advancement of paste technology and paste backfill system design, maximizing system reliability and efficiency.

This thesis has attempted to contribute to the body of knowledge related to paste backfill system design by meeting the following objectives:

- Identification of issues that still need to be resolved.
- Provide an updated compilation of paste case studies. The data and lessons learned from actual mine systems implementations should offer a valuable tool for streamlining and enhancing the design process for new paste backfill systems.

- Demonstrate the significance of using practical experience in an empirical approach to complement a theoretical approach to paste backfill design.
- Attempt to determine what factors have been important in the design and implementation of successful paste backfill systems and identify critical success factors. The pool of knowledge gained from practical experience provides a source of potential pitfalls and solutions to design and operating problems.
- Use a systems view of procedures and methodologies to provide a simple, effective overview of paste backfill design. This approach is intended to enhance the understanding of systems design and performance.
- Increase the motivation to develop paste technology for environmental benefit. As environmental stewardship takes an increasingly more important role in mine development and operation, there is a significant opportunity for mines to use paste backfill to reduce both the environmental impact of mining and the footprint they leave behind. The role of paste backfill in waste management, particularly for surface disposal of tailings, is largely untapped and needs to be expanded and developed.
- Develop a simple rationale for paste backfill design that can be used as a template for design.
- Establish key paste backfill design criteria and the data necessary to meet target criteria.
- Provide a database of references on range of topics related to paste backfill system design.
- Identify new and evolving issues related to paste backfill design.

- Recommend future research and development needs and priorities.

1.2 Backfill Functions

The primary functions that mine backfill serves are for ground support and control, provision of a working floor, and waste disposal.

1.2.1 Ground Support and Control

As a method of ground support, backfill can be used to improve mining recoveries and reduce dilution, while providing a safer working environment. With mining methods such as longhole stoping or room and pillar, backfill is placed to enable the recovery of pillars and secondary stopes. Backfill is also placed to provide a freestanding wall, which when exposed during the mining of adjacent stopes will allow minimal dilution.

As well as providing local support, backfill may be used for regional support, particularly in rockburst prone mining situations. The performance of backfill for local and regional support has been the subject of considerable research in South Africa, where the mining of deep, shallow dipping, narrow, gold bearing reefs presents many challenges. South African mines have a long history of severe rockburst and ground fall problems. Jager (1992) reported that an analysis of fatalities in three major South African gold mining districts in 1990 showed about 55% of all fatalities were rock related and of these fatalities, 52% were attributed to rockbursts. In addition to the serious safety issues, rockbursts and rockfalls also contribute to economic losses due to both decreased mining recoveries and increased dilution.

Modelling and in situ testing has been used to determine the potential for backfill to limit rockburst damage and overall seismic activity, improve strata control and

increase mining recoveries in South African mines. The main types of backfills used in South African mines have been sandfills, both cemented and uncemented; hydraulic slurry backfills or de-slimed fills; and rockfills; with limited use of high density slurry backfills and paste fills.

Regional support is applied to reduce the volumetric convergence in mined out areas and reduce stresses at the face. The concept of spatial rate of energy release (ERR) in deep mines has become an accepted criterion for determining the rockmass conditions and rockburst hazard in South African mines. Generally, an ERR of 30 MJ/m² is considered to be acceptable for the deep mines. ERR, rockburst damage, and seismicity are all related. Stabilizing-reef pillars or concrete pillars have been the most common method of reducing convergence. However, the use of permanent stabilizing pillars reduces ore recovery to 85% or less. As mining proceeds deeper, larger pillars will have to be left behind to maintain acceptable ERR's, further reducing the ore extraction. The increased cost of mining deeper coupled with the loss of potential revenue due to unrecovered gold from the pillars, will make deeper mining uneconomic.

Many South African mines have investigated the use of backfill to replace or supplement stabilizing pillars as a means of regional support. In their review of backfill practices Gürtunca and Gay (1993) found that the ability of backfill to provide sufficient regional support to control seismic events was inconclusive, possibly because the total percentage of backfilled areas in many mines was small. They also concluded that because stabilizing pillars are considerably stiffer than backfills, backfill can not in the short term, at least, affect regional seismicity.

Jolly (1993) investigated the use of backfilling at the Doornfontein Gold Mine from 1989-1992, where narrow stope, longwall mining occurs along the Carbon Leader reef, at depths between 2000 metres and 2800 metres. He reported that backfill resulted in generally better conditions, with fewer fall of ground accidents,

but higher seismicity levels occurring in backfilled areas. Although seismicity was increased in backfilled areas compared to non-backfilled areas, the damage to faces in backfilled stopes was considerably less. Proximity of backfill to the stope face was identified to be a key factor.

As a means of local support backfill stabilizes the hangingwall, resulting in reduced rockfall and rockburst incidence and damage. Gürtunca et al. (1989) reported that the reduction of convergence during rockbursts in filled stopes is considerable compared to conventionally supported stopes (timber props, timber packs, hydraulic props). Gürtunca and Gay (1993) cited a survey of rockfall and rockburst accidents in conventional and backfilled stopes conducted by Squelch and Gürtunca in 1991. This survey of South African mines showed there was a significant decrease in the accident rate provided that a minimum of 60% to 70% of the mined out area had been backfilled and that the face-to-backfill distance was 6 metres or less.

Hemp (1993) reported research on seismicity and ground motion in backfilled stopes and concluded that seismicity is not dependent on the presence of backfill, but is predominantly controlled by mine layouts and geological features. However, backfill was shown to control ground motion as peak particle velocities and accelerations, the development of low frequency surface waves, and vibration times were all reduced in backfilled stopes compared to unfilled stopes.

Spottiswoode and Churcher (1988) studied the effect of backfill on the transmission of seismic energy in a South African longwall mining situation. They analyzed data collected from a seismic network of geophones installed over a 2 square kilometre mining area at depths between 1.5 and 2 kilometres. Based on their analysis of the seismic data, they concluded that backfill shortens the effective length of the hangingwall beam, leading to an increase in resonance frequency of the hangingwall, a reduction of vibration levels, and a reduction in

differential movement between the roof and the floor. Because the resonance frequency is increased, the amount of energy radiated is decreased.

Quantitative measurements of backfill response have been hampered by a lack of instrumentation that can withstand the severe in situ conditions. Gay, Jager, and Piper (1988) reported on a quantitative evaluation of fill performance in South African gold mines. Stress cells similar to flat jack stress cells were placed in orthogonal nests inside backfill paddocks prior to filling. Cells were also used to measure stress on stope hangingwalls, down dip and along strike. These cells all failed due to shearing and lateral movement of the fills and corrosion. Attempts to measure strain with mechanical telescopic closure meters placed adjacent to the stress meters also failed, mainly due to buckling of the instrumentation. Lateral deformation was measured using extensometers, which proved to be more robust because the wiring was protected by extendible tubing. Subsequently, a Goodman Jack, typically used in civil applications for measuring stresses in soils and soft rock, was used in filled backfill paddocks to obtain stress measurements. Although performance of the instrumentation was sporadic, the authors were able to conclude that the backfill generally provided significant support benefits. They found that in situ load bearing properties as measured with stress and strain meters gave results similar to laboratory tests.

Backfilling in two separate areas of the Denison uranium mine, located at Elliot Lake, Ontario has been used successfully to control violent failure of pillars (Pritchard, Townsend, and Hedley, 1992). The authors reported that the backfill appeared to control the release of seismic energy. In an area containing cemented backfill, the magnitude of seismic events reached a maximum of 1.0 Mn, while in nearby unfilled areas, magnitudes up to 2.8 Mn were recorded. The authors concluded that this regional ground stabilization could be attributed to the use of backfills composed of de-slimed tailings and cementitious slags.

An evaluation of paste backfill performance conducted in 1995-96 at the Chimo Mine near Val d'Or, Quebec, used hydraulic pressure cells, which were placed orthogonally inside a cubic steel frame positioned at a depth of 20 metres in two stopes (Hassani, Fotoohi, and Doucet, 1998). The stress meters were positioned to give vertical, north-south (across the ore body) and east-west (along the ore body) stresses. Extensometers and vibrating wire stress measuring instruments were also used to obtain strain data. Data recorded over a 200 day period showed that the paste backfill absorbed a significant amount of energy. Mining of an adjacent stope commenced after backfilling of the first stope was completed. A major rockburst and subsequent high seismic activity occurred 115 days after commencement of backfilling. The paste backfill was shown to absorb a significant amount of energy during the monitoring period.

Installation of instrumentation in the second stope commenced after backfilling had begun. Mining activity took place adjacent to this second stope during the backfilling operation. Consequently, initial pressures measured in the second stope were higher than in the first stope. Again, the paste backfill was shown to absorb a significant amount of energy vertically and in the north-south direction, along the ore body. The authors concluded that the instrumentation performed well. They also concluded that the paste backfill protected the stope walls from sudden movement and absorbed energy through compressive strain, thus reducing the potential for rockbursting.

A key function of backfill is to improve mining recoveries and decrease dilution by providing stable support. Webbstock, Keen and Bradley (1993) reported that backfill at the Randfontein Estates Gold Mining Company's Cooke 3 Shaft, increased extraction from 78% to 95%, while maintaining safe working conditions.

Two different backfill systems, one using centrifuge de-watered tailings and the second system using cyclone classified tailings, were developed at the West

Driefontein operations (Bruce and Klokow, 1988). They reported that since the introduction of backfill, strata control in the highly stressed longwall panels improved significantly, face advance increased, and the accident rate due to rockbursts decreased, provided that backfill was within 10 metres of the stope face. The backfill enabled the increased extraction of otherwise unrecoverable ore from the barrier pillar.

Backfill may also be used to prevent caving and subsidence in near surface underground excavations. Hassani and Archibald (1998) reported that the first record use of hydraulic mine backfill in North America was in 1864 at a Pennsylvania coal mine, where the backfill was used to alleviate ground subsidence beneath the foundations of a church. Ground subsidence has primarily been a concern with the mining of flat-lying deposits, mainly coal.

Depending on the mining method used, backfill placement may be delayed or cyclical and this will affect the required backfill properties. With bulk mining methods, primary stopes are totally mined out first and then filled prior to recovery of adjacent mining areas. Hence, delayed backfill placement generally requires longer-term stability and the capability to create freestanding walls.

1.2.2 Working Platform

Cyclical mining methods, most commonly cut and fill or drift and fill type methods, can use backfill to provide a working floor, although in some bulk stoping methods, the top of a filled stope will also form the floor for the stope above. Cut and fill methods permit high recoveries and reduce dilution through selective mining, particularly in small, irregular orebodies. In overhand cut and fill, successive lifts are mined out and then backfilled, in a bottom up fashion, with each filled lift providing a foundation for mining out the next lift. In underhand cut and fill methods, successive cuts or lifts are mined out and filled, with mining proceeding downward so that the filled lifts form the roof for the next stope below.

Cut and fill mining was reported to be used by the Swedish mining company, Boliden Mineral AB in thirteen of sixteen mines (Krauland, 1989). Boliden has developed highly efficient mechanized cut and fill practices to achieve high mining recoveries from deposits with small, irregular orebodies and unfavourable rock conditions.

Cyclical fill placement requires early consolidation and strength to meet scheduling requirements the mining cycle. Cement contents are often higher in cyclical fills than in delayed fills in order to ensure a safe working floor for heavy mobile equipment. Load bearing capacity must be taken into consideration in the backfill design for provision of a working floor.

1.2.3 Waste Disposal

Backfill used for the purpose of ground support or to provide a working platform also serves a dual purpose of reducing the volume of surface disposal of waste rock and tailings. Where backfill has not been required for ground support, the underground disposal of waste materials has generally only been undertaken at sites that have no alternative areas on surface for waste disposal due to the high cost of transporting backfill underground as opposed to surface deposition. Underground disposal of waste development rock can also represent a significant cost saving in hoisting.

The global-wide environmental movement has changed both the psychology and the economics of mining waste disposal in recent years. Heightened public awareness and environmental activism have led to increasingly more stringent environmental regulations. A number of serious tailings dam failures resulting in environmental damage and/or loss of human life have helped to put the mining industry in a very bad light. The mining industry has come under intense pressure to improve waste management practices, resulting in a greater emphasis on the role of backfill in waste disposal.

The pressures of environmental stewardship and in some cases, permitting requirements, to reduce the footprint left by mining on surface have led to the investigation of the disposal of tailings and waste rock in underground voids, the co-disposal of mine wastes underground, and the development of dry-land disposal methods.

Another environmental advantage of backfill is improved health and safety working conditions. The presence of backfill underground increases the efficiency of underground ventilation and temperature control systems and reduces the risk of fire (Kamp, 1989).

1.3 Backfill Types

The most commonly used types of backfill are rockfills and hydraulic slurry fills. These fills may be cemented or uncemented, although with slurry backfills, in particular, binder addition is generally required to meet strength requirements for ground support.

In Canada, Falconbridge and Inco began adding Portland cement to the tailings backfills in their Sudbury area mines by the early 1960's. The use of cemented backfills has facilitated the adoption of bulk mining methods, leading to improved mining recoveries. Cement, however, is a high cost component of backfill. There has been considerable investigation and experimentation with alternative binding agents to Portland cement including ferrous and non-ferrous slags, fly ash, and ground glass.

1.3.1 Rockfill

Rockfill consists of waste rock or quarried material that is transported underground by truck or conveyor or is dumped down fill raises. Cemented slurries are often pumped underground to be mixed with the rockfill just prior to

placement in stopes or to be introduced to the stopes after filling. Cemented rockfill gives the highest strength and hence, the stiffest support of all backfill types.

Hassani and Bois (1992) reported that waste rock accounted for 64% of the backfill material used in Quebec mines in 1990. Where there is a ready source of rock such as development waste rock or a surface quarry or pit, the cost of material for rockfill is very low. Placement rates are high. There is no cost associated with drainage water, as the material is placed dry. The capital cost of supplying rockfill is moderate depending, of course, on availability of waste rock. In situations where a quarry must be operated to supply rockfill, there can be significant capital and operating costs. Operating costs of transporting and placing rockfill are high. It is difficult to achieve a tight fill with rockfills.

1.3.2 Hydraulic Slurry Backfill

Slurry backfills usually consist of classified mill tailings, alluvial sand or tailings/sand mixtures. Due to the fine grind required for metallurgical processing, there is often insufficient coarse material to meet backfill needs. Also, because only the coarse fraction of the tailings is used for hydraulic slurry backfills, the disposal of the slimes portion of the tailings remains a surface waste management problem.

The density of hydraulic slurry backfill generally ranges from 60% to 70% solids. One of the most important requirements of slurry backfills is high permeability, in order to promote drainage and rapid consolidation and to prevent potentially dangerous build-up of hydrostatic pressures in filled areas underground. Consequently, the use of hydraulic slurry fills creates large volumes of waste water underground. Stope preparation for slurry fills, including barricade construction, is more elaborate and costly than for other fill alternatives.

Drainage systems must be installed underground to collect the drainage water and pump it back to surface.

Capital costs for preparation and transportation of slurry backfills are relatively low. Transportation and placement operating costs are also relatively low. Cement costs, however, account for a significant portion of operating costs. A 1990 study of Quebec mines using backfill (Hassani, Bois and Newman, 1993) reported that backfill costs ranged from 1.3% to 16.4% of total operating costs. Of the mines using hydraulic fills in the Quebec study, cement costs contributed up to 43 percent of backfill operating costs.

1.3.3 Paste Backfill

The challenge of adapting to the environmental and economic pressures of tailings management has led to the development of paste backfill as an alternative to rockfill and hydraulic slurry backfill. Paste fill has seen limited use since its introduction in the late 1970's, however, in the past decade there have been significant advances in paste technology, resulting in more implementations of paste backfill systems.

Paste fill is a high density tailings backfill, incorporating the full particle size range of tailings, as opposed to conventional hydraulic slurry backfill which is produced by classifying tailings to eliminate the fines or slimes fraction of tailings. Paste fill is not defined by a specific solids density or percent of fines, however, as a general rule of thumb, a minimum of 15% minus 20 micron size particles is required to create paste properties (Landriault, 1995). The colloidal properties of the fines give paste unique water retention characteristics and, hence, unique rheological properties.

A conventional hydraulic tailings slurry exhibits Newtonian flow properties and will settle out if transported below a certain critical flow velocity. Paste, on the

other hand, exhibits pseudo-plastic behavior that results in plug flow, with the fines producing an annulus that surrounds and carries the material. When flow is stopped, the paste does not readily settle out. The water retentive properties of the fines result in minimal excess water in the paste.

There are many potential safety, environmental, and cost benefits associated with paste fill.

- From a geotechnical perspective, equivalent compressive strengths can often be achieved in paste backfills using less binder than for hydraulic slurry backfills. Paste backfill requires shorter consolidation time than conventional slurry backfill, reaching target safe strengths earlier. Drainage of excess water in slurry backfill can lead to particle size segregation and potential loss of binder. Paste fill, on the other hand, is homogenous and does not segregate, resulting in more uniform strength throughout the fill.
- Tighter fills can be achieved with placement of paste fill as opposed to both slurry fills and rockfills. There is an obvious cost benefit of placing more material in underground voids. From an environmental health and safety perspective, underground working conditions can be improved through increased efficiencies in ventilation and temperature control that can be realized with tighter fills.
- Cost benefits are achieved through reduced construction costs of bulkheads, elimination of drainage and pumping systems, shorter backfill cycle times, decreased binder costs, and reduced surface tailings dam capital and operating costs.
- Due to a minimal amount of excess water drainage from paste backfill, less elaborate fill barricades need to be constructed. Depending on mine regulations, barricades consisting of muck piles with simple fences on top

may be sufficient to retain fill until the fill plug has consolidated. After consolidation of the fill, the muck pile can potentially be removed and re-used. Cost savings are realized both in the reduction of costly barricade construction materials such as timber, shotcrete, and geotextiles and in the reduction of labour costs to install fill barricades.

- As well, systems for collecting fill drainage water and for pumping water and slimes back to surface retention and treatment facilities are not necessary. There should be minimal excess water associated with paste backfill placement other than flush water.
- Backfill cycle times can be reduced due to the elimination of staged pours to allow for consolidation and drainage. Actual pouring time of paste fill offers no time advantage over slurry backfill as pouring rates are similar. However, time saved in preparation of stopes for backfilling with paste and in consolidation of paste fill, shortens the mining cycle significantly, leading to improved productivity.
- Backfill costs are a significant component of operating costs. Binder costs, in particular, Portland cement, represent a major proportion of backfill operating costs. For every weight percent increase in cement addition, there is approximately one dollar per tonne increase in operating cost (Naylor, Farmery, and Tenbergen, 1997). Cost savings can be achieved by using paste fill, as equivalent compressive strengths can often be reached with lower binder content in paste fill as opposed to hydraulic slurry fill.
- Paste fill offers significant positive environmental benefits. Higher volumes of tailings can be placed in underground voids, thus reducing the amount of tailings leftover for surface disposal. As a result a smaller footprint is left on surface. The disposal of conventional slimes tailings on surface requires carefully engineered water retention structures with expensive on-going

geotechnical monitoring to ensure the stability of the foundation and dams. A reduction in overall volume of slimes requiring surface storage reduces the cost of surface dams considerably. If paste fill is also used for surface disposal, the need for conventional water retention tailings dams and systems for treating and transporting reclaim water can be eliminated. Reclamation of surface paste disposal areas can be undertaken on a continual basis instead of waiting until mine closure.

The primary disadvantages of paste fill have been associated with the technology of paste fill design and production. The mining industry has been averse to adopting paste technology due to these initial technological challenges, however, the rapid evolution of paste technology in the past decade has led to a growing number of paste plants in successful operation today. On-going research and development continues in many facets of paste technology including de-watering equipment design, paste rheology and transportation, in situ long-term stability of paste fills, geotechnical considerations including liquefaction potential, and reactivity of paste fill.

The use of paste has been limited, mainly due to the technological challenges presented by the de-watering of fine tailings and the transportation and distribution of paste. De-watering is a high cost component of paste production. The traditional and proven method of de-watering tailings is by thickening and filtration. Filtration is a high capital and operating cost unit operation. While flotation concentrates are generally thickened and filtered to reduce moisture contents for shipment to smelters, the cost of filtering the entire volume of tailings, which represents the bulk of the processing products, can be formidably high. Alternative methods of de-watering/thickening tailings have been developed and are currently being introduced in paste system design.

Transportation and placement of paste fills has been another technologically challenging aspect of paste system design. High density pastes, often over 80%

solids, can produce high pipeline pressures and offer significant resistance to flow over long horizontal distances. Paste pipeline transportation systems must be designed to accommodate these high pressures. Expensive positive displacement pumps may be required in order to move paste over long horizontal distances.

Despite the many advantages that paste backfill offers, technological challenges in the mixing, distributing, and placing paste backfill remain, which must be resolved. The overall reliability of paste backfill systems needs to be improved in order to provide consistent backfill quality and performance.

2.0 PASTE BACKFILL SYSTEM DESIGN

2.1 Backfill Systems Design Principles

Backfill system design rationales have been presented by Scoble and Piciacchia (1986) and by Hassani and Bois (1992), which employ an integrated stepwise process of evaluation, engineering design, implementation, and monitoring. These design rationales have been used to develop a rationale specific to paste backfill system design, which is presented in Figure 2.1. The characterisation of tailings material has been identified as the central component of paste backfill design. The physical, chemical, and mechanical properties of the tailings affect the ability of the paste to meet the required target design criteria and as such, the characterisation of tailings is critical to success in all phases of paste backfill system design from initial evaluation, through to design, construction, implementation, and on-going operation and monitoring. Paste backfill system design becomes an iterative process of engineering the various components of the system to meet target design criteria with the given tailings material.

Evaluation is an integral part of the design process. Engineering/economic evaluation essential to backfill design includes the:

- Determination of backfill function, evaluation of backfill type, and identification of backfill target properties; and
- Backfill material characterisation and evaluation

Before preliminary engineering design of the backfill system can be undertaken, the target design criteria must be established, based on the target backfill properties. The suitability of available backfill material to meet these specific target properties must then be determined. Target backfill properties are based on the backfill function, but may also be influenced by other site-specific factors.

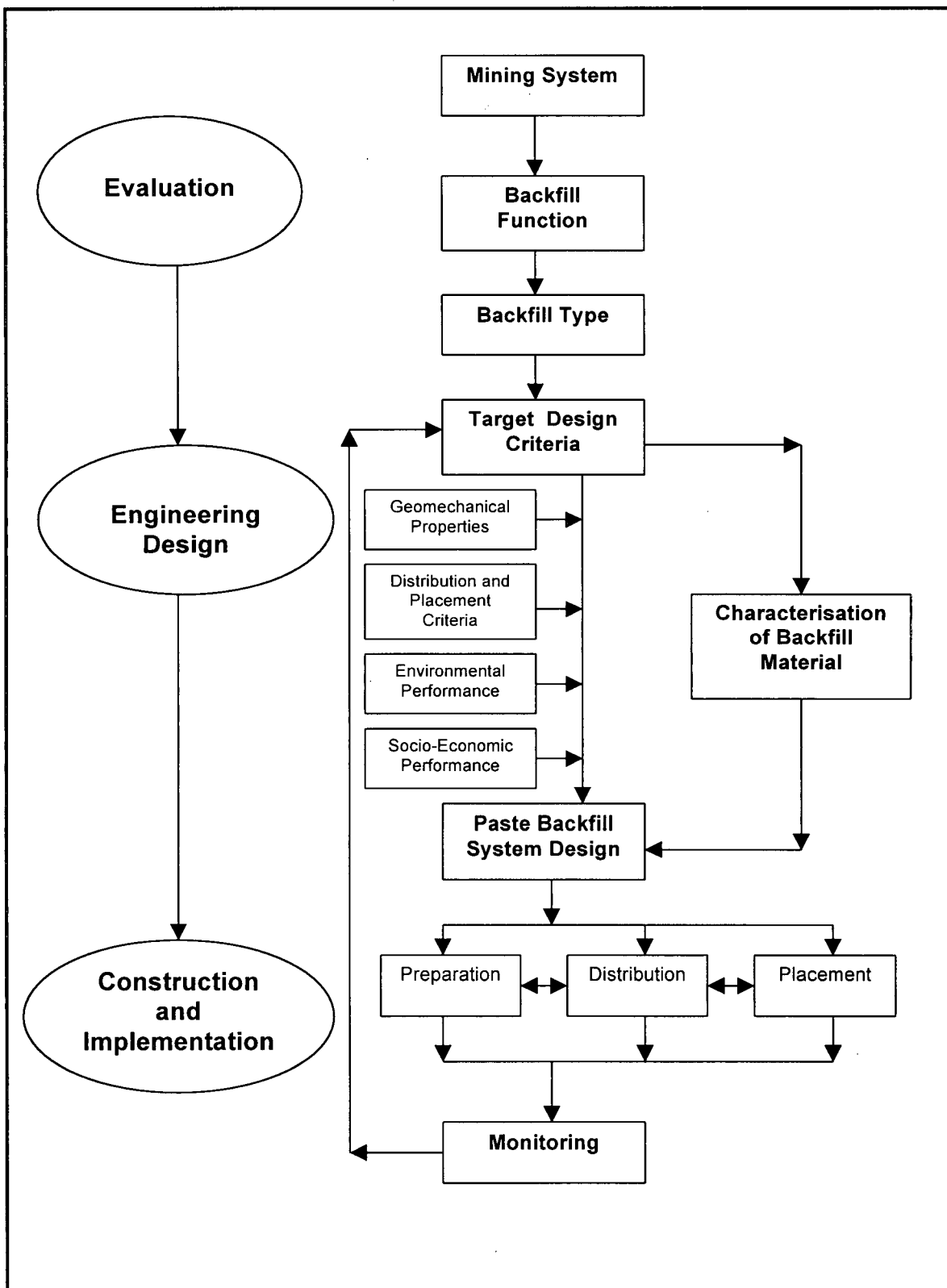


Figure 2.1 Paste Backfill System Design Rationale

The primary functions of backfill, as discussed in section 1.2, are for ground support and control; provision of a working platform; and for waste disposal, underground and/or on surface. Backfill function is predetermined by the mining system. The mining method selected is dependent on many factors as follows: the depth, size, dip, and structure of the ore body; ore geology and grade; and rock mechanics conditions including the rock mass rating of the hanging wall, footwall, and ore. Required backfill placement rates, backfill volumes, and logistical constraints for transporting and placing backfill are also based on the mine layout and mining system.

The selected mining method dictates the type of backfill required – delayed or cyclical. The required strength and curing time will be a function of the backfill type and rock mechanics conditions determined by stress, structure and rock mass. Backfill functions, suitability and availability of backfill material, and overall capital and operating costs of different backfilling alternatives will form the basis for the selection of backfill type and the identification of target backfill properties.

Regardless of the backfill type, the main design components of a backfill system are:

- Backfill preparation
- Backfill distribution
- Backfill placement

Key elements affecting each of these three main design components for paste backfill system design are presented in Figure 2.2. The quality and performance of the backfill is dependent on the careful engineering of each of the above components to meet the target design criteria. Compared to rockfill and slurry backfill systems, paste backfill systems require relatively complex engineering and design work for preparation and distribution of paste.

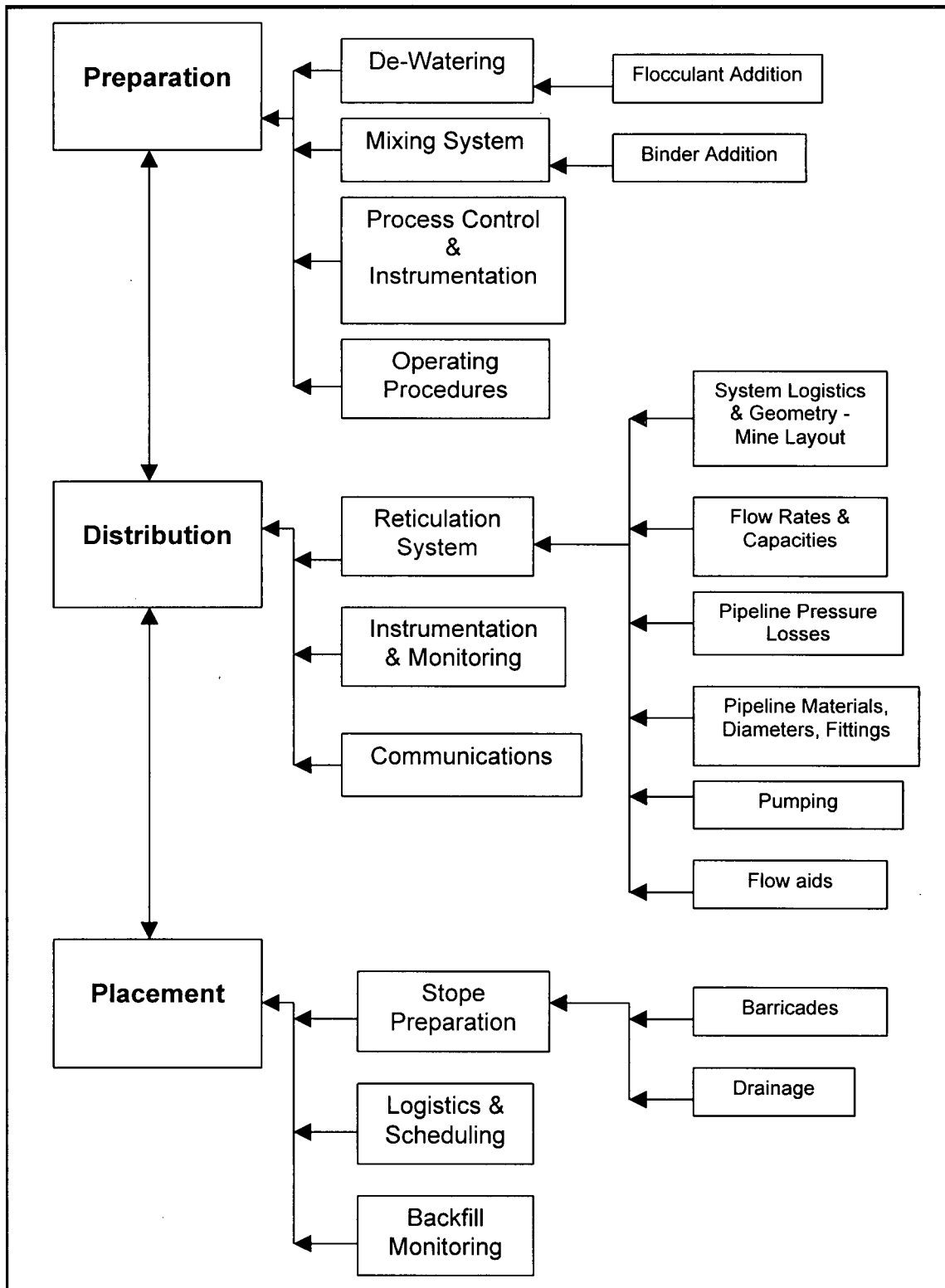


Figure 2.2 Elements of Paste Backfill System Design for Preparation, Distribution and Placement

For any backfill system once the backfill type has been selected and the target backfill properties have been identified, the target design criteria must be established. The available backfill material must then be evaluated to ensure that it meets target design criteria. For paste backfill systems the characterisation of tailings is critical to accurate engineering design. The physical, chemical, and mechanical characterisation of tailings will identify the range of geomechanical, geochemical, and rheological properties that will affect paste quality and performance. Many of the elements of design outlined in Figure 2.2 are dependent on tailings characterisation to establish the geomechanical properties and rheological behavior of the paste and the affect paste quality and performance. Design of each component of the backfill system will be based on the specific target design criteria. The system design must be optimized as a whole, taking into consideration the influence of tailings properties on each component part of the system.

Paste preparation entails de-watering of tailings followed by mixing to combine the de-watered tailings with binder and/or water to produce a paste, which meets the target design criteria. The most commonly used thickener and tank de-watering systems use flocculant-aided settling. De-watering/mixing systems may be designed as batch or continuous systems. Batch systems tend to allow better control of paste quality, particularly where variation in tailings is an issue. A schematic of a generic batch paste preparation system is depicted in Figure 2.3. In batch systems de-watered tailings, binder, and, if necessary, water are individually weighed and combined. The paste recipe is designed to meet the target backfill criteria, based on the properties of the tailings material determined in characterisation work. Process control systems based on relationships, such as slump:power draw or slump:moisture content, which are pre-determined through characterisation test work, are used to control the consistency of paste produced.

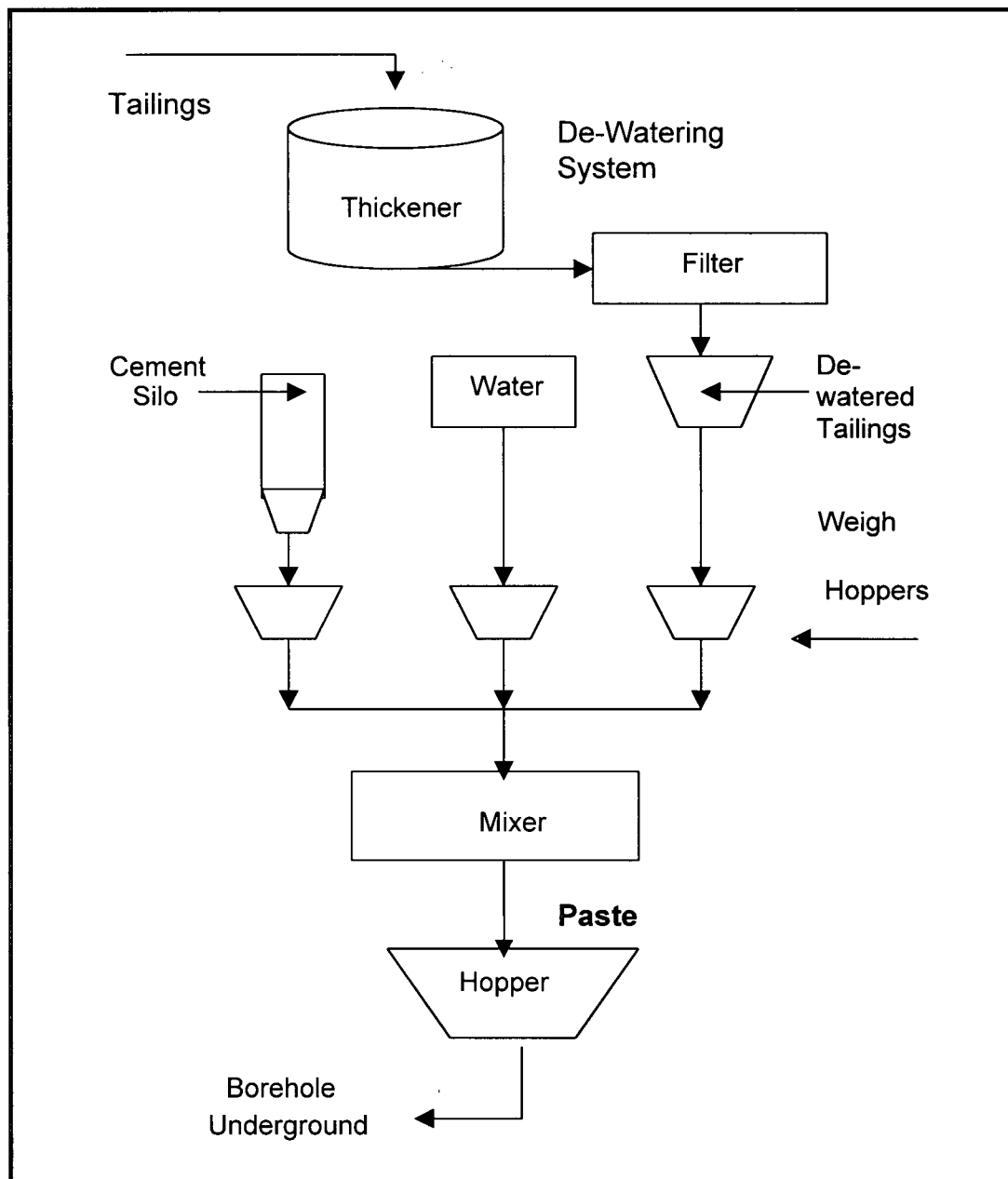


Figure 2.3 Generic Batch Paste Preparation System

Design of the paste distribution system also relies on data determined in characterisation test work. The distribution system design is based on meeting the logistical requirements of the given mining system within the rheological parameters of the paste. Again, characterisation test work establishes the boundaries for pipeline transportation -- gravity or pumped flow -- of a given

paste and provides the basis for pipeline materials selection. Instrumentation and monitoring of the reticulation system are also key elements of design for the distribution of paste.

Design for paste placement is primarily related to stope preparation. Bulkhead design and installation for paste backfill is one facet of backfill design, which is less complex than for other backfill types such as slurry backfills. As well, there is minimal drainage associated with paste backfill compared to slurry backfill systems.

The final element of backfill system design is monitoring of the system and the backfill quality and performance. Following the evaluation, design, construction, and implementation of a backfill system, on-going monitoring of the backfill and the operational components of the system should be established to ensure that the target properties of the backfill continue to be met. Monitoring and evaluation of in-situ quality and performance of the backfill provides valuable feedback regarding backfill strength and long-term stability. Monitoring of the backfill system equipment is an essential component of preventative maintenance and safety. Distribution of paste backfill is usually via pipeline transportation. To prevent costly production losses due to pipeline blockages or pipeline failures, paste backfill systems must be designed with adequate instrumentation in place to monitor line pressures. Pipeline wear should be well monitored, particularly at critical points in the distribution system such as in vertical shafts and at elbows, to prevent dangerous and costly pipeline ruptures.

2.2 Paste Backfill Target Design Criteria

Paste backfill design criteria will be based on target backfill properties, which will be dependent on the backfill function, the mining system conditions, and other site-specific factors. Key target design criteria include:

- Geomechanical Properties
- Distribution and Placement Criteria
- Environmental Performance
- Socio-Economic Performance

Where backfill is required for ground support or to provide a working floor, backfill strength is the primary geomechanical property. Backfill strength can be enhanced with the cement or other binding agents. A related geomechanical property of backfill is liquefaction potential, which is dependent on physical and mechanical properties of the tailings material. A number of physical, chemical, and mechanical properties of tailings affect the rheological behavior of paste backfills and, in turn, affect the geomechanical properties of the paste.

Distribution and placement criteria including system capacities and scheduling are based on the requirements of the mining system as well as the rheological properties of the material.

Environmental considerations have played a growing role in the determination of backfill target properties in recent years. New mines and existing operations face increasing pressure to reduce and limit surface waste disposal of tailings. Target backfill functions and design criteria are critical to improving underground environmental health and safety working conditions, particularly as mining companies investigate deep and ultra deep mining systems, also affects target backfill functions and design criteria.

Inevitably socio-economic performance of the backfill influences backfill design. As a primary resource, mining has a significant effect on the local, regional,

national, and global economy in terms of employment and income. Waste management is a serious issue for the mining industry and it is important to consider paste backfill systems for their potential to handle significant volumes of tailings. The advancement of technology that contributes to the sustainability of the environment will serve to enhance the continued economic viability of the mining industry. However, even though the technology may be available to meet the required geomechanical and environmental criteria and the logistical parameters for transportation and placement, the backfill system design will go no further if the cost is too high. The sophistication of paste backfill systems contributes to relatively high capital costs, but it is important that some of the less tangible cost benefits such as those related to environmental factors and potential increased mining recoveries be accurately factored into trade-off studies comparing paste backfill to alternative backfill methods.

Paste backfill design criteria are based on the target properties determined from the required backfill function, mining system criteria, and site-specific requirements. The target design criteria important to paste backfill design are discussed in the following sections in this chapter and illustrations of paste backfill design in practice are presented in chapter 3.

2.2.1 Geomechanical Properties

The primary geomechanical target property is backfill strength. In addition to backfill strength, liquefaction potential and rheological behavior are key geomechanical considerations for paste backfills. Many inter-related physical, chemical and mechanical characteristics of tailings material influence the geomechanical target properties of paste backfill.

Backfill permeability is a basic geomechanical property that is chiefly of concern with hydraulic slurry backfills. Permeability is an important property of hydraulic backfills because of the large volume of free water associated with them. The

drainage of this free water from placed backfill is dependent on the permeability of the fill. The consolidation and ultimate strength of hydraulic slurry backfill depends on the timely drainage of the backfill. Liquefaction potential is of greatest concern for unconsolidated, undrained backfills. With low permeability paste backfills, there is little drainage of free water. Concern for liquefaction potential in saturated paste backfills has been raised and this topic is discussed further under the heading "Liquefaction Potential" later in this section.

Strength

The strength of a paste backfill, as with any other backfill type, is a measure of its ability to provide adequate regional and local ground support and to bear loads. The importance of backfill in providing local and regional ground support has been discussed in section 1.2.1.

As a target backfill property, the required strength of a paste fill will depend on its intended function and site-specific factors pertaining to rock mass quality. If the function of the paste fill is to provide a working floor, as in a cyclical mining method, the curing time must be short and the fill must provide early strength to support personnel and mechanized equipment. For delayed type backfilling, the paste fill must achieve and maintain longer term stability and be capable of providing a freestanding wall to enable pillar recovery and the mining of secondary stopes with minimal dilution.

Backfill strength can be greatly enhanced by the addition of binding agents. The use of cemented backfills has facilitated the development of bulk mining methods, which have increased mining recoveries and improved safety and working conditions in underground mines.

Cement and Alternative Binding Agents

Conventional hydraulic slurry backfills and paste backfills generally require the addition of a binding agent, in order to provide adequate strength for ground support. Where the function of paste fill is strictly for underground waste disposal, the addition of a small amount of binder is always recommended to reduce the liquefaction potential due to dynamic or cyclical loading caused by nearby blasting and seismic events (Landriault et al, 1998). Binder addition enhances liquefaction resistance by providing increased cohesion of particles.

The most common binding agent used in backfills is Portland cement. Portland cement, containing lime, iron, silica, and alumina components, sets and hardens in hydration reactions. The five types of Canadian Standards Association (CSA) Portland cements are shown in Table 2.1.

Table 2.1 Portland Cement Types

1)	Type 10	Normal Portland cement	General purpose cement
2)	Type 20	Moderate Portland cement	Used for precaution against moderate sulfate attack
3)	Type 30	High-early-strength Portland cement	High strengths usually within 7 days
4)	Type 40	Low-heat of hydration Portland cement	Develops strength slower than No. 10 where heat generation must be minimized
5)	Type 50	Sulfate-resistant Portland cement	Used where concrete exposed to severe sulfate action

In addition to Portland cements, there are other cementitious materials, which can be used in combination with Portland cement or in some cases to replace cement to achieve required backfill strengths. These materials include natural pozzolans, fly ash, ground granulated blast-furnace slag, silica fume, and ground glass. Pozzolans are siliceous or aluminosiliceous materials that will, in the presence of water and lime, react to form cementitious compounds. As cement is a high cost component of backfills and pozzolanic materials, for the most part,

are available as by-products, there is often a significant cost advantage to using supplementary cementing materials in backfills.

The three groups of supplementary cementing materials recognized by the CSA Standard A23.5 (Kosmatka et al., 1995) are shown in Table 2.2.

Table 2.2 Supplementary Cementing Materials

Pozzolans	
Type N	Raw or calcined – occur naturally – includes volcanic glass, diatomaceous earth, opaline cherts, shales, tuff, pumicites
Type F	Fly ash produced by burning pulverized anthracite or bituminous coal
Type C	Fly ash produced by burning pulverized lignite or sub-bituminous coal
Granulated Slags	
Type G	Ground granulated blast-furnace slag – a glassy material formed when molten slag is cooled rapidly – has little or no cementitious properties without an activator
Type H	Cementitious hydraulic slag – similar to Type G, however, displays some hydraulic activity when mixed with water alone
Silica Fume	
Type U	Produced from silicon or ferro-silicon alloys containing at least 75% silicon

There is a wealth of information on the effects of cements and supplementary cementing materials on concretes including water requirement, air entrainment, workability, segregation, heat of hydration, setting time, curing, pumpability, shrinkage, carbonation, strength, and sulfate attack of concretes. Depending on the type and source of fly ash, slag or silica fume, the effects on various properties of cemented materials can be markedly different. Based on the knowledge of the effects of supplementary cementing materials on concretes, it is essential that mines considering the use of alternative binders fully investigate the ability of the final backfill product to meet the mine's target backfill properties.

An important consideration in the use of supplementary cementing materials as binding agents like fly ash and blast-furnace slags is their effect on the strength gain of the backfill. Pozzolanic materials typically retard the setting time of cement and increase the curing time. Therefore, a backfill made with pozzolanic binders may not develop the same early strengths as a backfill made with normal Portland cement. Higher long term backfill strengths, however, can often be achieved with supplementary cementing materials than with straight cement addition.

Commencing in 1933, backfill experiments were conducted at Noranda's Horne mine using a backfill composed of de-slimed pyrrhotite tailings (3%-10%) and granulated reveratory furnace slag (90%-97%). Backfill was required for pillar recovery in areas where open stoping mining methods were employed and for cut and fill stopes. Between 1938 and 1952 13,500,000 tons of backfill were placed underground, with reportedly negligible dilution or caving of exposed free-standing fill walls (Patton, 1952). Based on the success of the original Horne mine backfill recipe, Noranda investigated the use of a slag/pyrrhotite tailings backfill mixture for the Chadborne project in the early 1980's (Nantel and Lecuyer, 1983). Their test work indicated that a backfill exhibiting similar strength characteristics to the Horne mine backfill could be produced, with cementation of the slag particles dependent on the pyrrhotite tailings content.

Landriault et al. (1998) have shown the general behavior of different binders – Portland cement, iron blast furnace slag, and fly ash – on backfill strength and curing time. They reported that siliceous tailings pastes using a Portland cement binder will achieve 65% of ultimate strength in 7 days and 95% of ultimate strength in 28 days. Fly ash mixture siliceous tailings pastes will reach the same strength as straight Portland cement backfills by 28 days, but that strength is only 60% of the ultimate strength that the fly ash mixtures will produce as shown in Figure 2.4. They also found that iron blast furnace slags, without the addition of cement or lime as a catalyst will not generate backfill strength for 14 days, but

will produce an ultimate strength much greater than Portland cement or fly ash backfills over time.

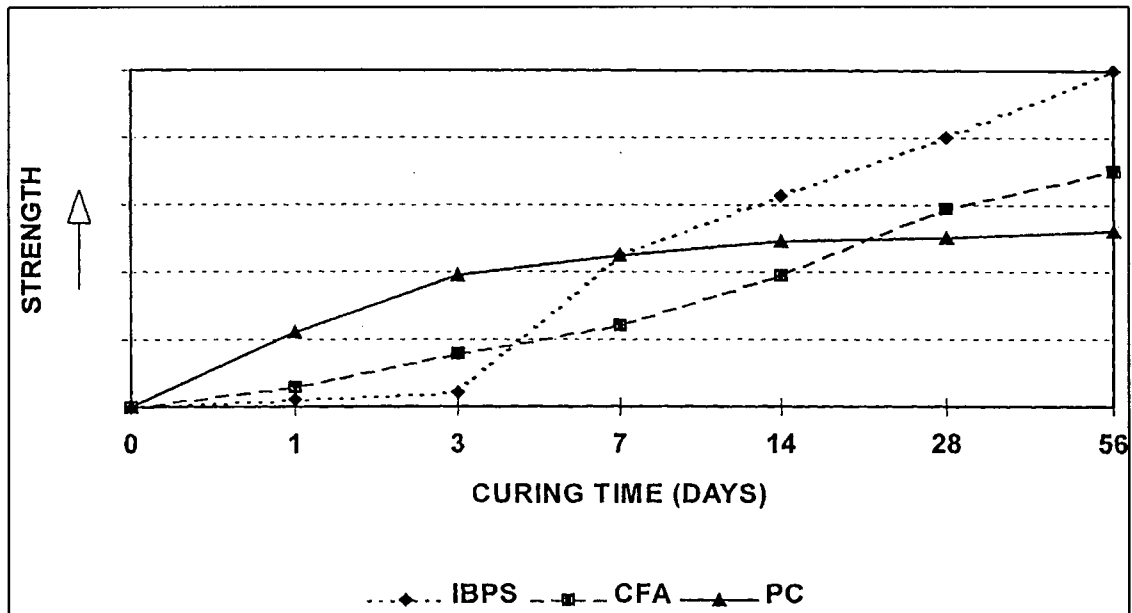


Figure 2.4 Siliceous Tailings Paste Backfill Strength with Different Binders (After Golder Paste Technology, 1997)

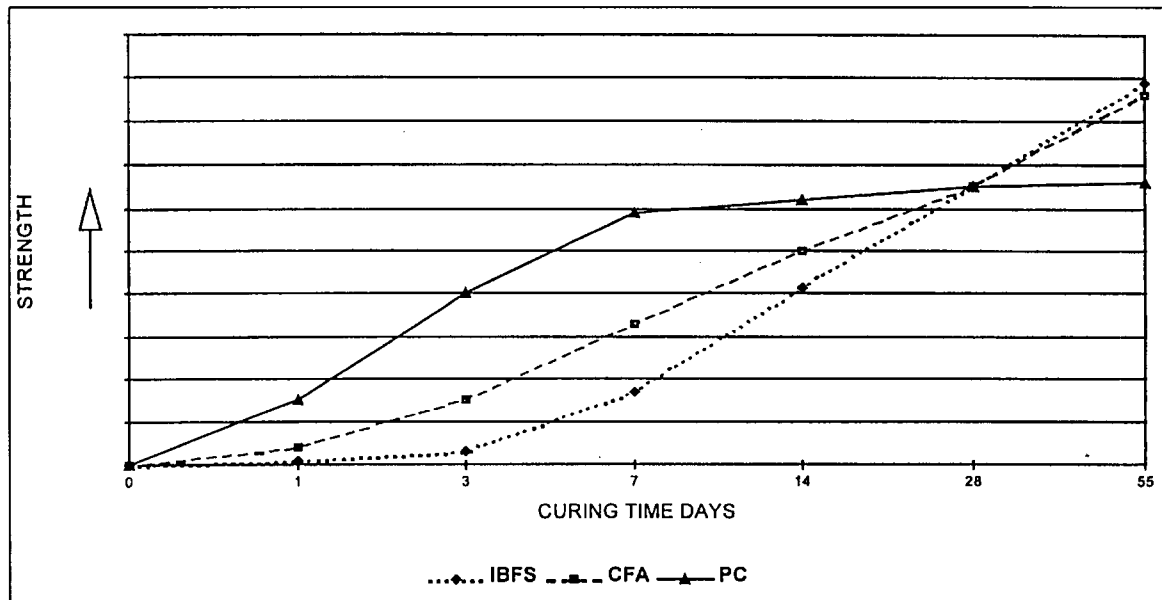


Figure 2.5 Pyritic Tailings Paste Backfill Strengths with Different Binders (After Golder Paste Technology, 1997)

A significant factor that influences the effect of curing time and backfill strength of alternative binders is the mineralogy of the tailings material. Pyritic tailings paste backfills with fly ash binders will produce higher early and long term strengths than siliceous tailings paste backfills, while iron blast furnace slags in pyritic tailings pastes produce even higher early and long term strengths as shown in Figure 2.5.

Research conducted in Canada using ground waste glass/Portland cement mixtures has demonstrated that equivalent or higher strengths compared to straight Portland cement binder addition can be achieved in paste backfills, with significant cost savings (Archibald, Chew, and Lausch, 1998). Strength characterisation test work on slurry backfills showed that partial replacement of Portland cement binder with ground waste glass yielded equivalent or higher strengths for curing times up to 224 days. Subsequent strength characterisation test work was conducted using paste backfills from four Canadian mines. The paste strength characterisation tests showed that early strengths, up to 28 days, were all lower for ground waste glass/Portland cement mixtures as opposed to straight Portland cement binders. For longer curing times, generally over 56 days, different blended ground glass/Portland cement mixtures, proved stronger than straight Portland cement. Maximum strength performance and optimum ground glass blend varied significantly depending on the specific tailings used.

Still another type of binder that has been introduced in recent years is silicate based activating and gelling agents. These binders are combined with ordinary Portland cement and/or slags and have been reported to produce high, early strengths and long-term chemical stability. Silicated gel binders tie up significant volumes of water, reducing the amount of water draining from slurry backfills. Silicated backfills have been used with both classified and total tailings.

Operational advantages offered by the use of silicated fill systems with hydraulic slurry backfills at 65 - 68% solids, which have been tested in South African mines

include: reduced pipe wear due to higher slurry relative density; reduced pipe wear due to binder; reduced binder cost; and improved fill performance (Spearing, Harrison, and Smart, 1993). One disadvantage to the use of silicated binders is that the activator has to be mixed into the slurry just before placement in the stope to prevent hardening or "setting up" of the binder in the distribution pipeline. This necessitates a separate distribution line to the individual stopes and equipment for introducing and mixing the binder with the slurry.

