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ROCK SLOPE STABILITY ANALYSIS USING MORGENSTERN-PRICE METHOD

BY

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FACULTY OF GRADUATE STUDIES & RESEARCH

The undersigned certifies that he has read, and recommends to the Faculty of Graduate Studies and Research for acceptance, a report entitled ROCK SLOPE STABILITY ANALYSIS USING MORGENSTERN-PRICE METHOD, submitted by Steve Tan in partial fulfillment of the requirements for the degree of Master of Engineering.

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Supervisor

Date May 1, 1975

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ABSTRACT

With the increasing size and depth of the open pit mines being constructed for the immediate future, the importance of the stability of the pit wall can not be overemphasized. A miscalculation in the performance of the wall may mean loss of life, damage to property and may produce severe financial consequences. The recognition of this fact together with today's technological knowhow in rock mechanics has resulted in a more economic and safety conscious open pit design. Systematic evaluation of the stability of the pits has become a part of the mine maintenance routine work.

The Gaspe Copper Mine, a wholly owned subsidiary of Noranda Mines Ltd., is an open pit operation designed to go to a maximum depth of 1,000 feet with a proposed 43^{o} slope angle. Present development is at the 250 foot level. It is feared that when mining progresses to a much greater depth, a slide would occur taking with it the smelter and the adjacent buildings.

The report deals with the stability investigation of a certain section of the Gaspe Copper Mine's ultimate pit geometry. A total of nine possible slip surfaces were analyzed first using the simpler wedge analysis and then with the more exact Morgenstern-Price method. The lowest factor of safety (for slip surface A-B-C) found is about 0.5, indicating that the slope is unstable. Analysis of the same slip surface A-B-C was then extended to a situation in which the groundwater level was lowered from 425 feet to 1,005 feet below the surface. The factor of safety in this case has risen to about 1.2, an acceptable condition.

CHAPTER I

INTRODUCTION

For years engineers have been constantly facing rock slope stability problems in their design work. However, insufficient effort has been devoted so far to studying the influential factors and the failure mechanisms which control the stability of these slopes. Often, the design criteria for a pit wall used by the practicing engineers were based on past experiences and rule of thumb; i.e., the slope angle is generally taken to be between 37^{o} and 45^{o}, and the ultimate angle of the slope is derived from some form of trial and error during the production period. A change of one or two degrees seems to cause insignificant effects on the economic success or failure of the project.

In recent times the continuing depletion of high grade mineral deposits and the vast increase in raw material consumption have opened up a new horizon for the mining industry. Ore bodies which were considered uneconomical to be developed previously have now become economically feasible. With the advent of large-volume earthmoving equipment, open pit mines are being planned and operated to depths and sizes that were unheard of before. As the pit gets deeper, the angle of the slope begins to play an important role in the profit eveluation of the operation. Brawner (1971) indicates that for a pit 3,000 feet deep, an increase of one degree in the slope angle represents an additional excavation cost of 5 to 15 million dollars. It is therefore obvious that substantial savings can be made if one utilizes the maximum slope angle without jeopardizing the safety of the people working in the vicinity.

Although the costs dictate the continuation or the desertion of an operation, the safety of the men and the equipment working in the area remains to be the first concern in any open pit design. In engineering the degree of safeness of a cut wall is expressed by a term known as the "factor of safety". It is defined as the ratio of the sum of all forces which may be mobilized for resisting failure to the sum total of all the disturbing forces, or as the ratio of available to mobilized shear strength. A factor of safety less than one indicates that an unstable condition exists and that failure is likely to occur. On the other hand, a factor of safety greater than one implies that the possibility of a major catastrophe is not likely to happen. The higher the value, the lesser is the chance of a slide.

The evaluation of the factor of safety involves elaborate procedures. With the aid of the modern computer technological advances, these calculations can now be performed in greater detail. The reliability of the analysis depends a great deal on the ability to establish the operative field shear strength and to some extent the method employed in the analysis, as well as the ability to estimate the ground water level conditions.

