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GROUND CONTROL IN TRANSVERSE CUT AND FILL MINING OPERATIONS AT INCO

P. H. Oliver[1]

Abstract

Inco has successfully utilized yield and span to control the effects of a high horizontal field stress condition on transverse cut and fill mining for forty years and yield to alleviate rockburst conditons for 35 years. The approaches used were developed empirically.

Ground conditions associated with deeper mining suggested the existence of high horizontal stresses which were subsequently confirmed by measurements. A new theory was required to explain the effectiveness of the procedures. It was found that beam-column theory, as applied to a thin, tabular and near horizontal orebody to control high horizontal stress effects in room backs, was similarily applicable to transverse cut and fill mining. The theory demonstrates the validity of the techniques used by Inco in the Sudbury operations and suggests means to effect further improvements.

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1. INTRODUCTION

Ground control in transverse cut and fill mining at Inco's Sudbury operations, involves yielding pillars and sequenced headings to alleviate rockburst conditions. Yielding pillars in conjunction with stoping spans, control stress conditions in the immediate stope area. The yield effect is influential in governing the mechanism of stress relief in crown pillars. The concepts of sequence and yield were developed over thirty years ago when the principal stress was assumed to be vertical and the mining method was overhand square set with crushed rock fill.

The two concepts have remained virtually unchanged and are as valid today as they were thirty years ago, in spite of the many changes in mining in the interim period and the ever increasing depth of mining operations. Roof-bolts and screen have replaced square sets; hydraulic sand-fill, followed by cemented hydraulic sandfill has replaced crushed rock as the fill medium; scooptrams have replaced the slusher which had replaced the hand shovel.

Over the past decade, it became increasingly apparent that the accepted theories relevant to the success of sequence and yield, could not adequately explain many of the observed conditions. There were strong indications of high horizontal stresses, but no means to explain the observed reduction in stresses in the stope backs associated with rib pillar yield.

The measurements of the stress state at depth by Dr. Herget (1) in 1974 confirmed the existence of high horizontal stresses, and emphasized the need to develop new theories to explain why the yielding pillar mining approach worked.

2. HISTORY

Fill method ground control began with the first layout of transverse stopes and rib pillars at Frood Mine in the thirties. Stopes were 45 ft. wide and the rib pillars 35 ft. wide. The primary mining recovery factor was 64%. Cut and fill mining was used and the fill material was crushed rock.

The sequence consisted of dividing the orebody into blocks, with each block having a strike length of from 400 to 500 ft. Barrier rib pillars, generally 66 ft. in width, separated the blocks. Major floor pillars, 50 ft. thick, were left every second level to control level subsidence (Figure 1). Mining on a level began with the blocks on the extreme ends of the orebody and new blocks were progressively started, working inwards from the end blocks.

It was soon found that artificial ground support was required to control the backs of the stopes and mining was

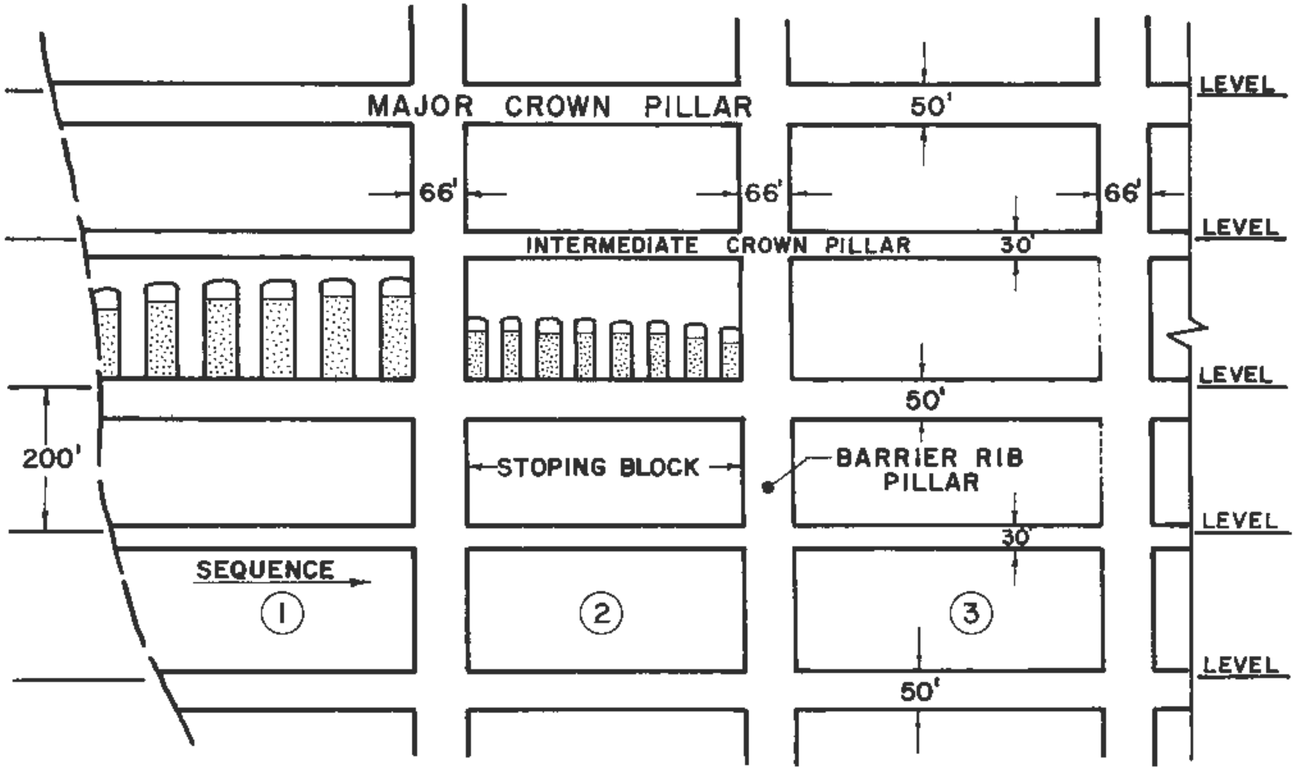

FIGURE I FROOD MINE BASIC LONGITUDINAL LAYOUT converted from cut and fill to square set and fill.

As additional stopes were opened, increasing the length of the stoping blocks, heavy ground developed when mining reached approximately 40 ft. from the level above. It was found necessary to reduce the width of the stopes and pillars to 5 sets and 4 sets respectively, which resulted in stoping widths of 30.5 ft., including overbreak, with the width of the remaining pillar being 19 ft.

The 5 set stoping width, in conjunction with 4 set pillars, provided a primary recovery factor of 62%. The fact that the pillars fractured was observed and it was noted the fracturing of the pillars resulted in improved ground conditions in the stopes. The 5 set stoping width provided optimum ground conditions from the sill to the crown and was an efficient width for square set mining with hand mucking. For the reasons outlined above, a span which suited the mining method and a stope to pillar relationship which optimized ground conditions, the 5 set stope and 4 set pillar became the standard for transverse fill method mining. There was no theory involved in arriving at this stope-pillar geometry, just the tried and true system of observing what works and applying the knowledge gainfully.

The sequence, mining from the limits of the ore in towards the center, plus the major barrier rib and crown pillars, provided a fertile field for rockbursts. The increased incidence of rockbursts coincided with the severe rockburst situation in the Kirkland Lake area and resulted in the setting up of the Rockburst Committee by the Ontario Mining Association to investigate ways and means to predict and control rockbursts and to co-ordinate the investigative studies of the individual mining companies. The key document resulting from the efforts of the committee was the report by Morrison (2), published in 1942, outlining the doming theory and the importance of sequencing towards the coalescing and enlargment of domes. The effectiveness of the approach outlined by Morrison has been proven by the marked reduction in the frequency of rockbursts and rockburst related injuries beginning in 1941.

Inco has freely used Morrison's report as a reference. Stoping begins on a lead level and progresses outward from the lead stope (Figure 2). No barrier pillars are left and small rock inclusions are slotted through to avoid the establishment of rockburst prone remnants.

The standard stope-pillar geometry, applied above the 4000 level horizon, required modification for mining the heavy massive sulphide ores at Creighton Mine below the 4000 ft. horizon. Stope widths were reduced to 3 sets for a mined width of 19.5 ft., including overbreak, and pillar widths were reduced to 3 sets for an effective 13.5 ft. width.

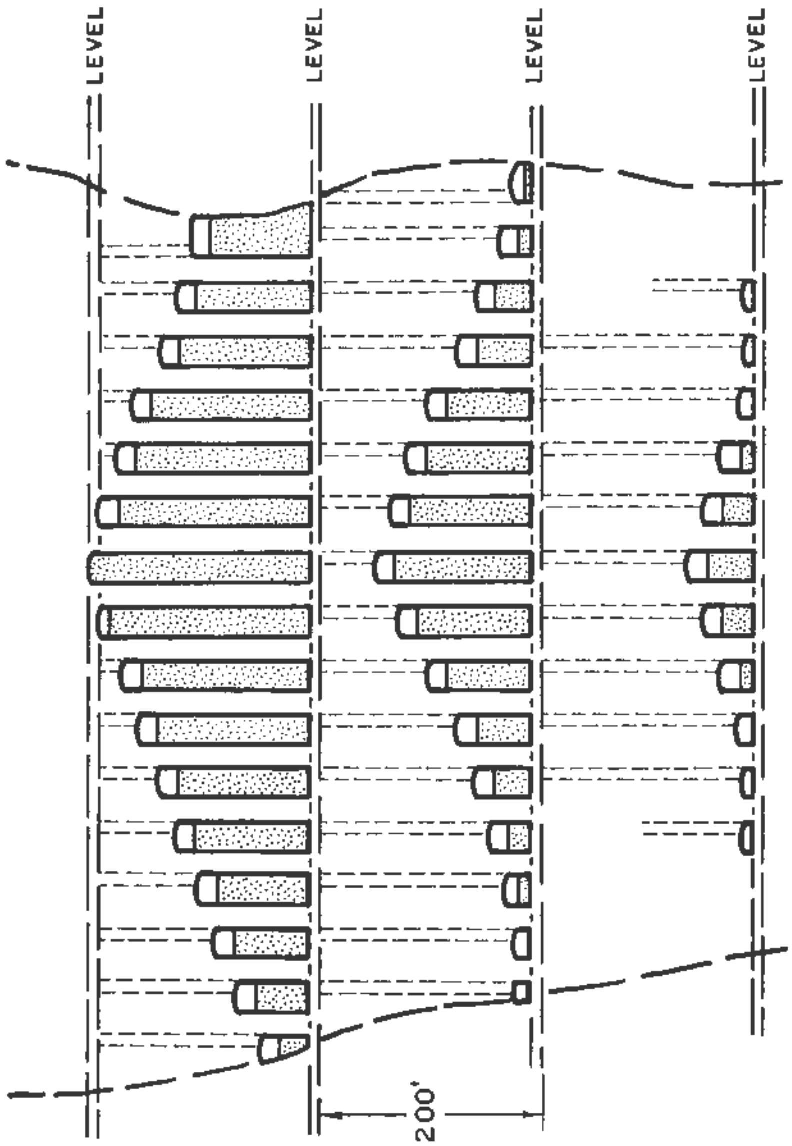

FIGURE 2 CONVENTIONAL TRANSVERSE FILL METHOD MINING SEQUENCE

The primary extraction ratio was reduced from 62% for the 5 set stope, 4 set pillar geometry to 59% for the 3 and 3 geometry.

In the early fifties, hydraulically placed classified tailings and alluvial sand replaced crushed rock as the fill medium. Towards the end of the fifties, the increased application of roofbolts began to spell the end of square set stoping.

In the early sixties, the use of portland cement to consolidate hydraulic fill came into universal application to improve ground conditions during pillar recovery operations by curtailing fill losses from mined areas and by reducing fill recompaction. The improvement of the fill characteristics was achieved without increasing primary mining method costs in that expenditures for cement were offset by the savings in labour and materials required to construct plank slushing floors and gob fences.

In the mid sixties, mechanization of cut and fill mining was started by the scooptram. There were no changes made to the stope-pillar geometries. There was a significant change in sequencing in that it became necessary to change from a sloping sequence line to a series of steps, with each step consisting of the 4 or more stopes making up a mechanized unit. The lead unit approaches the level above in an essentially flat multiple stope front (Figure 3).

In the seventies, in several areas, the primary mining recovery factor was increased by increasing the stoping span or reducing the pillars or a combination of the two approaches.

3. OBSERVATIONS

The following observations of ground behavior associated with transverse and longitudinal cut and fill mining were made by the author from the year 1960 to the present time. The observations pertain to a number of Inco's Sudbury area mines at depths ranging from 1500 to 6600 ft. below surface. The term, yield, refers to any case where pillars undergo non-violent stress relief.

1. There is some critical height at which a narrow rib pillar will fail by yielding.
2. Pillars burst if failure occurs before the critical height is reached.
3. Back conditions in the stope generally improve when the adjacent rib pillars experience initial failure.
4. The lead stope in a normal sequence experiences the highest concentration of stress in the crown area and at a lower elevation than is the case with adjacent stopes (Figure 4).

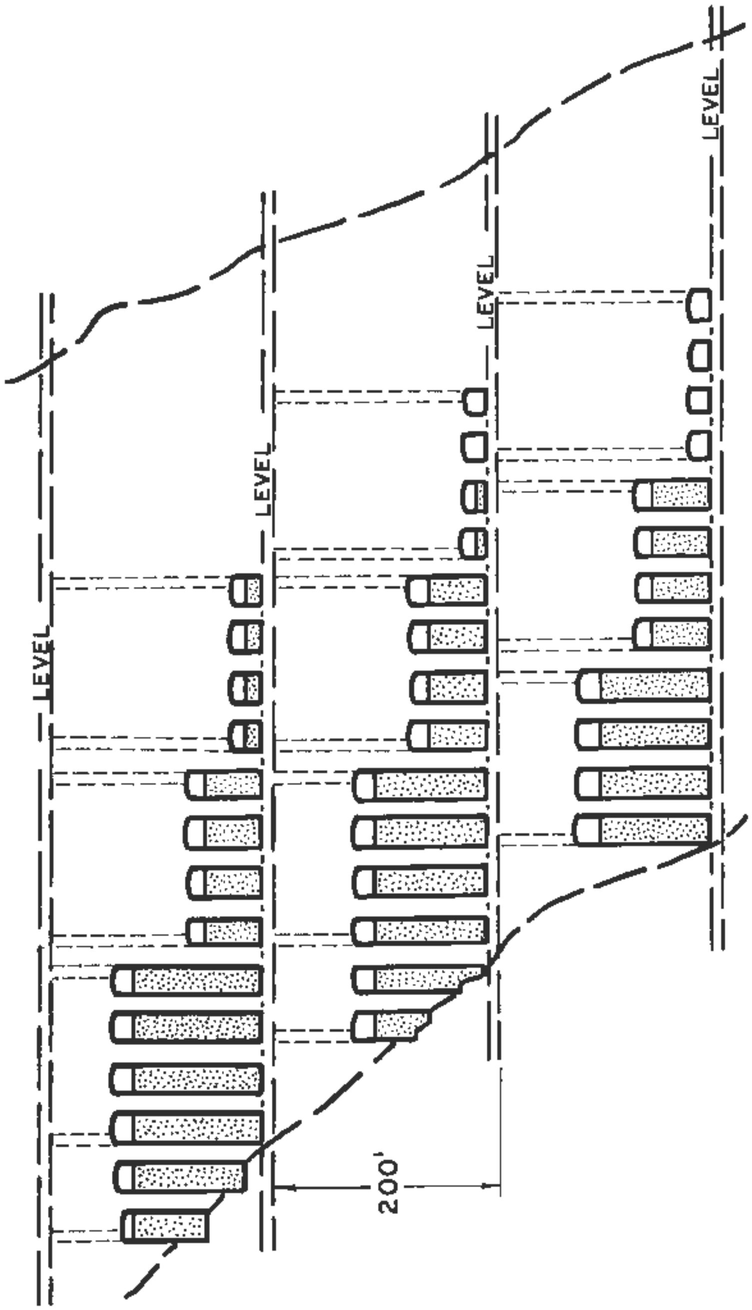

FIGURE 3 A MECHANIZED TRANSVERSE FILL METHOD MINING SEQUENCE

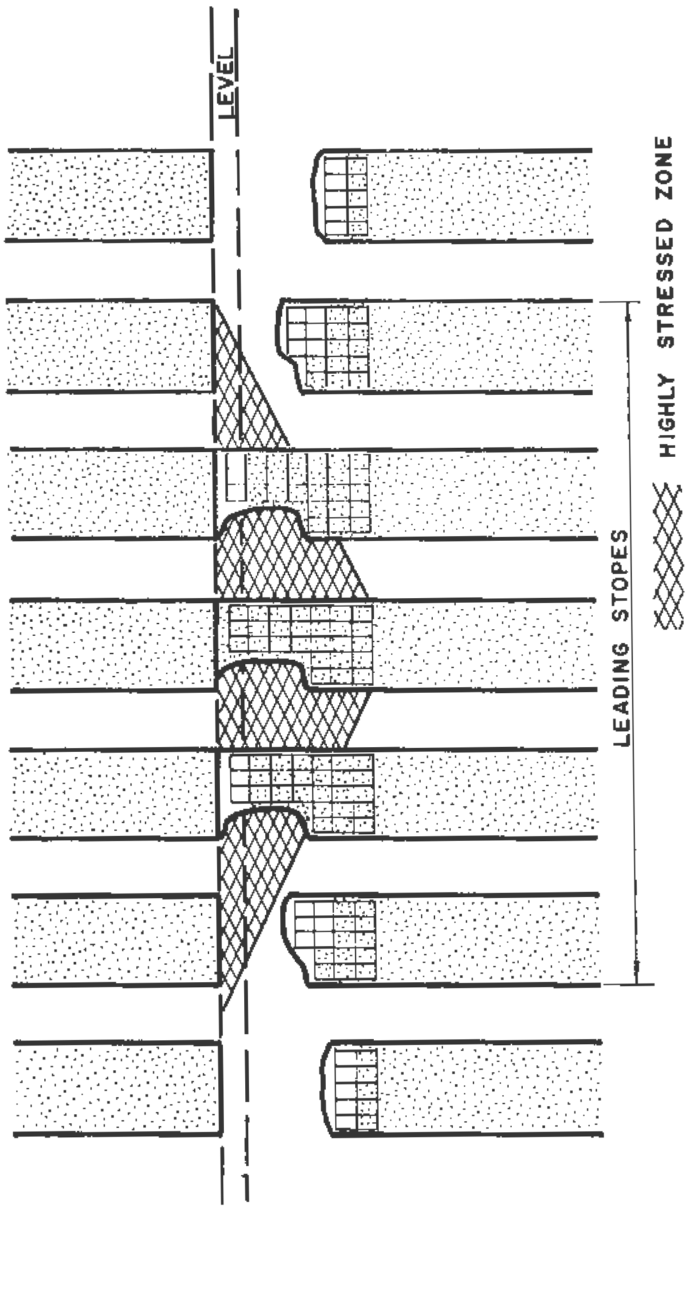

FIGURE 4 HIGHLY STRESSED ZONE ENCOUNTERED BY LEADING STOPES

5. After several stopes have mined up through the crown, the effects of high horizontal stresses disappear provided the footwall to hangingwall dimension exceeds some critical value.
6. Where the footwall to hangingwall span is less than some critical dimension, repetitive crown bursting occurs as each stope approaches the level.
7. The addition of cement to the hydraulic fill stiffened the rib pillars and significantly limited the lateral expansion of the rib pillars.
8. Increased compressive shear arching took place at one mine following the introduction of cemented sandfill.
9. Stoping span reductions are necessary for mining through the critical crown region.
10. Longitudinal cut and fill stopes could be mined with low arch where mining widths ranged from 30 to 40 ft. Beyond 40 ft., increased mining widths meant an increased frequency of joint oriented unpredictable ground falls.
11. In areas where multiple stopes are silled simultaneously for mechanized mining, all of the pillars fail at a common height. Failure is by bursting if it occurs during the mining of the first and second cuts. Non-violent failure occurs if the onset of pillar yield does not begin until the third cut is mined.

12. The burst type of rib pillar failure in mechanized mining occurs at depth. At shallower horizons, failure is by non-violent yield.
13. The nature of the violent failure of rib pillars in deep mining can be changed by destressing the ribs as silling progresses, but the degree of damage to the rib pillars at the time of failure is almost identical to the case where the pillars have not been destressed.
14. Where rib pillar failure occurs during the silling phase of a mechanized unit at depth, failure is not progressive with silling. Some critical minimum area must be silled before failure occurs. The area of failure then takes in most of the silled unit, with the level of damage highest in the central area and with little to no damage close to the abutments. Damage is confined to the rib pillar walls and shoulders. There is no back damage.
15. An increase in the stoping extraction ratio, within limits, leads to a reduction in the height of induced compressive shear arching of the stope backs.
16. Where the primary stoping extraction ratio is close to the optimum, localized pillar weaknesses can lead to the development of horizontal fractures above the back of the stope and above the pillar. Blocky falls of ground are liable to occur in the vicinity of

weakened pillars.

17. The lead unit of a mechanized cut and fill sequence can, under the right circumstances, mine up into the critical crown failure region in a flat front and cause the crown to yield rather than burst.
18. The stress level at the center of one pillar was measured just prior to yield failure and was found to be in the order of 4000 psi., as opposed to the 20,000 psi. uniaxial compressive strength of the pillar material.

4. HISTORICAL CONCEPTS OF GROUND CONTROL AS APPLIED TO TRANSVERSE CUT AND FILL MINING

4.1 Field Stresses

It was assumed the principal stress was vertical and related to overburden weight. Some consideration was given towards the possibility of hydrostatic loading to explain the stresses in the crown pillars.