A similar type of binder, SUNGERIC (SBB), has been developed in China, based on a high water binder developed there in the early 1990's. The SBB has been reported to reach 50% to 60% of its 28-day strength within one day. The SBB binder was tested in Canada at Inco in October 1997 with a hydraulic sand fill (Sun, Baiden, and Grossi, 1998). In the Inco test 450 tonnes of backfill was batch mixed and then poured 40 metres down a borehole to a test stope. The backfill consisted of 75% solids hydraulic sand fill with a 35:1 sand:binder ratio. The first round was blasted 24 hours after filling was completed with negligible dilution reported.

One topic of considerable importance in backfills is the long term strength of backfill in the presence of reactive chemicals, in particular sulfates. Resistance to sulfate attack on concrete is well documented. Mangat and Khatib (1995) reported on investigations of the influence of fly ash, silica fume and slag to sulfate resistance in concrete. They found that replacement of Portland cement with 22% to 32% fly ash, 5% to 15% silica fume, or 80% blast furnace slag improved the sulfate resistance of the concrete.

Investigations of sulfate reactivity available from the concrete industry provide a useful basis for predicting potential effects of sulfate attack in cemented paste backfills. Where the function of paste fill is for delayed backfill methods, the potential loss of backfill strength due to sulfate attack can have serious implications. Mineralogy and the potential geochemical reactivity of paste backfill

should always be evaluated in tailings characterisation work prior to backfill design. The chemical reactivity and geomechanical response is material specific and should always be considered in strength characterisation test programs for backfill, particularly for sulfide tailings. The topic of sulfate attack in paste backfills is discussed further in Section 4.7.1.

Liquefaction Potential

Liquefaction is described by Poulos (1985) as a phenomenon wherein the shear resistance of a soil mass decreases when subjected to monotonic, cyclic or dynamic loading at constant volume, which causes the mass to undergo large unidirectional shear strains. In other words, the soil mass fluidizes with the potential to build pressures that are considerably higher than the normal hydrostatic pressure due to the mass of water alone. Saturated soils are composed of cohesionless, granular particles, making them vulnerable to liquefaction.

Consolidation of soils occurs when the particles move closer together or become more tightly packed as water drains from the soil mass. The weight of the soil and any applied load make up the total stress on the soil. The pressure or load carried by the soil particles in contact with each other is the effective stress, while the pore pressure is the pressure of the water in the voids. The total stress minus the pore pressure is the effective stress. The rate at which the pore water pressure dissipates will be dependent on the permeability of the soil. If the soil has poor permeability, when pore water pressure increases, the shear strength of the soil will decrease, along with the effective stress.

Tailings material can be considered to be a soil mass under certain conditions. If the tailings material is fully saturated, unconsolidated, uncemented or otherwise cohesionless, and is subjected to dynamic loading, it may have the potential to liquefy. Seismic loading may have the potential to trigger liquefaction.

There has been considerable research and evaluation of liquefaction potential in backfills. While hydraulic slurry backfills may be liquefaction prone immediately after placement, they have not been shown to have high liquefaction potential, particularly when cemented. Because a key criteria for hydraulic slurry backfills is high permeability, they tend to drain rapidly, reducing the risk of build up of excess pore pressure.

Cementation increases the cohesion of particles in backfill, increasing the shear strength. Clough et al (1989) showed that saturated cemented sands subjected to seismic loading can liquefy, but have greater resistance to liquefaction than uncemented sands. Their study also demonstrated that nonhomogeneous cementation of sands decreases the resistance to liquefaction. The resistance of the more weakly cemented layers will dominate. Because hydraulic slurry backfills generally have high permeability to drain free water, they consolidate rapidly and can be considered to be unsaturated soon after placement.

Concern has been raised for paste backfills, however, because they remain in a highly saturated state. A limited number of laboratory and in situ studies have been conducted on the liquefaction potential of paste backfills.

A comprehensive test program on the liquefaction potential of a weakly cemented paste backfill was undertaken at the Dome Mine, in Timmins, Ontario (Aref, Hassani, and Churcher, 1989). This test work formed part of a major project on liquefaction potential for high density backfills conducted by Placer Dome Inc. (DSS File No. 15SQ23440-6-9039, 1991).

The test work and analysis was based on the concept of steady state of deformation (Poulos, Castro, and France, 1985), whereby dilative soils are not considered to be susceptible to liquefaction because their undrained strength is higher than their drained strength. This concept is dependent on a critical void ratio, which is the void ratio at which the volume of the material does not change

when sheared. The undrained steady state strength is sensitive to the void ratio, which can be very difficult to accurately determine, particularly the in situ void ratio. The liquefaction potential becomes a stability analysis with the factor of safety against liquefaction (F_L) determined by:

$$F_L = \frac{\text{Undrained steady state shear strength}}{\text{Driving shear stress}}$$

They further state that only contractive soils can be susceptible to liquefaction and even then, only if the driving shear forces are high enough.

For the Dome Mine study both laboratory and in situ testing were undertaken. Undisturbed samples were recovered from backfilled stopes using a continuous flight auger device with a modified split barrel sampler with tungsten carbide cutting edges. Paste backfill samples, both undisturbed and reconstituted, containing 1:30, 1:40, and 1:50 cement:tailings ratios were tested in a series of consolidated undrained triaxial tests at confining pressures varying from 34.5 kPa to 345 kPa. The test procedure used back pressure to ensure that the samples were fully saturated. The consolidation pressures used were typical values measured in underground stopes at the Dome Mine. The authors observed initial pore water pressure increases at low strain followed by dilation of the samples leading to decreased pore water pressures. They observed that as strain increased, the pore pressures dropped sharply, generally becoming negative. At a high confining pressure, 345 kPa, a high positive pore pressure was generated.

Different cement:tailings ratios (1:30, 1:40, 1:50) produced similar peak strength plots with similar internal angles of friction and different cohesion intercepts. The angle of friction varied from 31° for the 1:30 cement:tailings sample to 33° for the 1:40 and 1:50 cement:tailings ratio samples. Higher binder additions produced greater cohesion, with cohesion values of 53 kPa for the 1:30 cement:tailings samples and 20 kPa for the 1:40 and 1:50 cement:tailings samples.

The in situ testing consisted of piezometer cone penetration tests to measure and evaluate geotechnical parameters including relative density, angle of internal friction, and dynamic pore water pressure. The piezometer cone testing results all indicated that the paste backfill material was unsaturated with dilative behavior. The average in situ void ratio was 0.93.

In conjunction with the Dome liquefaction test program, blast tests were conducted in the area adjacent to the filled stope, in order to investigate the paste backfill behavior under dynamic loading conditions (Mohanty, 1988). Triaxial accelerometers were used to measure the energy transfer, peak particle velocity and peak particle acceleration through the rock and paste backfill. Amplitude and frequency attenuation was noted in the paste backfill, with typical accelerations of 1.0 g measured inside the fill and a maximum acceleration of 2.3 g measured just inside the backfill. While the duration of the detonation was 1 msec, the duration of the vibrations within the backfill was 60 msec.

The analysis of test results from the Dome Mine test project led the authors to conclude that liquefaction is unlikely to occur for paste backfill with a cement:tailing ratio of 1:30 and a void ratio up to 0.96. They also concluded that liquefaction resistance can be improved by increasing the cement content of paste backfill.

Ouellet, Bidwell, and Servant (1998) conducted laboratory and in situ test work on paste backfills from four mining operations in Quebec. The laboratory testing consisted of triaxial testing on prepared samples with cement ratios of 3.0% to 6.5% by dry weight. They used the test protocols of a consolidated drained test. The in situ void ratios of the typical paste backfill studied varied from 0.8 to 1.05. They observed contractive behavior during shear for all the backfills in the study.

The in situ testing was conducted using a Cambridge self-boring pressuremeter. Pore water pressure and total applied pressure were measured with time in order

to determine the degree of saturation of the in situ sample. They determined that if under shearing the total pressure and the pore water pressure increased in phase, then the backfill was saturated, while if the pore water pressure did not increase in phase with the total pressure, then the material was not saturated. The results indicated that some of the backfill tested was saturated. The measurement of water content on samples taken from backfilled stopes confirmed high saturation levels.

They determined that the load history of the backfill affects the apparent cohesion. For uncemented samples there is no cohesion, only a frictional force. For cemented samples as the pressure increases, the cement bonds are progressively destroyed. They concluded that once the backfill has become destructured by loading from rock mass convergence or blast induced damage, it will lose its cohesion and behave like an uncemented fill. In that event, if the backfill has no effective cohesion and is saturated, they concluded there could be potential for liquefaction.

The Golden Giant Mine at Hemlo, Ontario switched from using cemented rockfill to paste fill in July 1996. A study involving laboratory testing and stability analysis of paste fill (Pierce, Bawden, and Paynter, 1998) was conducted to facilitate the optimization of the paste fill design. Paste fill samples were tested in confined, unconfined and consolidated undrained triaxial compression. Confining pressures for the consolidated undrained triaxial tests varied from 103.5 kPa to 414 kPa. Positive pore pressures were observed during shear in the triaxial tests, hence, the undrained shear strength was less than the drained shear strength. The minimum 28 day uniaxial compressive strength obtained for the Golden Giant paste fill was 213 kPa. The authors concluded that liquefaction of material with this strength was unlikely. They recommended that cyclic triaxial tests could be conducted to determine the in situ strength required to prevent liquefaction under cyclic loading conditions.

In each of the three test programs reported above, the final conclusion reached was that the cemented paste backfills were unlikely to exhibit liquefaction. As the test protocols varied in the three test programs, they should not be directly compared. The only concern expressed appears to be for the short time immediately after backfill placement, prior to when any cementation/consolidation has occurred, which gives the backfill enough cohesion to provide liquefaction resistance.

Echo Bay Mines Ltd. commissioned a paste backfill plant in the fall of 1994 at its Lupin gold mine near Contwoyto Lake, N.W.T. Paste backfill was successfully used until the mine closure in 1997. In conjunction with the use of paste backfill, a sub-level retreat over/under consolidated fill mining method was used to limit dilution and minimize mining costs. The mining method involved opening slot raises directly in the paste backfill and the mine reported (Sandhu, 1996) that they experienced no problems with liquefaction under induced cyclic vibrations from drilling and blasting.

Liquefaction resistance was also investigated on silicated slurry backfills, which have a relatively high water content, to determine the potential effect of sudden closure due to seismic activity in deep South African gold mines (Spearing, Harrison, and Smart, 1998). Dynamic triaxial loading tests were conducted on silicated samples and uncemented classified tailings samples. Neither specimen showed liquefaction potential.

2.2.2 Distribution and Placement Criteria

The logistics of backfilling, which encompasses many facets of preparation, transportation, and placement, should be an important consideration starting with the initial stages of backfill system design. Target logistical properties, which will influence the subsequent paste backfill system design, should be identified early in the design process and require input from geology, mine planning, and mine

operations departments to ensure that all future backfill requirements can be accommodated. Scheduling of distribution and placement of backfill can be a crucial part of meeting mine production targets.

The overall mining system determines the required volumes and rates of backfilling. Depending on the backfill function, filling must accommodate delayed or cyclic backfill scheduling. Required strengths and curing times must meet design targets in order to prevent delays in gaining access to mining production areas. The logistics of distribution depends on ore body dimensions, depth, access, mine workings, and location of surface infrastructure.

Time must be scheduled for the construction of barricades and the installation of fill lines and instrumentation. Pouring time as well as waiting time for curing, for example, backfill plugs, must also be factored into backfill scheduling to meet mine plan requirements.

The unique rheological behavior of paste backfills, which is discussed in more detail in the following section, has a significant effect on paste mixing, distribution, and placement. The rheological properties of paste fill determine the flow resistance and pipeline pressure losses. The pipeline pressures, pressure gradients, and required capacities affect the engineering design of paste backfill pipelines. Paste fills exhibit higher pipeline pressures than conventional slurries and consequently, heavier pipe and stronger support for pipelines is generally required. Safety and pipeline wear considerations are other areas of concern that must be addressed in paste fill transportation system design. Where pumping is necessary, high capital cost positive displacement pumps or double piston concrete-type pumps are required and the rheological properties of paste will greatly affect the pumping capacities.

Rheological Behavior

Rheology is the study of the deformation and flow of matter. Paste backfills are homogenous mixtures of granular particles with just enough water to fill the voids between the particles. The fine particles give pastes their unique rheological properties. The colloidal electrical charge on the fine particles forms bonds with water molecules. Typically a paste must have a minimum fines content of 15% minus 20-micron particles in order to exhibit these water retentive properties.

A typical Newtonian fluid, such as water, oil, or conventional hydraulic slurry, exhibits a critical flow velocity, below which particles will settle out. High volumetric concentration paste fills, however, exhibit complex non-Newtonian flow properties. Paste backfills produce a plug flow whereby a portion of the fines create an lubricating annulus inside the pipe wall and act as a carrier fluid to support the flow of the balance of the material. A profile for plug flow of a Bingham type paste is shown in Figure 2.6. The plug moves at constant velocity

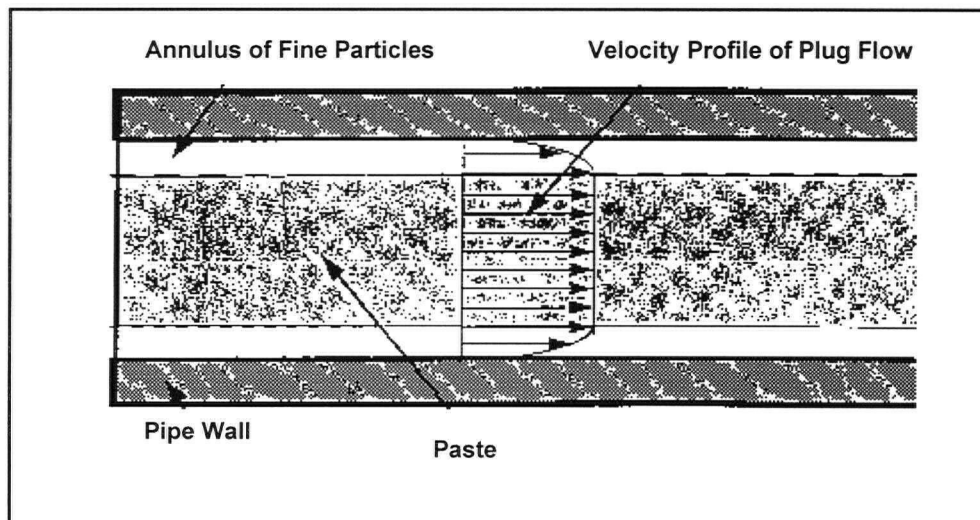


Figure 2.6 Velocity Profile – Plug Flow of Paste (after Landriault, 1995)

while the velocity profile surrounding the plug will depend on the viscosity and shear stress of the annular material.

Viscosity is a resistance to flow. Newtonian fluids show constant viscosity, independent of shear rate or the type of flow. Figure 2.7 shows rheological models for Newtonian and non-Newtonian fluids. These rheograms plot shear stress versus shear rate.

Bingham plastics, sometimes termed viscoplastic fluids, exhibit a resistance to flow up to a minimum shear stress or yield stress (τ). In his study of rheological properties of gold slimes, Horsley (1982) found a major reduction in pressure gradients could be obtained by using phosphate additives. He used sodium tripolyphosphate to effectively reduce the yield stress to a negligible value. In order for the paste to regain its necessary strength and energy absorption characteristics, the effects of the phosphate additives would have to be neutralized prior to placement. In their

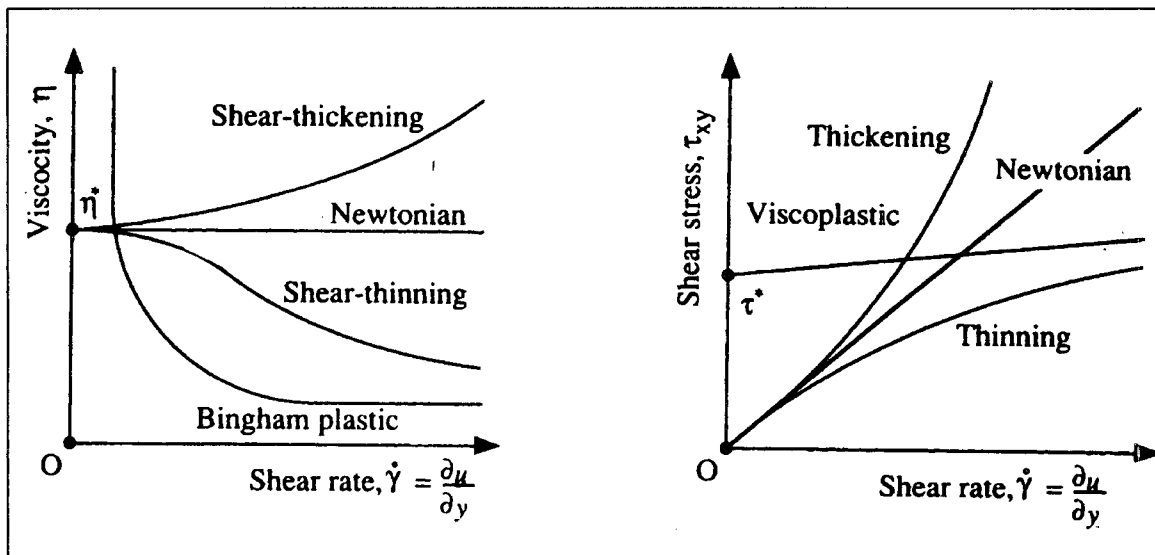


Figure 2.7 Viscosity and Shear Stress as Functions of Shear Rate
(After Papanastasiou, 1994)

discussion of the transport and rheological characteristics of paste backfills, Verkerk and Marcus (1988) noted that the addition of phosphate additives would not necessarily cause the break down of yield stress in all slimes.

With non-Newtonian fluids, the viscosity will depend on the shear rate. For shear thinning fluids, viscosity will decrease with increasing shear rate. Thixotropy is a phenomenon whereby the internal structure of the fluid gradually breaks down over time, resulting in a decrease in the pressure gradient at constant flow rate or an increase in flow rate at constant pressure gradient (Slatter and Lazarus, 1988). They reported that gold slimes slurries exhibited thixotropy. Depending on the transportation distance (time), this phenomenon may affect the pressure gradients in pastes.

Rheological properties of paste influence the engineering design of tank de-watering systems, affecting retention time, rate of paste formation, and underflow discharge systems.

Rheological behavior of paste is also an important consideration for potential surface deposition of pastes (Cincilla, Cuning, and Van Zyl, 1998). They have theorized that under a surface paste deposition scenario the rate of deposition can be greater than that of the currently best managed subaerial systems, because the consolidation process, which slurried tailings must undergo through settling and drainage, has been accelerated through mechanical de-watering.

The rheological behavior of paste is highly dependent on particle size distribution, in particular, the minus 20-micron fraction. However, other physical and chemical properties of the paste material affect the flow resistance. Chemical content and mineralogy have also been shown to have a significant effect on paste rheology. Higher flow resistances have been demonstrated by pyritic tailings paste fills as opposed to siliceous tailings pastes (Landriault et al., 1998).

The complex flow properties exhibited by paste backfills have a significant effect on paste target geomechanical properties and paste system logistical design criteria. Optimization of design requires a systems approach. The design goals

may conflict. For example, in the selection of the optimum paste backfill recipe for strength, changes in binder addition and percent solids will influence the slump or rheological behavior of the paste, which will in turn affect the pipeline pressure losses in distribution. In general, the effect of rheological behavior of the paste on backfill strength and on transportation/placement must be considered jointly in paste backfill system design because many properties of the tailings are interrelated.

2.2.3 Environmental Performance

In terms of environmental management, the mining industry has undergone a revolution in the past quarter century. Protection of the environment is at the forefront of all mining related activities, from initial exploration stages, through development and construction of mines, operations, reclamation, and long after mine closure. At all stages of mining, an effort is required to limit the footprint mining leaves on the environment. Mining generates an enormous volume of waste products including process tailings.

As a means of waste management, waste disposal is one of the primary functions of backfill. In the past, mines often placed waste rock underground for both support and waste disposal. The use of hydraulic slurry backfills has helped to decrease the volume of tailings on surface. Classified tailings backfills, however, use only the coarse fraction of tailings, leaving the slimes fraction for surface disposal, which from a perspective of waste handling is more challenging. In addition, the large volumes of water used to transport the hydraulic slurry backfills underground must be collected and returned to surface. As mineral processing technology has evolved, finer grinds have been used to increase metal recoveries. Consequently, the coarse fraction of tailings may not be sufficient for required backfill volumes.

With the development of paste backfills, it is now possible to use full plant tailings, eliminating a large portion of slimes disposal on surface and increasing the overall percentage of total tailings that can be disposed of underground and minimizing the surface disposal area required for tailings. As well, paste backfills eliminate the handling of excessive amounts of drainage water.

As a method of surface deposition of tailings, paste shows great promise. Subaerial or dry land deposition of process slimes, including bauxite red muds in Australia and uranium tailings in South Africa, has been successfully employed for many years. A growing understanding of the rheological behavior of pastes in relation to geotechnical properties required for surface deposition has shown that surface deposition of paste may be a viable option for tailings disposal.

Many existing operations, producing long beyond their original mine life, are faced with limitations of available surface tailings disposal area. New mines must meet increasingly more stringent environmental regulation. Process tailings are of particular concern for their potential to contaminate ground water through dissolution of heavy metals and acid generation. As a result, even in underground mining scenarios where backfill is not required for support functions, the use of paste backfill strictly for waste disposal has become more feasible.

Backfill also serves a beneficial function in underground environmental control by improving the efficiency of ventilation and by reducing stope heat load. Improved ventilation and temperature control results in better working conditions for miners. This subject has been of particular interest to South African mine operators. An investigation of the implications of backfilling on underground environmental control (Matthews, 1988) showed that heat load can be halved with complete backfilling of mined out areas. He also reported that theoretical models have shown that chilling of backfill prior to placement may be beneficial in reducing heat load. It was shown that the heat produced from hydration of

cemented backfill had a minimal effect on overall heat load. Simulations of a longwall mining ventilation system showed that backfilling leads to higher air velocity at the face, leading to increased cooling and better distribution of air within the stope. However, when the backfill is kept very close to the face, at a distance less than 5 metres, there may be diminished air flow and hence higher temperatures at the face. Matthews also noted that with the increased use of backfill, less timber support is generally necessary, reducing the fire hazard. As well, the presence of backfill can provide containment of fire.

Another potential environmental effect of backfill is the containment of radiation. Radon and thoron flow from stope surfaces is inhibited by backfill. As well, backfill seals off contaminated areas, preventing the circulation of contaminated air (Bates, 1992).

2.2.4 Socio-Economic Performance

Mining is a business. Few businesses can be sustained if they consistently fail to turn a profit. Backfill often accounts for a significant percentage of mining costs. Given these harsh realities, there can be very few mining operations for which cost targets can be neglected.

Target economic properties are based on the ability of a paste backfill to meet the other target properties – geomechanical, logistical and environmental – at justifiable cost. Paste backfill must be found to be a feasible alternative to more conventional backfill types such as rockfill or hydraulic slurry backfills.

There are, however, site specific factors which must be taken into account in conjunction with the cost of meeting other target paste backfill properties. For example, mine location can have a significant influence on cost. Where mines are remotely located, distant from supply sources, transportation costs may be prohibitively high. Cement is a notably high cost component of backfills. The

cost benefit of using paste backfill may be a reduction in cement usage as equivalent backfill strengths can often be achieved with less cement than for hydraulic slurry backfills.

Another high cost component of many remotely located mines is power. Where cheaper hydro-electric power is not available, mines frequently rely on diesel generated power. Power costs may, for example, influence the type of de-watering technology selected for producing paste backfill. The design of paste backfill systems using newer tank de-watering methods rather than filters, which are high power consumers, may be considered for remote mines.

Backfill also places a role in the socio-economic impact of mining, which is closely associated with the environmental and mine permitting process. Resource industries are the basic building blocks of an economy, upon which secondary and tertiary industries are developed. Mining is a key primary resource. Mining traditionally provides high paying jobs. Through both income and employment multipliers, mines make a significant contribution to the local, regional, and national economies. Through the many functions backfill serves, it contributes to the existence and continued viability of mines.

2.2.5 Summary

This chapter has presented a paste backfill system design rationale and identified key target design criteria for paste backfill systems, based on the required backfill functions. Four primary categories of target design criteria have been defined – geomechanical properties, distribution and placement, environmental performance, and socio-economic performance.

Two key geomechanical properties are paste backfill strength and liquefaction potential. These properties are influenced by binder addition. Paste rheological properties also affect geomechanical properties.

Distribution and placement target properties are influenced by the unique rheological behavior of paste backfills. Paste rheology has a significant effect on de-watering, transportation, placement and stability of paste fills. The importance of backfill scheduling as a logistical target property cannot be overstated. The production and placement of paste backfill must be capable of meshing with rest of the mining system. Paste production volumes and delivery capacities must be sufficient to meet the mine plan requirements and be flexible enough to accommodate hiccups in the mining system.

Increasingly, environmental considerations and waste management practices are dictating target backfill properties. As a waste disposal alternative, paste backfill maximizes the volume of full plant tailings that can be placed underground and hence, reduces the size of the tailings footprint left on surface. Surface deposition of tailings pastes is a promising alternative to conventional surface tailings disposal. Because paste backfills give tighter fills than alternative backfill types in underground voids, they improve the health and safety working conditions attributed to backfilling – more efficient ventilation, reduced heat load, and reduction of fire hazard.

Paste backfill, like any other alternative backfill method must meet capital and operating cost targets. Paste backfill offers many potential cost benefits, which should be weighed against alternative backfill types.

3.0

PASTE DESIGN IN PRACTICE – CASE STUDIES

Each mining operation will develop site-specific target criteria for paste backfill design within the general categories of geomechanical properties, distribution and placement criteria, environmental and socio-economic performance. Six mine paste backfill systems have been studied and reviewed. Their specific design target criteria are discussed, along with brief outlines of their paste backfill systems. Based on these case studies of paste design in practice, a number of critical success factors, which affect paste backfill system design, are identified and discussed in section 3.7.

3.1

Louvicourt Mine

The 4,000 tonne per day (tpd) Louvicourt underground base metal – copper, zinc, silver, and gold – mine operated by Aur Resources Inc. commenced commercial production in January 1995. The mine is located near Val d'Or, Quebec within easy access of existing power and transportation infrastructure.

The original mine design, based on geological interpretation from surface drilling, planned for mechanized room and pillar mining with backfill, to mine five sub-vertical lenses that appeared to be stacked en echelon. Backfill was determined to be required for pillar recovery and to provide regional stability. Subsequent definition drilling changed the geological interpretation of the ore body to a single massive, highly folded ore lens with thicknesses up to 60 metres. The ore body is located at depths from 450 metres to 860 metres. Stope dimensions are 15 metres wide, 25 metres high and up to 45 metres long. As a result of the new interpretation of the ore body, the backfill function remained the same, however, it was determined that stiffer fills than had originally been anticipated would be required to provide adequate support for mining of secondary stopes.

While the primary backfill function for the Louvicourt Mine is related to ground support, from an environmental perspective, the disposal of a substantial portion of the sulfide tailings underground, was identified as an added benefit.

Economic justification for a paste fill system was determined through a cost comparison of dilution, stope cycle time and sequencing, \$/tonne operating cost of backfill, and differential capital costs of paste fill versus, rockfill, high density fill and conventional slurry backfill (Lacombe, 1995).

When the decision was made to build a paste backfill system at the Louvicourt Mine, there was no bulk ore sample available for full-scale paste characterisation test work. Therefore, a laboratory scale test program was developed with the assistance of the Chamber of Mines Research Organization of South Africa (COMRO). A 50-kilogram composite sample of flotation tailings from metallurgical test work was used for the Louvicourt test program. Four relationships critical to design were targeted for investigation as follows:

- 1) pressure loss:particle size distribution
- 2) pressure loss:percent solids
- 3) pressure loss:binder addition
- 4) compressive strength:binder addition

Subsequent to the laboratory test program, a pilot-scale flotation test program provided tailings for surface loop testing conducted by Lakefield Research. The tailings material used in the surface loop test program had a higher sulfide content than the original laboratory test sample with an ore specific gravity reported to be 4.1 to 4.2. Due to a finer grind than had originally been planned, the tailings size distribution was reported to be 80% minus 30 micron to 70 micron, as opposed to the original size distribution of 80% minus 55 micron. The minus 10-micron fraction varied from 28% to 35%.

Surface loop tests were conducted at line velocities from 0.5 to 1.3 metres/second in 100 mm and 150 mm diameter steel pipes. Pressure losses

were reported to range from 11 kPa/m to 18 kPa/m. Tests at varying moisture contents indicated a high degree of sensitivity of pressure loss to moisture content. When the paste plant was first commissioned, a closed loop was constructed on surface to fine-tune the system for two months prior to sending the first paste backfill underground. Adjustments were made to the water and binder addition points in the screw mixer and to the angle of the screw mixer to optimize the mixing process.

A feedback loop was originally supposed to be used for controlling the density of the paste. The operators had anticipated that this would be required due to variation in tailings solids produced by variation in sulfide content of the tailings. They discovered that this feedback loop did not perform up to expectations due to difficulty with instrumentation used for density measurements. Subsequently, a feed forward loop was added to improve the water addition adjustments. It was also found that the hydraulic pressure of the actuators for the pistons in the concrete pump was directly proportional to the pressure in the line feeding the borehole. This relationship could be used as a guide to paste consistency.

The design strength criteria was for 0.7 MPa after 7 days curing for the backfill plug (first 4 metres of fill height) with 1.2 MPa strength required after 56 days curing to provide a free-standing wall with minimal dilution for secondary stoping. The mine plan allows for approximately 90 days between primary and secondary stoping. Originally, the binder addition was planned to be a cement:fly ash (60:40) mixture, however, pilot plant test work determined that a ferrous slag mixture would give much higher long-term strengths. In order to ensure rapid curing of the paste plug, 6% by weight pure Portland cement is used for the plug. To obtain long term higher strengths for the remainder of paste backfill, binder addition consists of a 4.5% by weight mixture of ferrous slag: Portland cement, 20:80, which results in 56 days strengths of 1 MPa (Bissonnette, 1996).

In order to meet the target of returning approximately 60% of daily mined tonnage or 2400 tpd underground as backfill and to allow for scheduling inefficiencies, paste production capacity was designed to be 200 tonnes/hour. Another logistical design factor was provision of 14-hour surge capacity to allow for routine SAG mill shutdowns.

Approximately 15% of the stopes were outside the range of a target 3:1 vertical:horizontal distance ratio for gravitational transportation of the paste fill. Consequently, the design included a concrete pump to push paste into the cased boreholes down to the 475-metre level. The pump capacity was designed to provide adequate line pressure for horizontal flows to the stopes, based on preliminary rheological evaluations of the paste. The installation of check valves to allow airflow into the boreholes solved initial problems with severe vibration in the distribution lines.

After mining of secondary stopes commenced at the Louvicourt Mine, problems were encountered with sloughing of paste backfill. The high pyrite content of the Louvicourt tailings had already been targeted as a potential cause of long-term backfill instability. Considerable investigation had been conducted to optimize the paste recipe and binder to continue to meet target backfill strengths and reduce the brittleness of the backfill. To complement the paste backfill strength/binder recipe test work, an investigation of blasting practice was conducted (Liu et al, 1998). Blast tests were carried out in a secondary stope to determine the extent of blast damage and the critical peak particle velocities. Instrumentation was installed to monitor vibrations and borehole cameras were installed to visually monitor damage. The effects of different charges, delays, and blast sequences were also evaluated. Cavity measurement surveys were conducted subsequent to mucking out of the test stope to determine the stope size and resulting sloughage. It was determined that modifications to the blasting practice could be introduced to reduce ground vibration and subsequent blast damage.

3.2 Golden Giant Mine

Battle Mountain Gold Company's Golden Giant Mine commenced production in 1985 and currently processes 3000 tonnes per day. The mine is located within one kilometre of the Trans-Canada Highway and power supply, near Hemlo, Ontario.

The primary function of backfill at Golden Giant is to provide adequate strength in the free-standing stope walls in order to maximize recovery from the high grade ore body. In some areas the backfilled stopes are undercut and must remain stable. The tabular ore body dips from 60° to 70° and varies in thickness from 3 to 40 metres. The mining method is blasthole open stoping with sublevel intervals at 25 and 33 metres and strike lengths of 15 to 20 metres. As mining progresses upward, the backfill acts as the mucking floor for the stope above.

From 1986 until 1994, the mine used cemented rockfill, requiring a relatively high cost quarrying operation. Existing tailings capacity was limited and in an effort to reduce surficial tailings disposal, the mine first developed a high density backfill plant, which operated from 1994 to 1996. Due to the fine grind, 90% minus 74 micron, the high density plant, which relied on cyclone classified tailings, could not provide the required backfill volumes. The paste backfill plant, which was commissioned in July 1996, allowed the mine to delay construction of a new tailings basin by three years and to reduce the overall size of the new tailings basin.

The economic motivation for switching to paste backfill at the Golden Giant Mine was two-fold – to reduce operating costs by shutting down the rock quarrying operations and to reduce the capital cost for construction of a second tailings pond (Dodd and Paynter, 1997).

Plant design criteria were based on an average binder content of 4.5% by weight, using mixtures of Portland cement and fly ash. Laboratory testing and stability modelling were used to determine factors of safety for varying binder content over different stope widths. The average 28-day strengths varied with the percent binder addition from 0.2 MPa for 3% binder to 0.5 MPa for 7% binder, while the average 112-day strengths varied from slightly less than 0.2 MPa for the 3% binder content to 1.2 MPa for 7% binder content (Pierce, Bawden, and Paynter, 1998). The test work also evaluated liquefaction potential, discussed in Section 2.2.1. Generally a stiff, low slump (6") paste is poured for the plug, followed by a less stiff (lower binder addition), low slump paste to a height near the top of the stope where the slump is then increased to approximately 10" to flatten the angle of deposition and form the floor for the stope above.

The mineralogy of the tailings is quartz, pyrite, barite, biotite, and muscovite mica. The average ore specific gravity is 2.86. Average reported fill properties include: a fill bulk density of 2 tonnes/m³, moisture content of 25%, porosity of 50% and 99% saturation. The minus 20-micron fraction of the tailings averages 27%, with a coefficient of uniformity of 13.

The target capacity of paste backfill plant was 125 tonnes/hour in order to supply an average of 2,036 tpd of paste backfill underground. The excess design capacity allows for scheduling flexibility. The distribution system consists of two 8" (200 mm) 350 metre long boreholes from surface, underground boreholes up to 200 metres long and 8" (200 mm) schedule 40 steel pipe. Pressure transducers are used to monitor line pressures in the distribution system.

When the plant was first commissioned in June of 1996 a surface loop was constructed from the discharge of the concrete pump to the tailings pump area and this loop was used for fine-tuning of the paste system. Early problems encountered and solved included leaks in the vacuum lines for the filters and difficulty discharging cake from the filter bags.

An initial test of the underground system was performed using water and air to test all of the components including the borehole casings, pipe connections, and anchor bolts. An underground test stope was then used to refine the pour procedures and to determine some of the important relationships for controlling paste quality. These relationships included mixer power draw:slump, filter disc speed:filter cake tonnage, and filter cake conveyor speed:paste consistency. It was discovered that the power draw on the mixer was not a reliable measure of slump because a build up of paste inside the mixer increases the power draw irrespective of the paste slump. Operators also found that by putting a variable speed drive on the filter cake conveyor, the feed to the mixer was more uniform, resulting in a more consistent paste material.

Subsequent to commissioning, the plant has reportedly performed well. The mine has continued to work on improving the quality of the paste to accommodate variations in ore mineralogy and feed rates.

3.3

Macassa Mine

The Macassa Mine, owned by Kinross Gold Corporation since 1995, has been in production since 1933. The mine is located near Kirkland Lake, Ontario with access to good infrastructure. The mine workings extend over a horizontal area of 2,750 metres and to depths of 2,150 metres, with current workings between the 1,295-metre level and 2,150-metre level. Following a major rockburst in November 1993, a review of the mining system recommended changing from underhand cut and fill mining using cemented rockfill to mechanized longhole stoping with alternative types of backfill.

Paste fill was evaluated as a backfill alternative due to the advantages it offered over rockfill, including higher placement rates, the ability to fill more openings underground, lower costs, and potential better support and control of rockburst prone areas. Construction of the paste fill plant commenced in November 1995 and the plant was commissioned and ready for production by April 1996.

Due to the extremely fine grind of the gold mill tailings – up to 90% minus 20 micron – an engineered paste backfill was required. A mixture of tailings and alluvial sand was used to alter the particle size distribution and gradation, in order to achieve the suitable rheological properties and strengths. Strength testing with various mixtures of tailings and sand showed that a 25%:75% tailings:sand mixture gave the best results. The primary function of the backfill at the Macassa mine is to provide a stiff fill. The engineered paste fill is a 25%:75% mixture of tailings and alluvial sand with a binder mixture of 1:1 Portland cement:fly ash. The 28-day UCS of a 25%:75% tailings:sand paste with 5% binder mixed at 75%:25% Portland cement:fly ash was 157 psi (~1.1 MPa).

Surface loop tests were conducted at Inco's test loop facility in Sudbury, Ontario, using 25% tailings:75% alluvial sand mixtures. Although most plants target a paste backfill slump in the 6" to 7" range, a higher slump of 9" was selected for

the Macassa operation due to the anticipated long horizontal transport distances of up to 1200 metres. Pressure losses were found to be low, 0.03 psi/ft (6.8 kPa/m). Gravity test work conducted at Inco's Stobie mine test facility also returned very low pressure losses.

Plant capacity was required to produce ~ 150,000 tons (13,600 tonnes) of paste backfill annually or roughly 400 tons per day.

The Macassa operation became the first commercial operation to use the Paste Production Storage Mechanism (PPSM) method of "flash thickening" their tailings. The very fine tailings, up to 90% minus 20-micron, leaves the processing plant at ~ 30% solids and must be de-watered to 60% solids, prior to mixing with the alluvial sand and binder for the designed paste recipe. The tailings from the mill is diluted to 10-15% solids to optimize settling in the PPSM tank, which is 22 feet (6.7 metres) in diameter and 34 feet (10.4 metres) high. Paste is transferred in roughly 2 tonne batches from the PPSM unit to the mixer.

At the time that construction of the paste plant was initially proposed, underground production was primarily near the mine's No. 3 shaft. Future production was anticipated to come from the No. 2 shaft, located 1 kilometre from the No. 3 shaft. Therefore, the paste plant was constructed adjacent to the No. 2 shaft, where paste would be directed down a borehole to the underground workings. When the plant was commissioned in April 1996, paste was trucked from the plant to the No. 3 shaft area, dumped into a hopper, and fed underground via a 12 ¾" (325 mm) uncased borehole to the 4250 level. Paste is then distributed using a network of vertical and inclined boreholes and piping at 6" (150 mm) diameter. Backfill is placed down to the 6700 level and covers a total distance, vertical and horizontal, of up to 3200 metres from surface. The pour point is monitored using a video camera.

The economic justification for switching to paste backfill at the Macassa operation was a reduction in operating costs from \$40 per ton for rockfill to ~ \$11 per ton for paste backfill (Naylor, Farmery, and Tenbergen, 1997). Additional undetermined cost benefits were anticipated due to the recovery of otherwise inaccessible ore made possible by the use of paste backfill in some areas.

3.4 Henty Gold Mine

The Henty Gold mine, operated by Goldfields (Tasmania) Limited, is located in Western Tasmania approximately 30 kilometres north of Queenstown. The mine is located in a highly sensitive environmental area, adjacent to the South West Conservation Area, surrounded by World Heritage areas.

The steeply dipping, narrow vein ore body is mined by a mechanized underhand cut and fill method. The backfill provides passive ground support, minimizes rock exposure and ensures maximum recovery from the high grade gold ore body (Henderson, Jardine, and Woodall, 1998). The fill also provides a marker horizon for the underhand cut and fill mining. The cut and fill stopes average 2.5 metres in width and are up to 6 metres in height. The top lift of a mining level is mined up to the base of the sill beneath the backfilled level above. A target strength of 2 MPa is required for support for the underhand cut and fill.

A target production rate of 30 tonnes per hour or 15 m³/hour of paste fill is required to meet the cyclic mining schedule at the Henty operation.

In addition, an environmental permitting constraint placed on the Henty operation required that all acid generating waste rock had to be placed underground. Paste backfill is blended with potentially acid generating waste rock to comply with the permitting requirements, to increase backfill strength, and to reduce the amount of binder required to achieve minimum strength requirements.

The initial design for a paste backfill system at the Henty Mine was based on using a 15 dtph drum filter to be located beside the processing plant to provide a filter cake to which a small amount of water would be added in a mixer to form a paste. This uncemented paste would then be pumped uphill 1.2 kilometres to the mine portal, where cement would be added using an inline mixer prior to sending the paste underground via either a borehole or a mine decline. Surface pump

loop tests conducted in 1994 indicated pressure losses would range from 6 to 12 kPa/m.

The initial system was commissioned in March 1996. However, the pumping system between the plant and the mine portal proved to be unreliable and the drum filter operating capacity was 33% lower than design requirements. The reduced filtration capacity was attributed to mineralogy and the fineness of grind.

As a result of the difficulties the paste backfill system was re-designed. The paste recipe was altered to use a mixture of acidic waste rock and tailings. Laboratory strength tests were conducted to confirm that the required short term strengths could be achieved for cut and fill mining. The re-designed paste recipe has been reported to achieve strengths of 2.7 MPa after 7 days curing and 3.5 MPa after 56 days (Evans, 1998). Additional laboratory and surface loop testing was conducted to provide information for re-design of the underground distribution system. The cement addition component of the system was re-located to the backfill plant to eliminate the inefficient pumping of the paste from the plant to the borehole location. The final paste product is now transported in batches to the portal using contract cement trucks where the paste is poured into a hopper feeding the 200 mm diameter, 405 metre long borehole. Schedule 80 steel pipe was used for the main distribution system, with HDPE pipe used in the stopes.

The technological problems related to de-watering the tailings and pumping of the paste that were encountered with the initial design of this plant illustrate the importance of conducting comprehensive characterisation test work prior to design. The experience at the Henty operation shows the significance to design of understanding completely the tailings mineralogy and rheology.

3.5 Brunswick Mine

Noranda Inc.'s Brunswick Mine, a 9,000 tpd underground zinc, lead, copper, silver producer located 27 kilometres from Bathurst, New Brunswick, has been in operation since 1964.

In the past the mine has used cemented and uncemented rockfill supplied from waste rock and a surface quarrying operation. Various mining methods have been used in the past including blasthole open stoping and cut and fill mining. The current mining system uses blasthole open stoping with delayed backfill (94%), modified Avoca method (5%) and cut and fill (1%) (Cooper, Moerman, Konigsmann, 1999). As mining has proceeded at Brunswick ground stresses have developed leading to increased seismic activity.

To address the ground support problems, improve ore recoveries, and reduce the volume of surficial tailings disposal, the decision was made to switch to paste backfill in 1997. The Brunswick paste backfill plant was commissioned in June 1998. Prior to and during successive stages of engineering design, comprehensive tailings characterisation work was conducted including laboratory test work, surface loop testing, pilot plant testing, and underground test pours. Tailings rheological behavior, strength properties, mineralogy, and settling properties were all investigated extensively. The chemical reactivity of the tailings, in particular, self-heating of sulfides was also studied. On-going studies continue to be conducted regarding the long-term stability of the paste backfill, based on potential sulfate attack (section 4.7.1).

One of the functions of the paste backfill is to minimize ground movement through tighter filling of voids than is possible with rockfill. To achieve higher recoveries, pillarless mining has been implemented using smaller stopes with quick filling of mined out stopes. The paste fill is also being used to stabilize caved areas to regain mining access to otherwise unrecoverable ore. By

shortening the mining cycle, the paste fill also contributes to improved stability and recovery.

Stope sizes range from 5 to 30 metres wide and up to 30 metres high. The backfill strength required for exposure of a free-standing wall at the maximum height is 600 to 700 kPa, however, the target strength selected for the paste backfill design was 1 MPa. Originally the mine planned to fill the stopes continuously with a higher binder addition paste of 7% for the whole stope, in order to minimize the time for change-overs and to reduce the amount of flush water. Now however, a paste fill plug is poured to 2 metres above the brow and allowed to set for approximately 72 hours prior to filling the balance of the stope. The mine has generally been using a 5% binder addition for both the plug and the primary stopes. By pouring the plug first and allowing it to set, the mine has realized significant savings in binder costs. In some cases 7% binder addition is used to achieve a higher early strength for the plug. To prevent liquefaction in unexposed fill the minimum target strength is 150 kPa, which is achieved with a 2% binder addition. To minimize sulfate attack type 50 Portland cement was selected as the initial binder for the paste backfill design.

Strength test work has shown there is high variability in the paste fill strength, from 0.3 MPa to 1.0 MPa, due to variability in tailings mineralogy and particle size distribution. In addition there have been some problems related to calibration of the cement addition equipment and inadequate mixing of the cement, which are being addressed. Based on extensive test work that has been conducted related to internal sulfate attack on the Brunswick paste, modifications to the paste plant are presently being made to accommodate a switch to a Type 50 Portland cement:Type F fly ash (60:40) binder. The test work has shown that this binder recipe will mitigate the effects of sulfate attack (Moerman, 1999).

A target design capacity of 360 dry metric tonnes per hour (dmtph) was established to meet mine-scheduling requirements. To minimize pipeline pressures, the borehole and piping system was designed to maintain a maximum stope distance of 400 metres from any borehole. The underground distribution system was designed to maximize the efficiency of distribution throughout the complex, multi-lensed ore body and to minimize downtime due to blockages. Three inclined boreholes at approximately 70° -- two boreholes for active service and one spare -- were drilled from surface to the 375-level. Underground boreholes for the four main distribution systems were drilled from the 375-level. The underground boreholes varied in length from 50 to 160 metres. Again, spare boreholes were drilled in three of the four underground distribution systems to minimize potential downtime due to potential blockages. The main distribution pipe is schedule 80 steel pipe and the secondary piping is schedule 40 steel. All of the pipelines are open-ended with no valves, in order to minimize potential ruptures due to blockages.

The mine has also installed an underground monitoring system, which includes pressure sensors at key points in the distribution system. Mobile cameras are used to monitor the pours and stationary cameras are used to monitor the elbows at the base of the surface boreholes.

From an environmental perspective the switch to paste backfill offers significant benefits. In the future only about 35% of the process tailings will have to be disposed of on surface, resulting in a considerable reduction in the final height of the tailings dam and a decrease in geotechnical and reclamation costs related to the dam structures.

Estimated direct operating cost savings of \$0.45 per tonne milled are attributed to using paste backfill instead of cemented and uncemented rockfill. Additional savings of \$0.43 per tonne milled attributable to reduced overhead costs are also expected.

3.6 Lupin Mine

The 2000 tpd Lupin gold mine, owned by Echo Bay Mines Ltd., is located 80 kilometres south of the Arctic Circle. Access to the property is by air year round and by winter ice-road during a 12-week period from January to March annually. All bulk freight including a year's supply of diesel fuel, reagents, and cement must be transported to the mine over the winter road.

The Z-shaped deposit is a steeply dipping sulfide-bearing iron formation mined in three zones. The mine produced from 1982 until its closure in 1996 at which time the Centre and West Zones were being mined below the 1130 metre level. The original mining methods used for exploiting the ore body at Lupin were sublevel open stoping, a limited amount of shrinkage mining, and some trials with Alimak Platform mining. Permafrost extends to a depth of 450 to 500 metres below surface and there is a limited amount of ground water. The mine air was not heated to maintain the permafrost conditions. As mining progressed deeper and higher stresses were encountered, excessive dilution became a problem.

After considerable engineering and economic evaluation, a decision was made to switch the mining method to sublevel retreat under/over consolidated fill, using paste backfill. The identified functions of backfill for the Lupin mine included ground support, improved mining recoveries through a reduction in dilution, and the provision of a working floor. Conventional hydraulic slurry backfill was not possible due to inadequate volumes of coarse tailings. In addition, the mine did not want to introduce excess water into the mine, which would be necessary with a hydraulic backfill. Rockfill was an uneconomical backfill option. Paste fill gave the added advantages of reduction of surficial tailings deposition, shorter cycle time, and a lower binder cost than for hydraulic fill. The Lupin mill used a conventional Merrill Crowe process for recovering gold, producing filtered tailings cake at approximately 82% solids. Thus, the need for installation of high capital

cost de-watering equipment was eliminated, which was a big economic plus for the introduction of a paste backfill system at this operation.

Initial paste characterisation test work was conducted starting in 1992 and included mineralogy, particle size distribution, moisture/slump relationship, and strength and rheological testing. The mineralogy of the tailings particles was found to consist of 1) iron formation – amphiboles, predominantly hornblende and grunerite; ~ 10% sulfides including pyrrhotite, arsenopyrite, and pyrite; minor quartz, graphite, and carbonates; 2) phyllites – sericite, chlorite, biotite; and 3) quartzite. The average specific gravity of the iron formation was found to be 3.16 and the specific gravity of phyllite/quartzite material 2.75. It was assumed that with minimal dilution the ore specific gravity would be 3.16. The average particle size distribution of the tailings was 46.6% minus 20 micron, with a range of 38% to 55% minus 20 micron. Rheological test work confirmed that a stiff paste could be produced having a solids density of 79%. When this paste was left uncemented for 21 days, bleed water would result in a final density of 83% solids.

Surface loop tests were conducted in Edmonton to determine the flow behavior of the Lupin paste.

The paste backfill plant was constructed starting in January 1994 and was in full operation by early 1995. A cement storage facility was included in the design to provide storage for a one-year inventory of cement hauled to the mine over the winter road.

The target paste backfill strength was determined to be 0.5 MPa, which could be achieved with a 3% binder addition of Portland cement. Slot raises were excavated directly in the paste backfill for the adjacent stope within 7 to 10 days of filling. The cement cost was reported to be 80% of the total operating costs of the Lupin paste backfill system (Sandhu, 1996). Strength testing using

cement:fly ash mixes was conducted. The cost of freight to the mine from the South is the overriding economic factor and a tonne is a tonne no matter whether it is cement or fly ash. There would be no significant cost advantage to using a fly ash mixture, particularly after factoring in the capital costs of building extra silage facilities and installing new delivery and mixing systems for the paste preparation plant. Test work was also conducted on dolomitic rock from a source located near the Lupin mine, which might have provided a cheaper binder source. Unfortunately the dolomite did not display any cementitious properties.

The target design capacity of the paste plant was 110 tph to ensure scheduling requirements would be met. The proposed backfill plan included filling of existing upper mine voids at a rate of 2,300 tons per day (stpd), after which the backfill production would be reduced to 1,150 stpd to maintain the on-going backfilling of operating stopes. Paste was pumped a distance of 215 metres from the mixing plant to the shaft column and then travelled under gravity flow down the shaft in a vertical 150 mm diameter schedule 80 steel pipe. At the 650-metre and 890-metre levels sweeping 90° elbows were used to connect the shaft pipeline to the main distribution lines leading to the working areas. Due to the favourable vertical:horizontal ratios there was adequate pressure for gravity flow of the paste to the stopes. Pipe wear on the shaft column and the elbows was monitored and was not found to be excessive. A small, 15 mm diameter vent hole in the pipeline at the shaft collar was used to prevent cavitation and surging in the distribution system.

Air quality was monitored underground to ensure that working conditions remained safe with respect to potential generation of hydrogen cyanide gas or ammonia gas and discharge water was monitored to ensure that cyanide levels were within acceptable limits. The use of paste backfill resulted in a significant reduction in the amount of tailings that must be disposed on surface at the Lupin mine, which was an important environmental plus.

3.7 Critical Success Factors

The target paste backfill system design criteria identified at the six mines discussed in sections 3.1 through 3.6 are summarized in Table 3.1. These target backfill design criteria have been identified from the literature review of the paste backfill system implementations at the four gold mines and two base metal mines under the general categories of geomechanical properties, distribution and placement criteria, environmental performance, and socio-economic performance. The specific target properties for each of the operations are based on site-specific mining conditions, which dictate the required mining method and backfill function.