Many slope stability considerations have been published in the literature since Terzaghi (1962) calculated the critical height of a vertical wall cut through solid rock. Numerous methods were introduced, all of which make use of the principle developed in soil mechanics. Among the more widely used methods of stability analysis on hard rock are the wedge analysis and the Morgenstern-Price method, the latter being the more exact method as will become obvious in subsequent chapters.

The purpose of this report is to investigate the performance of a certain section of the Gaspe Copper Mines pit wall, as the mining activity progresses to its maximum depth, using both the Morgenstern-Price and the simpler Wedge Methods of stability analysis. Several potential slip surfaces were considered and the results and recommendations are discussed in Chapter VI.

It has been demonstrated by Muller et al (1970), Goodman et al (1968), and many others that failures in rock slopes tend to be confined to structural discontinuities, hence any rational stability analysis requires a sound knowledge of the geological environmental factors of the area under investigation. A description of the different factors which influence the stability of the slope and the various measures which can be taken to ensure a more stable condition are presented in Chapter II. A brief review of the Morgenstern-Price Method is incorporated in Chapter III. Chapter IV describes the sampling and testing procedures followed to obtain the shear strength parameters of the material(s) involved in the study. Chapter V looks at the regional and local geology surrounding the Gaspe Copper Mines and the specific features needed for a realistic consideration of a possible slide.

CHAPTER II

SPECIFIC FEATURES OF STABILITY OF ROCK SLOPES

In a study of the stability of rock slopes, there is a variety of features which are relevant to the accuracy of the analysis. Among the more important ones are:

(1) In-situ stress - Contrary to the common belief that the vertical stress is the only dominant factor in rock mass, high horizontal stress several times that of the vertical has been recorded (Hast 1967). This abnormally high horizontal stress coupled with the quick response of rock to unloading, such as a pit or stripping of overburden, causes bulging and tension cracks around the periphery of the excavation; thus reducing the shear strength of the rock.

(2) Structural discontinuities - This includes the joints, the faults and fault zones, and the planes of weakness such as contact planes and alteration zones. Unlike faults and fault zones, joints often occur in sets of two that are at a right angle to one another. Because of the closely spaced nature of these joints, the pattern is rather difficult to detect with the various geophysical exploration tools during the planning stage. Hence, trenches are required in order to map out these joints. The orientation (strike and dip) and the frequency of occurence of these discontinuities are statistically determined using a graphical method commonly recognized as the stereonet, see Fig. 1 and 2. For further information regarding the use of stereonet, the reader is referred to a paper by Hoek et al (1973).