4.2 Yielding Pillars

Ground control practices revolved around the concept of using yielding pillars and sequenced mining. Yielding pillars were weak pillars and incapable of supporting the superincumbent load; therefore the major part of the superincumbent load was transferred to the perimeter of the mined area. The yielding rib pillars supported only the immediate ground above the stope back.

A yielding pillar was considered to be a narrow pillar, generally less than 30 ft. wide. Any pillar wider than 30 ft. was considered to be strong and liable to burst.

4.3 Sequence

Sequencing controlled the gradual enlargement and subsequent merging of the individual level stress domes. The sloping sequence line limited the volume of ground contained in the critical crown failure region by creating a wedge shaped crown with just the tip of the wedge in the critical crown bursting zone. The tip could either yield or fail in a series of small rockbursts. Small rock remnants were slotted through to eliminate a source of major rockbursts.

5. THE NEW CONCEPT OF GROUND CONTROL

The historical concepts of ground control offered satisfactory explanations for parts of the observed phenomena. However, with ever deeper mining, the mechanization of cut and fill operations which altered sequencing, and experimentation with varying stoping widths and primary extraction ratios, it became increasingly apparent the concepts were incomplete because of the faulty assumption of superincumbent loading.

5.1 Field Stresses

Deeper mining provided a clue as to the nature of the field stresses through ground behavior in development openings. High horizontal stresses were inferred and subsequently verified by stress measurements. The principal stress is near horizontal and in the areas pertinent to the listed observations strikes roughly parallel to the erruptive contact or normal to the direction of transverse fill method mining. The least stress is near vertical and in line with superincumbent loading. Details of the study are covered in the paper by Herget et al (1) which was prepared for presentation at this Symposium.

5.2 Controlled Sag Alleviates Horizontal Stresses

The established fact of a high horizontal field stress state explained the high stresses encountered in crown pillars and the behavior of the crowns at various stages of mining. High horizontal stresses confused the picture with respect to the improved stoping conditions associated with pillar yield until it was realized the concepts of back deflection (sag) related to stope span and pillar yield, as outlined by J. Parker (3), could be applied to transverse cut and fill mining. While Parker's (3) concepts were developed from observations of the behavior of openings in a room and pillar mining operation of a near flat, tabular deposit, they were based on guidelines outlined by C. L. Emery with respect to the behavior of beam-columns.

The deflection of the back of an opening, if downward, sets up a horizontal tensile stress component which will, depending on the degree of deflection, offset high horizontal compressive stresses in the back of the opening. If there are a series of openings separated by yielding pillars, there will be an overall abutment to abutment roof sag which will effectively reduce horizontal stresses in the backs of the openings even though the spans of the individual openings may be too narrow to provide stress relief by themselves. The height of the openings is insignificant towards stress relief outside of the relationship between pillar height and yield.

Horizontal stress relief through back sag explained phenomena related to span and yield. It explained why stoping ground conditions improved when the pillars yielded; why severe induced arching occurred in narrow openings while little to no arching was evident in adjacent, moderately wide openings; why tensile conditions were common to very wide openings. It explained why adjusting the primary stoping recovery factor by slotting out parts of the ribs could lead to improved back conditions and if carried too far to tensile back conditions. It explained the relationship between local pillar weaknesses and localized tensile back conditions.

The relationship between sag and the stress state of the stope back points out the importance of distinguishing the nature of the ground problems in a working place. Induced compressive shear arching can be alleviated by increasing the stoping span, by increasing pillar yield through reducing pillar width or by a combination of the two approaches. Tensile back conditions, characterized by unpredictable, joint oriented falls of ground, are indicative of excessive span, excessive pillar yield, a combination of the two, or the lack of elevated horizontal field stresses. Where the condition can be traced to excessive span or yield, conditions can he improved by reducing span, by increasing the pillar width to reduce yield, by decreasing pillar yield through improved lateral support, or any combination of the above approaches.

A wrong assessment of the nature of the ground problem can lead to a worsening of the condition. The miner's rule of thumb approach of reducing span when difficult back conditions are encountered is wrong when the condition is characterized by induced compressive shear arching.

6. YIELD

6.1 A Definition of Yield

Yield is considered to be any case where moderately to highly stressed ground undergoes stress relief in a non bursting manner.

6.2 Yield As A Rockburst Control Mechanism

Inco has been using sequencing to alleviate rockburst conditions for 34 years. Observations noted earlier in this paper and which pertain to crown pillars indicate that crown pillars can be made to yield under the correct circumstances in spite of their being the reverse case to rib pillars. Rib pillars have constant width and increasing height, while crown pillars have constant height and decreasing width (Figure 5). Under favourable circumstances, both rib and crown pillars can be made to yield and under less favourable circumstances, both pillar types will burst.

It has been concluded that yield is the prime mechanism of rockburst control and that sequence is a means to establish conditions favourable towards yield. The successful application of sequencing at Inco's Sudbury operations is felt to be due in a large part to the widths of the orebodies which have contributed a key element to the yield phenomenon.

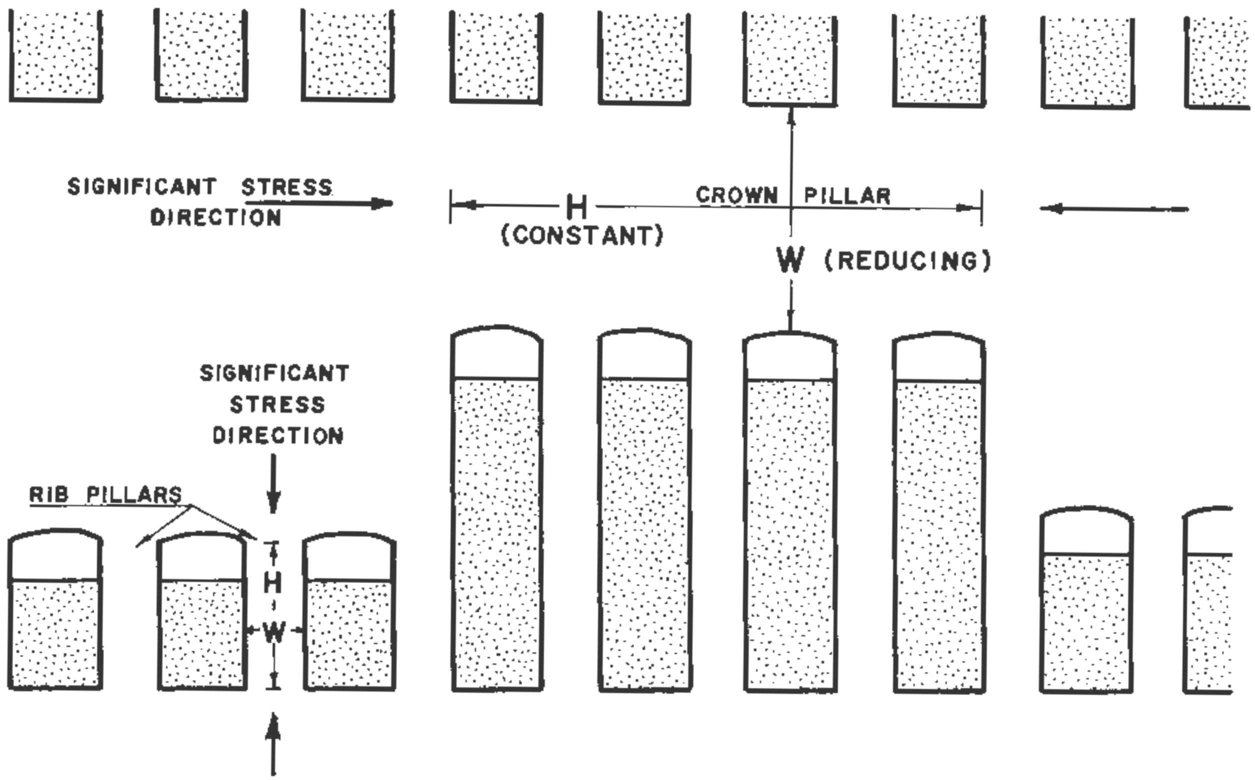

FIGURE 5 HEIGHT & WIDTH FOR RIB & CROWN PILLARS

6.3 The Unknowns of Yield

Current literature does not offer a suitable explanation for the yield phenomenon. It provides neither a means of determining the parameters governing yield, including the influence of lateral restraint, nor does it provide a means of determining post-yield properties. The lack of understanding of yield phenomena has deferred mining improvements as a result of the necessity to use trial and error in developing methods for utilizing yield. The problem with trial and error, in the absence of theory, is that once something works, experimentation tends to stop. The factors which make the system work are not understood and the effects of subsequently changing one or more of the factors may not be recognized.

Inco continued with the 5 set stope, 4 set pillar geometry, above the 4000 ft. horizon, through to the end of the sixties, although the support nature of the fill changed twice, to hydraulic fill in the early fifties and to cemented hydraulic fill in the early sixties. Experimentation with increased stoping spans and primary mining recovery factors did not begin until the seventies. That the results were favourable was credited to differences in conditions between the areas of mining with increased primary recovery factors and past areas mined with conventional geometries. The one identifiable unique change in conditions is that of increased pillar rigidity associated with the use of cemented hydraulic fill.

6.4 Suggested Parameters Governing the Behavior of Yielding Pillars

The Inco experience with yield has demonstrated the importance of yield as a mining tool. It has not, however, been possible to develop hard and fast rules for the design of yielding rib and crown pillars. Observations of yield behavior indicate the accepted means of calculating critical pillar stress do not apply in the case of yielding pillars.

It does appear that a yielding pillar might be defined as one having a width to height ratio of 0.5 or less. The width to height ratio, may in some cases, be greater than 0.5, but it will never be greater than unity.

Pillars appear to behave in a like manner to columns, except that in the case of rock pillars in comparison to steel columns, the slenderness ratio effect on instability is greatly increased; possibly because of the lack of in situ tensile strength.

It is suggested the curves developed from the Generalized Euler or Tangent Modulus Formula for columns, plotting critical stress against the slenderness ratio, may parallel measured values for rock pillars except that the slenderness ratio at the proportional limit will correspond to a W/H ratio of between 1.0 and 0.5 (Figure 6). It is significant that should this hypothesis prove valid, a yielding pillar will be similar to a long column up to the point of critical stress. The critical stress level will be a function of the elastic modulus and the slenderness ratio and not a function of the uniaxial compressive strength as is the case with current pillar strength theories.

It is hoped the recognition of the importance of yield with respect to safety and economics will stimulate research into yield phenomena now that such research has been made possible through stiff testing machines. The behavior of pillars should be studied to determine the relationship between critical strength and the W/H ratio; post yield physical properties; the effect of lateral restraint on the critical strength and post yield properties with particular emphasis placed on the role of fill with respect to rib pillars and yielding rib pillars with respect to crown pillars.

7. FILL

Fill is considered by the majority of mine operators in Canada to be primarily a working platform and secondarily a medium which limits the volume of open ground in a mine. Inco experiences with three types of fill in transverse fill method mining, backed up by the new understanding of the

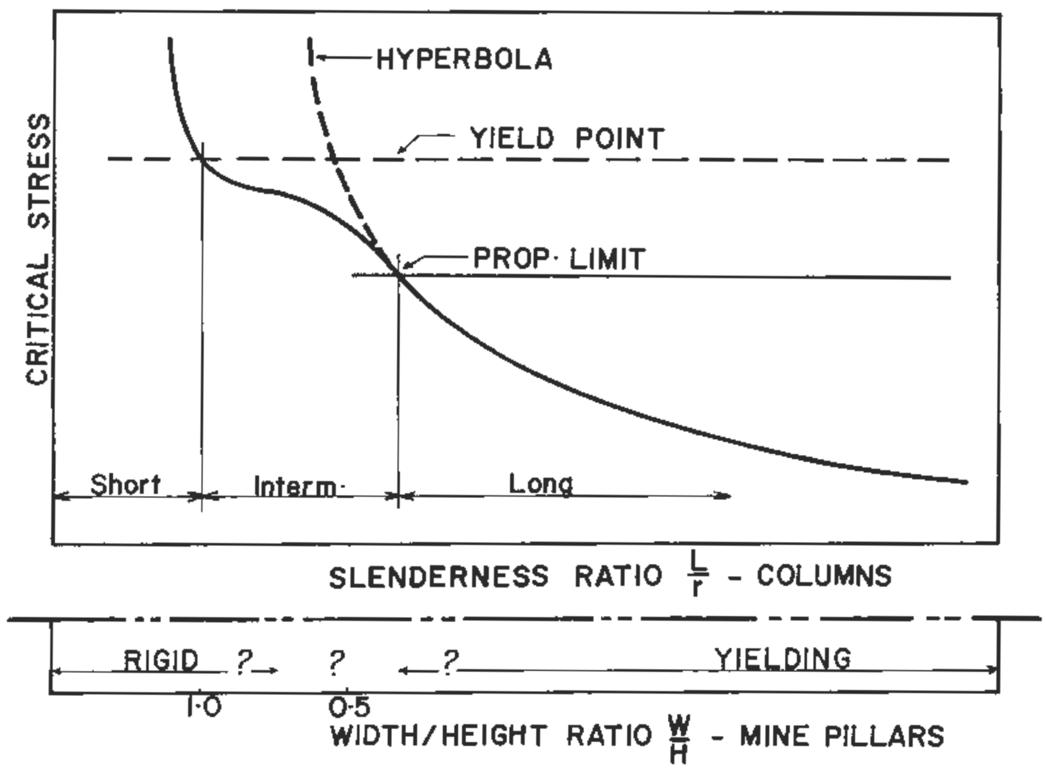

FIGURE 6· SUGGESTED RELATIONSHIP BETWEEN COLUMNS & MINE PILLARS role yielding pillars play towards controlling stoping ground conditions suggests the need to review the role played by fill. The parallelism, although somewhat delayed between improvements in the fill characteristics with respect to rib pillar support and improvements in fill method mining economics related to reduced back support requirements, increased stoping spans and increased primary recovery factors, is too distinct to be ignored. There is a possibility the point of diminishing returns has not yet been reached and further economic benefits may still be possible through the use of stronger fills.

8. CURRENT AND FUTURE PLANS

8.1 Current Transverse Cut and Fill Layouts

The understanding of the mechanism of roof sag towards controlling horizontal stoping stresses has been too recent to have influenced current mining layouts. The present variations in stope-pillar geometries, ranging from stopes 22 feet wide with 14-foot rib pillars to 42-foot stopes and 18-foot pillars (the dimensions include overbreak), are being assessed. A flexible approach is in order until such time as more specific design parameters become available.

APPLICATION OF HIGH-SPEED PHOTOGRAPHY TO ROCK BLASTING AT CANADIAN INDUSTRIES LIMITED A REVIEW

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Abstract

The role of high-speed photographic techniques in studying and optimizing blasts in open pit mines is reviewed. Among the qualitative data obtained through these studies are firing sequence, occurrence of miss holes, the degree of confinement of stemming and nature of muck pile formation. The quantitative data includes velocity of fly rock, spalling stress and heave energy required for muck pile formation. A typical camera and playback system and field procedures for photographing blasts are described. It is found that the uplifting velocity 'V' on the bench during the gas expansion stage of a blast obeys a simple power law for cylindrical charges,

$$V = K\left(\frac{R}{W^{1/2}}\right)^{-n}, \text{ R being the distance and}$$

'W' the charge weight. For blasts in iron ore formation 'n' is found to be nearly equal to unity. For large open pit blasts the velocity of fly rocks is about 15 m/s whereas excessively loaded boreholes may result in fly rock velocity of 50 m/s or higher.

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1. INTRODUCTION

The use of high speed photography in studying short-lived phenomena is not new and its principles are well established. The range of high speed photography may vary from a few dozen frames per second with time resolution of the order of several milliseconds to over a million frames per second with time resolution in the sub-microsecond range. This paper is concerned with the applications in the medium-high speed range i.e. in the 100∿2000 frames per second, in the field of rock blasting.

The first comprehensive studies on the use of a high-speed camera in blasting studies were carried out at the U.S. Bureau of Mines (Blair, 1960). Various photographic techniques have been in use for over a decade at Canadian Industries Limited in assessing blast results, but since the early seventies high-speed photographic studies have formed an integral part of rock blasting research at the CIL Explosives Research Laboratory. The results from these studies are routinely used to confirm blasting predictions and provide added insight on the mechanism and geometry of blasting (Lang and Favreau, 1972; van Zeggeren and Chung, 1973). This paper reviews the principles, field procedures and general applications of high-speed photographic studies at CIL. This is followed by a picture show of typical blasts as recorded by the high-speed camera.

2. DESCRIPTION OF EQUIPMENT

As shown in Figure 1, the filming system consists of a LOCAM Camera equipped with an automatic exposure control unit, a junction box and a portable electric power generator. The camera operates on 115 VAC 60 Hz. The frame rate is adjustable for any rate between 16 and 500 frames/second with an accuracy of ±1% or ±1 frame; the film used for blasting studies is Kodak high speed (160 ASA) 16 mm EKTACHROME Type 7241 EF 449. Accessory lenses ranging from 16 mm to 150 mm are available to cover a wide range of field of view.

Also installed inside the camera compartment is a heating element for use in cold weather. Since this camera consumes not more than 300 watts, a lightweight Honda portable generator Model 300E can supply enough power to run the system.

The function of the junction box is to control the camera operation; to start the camera by breaking the trigger wire and to switch off by a built-in timer.

At maximum frame rate (500 fps), the camera will not consume more than 10 feet (or 400 frames) of film to accelerate; therefore at a frame rate of 300 fps it is sufficient to give 4 seconds for the run up, 6 seconds

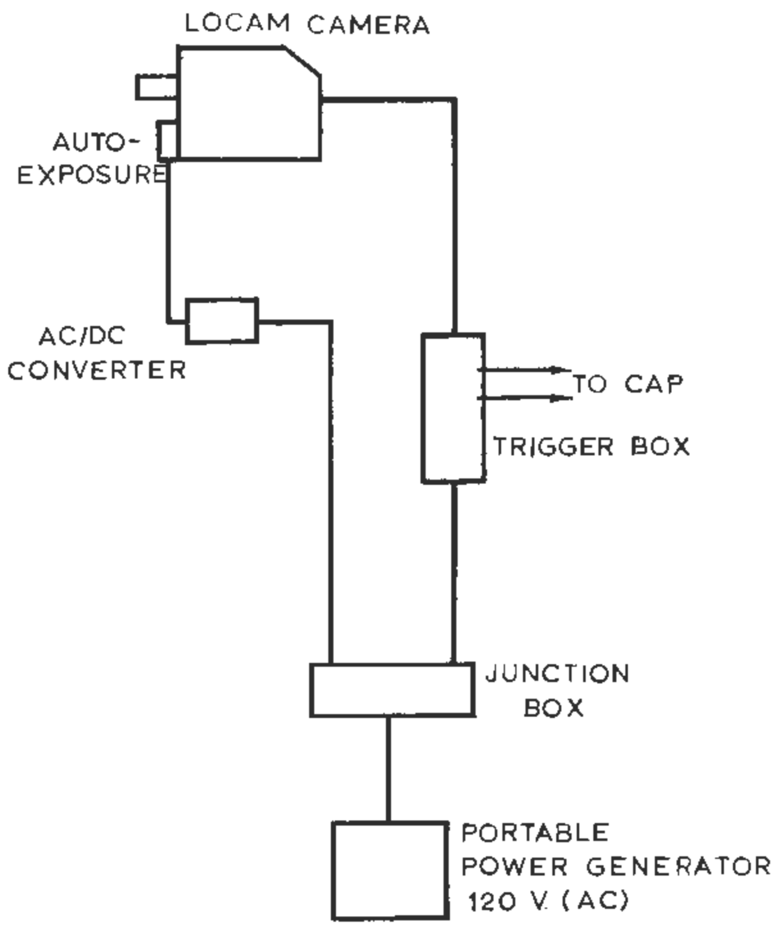

FIG.1 PHOTOGRAPHIC SYSTEM

for recording the blast action and about 4 seconds for taking the panoramic view after blast.

3. FIELD PROCEDURE

To film a blast, the camera must be properly located and protected by a shelter. For example, in the case of filming the face movement a suitable location would be on one level above, 600 to 800 feet away and looking at an angle of about 30° to the face (Figure 2). At least four visible targets (.5 x .5 m^2) arranged in three groups are hung over the crest and rested on the bench face; in the case of filming the top of the bench, a group of at least nine targets are placed behind the last row or on the side adjacent to the blast. All these markers, as well as the camera location, are painted and surveyed into a blast plan for use as dimensional controls in the film analysis.

The trigger wire of the junction box is connected to the blast initiation with no delay while a long delay (about 4 seconds) is placed between the trigger and the blast.