Based on these case studies of paste backfill design in practice, a number of critical success factors have been identified, which influence the ability of the paste backfill system to meet the target design criteria and to deliver consistent backfill quality and performance. They are as follows:

- Design based on the complete knowledge of the physical, chemical and mechanical characteristics of the backfill material (tailings)
- Holistic or operation-wide approach to paste backfill system design and operation
- Recognition of the significance of scheduling and logistics
- Adequate commissioning and “de-bugging” period
- Monitoring and on-going re-evaluation of backfill quality and performance

Table 3.1 Target Paste Backfill Design Criteria – Case Studies

Operation	Geomechanical Properties	Distribution and Placement Criteria	Environmental Performance	Socio-Economic Performance
Louvicourt Mine, Val d'Or, Quebec	0.7 MPa 7-day strength for fill plugs 1.2 MPa 56-day strength for delayed backfill	2400 tpd paste fill required 200 tph design capacity 60% of tailings to be placed underground	Underground disposal of sulfide-rich tailings Reduction of volume of surface tailings	Show economic justification based on dilution, operating cost, capital cost, cycle time, sequencing
Golden Giant Mine, Hemlo, Ontario	Required free-standing walls with delayed fill minimizing the dilution 33 metre stope heights	125 tph design capacity	Reduce surface disposal of tailings and defer construction of second tailings dam	Reduce backfill operating costs – alternative to high cost rockfill quarrying Reduce cost of constructing second tailings basin
Macassa Mine, Kirkland Lake, Ontario	Requires a stiff fill for high rockburst prone conditions 28-day strength ~1.1 MPa	~ 400 tons per day Total vertical and horizontal distance of 3.2 kilometres from surface	Provide a safer working environment	Reduce operating costs – alternative to high cost rockfill
Henty Gold Mine, Tasmania	2MPa strength for underhand cut & fill mining support Maximize ore recovery of high grade gold mine	30 tph required to maintain cyclical mining method	Placement of acid generating waste rock underground Reduce surface volume of tailings	Required cost effective means of mining high grade ore body in area with strict environmental regulations
Brunswick Mine, Bathurst, New Brunswick	Improve ground support and increase ore recoveries 1 MPa strength in primary stopes with stope heights up to 30 m 150 kPa minimum strength to prevent liquefaction for unexposed fills	360 dmtph required Maximum stope distance from borehole 400 metres	Reduce surface tailings deposition	Alternative to high cost rockfill Increase ore recoveries
Lupin Mine, N.W.T.	Reduce dilution Minimum ~ 0.5 MPa strength with stope heights up to 40 m	110 tph design capacity	Reduce volume of tailings on surface	Increase mining recoveries in a cost effective manner within logistical constraints of a remotely located operation

All of these case studies of paste backfill design in practice illustrate the importance of undertaking a comprehensive tailings material characterisation test program as a basis for reliable system design. Mineralogy and particle size distributions, in particular, can have a profound effect on the geomechanical and rheological properties of the resulting paste material, upon which the design of the mixing, distribution, and placement components of the system depend. The scope of characterisation test work must be such that it will encompass the full range of material variability in order to provide the basis for design of a paste backfill system that will produce a paste of consistent quality and performance.

Existing mining operations often have an advantage over new mining projects, because they have a working knowledge of the characteristics and the range of variability of their tailings. Adequate and representative samples of tailings material are also readily available from operating mines for paste characterisation test programs. The paste characterisation test work for new mines, on the other hand, can be hampered by insufficient sample size and a lower level of confidence that the material is representative of full production tailings. The tailings from metallurgical bulk sample test work may not necessarily be representative of final plant tailings. Many important geomechanical and rheological properties of the paste backfill will depend on the mineralogy, particle size distribution, and specific gravity. It is critical to ensure that the samples of tailings material used for paste characterisation test work, particularly with new operations, are representative.

As part of the characterisation test work and evaluation process, it is important to be aware of potential issues related to physical, chemical, and mechanical properties of a specific tailings, which may affect paste backfill quality and performance. These issues, which are discussed in more detail in the following chapter include sulfate attack, geochemical leaching, and acid generation. For example, operations with high sulfide content in their tailings, should be particularly careful to ensure that the long-term stability of their backfill is not

jeopardized by sulfate attack, by conducting adequate strength/binder alternative test work.

A holistic approach to the paste backfill system design from conception through implementation and operation is essential. This approach represents for many mines a dramatic change in mining culture. Traditionally in mining there has been a "great divide" between the mine and the mill, with somewhat of a "black box" attitude regarding each other's operations. This division has to be broken down because reliable paste backfill system design depends on input and cooperation from all departments starting with geology and mine planning, through to mine operations, the mill, and waste management personnel. Because mineralogy can have such a profound effect on paste backfill quality, it is important to have as much input from mine geologists into the design phase as possible regarding potential variability in mineralogy in future mining areas. Another example of interdepartmental cooperation in backfill design, which may appear somewhat trivial, is the provision of surge capacity to accommodate regular sag mill maintenance shutdowns. Mills usually have surge capacity in coarse or fine ore bins to accommodate fluctuations in mine production, however, the backfill requirements for mine scheduling must be taken into consideration along when planning the backfill system design. A holistic approach to paste backfill design will contribute to the reliability of the resulting system.

Each of the case study operations have identified the increase of ore recovery to be a key objective. It is therefore, essential that the paste backfill system be designed to meet the scheduling and capacities of backfilling required for the mine plan and have the capacity to accommodate future development demands.

Paste backfill systems are not cheap. The total reported cost of the Brunswick mine paste backfill system, for example, was \$22.3 million (Cooper, Moerman, and Konigsmann, 1999). Although actual paste backfill system costs have not been a specific focus of this thesis, the capital and operating costs of a paste

backfill system must meet "economic" design targets or they will not proceed. The mining industry today, more than ever, depends on low cost, high productivity operations. The reliability of the paste backfill system design is critical to maintaining production targets and it is therefore, imperative that a "total mine" approach be taken to evaluating cost-benefits.

Commissioning or "run-in" of a new plant is an important phase of plant design and implementation. It is critical, therefore, to provide sufficient time in the design schedule for plant commissioning and training of backfill operators. Fine-tuning of process control equipment is generally required to optimize backfill quality. All components of the mixing plant and distribution system must be systematically checked out. Startup and shutdown procedures, routine and emergency, must be established and practiced. Most of the case study operations reviewed conducted closed circuit trials or test pours to underground stopes or tailings impoundments in order to refine and optimize their systems.

Paste backfill distribution systems are not as forgiving as hydraulic slurry backfill systems. A small change in solids density in a paste can result in line blockages, dangerous pressures, ruptured pipelines, and costly downtime. The operators of paste backfill systems must be well versed in operating procedures. The importance of thorough commissioning, training, and development of operating procedures to paste backfill system reliability cannot be understated. The effect of plugging a main borehole from surface, for example, can be profound. First, there is the effect on mine scheduling and production. If backfilling falls behind schedule, then required stopes may not be ready for mining, resulting in potential production losses. Subsequent mining recoveries may also be lower due to dilution, which might have otherwise been preventable, had backfill been in place. Safety issues may also arise due to delays in backfilling where ground support is required. Added to the production losses is the cost of unplugging or re-drilling boreholes.

Once the paste backfill system is in operation, the monitoring of the system and re-evaluation of backfill quality and performance is vital to ensure that backfill target criteria are met on a continuing basis. All of the case studies reviewed have employed varying levels of instrumentation to monitor the paste backfill system operation. In addition paste backfill quality and performance is evaluated through on-going sampling and test work.

The design and practical operating experience these case study paste backfill operations offers an invaluable source of information and a potential guide for improved design of new systems.

4.0 PHYSICAL, CHEMICAL AND MECHANICAL CHARACTERISATION

Having identified the key paste backfill design criteria, the suitability of available fill material to meet paste design targets must be investigated. Based on the design objectives, the physical, chemical and mechanical characteristics of the tailings must then be evaluated in order to ensure that the backfill product will fall within the required parameters.

The characterisation of the tailings will form the basis for the appropriate engineering design of a paste backfill system including de-watering and paste preparation, backfill transportation and emplacement, environmental, and socio-economic performance.

Many of the physical, chemical and mechanical properties of tailings are inter-related. A critical component of characterisation test work is the determination of the full range of variability in tailings material properties. Variability of tailings properties can have a significant influence on key design criteria. In order to optimize the paste system engineering design, the effect of variability in tailings properties must be fully understood.

Characterisation test work can encompass laboratory, in situ, surface loop, and pilot plant testing. Most preliminary characterisation work involves laboratory test work and analysis. Surface loop testing forms the basis for engineering design of paste distribution and placement systems. In situ test work and pilot plant testing provides valuable input for engineering design. The accuracy of laboratory testing, modelling and predicted backfill response can be verified through in situ measurements. Pilot plant testing enhances paste backfill engineering design by providing feedback on paste performance under actual site specific conditions.

There are many physical, chemical and mechanical characteristics of tailings/fill material that influence paste performance. Characterisation of tailings for paste fill design should include the investigation of the following properties:

- Particle Size Distribution
- Soil Index Properties
- Sedimentation/Filtration Characteristics
- Consolidation Behavior
- Strength
- Rheological Characterisation
- Mineralogy and Geochemical Analysis

4.1 Particle Characteristics

Particle size distribution is one of the most critical parameters to be considered in the evaluation of tailings material for paste backfill. The overall particle size distribution, gradation, and particle shape of the tailings material will affect paste backfill permeability, porosity, strength, and rheological behavior. It will ultimately have a significant effect on the engineering design criteria for paste preparation, distribution and placement.

Standard particle size analysis, using wet or dry screening procedures, produces an accurate particle size distribution ranging down to 400 mesh (Tyler screens) or 38 micron particle size. Sieving or screening methods are suitable for profiling particle size for most mineral process unit operations including grinding, hydro-cycloning, and thickening, as well as for conventional hydraulic slurry backfills which require classification to remove the finer fractions of tailings. For paste backfill applications, however, the fines content (minus 20 micron fraction) must be accurately measured. Particle size analysis for paste characterisation should be conducted using cyclosizing or, preferably, laser analysis.

Particle size distributions are usually depicted as a semilogarithmic plot of cumulative percent passing versus particle size in microns.

4.1.1 Gradation

A flat particle distribution curve indicates a larger size particle size range, while a steep curve indicates a narrow size range. A smooth, concave distribution curve depicts tailings material that is well-graded, where the size fractions are relatively evenly represented. More poorly graded tailings may have uniform distributions with a disproportionately high amount of tailings within a narrow size range or they can have gradational gaps, where there may be a low proportion of intermediate size ranges.

Two commonly used soil mechanics indices of size distribution are the coefficient of uniformity (C_u) and the coefficient of curvature (C_c). These indices are determined as follows:

D_{10} = size at which 10% of particles are smaller

D_{30} = size at which 30% of particles are smaller

D_{60} = size at which 60% of particles are smaller

1) Coefficient of Uniformity (C_u)

$$C_u = D_{60} / D_{10}$$

2) Coefficient of Curvature (C_c)

$$C_c = [D_{30}]^2 / [D_{60} - D_{10}]$$

Well-graded soils (tailings) are generally considered to have a coefficient of uniformity (4 to 6) and a coefficient of curvature (1 to 3). High coefficients of uniformity indicate large ranges of particle size. Because of their wide distribution of particle size, paste backfills may have coefficients of uniformity that range from 2 to 500. Well-graded tailings will be capable of packing more densely, giving higher friction angles between particles.

4.1.2 Particle Shape

Particle shape will influence the way packing density and interparticle bonding or friction angles. Most tailings particles tend to be rough and angular, due to the blasting and comminution of particles in mineral processing. Angular particles will pack with higher friction angles than rounded, smooth particles.

4.1.3 Paste Particle Size Distributions

Landriault (1995) categorized tailings from Canadian hard rock mines as fine, medium or coarse according to percent fines below 20 micron, as shown in Figure 4.1 that illustrates typical paste fills and compares them to a typical classified hydraulic slurry backfill. He categorized full plant tailings backfills generally as follows:

Table 4.1 Tailings Size Classification for Paste Backfill

Full Plant Tailings Classification	Fines Content (Minus 20 Micron Fraction)
Fine	60% - 90%
Medium	35% - 60%
Coarse	15% - 35%

The amount of water held by the tailings is dependent on the colloidal properties of the fines. The higher the percent of fine particles, the greater will be the surface area available to bind to water molecules.

4.2 Sedimentation/Filtration Characteristics

The sedimentation (settling) characteristics of tailings material are of key importance in paste preparation and transportation system design. Tailings dewatering generally employs mechanical gravitational or centrifugal methods of solid-liquid separation.

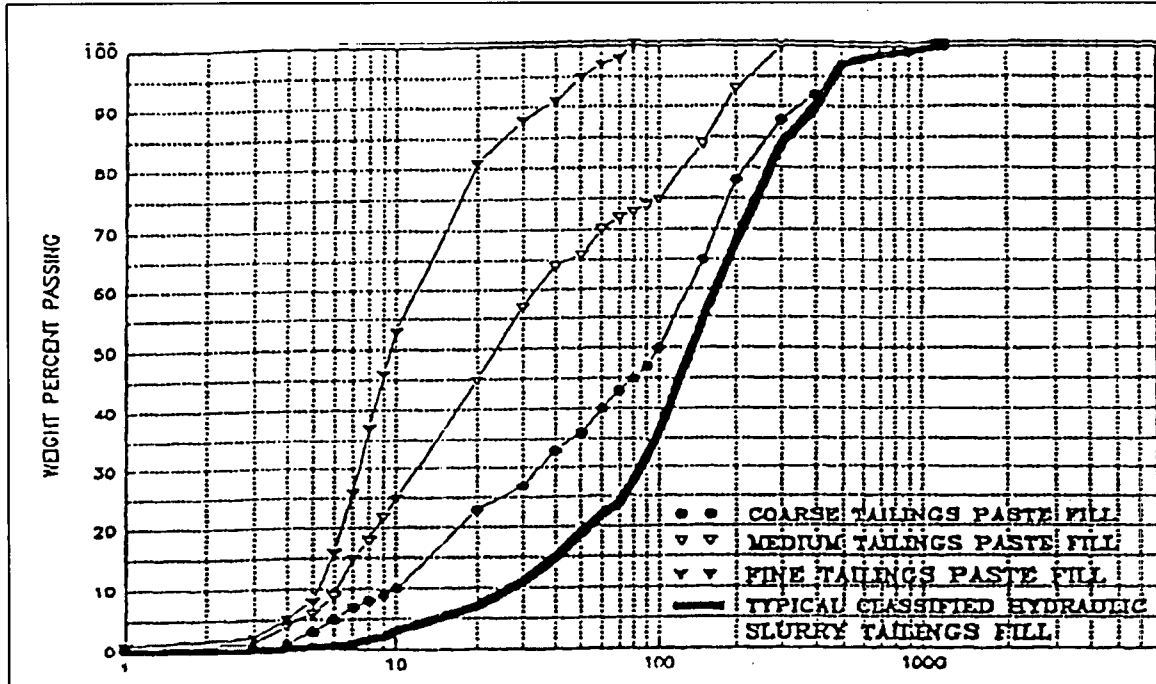


Figure 4.1 Particle Size Distributions of Fine, Medium & Coarse Paste Backfills versus Typical Hydraulic Slurry Backfill (After Golder, 1995)

Particle size becomes the most influential factor in settling rate (Loubser, 1988). Settling of the coarse particle tailings fractions is mainly influenced by gravitational forces. For larger particles, where gravitational forces dominate, the specific gravity of the particles will affect settling rates. The settling rate of the fine particles or colloids, which have higher surface area and relatively low mass, is affected by surface electrical charges. Particle shape also influences the settling rate with spherical shaped particles tending to settle more rapidly than elongate type shaped particles.

Flocculants or coagulants are often used to improve the efficiency of sedimentation and filtration of tailings. An understanding of the colloidal surface chemistry and electrical charge of the tailings is critical to establishing the reagent addition required to optimize paste de-watering systems design.

4.2.1 Backfill Segregation

Conventional hydraulic slurry backfills tend to be non-homogeneous or segregated due to differential settling and the loss of fines, including cement, in drainage from the backfill. As a result the backfill mass can be stratified with planes of weakness created by the layering. In addition, cement content may vary throughout the backfill mass. Backfill segregation may cause reduced backfill strengths that are unacceptable.

Backfill segregation in rockfills can also be a problem. Four mechanisms responsible for segregation in cemented rockfills are poor quality control, in-flight transportation, impact, and gravity flow (Bloss and Greenwood, 1998).

Paste backfills, however, by their nature are more homogeneous and do not tend to exhibit the differential settling on placement that occurs with hydraulic slurry backfills.

4.3 Soil Index Properties

In addition to particle size distribution there are a number of other physical and mechanical parameters used to characterise soils (tailings) that influence the geomechanical properties of backfill. These soil indices include water/solids content, saturation, void ratio, porosity, permeability, bulk density, specific gravity, liquid and plastic limits. The soil indices exhibited by paste backfills can be uniquely different from typical slurry backfills due to the water retentive properties of pastes.

4.3.1 Water/Solids Content

The water content (w) or moisture content is expressed as a weight percent. In most mining/mineral processing applications the water content is the ratio of the mass of the water over the mass of the water and the solids, or:

$$w = \frac{\text{Wet weight} - \text{Dry weight}}{\text{Wet weight}}, \%$$

As a note of caution, in many soil mechanics applications the water content (w) is expressed as:

$$w = \frac{\text{Wet weight}}{\text{Solids weight}}, \%$$

The water content is simply determined by weighing a representative sample of tailings to obtain the wet weight, then drying the sample at a maximum temperature of 105° C to avoid oxidation of sulfide minerals in the tailings and re-weighing the dried sample to obtain the dry weight.

4.3.2 Saturation

The degree of saturation (S_r) is the ratio of the volume of the water to the total volume of void space (air and water), or:

$$S_r = V_w / V_v, \%$$

The degree of saturation may vary from 0% for a completely dry soil to 100% for a fully saturated soil.

4.3.3 Void Ratio

The void ratio (e) is the ratio of the volume of the voids to the volume of the solids, or:

$$e = V_v / V_s, \%$$

4.3.4 Porosity

Porosity (n) is the ratio of the volume of voids to the total volume, or:

$$n = V_v / V, \%$$

Void ratio and porosity can be related as follows:

$$n = e / (1 + e), \%$$

4.3.5 Permeability

The permeability is a measure of the flow of a fluid through a porous medium according to Darcy's law, which states:

$$q = Aki$$

Where
 q = volume of water flowing per unit time
 A = cross sectional area of soil corresponding to the flow
 k = coefficient of permeability (metres per second)
 i = hydraulic gradient

The coefficient of permeability will depend on the size of the pores (interconnected voids), which is dependent on the particle size and shape. The void ratio, therefore, is a determinant of the coefficient of permeability. Temperature/viscosity effects will also influence the coefficient of permeability.

The percolation rate and permeability are critical parameters in the design of hydraulic slurry backfill systems, where the ability of the backfill to freely drain excess water is of key importance. A general rule of thumb for hydraulic slurry backfills is a 10 cm/hr percolation rate. The percolation rate is highly dependent on the particle size distribution, in particular, the percent fines and in porosity. An increase in the fines fraction of a slurry backfill will significantly decrease the percolation rate.

Due to their high fines content pastes have very low permeability and, therefore, percolation rate is not a significant design factor for paste backfills.

4.3.6 Bulk Density, Unit Weight, and Specific Gravity

The bulk density of soil (ρ) is the ratio of the total mass to the total volume, or:

$$\rho = M / V, (\text{kg/m}^3)$$

The unit weight (γ) is the ratio of total weight to the total volume. The total weight is mass times gravitational force (9.81 m/s^2).

$$\gamma = Mg / V, \text{ kN/m}^3$$

Specific gravity (G_s) of a solid, liquid or slurry is defined as the ratio of the weight of a given volume of the material to the weight of an equal volume of water at standard conditions of 1000 kg/m^3 at 4° C , or:

$$G_s = M_s / (V_s \rho_s) = \rho_s / \rho_w$$

4.3.7 Liquid and Plastic Limit

Plasticity, defined as the ability of a soil to undergo unrecoverable deformation at constant volume without crumbling, can be an important consideration for fine-grained soils. Plasticity is attributed to a high content of clay particles. The water content will determine whether the soil is in a liquid, plastic, semi-solid or solid state. In a plastic state, the net interparticle forces will be strong enough to maintain cohesion between particles and at the same time allow particles to freely slide relative to each other. If the water content of the soil increases, there will be a decrease in net interparticle attractive forces and the volume of the soil will increase.

Soil will show plastic behavior over a range of water content from the upper liquid limit (w_L) to the lower plastic limit (w_P). The plasticity index (I_P) is the water content range, or:

$$I_P = w_L - w_P$$

4.4 Consolidation Behavior

According to soil mechanics theory consolidation is a gradual reduction in the volume or densification of a fully saturated soil of low permeability due to drainage of pore water that dissipates excess pore water pressure. Prior to consolidation the soil is in an undrained state. As excess pore water gradually drains from the soil mass, the solid particles move closer together and interparticle forces increase, resulting in a reduced volume and increased effective vertical stress. The soil is considered to be in a drained state once excess pore water pressure has been dissipated. This consolidation is usually considered to be one-dimensional, with deformation in a vertical direction. Drainage of the excess pore water is dependent on the permeability of the soil mass.

Conventional hydraulic slurry backfills, which are generally designed to be highly permeable, should drain rapidly after placement and, therefore, will consolidate rapidly. Paste backfills, however, due to the high fines content, will not drain a significant amount of water.

In their discussion of geotechnical factors affecting the surface deposition of paste tailings, Cincilla, Cuning, and Van Zyl (1998) have theorized that the mechanical de-watering of fine tailings to create low slump, low moisture content, low void ratio pastes, by-passes the primary settlement process or consolidation required in conventional slurries. They identified the maintenance of partial saturation (reduced void ratio) in the paste as critical from the geotechnical standpoint of maximizing storage volumes and resistance to liquefaction.

4.5 Strength and Stiffness

4.5.1 Strength

Backfill strength is determined by the interparticle frictional forces and by cohesion. The shear strength parameters of uncemented backfill, apparent cohesion and internal angle of friction, are given by the Coulomb equation:

$$\tau_f = c + \sigma_f \tan \phi$$

where c = cohesion (or apparent cohesion)
 ϕ = angle of shearing resistance (or internal angle of friction)

Apparent cohesion is created by surface tension forces in pore water. The interparticle surface tension will decrease as the amount of water between them decreases. At zero saturation, there will be no surface tension and hence no apparent cohesion. A fully saturated backfill will also exhibit no apparent cohesion.

The internal angle of friction results from the interlocking of solid particles and will be dependent on grain size and shape and the packing density.

4.5.2 Stiffness

The stiffness of backfill material is defined as its resistance to deformation. The addition of cement to backfill increases its stiffness. Where the backfill is required to provide a free-standing wall for pillar recovery or where the backfill is placed to limit convergence, increased stiffness is important. Backfill has a modulus of elasticity that is significantly lower than intact rock – typical Young's modulus (E) values for intact rock and backfill are 70 GPa and 0.5 GPa, respectively. There is a trade-off between stiffness and brittleness as cement content of backfill is increased. As the backfill becomes more brittle, it will be more susceptible to fracturing.

4.6 Rheological Characterisation

Rheological characterisation of tailings is one of the most critical considerations in paste system design because of the wide ranging influence of paste rheology on all aspects of paste systems -- preparation, transportation and placement.

A developer of paste production systems, EIMCO, cites the importance of rheological characterisation of pastes for thickener sizing, tailings impoundment design, underflow pumping, and control systems (Klepper, Laros, and Schoenbrunn, 1998). Rheological profiles of pipeline pressure losses are essential to engineering design of distribution systems and pipeline materials selection.

One rapidly developing area of paste technology is related to surface deposition of tailings. Methods of dry stacking and slope deposition require that the tailings material develop a yield stress. Surface deposition design is based on the rheological properties of the tailings material. Boger (1998) has stated that a combination of an understanding of the rheology of high density suspensions with an understanding of the environmental challenges facing the mining industry has led to improved methods of waste minimization. Slope deposition methods have been applied in the aluminum industry for deposition of bauxite red muds in Australia and in Quebec. The technology of dry stacking is also being evaluated and applied to disposal of oil sands, potash, phosphate, and clay tailings.

The rheological behavior of a given paste is highly dependent on the particle size distribution, specifically the fines fraction, but may also be affected by the mineralogical/geochemical composition of the tailings. Rheological behavior can be affected by many variables including particle size distribution, solids density, and surface chemistry. Temperature, pH, and reagent addition will also affect paste viscosity. Rheological characterisation is crucial to developing an

understanding of the range of rheological behavior in a given paste in order to design a paste system that will produce a consistent product.

4.6.1 Slump

A key measure of paste consistency, slump has been adopted from the concrete industry. The conventional slump test (CSA Test Method A23-5C or ASTM C143) measures the vertical distance a sample spreads or "slumps" when released from a slump cone at a starting height of 305 mm (12 inches). The higher the slump, the more fluid is the paste. Typical low and high slump paste backfill samples are illustrated in Figures 4.2 and 4.3.

Slump versus solids density profiles can be developed for a specific tailings material. Because slump is a measurement of paste consistency, it can be used in paste mixing control systems through slump versus power draw or torque parameters to obtain a uniform fill product. Variability in particle size distribution affects slump by changing the porosity and hence, the water content of the resulting paste. If the particle size distribution is well graded over a wide range, porosity will be low resulting in a higher solids density at a given slump.



Figure 4.2 Typical Low Slump Paste Backfill



Figure 4.3 Typical High Slump Paste Backfill

Many operations using paste backfill have discovered that slump can be highly sensitive to moisture content. At BHP World Mineral's Cannington lead-zinc-silver operation in Australia a change of approximately 0.5% in moisture content results in a 1-inch change in slump (Skeeles, 1998). Similarly, at Battle Mountain Gold's Golden Giant operation near Marathon, Ontario, slump has been found to be equally sensitive to moisture content (Dodd and Paynter, 1997). When the slump is highly sensitive to moisture content it is difficult to develop a continuous paste production system that will maintain a high level of paste consistency. For this reason many operations have chosen to design "batch" type systems for mixing paste, which can then be transferred to a surge or gob hopper to provide a continuous feed to the distribution system.

4.6.2 Mineralogical Influence on Paste Rheology

The mineralogical composition of tailings has been shown to influence pipeline friction losses, depending on the pipe material. Landriault et al. (1997) have indicated that some pyritic tailing pastes have shown unusually higher flow resistance in steel pipe than in plastic pipe, while siliceous tailing pastes have not

shown this same difference (Figure 4.4). The particle size distributions of these mineralogically different pastes were similar.

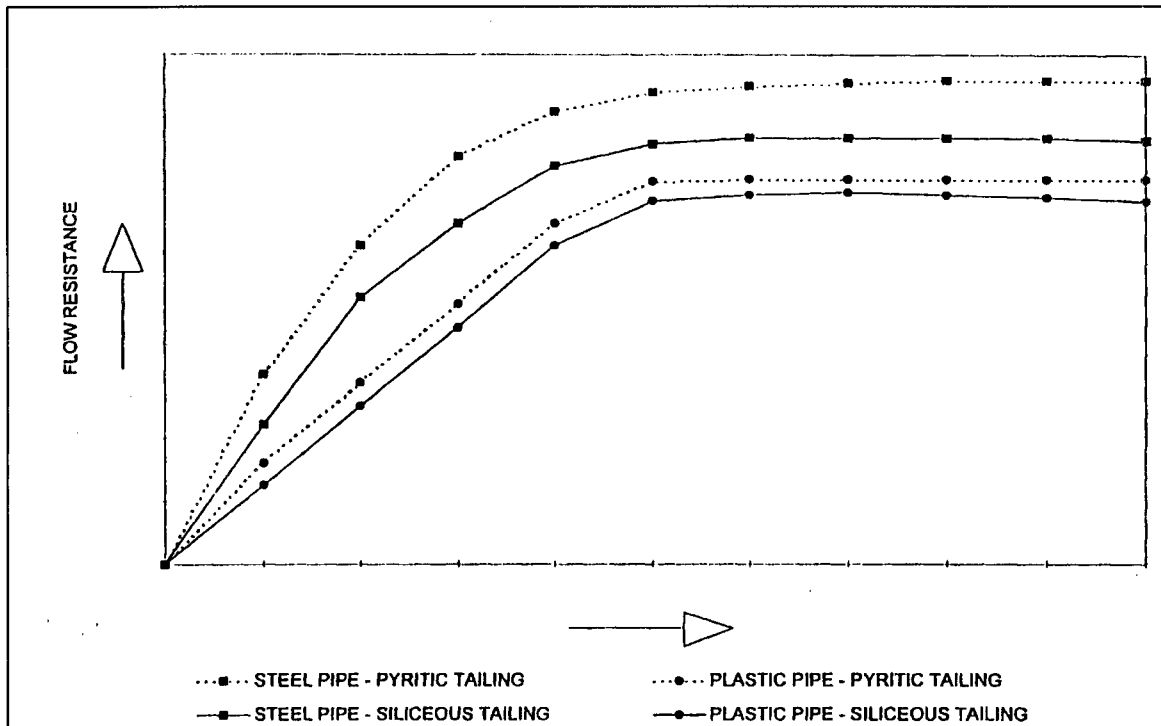


Figure 4.4 Effect of Mineralogy on Pressure Losses (After Golder, 1997)

4.7 Mineralogical and Geochemical Analysis

The mineralogical/geochemical composition of tailings, can have significant ramifications regarding the long term strength, rheology and environmental stability of paste backfills. Mineralogical composition and geochemical reactivity of tailings can significantly effect cementation properties and paste backfill strength. Environmental performance of paste for underground backfill and in particular for surface disposal is dependent on leachability of heavy metals as well as acid generating potential.

Potential geochemical reactions that should be considered in paste backfill design include:

- Chemical reactions related to cement hydration
- Oxidation reactions, particularly of sulfide minerals
- Leaching of metals and ions
- Production and release of toxic gases and compounds

Many mine tailings are sulfide-rich. Consequently, there are many potential chemical reactions related to the sulfide minerals that compose backfills, which should be considered including cementation effects, self-heating of sulfides, acid generation, and leaching of heavy metals and ions.

Paste backfills generally have a minimal amount of excess water available to leach due to the colloidal properties of the fine tailings particles in the paste. Because of the low permeability of pastes there is less risk of reaction between ground water, mine water or precipitation than there is with more permeable materials.

4.7.1 Mineralogical/Geochemical Effect on Cementation and Paste Backfill Strength

The mineralogy of the tailings can have a significant effect on the backfill strength achieved. As discussed in Section 2.2.1 pyritic tailings have been reported to produce higher strengths than siliceous tailings, depending on the given binder used.

Some sulfide-rich tailings, those containing pyrrhotite in particular, exhibit self-cementing properties. The degree of sulfide reactivity can vary, depending on the type of mineralization present, whether or not there are combinations of sulfides, and the extent to which oxidation-enhancing conditions such as moisture content, oxygen, temperature, low pH, and bacterial activity are present. Particle size and shape can also influence the tailings sulfide reactivity. Finer

The potential loss of physical strength and long term stability of paste backfill, particularly at operations where sulfide-rich tailings are produced is an important area to be evaluated prior to paste backfill system design. Several Canadian paste backfill operations have experienced difficulties with long term stability of paste fills due to sulfate attack and related geochemical alteration.

Ouellet, Benzaazoua, and Servant (1998) investigated chemical alteration in a paste backfill at a Canadian polymetallic mining operation that was experiencing unacceptably high dilution when secondary pillar recovery commenced. The paste backfill was produced from sulfide-rich pyritic tailings using a 60:40 mixture of Portland cement:fly ash as the binding agent. Results from triaxial and compressive testing of laboratory samples and cores recovered from an in situ sampling program indicated significantly lower backfill strengths than original pilot plant test work had predicted.

Subsequent mineralogical and microscopic examination of the samples showed high porosities and the presence of gypsum and ettringite rather than portlandite minerals. Expansive gypsum minerals cause fracturing and reduce the cohesive structure of the backfill. The authors concluded that this demonstrates the importance of investigating the long term stability of backfills due to chemical alteration, particularly when sulfide-rich tailings are used.

During the course of laboratory and stability analysis test work on paste fill from the Golden Giant Mine, oxidation was noted on samples that were cured under sealed conditions for 28 days and then removed from their molds and allowed to continue curing under conditions of 85% humidity (Pierce, Bawden, and Paynter, 1998). The paste fill samples were reported to be composed of approximately four to five percent pyrite. After 56 days curing there was a rim of oxidized material on the outside of the samples which exhibited little strength. At 112 days curing time, the oxidation had extended through the 5 centimetre diameter samples and uniaxial compressive strengths at all binder addition rates had

decreased below the 28 days strengths. The authors noted that with longer cure times oxidation might seriously affect fill stability.

The effect of tailings and binder geochemistry on paste backfill strength has also been the focus of considerable research at Noranda's Brunswick Mining Division (Bernier, Li, and Moerman, 1999). During the course of characterisation test work prior to construction of the paste backfill plant at the Brunswick Mine, it was discovered that maximum paste backfill strengths were reached at anywhere from 90 to 120 days curing, after which strengths deteriorated.

The Brunswick tailings typically contain 28% to 37% total sulfur, corresponding to 52% to 69% pyrite. Mineralogical analysis shows pyrite to be the dominant mineral (40% to 64%), other sulfides include pyrrhotite (5% to 14%), as well as sphalerite, galena, and chalcopyrite, with gypsum reported as the main secondary sulfate mineral (0 to 4.48%).

Because initial strength characterisation tests indicated that sulfate attack was causing a loss of strength after 90 to 120 days curing, Type 50 sulfate resistant cement was chosen as the binder for the original design of the paste backfill plant at Brunswick. Additional laboratory studies were conducted using alternative binders to examine the mineralogical/geochemical effect of tailings/binder combinations on strength development and to mitigate sulfate attack. Measurement of extractable aluminum (Al) concentration was used to calculate ettringite content of paste backfill samples.

Five different binder combinations were tested using a 95% tailings:5% binder recipe as follows:

- 1) Pure Portland cement Type 10
- 2) Pure Sulfate resistant cement Type 50
- 3) 20:80 PC Type 10:Slag 1 Mixture
- 4) 20:80 PC Type 10:Slag 2 Mixture

5) 60:40 PC Type 10:Fly Ash (Type F) Mixture

For all five of the binder combinations, ettringite contents either decreased or decreased and stabilized over the curing period from 3 to 120 days. Then at 120 days curing there was a sudden spike in ettringite content for all samples, with the pure cement samples, Type 10 and Type 50, and the 60:40 PC:FA mixture showing the largest ettringite increases. This was interpreted to be secondary ettringite formation and was not as high as the primary ettringite formation. Compressive strengths also decreased after 120 days with strength losses at 360 days as a percent of maximum UCS reported to be:

- 35% for the pure type 10 Portland cement
- 24% for the pure type 50 sulfate resistant cement
- 35% for the 20:80 PC10:S1 mixture
- 7% for the 60:40 PC10:FA mixture

The loss of backfill strength was assumed to be due to secondary ettringite formation. Additional paste samples prepared with 5% Type 10 Portland cement and cured for one year contained visible clusters of gypsum and minor ettringite, as well as fine cracks and oxidation veins.

The cement:fly ash mixture exhibited the highest resistance to internal attack, based on the test work. Consequently, the mine is currently planning to switch their paste recipe to a 60:40 blend of Type 50 sulfate resistant cement and 40% Type F fly ash.

The authors of the Brunswick study have raised a number of other important questions related to mineralogical/geochemical effects on development of strength in paste backfill including the potential effects of gypsum and/or clay minerals contained in the tailings, alkali aggregate reactions, alkali carbonate reactions, and alkali silica reactions.

4.7.2 Self-Heating of Sulfides

Oxidation of sulfide ores, concentrates, and tailings can lead to spontaneous combustion. Several mines in Canada have experienced underground fires due to self-heating of sulfides, including Cominco's Sullivan Mine at Kimberley, British Columbia and Noranda's Brunswick Mines at Bathurst, New Brunswick. In addition to the danger of fire, oxidation reactions can also lead to detrimental health and safety working conditions due to excessive heat generation, sulfur dioxide gas generation, and oxygen depletion.

Investigations by Rosenblum and Spira (1995) have identified pyrrhotite as the only sulfide mineral to be of concern for self-heating. The effect of a number of variables on self-heating of tailings was tested including pH, aging for one month under both dry and wet conditions prior to testing, pre-aeration, compaction, moisture content and particle size. Moisture content appeared to have the effect of increasing the self-heating rate at a moisture content of 3%, after which self-heating rate decreased. As expected, finer particle size material had a significantly higher heating rate than coarser material due to the reactivity from increased surface area.

As part of characterisation test work for the paste backfill plant at Brunswick mine, self-heating analysis was conducted on tailings and it was determined that a concentration of pyrrhotite greater than 14% would potentially cause self-ignition. Sampling data from the Brunswick mill over a two year period beginning in October 1996 indicated that pyrrhotite concentrations ranged from 4% to 9% (Cooper, Moerman, and Konigsmann, 1999).

Patton (1952) reported that the pyrrhotite:slag (3%:97%) backfill used at the Horne mine resulted in some release of sulfur dioxide gas and in heat generation, which was managed with adequate ventilation.

Another potential deleterious effect of chemical reactivity of sulfides in backfill, particularly pyrrhotite, is oxygen depletion. The mechanisms of oxygen depletion were studied in a high pyrrhotite content conventional hydraulic slurry backfill from Inco's Thompson mine (Bayah, Meech and Stewart, 1984). They found that oxygen depletion was rapid where water was flowing at moderate rates. The rate of oxygen depletion was also found to be affected by the pulp pH and the cement content. Monitoring of backfilled zones, increased ventilation, and better control of mine water flow away from sealed backfill areas can be used to ensure that safe levels of oxygen are maintained. Paste backfills, due to their lower permeability, may not present the same conditions for oxygen depletion as conventional slurry backfills. Mine water will not tend to flow through the paste backfill, which should reduce the opportunity for exposure to the reactive mineral particles. Because of their higher content of fines paste backfills may, however, contain much higher total surface areas of potentially reactive mineral particles. Where high sulfide content tailings are used for paste backfill, the mine should be alert to potential oxygen depletion in backfilled areas.

4.7.3 Volatization of Cyanide and Ammonia

Where cyanide is used in metallurgical processing, primarily with gold and silver ores, there may be potential for leaching of cyanide from paste backfill, which can result in the generation of dangerous hydrogen cyanide gas. Most precious metals processing operations employ cyanide destruction systems to reduce cyanide in the tailings to regulatory levels. The addition of Portland cement and other binders also aids in maintaining the high pH conditions in the backfill and preventing the formation of hydrogen cyanide.

South African gold plants typically neutralize cyanide-bearing gold tailings slurry backfills with ferrous sulfate prior to transporting underground (Pothas, 1988).

The addition of a premix of ferrous sulfate and cuprous sulfate was incorporated into Battle Mountain Gold's Golden Giant Mine paste backfill mix system to maintain free cyanide levels below regulatory thresholds of 60 ppm. This patented "Hemlo" water treatment process for removing cyanide essentially precipitates free cyanide as stable cuprous cyanide (Dodd and Paynter, 1997).

No hydrogen cyanide gas was detected from paste backfill at Echo Bay's Lupin operation. However, elevated levels of ammonia were detected at the underground crusher and loading pocket area when the paste backfill was first used at Lupin. The generation of ammonia was attributed to a reaction between the cement in the paste backfill and Anfo. Increased ventilation and improved blasting practices eliminated this problem.

Potential cyanide leaching from paste backfill should always be evaluated as part of the geochemical characterisation of tailings from plants using cyanidation processes.

4.7.4 Acid Generation

The potential for acid generation due to oxidative reactivity of sulfide-rich tailings is a serious concern. Particularly where paste is being considered for surface disposal, the evaluation of the potential for release of acid and toxic heavy metals into ground water is extremely important. Paste has been identified as having an environmental advantage over conventional tailings disposal due to its low permeability and the limited amount of free water. Portland cement added to paste backfills helps to buffer reactive sulfides and to reduce leaching of metals. For these reasons it has been proposed that paste can be engineered to reduce the risk of acid generation by encapsulation of potentially acid generating wastes with non acid generating paste and for co-disposal of wastes.

The mineralogical/geochemical composition of tailings will have a profound effect on the acid generation and neutralizing potential of a paste. Geochemical characterisation of tailings for paste backfill should include acid base accounting and the analysis of sulfur species present in tailings. Results of preliminary geochemical analysis will determine the necessity for further test work including short and long term kinetic testing.

4.8 Characterisation Test Work

Characterisation test work for paste backfill design can be carried out in several phases ranging from laboratory bench testing and analysis to full scale pipe loop and gravity flow testing and pilot plant testing. Initial laboratory work including particle size and mineralogical analysis, sedimentation characteristics, soil properties, preliminary rheological characterisation, and strength tests will establish the suitability of the tailings material for paste and provide the basis for on-going laboratory work, leading to flow loop testing and pilot scale test work. One important focus of characterisation test work should be the determination of the range of variability in tailings material and the evaluation of how the variability influences the quality of the paste and the ability of the paste to meet target design criteria. The degree to which tailings variability is important is highly site specific. Ore mineralogy and processing methods determine the downstream variation in tailings. The more information that can be gained on paste quality and response through characterisation test work, the more accurate the engineering design of the paste backfill system will be.

4.8.1 Laboratory Test Work

Initial laboratory analyses including mineralogy, particle size distribution, soil properties and rheological indexing will provide a preliminary evaluation of the suitability of the tailings material for paste. The degree of confidence associated with results of initial test work is dependent on obtaining representative samples

of the tailings material. In the case of new mines, where laboratory work relies on geological core samples or metallurgical bulk samples, this may be of concern. For operating mines, the collection of representative tailings samples should be less of a problem.

Depending on the results of initial laboratory analyses, further geochemical analysis, including acid base accounting, settling and de-watering tests, and geomechanical strength test work should be conducted. Backfill strength test work can be fairly comprehensive, particularly where alternative binder recipes are investigated.

4.8.2 Surface Loop and Gravity Flow Testing

Surface Loop Testing

Surface pump loop testing provides pressure loss/flow resistance data that is crucial to engineering design of the distribution and placement system for paste backfill. Preliminary system capacity targets are based on the mining system. The mining method, production schedule, and geometry of the mine (ore deposit) layout determine the required paste backfill capacity and delivery schedule. Based on these targets and preliminary rheological properties, a surface flow loop can be constructed to evaluate the flow response of paste through a range of pipe diameters and materials.

The surface loop consists of relatively long (~ 100 - 200 metres) sections of each pipe diameter/material being tested with uniformly spaced pressure transducers inserted into the pipeline to measure pressure losses. Paste material is pumped around the loop using a suitable pump, generally a positive displacement pump or concrete, double piston type pump. Depending on the size of the loop, roughly 6 m³ to 9 m³ of paste is required.

Starting with a low slump material, approximately 6" to 7", the paste is pumped around the loop at a range of flow rates. The low slump material can then be mixed with additional water to increase the slump to roughly 10" to 11" for a second series of pressure measurements. Flow aid additives may also be tested to determine their effect on pressure losses. The slump should be measured before and after each test and appropriate samples taken for particle size analyses and moisture content.

Flow data is adjusted for elevation changes in the surface loop and pump efficiency. Based on the test data, flow rate/pressure loss relationships can be determined for varying slump/moisture content pastes.

Gravity Flow Testing

The general experience with paste backfills has been that the flow resistance is considerably less for pastes transported under gravity flow than for pumped flow conditions. Therefore, while surface loop tests are very useful for determining pressure loss data for paste flow under varying flow conditions – pipe diameter, pipe material, and different flow rates – gravity flow tests more accurately reflect true operating conditions. Where the vertical:horizontal distribution ratios are unfavourable and preliminary investigations indicate that paste pipeline pressure losses may create a negative energy balance requiring pumping of paste backfills, gravity flow test work should definitely be conducted to define the pressure losses more accurately.

The drawbacks to gravity flow testing are that a large quantity of paste is required and the preparation of a pipeline and test stope is more expensive and time consuming. Underground gravity flow test work can be conducted in conjunction with pilot plant testing, which will produce adequate volumes of paste for gravity testing.

In gravity flow testing paste flow is monitored in a similar manner to surface loop testing, using pressure transducers uniformly spaced along the pipeline. Ideally, low and high slump pastes should be tested at varying flow rates to obtain slump/pressure loss relationships. Again, material sampling and slump cone testing is undertaken before and after each test.

4.8.3 Pilot Plant Test Work

Pilot plant testing is useful for evaluating and comparing different paste dewatering/preparation systems and for providing more accurate data for engineering design of paste backfill systems. Pilot plant testing should provide better operating data than laboratory or bench scale testing because the equipment tends to more accurately simulate full-scale operation. Variability in tailings material can be difficult to test on a lab scale. Because pilot plant test work generally involves larger quantities of material, there is less chance of introducing sampling errors to the test work.

4.8.4 In Situ Test Work

In situ test work provides feedback on backfill quality and performance after placement. Due to the high cost of in situ testing and a lack of robust, reliable instrumentation, this type of testing is not routinely undertaken. Backfill strength is one key focus of in situ testing. In delayed backfill situations where backfills have been identified to be susceptible to sulfate attack and degradation and where long-term stability of the fill may be of concern, monitoring and sampling of in situ backfill may be used to ensure that adequate backfill strengths are being maintained.

The challenge with in situ testing has been to find field instrumentation, which will minimize the disturbance of the material being tested and give accurate measurements of in-situ stress, stress-strain response and pore water pressures.

Accurate measurements are essential to determine the void ratio and degree of saturation of the in situ fill. Boreholes are often used for obtaining in situ measurements. Unfortunately, the borehole cavities may introduce error to measurements due to re-distribution of stresses. As well, the inside walls of the boreholes may be rough and uneven, which will affect measurements.

The collection of undisturbed samples is also challenging. Conventional diamond drill core recovery methods are generally not suitable because air and/or water used in drilling may be introduced to the samples. Techniques have been developed, which use modified double barrel coring machines. Core liners are inserted into the inner core barrel to confine samples during handling. Retrieved core samples can be sealed and frozen to minimize disturbance (Falconbridge, 1990).

Scoble, Piciacchia, and Robert (1987) reported on the successful adaptation of a soil mechanics tool, the Pencil pressuremeter, which they used in a test program involving in situ measurements of backfill in five Quebec mines. The pressuremeter can be inserted in a pre-drilled 35-mm diameter borehole or can be pushed into an intact backfill mass at depths of up to 10 metres. When used in conjunction with electronic piezocones or static cone penetrometers, a wide range of geomechanical data can be measured or derived, including modulus of deformation, shear modulus, limit pressure, angle of internal friction, pore water pressure, and strength.

In 1986 project work was conducted at Inco's Levack mine to determine the in situ support and fill properties of a high density backfill (Thibodeau, 1989). Laboratory and in situ test work was conducted to compare results. A range of analyses were made including particle size distribution, chemical composition, cement content, moisture content, bulk density, compressibility, and uniaxial compressive strength. The in situ instrumentation was placed in a mined out slice of an undercut and fill test stope prior to filling and included 36 electronic

total earth pressure cells, 3 electronic piezometers, 2 Young's modulus cells, 4 thermosistors, and 7 convergence monitors. Monitoring of the instrumentation was conducted while the undercut beneath the filled slice was mined out. Difficulties were encountered with calibration of the instrumentation and with temperature sensors and pressure transducers that failed to operate. Attempts to recover in situ samples through coring met with limited success and only 3 intact samples were recovered for testing. The compressive strength of the in situ samples was found to be 86% higher than the strength of the cast samples. Although the tests showed that the high density fill was competent throughout and provided very stable support, the difficulties encountered with the instrumentation showed the need for the development of more robust instrumentation.

5.0 PASTE CHARACTERISATION – A CASE STUDY

This case study is based on paste characterisation test work conducted at Boliden-Westmin's Myra Falls underground operation, which is located near the south end of Buttle Lake, inside Strathcona Provincial Park approximately 90 kilometres southwest of Campbell River, British Columbia. The author participated in the test programs that were conducted over the summer of 1998, in conjunction with mine personnel and paste consultants.

5.1 Case Study Background

This 3500 tonne per day base metal operation had been evaluating the feasibility of switching its conventional hydraulic slurry backfill system to paste backfill. Mining of the volcanogenic massive sulfide deposits at this operation commenced in 1966. The mine produces copper and zinc concentrates with significant gold and silver values. Current geological ore reserves (proven, probable, and possible) are in excess of ten years.

The mining methods used at the operation include room and pillar mining and sublevel retreat longhole stoping, as well as some cut and fill. Two main production areas -- the Battle-Gap and the H-W zones -- located approximately 2 kilometres apart and at different elevations, are currently being exploited (Figure 5.1). These production areas have different ore and host rock assemblages with highly variable grade. Consequently, grade control has proven to be a significant challenge for the geology and mining departments. An integrated grade control system has been developed to facilitate blending of ore and to monitor production (Bakker and Sawyer, 1999). Downstream, this variation in ore grade has been equally challenging in the processing plant. From the outset, the variability in mineralogy had been identified as an important parameter to be considered in the characterisation test programs, due to its potential effect on paste quality and performance.

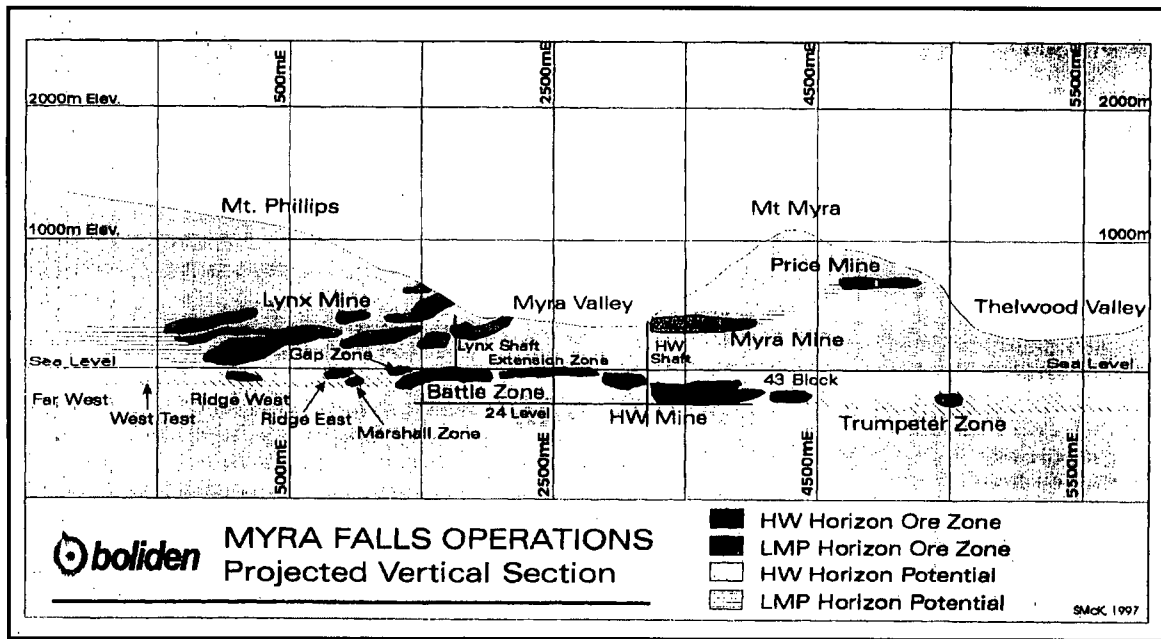


Figure 5.1 Longitudinal Section - Myra Falls Ore Zones (Bakker and Sawyer, 1999)

One of the driving forces for a paste backfill system at this operation is the improved geotechnical attributes of paste tailings for construction of the surface tailings management facilities. The tailings are deposited sub-aerially in thin layers using spray bars. The solids settle with water draining to a decant pond. (Figure 5.2). The existing surface tailings disposal facility, is approaching its design capacity and the conversion to paste tailings disposal offers potential alternatives for long-term tailings management at the site

The concentrator produces approximately 3000 dry tonnes per day of tailings. Prior to 1999, the existing slurry backfill system was not supplying enough backfill for all of the mined out areas underground. The volume of classified tailings available for fill was approximately 40% of the tailings produced. The existing backfill system was placing roughly 1200 dry tonnes per day or one third of the daily mine production.