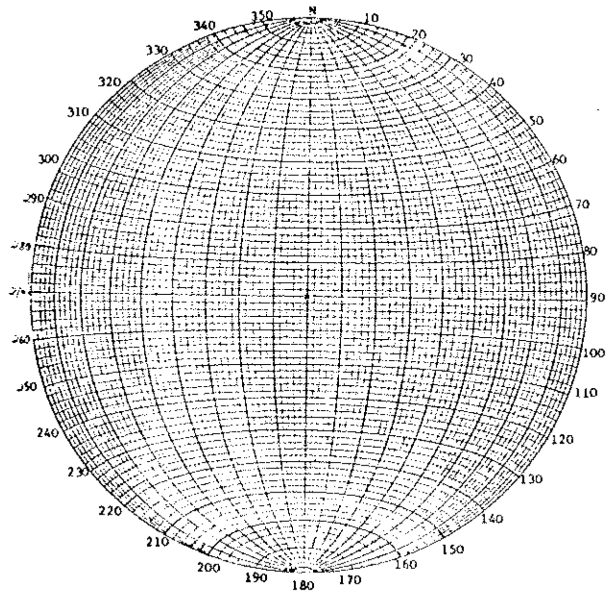

Fig. 1 Equal Area Stereonet

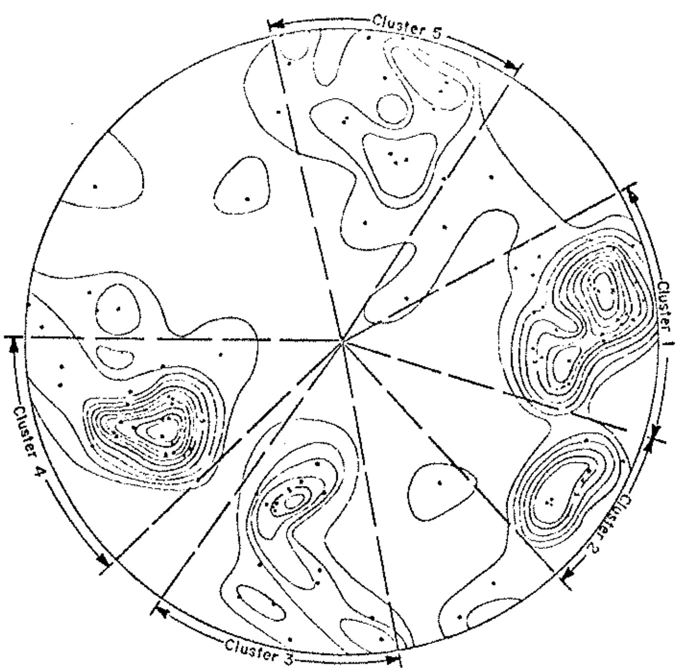

Fig. 2 Contours showing frequency of occurrence of joints.

Structural discontinuities affect the stability of rock slopes in two ways:

(a) Shear strengths along the plane of discontinuity are generally lower; consequently, the factor of safety is lower. The reduction in shear strength may be the result of a number of circumstances:

   (i) loss of cohesion due to the fact that the material is fractured

   (ii) the presence of the opening allows the water to flow in resulting in a reduction of the effective normal stress and the shear strength

   (iii) alteration products filling the openings act as some form of lubricant

   (iv) large displacement as in the case of a fault reduces the shearing resistance from peak to residual, and/or

   (v) a combination of the above.

(b) Depending upon the location and the orientation of the pit geometry, the structural discontinuities may be situated in such a way that it enhances the possibility of a major rock slide, Fig. 3.

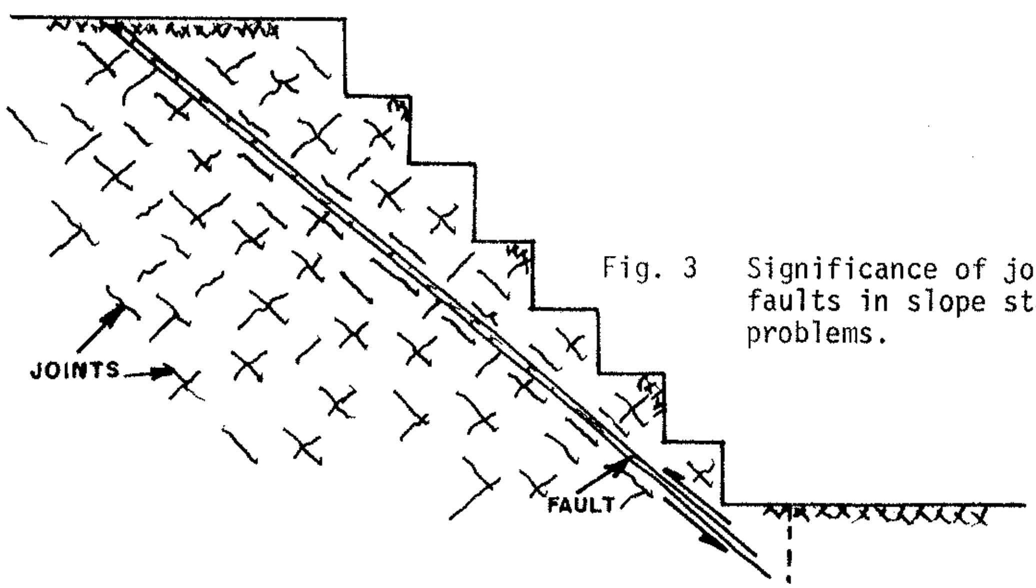

Fig. 3 Significance of joints and faults in slope stability problems.