4. APPLICATIONS

Both qualitative and quantitative information can be obtained from high speed photographic studies. The qualitative data which may include information on the

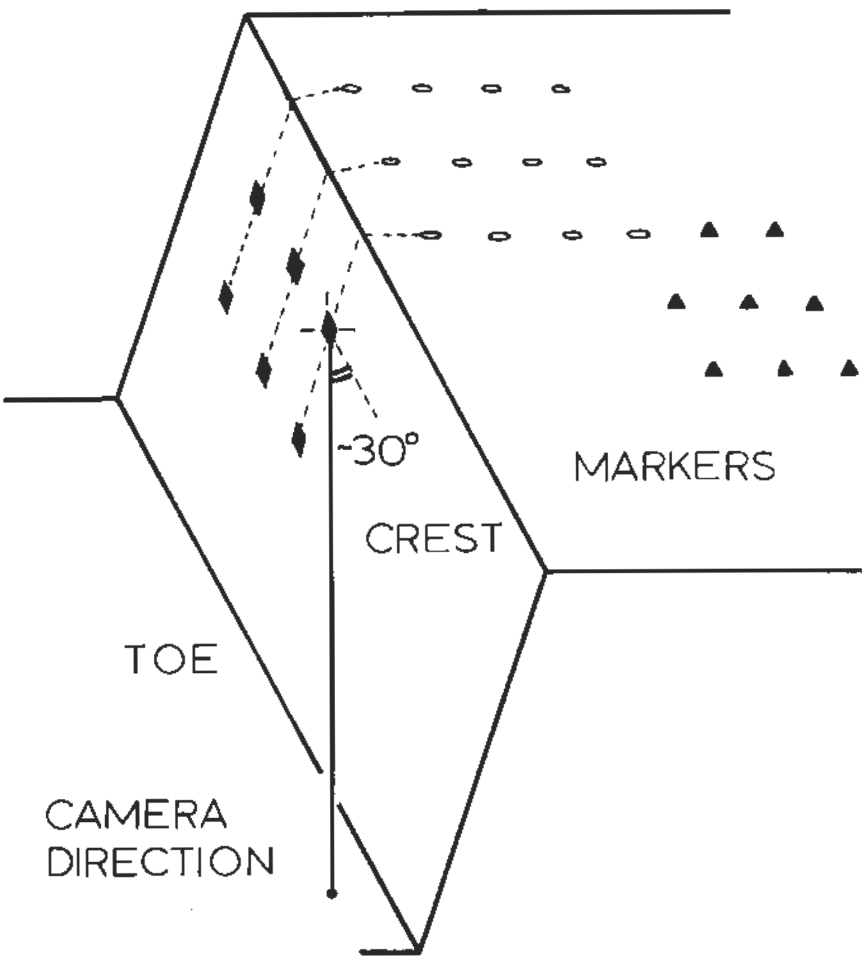

FIG. 2 CAMERA VIEW

firing sequence of multiple row blasting and correlation with respective delay periods, location of miss holes, degree of confinement by stemming and initial rock movement can be of immediate benefit to blast design and planning. In addition, the colour of escaping gases may also indicate possible deflagration or changes in the explosive composition.

The high speed photographic technique is particularly suited to quantitative measurements of a large number of parameters for improved blast efficiency. The accuracy of measurement is controlled only by the time resolution of the camera and the dimensional controls such as markers or grids on the bench. The measurements include particle velocity on the bench face, and hence the spalling strength, the time of critical rock movement on the bench face, velocity of fly rocks, the uplifting velocity on bench top near adjacent holes, determination of delay time for optimum rock movement, fragment size, estimation of required heave energy from the rock mass motion and estimation of borehole pressure at the instant of ejection of stemming. The last two parameters are of particular importance in predicting explosive action and blast patterns in rock through analytical models.

5. DATA ANALYSIS

The processed film is played back through a L-W Photo-Optical Analyzer (Model 224-A) for visual inspection of the entire blast. For a more critical examination and measurement of displacement of a particular target at a given time, the film is played back frame by frame. Since a pre-set scale system is set up on the bench and on bench face by means of markers, tedious methematical conversion process in obtaining the true displacement values is largely eliminated. The inclination of the optic axis due to camera location and the possibility of target moving off non-parallel to the film plane are taken into account in determining displacement; the latter is accomplished by averaging. The accuracy of displacement measurement is estimated to be 10%.

A cross-section of the bench with markers (M) is shown in Fig. 3. A typical blast pattern for an open pit mine in hematite ore formation is shown in Fig. 4. The corresponding displacement vs. time curves are shown in Fig. 5. In this particular case the velocity of free face is considerably larger than that on top of the bench. The initial large scale rock movement on the bench face occurs at about 100 msec. after blast initiation and at about 200 msec. on top of the bench for the pattern and delay shown in Fig. 4.

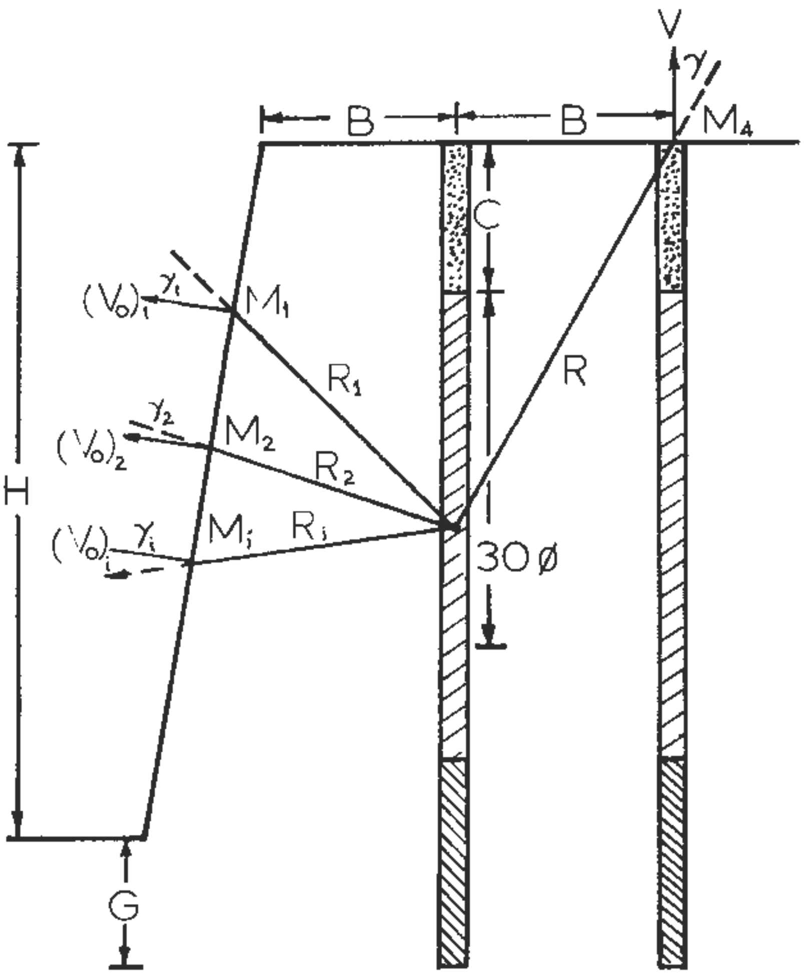

FIG.3 BENCH CROSS SECTION

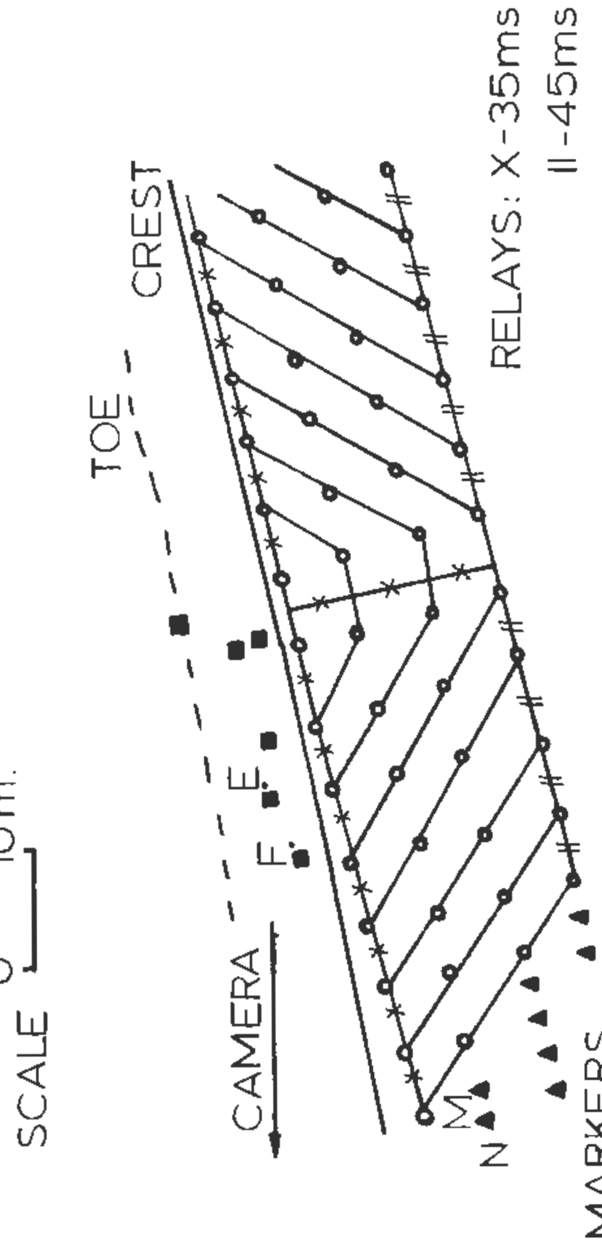

FIG. 4 BLAST PLAN

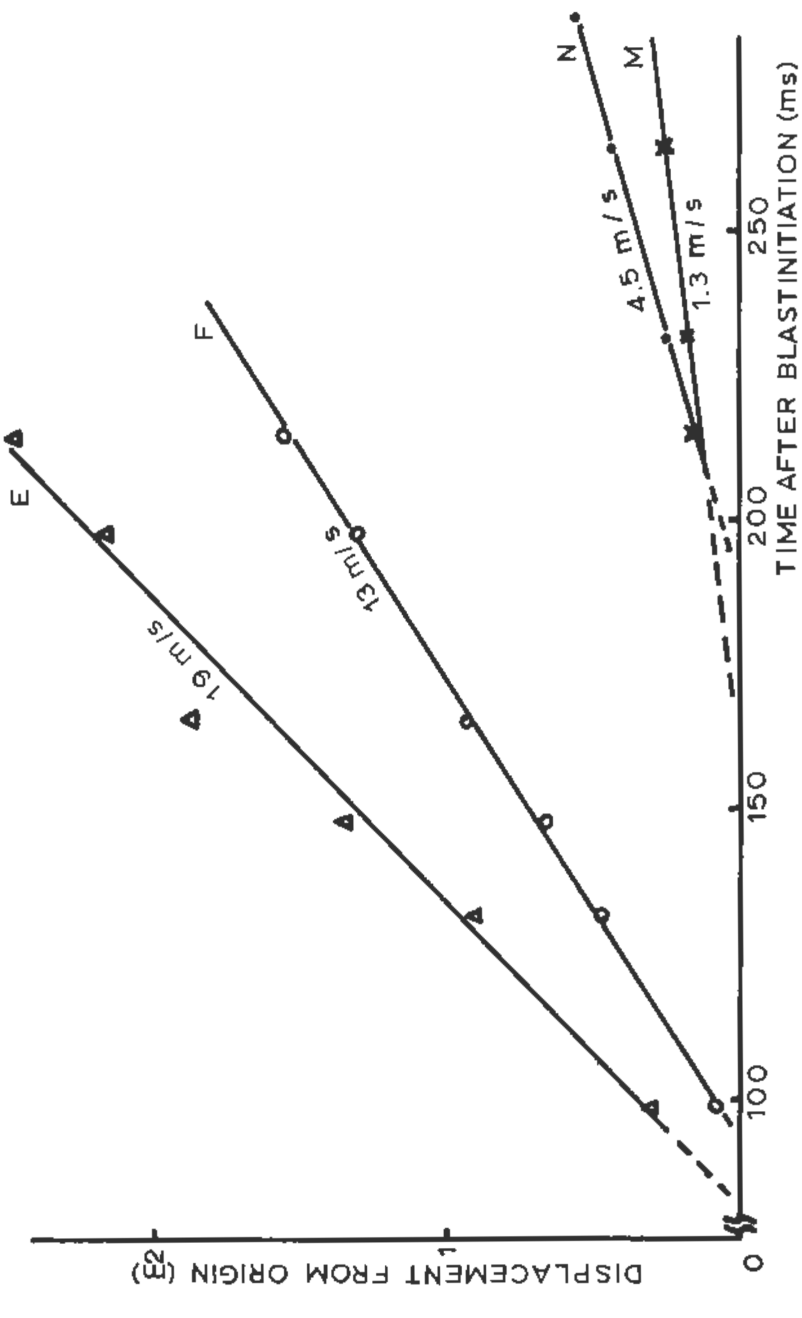

FIG 5·DISPLACEMENT VS TIME

Fig. 6 shows variation of uplifting velocity on the bench top vs. scaled distance ($R/W^{1/2}$) in another iron ore mine. 'R' is the distance (m) from the centre of gravity of the explosive column and 'W' is the total charge weight (kg). It is seen that the data can be fitted empirically by the following:

$$V = K \left[\frac{R}{W^{1/2}}\right]^{-n}$$, where K = 2.24 and n = 0.93. The constants 'K' and 'n' are characteristic parameters for the particular explosive-rock combination and include attenuation and geometrical spreading factors. Thus from a set of data obtained from film analysis the particle velocity at any point on the bench may be calculated. An important blast design parameter which can be estimated from these velocity values is the absolute maximum permissible delay time for the nearest delayed shot. This is the time for maximum extension which a section of detonating cord in the delayed borehole can sustain before failure. However, caution must be exercised in estimating particle velocity from the bench top to that of the bench face as different conditions prevail for the two geometrics as shown in Fig. 5.

Typical flyrock velocities for an iron ore mine are shown in Fig. 7. These velocities are much higher than normally encountered in blasts of proper design. These figures also show subsequent increase in flyrock velocity

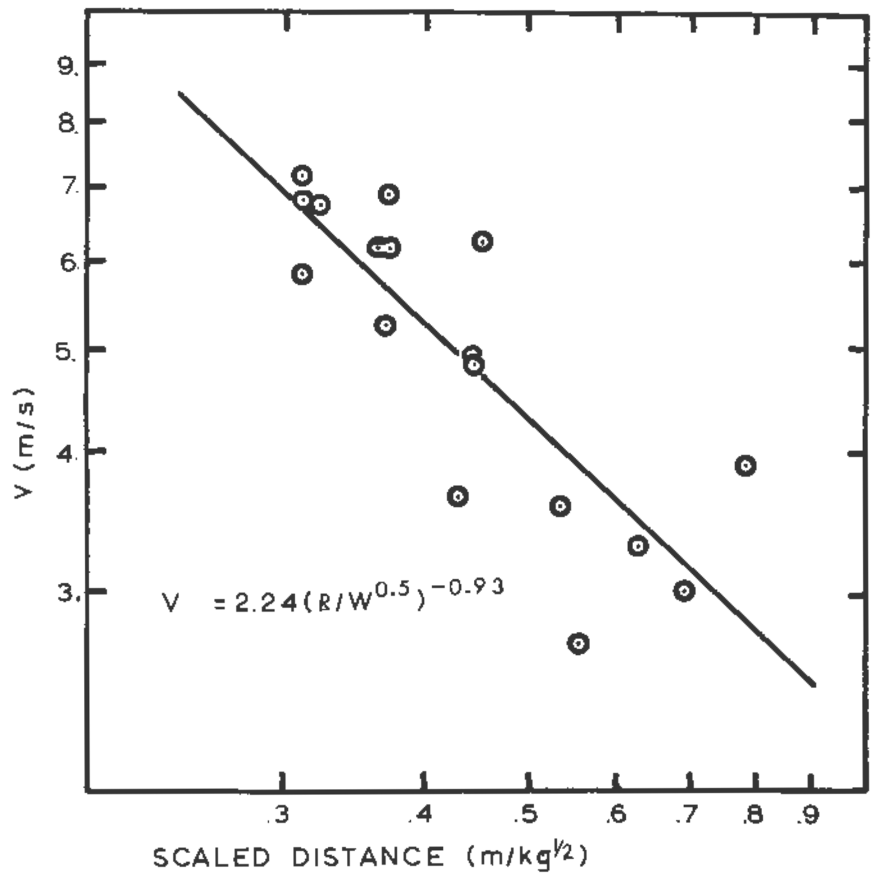

FIG. 6 UPLIFTING VEL. VS SCALED DISTANCE

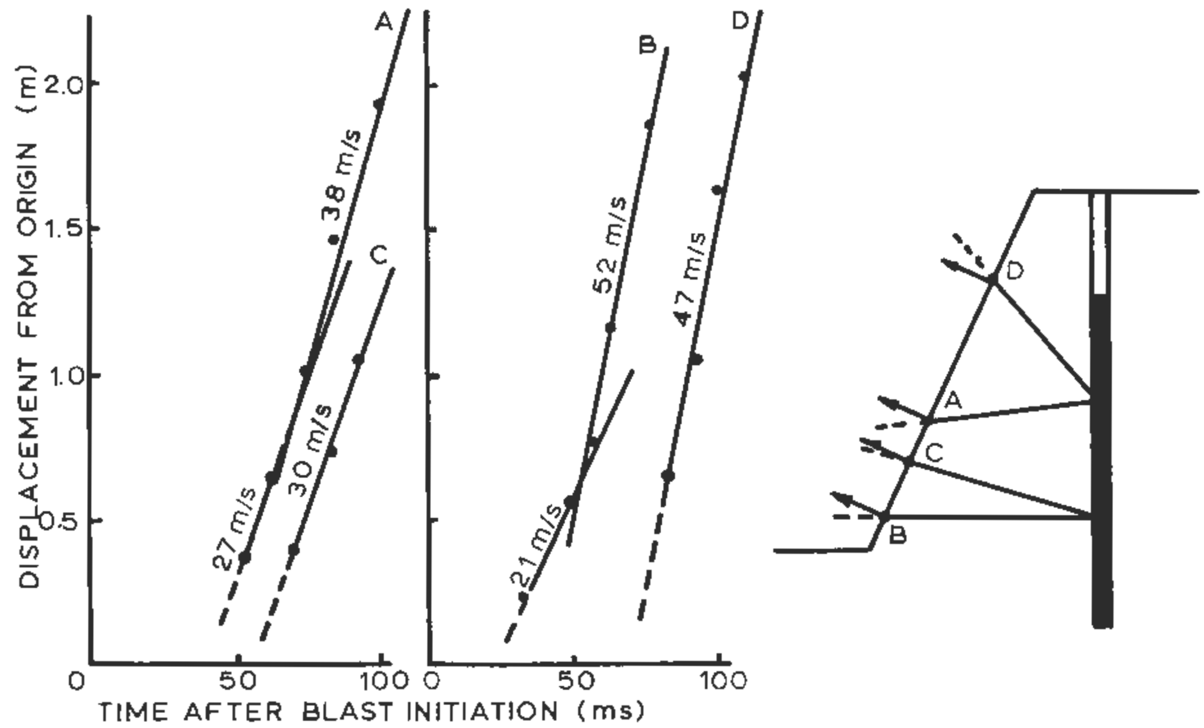

FIG.7 FLY ROCK PROFILE

due to multiple collisions.

6. CONCLUSIONS

The high-speed photographic technique is a simple and direct means of studying blast results in open pit mines. In current open pit practices, frequently involving over 100 ktons of rock per blast, proper selection of explosive and optimum design of blast patterns is of crucial importance. High-speed photographic studies provide both qualitative and quantitative data for greater blasting efficiency and avoiding costly errors. As a research tool such photographic studies can provide important data on explosive energy partition, determination of optimum delay between rows and general verification of theoretical blast predictions.

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SLOPE MONITORING AT BRENDA MINE

LES PROBLEMES POUR SURVEILLER DES PENTES DE LA MINE BRENDA

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D. Pow**
M. Keast***

Abstract

The problem of monitoring slopes at a particular mine are discussed. Methods of measuring wall movement are examined and the combination of electronic distance measurement and theodolite angular measurement chosen. To decrease the required manpower, an electronic geodimeter combining both functions was bought.

In its simplest form of operation, the distances are measured to ±6 mm and angles to ±5 sec. By using more sophisticated techniques, distance errors can be reduced to ±2 mm. The cost of good reflecting targets is such that a method of utilizing inexpensive short range reflectors was developed.

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The accuracy of the machine is found to be quite adequate for monitoring failing areas.

Resumé

Les problèmes pour surveiller les pentes d'une mine particulière sont discutés. Les méthodes pour mesurer le mouvement des parois sont examinées. La combinaison de la mesure electronique de distance et de la mesure angulaire avec theodolite a été choisie. Pour diminuer la main d'oeuvre requise, un géodimètre incorporant les deux mesures fut acheté.

Dans sa forme la plus simple d'opération les distances sont mesurées à ±6 mm et à des angles de ±5 sec. En utilisant des techniques plus sophistiquées les marques d'erreur peuvent être réduites de ±2 mm. Le coût de cibles qui réfletent bien est tel qu'une méthode utilisant des reflecteurs à courte distance et peu coûteux fut développée.

La précision de cette machine fut trouvée suffisante pour surveiller les zones instables.

1. INTRODUCTION

1.1 Brenda Mine

The Brenda open pit is located 26 miles west of Kelowna in the southern interior of British Columbia. The mine elevation is 5000 feet above sea level. Temperatures range from a minimum of -30°C to 30°C with freezing conditions experienced from mid-September to mid-April during which an annual snowfall of 400 - 800 cm is experienced.

A low grade copper-molybdenum ore is mined at the rate of about 27,000 tons per day with a similar amount of lower grade and waste material being stockpiled. A fractured quartz diorite rock gives a competent north wall and good east and west walls. The low grade nature of the deposit dictated a mine plan consisting of several pits inside each other. Pit One is nearing completion. Its north and east walls are being removed as Pit Three deepens. Pit Four has just commenced and will eventually produce new walls all around as shown in Figure 1.

1.2 Slope Monitoring Considerations

With wall heights in the order of 1000 feet planned for the future, management thought it wise to have a simple yet effective method of monitoring these walls, to determine if movement is occurring and if failure was imminent.