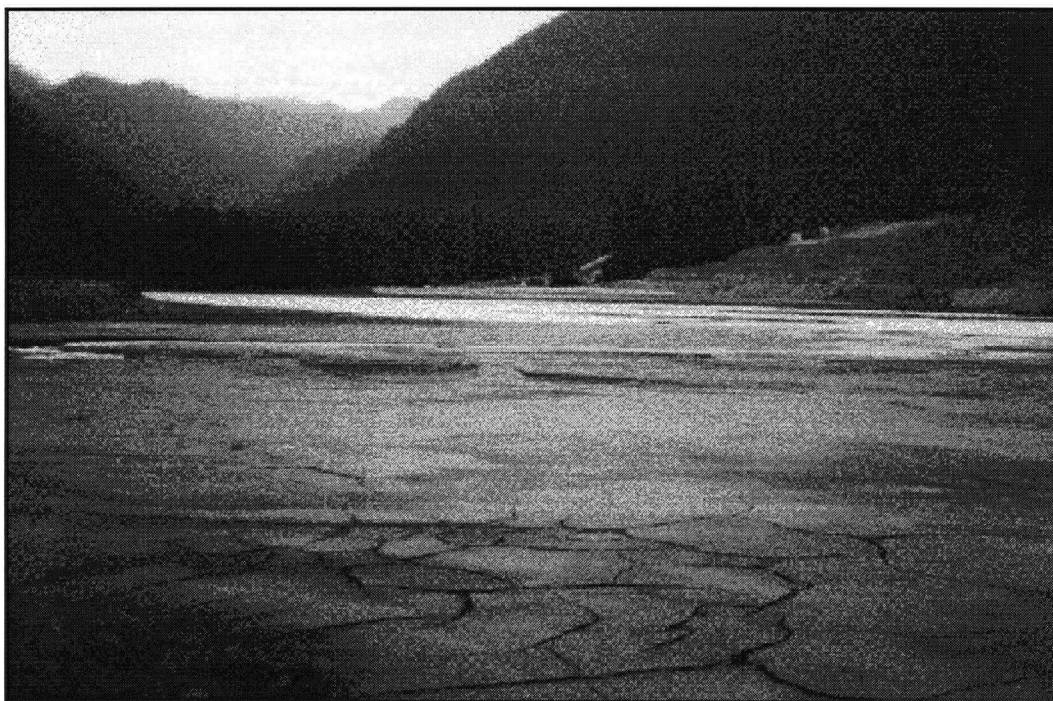


Figure 5.2 Myra Falls Surface Tailing Facility

A switch to a paste backfill system using full plant tailings would allow the operation to place up to 65% of its daily production in new and existing underground voids. In one possible scenario, the balance of paste produced would be deposited on surface behind cemented paste/waste rock berms in an existing mined-out open pit, eliminating the need for additional conventional tailings dam structures constructed of borrow materials. The use of this previously disturbed area for tailings disposal could reduce the requirement for further surface disturbance, resulting in a smaller overall surface footprint at this mining operation.

Subsequent to the paste characterisation test work, a number of changes have been made to improve the efficiency of the slurry backfill system. New hydraulic switch valves have been installed and the cyclone settings have been fine-tuned. Shotcrete bulkheads are being installed in place of conventional timber bulkheads. As a result of these changes, the recovery of cyclone underflow for

backfill has improved from 40% to 55%. Approximately 40% of the mine development waste is being used to supplement the slurry backfill. The mine now reports that they are targeting a maximum recovery of 60% from cyclone underflow, which will provide sufficient slurry backfill to fill the voids generated by new mining. As well, the mine has been reclaiming classified tailings by dredging one of their ponds to produce backfill for filling existing voids. The mine has been able to resolve the challenges it was facing with its underground backfilling system without implementing a new costly paste backfill system. The backfill system may eventually be converted to a paste backfill system, however, depending on future surface disposal requirements (Isagon, 1999).

Ground control conditions at this operation require the use of backfill for support in many areas. Where room and pillar and longhole stoping methods are employed, then delayed backfill is used. Some cut and fill mining is carried out in more steeply dipping, narrow zones of the ore lenses and additional cut and fill mining is currently planned in one of the main production areas.

In addition to the environmental/socio-economic advantages of using paste backfill, which could prevent premature closure of the operation (closure prior to the ore deposits having been fully exploited), there are many other potential advantages of using paste backfill at this operation. These advantages include shortened mining cycles, tighter fills, and reduced operating costs related to construction of barricades and collection and pumping of drainage water.

5.2 Case Study Paste Design Criteria

The paste backfill functions identified for this project were:

- Ground Support – Delayed backfill in longhole and room and pillar stopes
- Working Floor – Cyclical backfill in cut and fill mining stopes
- Waste Disposal – Underground and Surface

Based on these backfill functions the preliminary target design criteria that have been identified for paste backfill system design at this operation are:

- **Geomechanical Properties** – The paste backfill target strength will be dependent on the function of the backfill. Where paste is required to provide a working floor for cut and fill, backfill strength of 1 MPa would be required with a lower strength of 0.3 MPa to form the base beneath the immediate mucking floor. In primary stopes that will require a free-standing wall to allow mining of secondary stopes, the backfill strength will be dependent on the size of the stope. The stope size at this operation varies from 15 x 15 x 12 metres high to 15 x 18 x 30 metres high, with required backfill strengths consequently varying from 0.3MPa to 0.5MPa. For secondary stopes a plug of 0.3MPa strength is currently poured up to the brow height, with 0.1MPa for the balance of fill in the stope to provide sufficient liquefaction resistance.

For surface deposition using either a slope deposition or dry stacking method the paste must develop a certain minimal yield stress. The rheological behavior of the paste must be investigated to determine the suitability of the material to slope deposition. The mine wants to place paste on surface at the highest weight percent solids possible, in order to reduce the amount of surface disturbance required for tailings disposal. Where cemented paste is planned for berms, the strength requirements will also have to be determined in early test work.

- **Distribution and Placement** – Design capacity of the paste backfill plant would be 180 dtph. This design capacity was selected in order to optimize the mining cycle and to provide for an increased mining production rate in the future. The distribution system must be capable of transporting and placing paste underground over horizontal distances up to 2 kilometres from the plant, maximizing the use of gravity flow as opposed to pumping of paste backfill. The rheological trade-off between paste slump and required

strength/binder addition must be optimized. Low slump paste requires less binder to achieve target backfill strengths, but results in increased pressure losses in transportation and hence, higher costs associated with transportation. Similarly the surface deposition of paste also requires a trade-off between volume placed with cost efficiency of pumping over distances up to 1 kilometre, where gravity flow is not possible.

- **Environmental Performance** – The primary consideration for this project is to minimize the surface deposition of tailings by maximizing underground disposal of paste and by maximizing the solids density of the paste deposited on surface to reduce the volume of tailings. Surface deposition of paste will reduce the environmental impact of tailings deposited on surface.
- **Socio-Economic Performance** – The project would have a significant influence on the economy through employment (direct, indirect, and induced), purchase of goods and services (direct and indirect), and taxation. Closure of this operation will have a profound employment and income effect on the local and regional economy. The surface area available for tailings disposal limits the mine life. The capital and operating costs of introducing a paste backfill system that will extend the mine life by maximizing ore extraction underground and by reducing the surface waste disposal of tailings must be economically feasible.

Cement and power are significant operating cost components of the paste backfill system. Cement consumption must be minimized through the design of paste backfill that meets strength criteria and maintains rheological behavior that minimizes transportation and placement costs. Design of the paste backfill system at this operation must also take into consideration the cost of using diesel generated power. The mine currently supplements available hydroelectric power supply with diesel generated power. Any additional future power requirements will have to be met with higher cost

diesel generated power. The power consumption associated with de-watering equipment, in particular, will be an important consideration in the choice of de-watering equipment for this operation.

5.3 Case Study Paste Test Work

In order to evaluate the viability of switching to paste backfill, mine personnel in conjunction with consultants, have carried out a program of tailings characterisation test work to ensure that a consistent paste fill can be produced that will meet the identified geomechanical properties, distribution and placement parameters, environmental performance, and socio-economic performance design targets for the project.

The influence of variability in physical, chemical, and mechanical properties of the tailings on paste backfill quality has been a key facet of the characterisation work. Mineralogy, particle size distribution, and specific gravity are three significant components of tailings variability at this mine which will affect the paste backfill system design. Engineering design must incorporate the capability to compensate for tailings variability in order to provide a consistent paste backfill product.

Over a period of three years, a range of characterisation test work has been conducted on tailings material including laboratory analyses, bench scale testing, surface loop testing, and pilot plant test work. Preliminary studies of underground distribution and placement systems have been undertaken and consultants have also prepared preliminary capital and operating cost estimates on de-watering alternatives for the proposed paste backfill system.

Laboratory test work has consisted of particle size analysis, mineralogical analysis, strength characterisation, bench-scale sedimentation, filtration, and de-watering tests, triaxial compression test work, and rheological characterisation.

Two tank methods of producing paste – fluidization and deep cone thickening – have been pilot tested at the mine. In 1996 one of the two existing slurry backfill tanks at the mine was retro-fitted to conduct pilot test work using the CANMET proprietary fluidization method. The CANMET fluidization paste production method, marketed by MAG Engineering International, uses a batch process. Tailings are introduced through a feedwell at the top of a conical-bottomed silo, along with flocculant to aid in gravity settling if necessary. Compressed air is introduced through fluidization nozzles to facilitate densification and mixing of the paste. In conjunction with other physical and mechanical characterisation test work, paste preparation trials were conducted using the retro-fitted fluidization tank on site. Gravity flow tests to an underground stope were also carried out in 1996 using fluidized paste fill.

In May 1998, after the installation of a new feedwell in the fluidization tank, additional paste preparation trials were conducted. A surface loop was constructed on site for loop testing using paste prepared in the fluidization tank. Gravity flow tests to an underground stope were also carried out during the summer of 1998.

In August 1998 a pilot test program was conducted on site using an Eimco deep cone thickener pilot plant. The Eimco deep cone thickening system is designed to provide a continuous paste underflow. Tailings are fed into the top of the thickener with flocculant addition as necessary, using a patented E-Duc system and an expert control system to optimize settling and de-watering of tailings.

Pilot test work using the two different de-watering systems contributed to the database of paste quality and variability. The pilot test work also provided an opportunity for engineering and operating personnel to compare the performance of the alternative de-watering systems.

To complement the material characterisation efforts, test work was undertaken by the author, at the request of the mine. The author worked as a liaison between the mine and the paste consultants, conducting on site test work, collecting data for the preliminary design of the underground and surface distribution system, and assisting with the pilot test programs.

Results of the characterisation test work and discussion of the potential effect of tailings variability on paste backfill design for the case study project are presented in the following sections.

5.3.1 Particle Size Distribution

Particle size distribution is identified as being a very critical parameter for paste due to its effect on permeability, porosity, strength, and rheological behavior.

Samples were collected during the May 1998 paste trials and sent for analysis using a Model 2010 Galai particle size analyser at the CANMET laboratory in Sudbury, Ontario. The particle size distributions are shown in Figure 5.3. The minus 20-micron fraction ranged from 20.3% to 42.3% in the eighteen samples, with an average of 35.4% passing 20 micron. This would place the tailings at the lower range of medium size for paste classification using paste classification criteria (Landriault, 1995). Other particle size analyses conducted in 1995, 1996, and 1997 have been reported to range from 33% to 56% minus 20-micron.

Variability in the paste particle size distribution will affect the water content of the paste, which in turn will affect the rheological behavior of the paste, settling rates, and paste backfill strength. Because the colloidal particles affect the water retention properties of the paste, the water content of the paste will increase as

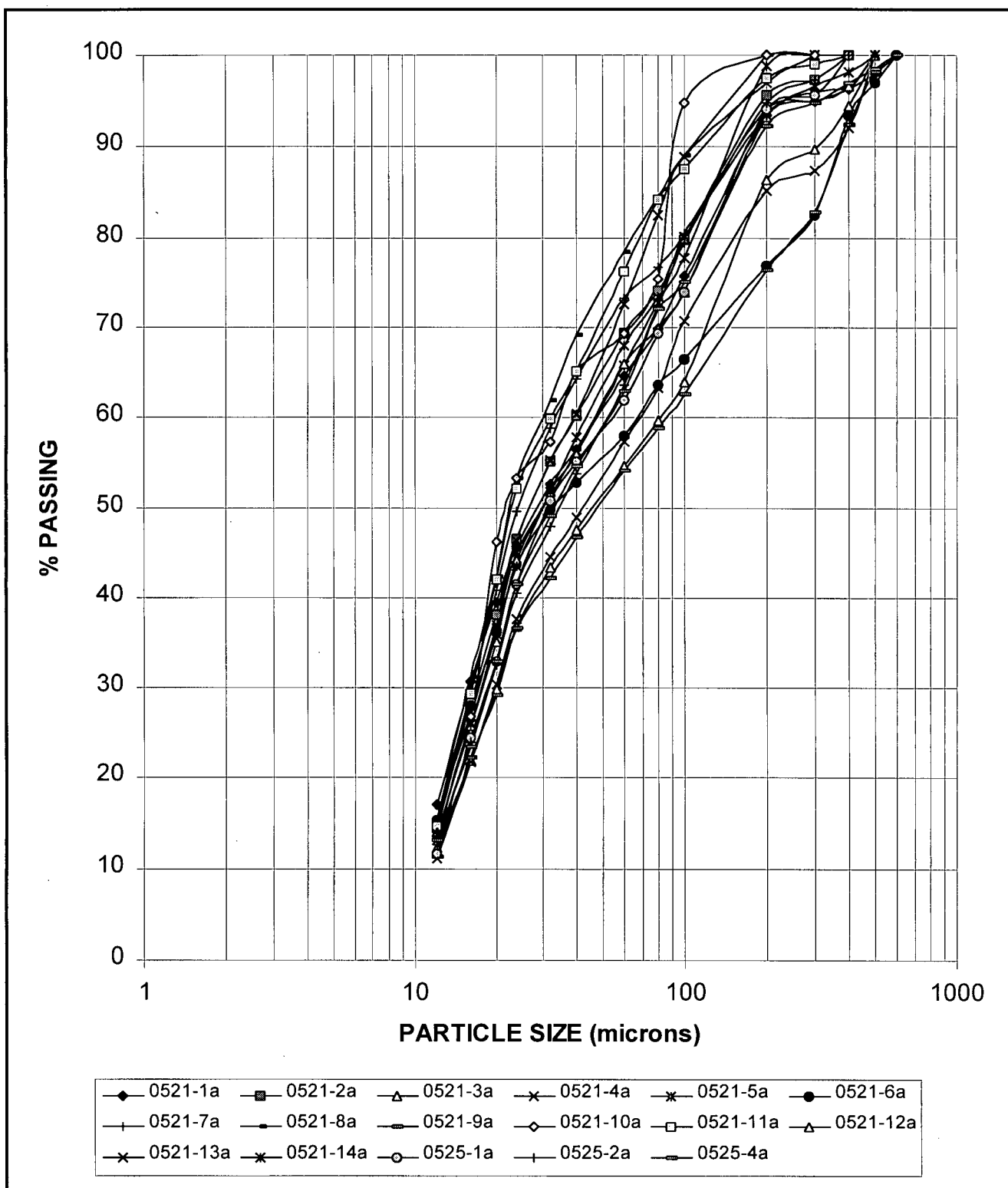


Figure 5.3 Particle Size Distributions (After Golder Paste Technology Ltd., July 1998)

the percent minus 20-micron material increases. This will have a significant effect on the paste backfill strength and the binder addition required.

5.3.2 Mineralogy

The high degree of variability in mineralogical/geochemical composition of the ore from the two distinct production areas of the mine is reflected in high variability in the composition of the tailings. Variability in mineralogy will influence the specific gravity and particle size distribution of the tailings, which can potentially affect the settling and filtration properties and the overall rheological behavior of the paste.

Mineralogical/chemical analyses using x-ray diffraction were conducted on eighteen samples collected during the course of the pilot test work in May 1998. The relative proportions of crystalline mineral assemblages (Table 5.1) show pyrite to be the major mineral present in the tailings samples. Pyrite was reported to be the dominant iron mineral in all of the samples, with some minor to trace iron in phyllosilicate and carbonate minerals.

Table 5.1 Crystalline Mineral Assemblages in Tailings Samples

Major	Moderate	Minor	Trace
Pyrite	Quartz	Mica, chlorite	Barite, calcite, dolomite

The identification of pyrite as a dominant mineral in the tailings may be significant to paste performance and quality, in particular, rheological properties and paste backfill strength. As discussed in section 4.6.2 some pyritic pastes have exhibited higher flow resistance than siliceous tailings pastes. The possible effect of pyrite on paste flow resistance may be significant when considering the potentially unfavourable vertical:horizontal ratio for paste distribution, both on surface and to some of the underground production areas at this operation. As pyrite is a reactive sulfide mineral, the environmental ramifications of placing

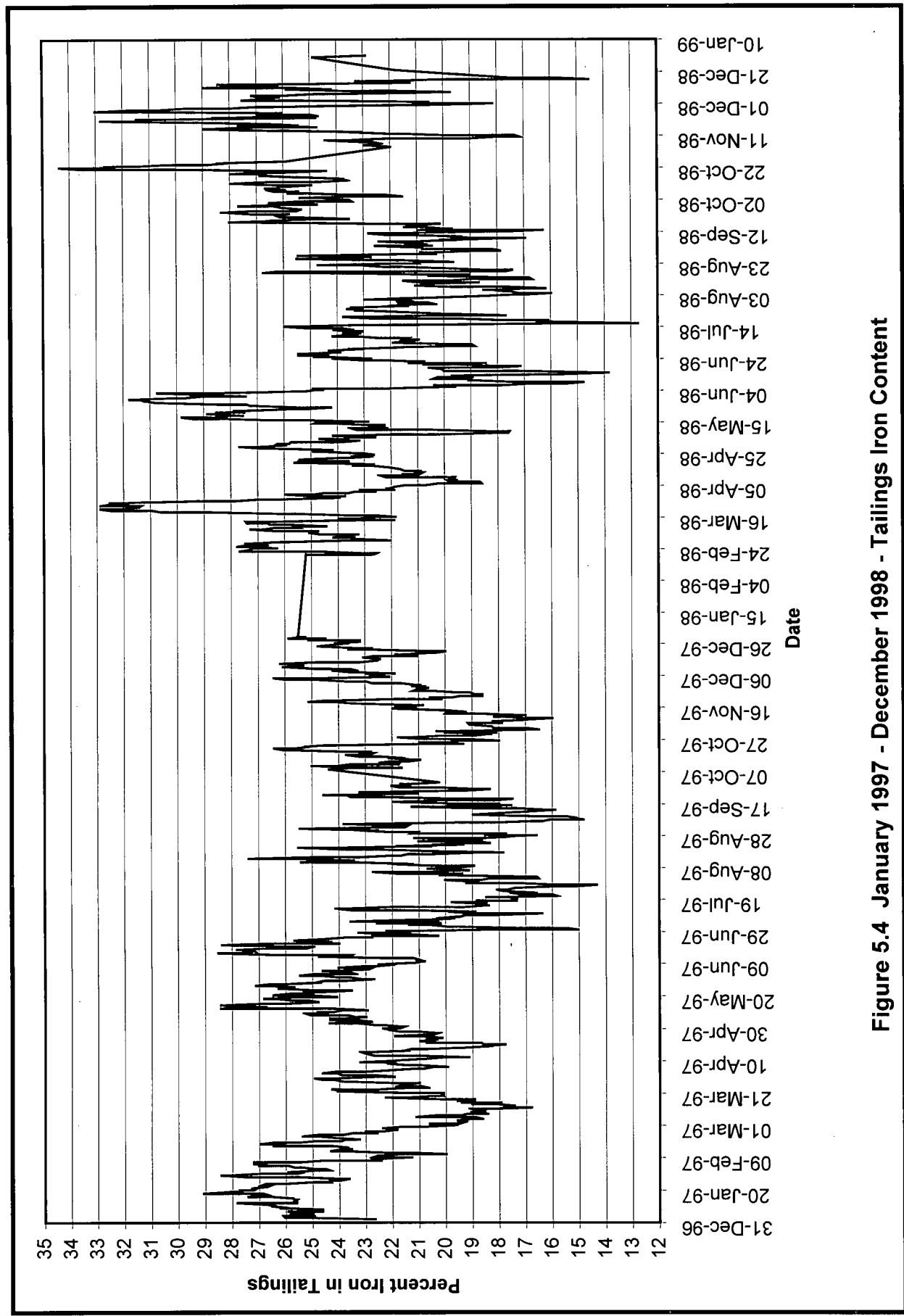
pyritic paste, as well as the influence on long-term backfill strength due to sulfate attack, are also important considerations that require investigation.

Tailings iron content and the 12-hour variance in tailings iron content has been extracted from the metallurgical assay data for the mine, for two full years, 1997 and 1998, to illustrate the degree of variability in mineralogical composition of the tailings. A plot of tailings iron content over a two year span from January 1997 through December 1998 is shown in Figure 5.4. The descriptive statistics and relative and cumulative frequency histograms for the tailings iron content and 12-hour variance are also presented in Table A-1 and Figures A-1 and A-2 in the Appendix. Both the two-year iron content of the tailings and the 12-hour variance in iron content are normally distributed. Over the two-year period, the iron content in the tailings ranged from a minimum of 12.7% to a maximum of 34.4% with a mean iron content of 22.8%. The standard deviation for the tailings iron content is 3.38%. Plots of tailings iron content and 12-hour variance in tailings iron content on a monthly basis from January 1997 through to December 1998 are presented in the Appendix (Figures A-3 through A-50).

Sample plots of tailings iron content and 12-hour variance in tailings iron content for the month of August 1998 are presented in Figures 5.5 and 5.6. The August 1998 plots have been included to illustrate the variability in mineralogy that occurred during one of the pilot test programs which the author was involved in.

A notable 12-hour variance in iron content of the tailings occurred at the end of one of the pilot tests with the Eimco deep cone thickener. The iron content of the tailings dropped from 26.25% over the 12-hour day shift period to 17.7% over the 12-hour night shift period. The 12-hour iron assay for the following day shift remained low at 17.4%. On that particular day, August 18, 1998, feed to the pilot tank commenced at 8 a.m., was shutdown briefly between 6:30 p.m. and 7:30 p.m. and was then shutdown for the night at 9 p.m. The sharp drop in iron content of the tailings did not have a noticeable effect on tank performance, as

there was a limited volume of feed to the tank during the night shift time period, relative to the total tank volume, and the bed level had already been established. Regrettably, it was a missed opportunity to observe the effect of a large variance in mineralogy on paste quality.



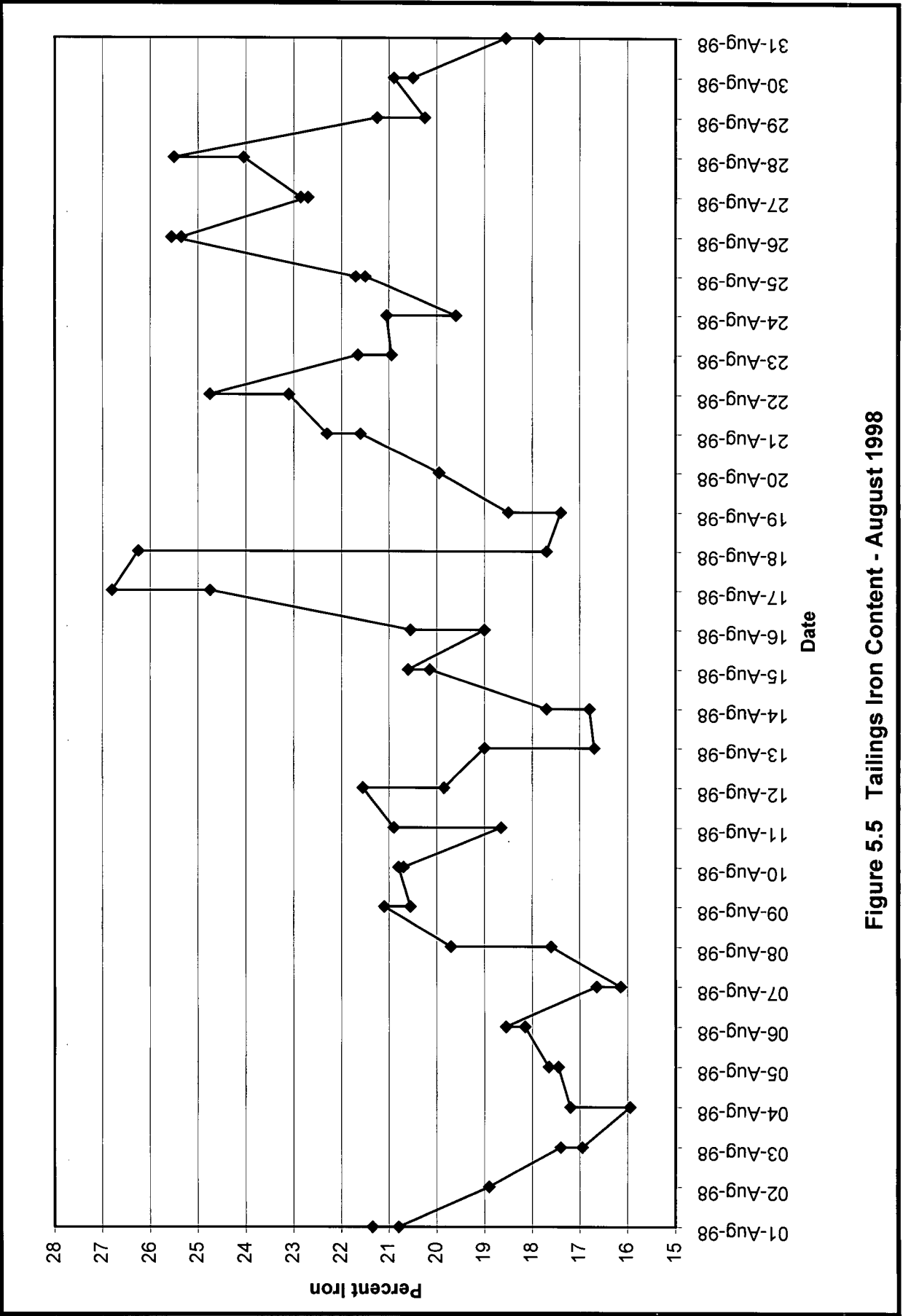


Figure 5.5 Tailings Iron Content - August 1998

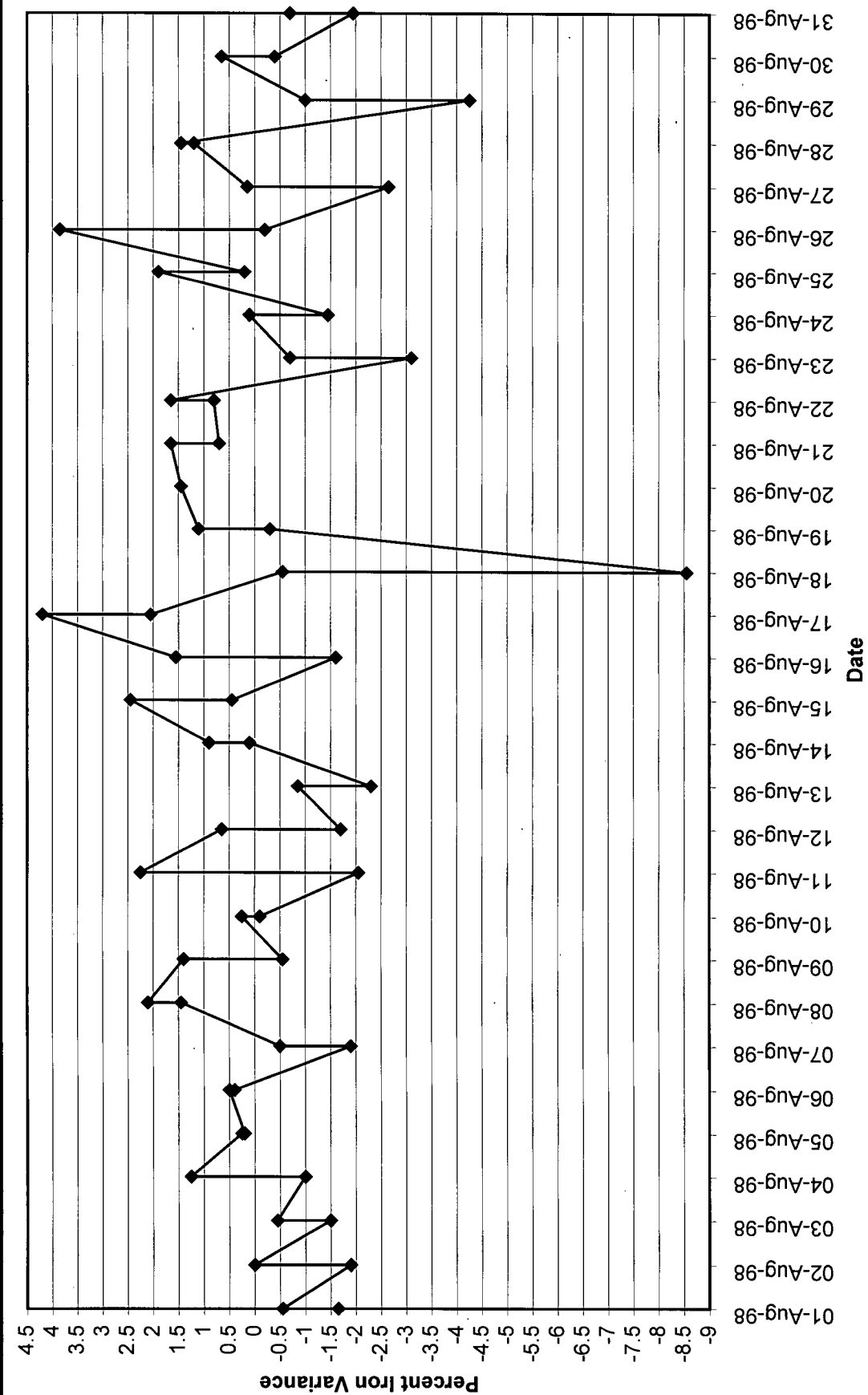


Figure 5.6 12-Hour Variance in Tailings Iron Content - August 1998

5.3.3 Specific Gravity

The variability in mineralogy affects the specific gravity of the tailings, which in turn will affect the weight of water and binder that must be mixed to provide a uniform paste product. Specific gravity measurements of the eighteen samples collected during the May 1998 test program ranged from 3.3 to 3.9, averaging 3.6. This is consistent with previously reported specific gravity analyses. The mine reports that variability in specific gravity can range from 3.2 to over 4.0.

Fluctuations in specific gravity can have serious implications when the design of the paste backfill system incorporates a continuous mixing system. The system is usually designed to provide a consistent slump product through calibration of the slump versus power draw of the mixer. This relationship is based on a particular average specific gravity. At a given specific gravity there will be a given slump/pulp density relationship. If there is a decrease in the specific gravity of the tailings, then the weight of the solids will reduce the power draw, which is equivalent to an increase in slump. The process control system then compensates by decreasing the water addition, which, with no change in the cement feed rate addition, will produce a lower slump material. Transportation of the lower slump material can potentially result in higher pressure losses and plugged lines. In addition, if the cement addition is constant, then at the lower water:cement ratios, the paste backfill strength will be higher than required, resulting in "wasted" cement, which is a high cost component of backfill.

The opposite scenario occurs if the specific gravity of the tailings increases significantly. The power draw on the mixer increases, equivalent to a lower slump. The control system compensates by adding more water, which decreases the pulp density and the mixer produces a lower strength paste backfill if the cement addition is constant.

The paste backfill system can be designed to account for the potential problems caused by variation in specific gravity, by using a batch mixing system, which allows for a volume/weight measurement to be translated into a specific gravity calculation. The process control system is calibrated to determine the appropriate water:cement addition for the measured specific gravity of the tailings in order to produce the consistent slump paste. On the basis of the high variability in specific gravity at this particular case study operation, paste consultants have recommended the use of a batch mixing system. They have also recommended that a filter be incorporated into the paste preparation circuit in order to further de-water approximately 50% of the Eimco thickener underflow, which would then be re-mixed with the balance of the thickener underflow, to provide a consistent, higher solids content paste.

5.3.4 Paste Backfill Strength

Unconfined compressive strength (UCS) test work has been conducted on paste backfill samples prepared both on-site by mine personnel and at off-site testing facilities by independent consultants. Figure 5.7 shows some of the on-site paste cylinder preparation work. As outlined in section 5.2, the mine currently requires a maximum strength of 0.5 MPa in primary stopes and 1 MPa for cut and fill working floors, with lower strengths for smaller primary stopes, secondary stopes, and for the base under cut and fill floors.

All UCS test work has been conducted using 100% ordinary Portland cement (Type 10) as the binder. Figure 5.8 presents a combination of 28-day UCS results from several different test programs. The results of the different programs should not be directly compared because the initial moisture content of the tailings differed.

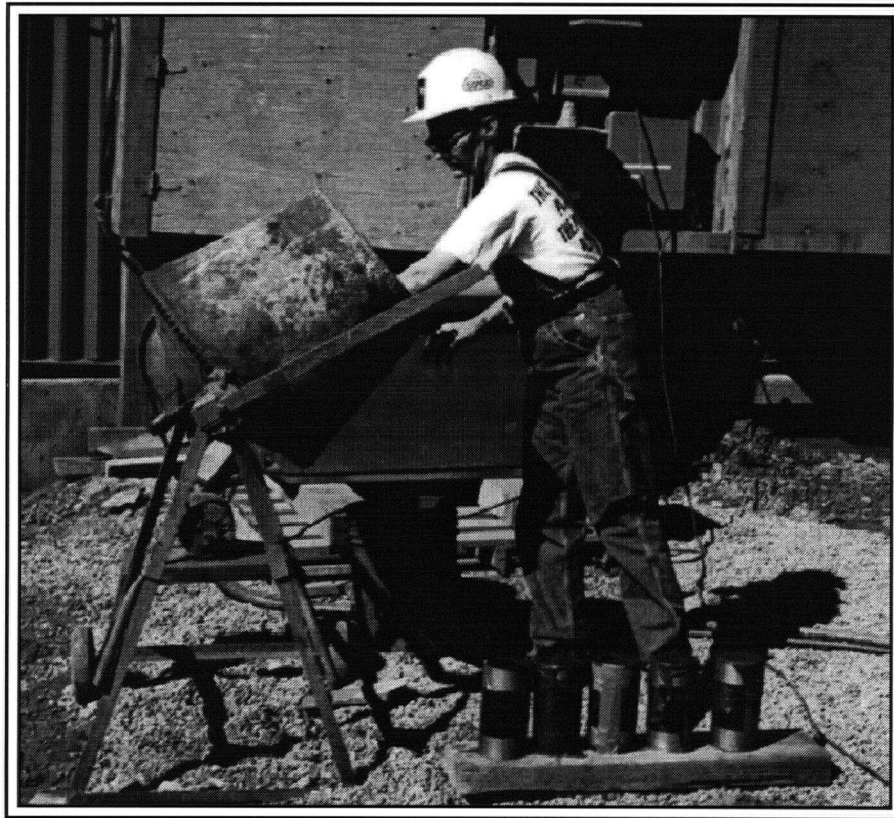


Figure 5.7 Casting Paste Cylinders for UCS Test Work – August 1998

The individual test programs do, however, show the effect of moisture content and weight percent of cement addition on the paste backfill strength. A test program was conducted in 1998 using weight percent cement additions of 2%, 4% and 6% with initial tailings solids weight percents of 73%, 76%, and 79%. The resulting 28-day strengths are presented in Table 5.2 and the three series are plotted as solid circles in Figure 5.8.

Table 5.2 Paste Case Study UCS Test Work – 28-Day Strengths

Weight % Cement	Slump (inches)	Weight % Solids	28-Day UCS (MPa)
2	> 11	73	0.10
2	~ 10	76	0.14
2	~ 7	79	0.21
4	> 11	73	0.21
4	~ 10	76	0.39
4	~ 7	79	0.52
6	> 11	73	0.49
6	~ 10	76	0.77
6	~ 7	79	1.04

At a constant weight percent cement addition, the 28-day strengths of the higher water content pastes (73% solids) range from 40% to 48% lower than the strengths of the lower water content pastes (79% solids). As cement is a high cost component of the operating cost for paste backfill, there is an obvious advantage to ensuring that the paste consistently meets a low slump/high solids density standard, in order to achieve the required backfill strength at the lowest cost in terms of binder addition.

A series of 4% cement samples at 73%, 76%, and 79% solids were also tested for 56-day and 90-day UCS. The results are shown in Table 5.3 and plotted in Figure 5.9.

Table 5.3 UCS – 4% Cement – Strength vs. Curing Time

Weight % Solids	28-Day UCS (MPa)	56-Day UCS (MPa)	90-Day UCS (MPa)
73	0.21	0.20	0.24
76	0.39	0.42	0.43
79	0.52	0.50	0.53

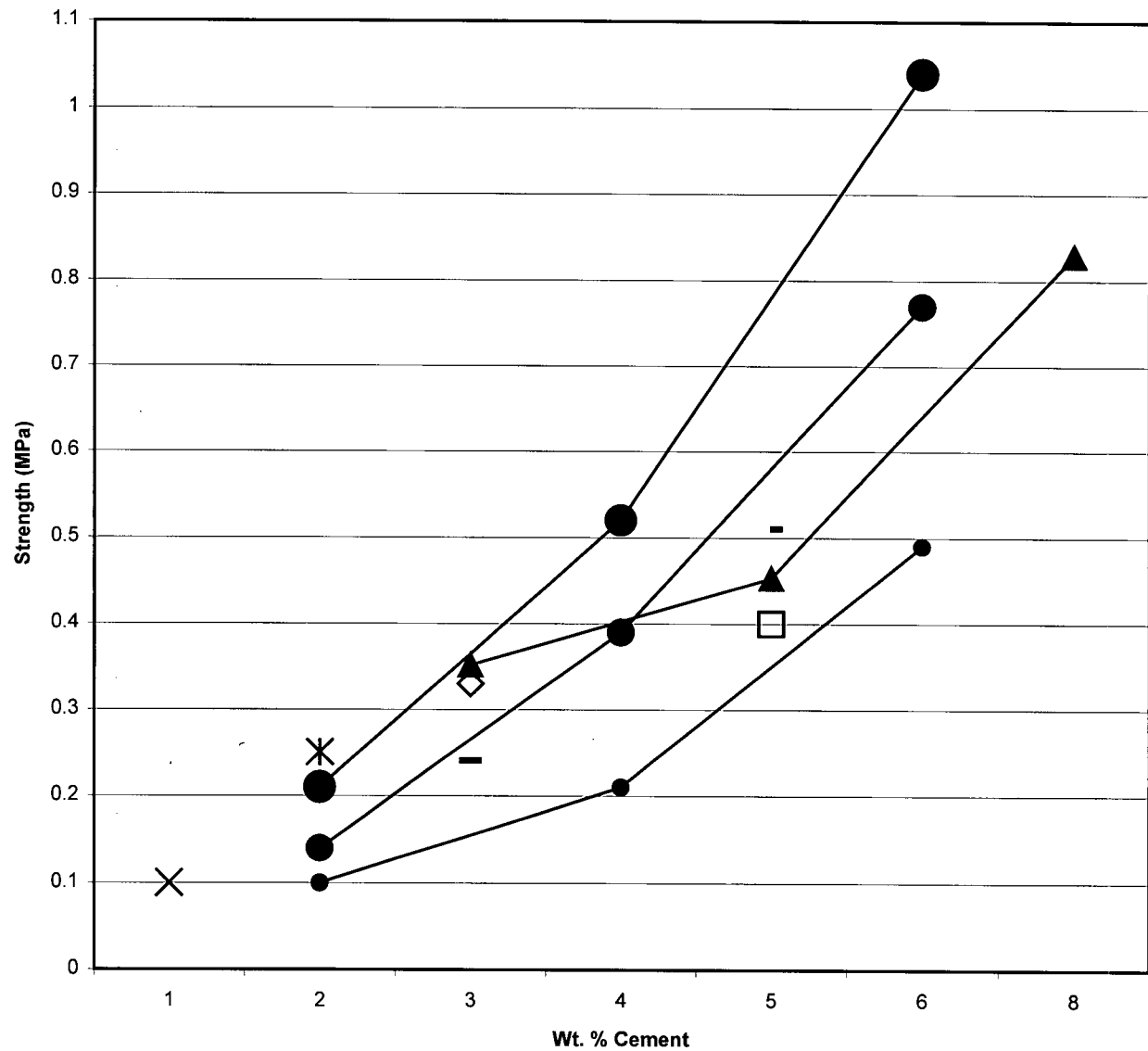


Figure 5.8 Unconfined Compressive Strength -- Paste Case Study Test Work -- 28-Day Strengths

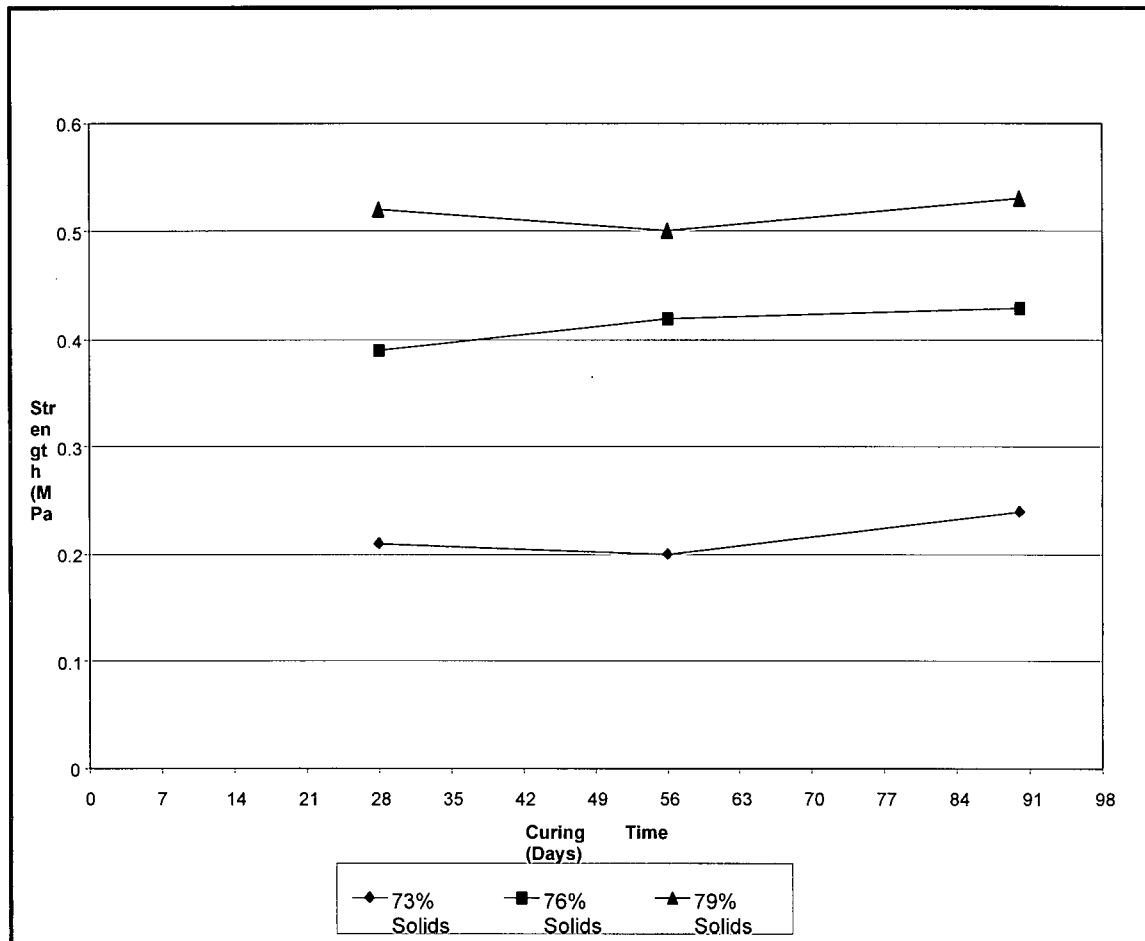


Figure 5.9 Effect of Curing Time on UCS – 4% Cement – 1998

The 28-day strengths vary from 88% to 98% of the 90-day strengths, which would be expected with pure Portland cement binder. No longer-term UCS testing beyond the 90-day samples has been conducted. It would be useful for the mine to investigate the potential effects of sulfate attack, which are discussed in Section 4.7.1. Based on the experience of other paste backfill operations using sulfide tailings, it would be advisable that long term UCS test work be conducted to confirm that target paste backfill strengths can still be met for the delayed backfill applications at the case study mine.

Additional cylinders were cast using paste produced during the Eimco deep cone thickener pilot plant test program in August 1998. Samples were prepared with

3% and 5% cement addition and were shipped off-site for UCS testing. The results are shown in Table 5.4 along with the data for 4% cement samples cast at 73% solids in the July 1998 test series conducted by paste consultants. The data from the two different tests is consistent. The 5% August sample, however, returned a relatively high 28-day strength of 0.51 MPa, which then dropped by 25% after 56-days curing time. As the August samples were cast on-site and then shipped out to the test lab, the curing conditions would not have been consistent with the standardized procedures used with the other tests conducted off-site by paste consultants and, therefore, it would be difficult to draw any direct conclusions from the test results.

Table 5.4 UCS Paste Tests – July and August 1998

Sample Description	28-Day UCS (MPa)	56-Day UCS (MPa)
3% Cement – 71.9% Solids – August 1998	0.24	0.23
4% Cement – 73% Solids – July 1998	0.21	0.20
5% Cement – 72.4% Solids – August 1998	0.51	0.38

5.3.5 Triaxial Compressive Strength

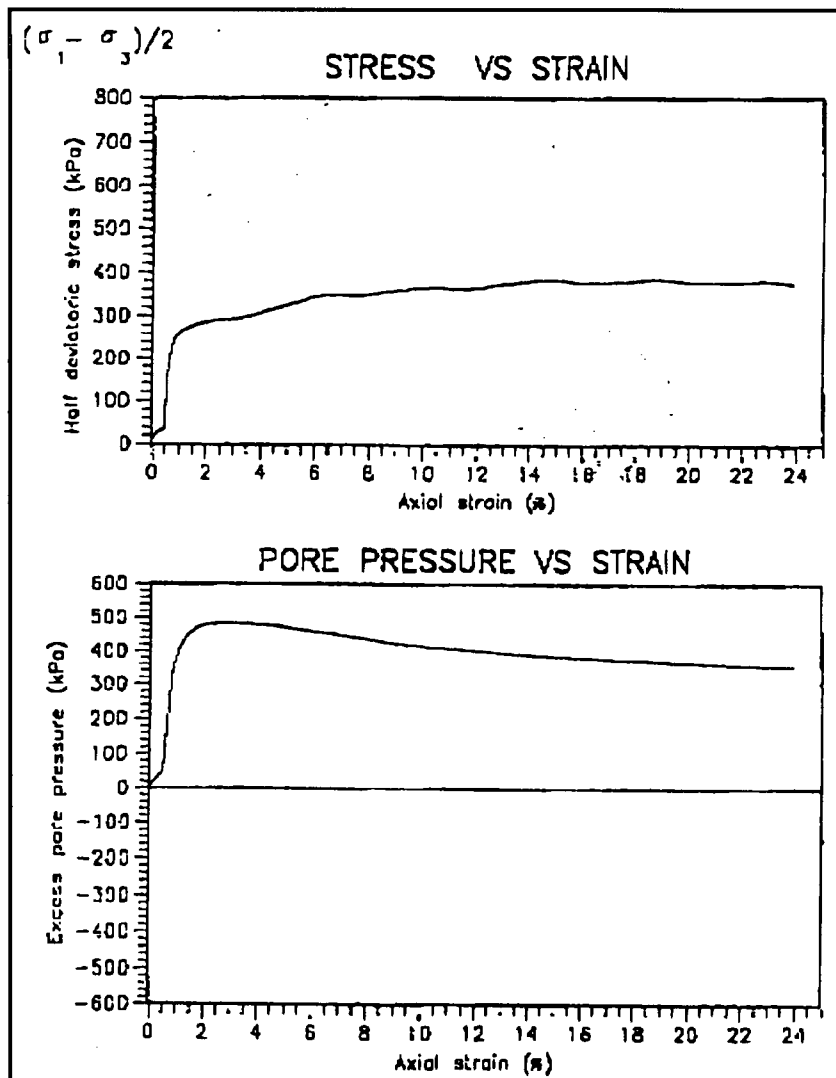
A series of undrained triaxial compressive tests were conducted in 1996 on both uncemented and cemented paste fill samples, using confining pressures of 207 kPa to 828 kPa. The uncemented samples cast at 87% solids, representing unconsolidated paste fill, were tested after a curing period of 7 days. The cemented samples, which were cast with 5% cement at an 8 inch slump, were tested at 14 and 28 days. Pore pressure measurements were not taken. The resulting friction angles and cohesion obtained are shown in Table 5.5.

Table 5.5 Triaxial Compressive Strength – 1996

Sample Description	Friction Angle (ϕ)	Cohesion c (kPa)
Uncemented – 7 Days Curing – 13% Moisture	28° - 29°	241 – 310
Cemented – 5% - 14 Days Curing	29° - 31°	483 – 621
Cemented – 5% - 28 Days Curing	27° - 32°	379 – 552

The cohesion would be expected to increase with binder content and curing time. Cohesion was higher in the cemented samples than the uncemented samples, as expected. However, the cohesion did not improve with curing time in the cemented samples. In addition, the cohesion of the uncemented sample was relatively high compared to the cemented samples. The low initial moisture content of the uncemented sample may have been a factor in this result. The effect of binder content was not tested.

In 1998 single stage triaxial undrained compressive tests were conducted on a cemented and an uncemented sample at 600 kPa confining pressure. No Mohr failure envelopes were developed to determine friction angles and cohesion. The cemented (3%) sample cast at 80% solids had been cured for longer than 28 days. For the cemented sample, the deviator stress climbed steeply and almost linearly initially and the slope then flattened and increased very slowly to a peak half deviator stress of approximately 300 kPa (Figure 5.10). The corresponding pore pressure curve also climbed steeply at first and then slowly decreased (Figure 5.10). As positive pore pressure was generated, the undrained shear strength would be less than the drained shear strength. The uncemented specimen showed an initial steep increase in half deviator stress that was approximately one half the level of the cemented specimen and then increased slowly at a steeper slope than for the cemented sample. The pore pressure curve generated from the uncemented specimen was similar to that of the cemented specimen, however the pore pressure was lower.



**Figure 5.10 Single Stage Undrained Triaxial Test of Cemented Sample
(After Kloth-Crippen, 1998)**

5.3.6 Sedimentation and Filtration Test Work

In conjunction with the pilot test programs conducted at the mine, a number of sedimentation and filtration tests have been conducted by consultants or equipment/reagent suppliers. All of the test results have indicated that the tailings from the case study operation have good settling properties, mainly due to the high specific gravity.

Settling tests conducted in 1996 using a starting pulp density of 30% with full plant tailings gave final densities after 24 hours ranging from 71% to 77%. Settling tests using flocculant were also conducted by two independent suppliers in 1998 and returned settling rates in the range of 0.3 to 0.4 cm/s. These tests formed the basis for determining initial flocculant dosages for the pilot test work. During the course of both pilot programs conducted in 1998, it was found that flocculant dosages could be reduced from the initial predicted dosages. In the tank de-watering/thickening methods, the starting density of the feed is an important consideration for optimizing the flocculant aided settling rate.

The results of bench scale paste production tests conducted by two different equipment suppliers in 1998 were very similar. These tests were conducted on tailings samples drawn at the same time and shipped from the mine to the suppliers' testing facilities. One supplier reported producing a consistent paste in the 74% to 78% solids range, while the second reported a consistent output of 76% solids paste.

Bench scale thickening and filtration tests were also carried out on tailing samples in order to provide the basis for costing of a conventional thickener/filtration method of producing paste as one of the options for the paste system design for the operation.

5.3.7 Surface Loop and Gravity Flow Test Work

Surface loop pump tests and gravity flow tests are essential to determine the production scale flow parameters for design of a paste backfill system. A surface loop test requires a minimal amount of paste material (6 m^3 to 9 m^3) and can be constructed to test different pipe diameters and materials. A gravity flow test requires a significantly larger amount of paste and is more costly in terms of preparation and time. In order to conduct a gravity flow test, a pilot scale paste

production plant must be constructed. The gravity flow test will, however, produce more accurate pressure loss data.

The retrofit of one tank at the mine's existing backfill plant to produce paste with a fluidization method has permitted several gravity test flow programs to be conducted since 1996. In August 1998 a mobile pilot deep cone thickener plant was trucked to the mine site and used for pilot testing of a continuous paste production method. A surface loop was constructed on site in 1998 for surface flow loop tests.

Surface Loop Test Work

In May 1998 a surface loop was constructed on site using sections of 200 mm (8") Schedule 40 steel pipe, 150 mm (6") Schedule 10 steel pipe, and 200 mm DR9 rated HDPE pipe. A truck-mounted Schwing positive displacement pump Model 1200 HDR was used to pump material around the loop. Four pressure transducers were mounted into spools between sections of the pipe. The pressure transducers were connected in parallel with standard 3-core wire to a laptop computer to record all of the test data. Paste tailings prepared in the backfill fluidization tank were pumped into a ready-mix truck for transfer to the loop test site. Photographs of the surface loop and one of the pressure transducers are shown in Figures 5.11 and 5.12.