(3) Groundwater condition - Water pressure reduces the effective normal stress and lowers the shearing strength accordingly (Morgenstern 1970). Chemical alteration due to the presence of water also lowers the strength. Furthermore, excessive water influx into the pit is a nuisance to the mining operation and can become very costly if not properly corrected. Therefore, a complete picture of whether the mine is located in a regional groundwater discharge area or recharge area or in some intermediate area is certainly advantageous, since without it erroneous conclusions can be drawn with respect to the suitability of drainage facilities and other remedial measures (Patton et al 1970).

(4) Earthquakes - Many of the earthquake-induced landslides have seriously affected mining operations in one way or another although no actual slope failure in open pit mines during earthquakes has been reported in the literature. In the last ten years studies into the influence on stability were carried out by investigators like Seed (1970), Goodman et al (1966), and others. Laboratory test models such as a block resting on an inclined plane subjected to a series of acceleration pulses simulate the actual behaviour of rock or soil during earthquakes. Whitman (1970) in his investigation has discovered that rockfalls begin to occur when the MM (Modified Mercalli) intensity reaches VII, Table 1, and the frequency of rockfalls and rockslides increases with increasing intensity. He suggests that the expected intensity of the ground motions must be determined on a probabalistic basis. The effect of earthquakes is entered in the stability calculations as an external force.

Table 1 (After Whitman 1970)

ABRIDGED MODIFIED MERCALLI INTENSITY SCALE

VI Felt by all, many frightened and run outdoors. Some heavy furniture moved; a few instances of fallen plaster or damaged chimneys. Damage slight.

VII Everybody runs outdoors. Damage negligible in buildings of good design and construction; slight to moderate in well-built ordinary structures; considerable in poorly built or badly designed structures; some chimneys broken. Noticed by persons driving motor cars.

VIII Damage slight in specially designed structures; considerable in ordinary substantial buildings with partial collapse; great in poorly built structures. Panel walls thrown out of frame structures. Fall of chimneys, factory stacks, columns, monuments, walls. Heavy furniture overturned. Sand and mud ejected in small amounts. Changes in well water. Persons driving motor cars disturbed.

IX Damage considerable in specially designed structures; well designed frame structures thrown out of plumb; great in substantial buildings, with partial collapse. Buildings shifted off foundations. Ground cracked conspicuously. Underground pipes broken.

(5) Blasting - There are two aspects in which blasting affects the open pit slope stability. The first aspect being that the blast waves produce ground motion similar to that caused by earthquakes. Secondly, blast waves induce fracturing of rock in the vicinity of the slope. Fracturing can be minimized by shortening the blast holes to prevent overbreaking of rock, as illustrated in Fig. 4, and by delaying the interval between shots detonated. Other means of controlling blasting effects were discussed by Oriard (1971). As a general rule, for harder rock, a 9 millisecond and longer delay interval is enough to ensure that no energy is fed back into the peak of the waves from later delay periods, but for softer material such as tar sand, it is necessary to use 100 millisecond delays to stop the additive effects of one charge. Edwards et al (1960), during their study of the effect of blasting on building damage, found that there exists a good correlation between

damage and peak velocity in the disturbance. They went farther and developed a scaling law for particle velocity versus scaled distance of the form:

$$V = \left(\frac{d}{W^{1/2}}\right) x$$

where, V = the particle velocity (in./sec.)

d = distance in feet from shot to observation point

W = charge weight in lb. per delay

K = constant, depending on the distribution of the charge and the material type, varies from 45 to 450

x = varies with material type and whether the longitudinal, vertical or transverse component is being measured

Pursuing the same line of thought, Bauer et al (1970) proposed the following particle velocity damage criteria for rock mass affected by blasting vibrations:

(a) 10 (in./sec.) .... no fracturing of intact rock

(b) 10 - 25 (in./sec.) .... minor tensile slabbing will occur

(c) 25 - 100 (in./sec.) .... strong tensile and radial cracking

(d) 100 (in./sec.) .... complete break-up of rock mass will occur

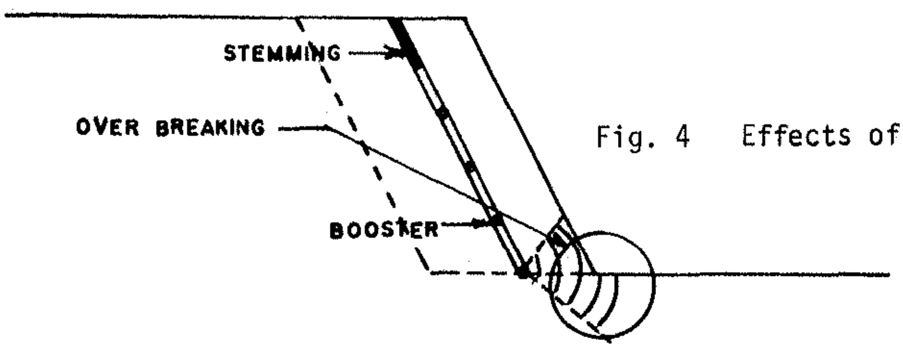

Fig. 4 Effects of overdrilling.

(6) Weathering - Deterioration of rock due to weathering within the life span of a mining operation is not common. However, there are cases reported in which newly exposed surfaces of shale have weathered considerably, losing much of their shear strength in only a few months. Shear parameters used in the analysis of these cases must be carefully chosen. At the present, there are no said rules on how the shear strength ought to be selected. Another phenomenon that has to be accounted for in a stability analysis is the change in the strength of rock with increasing depth. Given the same type of rock, the farther away from the surface, the lesser the weathering effect, hence the greater its strength will be. It should be noted that in each of the above factors, time plays an important role in degree of influence in the stability calculations. With this in mind, perhaps the performance of a slope should be checked at regular intervals so that the danger of a slide due to an unforeseen cause -- excessive rain or snow, pore pressure build-up, etc. -- can be rectified. Forms of monitoring a potential slope failure in open pits are described by Watt (1970) and Kennedy (1971).

After the warning comes the improvement on the slope. According to Golden (1970), stabilizing a slope is usually possible. But certain questions must be asked in order to select the correct method of stabilization. These are:

(a) How much stability is required?

(b) How much funding is allocated for the project?

(c) What causes the instability?

(d) For how long is it required?

Once these questions are answered, it takes little effort to decide which of the available methods of stabilization to use, i.e.:

(a) mechanical strengthening

(b) change shape of slope

(c) drainage (Hoek et al 1970)

CHAPTER III

MORGENSTERN-PRICE METHOD

Stability analyses generally fall in two categories (Krahn 1974): (1) stress-strain analysis, and (2) limit equilibrium methods. The difference between the two categories lies in their approach to solving a problem. In the stress-strain analysis, one uses the deformational properties of the material, hence the constitutive equations governing the material behaviour must be known. In limit equilibrium methods, one requires only that all equations of static equilibrium be satisfied, and no assumption is made on the deformation characteristics of the material.

Given below are the principles of one of the more advanced limit equilibrium analyses, the Morgenstern-Price Method. A complete discussion of the method is published in two papers by Morgenstern-Price (1965) and (1967). This method, unlike the wedge analysis, the slip circle by Bishop (1955) and others, or the analysis developed by Jennings (1970), places no restriction on the shape of the failure surface, making its applicability most attractive to rock slopes. The theory goes as follows:

Assuming we have:

(1) a slope surface, as in Fig. 5, defined by a line

$$y = z(x) \quad \ldots\ldots \quad (1)$$

(2) an arbitrary slip surface represented by an equation

$$y = y(x) \quad \ldots\ldots \quad (2)$$

(3) the position of thrust of internal water pressure given by

$$y = h(x) \quad \ldots\ldots \quad (3)$$

(4) the line of action of the effective horizontal force described by

$$y = y_t'(x) \quad \ldots\ldots \quad (4)$$

The equations (1), (2) and (4) are known, and (3) is unknown. A schematic illustration of the above assumptions is presented in Fig. 5.