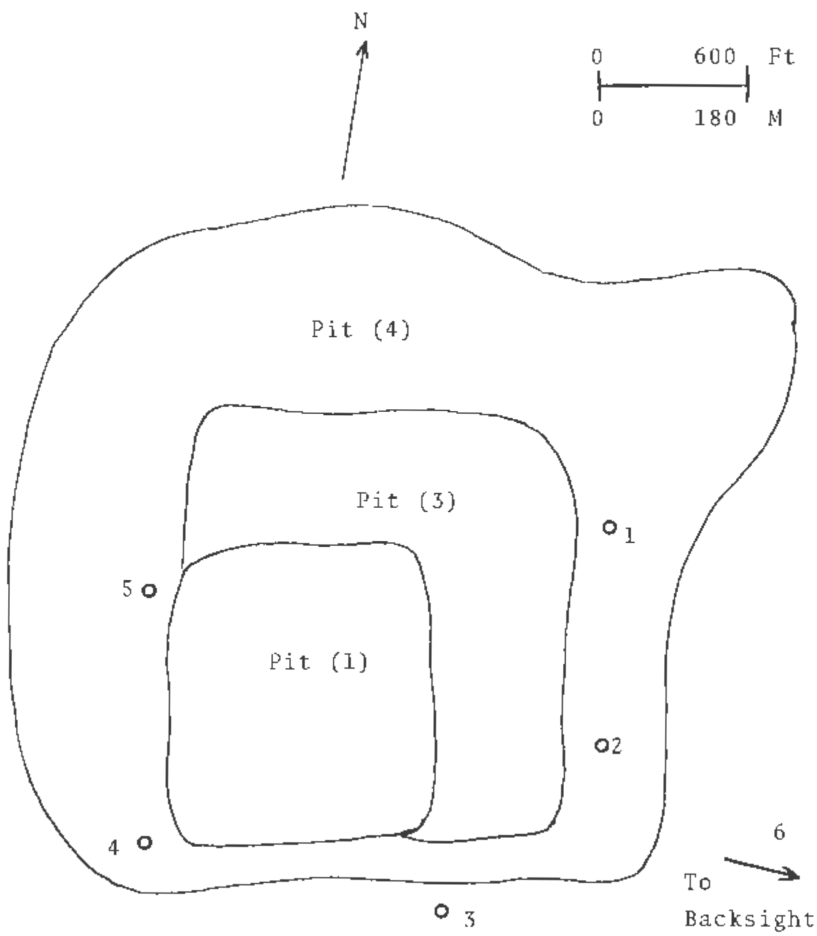


FIGURE 1

SKETCH PLAN OF PIT SHOWING SURVEY STATIONS

An investigation of the different methods available, their costs and manpower requirements when applied to the Brenda operation was made. Basically there are three methods available:

a) Deformation in a borehole

b) Photogrammetry

c) Surveying

The costs of any system are roughly proportional to the degree of accuracy required. Figure 2 shows the relationship between cost and the amount of movement easily detected. This data only applies to the Brenda operation and does not go into details of capital and operating costs.

For an unlimited sum of money, any movement can be detected. Published literature[1,2] indicates a wide range, from inches per month to inches per day, of movement during a slide. Some movement must occur due to the blasting and removal of vast volumes of rock which is of no immediate concern. However, unstable movements must be detected early enough to give management sufficient time to alter the mining plan or preferably, time to consider stabilization of an area.

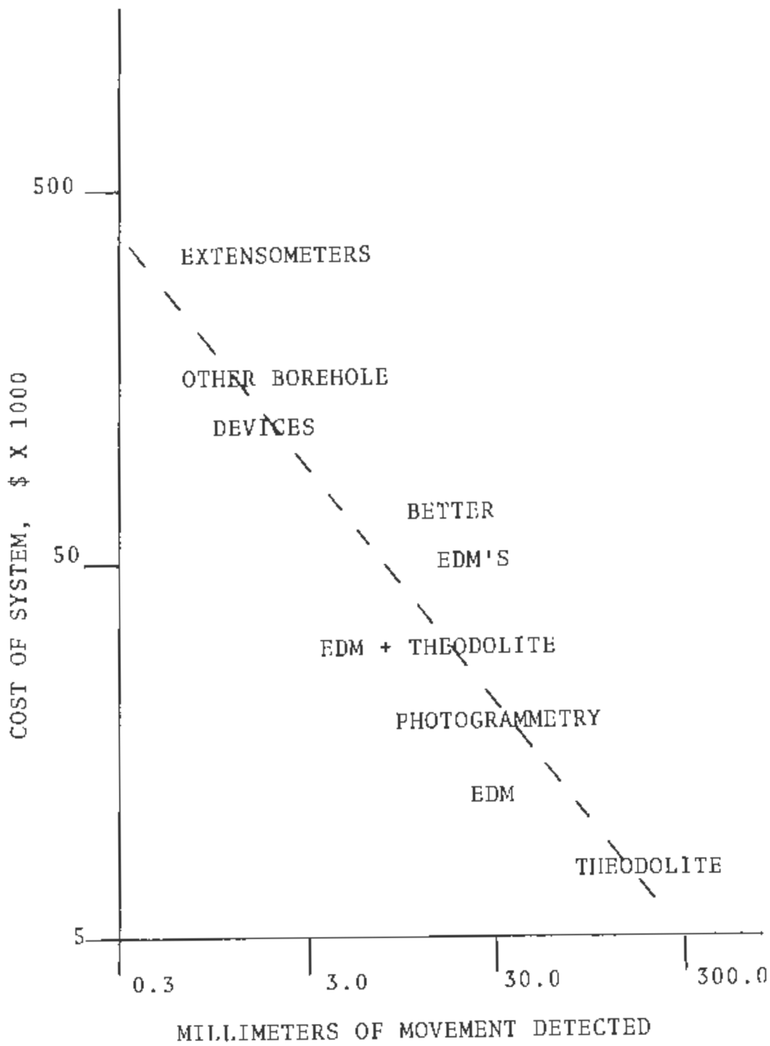


FIGURE 2

SOME METHODS, ESTIMATED COSTS AND ACCURACIES OF SLOPE MONITORING SYSTEMS

The system selected should require a minimum of man hours for the collection and analysing of data. It would be beneficial if the direction of movement was detectable, as this might indicate which fault plane should be strengthened if stabilization was to be carried out.

The choice of a system is limited by the mining method. Only the final wall can be adequately monitored using boreholes. This increases the cost and accuracy but gives no indication of how earlier pit walls are standing. Photogrammetry would not provide instant information unless the analysing system were installed at the mine. Even so, it would probably take considerable survey time to obtain the co-ordinates of set up and target points. This leaves survey methods as the final choice.

1.3 Survey Monitoring

As a simple example, the movement of a point around a one foot cube was simulated by sightings from three points 3000 feet away. Neither triangulation nor trilateration would indicate all movement. However, a combination of the two methods would indicate a one inch movement relatively easily. If instrument stations were set up to the best advantage, the amount of movement detected would be limited only by the machine accuracy.

The most inexpensive approach was to use the theodolites already available at the mine in conjunction with an electronic distance measuring unit (EDM). Many EDM's were examined and several were tested at the mine site. The helium-neon type was not thought to be better or worse than any other, but was relatively inexpensive and utilized a visible red light beam.

Manpower requirements for this system would be large because of the number of readings required with the theodolite. To measure distances the EDM is then mounted on the tri-brach and further readings taken. Data analysis would be an equal problem in terms of man hours.

AGA suggested that their Geodimeter 710 would alleviate these problems, but at a greater initial capital expense. Looking to the future it would be reasonable to assume that routine mine surveying would be automated to provide maps, etc. directly from recorded instrument data. A combination of the AGA Geodat 700 punch tape recorder and the 710 Geodimeter would reduce manpower requirements and instrument time. Monitoring and routine survey work would be a combined operation with data and maps presented in complete form from the Geodat punchtape.

A telephone conversation with Dr. D.G.F. Hedley at the Department of Energy, Mines and Resources confirmed that the instrument would meet the manufacturers specifications in most respects.

Management then decided to purchase the 710 Geodimeter after demonstration.

2. A BASIC MONITORING SYSTEM

2.1 Stations

Having concluded that surveying would provide most of the slope monitoring information required, concrete survey stations were installed around Pit 3 in the summer of 1974. These stations were made of weld mesh, rebar and old grinding rod. "Ready Rod" was welded to the rebar to hold an instrument platform. Concrete was poured into a 3 foot diameter cardboard mold around the reinforcing structure. A brass platform and forced centering bolt were added later. Figure 3.

Five main stations were set around the pit walls, and a further station outside of the influence of the pit was added to provide a stable backsight for all five main stations. With the arrival of the Geodimeter in December 1974, basic monitoring commenced with the surveying of the main stations.

2.2 Using the Geodimeter 710

Table 1 gives a brief description of the instrument. There are two main parts, the measuring and display units, joined by an electrical cable, Figure 4. A full description is available from AGA[3,4]. Briefly, angles are measured by means of horizontal and vertical pulse discs. The relative position of tracks on the discs is converted electronically to angular position.

Distance measurement is accomplished by determining the phase difference between transmitted and reflected light beams. Frequency (1) finds the nearest 5 metre, frequency (2) the nearest 0.001 metre. Frequency (3) is used to solve units of 250 metres up to 5000 metres.

The beam is reflected from either a cube corner glass prism ($50 - $200 each) or a plastic reflector ($2 - $5 each). Alignment of the prism can be 20° off, but the plastic reflector loses range quickly as the face is turned away from the beam. Returning signal strengths can be estimated using the meter on the display unit. Adjusting the angular verniers and beam expander to give maximum signal strength is a useful aid when measuring distances.

The display unit automatically gives various combinations of slope, horizontal and vertical distances with horizontal and vertical angles as required.

TABLE 1

AGA 710 Geodimeter

Cost	$22,000
Weight (including carrying case)	25 kg (55 lbs)
Power	12 V. DC

Distance Measurement

Expandable 1 mW He Ne gas laser

Modulation Frequencies	F_1	299.7	K Hz
	F_2	29970	K Hz
	F_3	30000	K Hz

Range 1 AGA prism	1700 m	(1 mile)
1 plastic reflector	200 m	(600 ft)
Resolution	1 mm	

Angular Measurement

Erect Image	30X magnification
Resolution	Horizontal Circle 1 sec
	Vertical Circle 3 sec

FIGURE 3 SURVEY STATION

FIGURE 4 AGA GEODIMETER

Since the machine utilizes the laser principle, eye safety must be considered. The one milliwatt laser emits an average power of 0.125 milliwatt per square centimetre and should not be stared at from a close range. The normal human reflex is to turn away from the visible beam and therefore the machine is considered safe when used as directed[5].

Safety precautions are incorporated in the machine and Brenda has undertaken others:

i) The laser beam is turned off when not required, e.g. when first sighting a nearby target through the telescope.

ii) There is a filter which is used when operating at short ranges.

iii) All mine operating personnel have been given a safety talk on the machine.

iv) Survey personnel are provided with safety goggles which do not transmit the red light.

v) Warning signs are present which tell the reader not to stare at the beam.

2.3 Factors Affecting Machine Accuracy

The beam is affected by the density of the medium in which it travels. A correction is therefore necessary for the variations of air pressure, temperature and humidity. The correction factor is obtained from a table and dialed directly into the machine. Over a distance of 1000 metre, about 1 mm error would be caused by an error in the temperature reading of 2°C, at constant pressure. Humidity changes have very little effect on red light.

Patches of warm and cold air over the beam path produce erroneous results whereas overcast windy mornings or nights produce the best results.

Measurements of pressure and temperature present problems. In most stability work measurements over the beam path are not always possible. The most practical system would be to place a maximum/minimum thermometer on a pole near the target and use an average of the target and instrument conditions for calculating the correction factor.

Temperatures around the pit vary by ±2°C at any time during the morning. Pressures vary due to elevation by ±1 mm Hg. The error caused by these changes is in the ±2 mm range, but does not greatly affect monitoring data using the same stations.

At Brenda it has been found that other climatic conditions, wind, cloud, snow and freezing rain have an effect on the machines operations. The cloud level occurs at the mine elevation during the winter months.

Dense fog can remain at the mine level for several days. Generally a reflection from a cube corner prism can be obtained although the target itself may have just disappeared from view.

Wind tends to disturb the instrument, but the spread of the beam is such that suitable distance measurements can be obtained. Snow and rain tend to give a greater spread of readings about a mean.

A major winter problem was the coating of the reflectors in ice, up to 1/2 inch thick. This problem was solved on a temporary basis by use of an insulated shield with an open front face. A 12V lamp bulb heated the inside of the shield and kept the reflector from freezing. The complete winterization of the system will be completed in 75/76.

Unfortunately there was no target housing available that was 100% satisfactory for mine purposes. For this reason a housing was made in the mine shop consisting of a 2 1/2" pipe coupling, welding road and a 5/8" bolt. Figure 5. This housing is now standard at Brenda, and fulfills all requirements.

FIGURE 5

TARGET HOUSING

FIGURE 11

SOUTH WALL
HAUL ROAD

3. IMPROVING THE SYSTEM

3.1 Routine Surveying From Main Stations

The initial system employed involved setting up on a main station. Readings were taken to the main stations, and reflectors mounted in 3" holes drilled in the rock faces were monitored. It was found that operators tended to bias their distance readings instead of reading the first reasonably stable number. At this stage the manufacturers stated accuracy of ±0.02 feet was not being attained. To overcome these problems, a regression analysis procedure was adopted. Readings were taken at various correction factors instead of dialing in the correct one. Using the various correction factors and distances a least squares equation was used to obtain the correct value of slope distance. This reduced errors significantly and gave the operators a better feeling for the drift in readings.

At AGA's suggestion, the instrument was converted to display metric distances. This solved the problem of accuracy because it removed on electrical circuit and stopped truncation of the last decimal of distance. An examination of the data showed that four readings, two face left and two face right, would be sufficient. The regression method was abandoned in favour of dialing the correction factor as before. A plot of typical data is given in Figure 6; a spread of ±6 mm is evident.

Angular measurements were taken using the reiteration method[6] on one quadrant. The instrument did not display the positive action of the normal theodolite. Consequently care had to be taken not to disturb the instrument after sighting a target, e.g. pulling on the cable to the read out unit. Typical data is given in Figure 7.

All angles are meaned to about ±5 second. The scatter and standard deviations agree with those published[7] for a 0.1 second theodolite.

It should be noted that vertical angles are corrected internally by an electronic plumb which is set during the initial machine calibration process. Face left and face right angles must be taken since the instrument does not hold its adjustment.

3.2 Normalizing Data to a Known Backsight

The error in day to day readings is from two main sources, atmospheric conditions and internal machine error. The atmospheric correction dialed into the machine may not have been correct since temperature and pressure readings cannot conveniently be taken at targets all around the compass. Present practice is to use the temperature and

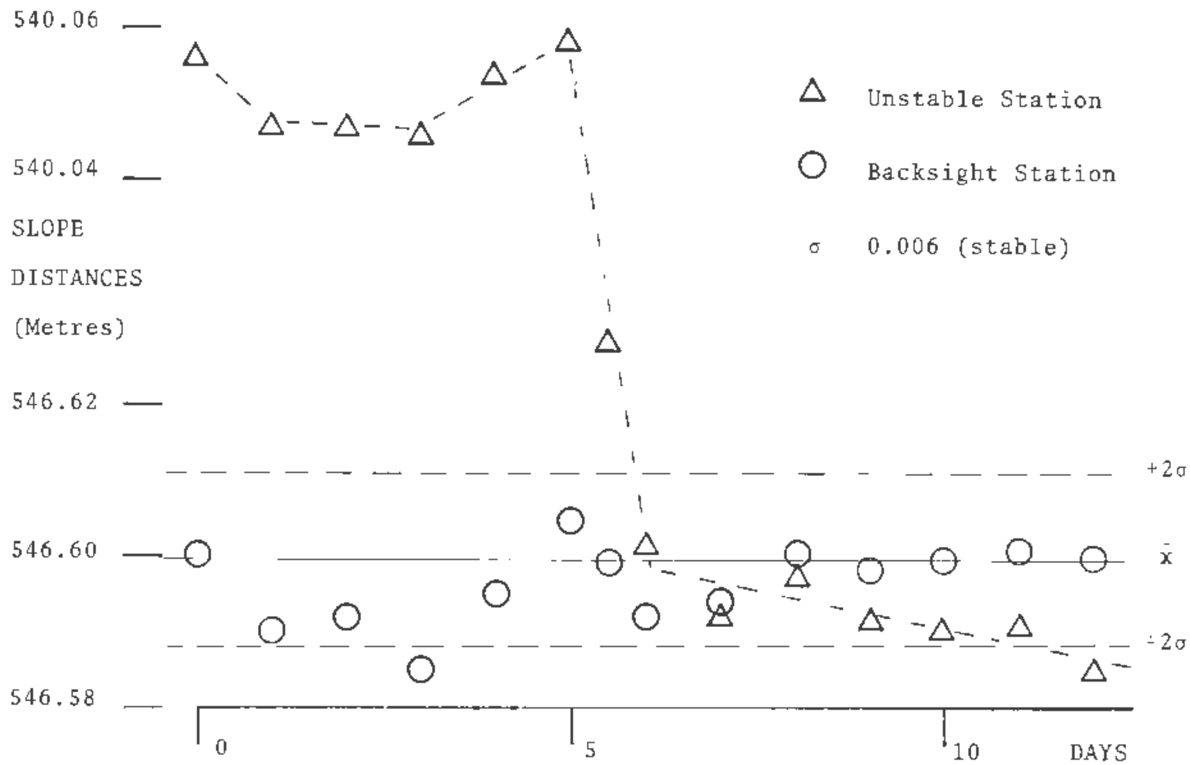


FIGURE 6 TYPICAL SLOPE DISTANCE DATA

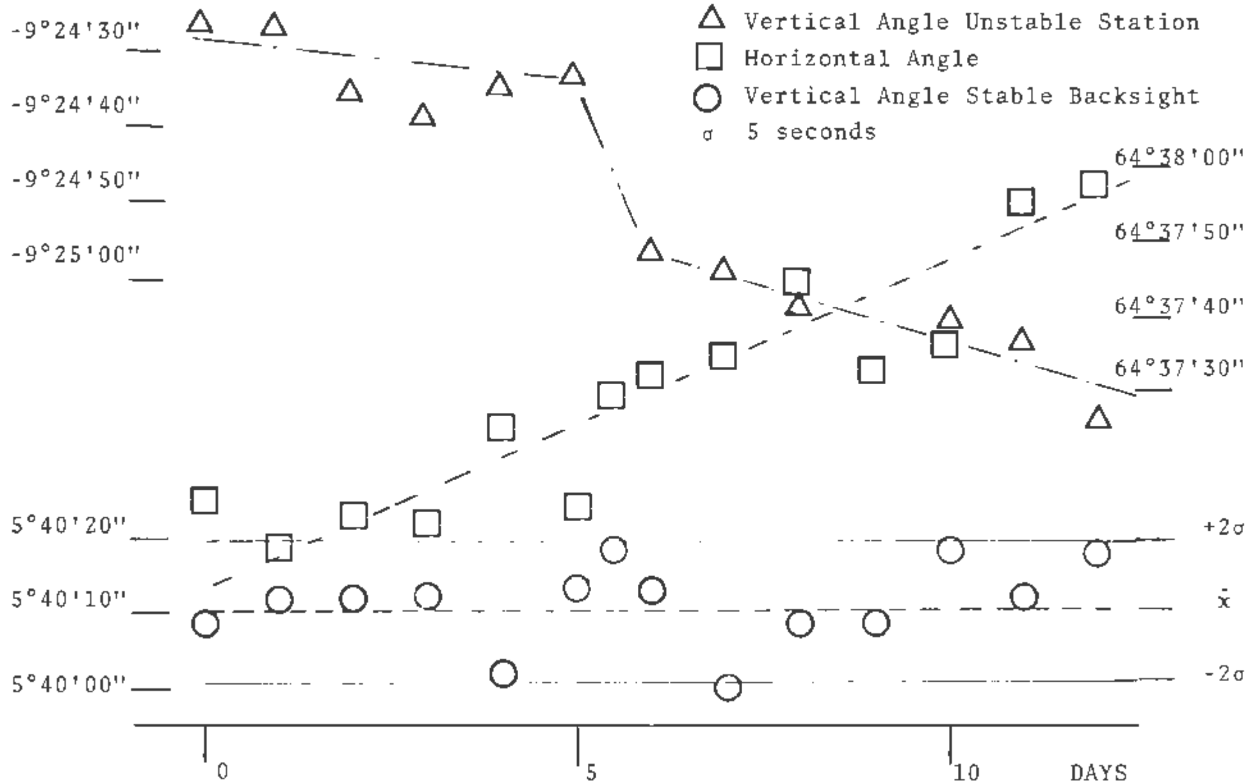


FIGURE 7 TYPICAL ANGULAR MEASUREMENTS

pressure correction at the instrument only.

Readings are taken from one main station to monitoring points and the main backsite. As all main stations are surveyed regularly, they can be considered stable. Any error in backsite slope distance is applied to all other distances measured from that particular set up.

Atmospheric corrections are proportional to the measured distance, and the instrument error can be considered to be constant for a particular set up.

The application of the correction could be made on a distance proportional basis or as a constant if all distances are of the same order. Limited experiments did not prove the "distance proportional" correction to be any better than the constant, so the simpler constant method was adopted. To obtain the best results it would be advisable to use a backsight distance approximately equal to the target distance.

The improvement in using this method is shown in Figure 8.

3.3 Forced Centering With a Reference Reflector

A procedure recommended by AGA[8].