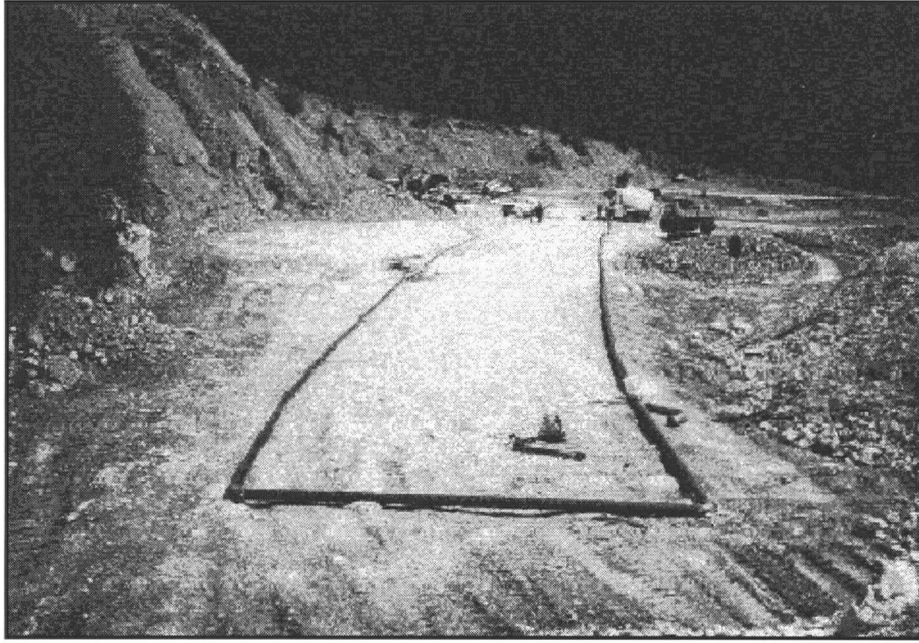


Figure 5.11 Surface Loop



Figure 5.12 Pressure Transducer on Surface Loop

Ideally, a surface loop test program would involve trials of paste material at two different slump consistencies to compare the flow rate/pressure loss relationship of the material at varying slump. A low slump material (7") would first be tested. Mine water could then be added to the material in the pump hopper to increase the slump for a second trial at a higher slump (10"). In the case study trial, the starting slump of the paste material obtained from the fluidization tank was 9.5". Therefore, the surface loop test was only conducted at the one slump consistency.

Tests were conducted over a full range of flow rates. In addition a final test was conducted to evaluate the effect of a flow-aid admixture at two different dosages.

The pressure losses for the target capacity of 180 dtph at different pipe material and diameters are shown in Table 5.6. The pressure losses in the steel pipe were found to be very consistent, increasing slowly as flow rate increased. The initial pressure losses in the HDPE pipe were lower than for the steel pipe, however, the pressure losses increased as flow rate increased.

Table 5.6 Surface Flow Loop Pressure Losses for 9.5" Slump Paste

Flow Rate	Pressure Loss		
	200 mm (8") Steel Pipe	150 mm (6") Steel Pipe	150 mm (6") HDPE
(dtph)	(kPa/m)	(kPa/m)	(kPa/m)
180	4.0	5.0	6.3

As the operation currently uses HDPE pipe in its slurry backfill distribution system, it would be advantageous to retain this pipe where possible for the proposed paste distribution system. Unfortunately the surface loop test results indicated that the use of HDPE pipe may be limited, depending on the horizontal distances paste backfill will have to be transported underground.

The addition of a flow-aid admixture showed an insignificant decrease in pressure losses. As the paste slump being tested was relatively fluid at 9.5", the effect of the admixture was not likely to be as pronounced as might have been with a lower slump material. Further testing with admixtures would be justifiable if the paste backfill system is designed to produce 7" slump material, due to the higher pressure losses that will be expected for long transportation distances. To this end, a supplier is currently testing flow-aid mixtures with low slump pastes for the mine.

Gravity Flow Test Work

The existing slurry backfill system at the operation uses three uncased, 113 mm (4.5") diameter, inclined boreholes to different levels. The 1998 gravity flow tests were conducted using the longest borehole, a distance of approximately 530 metres at an 87° incline. At the base of the borehole a section of 100 mm (4") steel pipe connects to a section of 100 mm diameter HDPE pipe, which is then connected to 150 mm (6") HDPE pipe. A pressure transducer is located near the base of the borehole to monitor the initial pressure and to ensure that the flow pressure does not exceed the 1.4 MPa (200 psi) rating for the HDPE pipe. Three additional pressure transducers were installed at roughly 100 metre intervals along a relatively flat section of the HDPE pipeline to the test stope.

Because the pumping and mix system used for the existing slurry backfill system is not suitable for paste backfill, the gravity trials were conducted with uncemented paste produced in the fluidization tank and sent directly down the borehole. Under the direction of paste consultants, gravity flow tests were conducted on two separate batches of paste prepared in the fluidization tank during May 1998. Additional gravity test pours were conducted during the summer of 1998 by the author and mine personnel.

As with surface loop pump tests, the slump of the paste material should be low (7") in order to evaluate the "high end" pressure losses that might occur. Unfortunately, the slump consistency of the fluidized paste was considerably higher than desired for the test, varying between 10.5" and 11.5" during the first test and 10.5" for the second test.

The 11.5" slump test results were indicative of flow behavior of high density slurry rather than paste. Pressure losses tended to increase exponentially with increasing flow rate. The 10.5" material produced results that were more indicative of plug flow behavior. Pressure losses were higher for the lower slump material, as would be expected.

Based on the gravity flow tests, the pressure losses for the gravity driven transportation of paste backfill were estimated to be 3.5 kPa/m for a 10.5" slump paste consistency. Paste consultants have indicated from prior experience that pressure losses for a 7" slump paste are generally between 1.9 and 2.6 times as large as pressure losses for 10" slump paste. Using this guideline, the operation might expect maximum pressure losses of 8 kPa/m for 150-mm diameter pipe in a gravity distribution system.

The pressure loss parameters established for preliminary design purposes, based on a 180 dtph capacity plant are shown in Table 5.7.

Table 5.7 Pipeline Pressure Losses for 7" and 10" Slump Pastes

	7" Slump Paste	10" Slump Paste
Weight Percent Solids	79%	76%
Pumped Pressure Loss at 180 dtph	9 kPa/m	4 kPa/m
Gravity Flow Pressure Loss at 180 dtph	6 kPa/m	2.7 kPa/m

Using these pressure loss parameters, requirements for a paste distribution system to the two production areas underground have been evaluated. A paste distribution system will require the installation of steel pipe, as the line pressures

will be too high to use the existing HDPE pipelines. Delivery of a 9.5" to 10" slump paste backfill to the production areas can be achieved with a gravity flow distribution system. It is also possible to deliver a 7" slump paste using gravity flow to most of the mining areas. The slump will degrade to a higher consistency (9" to 10") by the time it reaches the farthest zones. For mining zones and potential new developments at the greatest distance from the surface plant, it may be necessary to use admixtures to assist the paste distribution.

Distribution of paste for surface disposal will require pumping with positive displacement pumps, due to the long horizontal distances with negligible gravity assistance possible.

5.3.8 Pilot Plant Test Work

As discussed in section 5.3.1, two different paste production systems – fluidization and deep cone thickening – have been used in pilot tests to investigate potential paste performance at the Myra Falls operation.

The objectives of pilot testing were to verify the capability of producing a consistent paste backfill material from the full plant tailings and to obtain more accurate performance data than could be achieved using bench scale testing, in order to improve the accuracy of preliminary engineering design and cost estimates. The two de-watering methods pilot tested are relatively new technological applications to paste backfill design and have limited full-scale experience. These new technologies offer more risk compared to conventional thickening/filtration systems used in the majority of paste backfill operations. Pilot testing reduces the risk by verifying the capability of the technology to produce a paste product that meets the operation's specific target criteria.

Both of the pilot plant programs have confirmed that paste with a solids weight percent in the 72% to 78% range can be produced from the concentrator full plant tailings.

Fluidization Pilot Test Work

The retro-fitted fluidization tank using CANMET technology provided paste material for all of the gravity flow test programs conducted at the case study operation. A photo of the inside of the fluidization tank showing the fluidization nozzle rings is shown in Figure 5.13. The test work conducted during the spring and summer of 1998 indicated that the paste produced in the fluidization tank would achieve a maximum solids density of 74%, although earlier (1996) test work conducted in 1996 had produced higher density pastes closer to 80% solids.



Figure 5.13 Inside View of Retro-fitted Fluidization Tank

The design of the ideal fluidization silo requires a conical bottom and a roughly 2:1 vertical:horizontal geometry. The retro-fitted tank was flat-bottomed and had a 1:1 geometry, which may have influenced the ability of the tank to optimize the fluidization process. As well, the discharge was located at the outside wall of the tank, as opposed to a central discharge. During the gravity flow testing, it was often difficult to initiate flow. Some rat-holing of material occurred. Again, the geometrical configuration of the tank have contributed to this problem.

Deep Cone Thickener Pilot Test Work

A 4.6 metre high, 1.5 metre diameter Eimco pilot deep cone thickener was shipped to the mine site and set up for a two week pilot test in August 1998 (Figure 5.14). At a bed height of 2.7 metres, approximately 10 tonnes of paste was contained in the thickener.

Feed to the thickener was drawn directly from a feed launder receiving the concentrator tailings at the cyclone plant. At the time of the pilot program, the mill was only operating on one circuit or 50% of production. As a result the tailings percent solids was reduced to between 12% and 18% from the normally reported 22% solids. Earlier settling tests had indicated that flocculated settling would be optimized at 11% to 12% feed solids. Consequently, the pilot plant E-Duc system was not required for feed dilution. Flocculant was drawn directly from the feed tank in the cyclone plant and metered into dilution water to provide the appropriate dosage to be mixed with the feed slurry.



Figure 5.14 Eimco Deep Cone Thickener Pilot Plant

The pilot tests showed that the tailings could be thickened to a paste consistency between 72% and 78% solids. Yield stress data obtained from thickener underflow samples was determined using a Bohlin Visco 88 viscometer and a rheogram was generated of the yield stress data at different weight percent solids. Based on the rheological data obtained from the test samples, an operating range of 70% to 75% solids has been suggested for a full size thickener.

5.4 Significance of Characterisation Test Data

Over a three-year time period, the Myra Falls operation has undertaken a characterisation program to help determine the feasibility of switching its existing slurry backfill system to a paste backfill system. The main objective of the characterisation test work has been to evaluate the quality and performance of paste produced from full plant tailings at the operation. Physical, chemical, and mechanical characteristics of the tailings have been investigated to assess the ability of paste backfill to meet target design criteria identified from the required paste backfill functions for the operation. The test programs have determined design parameters upon which the preliminary engineering of the paste backfill system can be based.

Characterisation test work including laboratory testing and analyses, bench scale, surface loop, and pilot plant testing, has determined the viability of using the full plant tailings from this operation for paste backfill. A key focus of the characterisation test work has been the potential effect of variability in the tailings mineralogy on paste backfill performance. Variability in the tailings mineralogy is reflected in variability in particle size distribution and specific gravity, which can significantly affect the geomechanical strength and rheological behavior of the paste.

Ultimately, this variability must be addressed in all aspects of the paste backfill system design from paste de-watering and mixing to transportation and placement. The characterisation work at the case study operation clearly illustrates the effect of tailings variability on many key relationships including paste backfill strength/moisture content and slump/pressure loss. As a system, all of the components are interdependent. The paste backfill system design must accommodate the potential variability in order to produce a consistent paste, which meets all of the target criteria – geomechanical properties, distribution and

placement, environmental performance and socio-economic performance – for the operation.

Paste consultants have recommended that for the Myra Falls operation batch mixing of paste backfill with well-designed process control could alleviate the effect of material variability on paste quality. The characterisation work has determined the range of paste quality and performance that can be achieved, depending on the tailings material properties. This case study has illustrated how critical the physical, chemical, and mechanical characterisation of paste backfill material can be in determining the capability of meeting target design criteria and in forming the basis for reliable paste backfill system design.

Since its introduction in the late 1970's, paste backfill has slowly evolved as a viable alternative to rockfill and hydraulic slurry backfills in underground applications and as an alternative method for surface deposition of tailings. As companies look for ways to reduce the amount of surface disturbance created by mining, paste technology is being investigated for its ability to decrease the volume of surficial deposition of tailings. Environmental compliance is a growing issue for the mining industry. Paste backfill offers potential for reducing the environmental impact of acid generating wastes and heavy metals and for the conservation of scarce water resources.

Globalization of the mining industry and lower commodity prices have forced mines to decrease their costs and improve productivity in order to remain competitive. Efforts to improve ore recoveries have increased the use of both bulk and selective mining methods employing backfill. In addition, as deeper ore deposits are discovered and developed, where ground conditions, rock stresses, and environmental health and safety issues become critical considerations, the role of backfill will continue to grow. Paste backfills, depending on the given situation, offer many potential advantages – safety, environmental, and economic – over conventional backfilling methods.

After developing a design rationale for paste backfill systems, a key objective of this thesis has been to identify the components of the design process that are critical to reliable paste backfill design. In order to accomplish this, a thorough review of the literature including topics related to backfill type, backfill functions, paste backfill target properties and paste system design criteria has been undertaken. Backfill design methodologies have been examined and analysed. In addition, a survey of the design and operating experience of mines using paste backfill has served as an important source of data for the development of the resulting paste design rationale. Lastly, field experience gained by the author in

paste characterisation test work has also contributed to the analysis of design and issues related to paste backfill.

The paste backfill design rationale that has been presented in this thesis is intended to be a guide to design. Target design criteria are based on the required backfill functions, which are in turn dictated by the overall mining system and other site-specific conditions. The three main design components of paste backfill systems are related to preparation, distribution, and placement. Available backfill material must be evaluated to determine its suitability to meet the required backfill target properties. Evaluation is a critical element of successful paste backfill system design. After implementation of a backfill system, on-going monitoring is important to ensure that the backfill performance continues to meet design targets.

The primary paste backfill design criteria include geomechanical properties, distribution and placement criteria, environmental performance, and socio-economic performance. Paste design criteria are site-specific and will vary depending on the requirements of the given mining system. The suitability of tailings to meet the target paste backfill properties can only be determined through a comprehensive characterisation program to identify the physical, chemical and mechanical properties of the tailings material. This thesis has outlined the component procedures for such characterization.

Backfill strength and liquefaction potential are key geomechanical properties. Binder addition is important to meeting strength targets and to reducing the susceptibility of paste backfill to liquefaction. Selection of an appropriate paste backfill recipe is a critical element of the design process.

The distribution and placement systems must be designed to meet the mine scheduling targets, volumes, and rates, within the limitations of the rheological properties of the paste. Accurate design of the distribution system relies on

rheological characterisation test work, including surface loop and gravity flow testing to determine the friction losses and transportation constraints for the paste.

Waste management is a critical issue for the mining industry. The environmental performance of paste backfill for both underground and surface waste disposal has significant ramifications for successful mine permitting and operation.

The socio-economic considerations related to paste backfill are also an important part of the evaluation and design process. The socio-economic impacts of mining projects can be far-reaching. Capital and operating costs of paste backfill systems are significant and are governed by the limitations of available technology to meet the target design criteria and backfill properties required.

A review and analysis of case studies of the practical design experience of mines that have implemented paste backfill systems has formed the basis for the identification of critical success factors for paste backfill system design. These critical success factors include: design based on the complete knowledge of the physical, chemical, and mechanical characteristics of the backfill material; a holistic approach and participation in the design and operation of paste backfill systems; provision for adequate commissioning time; and monitoring/re-evaluation of backfill quality and performance. The practical experience gained from paste backfill system operators can provide a source of solutions to design and operating problems.

Despite the potential benefits of paste backfill systems, there have been relatively few field implementations. Considerable research has been conducted and advancements in paste technology have been made, however, there are still outstanding issues that must be addressed. Some of these issues are related to technological challenges. Other issues are related to design methodologies and the sophistication and reliability of paste backfill systems. As with all newly

developing technologies, there is often proprietary knowledge, which can slow the spread of technology. There may also be a reluctance to share technology and information with competing mining operations, consultants, manufacturers, academics, researchers, and suppliers. It often takes time for these barriers to be broken down to allow more widespread dissemination of information.

Paste, due to its composition, exhibits unique backfill qualities and presents technological challenges, which impact on paste backfill system design. The colloidal particles in the tailings are responsible for the water retentive properties, which differentiate pastes from slurries and, in turn, influence the strength and rheological properties of the paste. Many of the technological challenges are due to the geomechanical and rheological properties attributed to these colloidal particles.

De-watering of fine tailings in an efficient and cost effective manner is one significant technological challenge. The proven conventional de-watering method of tailings de-watering is through combined thickening/filtration, which is a high capital and operating cost option. Alternative tank-type batch and continuous de-watering systems have been developed. To date, the mining industry has been reluctant to adopt these alternative de-watering systems. Reliability and production of a consistent quality paste appear to be stumbling blocks to more widespread use of new de-watering technology. Where paste is being considered for surface disposition involving large tonnage projects such as oils sands, coal, aluminum, potash, and phosphates, the development of reliable, cost effective continuous de-watering systems is particularly important. Thus, despite the advances that have been made in de-watering of fine tailings, more research and, perhaps, better marketing of new technology is still required.

Transportation of pastes is another technological challenge. The unique rheological behavior of paste places constraints on its distribution. Surface loop and gravity flow testing provides vital data for predicting pipeline pressure losses

and for determining important relationships such as slump:solids density:friction loss, which form the basis for careful engineering design of paste distribution systems. Research has been conducted on the use of admixtures or flow aids to facilitate pipeline transportation of pastes. The performance of admixtures will be dependent on factors such as mineralogical composition and particle size distribution of a given paste and the effects of chemical additives on paste backfill strengths must also be considered. Pumping of pastes generally requires high-cost positive displacement pumps. Pumping research and development in the concrete industry continues to be a potential source of improved technology for pastes in mining.

The range of tailings variability can also be significant to paste design. The backfill system must be designed to accommodate variability of material properties, which may affect the paste quality and performance. Key tailings characteristics that impact on paste backfill are mineralogy and particle size distribution. Variation in mineralogy, affecting specific gravity and the percent of fine particles in the tailings, can affect the ability of the paste backfill to meet target strength properties and on the flow behavior of the paste. One or more of a number of relationships can be exploited to control the effects of variability in tailings and, hence, permit the production of a paste backfill that will exhibit consistent quality and performance. Some of the relationships that can be used include: moisture content:slump, power draw:slump, pressure loss:slump, specific gravity:pulp density, specific gravity:slump, and strength:binder addition. During the tailings characterisation program it is important to identify the specific relationships that will be useful in the design of the paste backfill system under consideration.

The case study paste characterisation work conducted at Boliden-Westmin's Myra Falls operation illustrates the significance of determining material variability to reliable paste backfill system design. Different ore mineralogy results in variability of specific gravity from 3.2 to 4.0 and variability in particle size

distribution from 30% to 42% minus 20 micron. In order to produce a consistent paste backfill quality and performance, the paste backfill plant must be designed to accommodate the material variability. Design of the transportation and placement components of the paste backfill system for the Myra Falls operation is another key area, which relies on accurate rheological characterisation of the tailings material. Because paste will have to be transported relatively long distances underground to reach the separate production areas without compromising the ultimate backfill strength, the relationships between slump, moisture content, and backfill strength must also be well understood.

Long-term stability of paste backfill is a key issue that must be addressed, particularly when the function of the backfill is to provide a free-standing wall. Considerable test work can be undertaken to optimize paste backfill strength through adjustment of the paste backfill recipe. The binder content, binder type, moisture content, and tailings particle size distribution can all be fine-tuned to provide a paste that meets target strength requirements. Sulfate attack is a major topic of concern, particularly with tailings that have high sulfide content. The effect of sulfate attack on backfill stability has been a significant challenge for several mines that have implemented paste backfill systems. It has been shown that the effects of sulfate attack can be at least partially mitigated through optimization of the binder recipe. Paste backfill operations have also found that by altering blasting practice, backfill dilution can also be reduced. The experience to date of mines with pyritic tailings, in particular, should serve as a lesson well learned regarding the necessity of conducting adequate mineralogical, backfill strength, and paste backfill recipe test programs during the design phase.

In addition to sulfate attack, other chemical reactions, including acid generation and leaching of heavy metals from tailings may impact on the environmental effects of paste backfill. Research and development work related to chemical/environmental impacts of paste backfill has been limited, partly due to the fact that the technology is relatively new and many environmental effects are

only manifested with the passing of time. Again, adequate characterisation work must be conducted to identify any potential chemical reactivity that should be addressed in the paste backfill system design.

The development of more robust, cost effective instrumentation and monitoring systems is another area of research and development, which will enhance the design and operation of paste backfill systems.

Other outstanding issues are related to costs, reliability, and sophistication of paste backfill systems. Production depends on reliable backfill total system performance. Few implementation case studies have shown rapid achievement of high reliability however, the six case studies reviewed all appeared to be operating successfully after addressing initial startup problems. Paste backfill systems require a fairly high degree of sophistication compared to rockfill and hydraulic backfill systems. Process control and instrumentation, monitoring systems, and operating procedures are more complex and may be perceived to be unsuitable for particular mining situations, for example in remote locations and where a skilled labour force may not be readily available.

Design and operation of reliable paste backfill systems depends on a holistic approach or in other words, the collaboration of geology, mine-planning, rock mechanics, milling, waste management, and mine operations departments. The gap between the mine and mill operations must be bridged. Over time metallurgical technology has evolved to the point where mineral processing operations are highly complex. In mill operations the optimization of metallurgical recoveries and the focus on quality control and performance necessitates careful attention to detail and an acute awareness of the potential downstream effects of any variation in the system. Instrumentation, process control systems, operational procedures, regular data collection and analysis, and comprehensive maintenance schedules are essential components of mill operations. New technology has contributed to a growing complexity in mining too. The same

operational approach found in mill operations must be carried underground. The traditional approach of mining by brute force must be adjusted to accommodate the new demands brought to mine operations by the evolution of technology.

As outlined above there are a number of outstanding issues that must be addressed in the field of paste backfill. New technology is evolving and research in many different facets of paste backfill design is still on-going. Environmental and economic pressures will continue to provide motivation to apply paste technology to a range of waste disposal and backfill functions.

In conclusion, the rationale for paste backfill systems design that has been developed in this thesis is intended to provide a useful template for paste backfill system design. An identification of critical success factors for paste design and important outstanding issues based on a review of practical case studies of mines that have implemented paste backfill systems, together with hands-on field experience gained in conducting paste characterisation test work has been used to present an empirical approach to facilitating improved design of future paste backfill systems.

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APPENDIX

Table A-1 Descriptive Statistics for Tailings Iron Content 1997-98

Mean	22.80582813	Skewness	0.198551602
Standard Error	0.097316206	Range	21.7
Median	22.75	Minimum	12.7
Mode	23.5	Maximum	34.4
Standard Deviation	3.481690422	Sum	29191.46
Sample Variance	12.1221682	Count	1280
Kurtosis	0.001638975		

Table A-2 Descriptive Statistics for 12-Hour Variance in Tailings Iron Content 1997-98

Mean	0.00328125	Skewness	-0.23834152
Standard Error	0.038744683	Range	14.1
Median	0	Minimum	-8.55
Mode	0	Maximum	5.55
Standard Deviation	1.386171913	Sum	4.2
Sample Variance	1.921472571	Count	1280
Kurtosis	2.752032596		