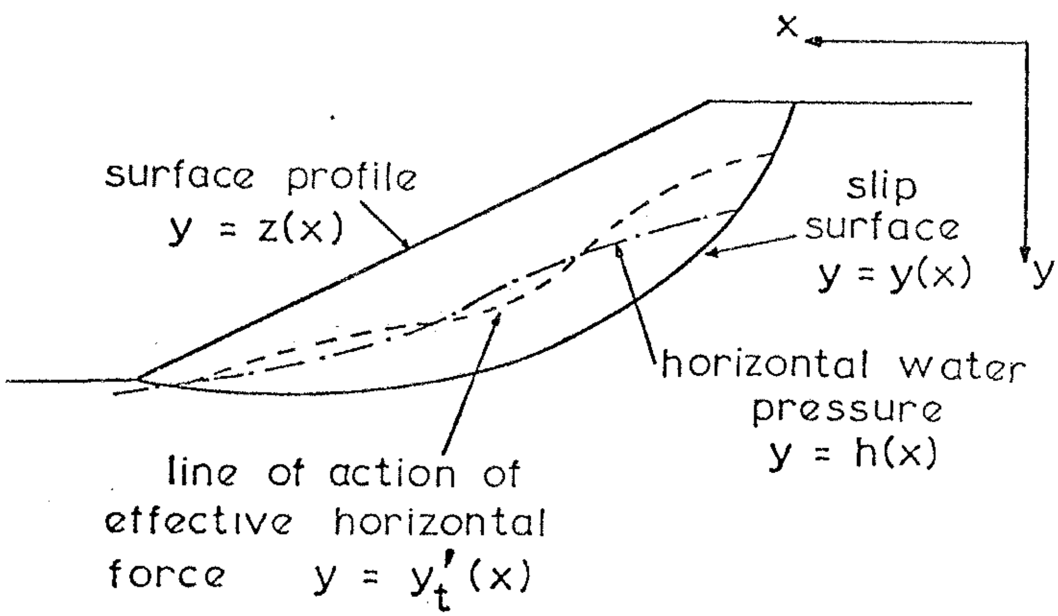

Fig. 5 Schematic illustration of the assumptions.

Consider now a vertical slice of width dx with forces acting on the mass as shown in Fig. 6.

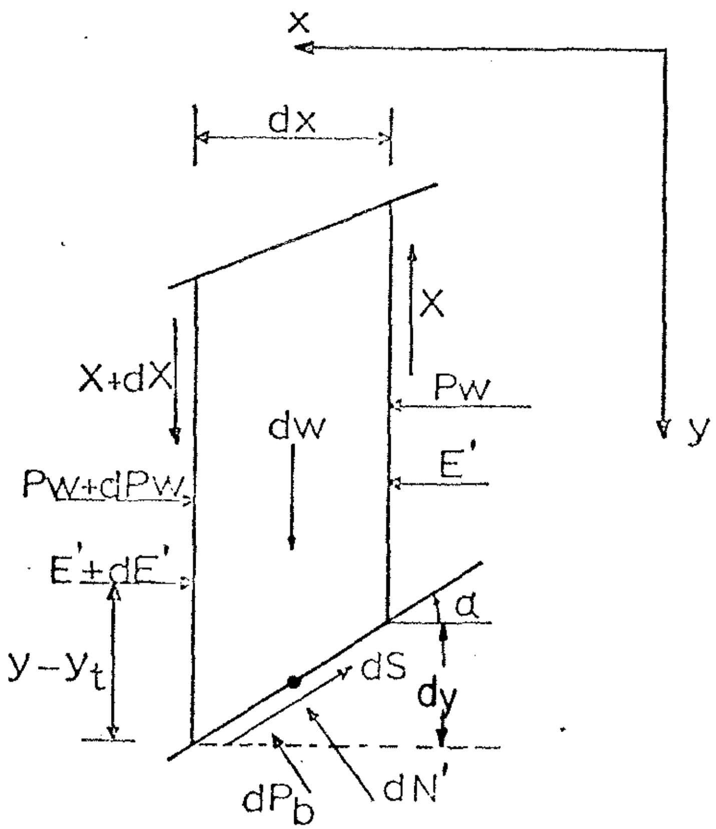

Fig. 6 Force diagram of the slice.