The instrument uses three frequencies to resolve the

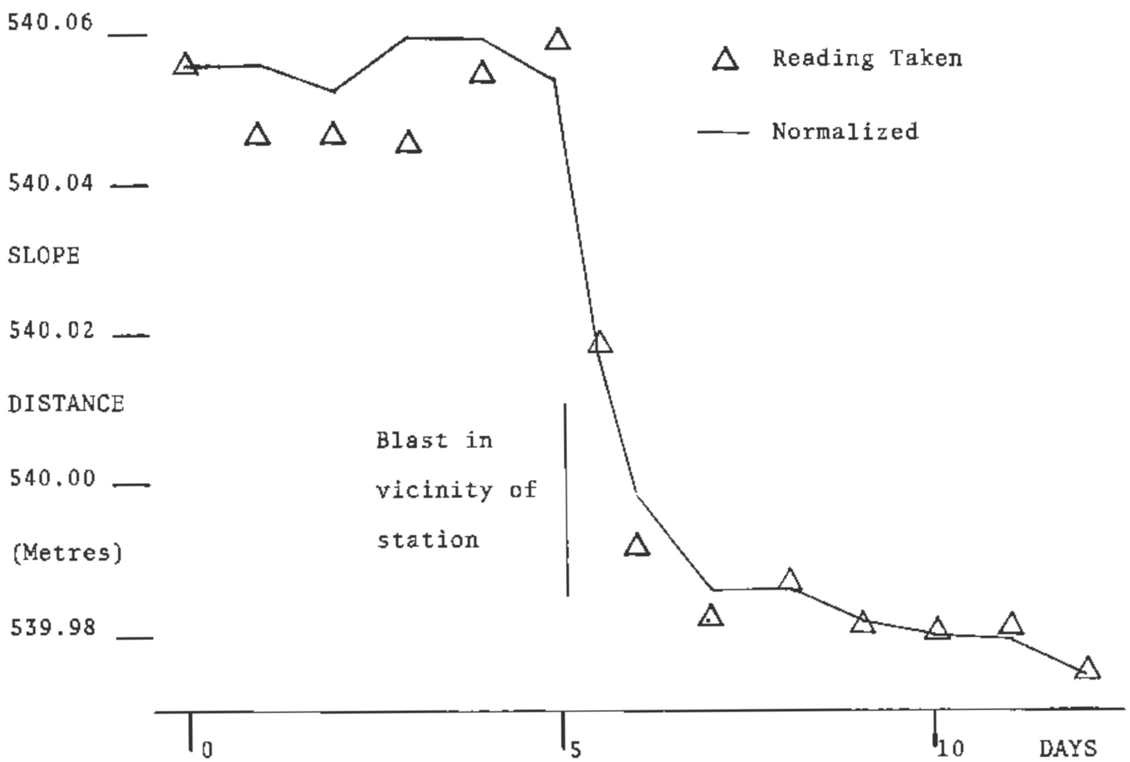


FIGURE 8 SLOPE DISTANCE NORMALIZED TO BACKSIGHT

target distance. The final 5 metre resolution utilizes a 10 metre wave length. The phase difference does not account for negative or positive parts of the wave form. The effective range of this final wave form is 10/2 or 5 metre.

Errors in the wave form would be eliminated if readings to a known backsight used the same part of the wave form.

If readings were taken quickly from monitor to reference to monitor etc., etc., the drift in readings would cease to have a great effect, and other internal machine errors would be reduced.

The only problems remaining are atmospheric corrections. If the reference and monitor are at the same distance and in the same area, then the atmospheric correction need not be considered. The computation of the monitor distance will always be corrected regardless of what is dialed into the machine.

It is more convenient to use a short reference distance. In this case the atmospheric correction is not dialed in, but computed and added to the final result later. Only the distance to the monitor is found using all three frequencies. From this point on, the machine is left in the 10 metre wavelength, and only the last two digits change for each reading. An example of the calculations required is given in Table 2.

The data plotted in Figure 9, shows that ±2 mm can be obtained without much effort. We feel that night surveys with great attention paid to setting of reference reflector and atmospheric data could reduce this to ±1 mm. The method is a little tedious, but would have an application where a fine degree of accuracy was required.

3.4 Using Inexpensive Plastic Reflectors

To adequately monitor a pit slope, a reflector would be required for about 50,000 square feet of face. A pit one mile wide would need about 200 reflectors. Problem areas would use as many again.

The variation in cost of using $2 plastic or $50 glass reflectors is $800 to $20,000. The best system would use mainly plastic reflectors.

Unfortunately the range of these units is not sufficient for sighting across an open pit. The instrument must then be brought close to the reflector.

In an operating pit a temporary station is not the answer. A method must be established to accurately find any instrument position and transfer this to the target. As benches are mined, the instrument position changes.

TABLE 2

Tabulation of Forced Reference Method

SLOPE DISTANCE

FACE LEFT		FACE RIGHT	
Station	Reference	Station	Reference
676.099	676.092	676.096	676.091
89	85	93	86
96	88	90	83
96	86	89	88
97	92	96	85
676.0954	676.0886	676.0928	676.0866

Mean Station 676.0941
Reference 676.0876 (16.0876)

As only frequency (2) used, readings to reference station are 676. Actual distance is 16.

Mean Temp. 49°F Pressure 24.73 ins

Correction Factor 68.5 (AGA Table)

Corrections

Station → $676.0941 \times 68.5 \times 10^{-6} = 0.0463$

Reference → $16.0876 \times 68.5 \times 10^{-6} = 0.0011$

Corrected for Atmospheric Effects

Station $676.0941 + 0.0463 = 676.1404$

Reference $16.0876 + 0.0011 = 16.0887$

But the reference station distance is 16.0900

Correction $= 16.0900 - 16.0887 = + 0.0013$

Station Distance $= 676.1404 + 0.0013 = 676.1417$

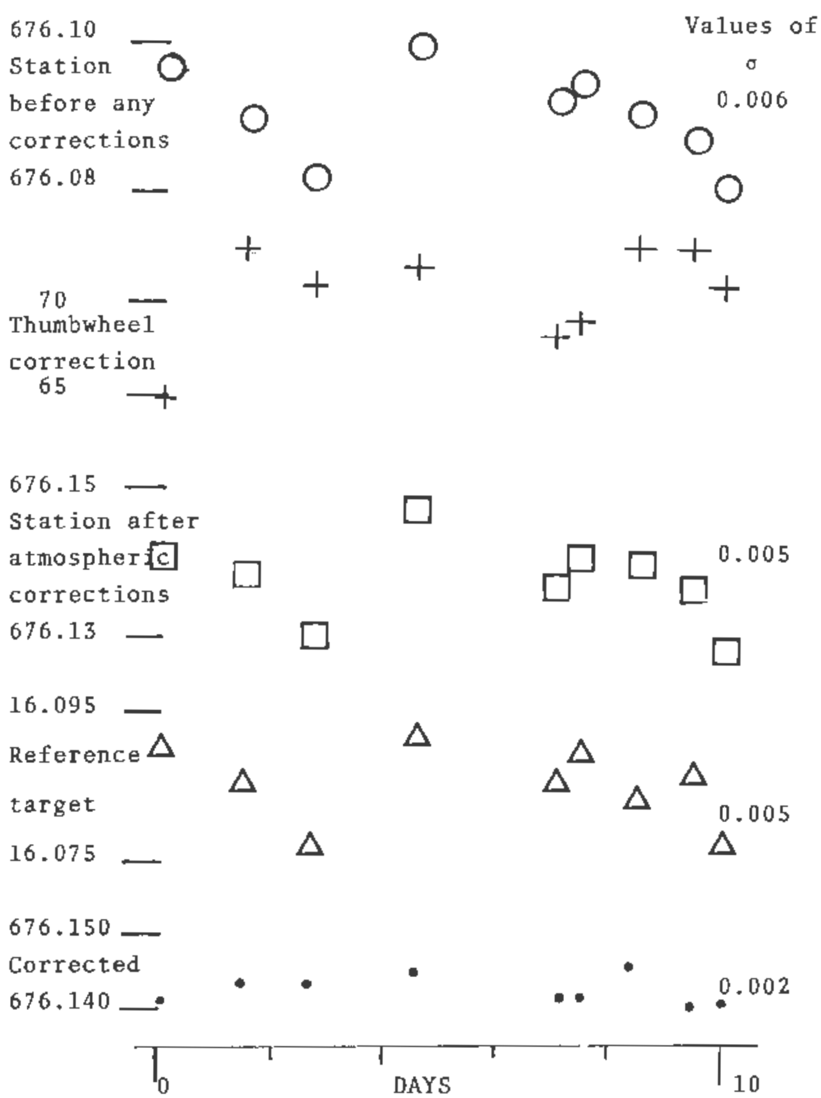


FIGURE 9 USE OF A FORCED REFERENCE REFLECTOR

Step one is to sight as many main stations around the pit as possible and use all the information to give a mean instrument position. As both angles and distances are measured, combinations of triangulation and trilateration can be used. The number of calculations and solutions are such that a computer program was written for the purpose.

A triangle comprising two main and one instrument station has two cosine rule and up to eight sine rule solutions for latitude and departure. Further main stations increase the solutions factorially. Elevations are simpler.

Instrument positions found from three stations generally have a spread of 3 to 10 mm in latitude and departure. Elevations have about half the spread. Errors in the system include computer error ($\pm$2 mm), atmospheric conditions ($\pm$2 mm). The mean co-ordinates are probably in the $\pm$5 mm range.

Transferring the co-ordinates from instrument to plastic reflector is quite simple.

As with every other system, an excessive amount of data is collected at the start. Presently four sightings are made onto all stations, using 0° and 90° quadrants. Readings are read into a tape recorder and cards punched directly. Eventually we hope to decrease the sightings to two per station on one quadrant only. Figure 10 shows typical data. As movement is known to be occurring in this area, the standard deviations are high.

A series of ten set ups would probably find a movement of 5-10 mm in any direction. At present a set up takes about 2 hours including keypunching cards.

The system is also useful in monitoring the main stations. Any movement of a main station affects the instrument co-ordinates. Experiments have shown that the moving main station can be easily isolated from the data.

4. A CASE HISTORY

The south wall at Brenda has two main jointing patterns. Analysed by photogrammetric methods these are:

a) strike 085/265 ±25°, dip 45° North ±5°

b) strike 075/255 ±10°, dip 72° South ±5°

The 45° dipping pattern is not continuous.

A study was undertaken to find the mechanical properties of the 45° North dipping set. A Hoek shear box was used for this purpose. Normal loads of 0 - 350 psi and shear of 0 - 250 were used. Cohesion intercept values varied between 0 and 20 psi, and friction angles of 25° - 35° found. A vertical hole was drilled to check water levels in the area.

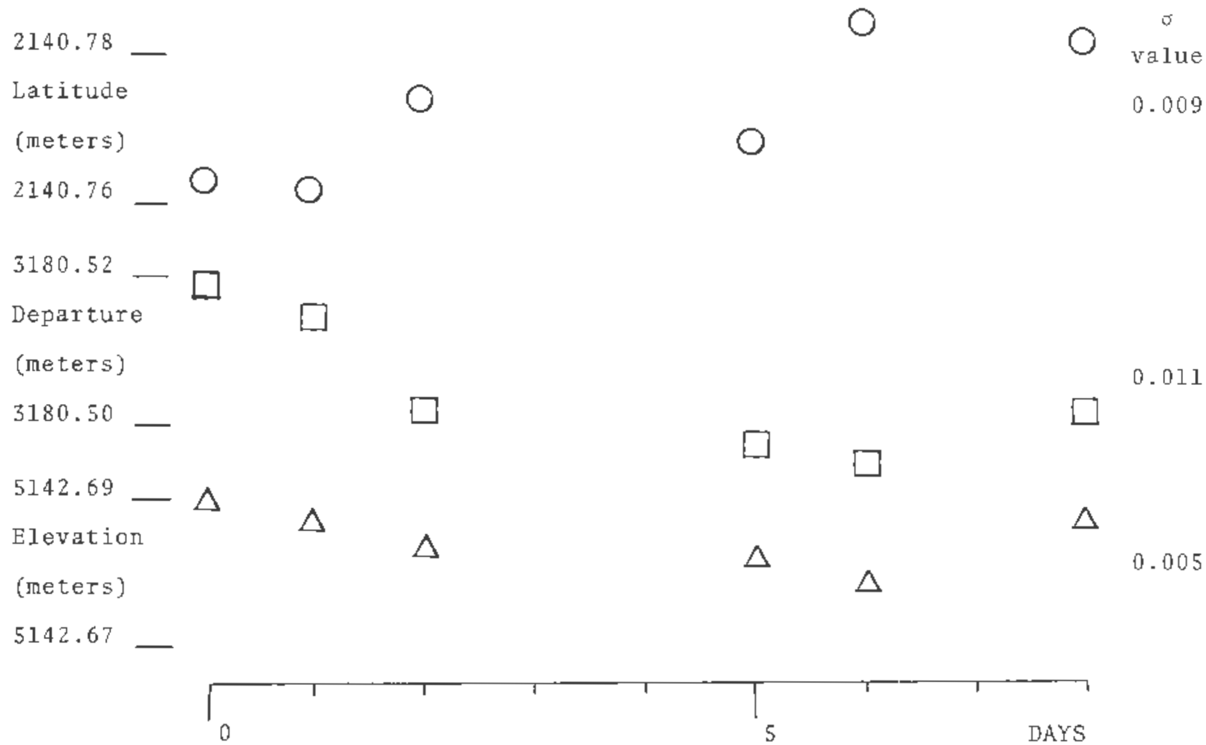


FIGURE 10 LATITUDE, DEPARTURE AND ELEVATION OF PLASTIC REFLECTOR FROM TEMPORARY STATION

A basic analysis was carried out using methods given by Hoek[9]. This showed the area to be very stable when dry. A high water table decreased the stabilities. A plane failure on the 45° joint system was assumed with tension cracks formed on the 72° plane.

The south wall area is shown in Figure 11. Stabilization of the area was not carried out, as the cost and time spent was not warranted. The haul road will not be used after Pit (1) is completed this year (1975). Further it would be possible to reroute traffic through Pit 3.

There are three main causes of instability in this case.

a) Natural jointing patterns.

b) Water pressure.

c) Blasting shocks.

The natural jointing pattern causes the bench face to break as a series of steps. The failure mode would be a combination of toppling and plane failure. Cable bolting would strengthen both planes, and has been well documented[10]. Water pressure would be a problem in spring, when pit pumping volumes increase from the normal 0 - 100 gal/min to up to 2000 gal/min for a few weeks. Theoretically, any weakness in the area would have appeared at this time.

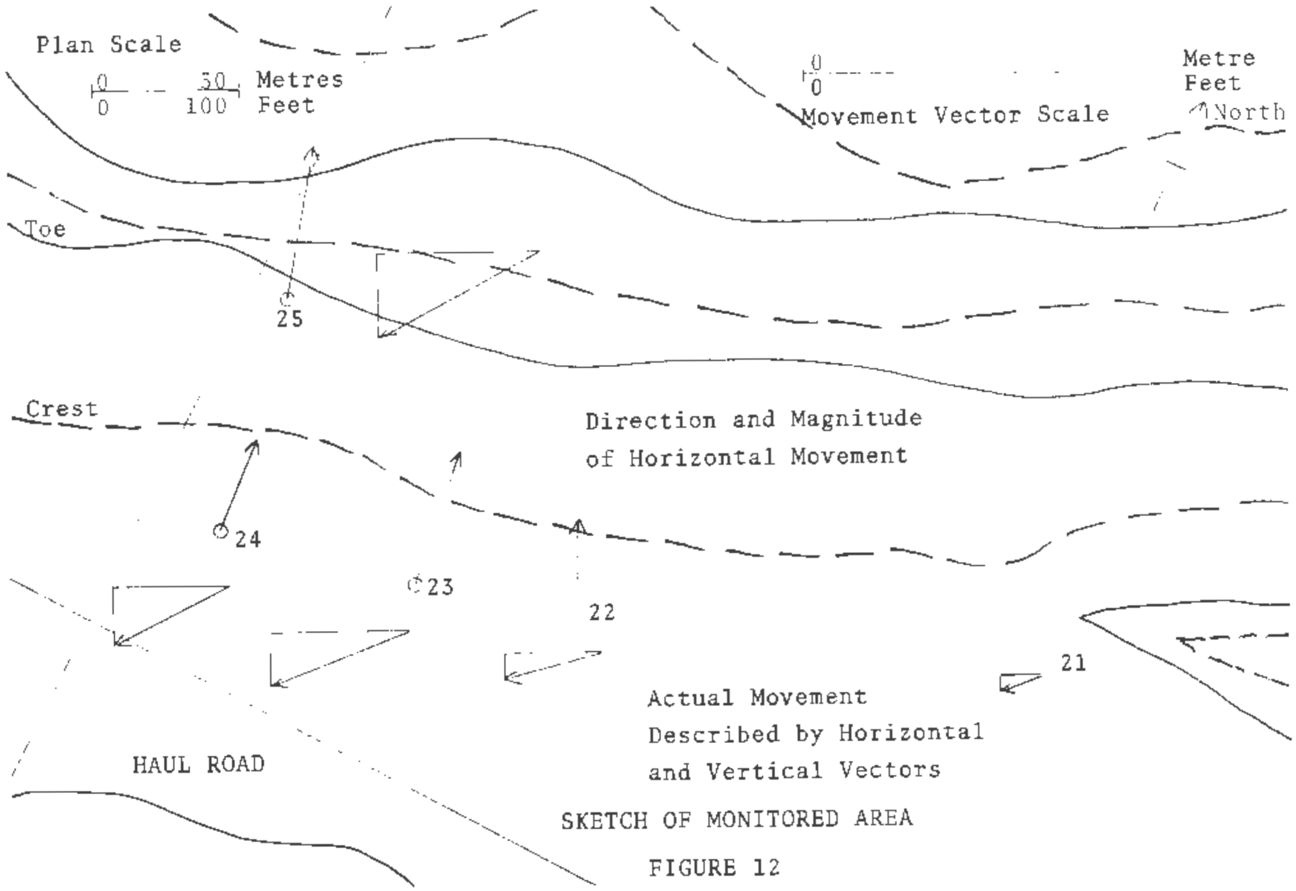
Any slope dewatering system would have to be designed for the spring peak, and consequently this might not prove feasible with gravity flow from boreholes.

The effects of blasting in the area eventually caused a small movement on a 'nose' on the south wall. Cracks appeared on the haul road. A more complete monitoring system was then installed to show the extent of the movement. Figure 12 shows the movement to date.

With water in the water monitoring borehole slowly dropping, the area should become more stable. Blasting immediately below the area and the resulting movement are shown for one station in Figure 13. The slope distances were measured from a station oblique to the movement. Despite this, the advantages of measuring distances rather that angles can be seen.

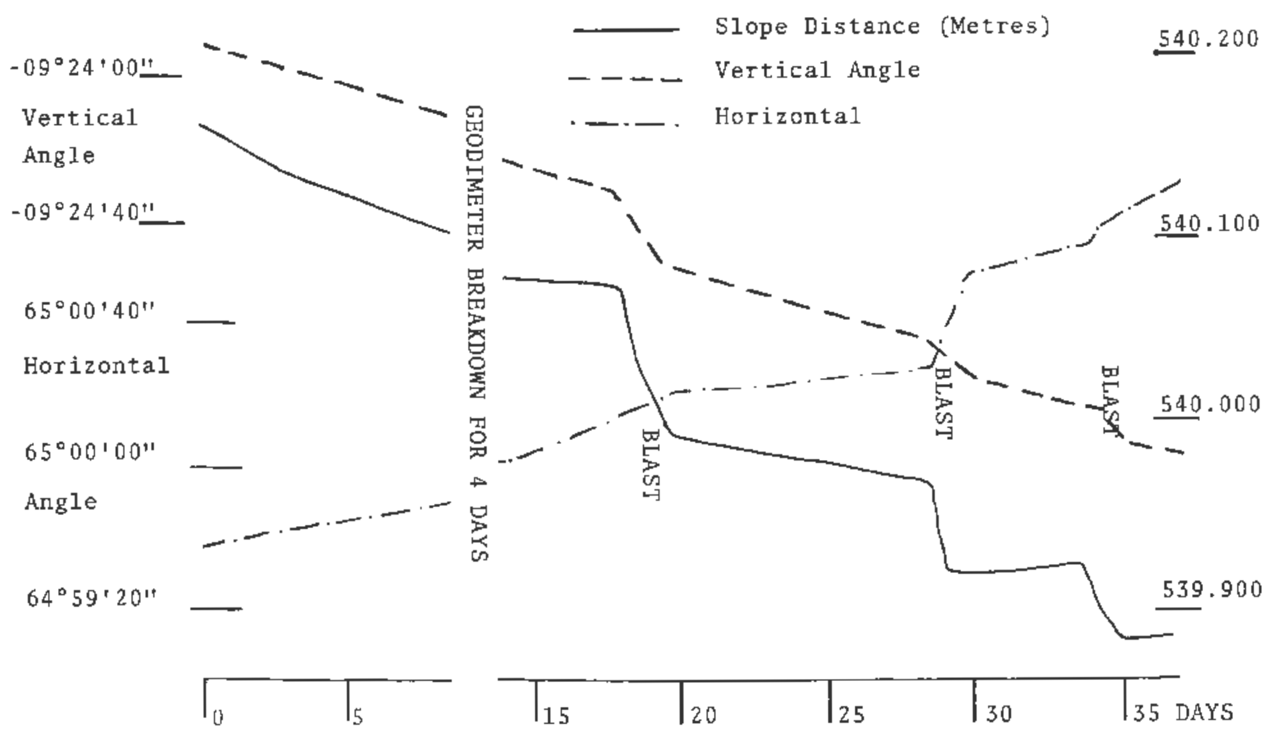
Some measure of stability would be attained by decreasing blast pattern sizes or increasing the number of delays used. This would decrease the scaled distance (distance from blast/weight of explosive$^{1/2}$) and decrease vibration[11].

The sinking cuts necessary to take out the next level gave $D/W^{1/2}$ of about 3-4 ft/lb$^{1/2}$ in the failing area. The relation between blasting and movement confirmed that the post shear trim blasting technique now in use for three



SKETCH OF MONITORED AREA

FIGURE 12



BLASTING EFFECT ON STABILITY

FIGURE 13

SLOPE MONITORING AT BRENDA MINE

years should be continued. (Loading 4 inch cardboard tubes inserted in the 12 inch drill holes. Spacing is 15 ft and subgrade 3 ft).

The area is now monitored using the plastic reflectors as in Section 3.4.

5. CONCLUSION

The EDM/Theodolite system is adequate for monitoring known problem areas. It also has the abilities to detect movements which apparently are of no immediate concern.