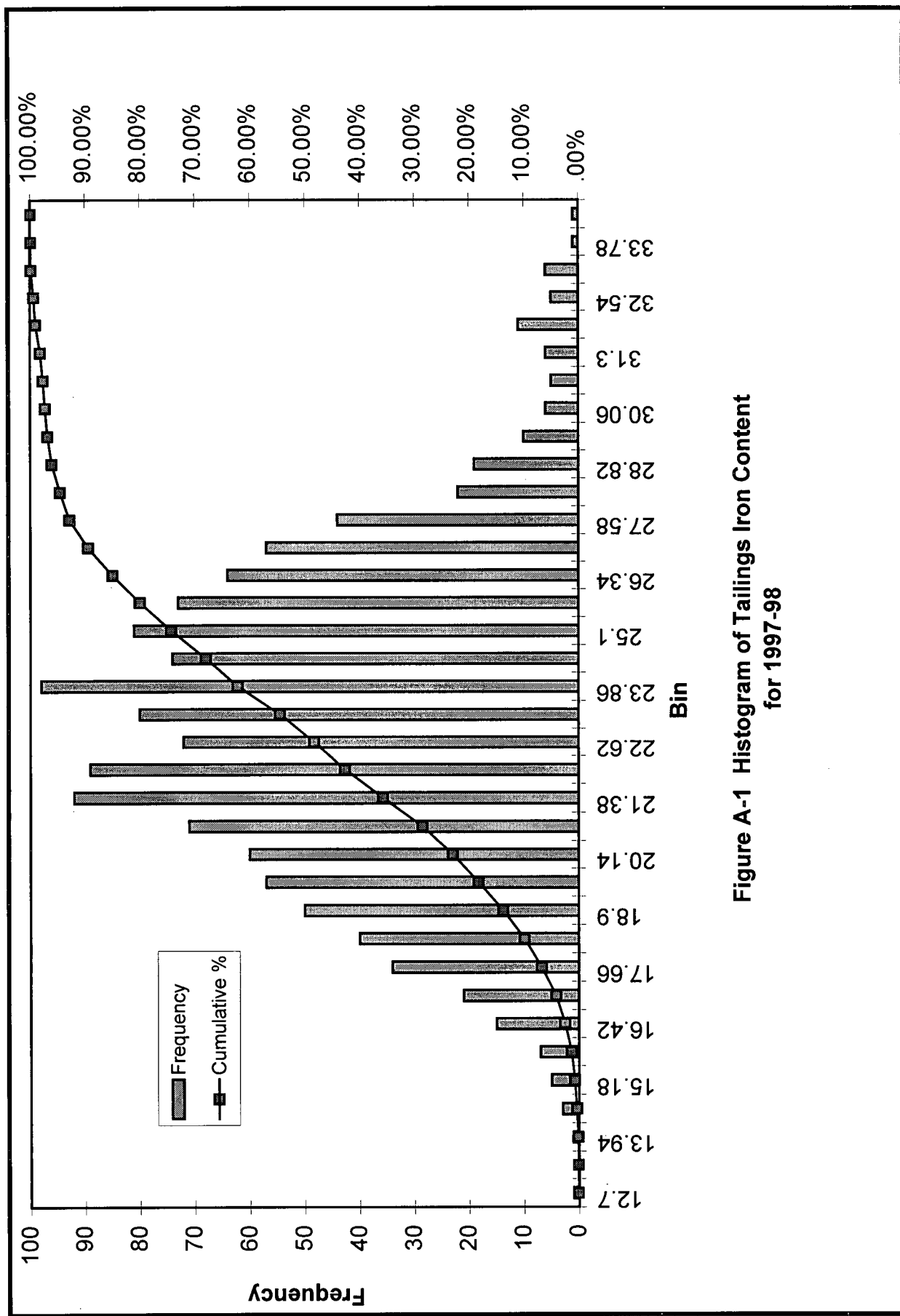


Figure A-1 Histogram of Tailings Iron Content
for 1997-98

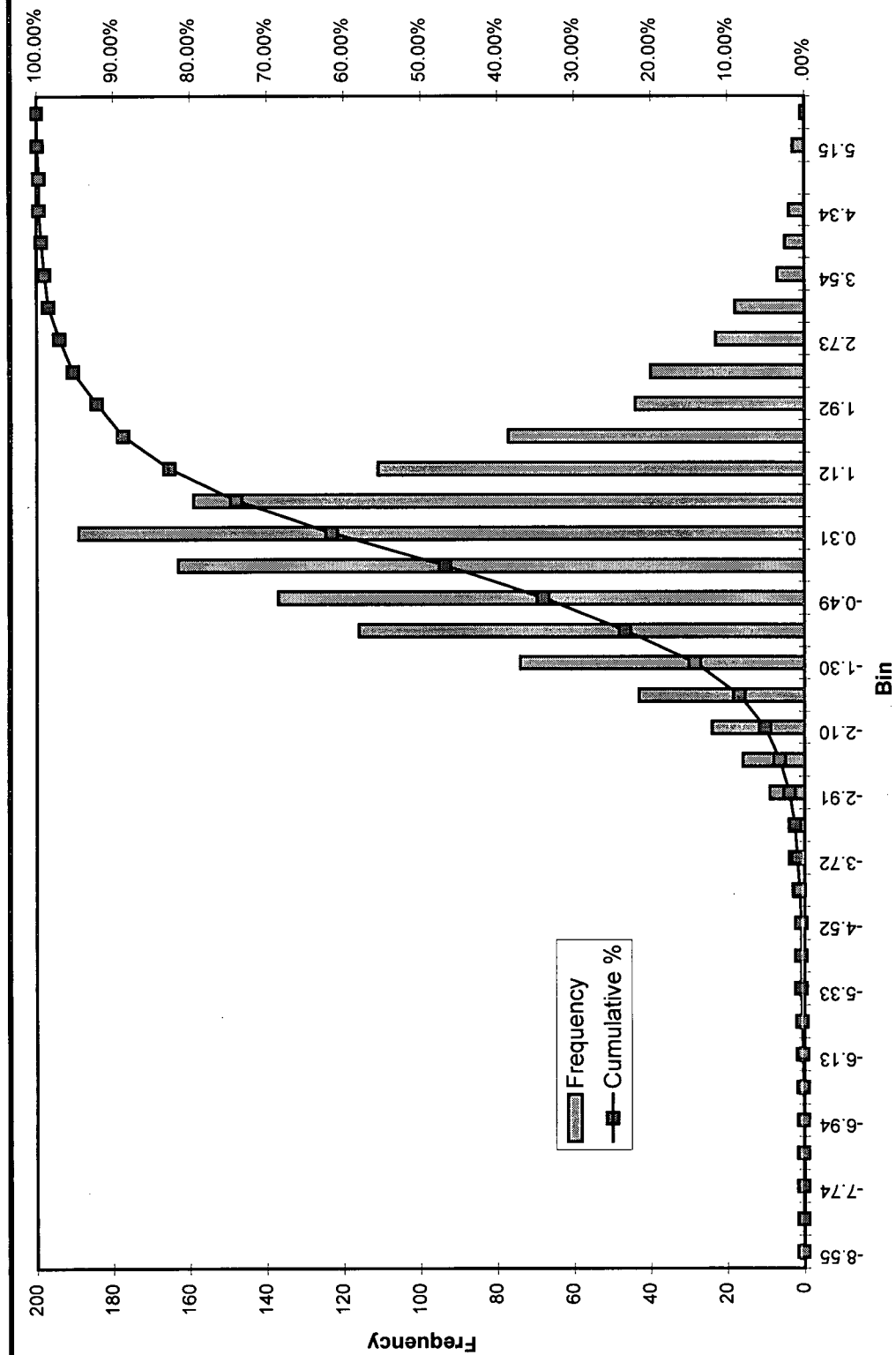


Figure A-2 Histogram of 12-Hour Variance in Tailings Iron Content for 1997-98

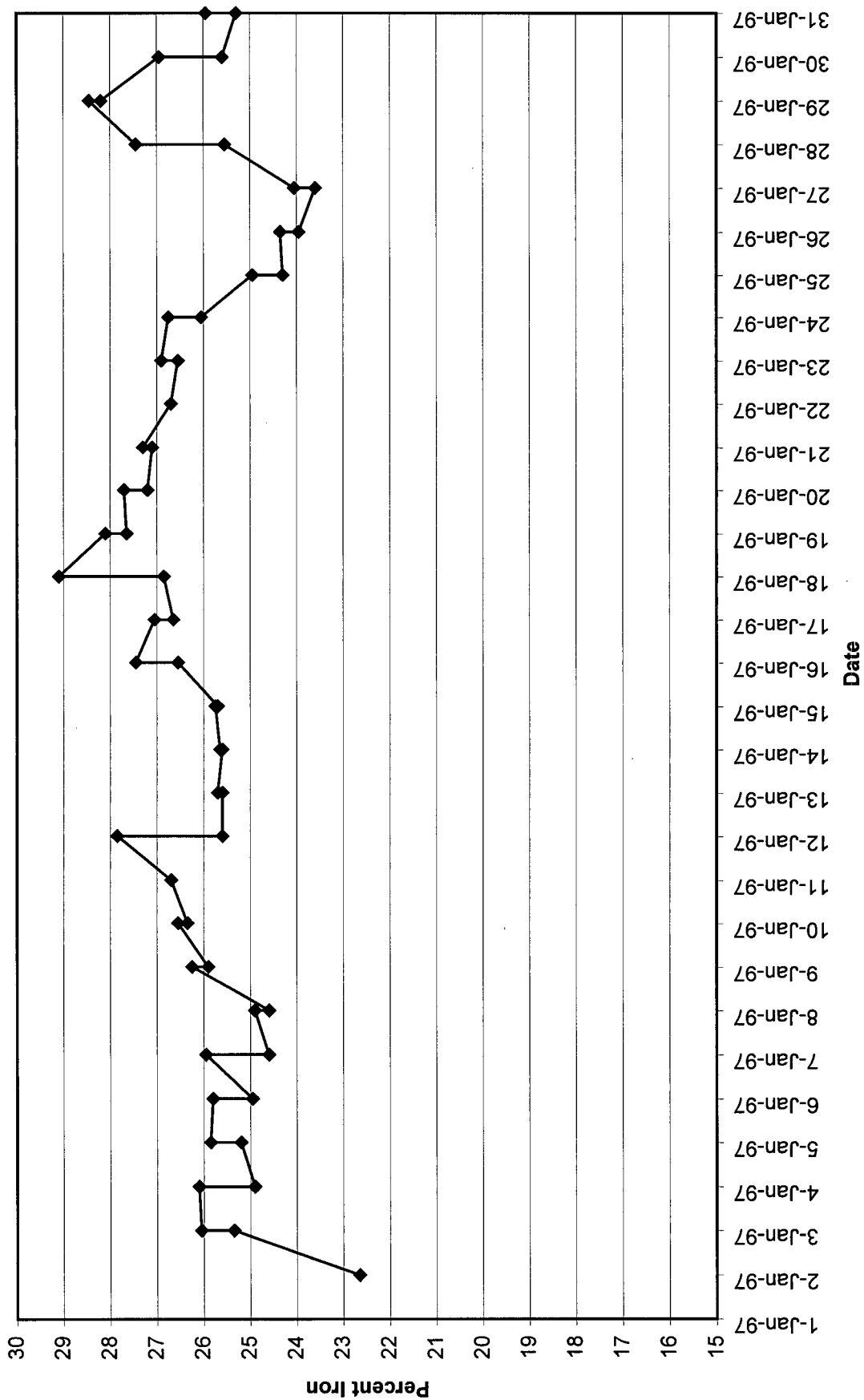


Figure A-3 Tailings Iron Content - January 1997

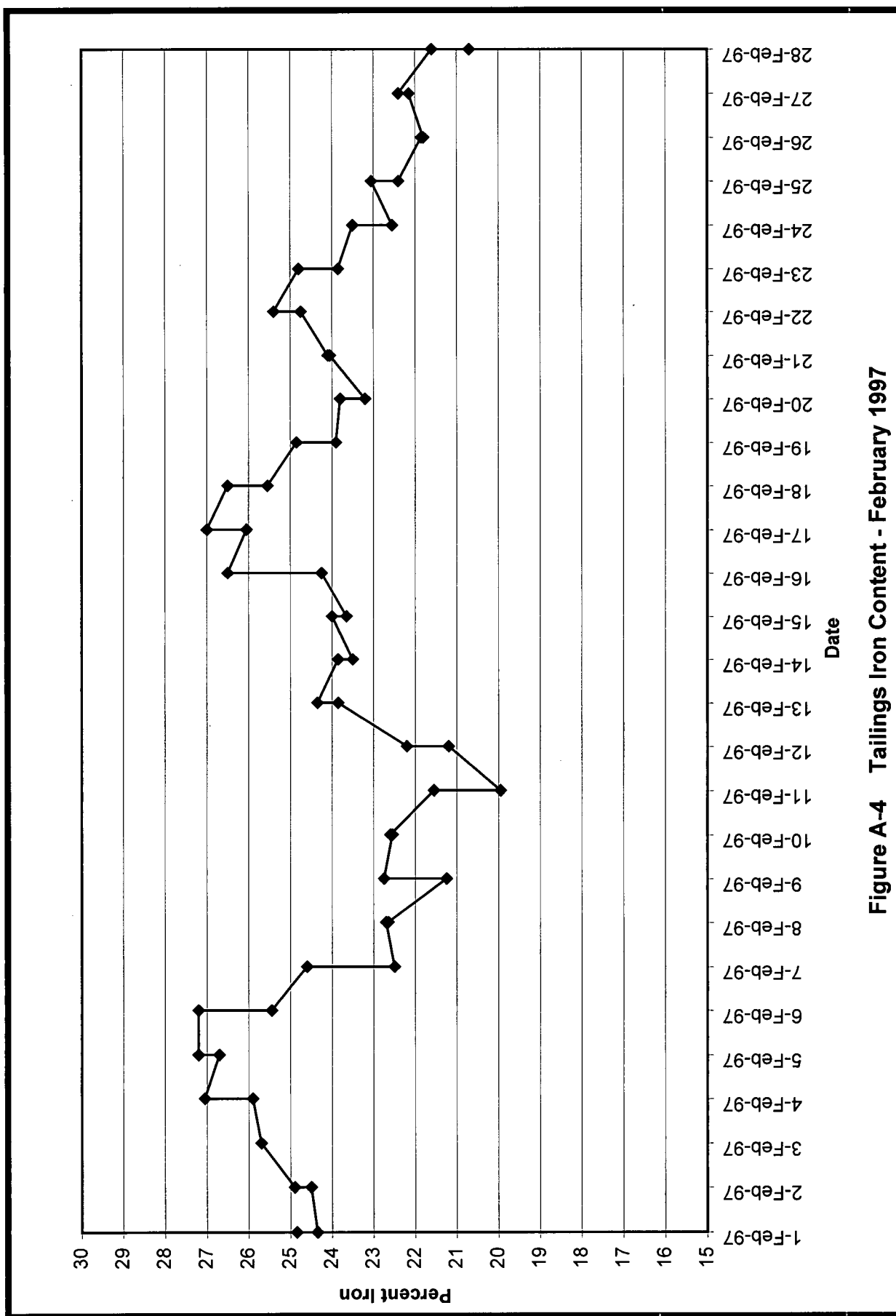


Figure A-4 Tailings Iron Content - February 1997

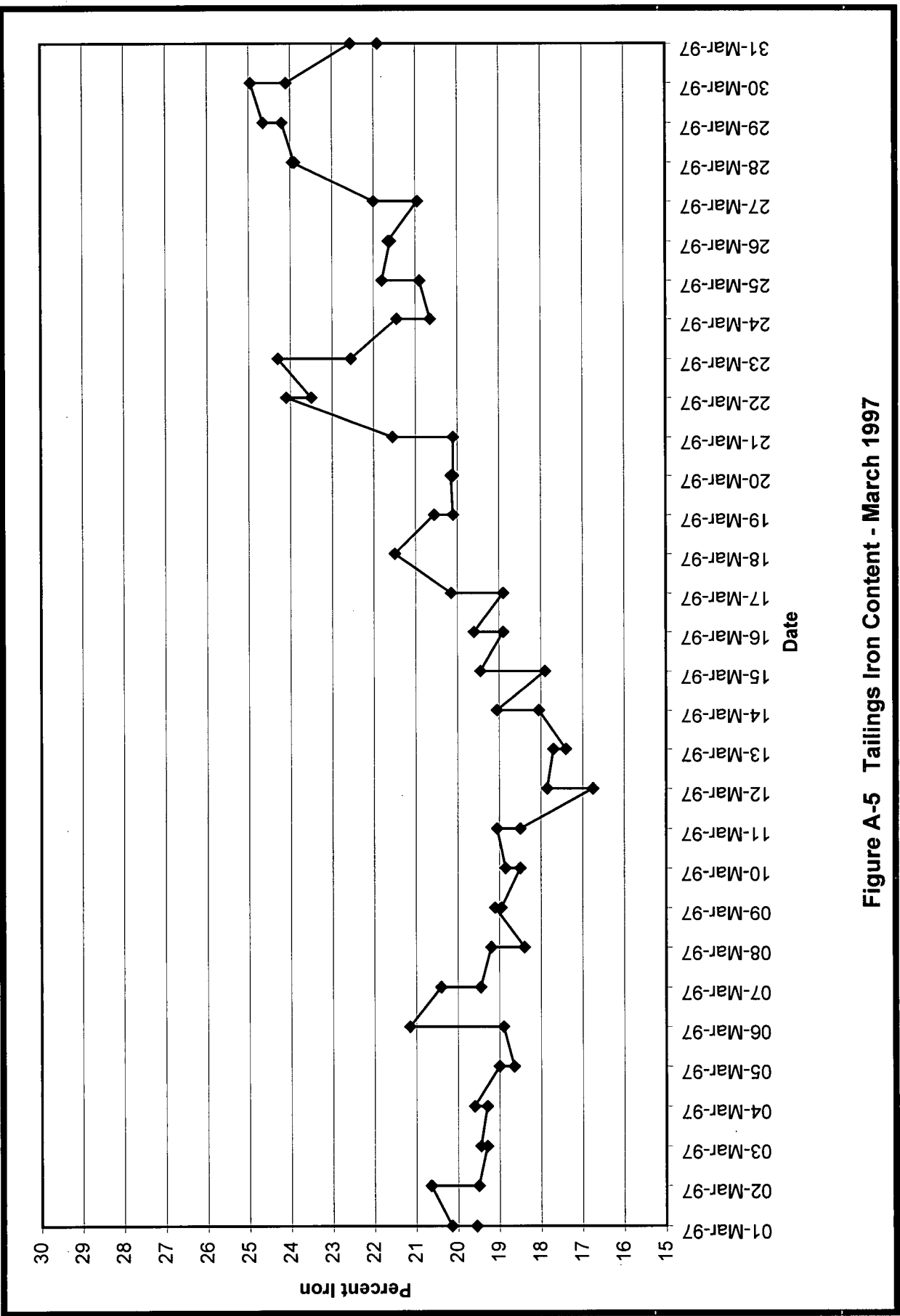


Figure A-5 Tailings Iron Content - March 1997

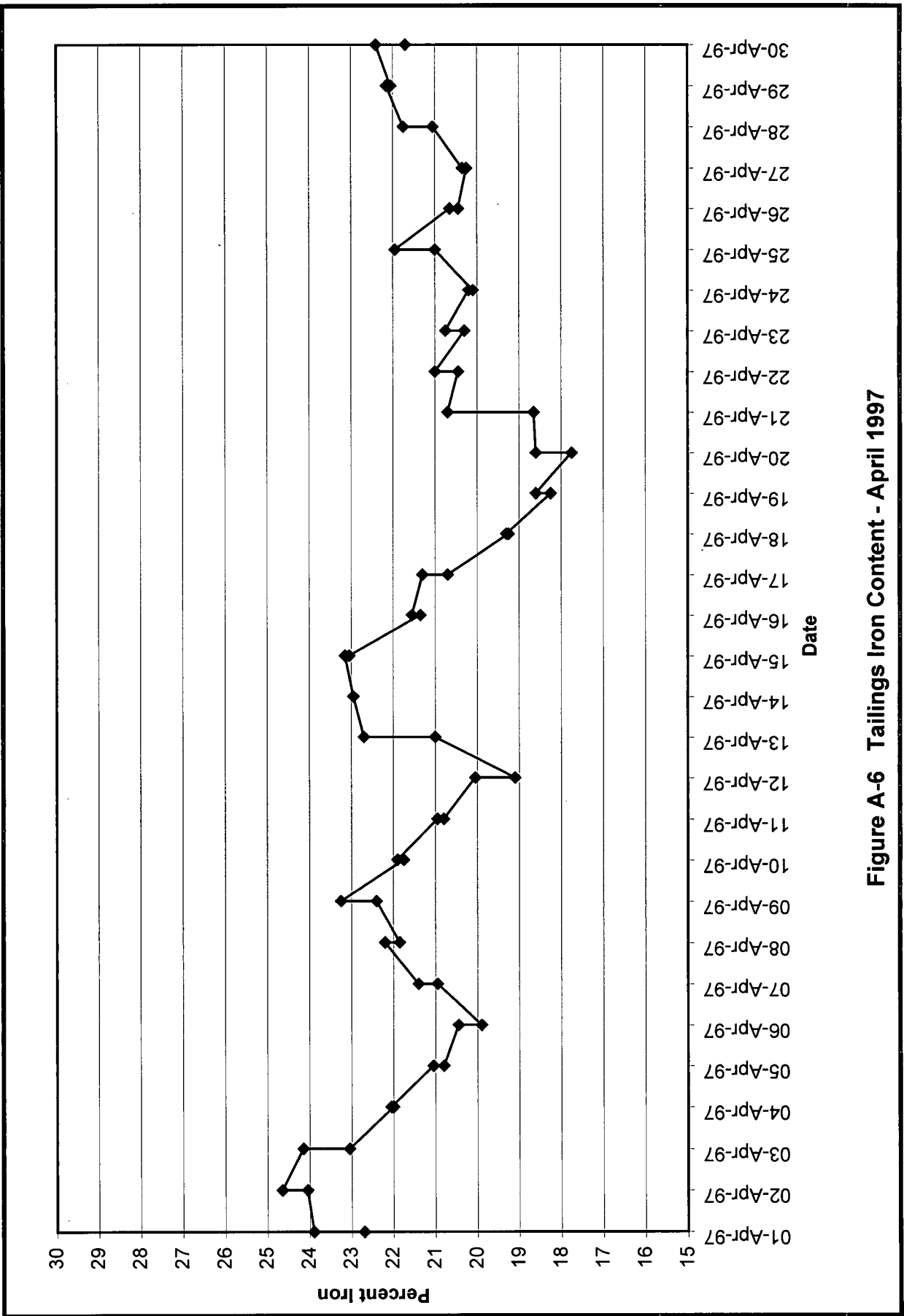


Figure A-6 Tailings Iron Content - April 1997

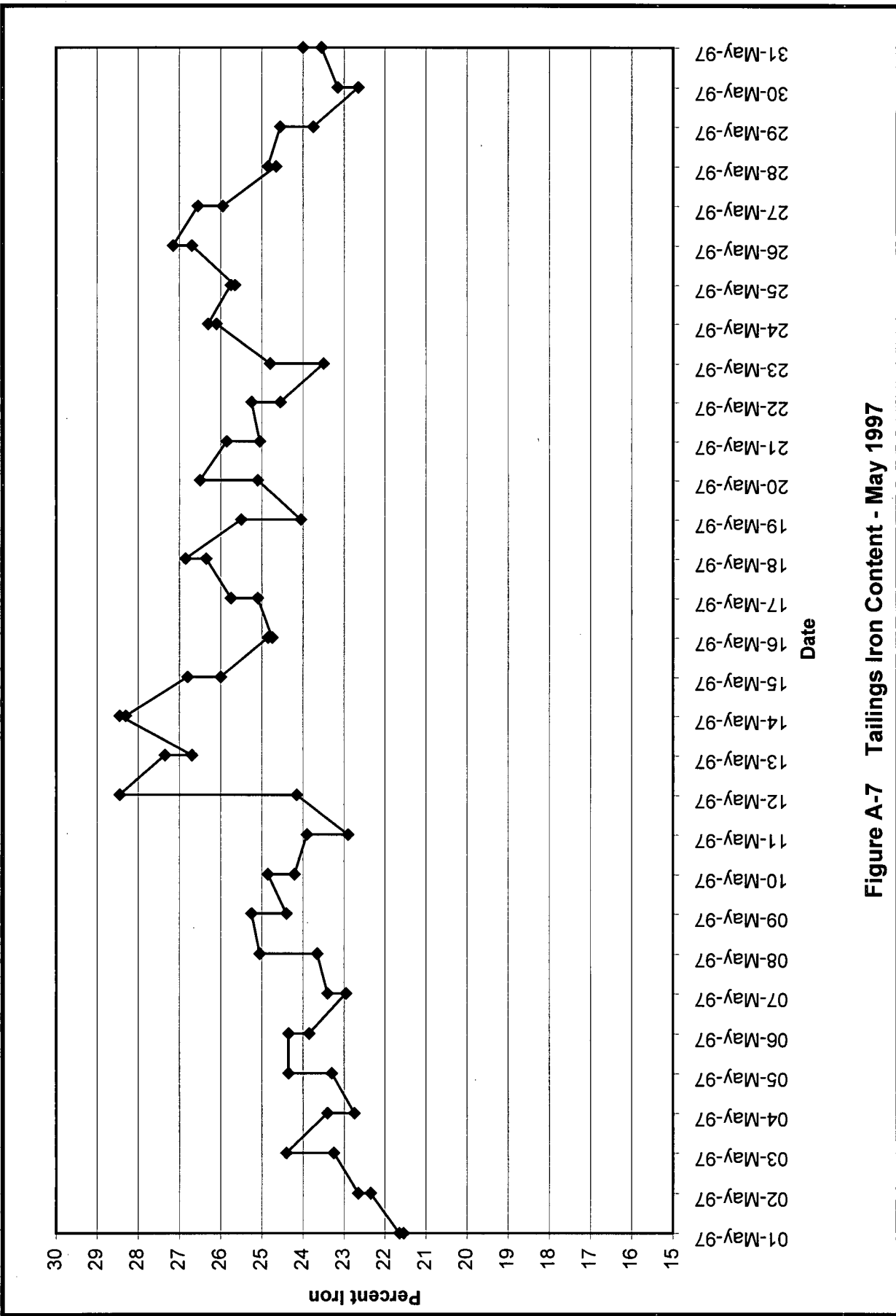


Figure A-7 Tailings Iron Content - May 1997

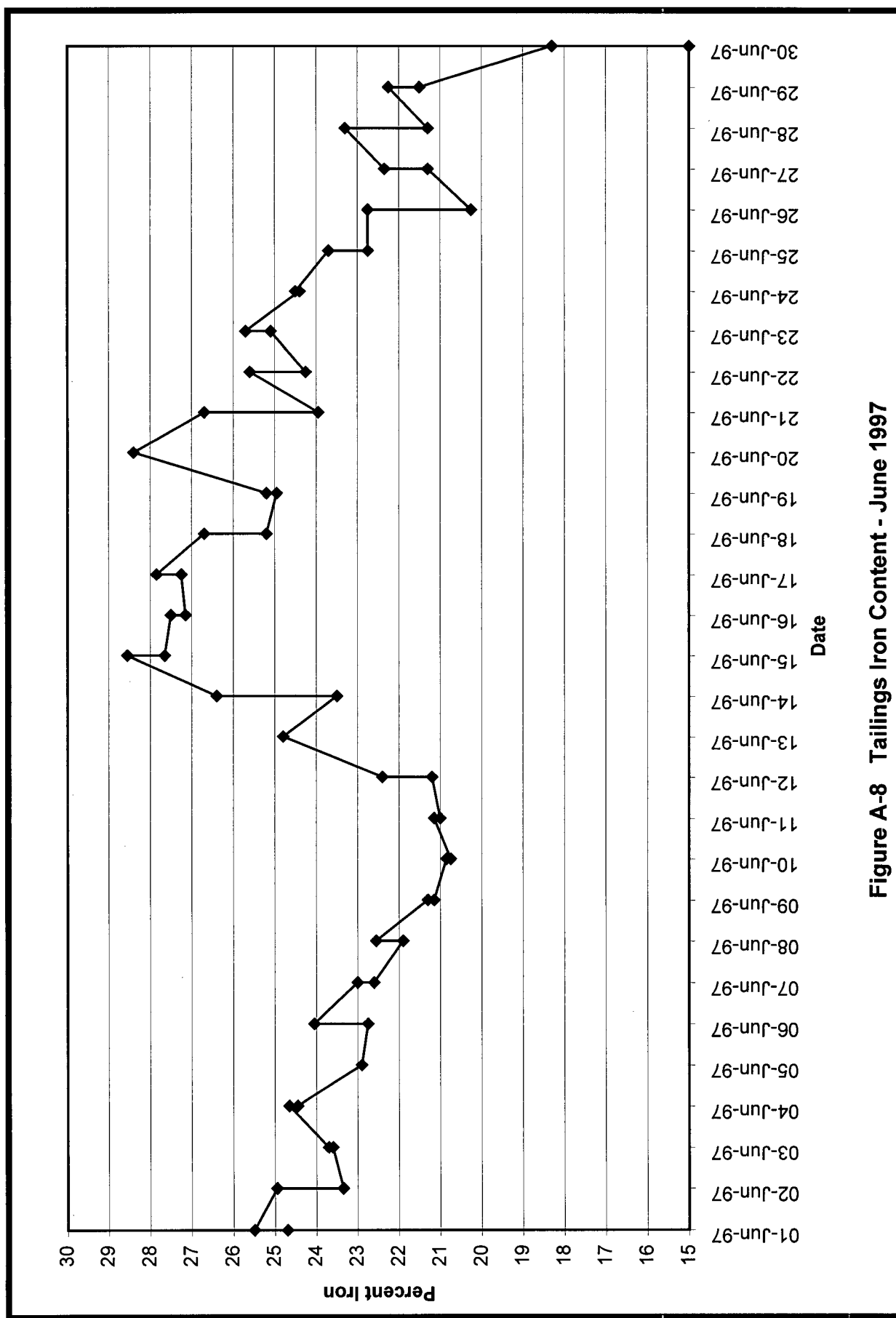


Figure A-8 Tailings Iron Content - June 1997

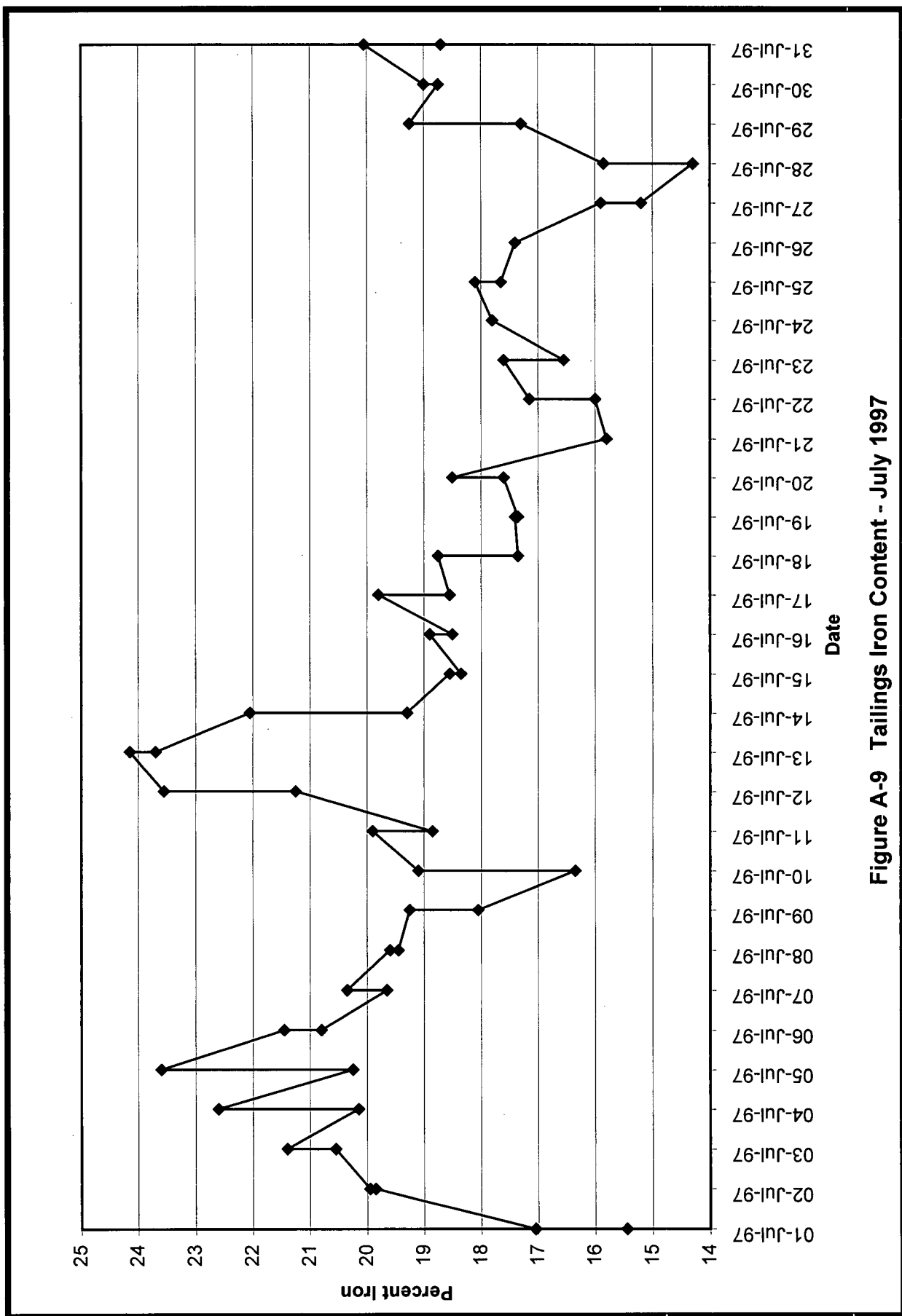


Figure A-9 Tailings Iron Content - July 1997

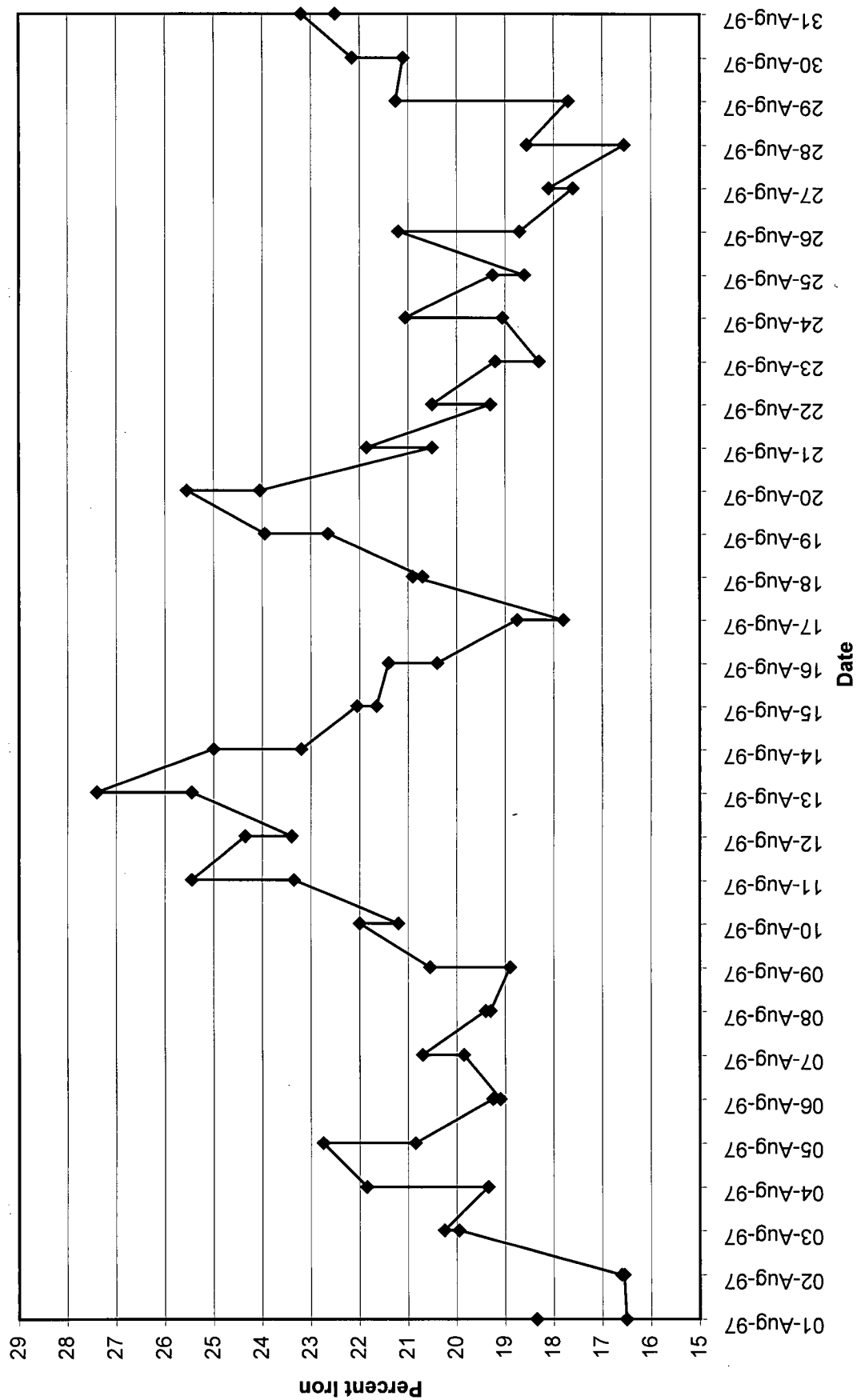
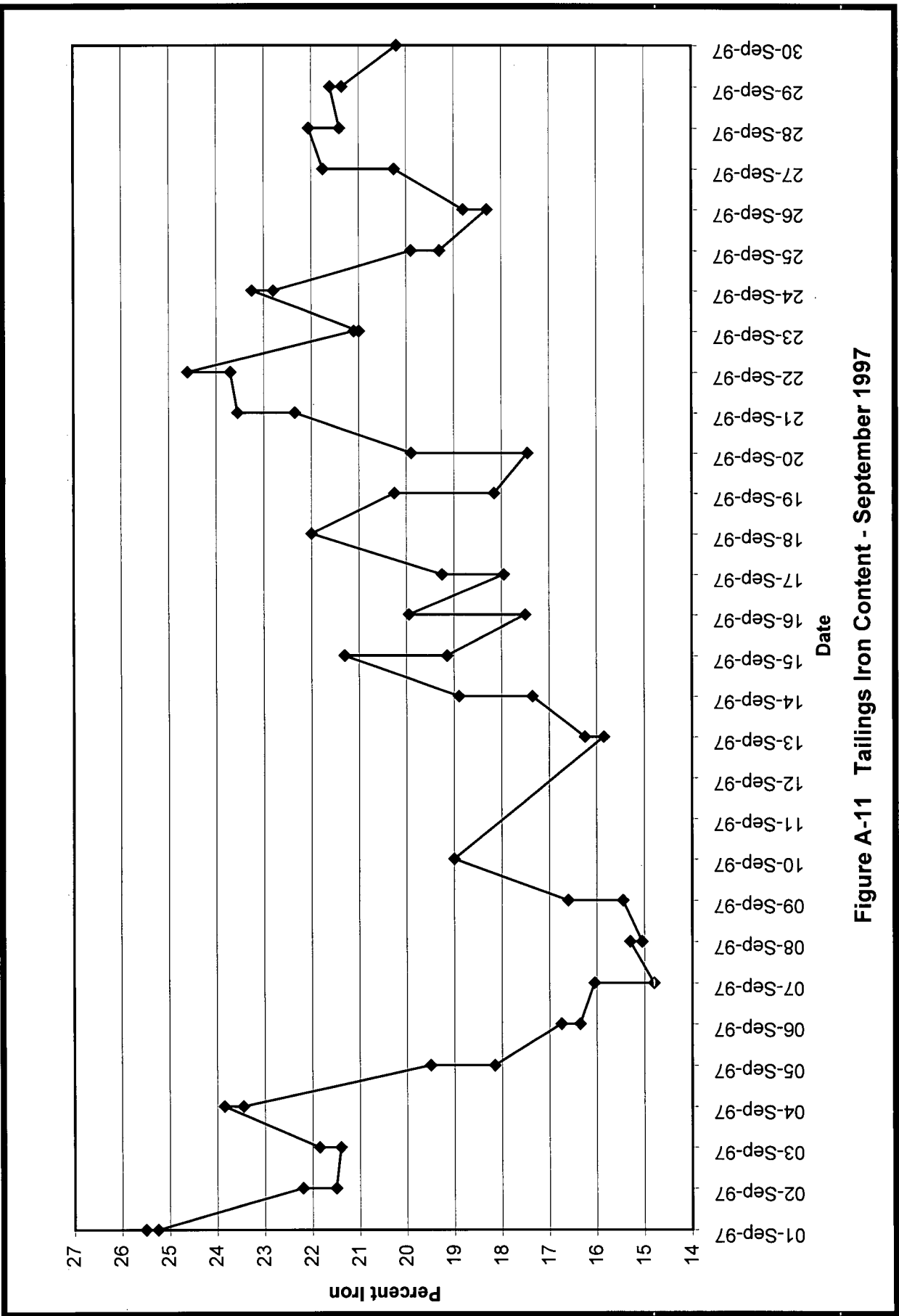
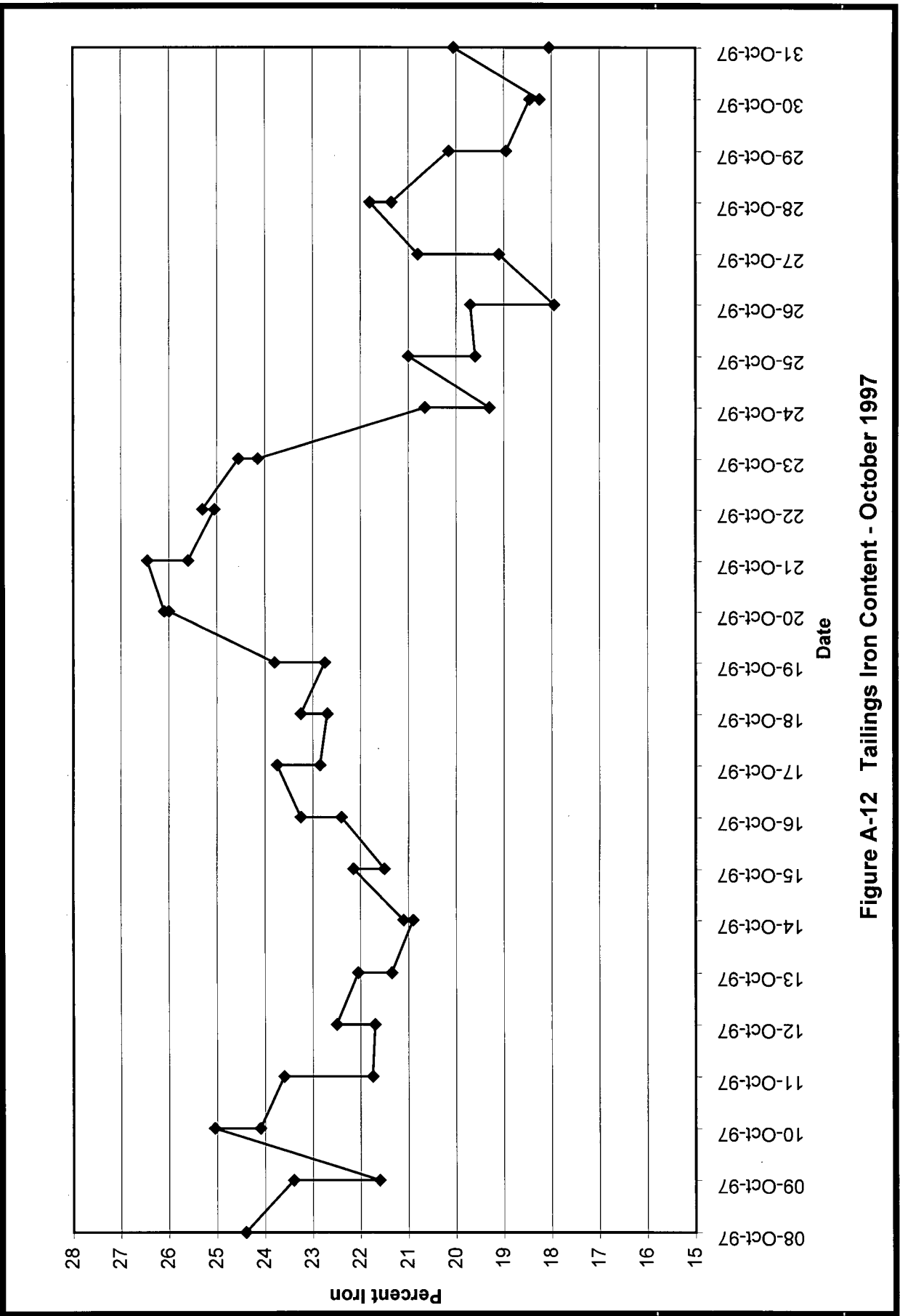


Figure A-10 Tailings Iron Content - August 1997





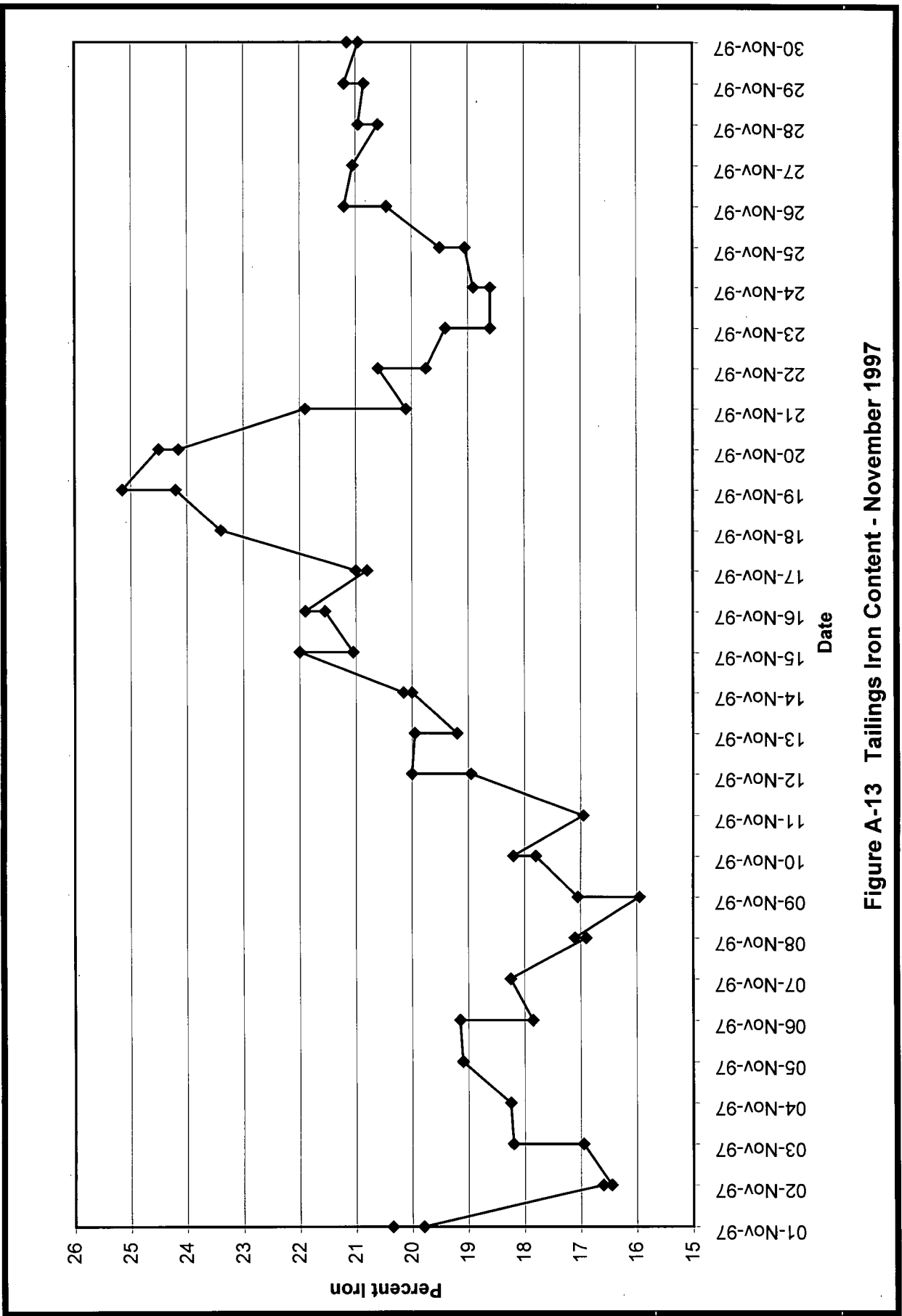
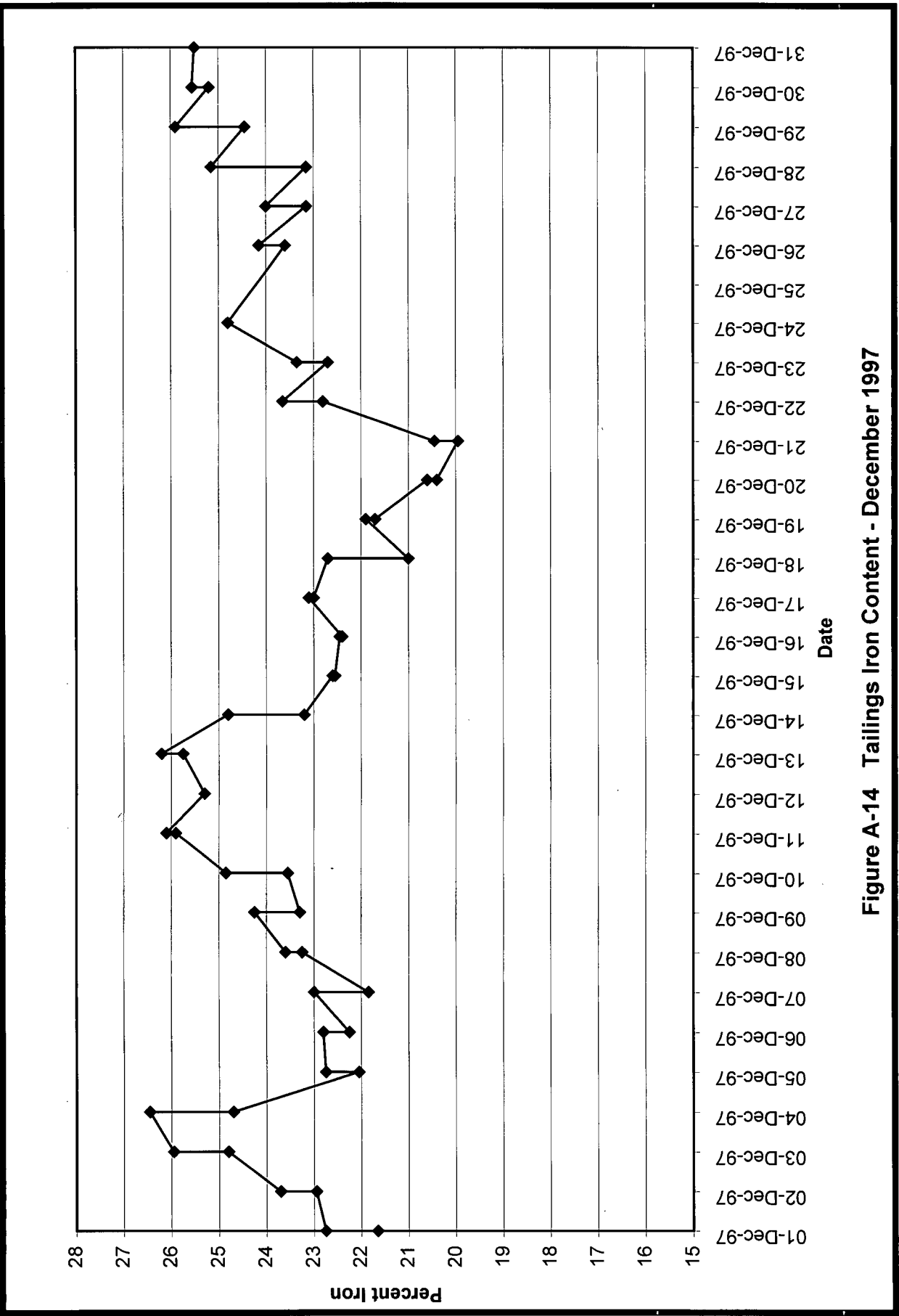


Figure A-13 Tailings Iron Content - November 1997



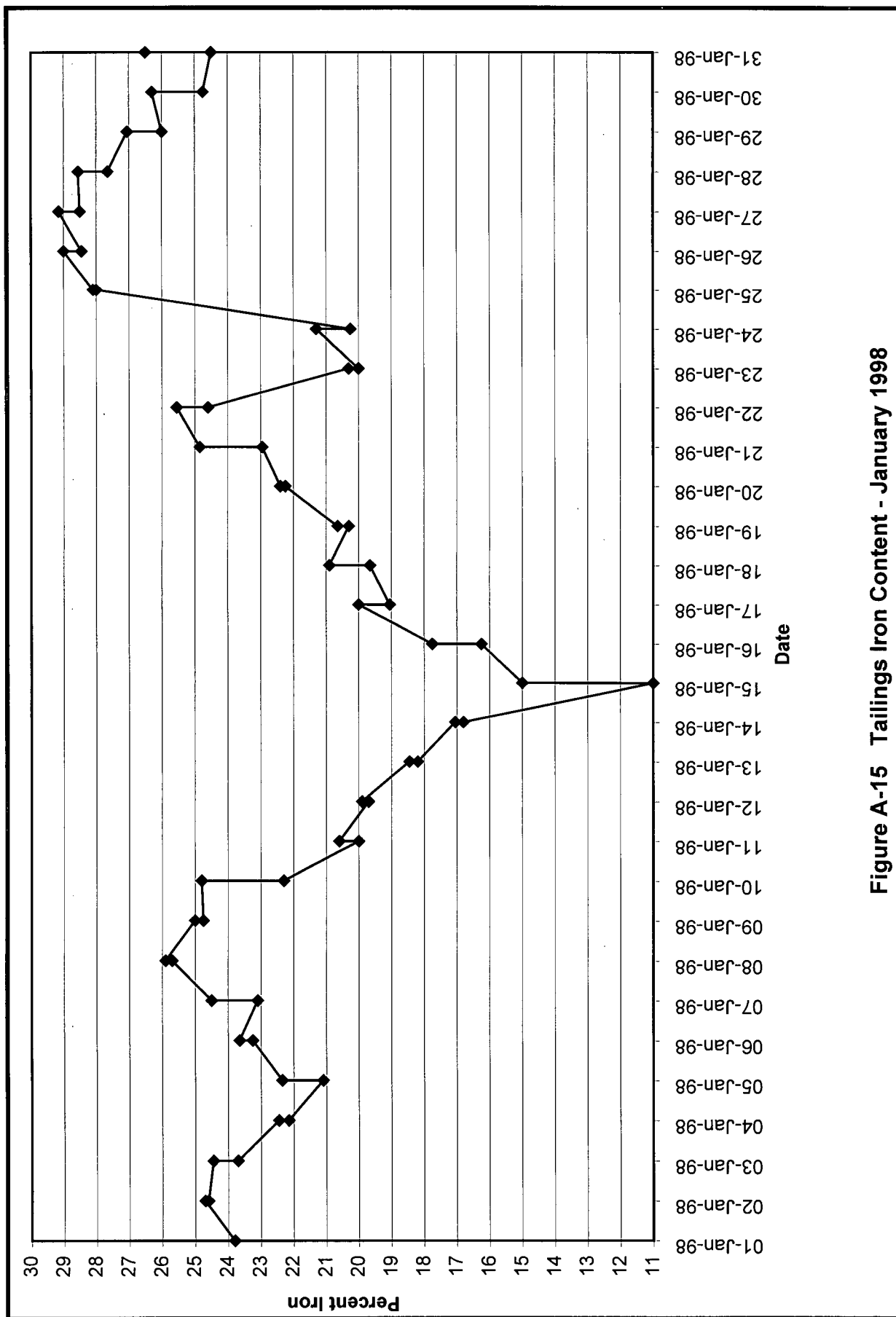


Figure A-15 Tailings Iron Content - January 1998

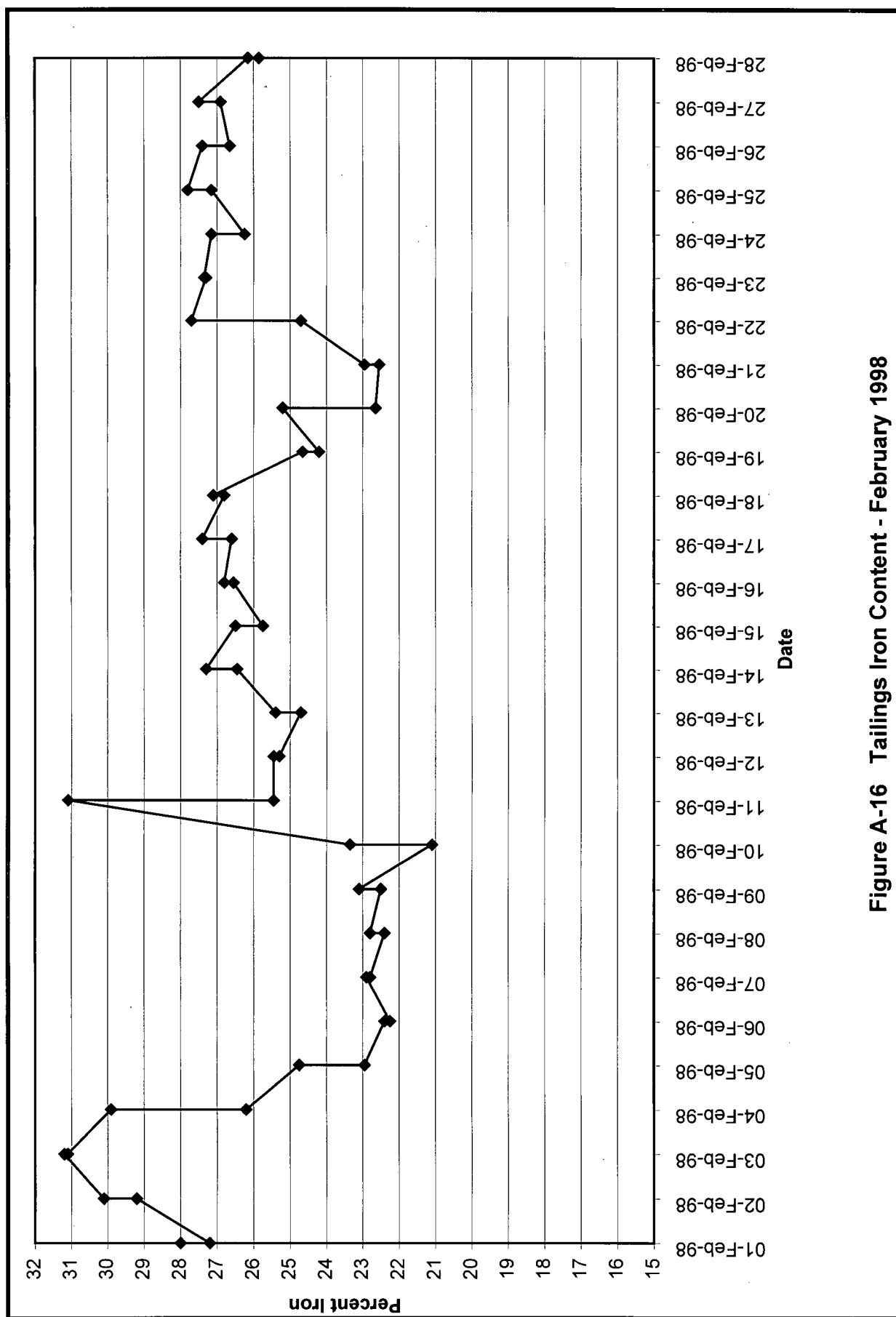


Figure A-16 Tailings Iron Content - February 1998

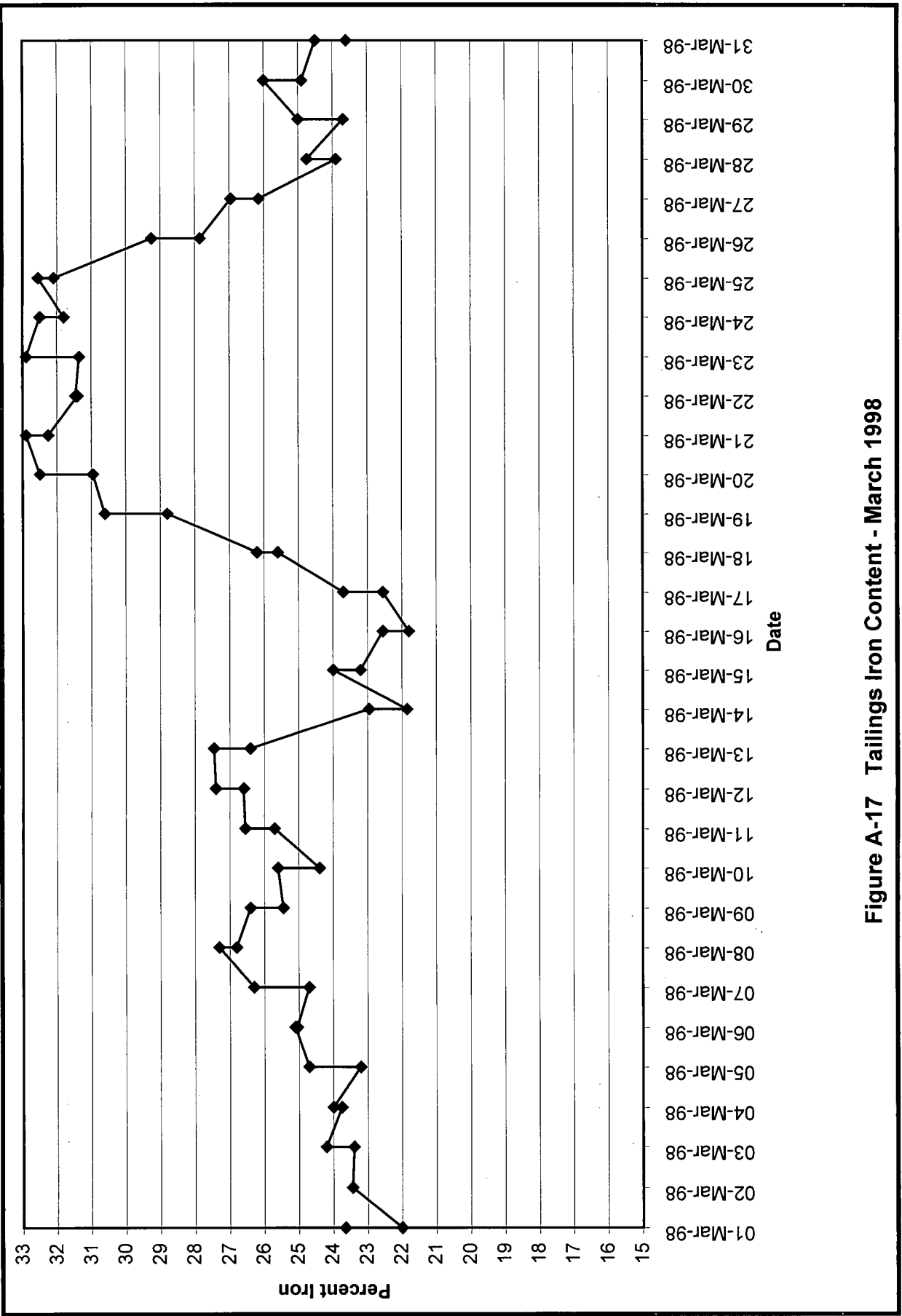


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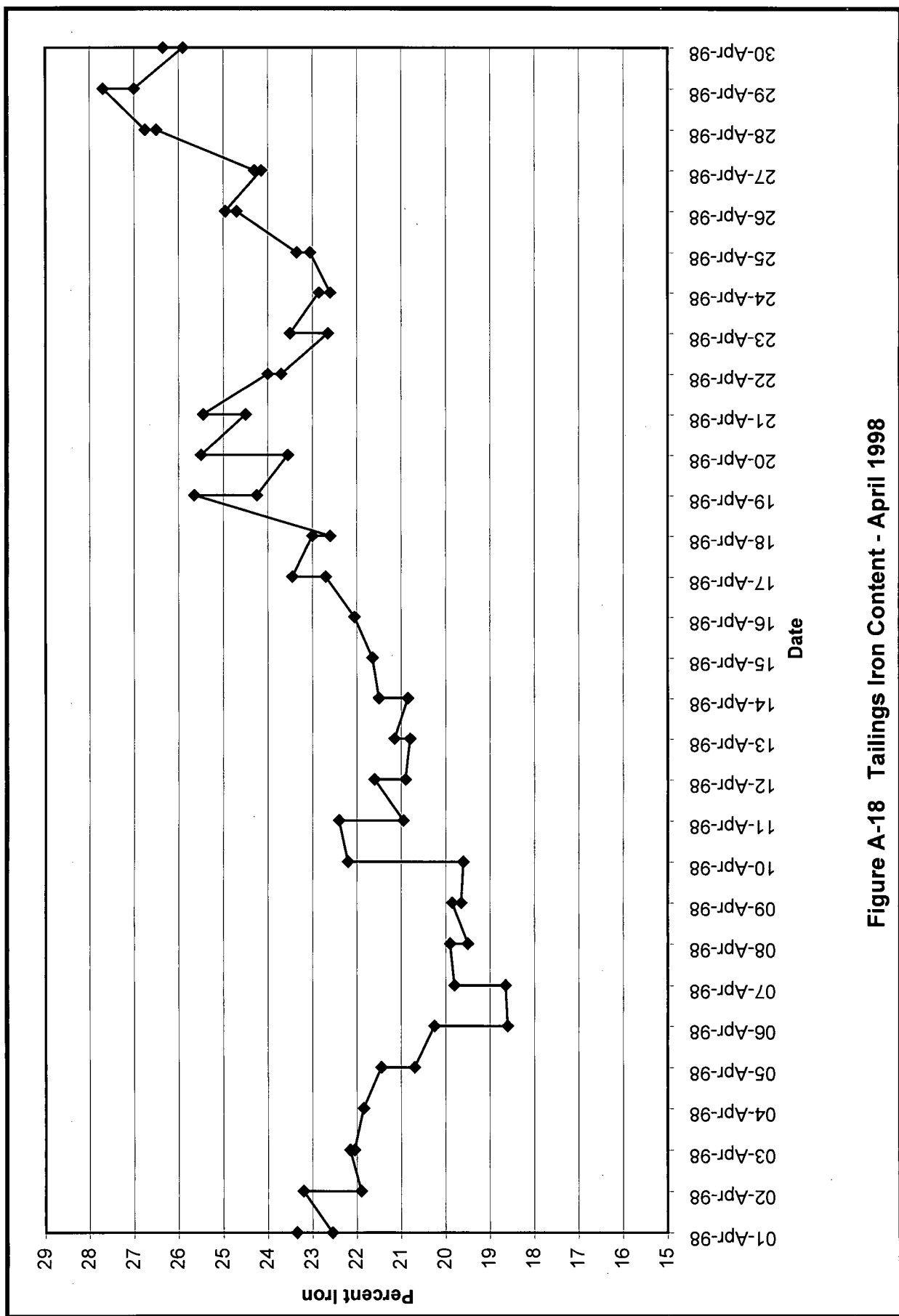


Figure A-18 Tailings Iron Content - April 1998

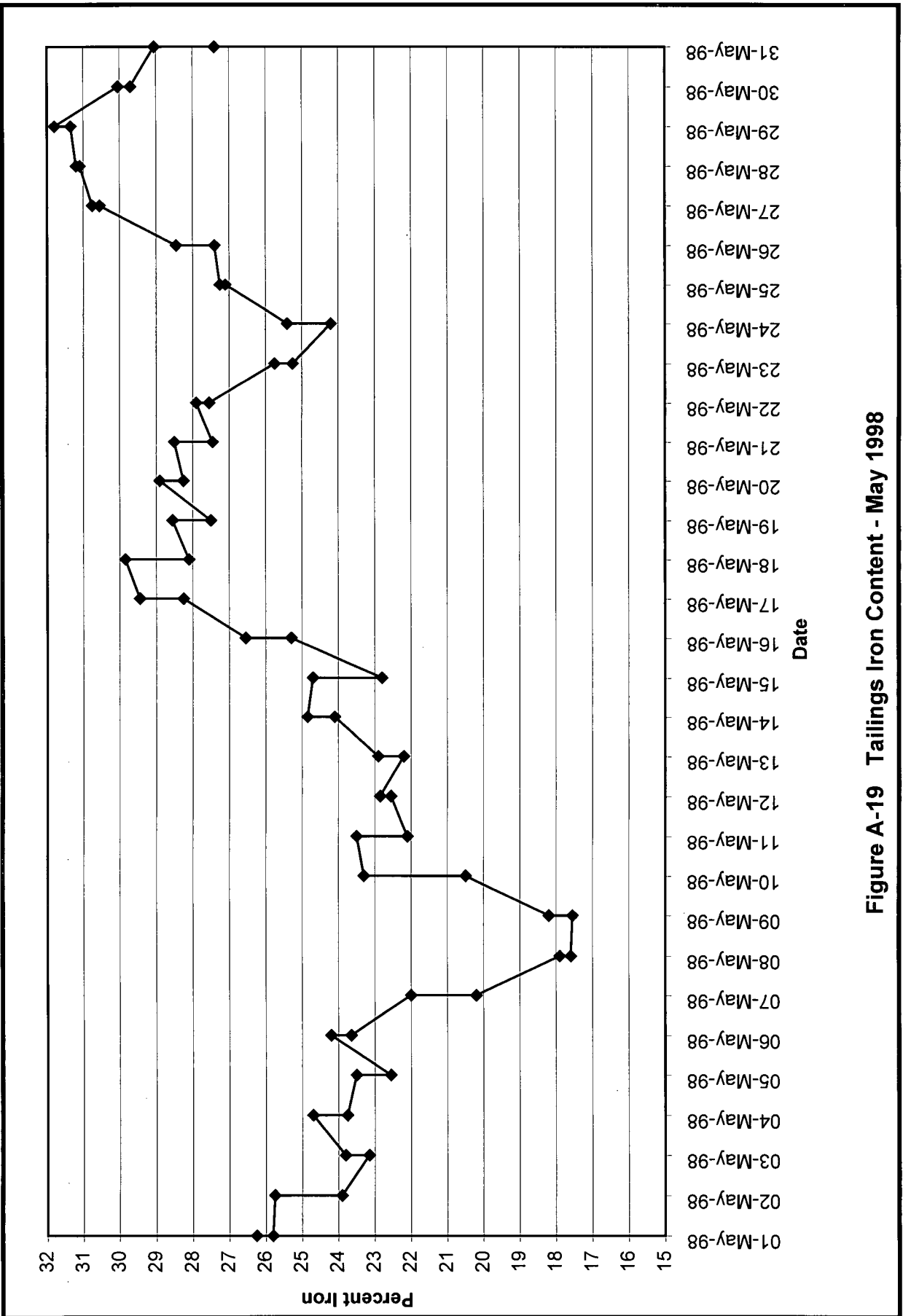


Figure A-19 Tailings Iron Content - May 1998

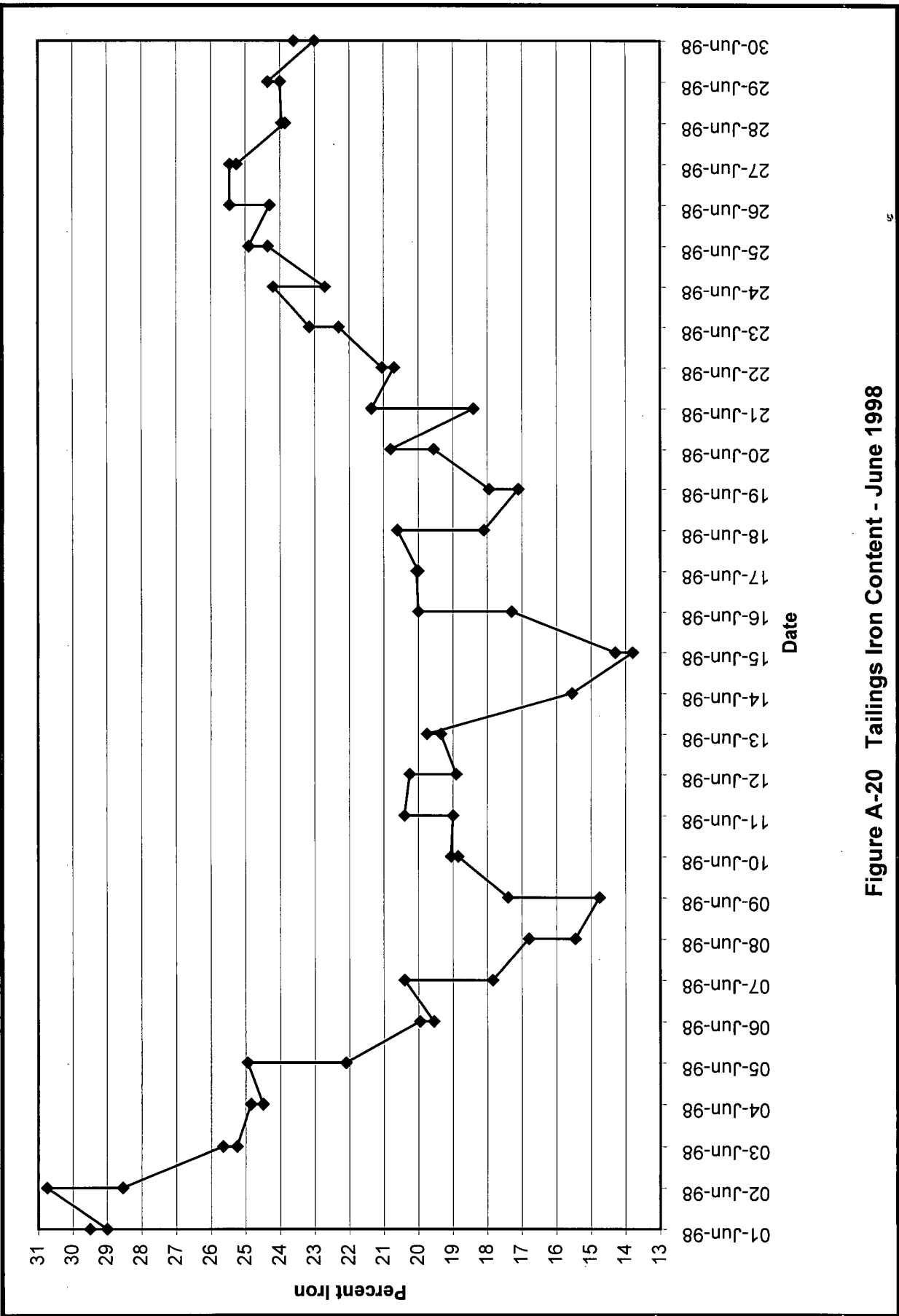


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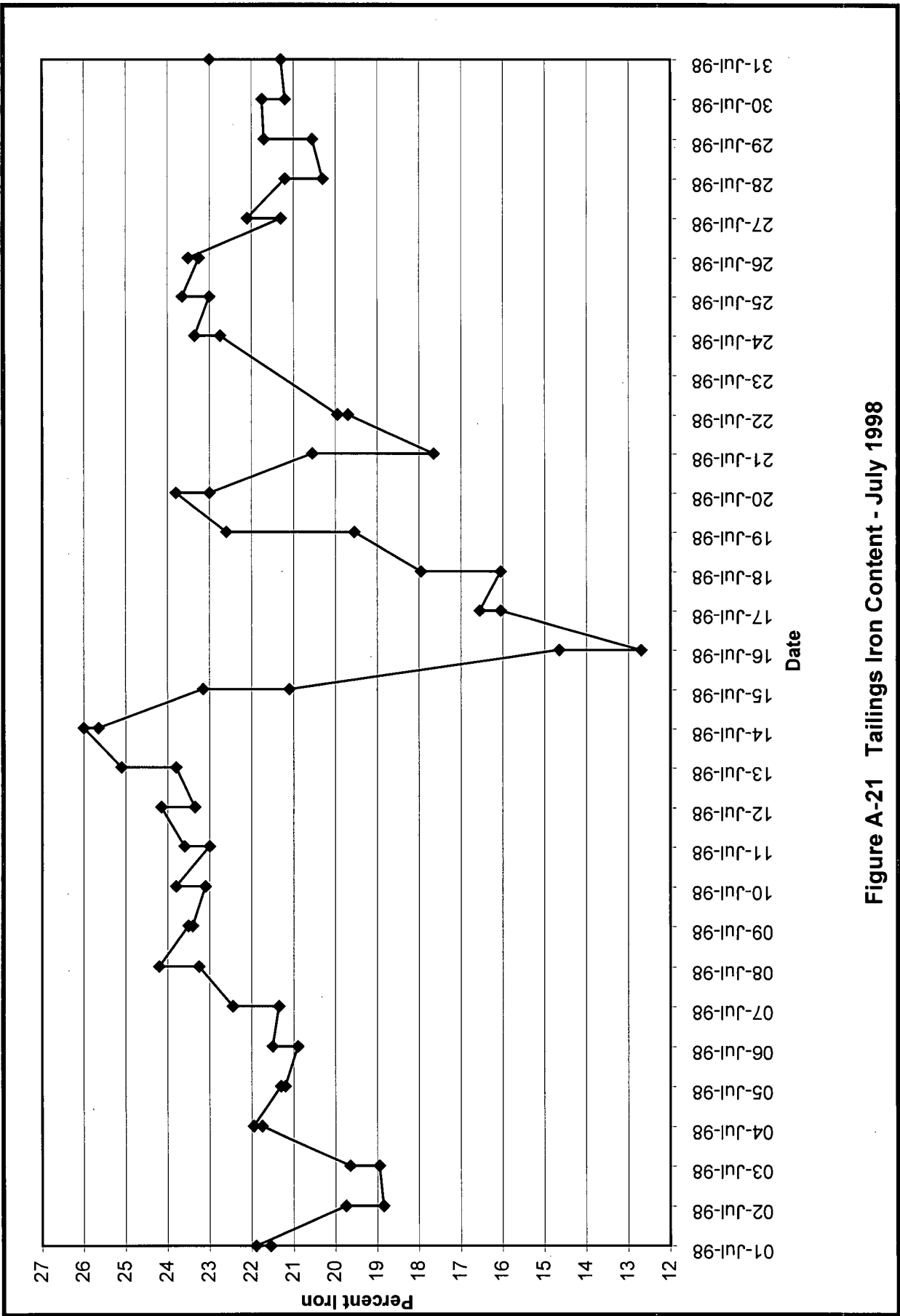
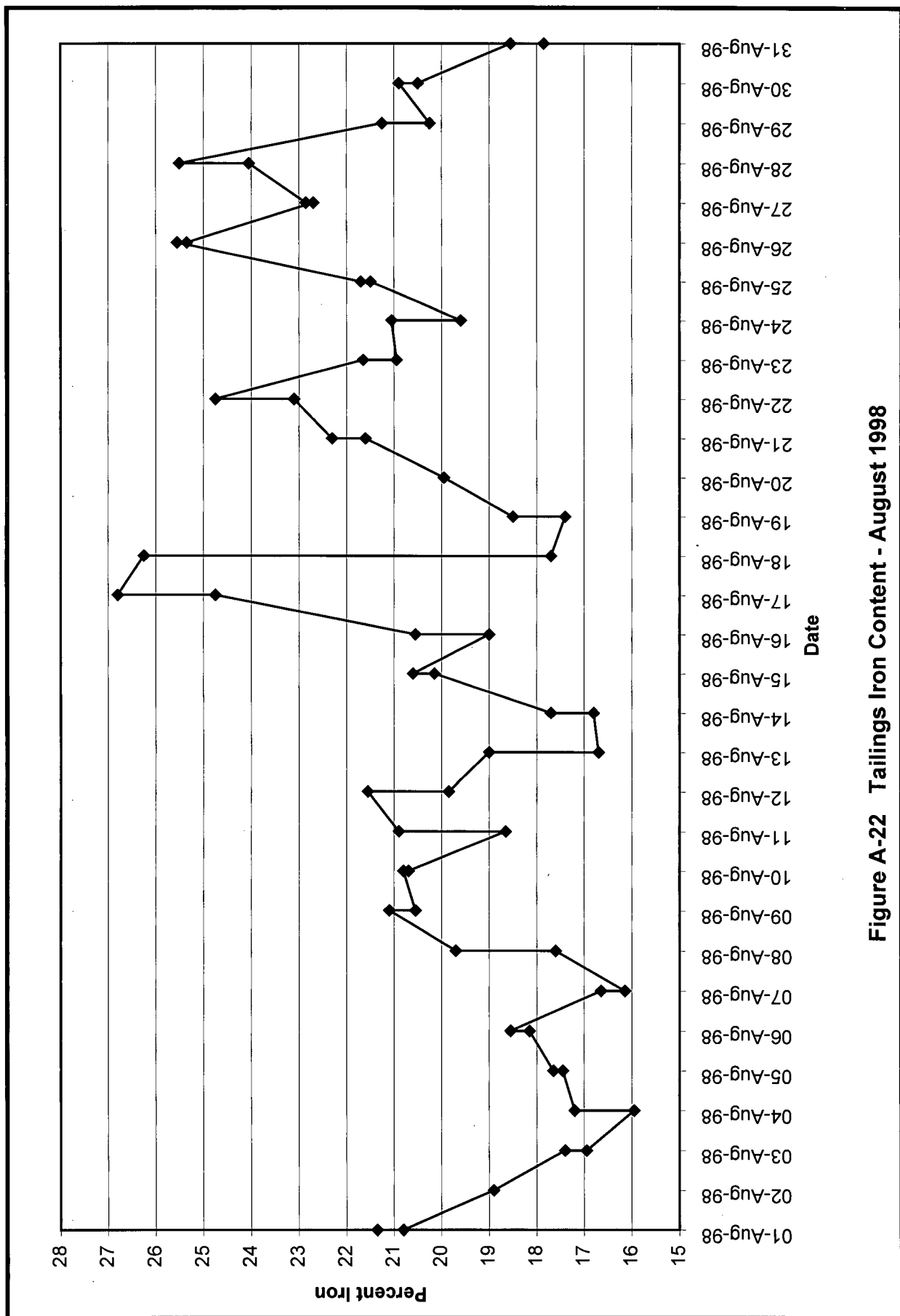