The calculation and display of data can be automated using an electronic geodimeter with a punch tape recorder. As a pit enlarges and workloads increase, manpower requirements can be kept at a minimum.

The use of inexpensive plastic reflectors and poor quality prisms would offset the high capital cost of the system.

ACKNOWLEDGEMENTS

We wish to thank the management of Brenda Mines Ltd. for permission to present this paper.

The assistance of the mine staff in the project was appreciated. We are grateful for the help given by Agatronics and many other geodetic instrument manufacturers.

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SUPER SCALE RESOURCE DEVELOPMENT

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1. INTRODUCTION

Large-scale resource development provides one of the most challenging fields open to geotechnical or earth science engineering.

The growing world demand for energy, water, and other raw materials has resulted in accelerated exploitation of

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the natural resources through what may be termed "superscale" projects. The term "superscale" is applied to projects of the magnitude of Syncrude, the Arctic Gas pipelines, the James Bay project, and others of similar multibillion dollar scope.

Because of the large scale of today's developments, they have the potential of interacting with nature on a scale previously unknown to man. The speed with which these projects are initiated may lead to adverse development features before the effects on environment or project operations can be recognized. After inception, so much funding will have been committed to the project that the developer might be unable to stop or change direction.

With the advent of superscale projects, there is a greater need for appreciation of natural site conditions. Immediate and long term effects of natural conditions on the project as well as the reverse effect must be considered in all phases of development.

2. SUPERSCALE PROJECTS

A comparison of conventional and superscale projects is considered in Figure 1. Whereas there have been many projects within Alberta and the Northwest Territories

	PROJECT TYPE	
	CONVENTIONAL	SUPER SCALE
SCOPE	Small — $50-100 Million -Gas Plants -Compressor Stations -Commercial complexes	Large — $1 Billion up — Syncrude — Arctic gas, oil lines — James Bay
IMPACT	Local Not large Relatively little disturbance	Vast — National & International Political Sociological Economic Environmental
MANAGEMENT & ENGINEERING	Close to problem & site Direct Communication	Remote — National International Complex Communications Often Third Party
REGULATION	Open & Flexible Ready implementation of Changes Existing regulations workable	Difficult to Assess & Modify Inflexible Existing regulations may be ineffective
APPRAISAL	Open	Obscure & Complex
INTERACTION OF PROJECT COMPONENTS	Slight	extreme

FIG. 1 CONVENTIONAL vs SUPER SCALE PROJECTS

involving the expenditure of up to one hundred million dollars, there is a recent trend to rapid development of multibillion dollar projects. Within these projects, there are profound differences in the range of impact on geological and environmental conditions. Management and engineering of such projects are also substantially different and the regulation of the development poses new problems for government.

Superscale projects require well planned advance studies and evaluation of all factors involved in the project. The economics of the development may be threatened if this is not done.

The development of tarsand is an outstanding example of the care with which superscale developments must be treated. As demonstrated in Figure 2, tarsand is a unique material affected by a variety of factors. There are several geotechnical fields which must be intensely considered in tarsand development. These include geology, ground water, material properties, climate, and other site resources. Each of these areas interact and has an impact on the important development components consisting of:

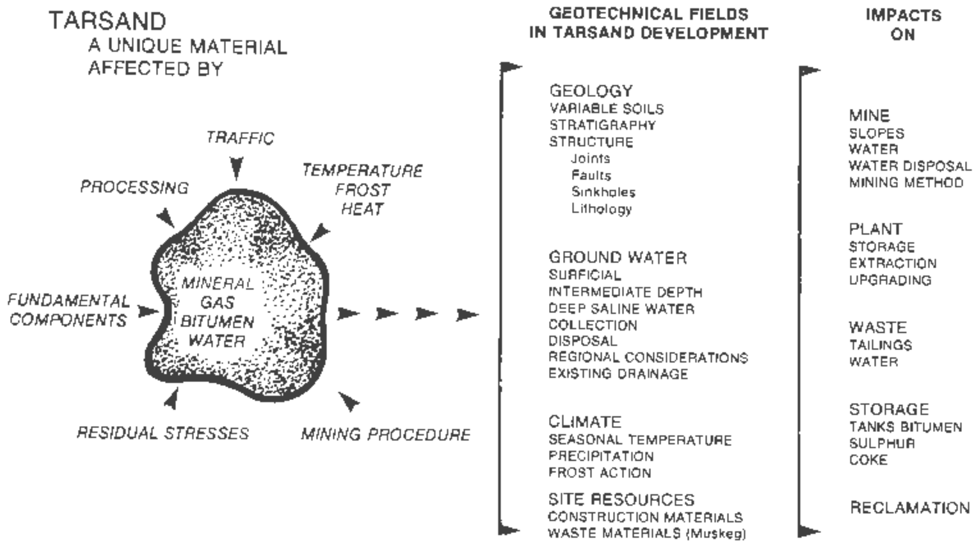


FIG. 2 FACTS OF SITE GEOTECHNOLOGY

1. the mine,
2. the plant site,
3. waste disposal,
4. product storage, and
5. site reclamation.

The management of this complex relationship provides a significant challenge to the areas of engineering, industry, and government.

3. SHORTCOMINGS OF THE PRESENT DEVELOPMENT FORMAT

Superscale development of natural resources in Alberta is in early formative stages; as a result, there is a general lack of experience or guidelines for developers, the engineering profession, and government to follow. It is desireable that an approach to project evaluation be developed which provides the most effective and economical solution.

Although geotechnical and ground water considerations are among the primary factors controlling development of large leases, the application of appropriate expertise to provide baseline information, and effective evaluation of that information, is occasionally overlooked.

In considering the nature of the baseline information, it is important to delineate the significant factors in quantitative terminology. Whereas the implications of this aspect of development are of limited impact in conventional projects, the problem may be profound in superscale projects as indicated in Figure 1.

4. SIGNIFICANCE OF DEVELOPMENT APPROACH

Concerns relating to development format manifest themselves in a dominant fashion in the field of earth science engineering. As outlined in Figure 2, the various natural conditions will have an impact on each element of the project and vice versa. The excavation of a large scale open pit mine may alter the regional ground water pattern to a significant degree. This alteration may be further compounded by the development of vast tailing retention areas. Failure to optimize both mining and tailing disposal development elements will result in a less than optimum economic development.

It is the goal of industry to extract resources with a minimum of expenditure. This is most likely to be accomplished if mining, plant, and waste disposal schemes are selected which interact most favourably with natural conditions. Failure to recognize and work with natural conditions may give rise to adverse circumstance in the planning and execution of:

1. mine layout,
2. plant location,
3. excavation equipment, and
4. waste disposal (method and location).

The optimum size, orientation, and depth of a mine will depend on the natural conditions present on site and on the equipment selected to mine the ore. If the site is highly structured, its effect becomes more significant and mining operations will work with or against nature depending on the layout selected. Mining operations are expensive and therefore, recognition of the above interacting factors is important if optimization is to be achieved.

Optimum selection of mine equipment is a function of tarsand properties, stable slope configuration, conditions in the mine, layout required for optimum performance of that equipment, and finally the efficiency of the equipment itself. Appreciation of these interactions is important because equipment costs are significant on superscale projects and lead times to secure a replacement of unique equipment are long.

Excavation of the Athabasca tarsands will intercept aquifers at various elevations. The pressures and quantities of water involved are critically important to slope stability and mine pit conditions. There are areas where extensive depressurization of the aquifers is necessary before mining can take place. Consistent with this depressurization is the need to dispose of substantial quantities of water that may be saline. The appropriate method of handling this water involves an evaluation of both local and regional hydrogeological conditions.

Large waste disposal areas must be developed to accommodate the large quantities of tailings which are produced by tarsand processing. The development of these tailings ponds has an impact on local hydrogeological conditions which must be evaluated. The failure of a disposal area has the potential of discharging silts and sludges which may ultimately pollute the entire downstream drainage system. This could have far reaching effects since the tailings are inherently toxic. Notwithstanding the environmental aspect of a dam or mine pit slope failure, there is the critical consideration of project economy. Either failure could result in plant shut down with losses measured in multiples of millions of dollars. This aspect has an effect on the economy of the project itself and on the surrounding community. It is desireable, therefore, that a systematic and effective method of appraising project components to be implemented.

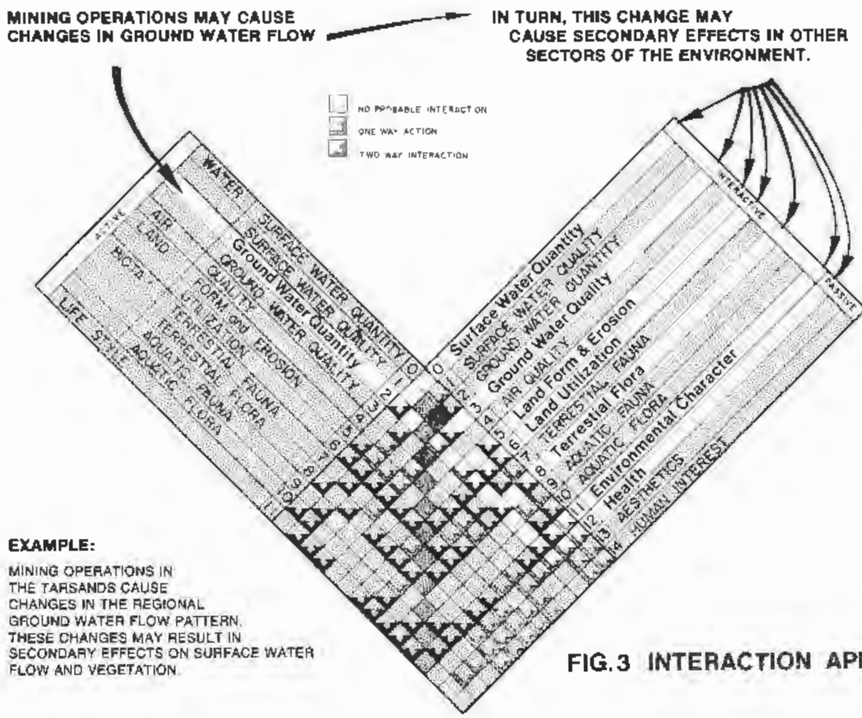
5. A METHOD OF APPRAISAL

An appropriate method of appraising geotechnical and environmental circumstances must consider all aspects of project development. The following suggested methods apply to superscale projects.

Secondary aspects, although still important, include indirect changes in the biosphere. These changes can only be predicted after the physical environment is understood.

The evaluation process, which of necessity is multidisciplinary in nature, should encompass all natural conditions and planned activities. The purpose is to appraise the effect of each factor on the site development and to adjust the development so that an optimum relationship exists.

All areas should be investigated bearing in mind the ultimate goal of reclamation.



EXAMPLE:

MINING OPERATIONS IN THE TARSANDS CAUSE CHANGES IN THE REGIONAL GROUND WATER FLOW PATTERN. THESE CHANGES MAY RESULT IN SECONDARY EFFECTS ON SURFACE WATER FLOW AND VEGETATION.

FIG. 3 INTERACTION APPRAISAL

6. THE OBSERVATIONAL APPROACH

Early this century, one of the most outstanding civil engineers, Dr. Karl Terzaghi, evolved what is now referred to as "the Observational Approach". This approach was originally designated by Terzaghi as pertinent to geotechnical matters. It is equally applicable to environmental matters and to large scale natural resource development. Details are outlined in Figure 4.

The observational approach involves formulation of an initial project concept on the basis of experience and judgement. Work proceeds on this basis, but research is initiated to explore areas of uncertainty (laboratory testing, analysis, and field exploration) and development proceeds under close observation. As information and experience develop, the initial concepts can be reappraised and design modifications made as necessary.

The observational approach is an iterative one that permits optimum development. It is superior to methods that attempt to cover every aspect of development in detail before start up but that have no mechanism to observe and learn from operations in progress. The most effective area in which to apply this technique is in the field of the earth sciences, where a total understanding of site condi-

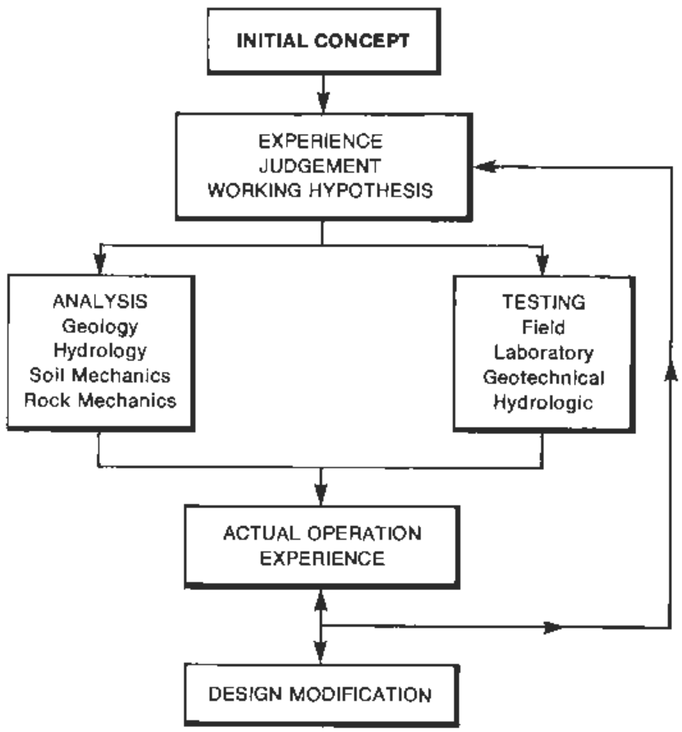


FIG. 4 OBSERVATIONAL APPROACH

tions or interaction is never possible. Major resource development falls within this category.

Collection and interpretation of pertinent baseline data is the first step. The initial data need not be all inclusive but only sufficient to support or lay the basis for amendment of the initial project concept. Part of the underlying philosophy in implementing the observational approach is the development of a working hypothesis does not require design for worst possible conditions provided the flexibility of recycling information into the stream of experience and judgement remains possible.

One of the potential downfalls in the management of superscale projects is that this flexibility may not exist. Flexibility must be present if the project is to benefit from the observational approach.

7. COORDINATION OF EFFORTS

In order to achieve full benefit of the observational apporach, it is necessary that three primary conditions prevail. These conditions are:

1. functional coordination of all development aspects,

2. the interaction of engineering efforts with the development industry and government agencies, and
3. continuity of effort from planning through construction and operation.

The functional coordination of civil engineering aspects is shown diagramatically in Figure 5. All phases of site development should be subjected to similar scrutiny if impacts are to be fully appreciated and effective optimization with nature achieved.

The interaction of the engineering, industry, and government functions is demonstrated in Figure 6. An open relationship must exist between these bodies in order to optimize the proposed development. The tasks of each segment are shown. Some areas of industry tacitly include an engineering function as well. It is this latter area which has led to the implementation of secrecy. However, with the exception of highly specialized and patentable processing techniques, it is suggested that secrecy in regard to geotechnical and environmental matters which impact on a regional basis is inappropriate. The functional coordination as demonstrated in Figure 5 is oriented dominantly to geotechnical and earth sciences areas.

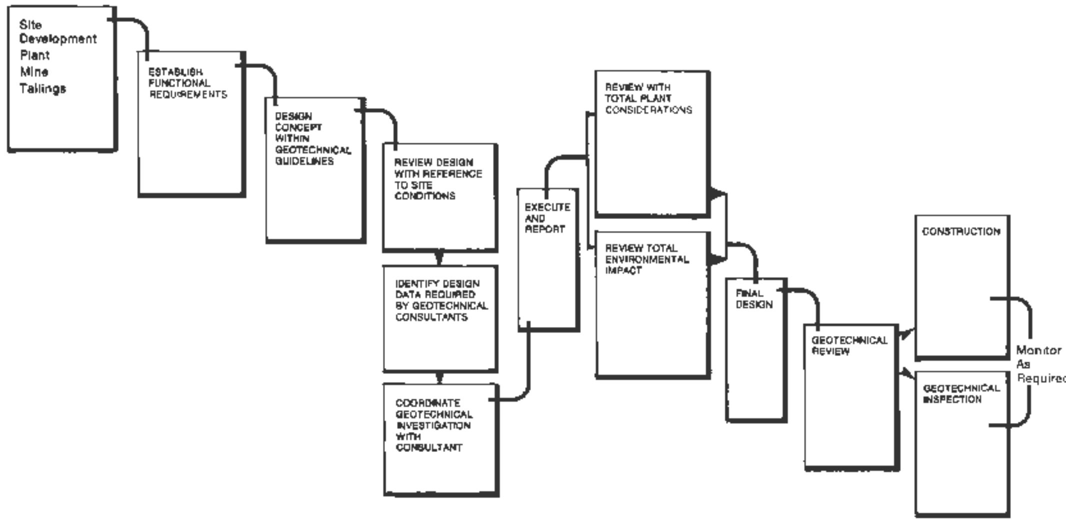


FIG. 5 FUNCTIONAL COORDINATION

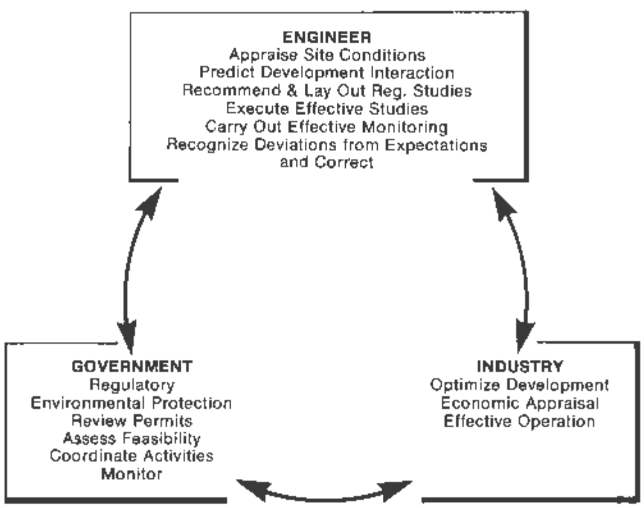


FIG. 6 DEVELOPMENT INTERACTION

Figures 4, 5 and 6 demonstrate that an interaction and iterative procedure is necessary in order to achieve the desired goals and this can only be achieved by the cooperative use of multidisciplinary teams. The need for continuity in the review process is obvious.

The function of government within the areas of development deserves special mention. Regulations on environmental protection are a natural function. Less obvious is the granting of permits, which is necessary because of the massive socio-economic impacts of these superscale projects. Continual monitoring of the environment and reassessment of development assumptions should also be carried out in close cooperation with government agencies.

8. RECOMMENDATIONS

The foregoing text suggests that the relationship between project development and natural environmental conditions must be effectively integrated on a logical basis in order to provide optimum development. The relationships have been shown and a method has been outlined for achieving this desirable goal.

Emphasis has been placed on the collection of rational baseline data necessary for appraisal leading to optimum development. The foregoing suggestions relate primarily to superscale projects which differ significantly from conventional projects. The significant features related to superscale projects are illustrated in Figure 1. Useful conclusions may be drawn from this figure in regard to necessary engineering activities. The following recommendations are suggested in order to optimize development of our natural resources.

1. A policy of recognizing and working with natural conditions can provide the basis for selecting optimum development schemes and should be implemented.

2. Appropriate studies to identify and appraise these natural conditions should be carried out prior to, during, and subsequent to development.

3. There should be effective coordination and integration of various site studies.

4. Project development should include design that recognizes environmental protection, reclamation and long term geotechnical effects.

5. Optimum use of Canadian professional capability should be encouraged and integrated to its maximum extent

so that our human resources may grow as our natural resources decline.

6. Licensing agencies should issue guidelines for proper development, review applications, and then monitor work in progress. Submissions for development should be reviewed on an economic basis and also on a technical basis that considers pertinent geotechnical and environmental factors. Several public hearings would be required to accomplish this.

7. The government and its various regulatory agencies should encourage coordinated cooperative studies of resource potential and development proposals. Information obtained could be used to monitor projects underway and also applied to future developments.

8. The observational method of approaching geotechnical and related environmental problems should be an underlying working philosophy in all developments which relate to the rearrangement of natural conditions.

It is concluded that no other development area is more amenable to the foregoing procedure than the exploitation of the Athabasca tarsands. The unique nature of this development requires a unique approach to deal with the problems. The approach recommended is a coordinated observational approach carried out by multidisciplinary teams.

The major challenge to the engineering profession at this time is the implementation of rational procedures which permit safe economical resource exploitation involving mining, extraction, waste disposal, and reclamation on a scale unique in the world. Major contributions to overall project development can be achieved by utilizing the organized philosophy of the observational approach which has been used in the field of geotechnical engineering for a number of years. Indeed, it is questionable if rational and economical projects can be accomplished without implementation of this process.

MONITORING OF THE HOGARTH PIT HIGHWALL, STEEP ROCK MINE ATIKOKAN, ONTARIO

C. O. Brawner[1]
P. F. Stacey[2]
R. Stark[3]

Abstract

The development of unstable conditions in the Hogarth Pit highwall at Steep Rock is described. The basis for the Company's decision to continue mining after institution of a detailed monitoring system is reviewed. The monitoring system included both triangulation and Electronic Distance Measuring surveying, extensometers, movement pins and seismograph and a visual guard. Mining operations were restricted to daylight hours and were revised to give the pit crews maximum protection.

Movements are described and the operational characteristics of each of the monitoring system elements are discussed.

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Mining was completed in March 1975, with more ore than anticipated being recovered.

In early May, movements started to accelerate, and the major failure occurred in the predicted toppling mode.

Finally, the paper describes the program from the Mining aspect. Reasons for choosing a monitoring approach are given, and the effects of the revised mining procedures discussed. The Company policy of continual communication with its employees played a big part in the success of the program, and the choice of pit crew members to form the visual guards had several advantages.

PART A - GEOTECHNICAL ASPECTS

INTRODUCTION

This paper describes the monitoring program developed to study movements in the highwall of the Hogarth No. 1 Zone of Steep Rock Mine at Atikokan, Ontario. As such, it forms a case history of a successful monitoring program which permitted removal of the recoverable ore reserves from that area of the mine, while at the same time providing a control safety for the operating crews.

The paper describes a case history of successful cooperation between a mining company and a consultant. This success was enhanced by the effective communication between the Company and its employees, and the use, where possible of all levels of Company personnel in the program.

It must be stressed that the aim throughout was to provide a practical solution to the practical problem of the safety of the mine crews.

HOGARTH NO. 1 ZONE

The Steep Rock ore zone is composed of a goethitic, soft iron ore horizon, which dips steeply to the west, and is overlain by ash rock, and underlain by paint rock and carbonate. The total Steep Rock Complex is divided by regional faults into three major sections, each of which originally outcropped under an arm of the Steep Rock Lake.

The section of the orebody currently mined by Steep Rock Iron Mines Ltd. is termed the Middle Arm Orebody, referring to its respective arm of the lake. This orebody is separated into several sections by faulting and folding. The major divisions from south to north are termed the Errington Zone, the Roberts Zone, and the Hogarth Zone. At the north end of the Hogarth Zone the ore terminates against the Bartley Fault, which strikes northeast-southwest and dips to the southeast at 85°. Northwest of the fault the original lakeshore was formed by a steep 300 ft. high wall comprised of a hornblende-biotite metadiorite.

The diorite contains sheared basic dykes which parallel the Fault. It also contains two well developed vertical joint sets which strike respectively parallel to and perpendicular to the major fault direction. It was these joint sets which controlled the development of the pre-mining lakeshore, and which also controlled the failure to be described. The entire failure was restricted to the diorite highwall, although the monitoring program covered the possibility of the onset of failure in the iron formation below.

The crest of the pit in the area of movement was covered by loose blocks from the excavation of a cut for the railway line to the pellet plant. This line runs approximately 150 ft. behind the movement area. The loose blocks caused initial problems in locating cracks, and subsequently added to the difficulty of traversing the area.

A plan of the Hogarth No. 1 Zone is shown in Figure 1 and an air photo of the highwall is shown in Figure 2.

ONSET OF FAILURE

A crack behind the crest of the diorite highwall was first noticed in October 1973 and became the subject of weekly observation. Further signs of instability developed in April, 1974 when an attempt was made to access the foot of the diorite wall at the 975 level on a berm containing the Bartley Fault outcrop. This level approximates to the ore outcrop below the original lake floor. A small blast on the hangingwall side of the berm resulted in the failure of a small wedge of diorite bounded by vertical joints. Talus spilled over the ore berm and down the iron formation face below.

As a result of the crack on the crest opening and extending, the Company installed a simple monitoring pin system, which was read on a daily basis.

In mid-August 1974 a second crack was noticed, approximately 150 ft. behind the crest, when craters appeared in the debris from the rock cut. At this point the pin monitors were supplemented by triangulation stations and three elementary wire extensometers constructed of drill steel and clothes line. On August 20th, movements increased after a period of heavy rain. At this

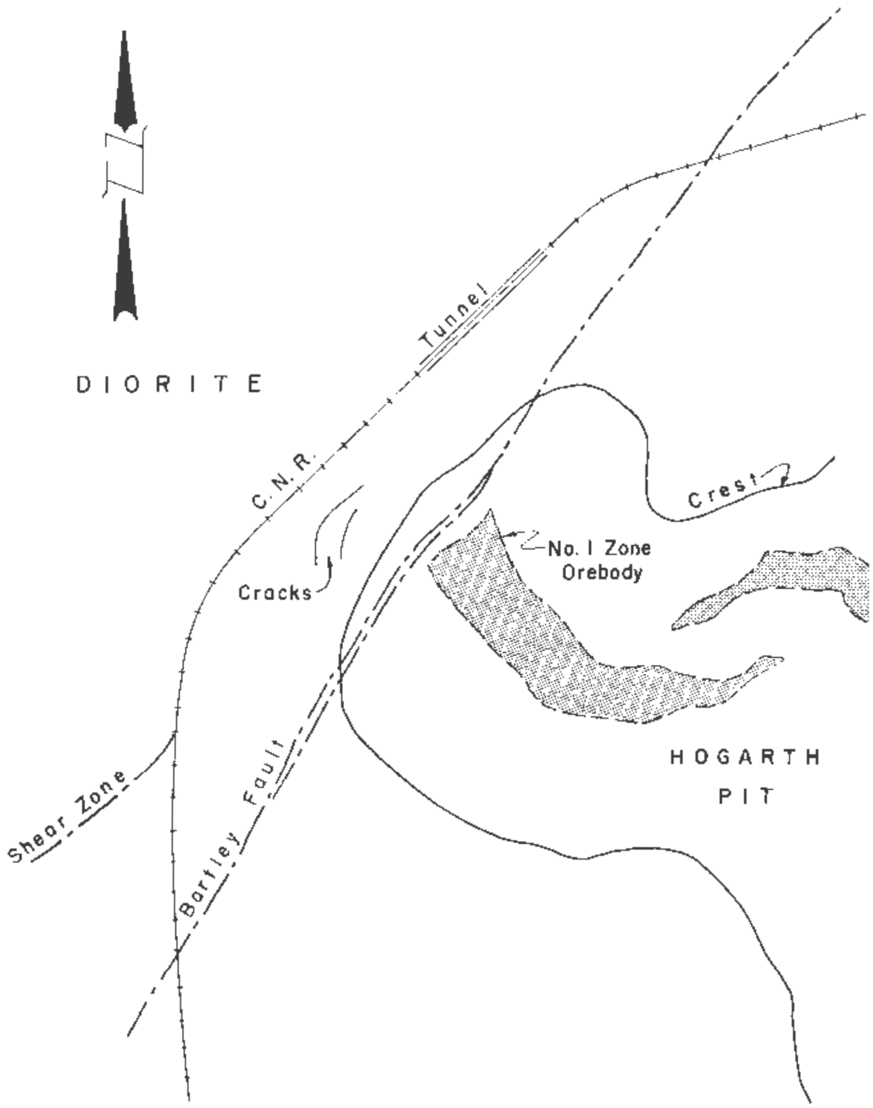

FIGURE 1 SCHEMATIC PLAN OF HOGARTH NO. 1 ZONE

point the advice of Golder Brawner & Associates was sought.

REVIEW OF ALTERNATIVES

It was apparent that a serious slope instability situation was developing. Accordingly, a temporary halt was called to mining operations in the No. 1 Zone while the movements were reviewed. A 5 mph Slow Order was placed on trains operating on the track behind the highwall.

The review indicated increasing movement involving between 200,000 and 250,000 cu. yds. of diorite. The safest procedure would have been to discontinue mining in the area. However, since the Hogarth No. 1 Zone was the only major short term source of ore, two alternatives which would allow ore production were considered. Attempts could be made to fail the slope by flooding the cracks, or blasting, after which the ore could be mined following a delay for clean-up of the debris; or a comprehensive monitoring system could be installed to provide warning of potential failure to the pit crews working below. The former course carried with it the risk of only partial failure, possibly resulting in a less stable slope which would terminate all mining activities. The latter course was felt to be a safe approach, since geologic evidence, existing monitoring data and the author's past experience, Brawner (1968, 1970, 1970, 1971, 1974) indicated a "toppling" mode of movement would be expected which would give ample warning of failure in the form of ravelling and an increasing rate of movement. This, combined with the requirement to maintain committed pellet shipments, influenced the Company's decision.

To assess whether cleft water pressures may be contributing to instability a piezometer was installed in a hole drilled from behind the crest. This indicated no water in the diorite behind the toe at the 975 ft. level. This hole remained dry until the standpipe was finally sheared by movement along the back crack.

MONITORING SYSTEM

When the decision was made to go ahead with a mining and monitoring approach, the monitoring system established by Steep Rock staff was expanded to include seven further pin monitors (Figure 3), seven further wire extensometers mounted on tripods (Figure 4) rather than drill steels, and an increased number of triangulation stations. The wire extensometers were tensioned with 40 lb. weights so wind, contact with bushes, etc. would not influence readings.

The results of the monitoring of the 14 triangulation stations indicated a "toppling" mode of failure, with essentially horizontal movement of the crest towards the east-northeast, and negligible movement at the diorite or iron formation toes. Daily triangulation by the pit survey crew was both slow and impractical as a long term program,

FIGURE 2 AERIAL PHOTOGRAPH OF HOGARTH NO.1 ZONE HIGHWALL - SEPTEMBER 1974
(The location of the main cracks is indicated. The arrow shows the site of the guard shack.)

FIGURE 3 MEASURING PIN MONITOR ACROSS BACK CRACK

FIGURE 4 EXTENSOMETER TRIPOD WITH LIMIT SWITCH

it was decided to use a laser Electronic Distance Measuring device (EDM) (Figure 5) shooting across the pit along the approximate line of movement. It should be noted that even if the line of sight is off line of the direction of movement by as much as 45° the scale of the time-movement graph is only reduced by about 30 per cent and warning of impending instability is still readily available. Initially 10 EDM reflector prisms were installed on the crest, on the 975 ft. berm and on the Iron Formation face. Figure 6 shows typical EDM data which confirmed the toppling mechanism. This general system remained in operation until completion of mining, with three further prisms being added on the 712 bench. The EDM monitoring was supplemented by periodic triangular surveys to obtain 3-dimensional vectors of movement.

Beside the overall "remote" EDM - triangulation monitoring system, movement of major individual blocks within the mass was monitored on a daily basis by the network of extensometers. Limit switches connected to a warning system involving sirens and flashing lights were attached to strategic lines, and a light on the tripod indicated that power was on and that the switch had not been operated.

At a later date, at the request of second opinion consultants employed by the Ontario Department of Mines, twelve extensometers equipped with limit switches and a

FIGURE 5 ELECTRONIC DISTANCE MEASURING DEVICE (EDM.)

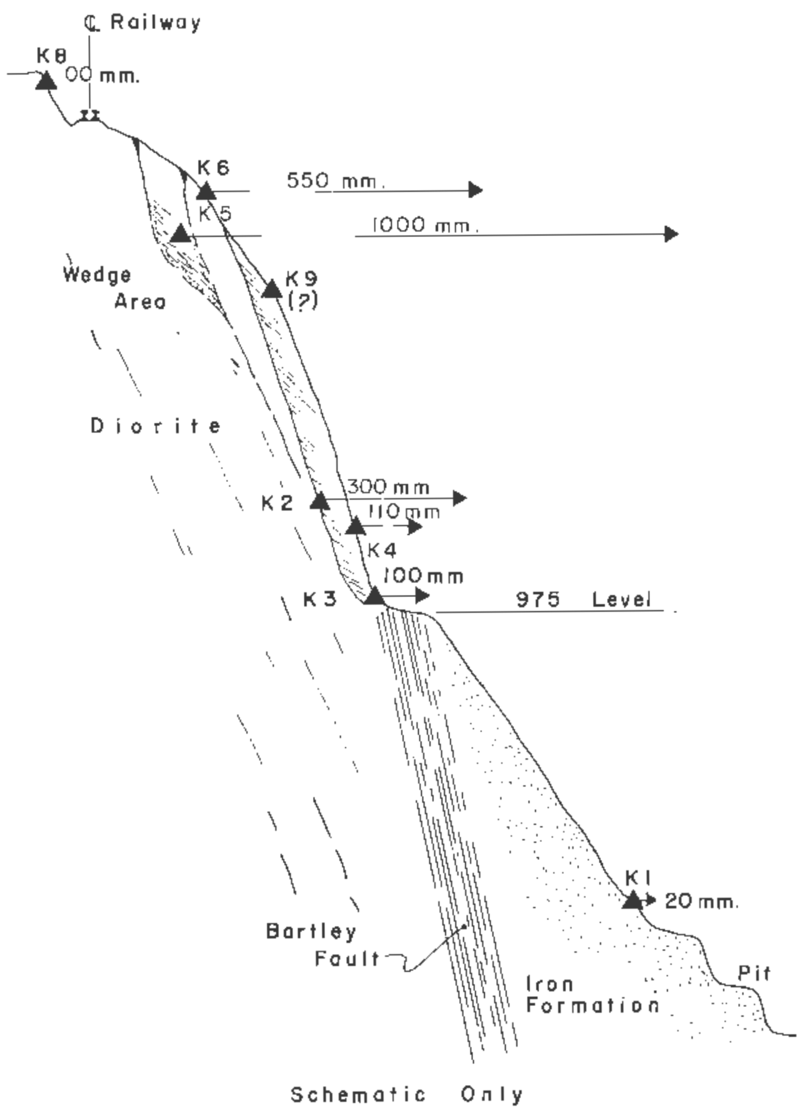

FIGURE 6 RELATIVE EDM MONITOR DISPLACEMENT SEPTEMBER TO DECEMBER 1975

micro-seismograph were added. Throughout the mining operation the limit switches were set for a predetermined movement and were adjusted daily. Prior to the onset of winter conditions this tolerance was 1/2 inch.

During the initial stages of the monitoring program the movements on the two major cracks were monitored at seven individual points using a three pin system. However, by mid-winter the movement at several points had made this operation too dangerous to continue.

An extremely important element of the monitoring system was a 24 hour visual guard by pit crew members. A heated shack was installed on the 1050 bench of the hangingwall. Two guards per shift had a complete view of the highwall as well as the warning lights and sirens. Radio and visual contact was maintained with the crews working in the pit below. This included an hourly radio check with each drill and shovel in the area. All events noted by the guards were recorded in a logbook which was initialled daily by the Mine Superintendent and by the Golder Brawner representative. Based on past experience it was envisioned that any major failure would be preceded by ravelling of rock blocks too small to be detected by the monitoring system, the guards were intended as the first line of warning to the pit crews.

The Kinemetrics VR-1/SS-1 seismograph system requested by the Department of Mines was installed in

December. The seismograph head was installed in the centre of the failure area, and the remote drum recorder was located in the guard shack.

The daily monitor reading and compilation were performed by a Golder Brawner field Engineer, who reported the results to the Steep Rock Senior Engineer and to Vancouver by Telex. Movement plots were posted daily in the Mine Dry Room.

In summary, the total system installed covered all levels of monitoring from the EDM direct measurement system and triangulation vectorial system, through the extensometers, which monitored major blocks within the mass, down to the visual watch for the failure of small pieces. Operational characteristics of each of the monitoring systems and the selection and psychological effect of the guards are discussed later.

MINING SEQUENCE

The monitoring system was complemented by a planned mining sequence. At the toe of the Highwall a 40 ft. wide berm with rock pile was left on the 712 level. This berm was intended to catch only the initial ravelling since a more desireable, wider berm would have greatly reduced ore reserves in the Zone. Below the 750 level, overall slopes in the Iron Formation were designed at 40° instead of the usual 42-1/2°.

Prior to the mining of each bench, a 100 ft. wide slot was drilled, blasted and mined along the foot - or hangingwall contact of the ore up to the proposed Highwall toe. At the same time a 100 ft. wide zone was drilled off along the base of the Highwall, perpendicular to the slot. The holes in the zone had a maximum loading of 1,200 lb. of explosive per delay and were fired into the slot. This approach was aimed at reducing the exposure of men and machines by relieving any stress in the toe and/or initiating any unheralded failure while the operating equipment was still well back from the Highwall. For want of a better word the procedure was termed "De-stressing".

At the same time a "Green line" was defined 300 ft. from the Highwall toe on each bench. This line, which was indicated in the field, became highly revered by the mining crews. It formed an inner boundary for all nighttime and bad weather operations. Further, no operations were permitted within the Green line for 48 hours after a de-stressing blast and for 12 hours after a production blast. It is interesting to note that when the slope failed, little, if any, material passed this line.

It was recognized that time was an important factor. Accordingly, a series of six high-powered lights were installed to illuminate the entire face, thereby providing conditions under which the initial daytime operations could be safely extended to a 24 hour basis.

This move was not accepted by the Union, and operations were restricted to good visibility conditions during daylight.

MOVEMENTS IN THE PERIOD SEPTEMBER - JANUARY

The "toppling" movement pattern established in early September continued until January 9th. Movement was restricted to the diorite above the ore berm with the two sets of vertical cracks opening up and propagating downwards. A small "wedge" on the east side of the main mass showed more rapid movement, and on several occasions loose material was scaled from this area by Steep Rock scaling crews.

The extensometer and EDM data showed that there were periods of constant movement rates separated by periods of more accelerated movement. The acceleration of movement appeared to correlate with heavy rainfall, there being a time lag in the order of 1 day. It is not certain whether blasting also had a contributing effect.

The triangulation surveys showed an interesting vectorial picture. The material between the face and the original "front" crack showed a horizontal:vertical movement ratio in excess of 3:1, with no appreciable movement at the toe, indicating toppling of this block. The block between the front and back cracks showed a horizontal:vertical movement ratio of approximately 1:1, indicating an outward slumping behind the front block. Monitoring of survey stations on the 975 Ore Berm and Iron Formation face below indicated no movement.

All major movement continued to be on an east-northeasterly azimuth, i.e. in the general direction of the EDM unit.

WINTER MONITORING

On January 9th prolonged sub-freezing winter conditions, commenced with a blizzard. These conditions generally persisted until after March 10th when mining was completed.

The sub-freezing conditions required variations in the monitoring techniques. Rapid temperature fluctuations from +30°F to -20°F experienced over a few hours would result in rapid contraction of the extensometer wires by as much as 7/8 inch, with resultant tripping of the alarm system. The maximum daily permissible movement on an extensometer limit switch was therefore increased to 1 inch to accommodate this phenomenon.

During extremely cold conditions the EDM readings across the pit were affected by the temperature gradients developed in the air. As a result there could be a difference of 2 cm. between a reading made in the early morning, and one taken in the late afternoon, particularly on still, clear days. This problem was overcome partially by referencing measurements to a stable station on the crest of the railway cut.

Blizzards or poor visibility resulted in the temporary cessation of operations, while the onset of thawing conditions resulted in a mandatory withdrawal behind the "Green Line".

Snow made negotiation of the boulders and cracks on top of the movement area a treacherous operation. Accordingly the cross-crack pin measurements were halted, and, by the time the snow had melted, many of the cracks were too wide to measure safely.

MOVEMENTS JANUARY - MARCH

Contemporaneously with the onset of permanent subfreezing conditions major movement essentially ceased. Failure was still restricted to the area in front of the original back crack, and the main mass was now divided into a series of slabs by the opening of vertical fractures.

Although daily movement dectectable by the extensometers and EDM had ceased, and variations in the triangulation survey data were generally within the limits of accuracy, there was evidence that the slope was still active. Particularly in cold weather water vapour was emitted from the major cracks. During the night shift when no equipment was operating in the zone, occasional small signatures appeared on the seismograph trace. In the daytime however, seismic indications of this type were masked by mine equipment signatures so that warning was not available from the seismic instrumentation. This confirms the experience of Kennedy (1972) on the practical limitations of the seismic monitoring technique.

Much of the movement in this period was restricted to a slight opening of the forward cracks on the face, and later to the occasional loosening of debris on the slope by freeze-thaw action.

COMPLETION OF MINING

The planned ultimate depth of the No. 1 Zone at the 600 level was reached in late February. With movement on the Highwall negligible, and horizontal drain holes drilled into the face indicating no water within 50 ft., circular arc analyses were performed on the Iron Formation to assess the effect of taking an additional lift. The analyses showed only a minor influence on the Factor of Safety. Accordingly an additional 20 ft. bench was mined. At the completion of this level there was just sufficient width of ore exposed in the floor to make an additional 20 foot deep scram or rob cut feasible.

With continuous monitoring the scram was drilled up to the toe of the previous bench and fired in one blast using a maximum loading of 1,200 lb/delay. An intensified monitoring system involving daily triangulation of 4 critical stations, twice daily extensometer readings and three daily EDM surveys was instituted, and, after a delay of 48 hours during which no movement was detected, mining recommenced.

It was planned to drill a series of exploration holes from the floor of the Hogarth Zone on the footwall side; the original plan to mine in at 40° slopes on all sides and double-bench on the way out for maximum ore recovery was modified to include double-benching on the hangingwall and highwall faces only.

Mining of the scram cut to the 560 level was completed on March 10th. In total, 970,000 tons of ore were recovered from the Hogarth No. 1 Zone compared with an original estimate of 933,000 tons, in spite of an ore loss on the footwall side. Part of this gain was due to the mining of the two additional 20 ft. benches.

MOVEMENTS MARCH - FAILURE

Two levels of highwall monitoring were established after completion of mining, one to cover drilling in the No. 1 Zone, and a second level aimed at monitoring for movements which could affect the safety of the railway.

Drilling started immediately after completion of mining operations, but shortly thereafter the onset of permanent thawing conditions, with resultant ravelling and small failures in the ash rock led to the temporary removal of the drill to a safer location.

By early May weekly reading of the extensometers and monthly triangulation of 12 selected survey points by Steep Rock personnel indicated a marked increase in movement on the Highwall. This coincided with a heavy rainfall. Increased activity was also noted on the seismograph. The movement followed the original pattern, with the vertical fractures opening from the top in a toppling mode, and the back block slumping behind the front block.

On May 21st a slab from the front of the main mass of about a hundred cu.yds. termed the Maple Leaf after the shape of its bounding crack, failed. Most of the material from this failure was contained on the 975 Ore Berm, none of it reaching the pit floor. This failure was preceded by minor ravelling, and was followed by accelerating movement, accompanied by increased seismic activity and constant ravelling particularly from the wedge area.

On June 10th the Wedge failed involving several hundred cu.yds., followed on June 11th by a further section from the front of the main mass. By this time both the 975 Ore and the catch berm at the 712 level were choked with debris, and large amounts of material started to reach the pit floor.