Figure A-21 Tailings Iron Content - July 1998



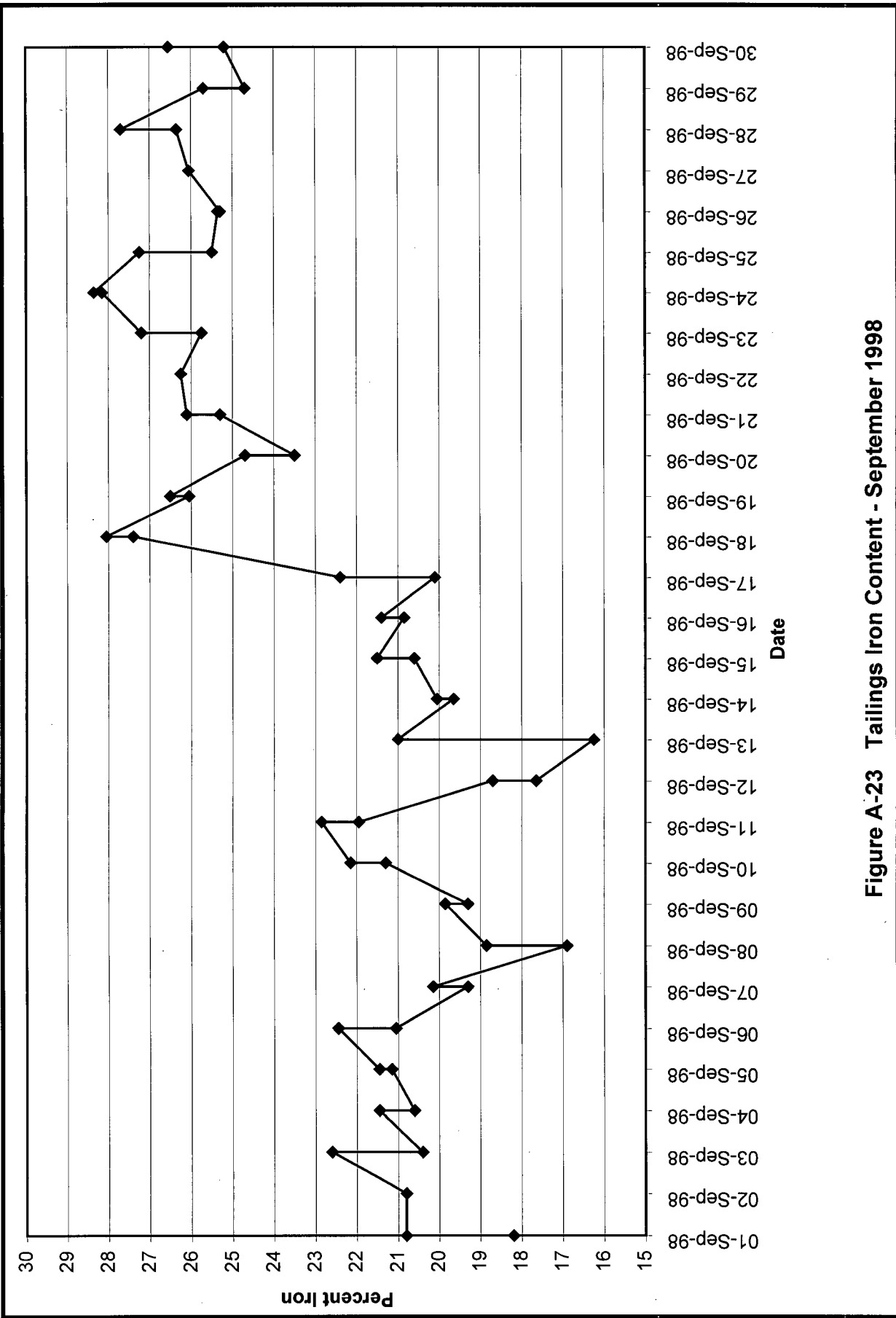


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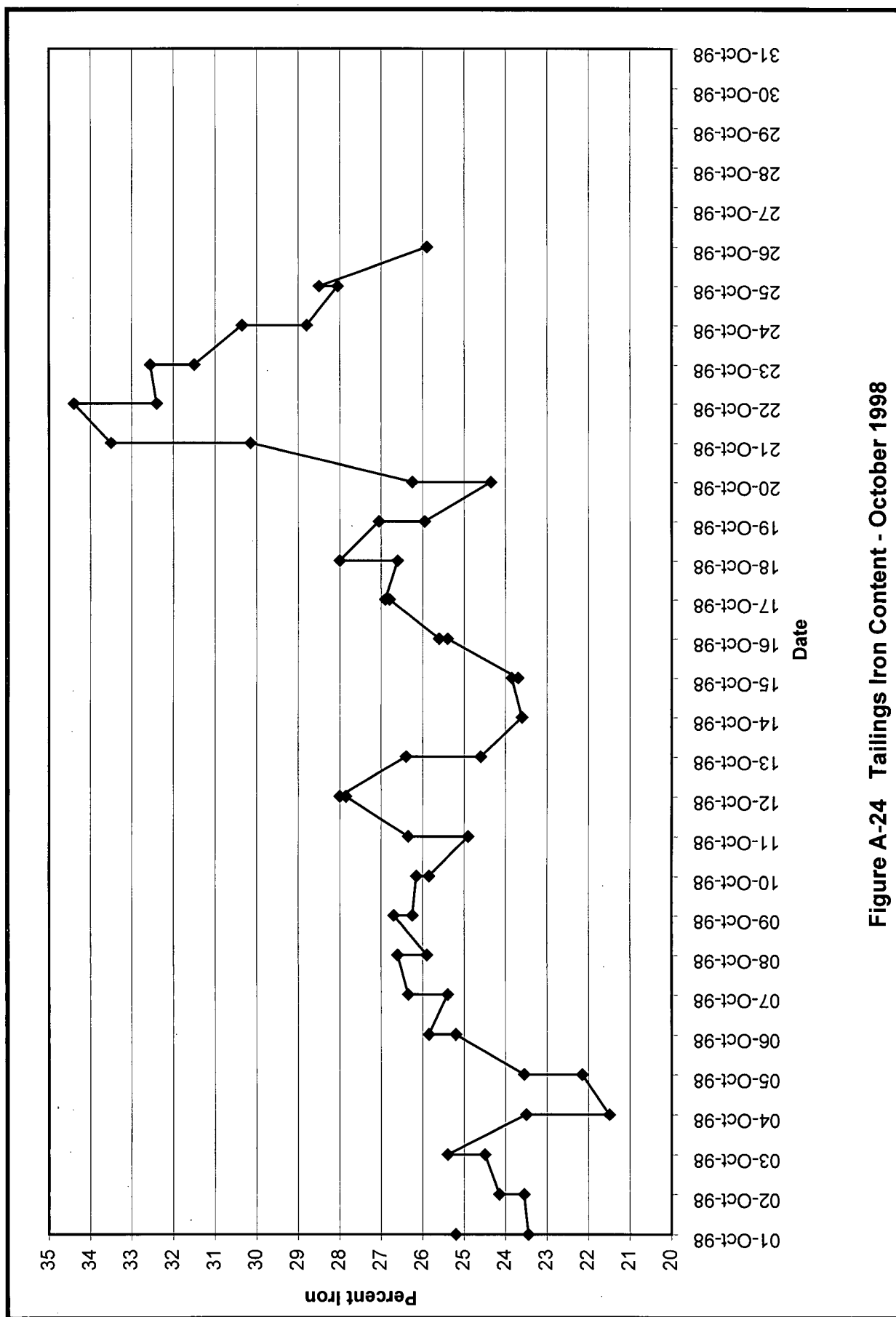


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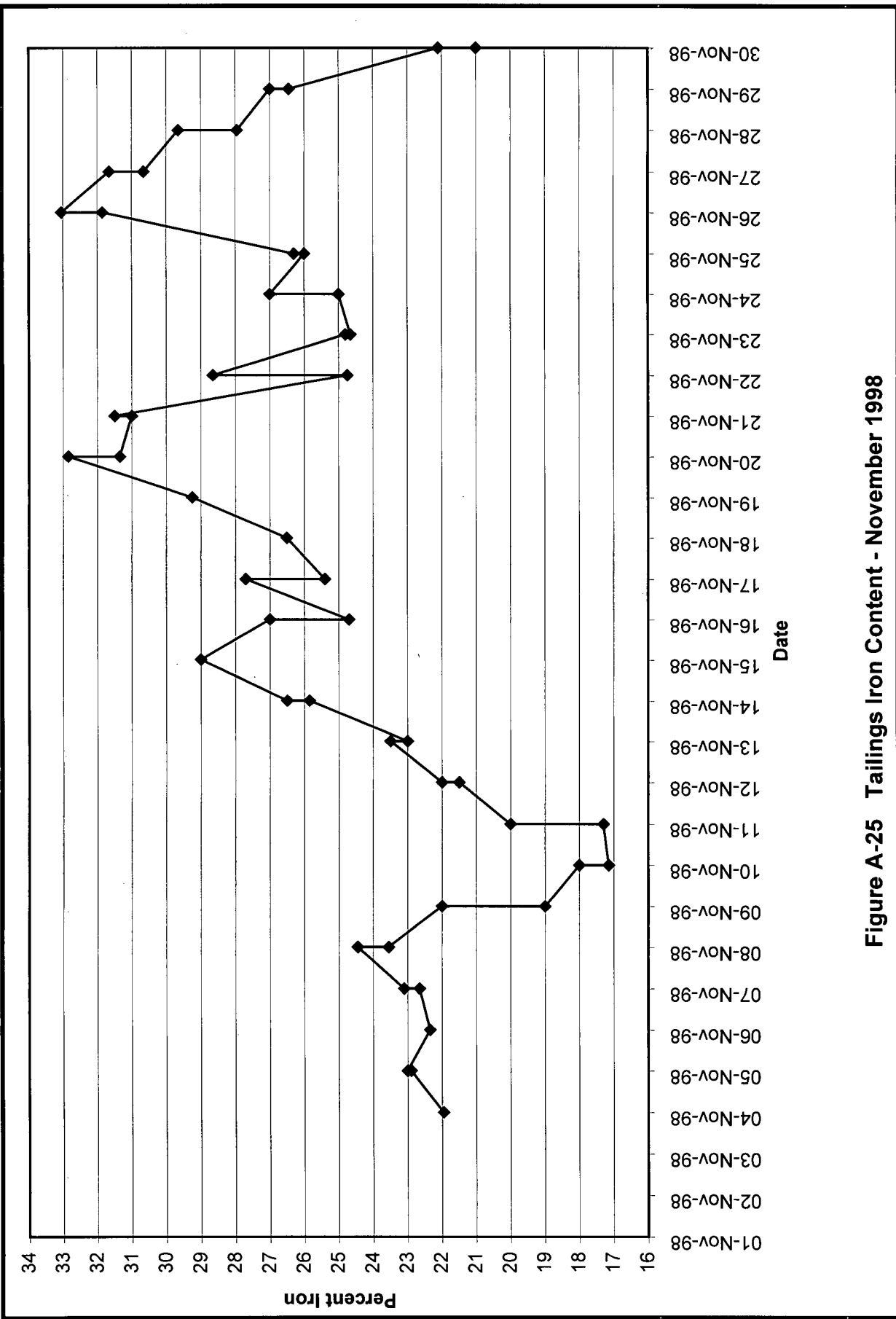


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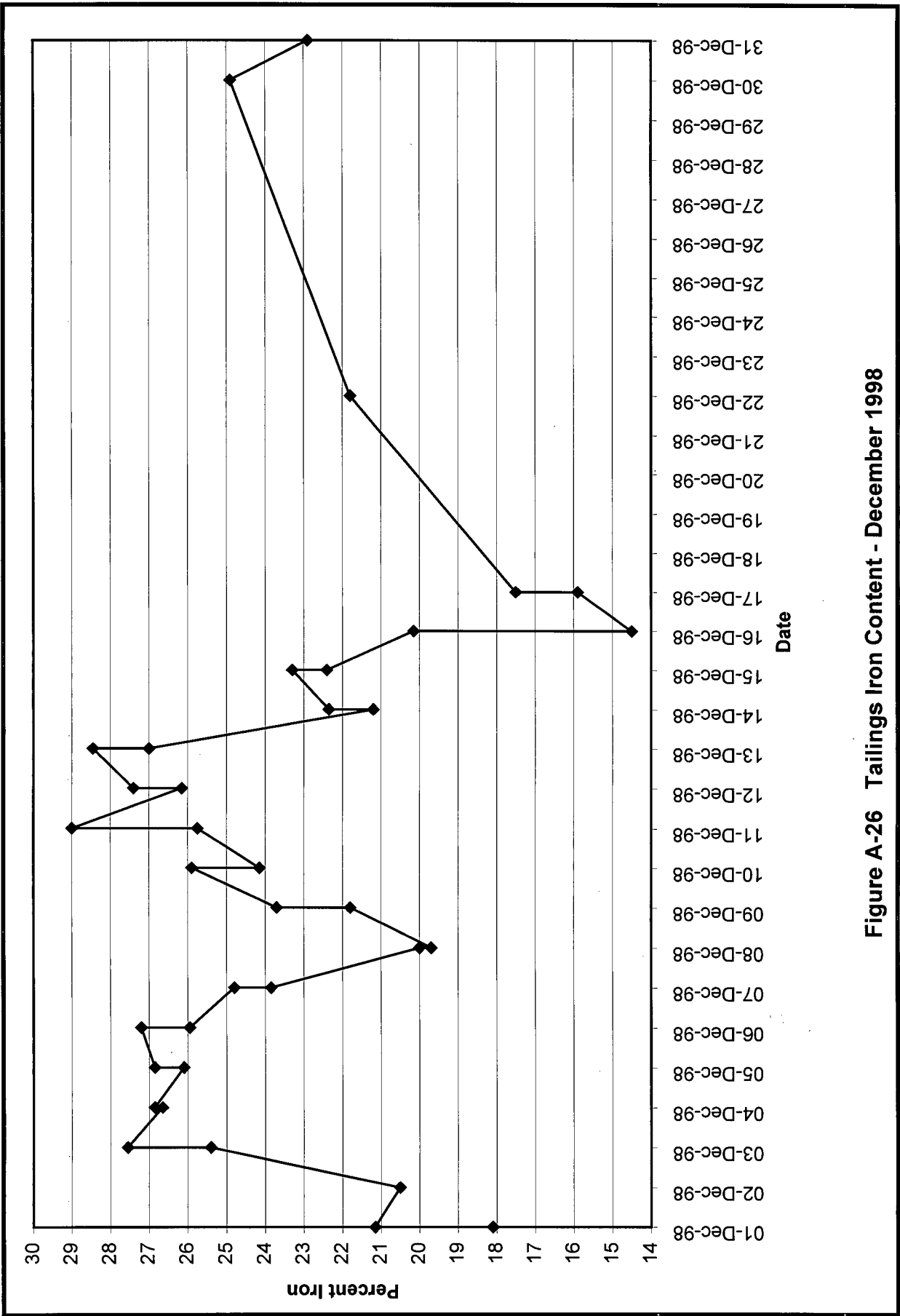
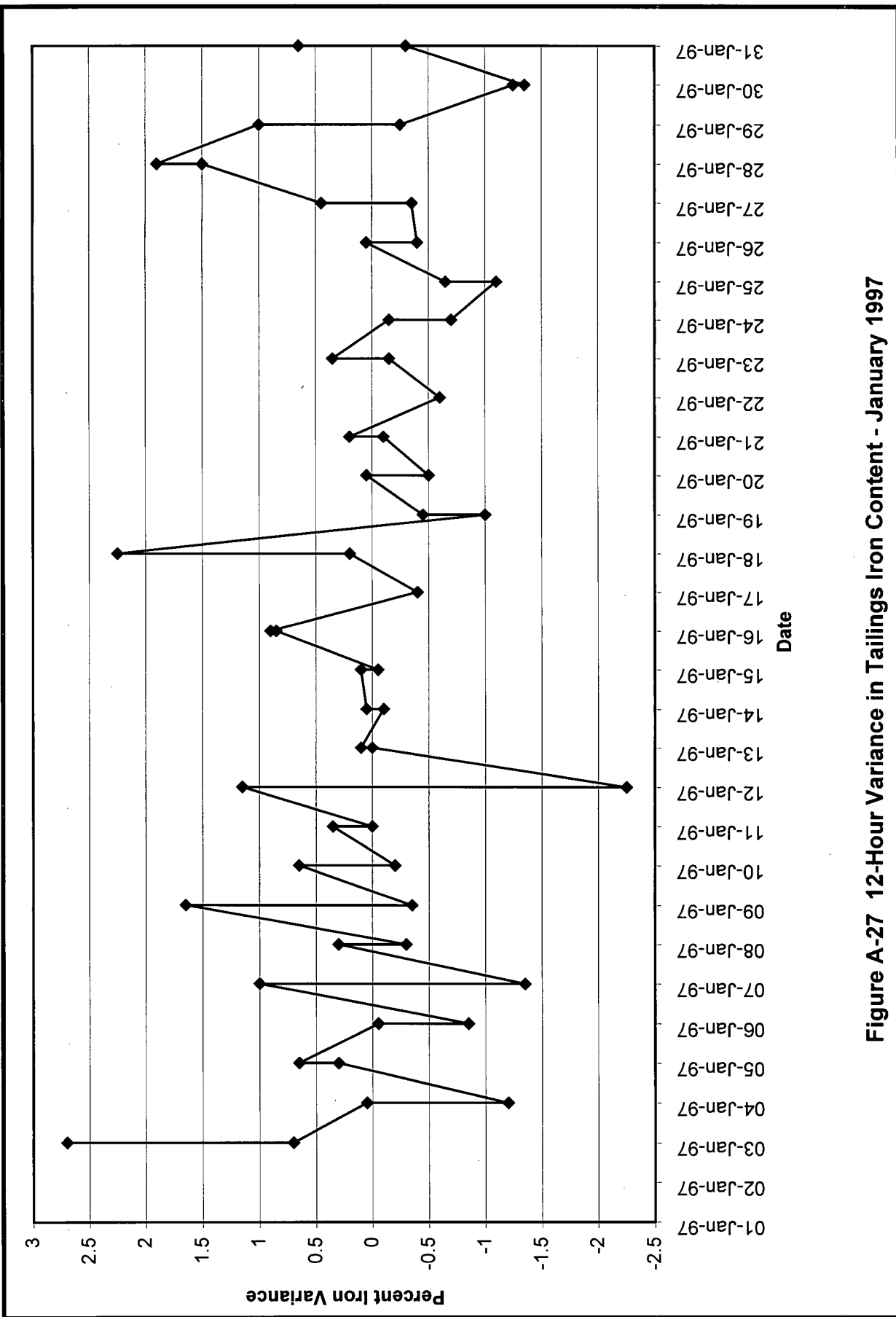
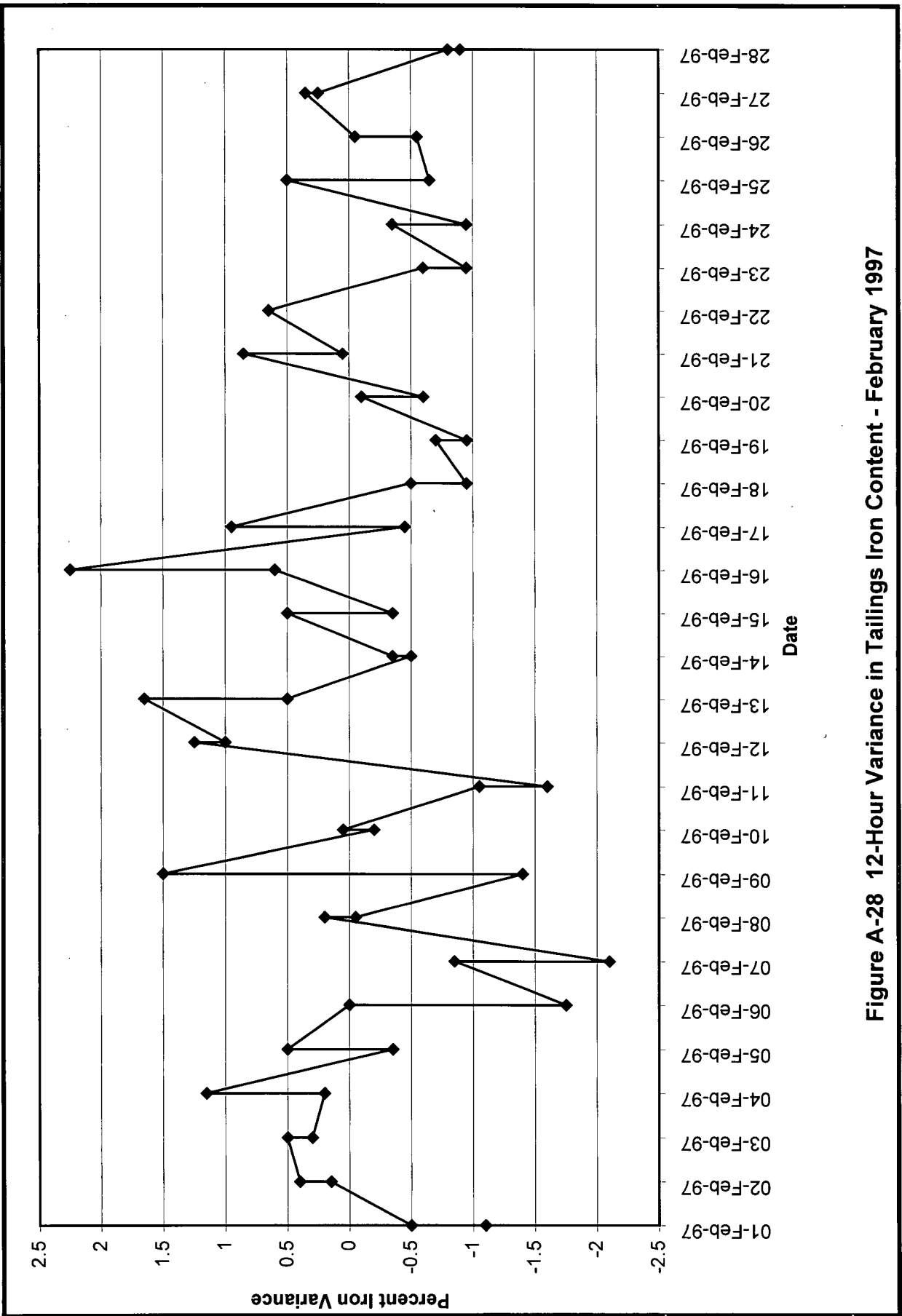


Figure A-26 Tailings Iron Content - December 1998





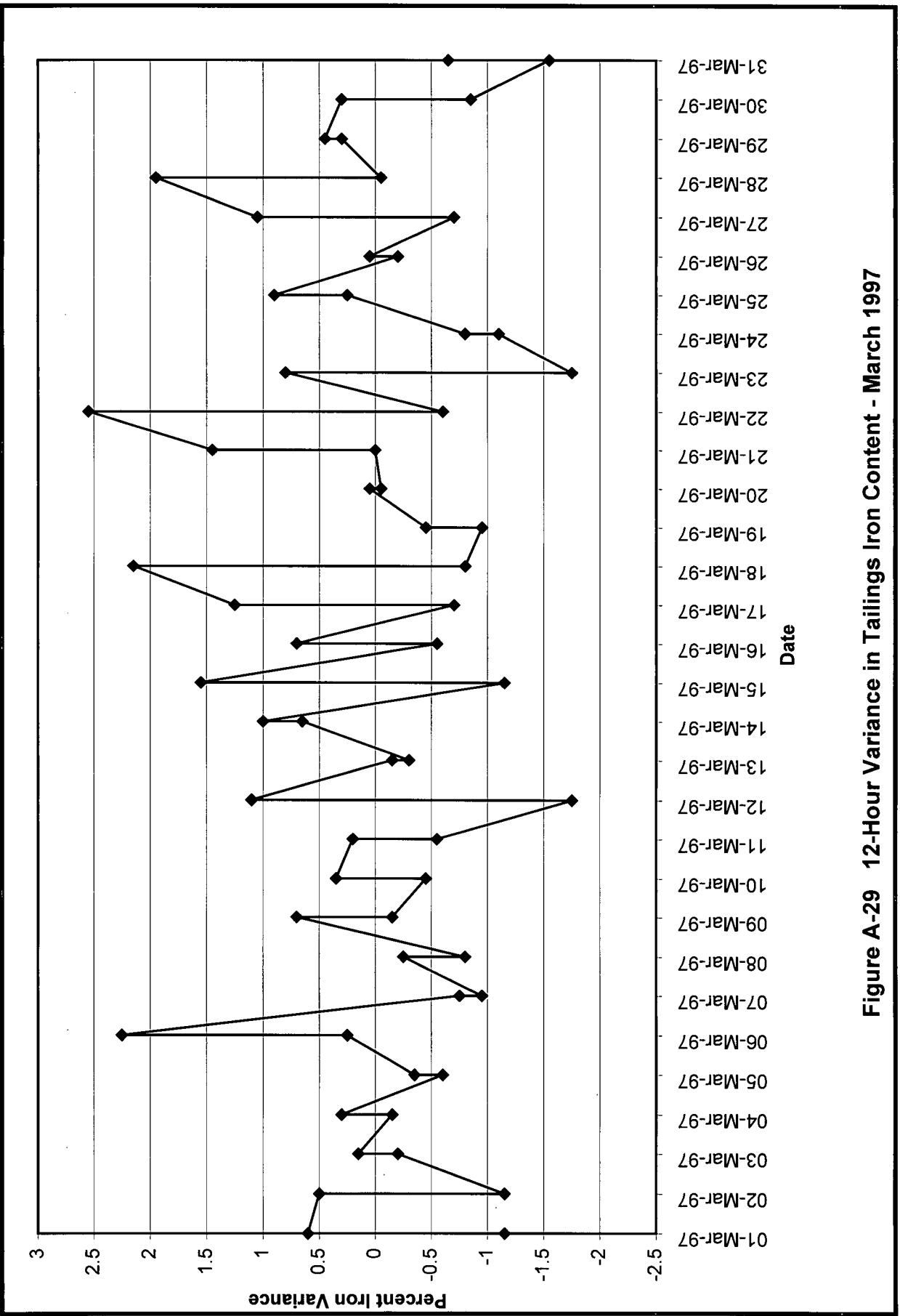
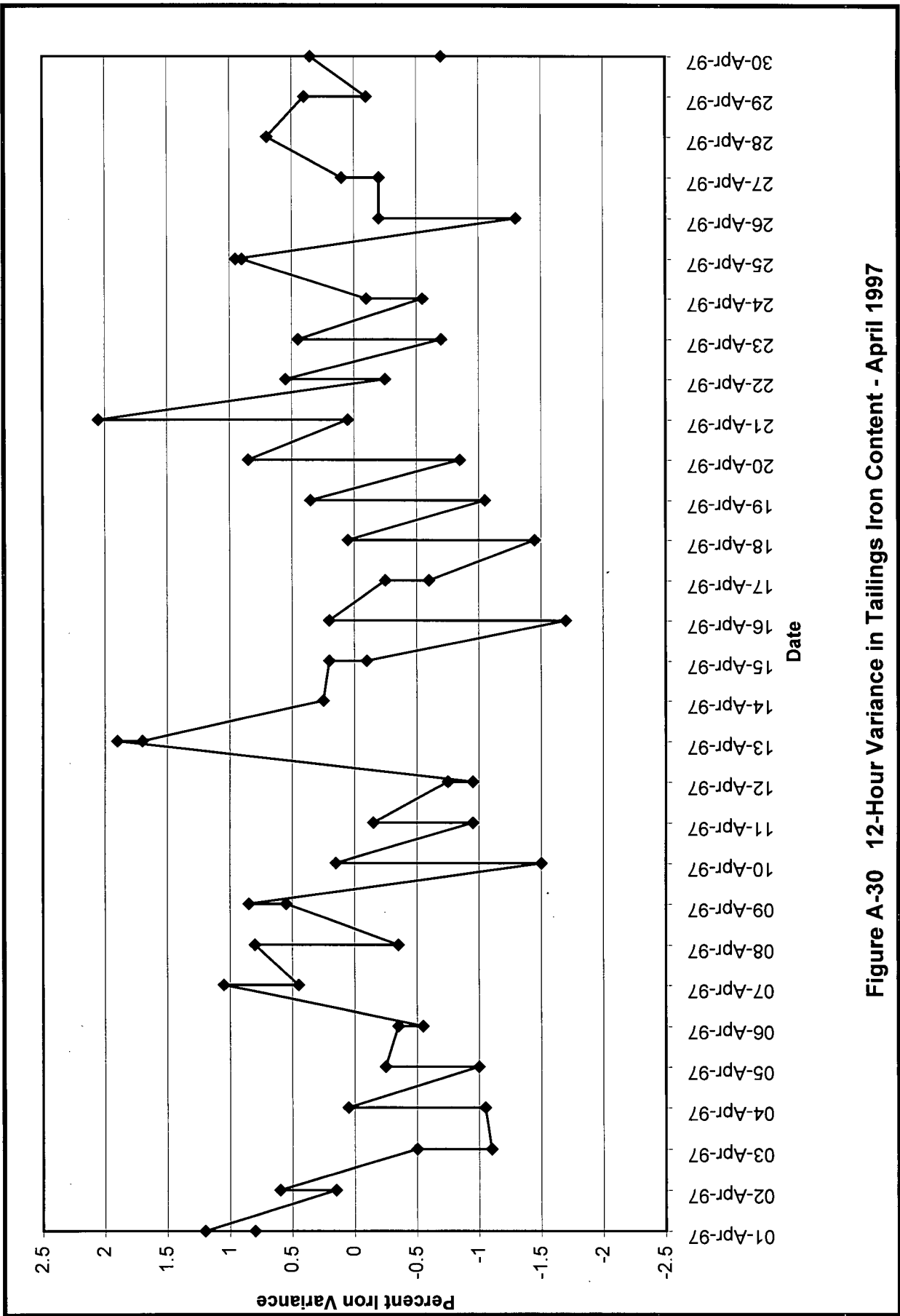
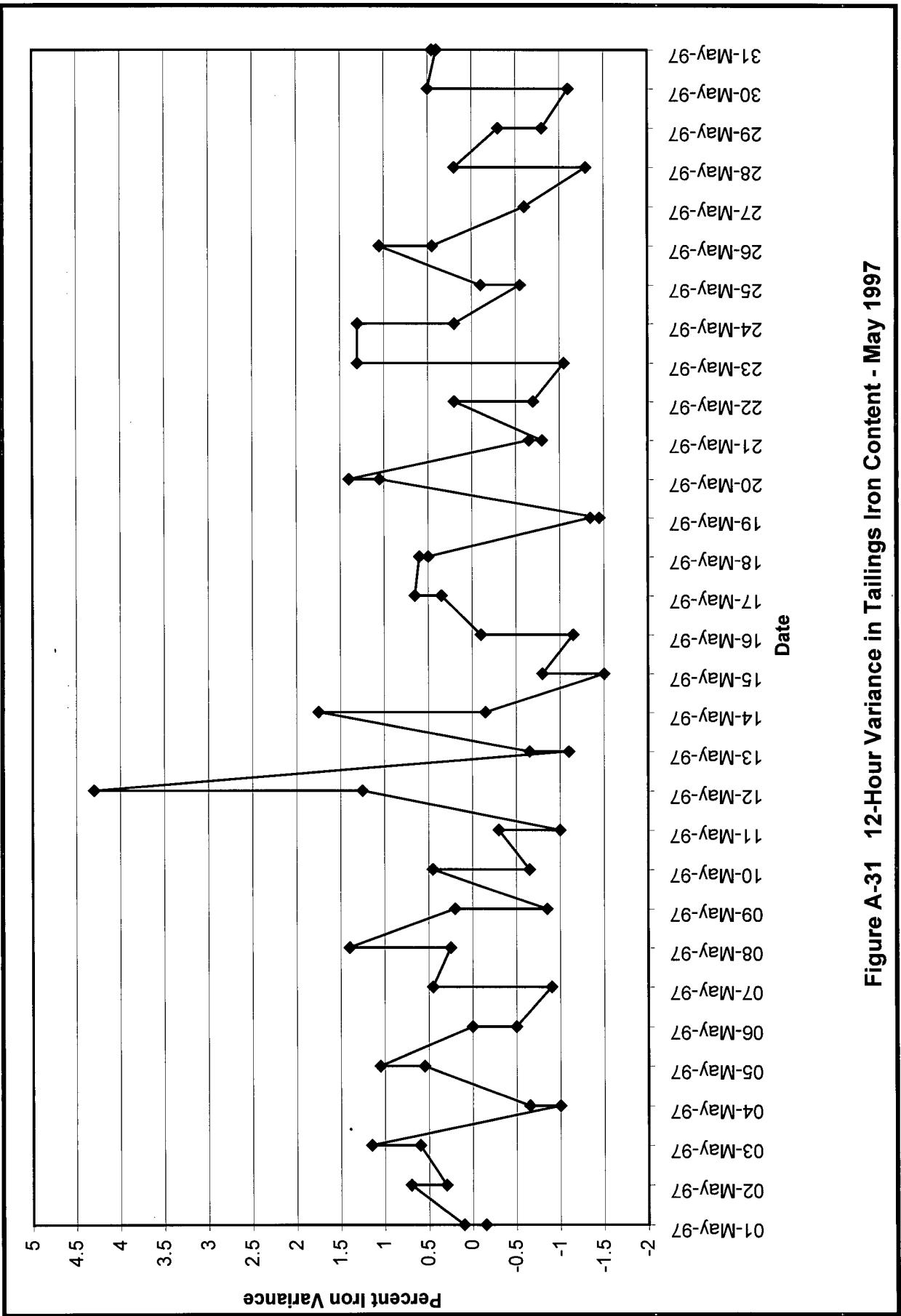
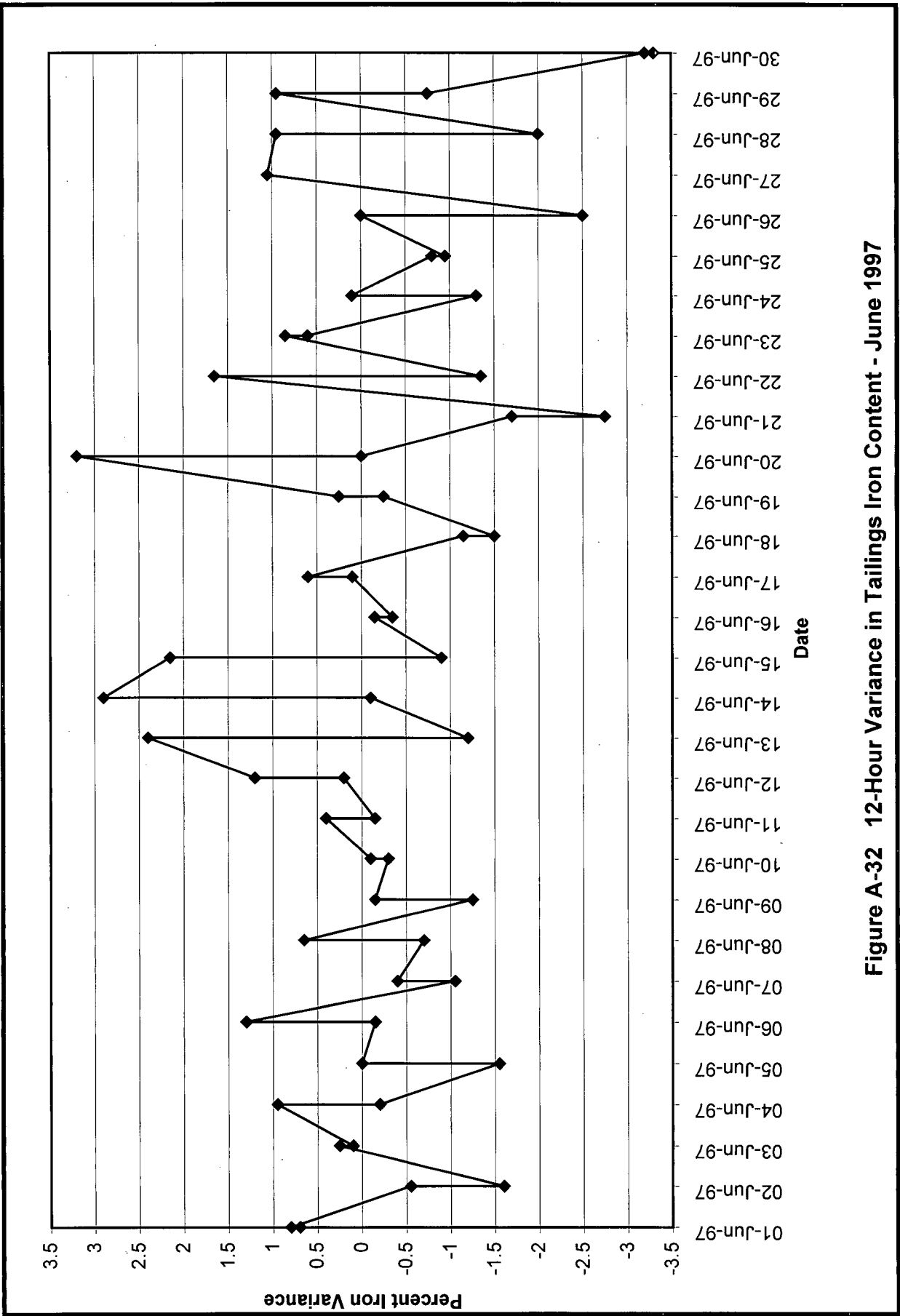
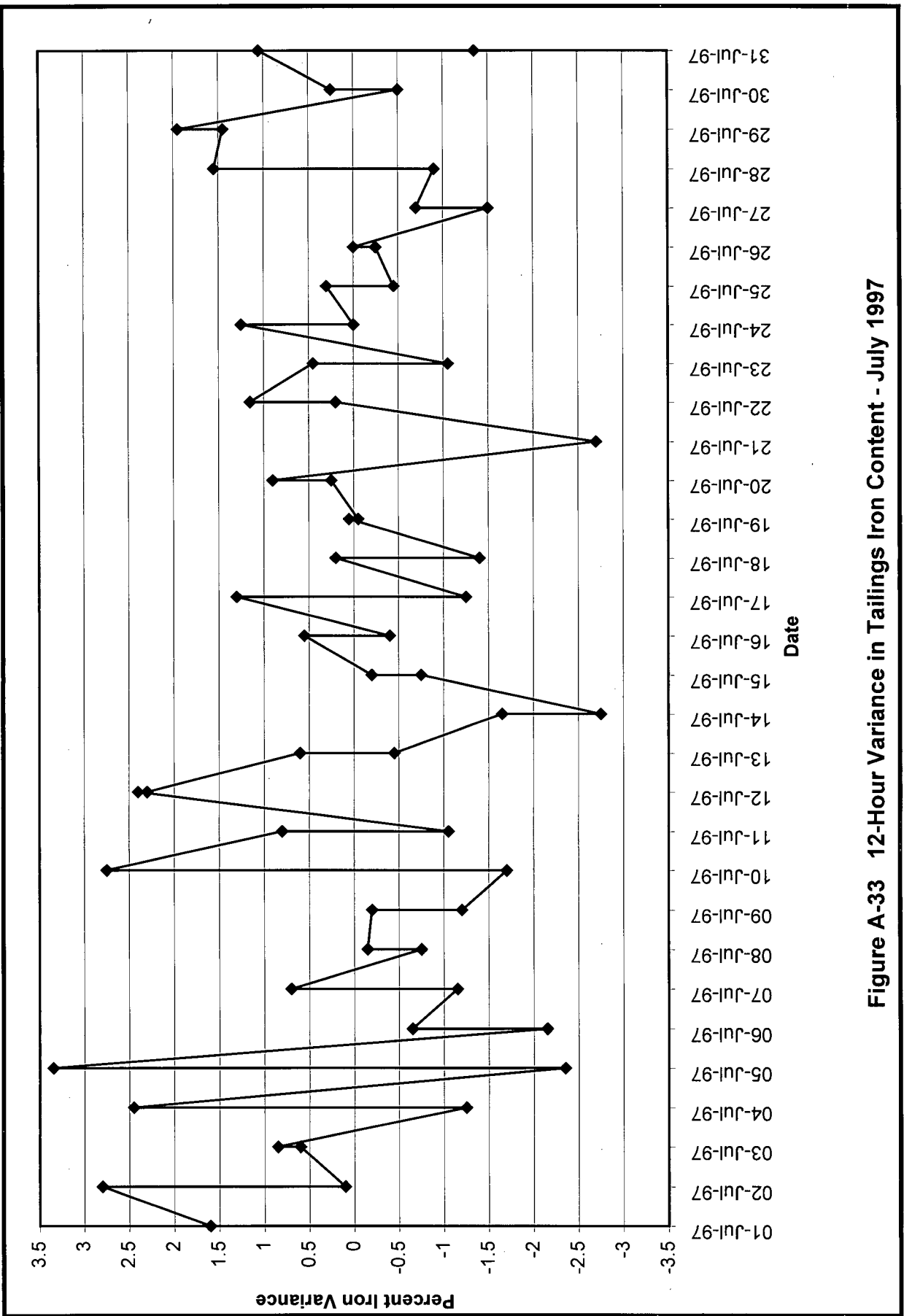


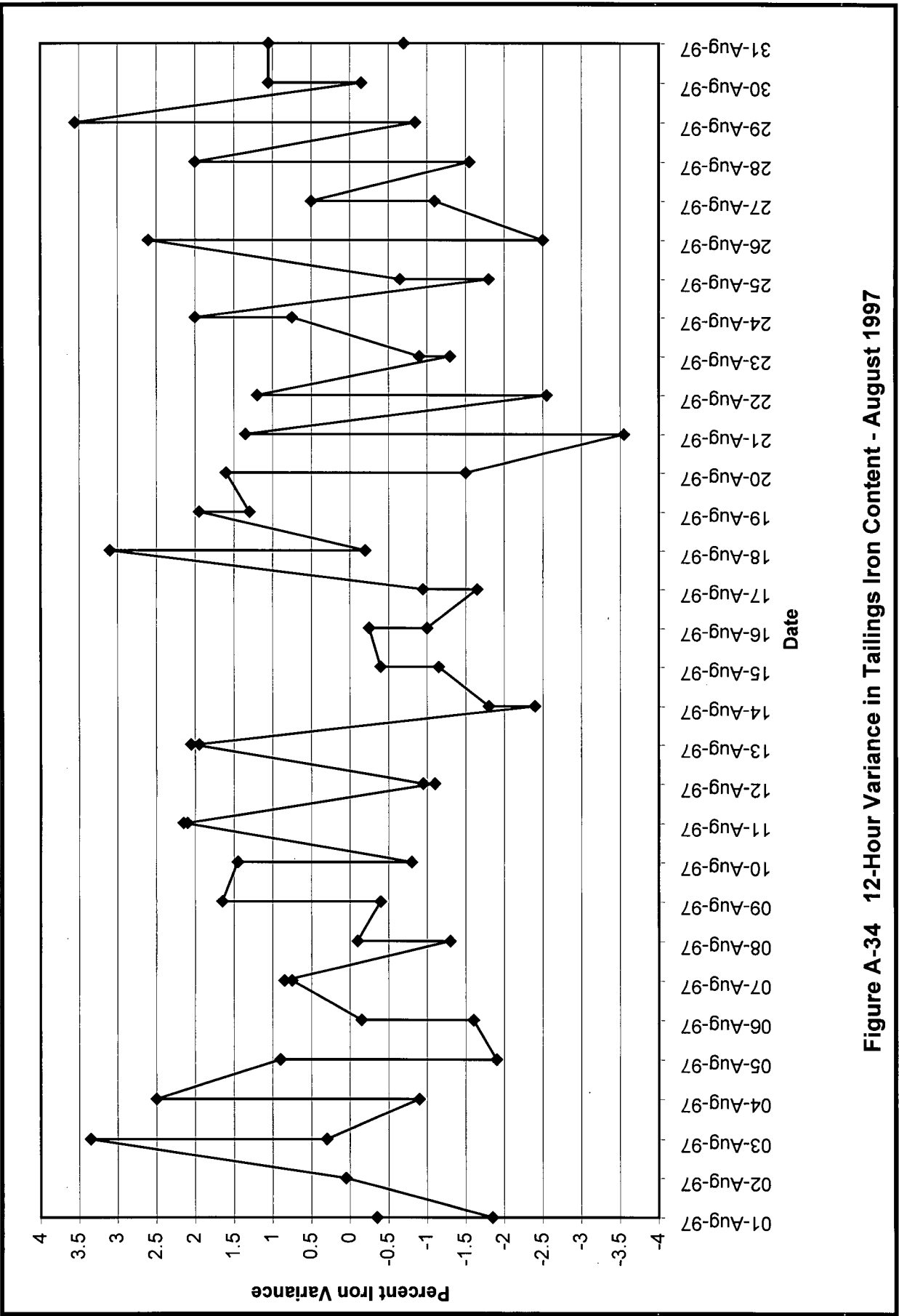
Figure A-29 12-Hour Variance in Tailings Iron Content - March 1997