As movement continued both sets of joints opened to divide the mass into a series of vertical columns. The pattern still remained that the front columns toppled forwards, while the back columns tipped forward and slumped. Most of the extensometers and EDM targets were destroyed in the period June 10th to 11th, but immediately prior to this date extensometers and laser targets in the centre of the mass were recording movements in excess of 1 ft./day (Figure 7). On June 23rd, 5 hours before failure, a triangulation point on the front of the old front crack was measured to have had a cumulative movement of 30.5 feet horizontally and 4.8 feet vertically since August 1974.

Heavy rainfall on June 20th induced renewed heavy activity, and by June 23rd the face was showing signs of major distress. The remaining mass was keyed in by a block on the west side. This was seen to be cracking badly, and ravelling was general over the whole face. At 9:00 p.m. the back mass slumped, pushing out the front columns. Finally at 9:15 p.m. the remaining material failed in three portions commencing in the east. The largest, westernmost column, estimated at about 50,000 cu.yds. failed in a typical toppling mode, whereas the remainder broke up and ravelled. See Figure 8 for failure sequence photos.

The results during the final days prior to failure indicated that even if failure had occurred while the mining operations were still in progress, the monitoring systems would have given ample warning. Thus, the decision to proceed with the mining under a controlled monitoring system was justified. Also, the instrumentation installed

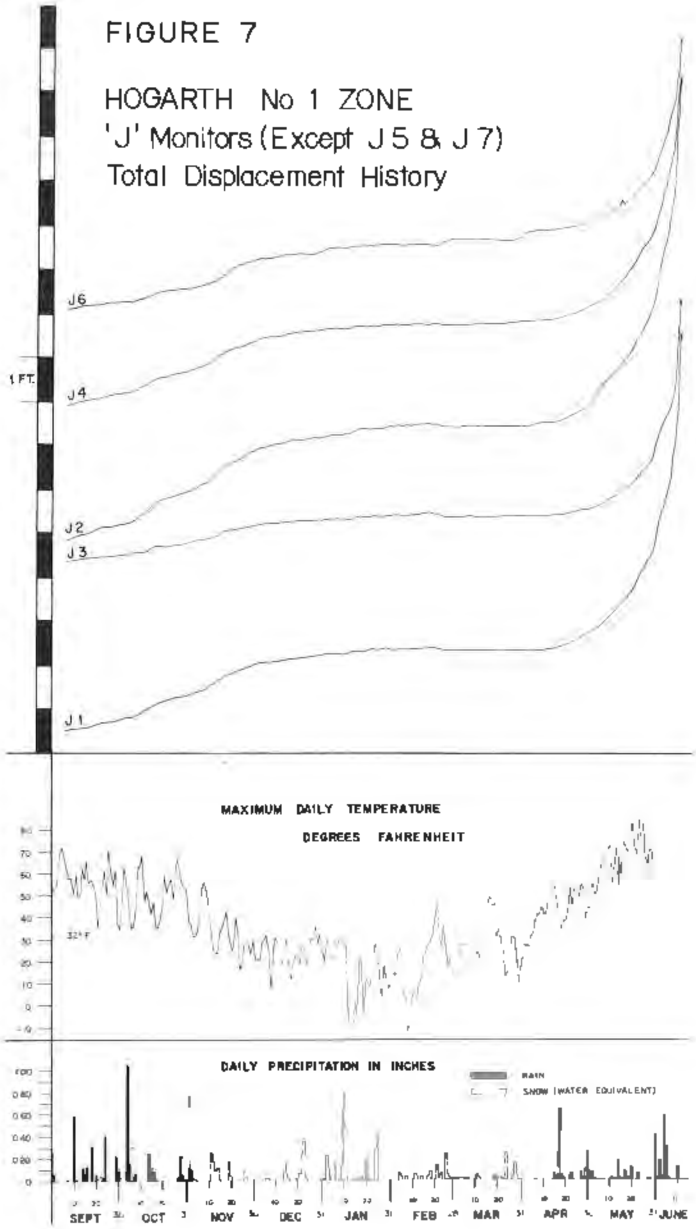

FIGURE 7 HOGARTH NO. 1 ZONE - 'J' MONITORS (EXCEPT J5 & J7) TOTAL DISPLACEMENT HISTORY

a) 9:00 p.m.

b) 9:15 p.m.

c) 9:15 p.m.

d) Morning June 24th

FIGURE 8 MAIN FAILURE JUNE 23rd 1975

during August proved totally adequate and the time delay and mine shutdown period required for additional instrumentation installation proved to be unwarranted.

Finally, the choice of a 300 ft. Green line was upheld, since little, if any, material extended beyond the marks remaining on the benches in the Iron Formation.

PART B - MINING ASPECTS

The decision to strip and mine the No. 1 Zone of the Hogarth Pit was not made until August, 1973. An expansion into the No. 1 Zone was not included in the original planning because the stripping ratio was fairly high and the ore zone was not as well explored as the other 3 zones of the pit. In addition, a hangingwall failure immediately adjacent to the highwall had terminated mining fourteen years previously, thereby making stripping much more difficult.

However, because of a shortfall in net excavation in 1972, the decision was made to strip and mine the area as part of the Hogarth pit design. Stripping in the No. 1 Zone began in September, 1973.

A crack had been detected as early as October, 1973 along the north wall of the No. 1 Zone, Hogarth Pit, outside the planned pit limits. Movement along the crack was not detected however, until April of 1974. Movement was of a minor nature and was not expected to have any adverse effects on the stripping and mining operation. In early August, when the stripping programme was almost completed and the mining programme was just beginning, movement accelerated. The ore resulting from the mining programme was required within two months to maintain a continual supply of ore to the pellet plant. In achieving this goal, the safety of men and equipment working in the area must be assured.

Steep Rock Iron Mines personnel have had considerable experience in dealing with wall stability problems, mainly associated with the hangingwall ashrock and footwall paint rock. However, because of inexperience in the diorite, and the tight mining schedule at that time, the decision was made to consult an engineering group who had had experience in this type of stability problem. Golder Brawner & Associates were approached because of their diversified mine stability experience and professional reputation throughout the world.

Golder personnel suggested two ways of dealing with the problem. These were:

a) to induce a failure, clean it up and resume mining, and

b) to continue to mine, after instituting a detailed monitoring system.

Two methods of inducing failure were considered. The first was to drill a series of diagonal holes under the area and blast them, thereby unloading the slope. The second was to flood the cracks with water.

The system of drilling diagonal holes was not considered practical under the circumstances. The holes would be expensive to drill and considerable time would be required to complete the programme. Of a greater importance was the fact that the cracks were opening continuously, and it would be extremely difficult, if not impossible, to complete the holes.

If the alternate method of flooding the area was used, approximately one month would be required to remove the failed material from the pit bottom. Costs over and above the extra cost to remove the failed material would be incurred in handling the large blocks of rock normally associated with wall failure, and in cleaning up the water afterwards.

If the failure did not occur when the area was flooded, it was reasonable to assume that the wall would be stable and would probably not fail for some time. However, if only a partial failure occurred, the situation would be worsened rather than solved.

Because of the unknown factors associated with failure induction, and the assurance that advance warning of any impending failure would be given, the decision was made to monitor and mine. There were limitations on this solution, as mining would be governed on the basis of monitoring results and daylight hours only. It was estimated that mining in this manner would still provide adequate ore to assure us of a continual supply of ore to the pellet plant. Should movement increase and a failure occur, it was anticipated that the clean-up could be completed and mining resumed with little or no disruption in the pellet plant operation.

The decision to monitor and mine was made on the basis that it was the most reliable system which could be adopted to achieve a continual ore supply to the plant.

Considering the economics involved, the decision was also the best alternative. The actual cost of monitoring the wall including a resident Golder representative and two guards full time was $163,300.00. The breakdown of this total is $23,700.00 for diamond drilling, $63,300.00 for consultant fees and $76,300.00 for operating costs including the cost of monitor installations, guards, etc. This cost was well below the estimated cost of the other alternatives.

RESTRICTION OF MINING SEQUENCE

The mining sequence as recommended by Golder Brawner had some adverse affects on the mining programme in one zone.

First of all, the slot type approach limited us to one shovel earlier in the programme than we had anticipated.

Because of the confined area, a two shovel operation was not very effective under normal circumstances. With the slot type approach it eliminated any chance to operate two shovels effectively.

The destressing blast limit of 1,200 lbs. per delay necessitated in a very low powder factor for the material against the wall. As a result, the material was not well broken and tough digging conditions were experienced.

The 48 hour waiting period after the destressing blast delayed the mining programme to some extent, particularly near the end of the programme. On the wider benches sufficient material could be left outside the 300 ft. limit to accommodate the shovel during these periods. However when the benches became narrower, this flexibility was lost.

It was obvious to us after the mining programme was completed, that the mining sequences as recommended by Golder Brawner were not only justified but well thought out. Although it delayed the mining programme to some extent, we feel that it was a major factor contributing to the successful completion of the programme.

RESTRICTION ON NIGHTTIME OPERATION

The restriction of daylight operation only was initiated originally because the visual monitoring could not be carried out at night time. It was recognized, however, that we were dealing with a moving mass and that failure would occur with time. It was the opinion of all the consulting people that we should complete the mining as quickly as possible so that we would be out of the area in the shortest possible time. The daylight hours only restriction would result in lengthening the time involved in completing the program and also result in many lost equipment hours and an overall inefficient mining operation. Steep Rock personnel installed a system of metal hilite lamps to illuminate the wall so that mining could be carried out on a twenty-four hour basis. Six 1500 watt lamps were used and were positioned such that the entire wall would be illuminated. Golder personnel inspected the area after installation and agreed that the lighting was adequate to monitor the wall visually and to operate on a twenty-four hour basis.

The plan was explained to a delegation from the union who visited the wall at night to inspect the lighting. They decided to take the request back to the members, and the majority of the membership voted not to work in the area at night time. As a result, mining inside the Green Line (300 foot limit) was carried out during day-light hours only to the end of the program.

Of all the restrictions imposed, this decision had the greatest economical impact. As estimate of 600,000 cubic yards of net excavation was lost throughout the operation because of the inefficient use of equipment resulting from the restricted operation in the Zone. It was necessary to use the largest shovel on the property in the No. 1 Zone because of its dependability and its elevated cab. Consequently, the shovel could not be used elsewhere and was idle during nighttime, i.e. for 15 hours per day. In terms of direct cost dollars, it is difficult to put a value on it. Indirect costs resulting from the 600,000 cubic yards shortfall, would be significantly higher than the direct costs, when the long term ore supply outlook is considered.

UNION INVOLVEMENT

Steep Rock Iron Mines Limited have established a policy over the years of keeping all employees informed continually of any changes or decisions which affect them directly or indirectly. This information is presented to them on the job by supervisors, by technical people, and through their union.

Because of the seriousness of the situation, an even greater emphasis was placed on following this policy in all aspects of the No. 1 Zone problem. When the wall movement initially accelerated in August 1974, and mining in the area was temporarily suspended, the union representatives were notified and the situation was explained to them.

Golder Associates completed their evaluation and the monitoring system was installed. When the recommendations were all met, the union executive attended a briefing session with the company and Golder's representatives. Subsequently, all open pit employees on each shift were briefed on the monitoring and mining system. One union member was selected daily as a guard to visually inspect the wall. At the request of the union, a second guard was added to each shift to reduce the monotony and to improve the visual inspection by allowing the guards to spell each other off after various time periods.

To further inform the employees, the monitor results were posted daily on the bulletin board in the mine dry area. Comments were also added to explain any abnormal movement recorded. The top of the wall was also inspected daily by the scalers and their comments were posted on a blackboard in the dry area.

Considering all aspects of the No. 1 Zone problem and subsequent solutions, we are convinced that the policy of keeping all employees informed is well worthwhile. We feel that our emphasis on communications has paid off with better employee relations, better union-management relations, and a greater desire on the part of most employees to complete the job successfully.

CHOICE OF PERSONNEL AS WALL GUARDS

The decision to use hourly rated employees as guards on the wall had an interesting psychological effect on those working in the area.

As mentioned previously, two guards were selected from each shift. However, the guard duties were rotated so that most employees working in the area took their turn. Only one guard was changed each day so that at least one of the guards had experience at all times. The guards were selected at the discretion of the operating foremen.

There were three advantages in using hourly rated personnel and rotating them. First of all, monotony was not a problem because of the rotation. Secondly, we created a situation where hourly employees were taking responsibility, to some extent, for the safety of their fellow workers. Thirdly, most employees had the opportunity to experience the difficulties and responsibilities of those guarding. This gave each of them a better appreciation of what was going on.

Steep Rock Management was concerned about factors which could affect the efficiency of the guards. Most of the guarding was required in winter weather, and there was a tendency to keep the window closed, thereby eliminating sound as a monitoring tool. A light was required in the shack, but it had a tendency to cut down on the effectiveness of visual monitoring. It also allowed the guards to read, which had an adverse affect because their eyes had to adjust to the distance change. The installation in the guard shack of the seismograph recorder had both positive and negative effects on the guards. The fact that it was located there, gave the men more confidence that any indication of movement would be noticed immediately. However, there was a tendency for the guards to watch the recorder rather than the wall, thus reducing the visual monitoring. The seismograph was not meant to replace, in any way, the visual monitoring system.

ACKNOLWEDGEMENTS

The authors wish to thank the Management of Steep Rock Iron Mines Ltd., both for their support and readiness to assist throughout the program, and for their permission to publish this paper.

The advice and assistance of many members of the Golder organization is also acknowledged. In particular the perseverance of the engineers who acted as field men is gratefully appreciated.

Finally, thanks are due to many members of the Steep Rock staff for their cooperation and assistance throughout the programme.

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WRITTEN DISCUSSION OF TECHNICAL PAPERS

DISCUSSION OF: "Monitoring of the Hogarth Pit Highwall, Steep Rock Mine, Atikokan, Ontario" by C. O. Brawner, P. F. Stacey and R. Stark. Presented at the 10th Canadian Rock Mechanics Symposium, Kingston, Ontario, September 1975.

SUBMITTED BY: P. N. Calder, Associate Professor, Department of Mining Engineering, Queen's University, Kingston, Ontario, K7L 3N6.

In this paper, the authors state, "During the night shift when no equipment was operating in the zone, occasional small movement signatures appeared on the seismograph trace. In the daytime however, seismic indications of this type were masked by mine equipment signatures so that warning was not available from the seismic instrumentation. This confirms the experience of Kennedy (1972) on the practical limitations of the seismic monitoring technique."

I was involved in the monitoring of the Hogarth Pit Highwall, having been engaged for this purpose by the Ontario Ministry of Natural Resources. It was upon my recommendation that the seismograph unit was employed as a monitoring instrument, however, I was not involved in the operation of the seismograph or the interpretation of the data.

My main purpose in recommending the seismograph was to provide a continuous sensitive warning system. The surface extensometers would only trigger the warning system following a total pre-set movement. In my opinion, if an amount of movement less than the magnitude required to trigger the warning system occured suddenly, the mass could fail. For example, if within a fifteen minute period the mass moved 1/4 of an inch, this would be very significant, but would not necessarily trigger the system.

Regarding the masking of the seismic noise generated by the moving mass, in the paper referred to (Reference 6) by Kennedy, he states, "With experience, it is possible to recognize with ease the cultural noise generated in the pit by drills, trucks, shovels, locomotives, etc. Earthquakes are also easily identifiable with a very definite wave form envelope. The rock noises emanating from an unstable zone are identifiable by their characteristic envelope shape, a frequency usually between six and nine Hertz, and a characteristic irregular amplitude."

I also felt that the seismograph would provide sufficient confidence in the ability to know the state of movement to permit night operations, and still feel this should have been possible. My feedback on the use of the seismograph in this instance was that it was a practical method of providing continuous monitoring and was accepted by the people involved on site as such.

DISCUSSION OF: "Rock Performance Considerations for Shallow Tunnels in Bedded Shales with High Lateral Stresses" by J. D. Morton, K. Y. Lo and D. J. Belshaw. Presented at the 10th Canadian Rock Mechanics Symposium, Kingston, Ontario, September 1975.

SUBMITTED BY: J. H. L. Palmer, Geotechnical Section, Division of Building Research, National Research Council of Canada, Ottawa, Ontario, K1A 0R6

The authors have presented many aspects of a complex problem. This discussion concerns one facet only, the measured horizontal stresses.

Reference has been made to in situ stress measurements performed by the writer at Thorold, Ontario. Unfortunately, one of the results cited in the text is misquoted and the error is rather significant. At Thorold, the major principal stresses varied from 1200 to 2100 psi (8.3 to 14.5 MPa) and the minor principal stresses varied from 760 to 1750 psi (5.2 to 12.1 MPa). There were <u>no tensile stresses</u>. The 200 psi tension attributed to the Thorold tests was, in fact, measured in the Collingwood formation. Every test in two different members of the Lockport formation and in the upper member of the Clinton group at Thorold indicated very high horizontal stress, on the average about 1400 psi (9.6 MPa) compression (i.e. about 20 times the overburden stress).

The Dundas and Collingwood shales are layered deposits characterized by distinctly different physical properties in some of the layers. Overcoring tests in such deposits generally have indicated high stresses in the "soft" layers and low (or tensile) stresses in the "hard" layers. The 200 psi tension was attributed to a particularly competent portion of the Collingwood formation. The writer agrees with the authors that it is difficult to explain completely the abrupt stress changes from 1000 psi compression to 200 psi tension, but there is no reason to doubt the test results. The magnitude of the tensile stresses might be questioned, since that is a matter of interpretation of the measured deformations and in situ deformation modulus; there is no doubt that the tests indicated high compressive stresses and tensile or essentially zero stresses in close proximity.

More data similar to those presented by the authors are required to permit full assessment of the extent and engineering significance of this phenomenon of high in-situ horizontal stress. The writer strongly urges anyone who has the opportunity to support more detailed geotechnical investigations of future tunnelling projects to include in contract specifications carefully planned monitoring procedures, and to publish the data.

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ADDENDA

CORRECTIONS TO VOLUME 1 OF SYMPOSIUM PROCEEDINGS

1. Substitute the figures below for the figures on pages 173 and 175 respectively.

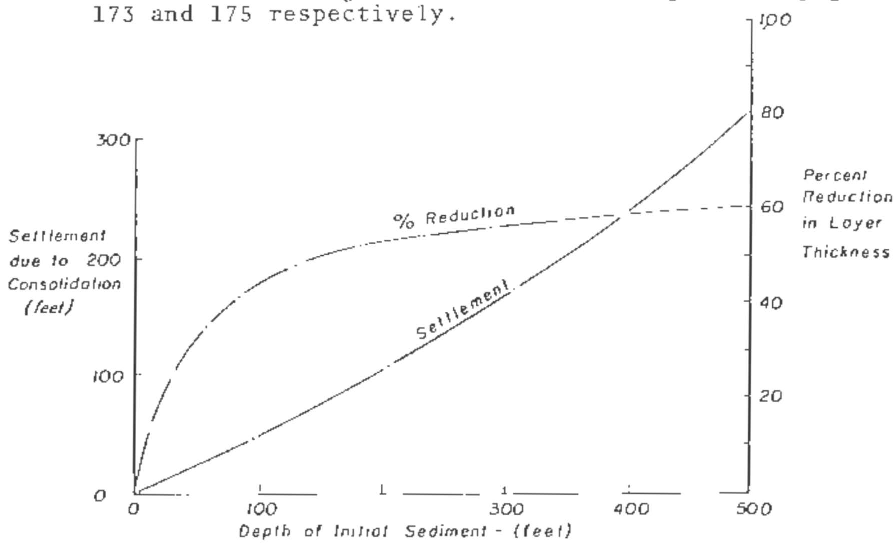

FIGURE 3 PREDICTED SETTLEMENT DUE TO CONSOLIDATION OF TAILINGS FINES

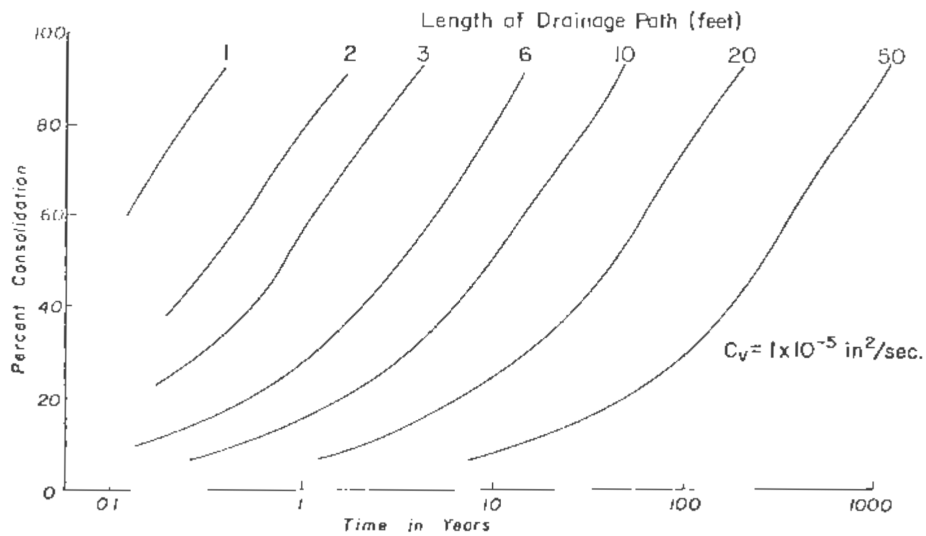

FIGURE 4 TIME RATE OF CONSOLIDATION FOR VARIOUS LENGTHS OF DRAINAGE PATH

2. Line 19 on page 418 delete the word 'strength'.

3. End of sentence line 7, page 420, add '(Morgenstern and Dusseault, 1975)'.

4. Last line page 424 add '(Bjerrum 1961)'.

5. Line 6, page 246 change 'figure 13' to 'figure 2'.

6. Line 12, paragraph 2, page 438 delete word 'result'.

7. Add the following set of references below. Note that these references are to replace those presently tabulated on pages 445-446.

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