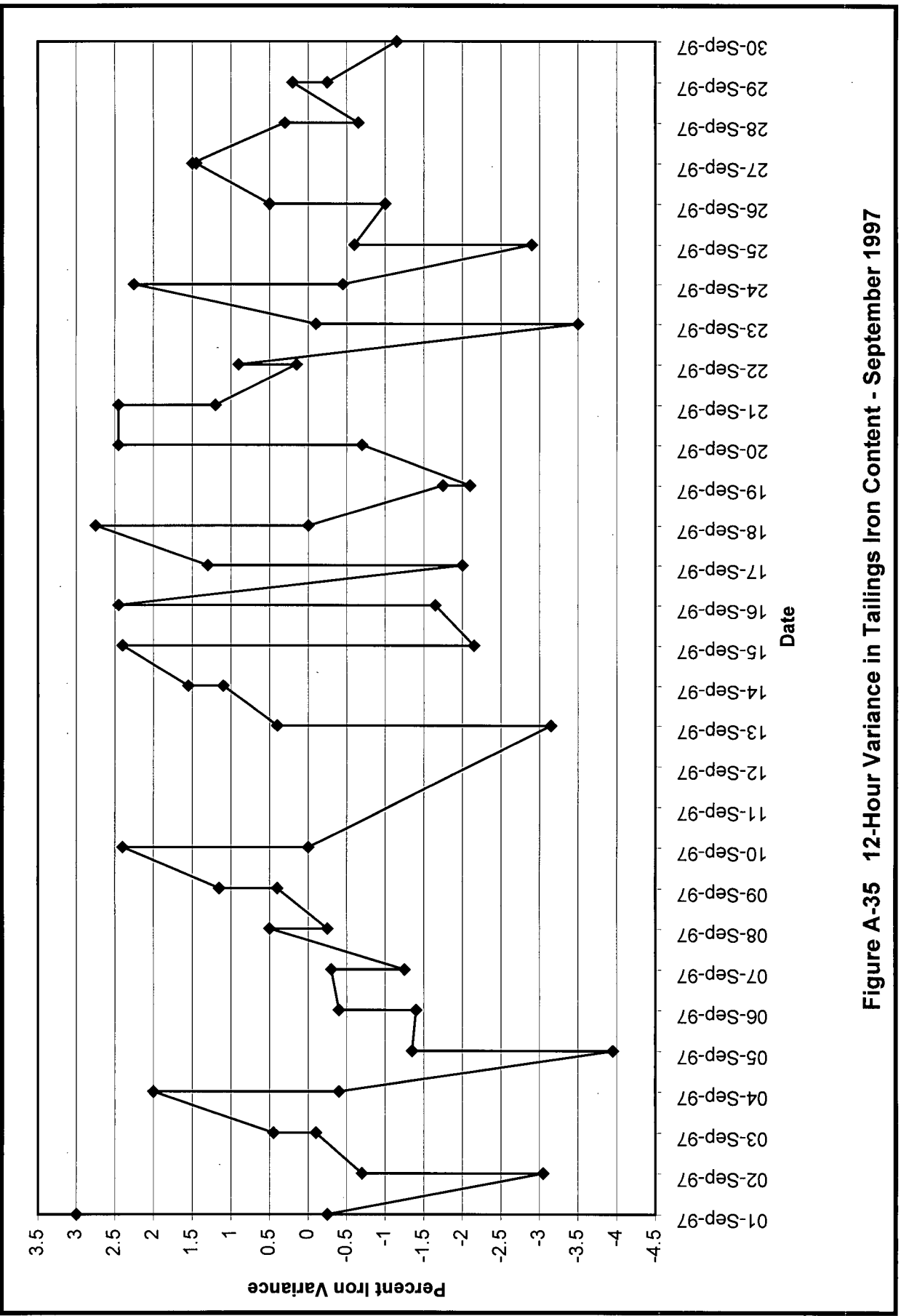


Figure A-35 12-Hour Variance in Tailings Iron Content - September 1997

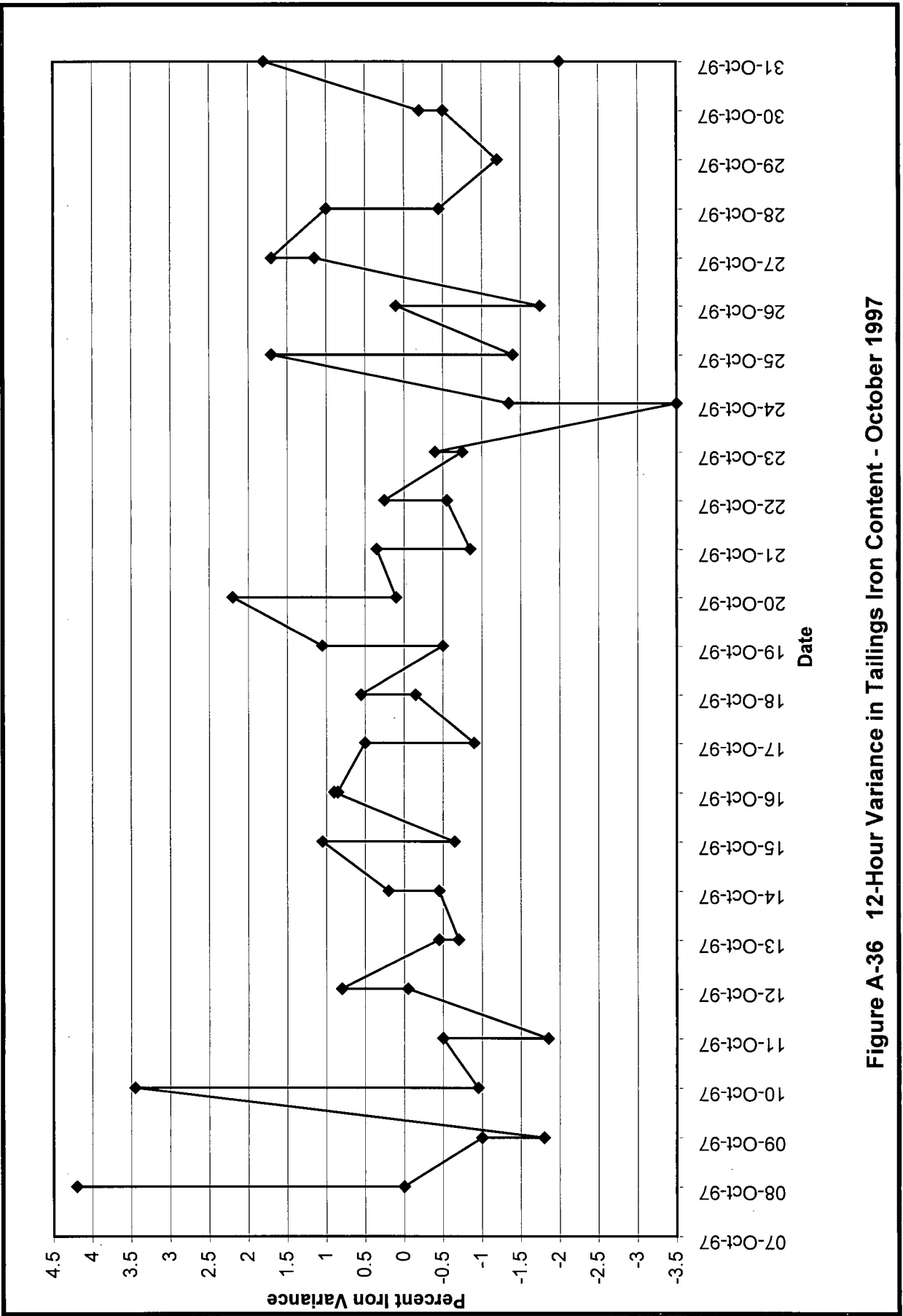


Figure A-36 12-Hour Variance in Tailings Iron Content - October 1997

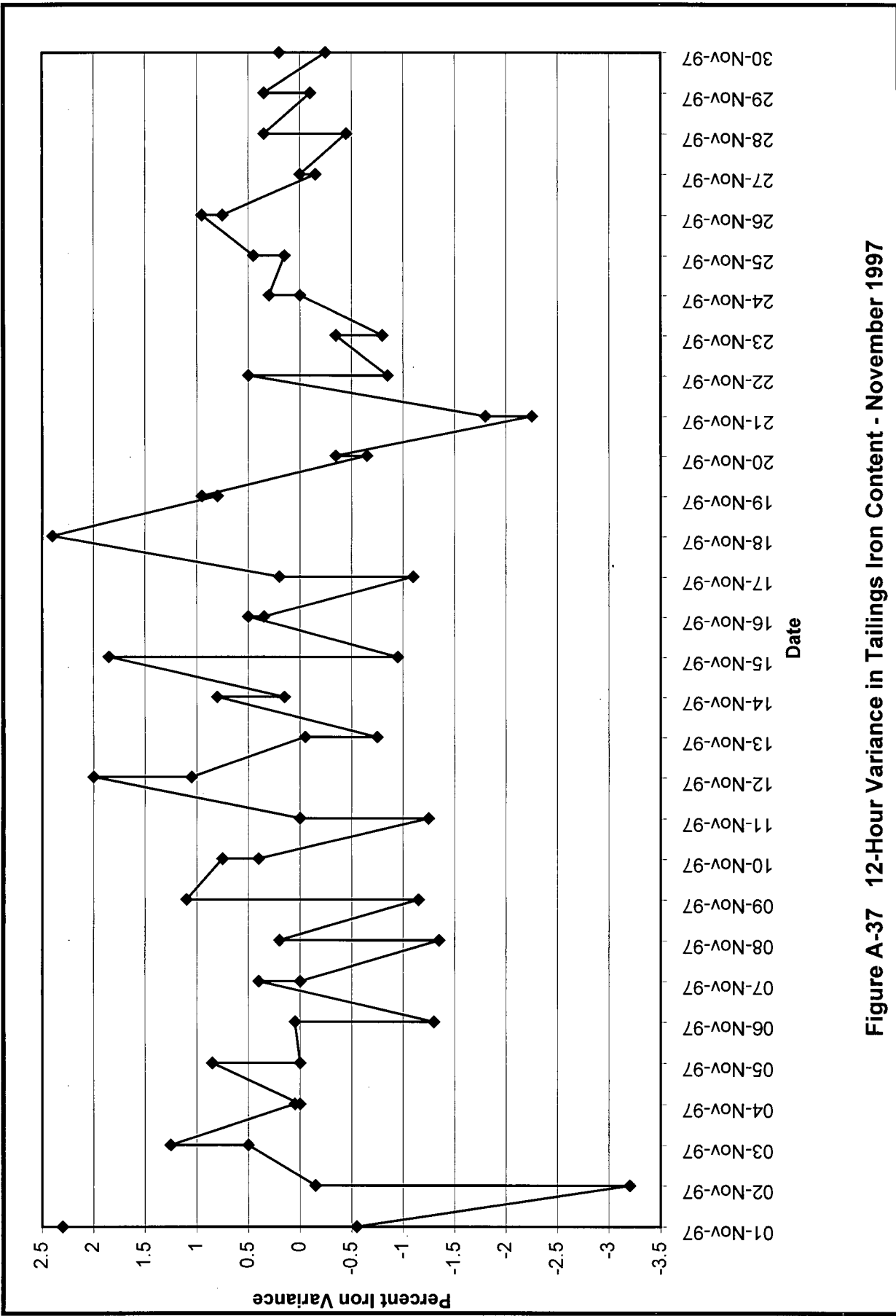


Figure A-37 12-Hour Variance in Tailings Iron Content - November 1997

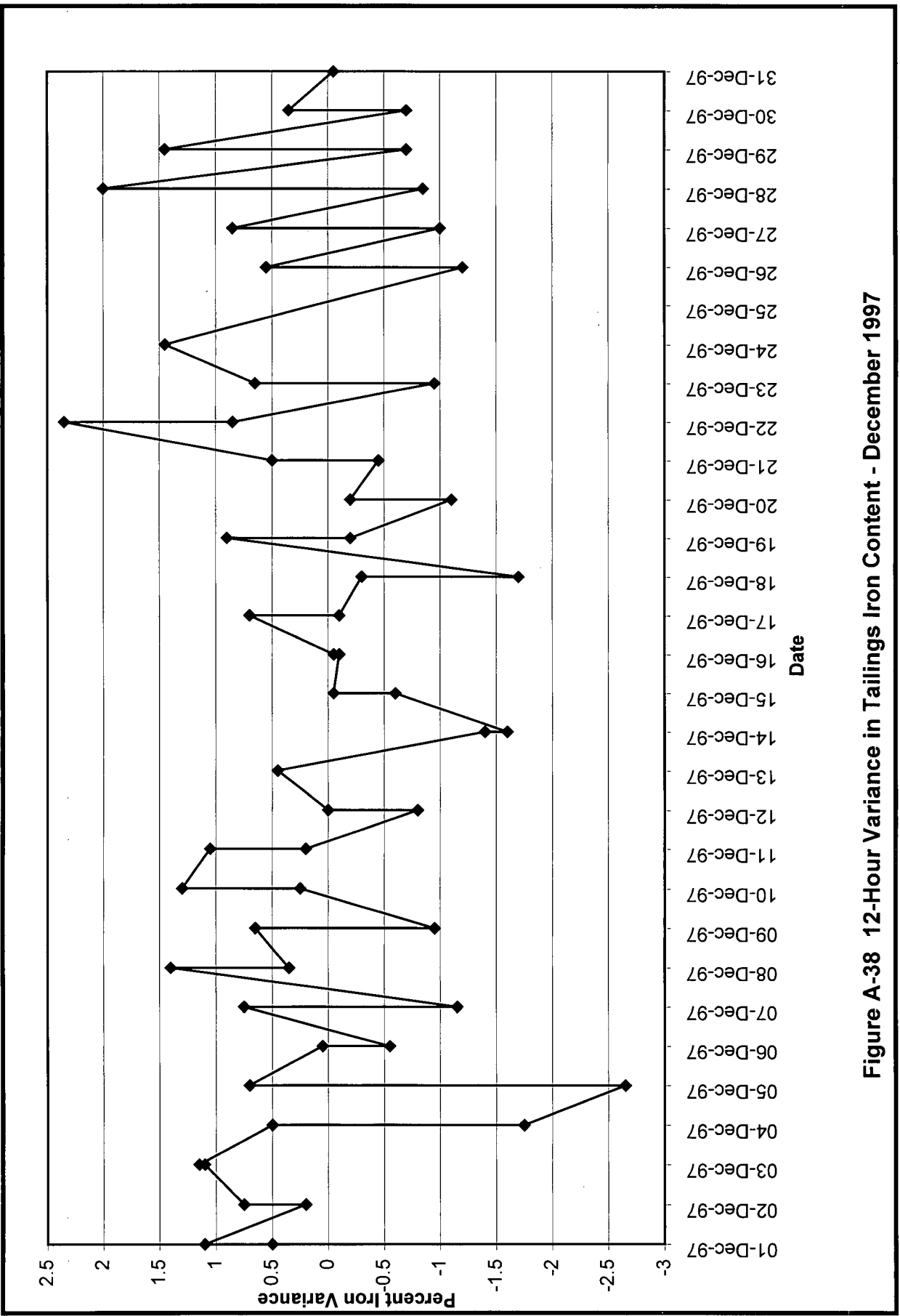
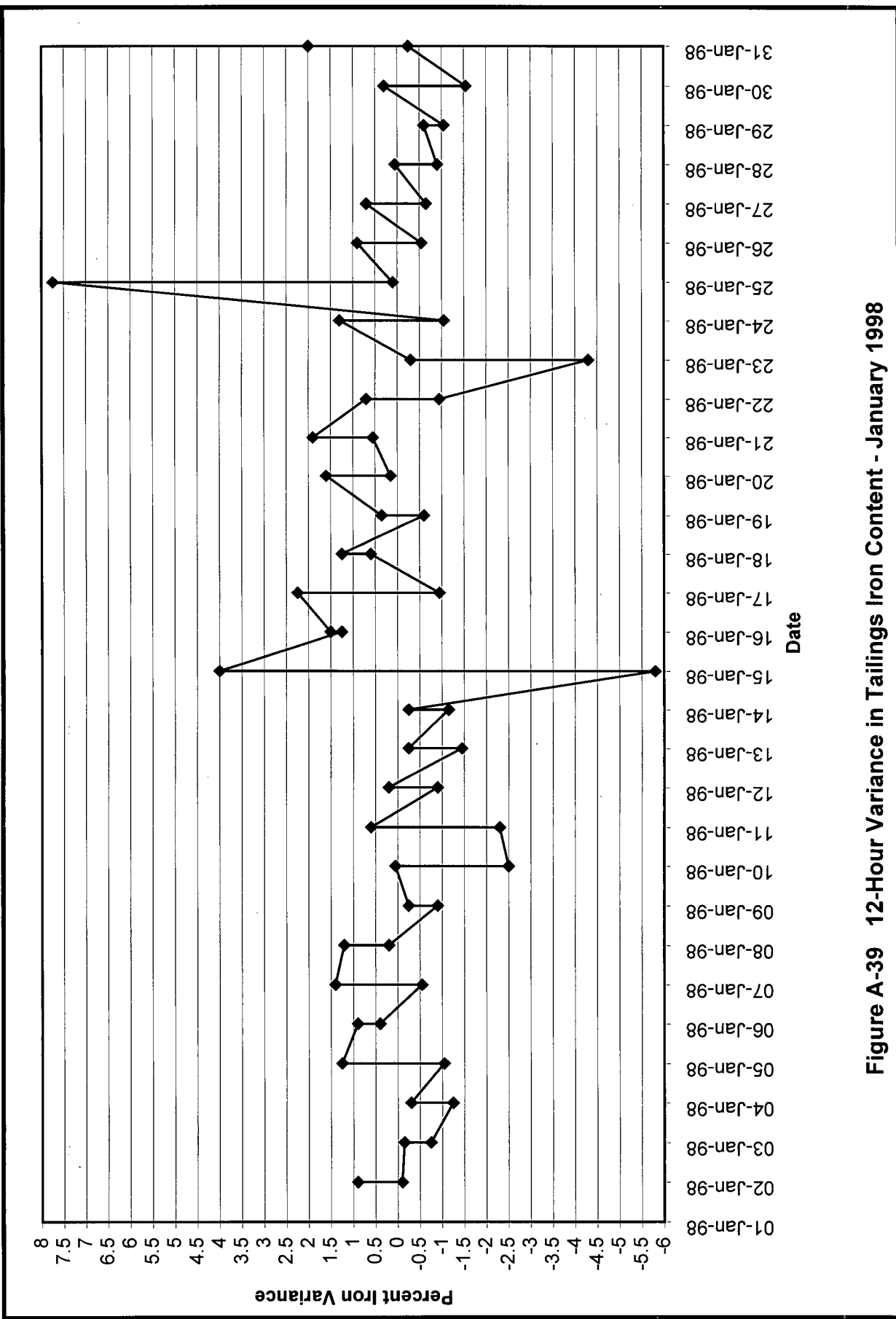


Figure A-38 12-Hour Variance in Tailings Iron Content - December 1997



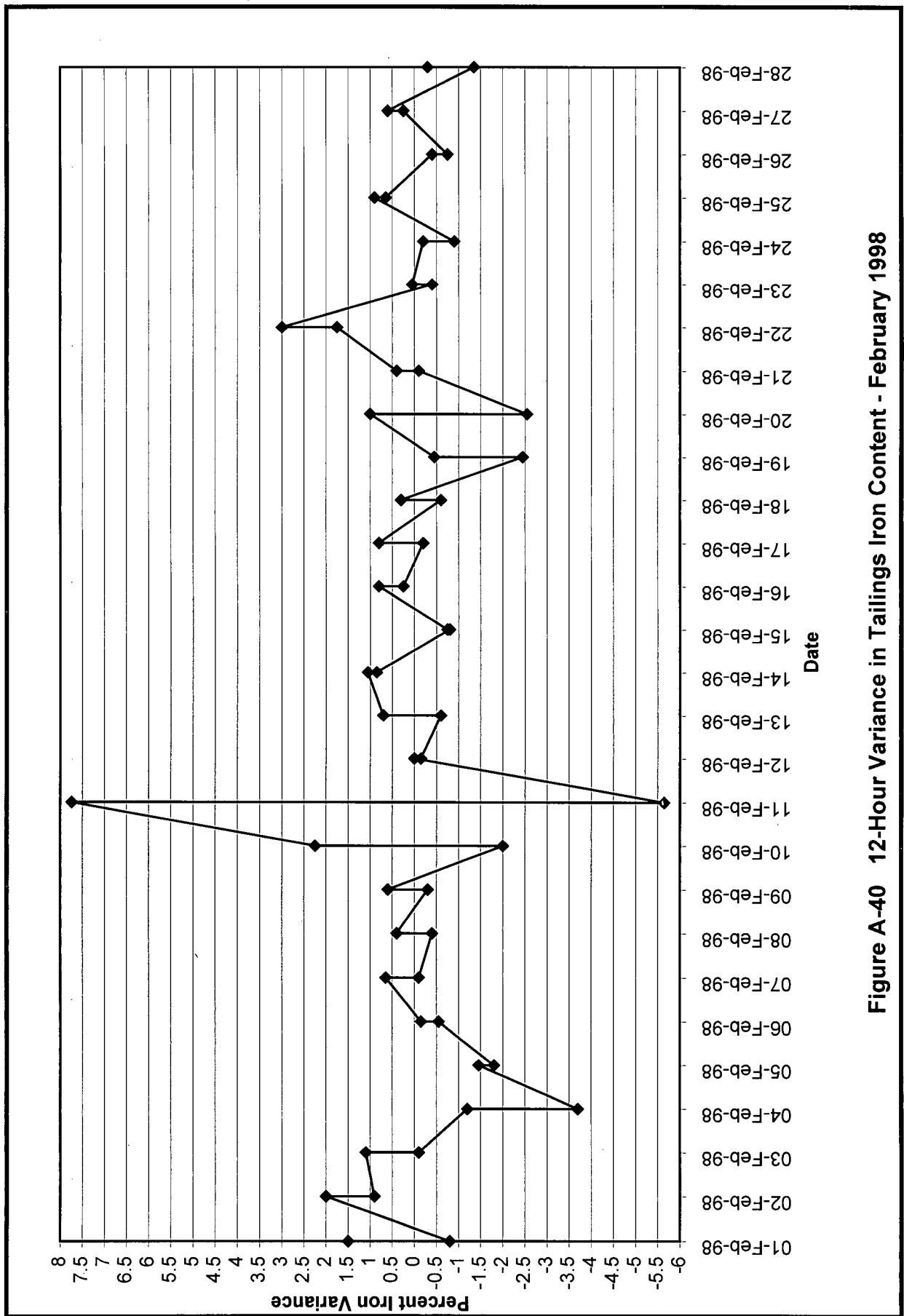
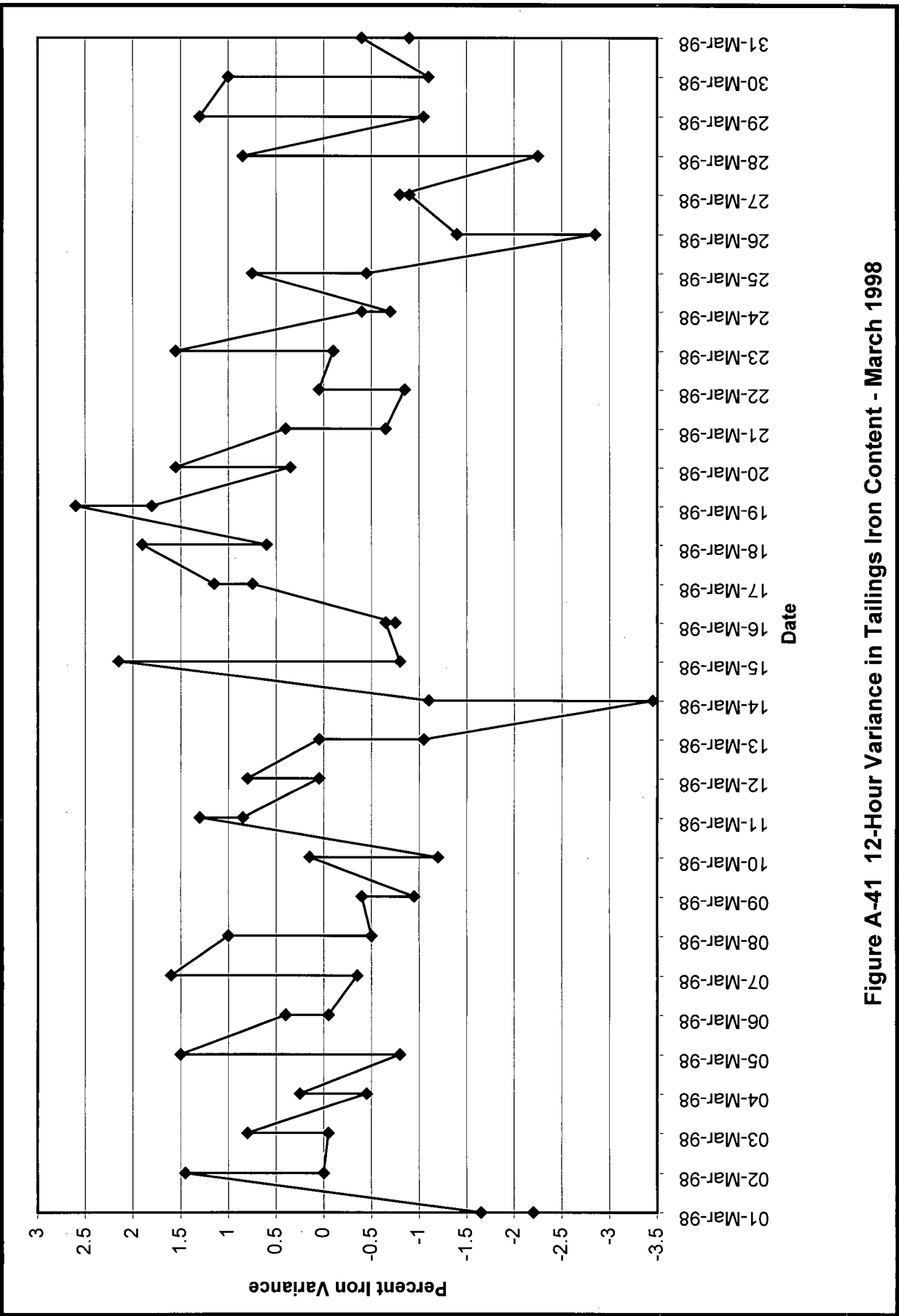


Figure A-40 12-Hour Variance in Tailings Iron Content - February 1998



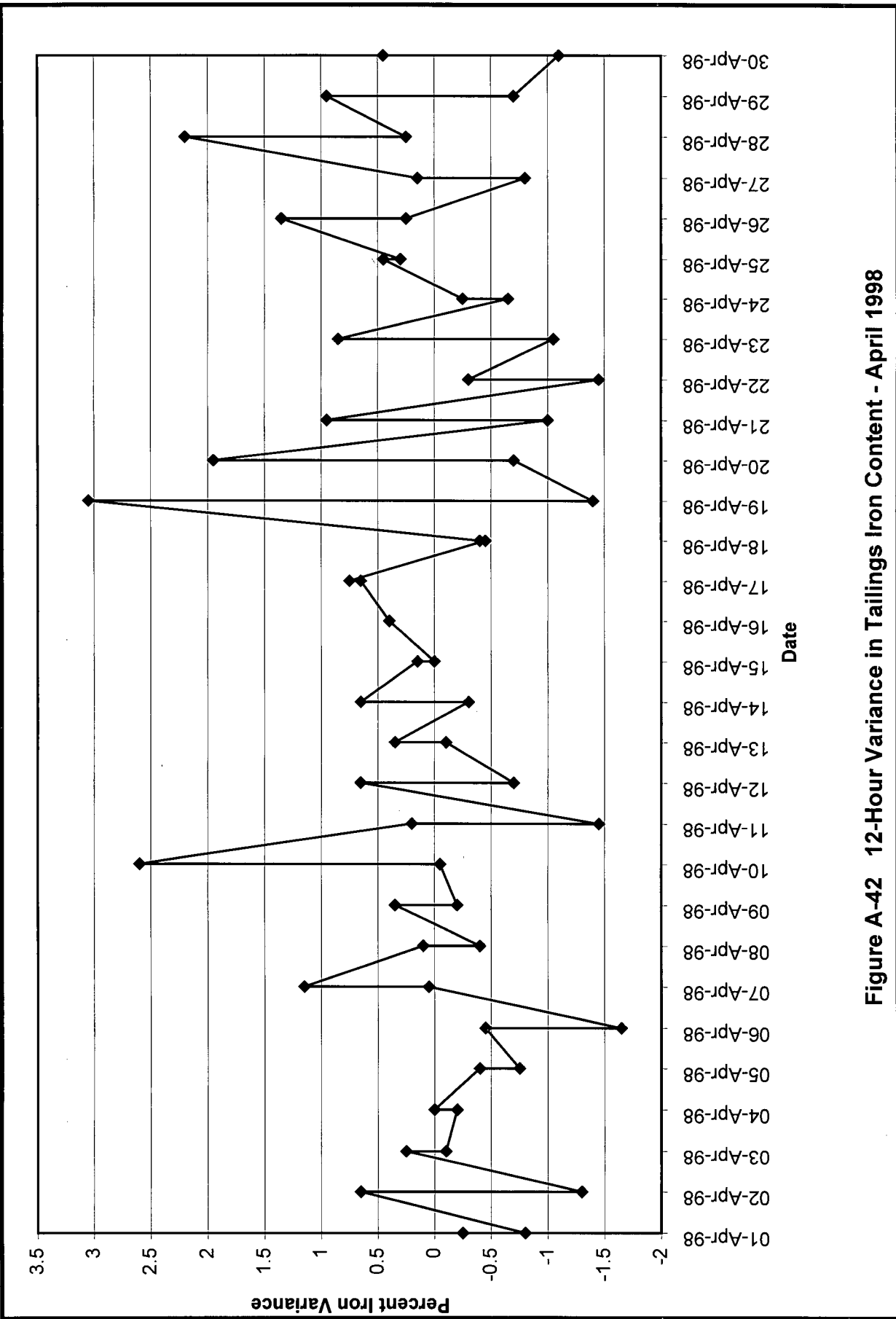


Figure A-42 12-Hour Variance in Tailings Iron Content - April 1998

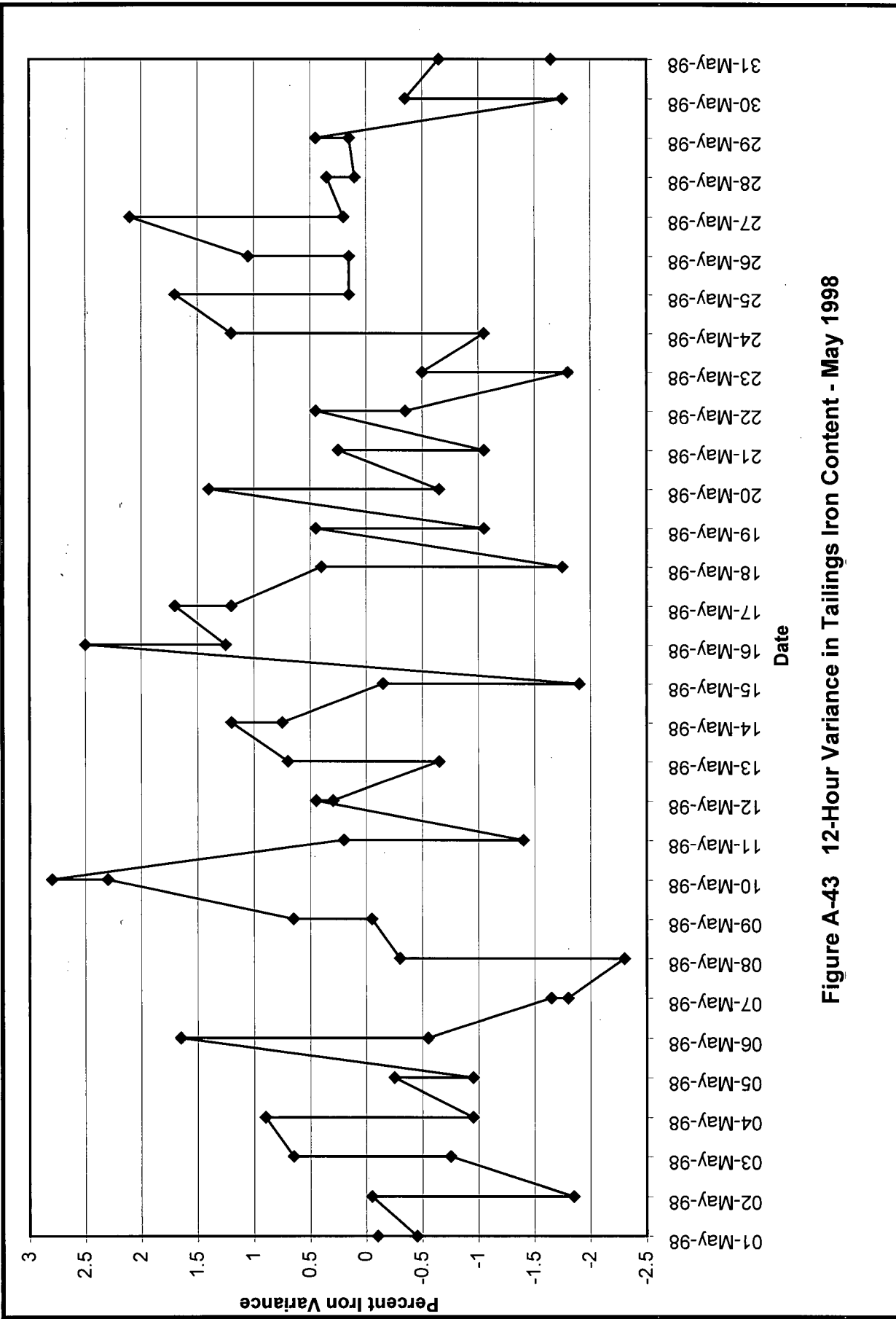


Figure A-43 12-Hour Variance in Tailings Iron Content - May 1998

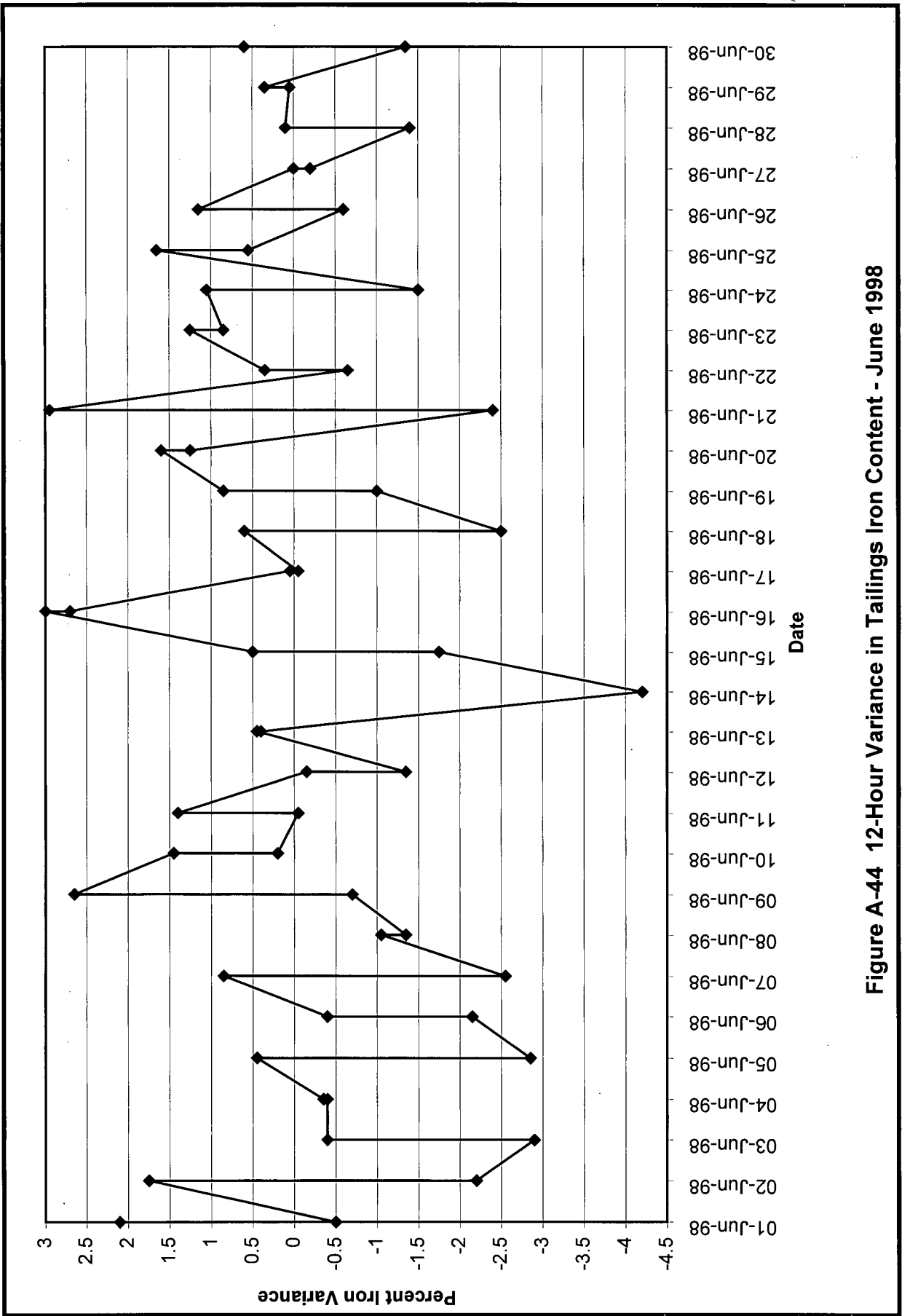
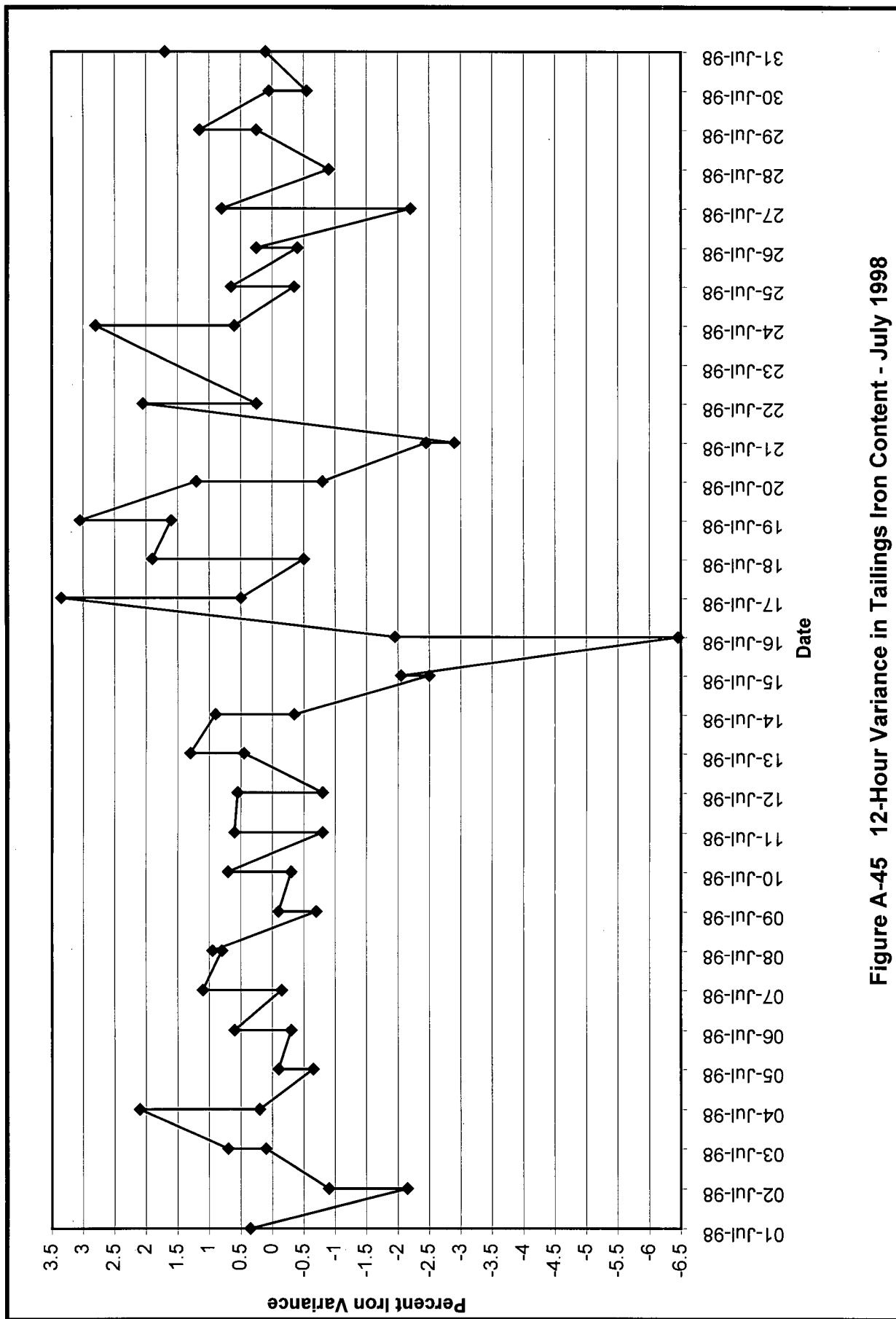


Figure A-44 12-Hour Variance in Tailings Iron Content - June 1998



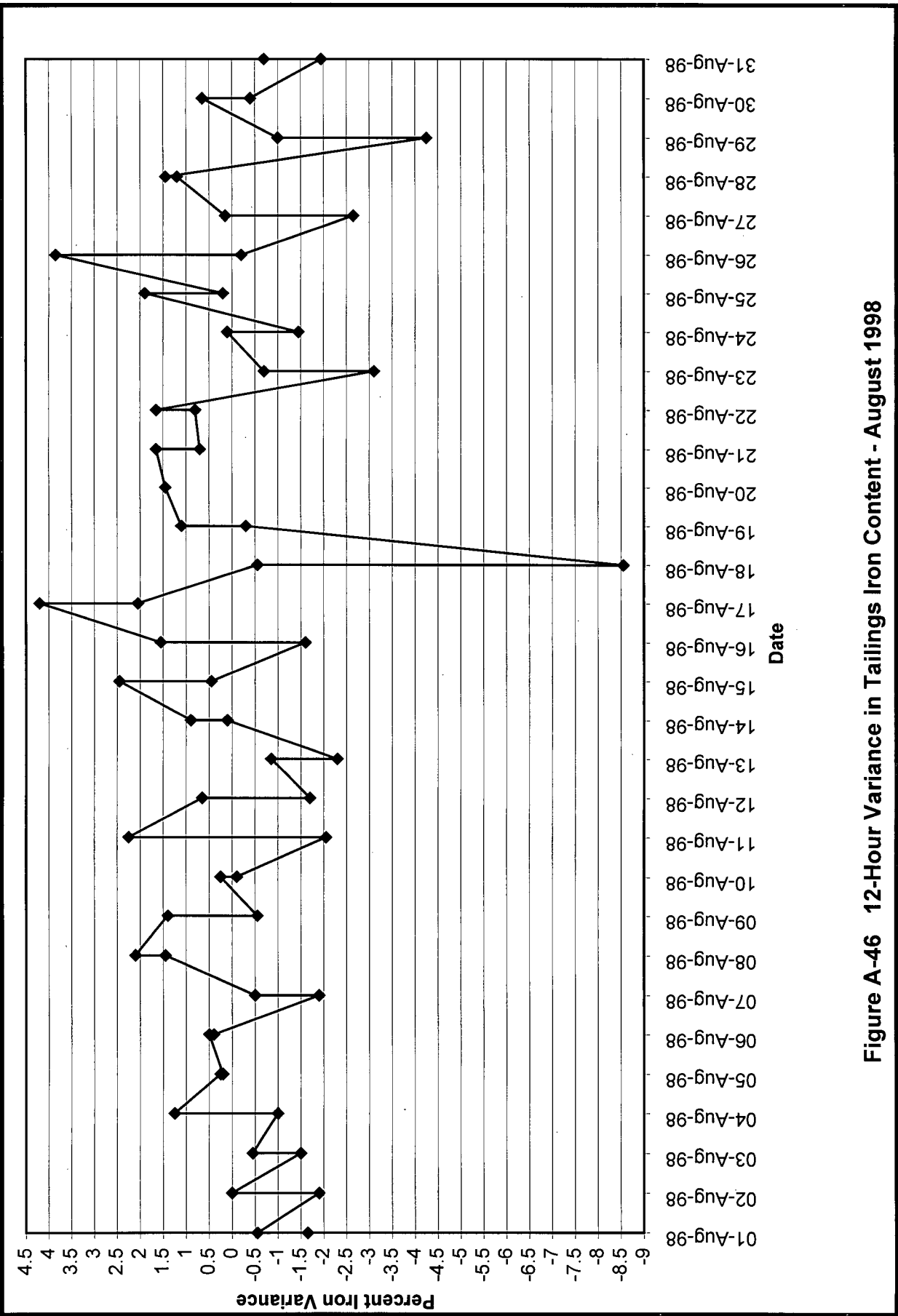


Figure A-46 12-Hour Variance in Tailings Iron Content - August 1998

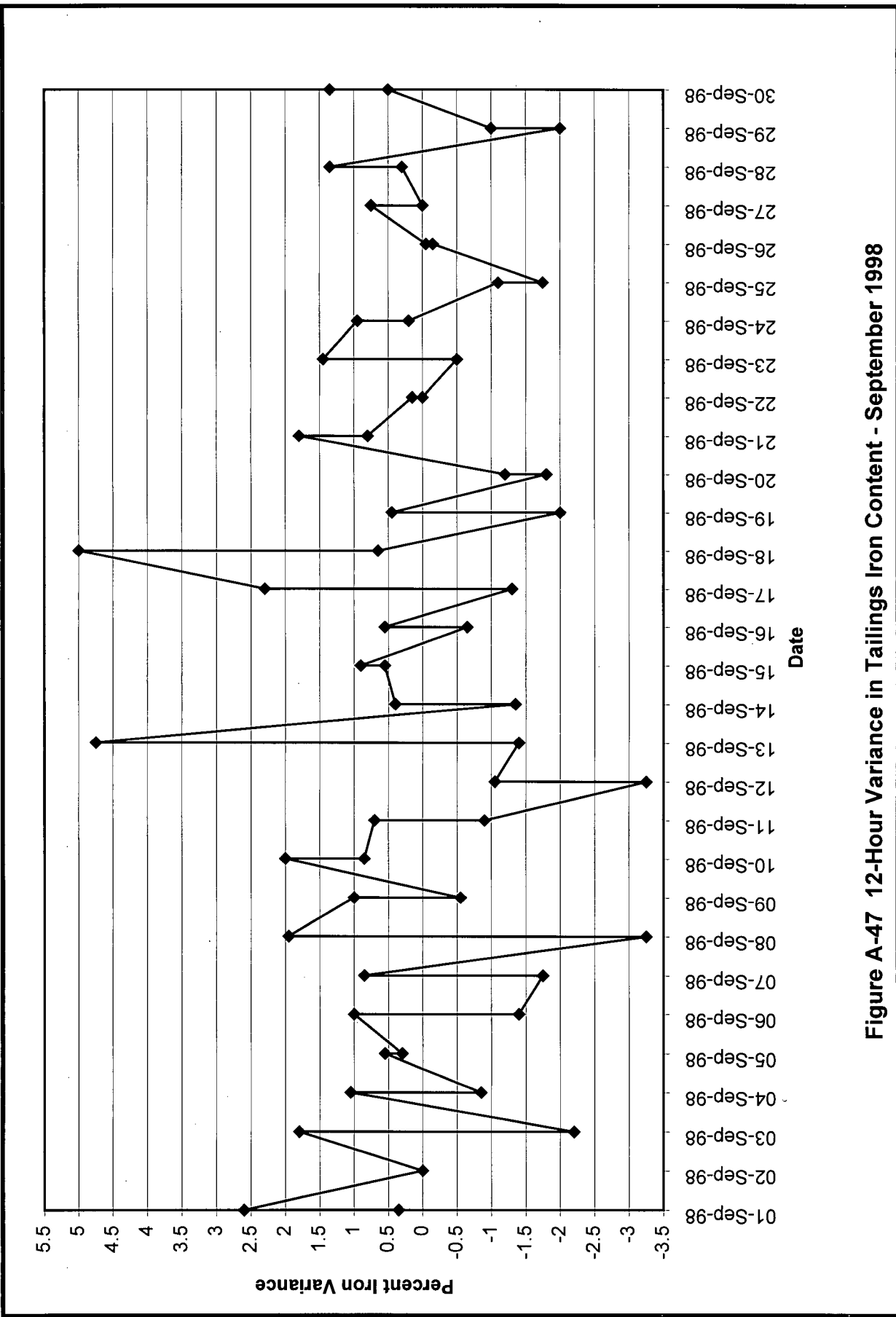


Figure A-47 12-Hour Variance in Tailings Iron Content - September 1998

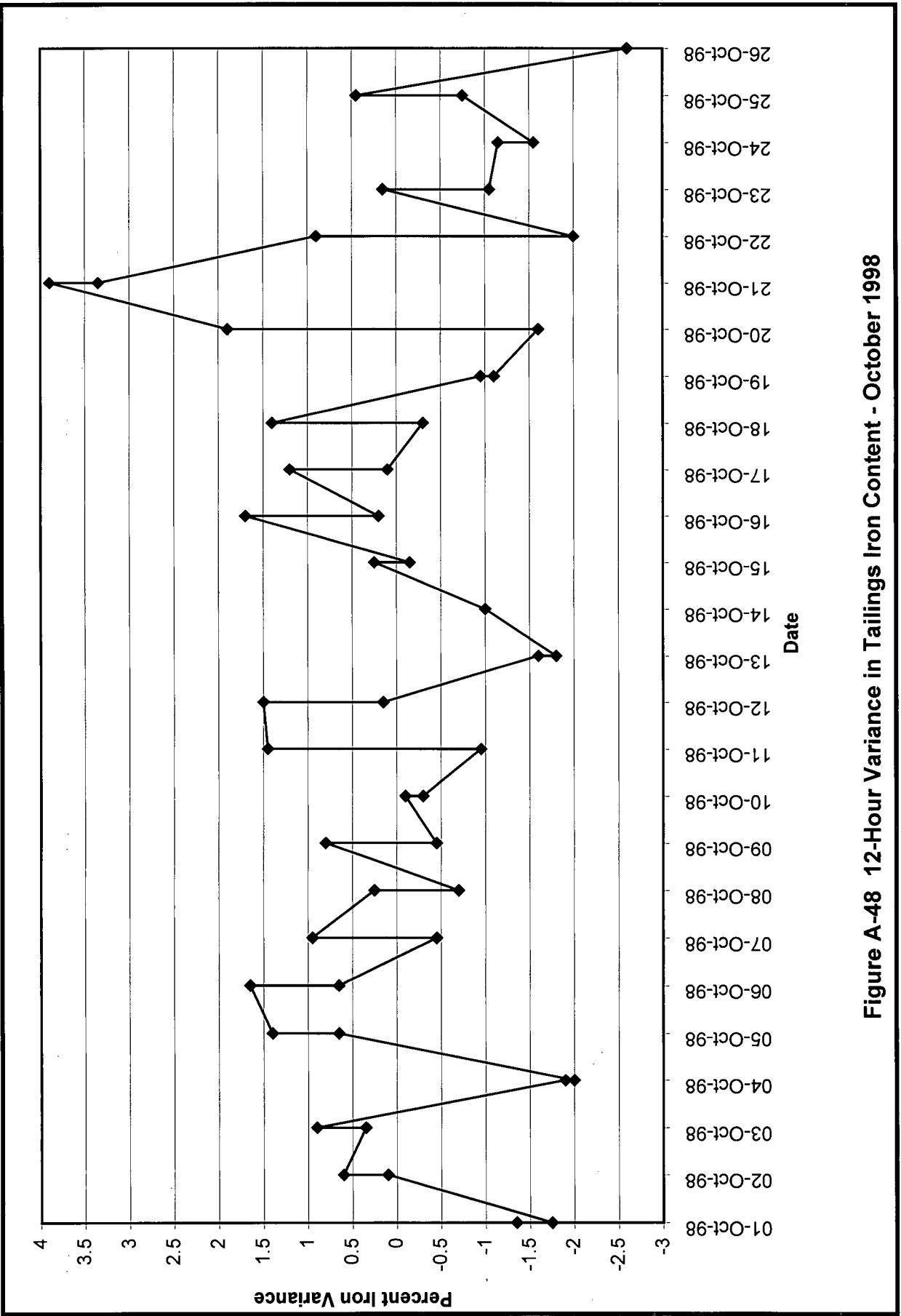


Figure A-48 12-Hour Variance in Tailings Iron Content - October 1998

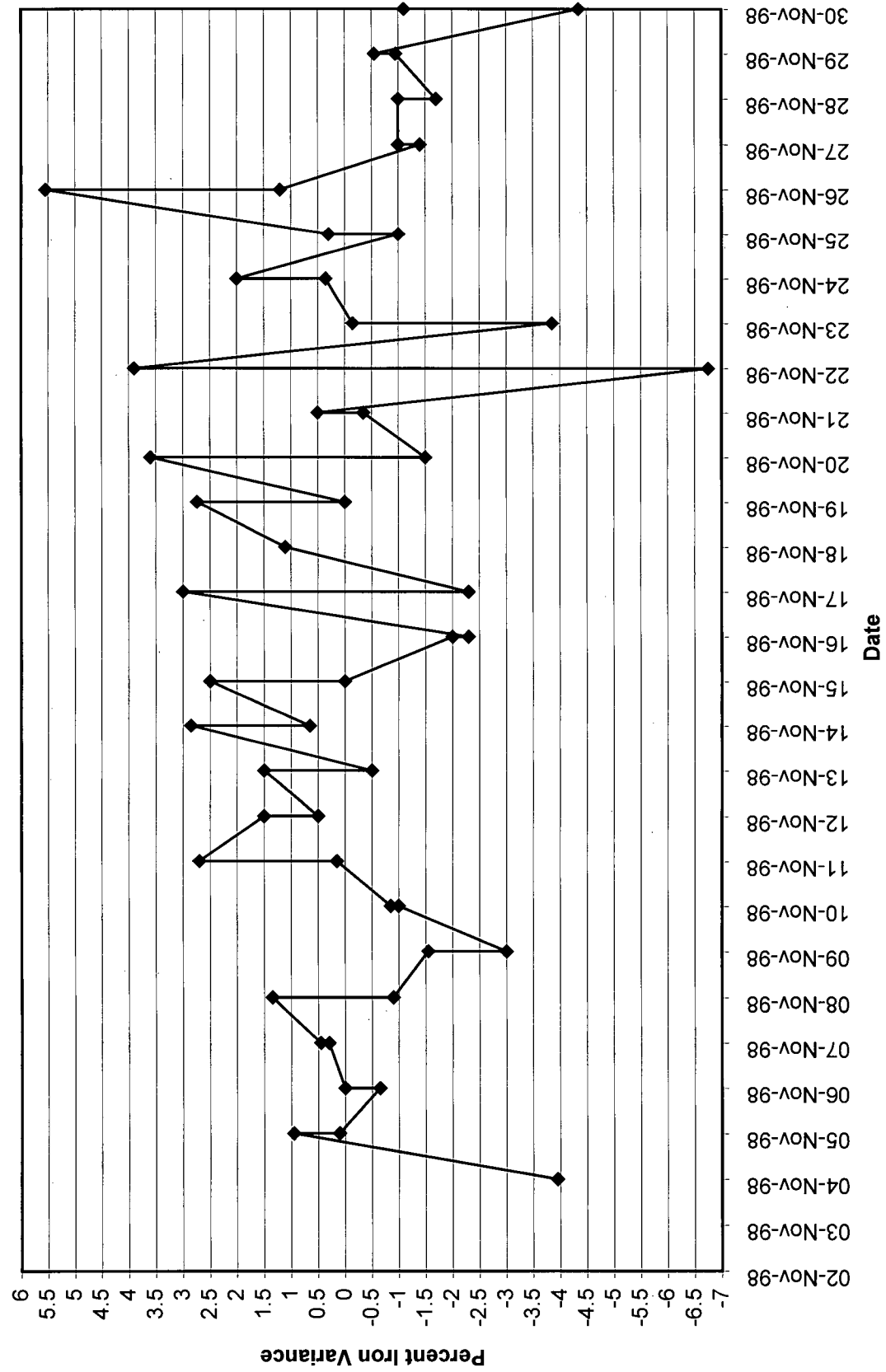


Figure A-49 12-Hour Variance in Tailings Iron Content - November 1998

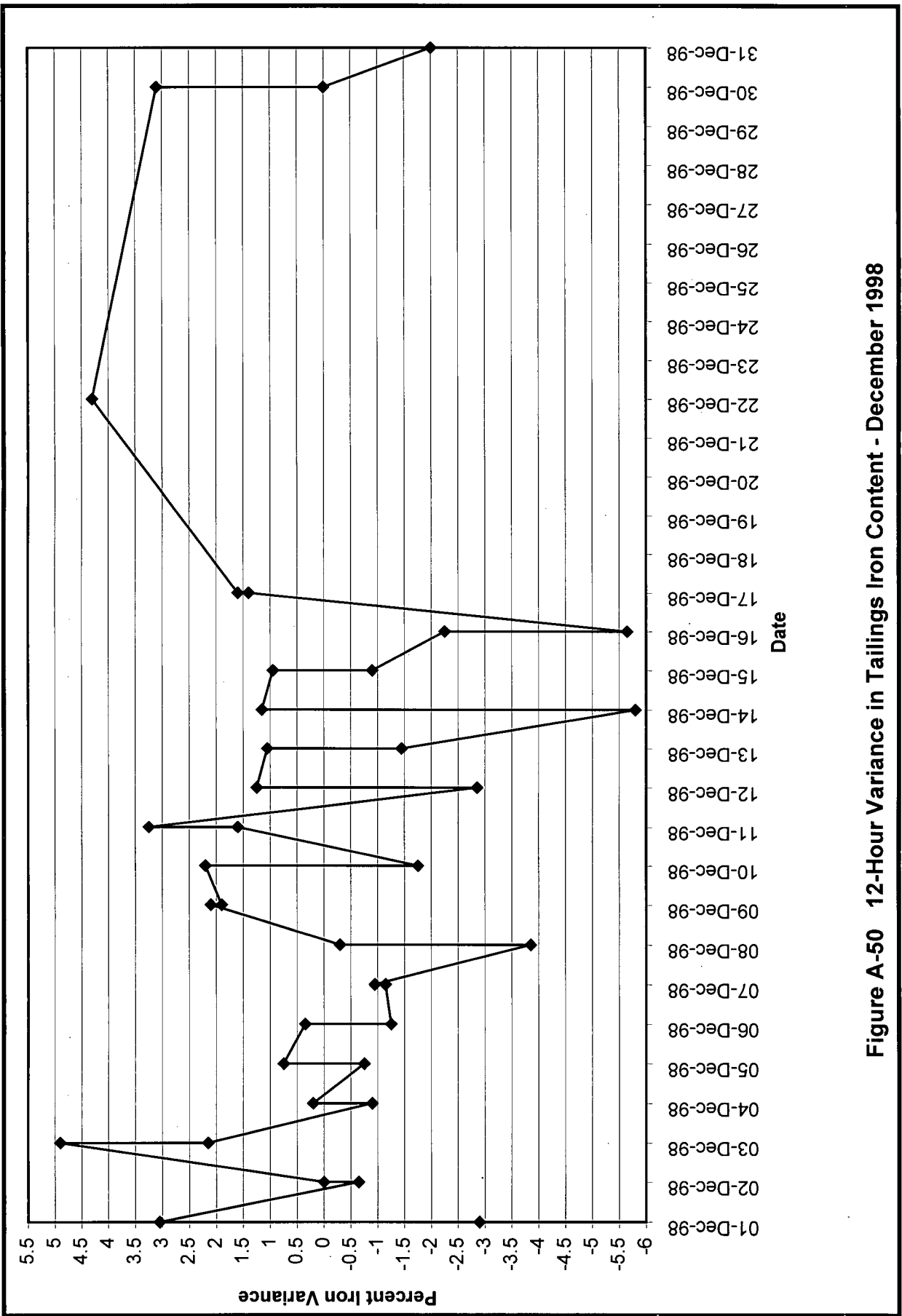


Figure A-50 12-Hour Variance in Tailings Iron Content - December 1998