In this figure, let:

E'	be the lateral thrust on the side of the slice in terms of effective stresses
Pw	be the resultant water pressure acting on the side of the slice
X	be the vertical shear force acting along the side of the slice
dw	be the weight of the slice
dS	be the shear force acting along the base of the slice
dN'	be the effective normal pressure
α	be the inclination of the base of the slice from the horizontal
dP_b	be the water pressure acting on the base of the slice

In order to avoid rotation, the sum of the moments taken about the center of the base of the slice must be equal to zero. By taking moments about the mid-point of the base of the slice and proceeding to the limit as $dx \rightarrow 0$, it can be readily shown that:

$$X = \frac{d}{dx} \ (E' \cdot Y_t') - Y \frac{dE'}{dx} + \frac{d}{dx} \ (Pw \cdot h) - Y \frac{dPw}{dx} \qquad \text{.............} \ (5)$$

Similarly, for conditions of equilibrium in the normal and parallel to S directions,

$$dS = dE' \cos \ + \ dPw \cos \ - \ dX \sin \ - \ dPw \sin \qquad \text{................} \ (6)$$

and

$$dN' + dPb = dW \cos \ - \ dX \cos \ - \ dE' \sin \ - \ dPw \sin \qquad \text{................} \ (7)$$

Also from the Mohr-Coloumb failure criterion (in terms of effective stresses),

$$dS = \frac{1}{F} \left[c' \ dx \sec \ + \ (dN') \tan \phi' \right] \qquad \text{...........................} \ (8)$$

where c' = cohesion intercept

ϕ' = the effective angle of shearing resistance

F = factor of safety

But careful examination of the above equations reveals that (6), (7), and (8) are really only one equation. Thus, we have on hand two governing equations with three unknowns, E', X, and $Y_t'(x)$. Clearly one more equation is necessary if the problem is to be statically determinate. A generalized assumption is made that

$$X = \lambda f(x) \ E \qquad \text{..} \ (9)$$

where f(x) is specified but λ must be determined from the solution, and a computer program is written to that extent by the Department of Civil Engineering of the University of Alberta. The effect of the f(x) specification on the evaluation of λ and F and a range of values assigned to f(x) are described in a Ph.D thesis by James (1970). Further discussion of other special assumptions relating X and E forces (Bishop 1955, Janbu 1954) or the line of horizontal thrust (Kenney 1956) is beyond the scope of this paper.

CHAPTER IV

DETERMINATION OF SHEAR STRENGTH PARAMETERS

The determination of the shear strength of the rock discontinuities involves three phases: (1) obtaining samples, (2) preparing test specimens, and (3) shear testing to derive a strength value necessary for stability analysis.

Sampling Procedure

As was mentioned before, failure in rock slopes tends to be confined to structural discontinuities. So the first step is to assess the governing type(s) of discontinuities of the area in question. This is accomplished by an extensive structural mapping. Upon recognizing the controlling features, a kinematically possible sliding mechanism is determined. Then it is possible to select the sites in which representative samples of the critical discontinuities could be taken for laboratory testing. Before the sample is removed from its original position, the strike, dip, direction of shear, location, and elevation should be marked clearly on each specimen and recorded in the field book for future references.

Sampling of rock is achieved in one of two ways:

(1) block sample, which utilizes no special equipment other than a pick and shovel. This method is easy to perform and costs considerably less than coring.

(2) cored sample is obtained with a masonry drill coring around the desirable discontinuities. The method is adaptable to both surfacial conditions and deep boreholes. It can be applied to any special directions, places, and depths. A picture of the portable coring unit developed by the

Department of Civil Engineering, the University of Alberta, is shown in Fig. 7. For complete description and specification of this machine, the reader is referred to Stewart (1974).

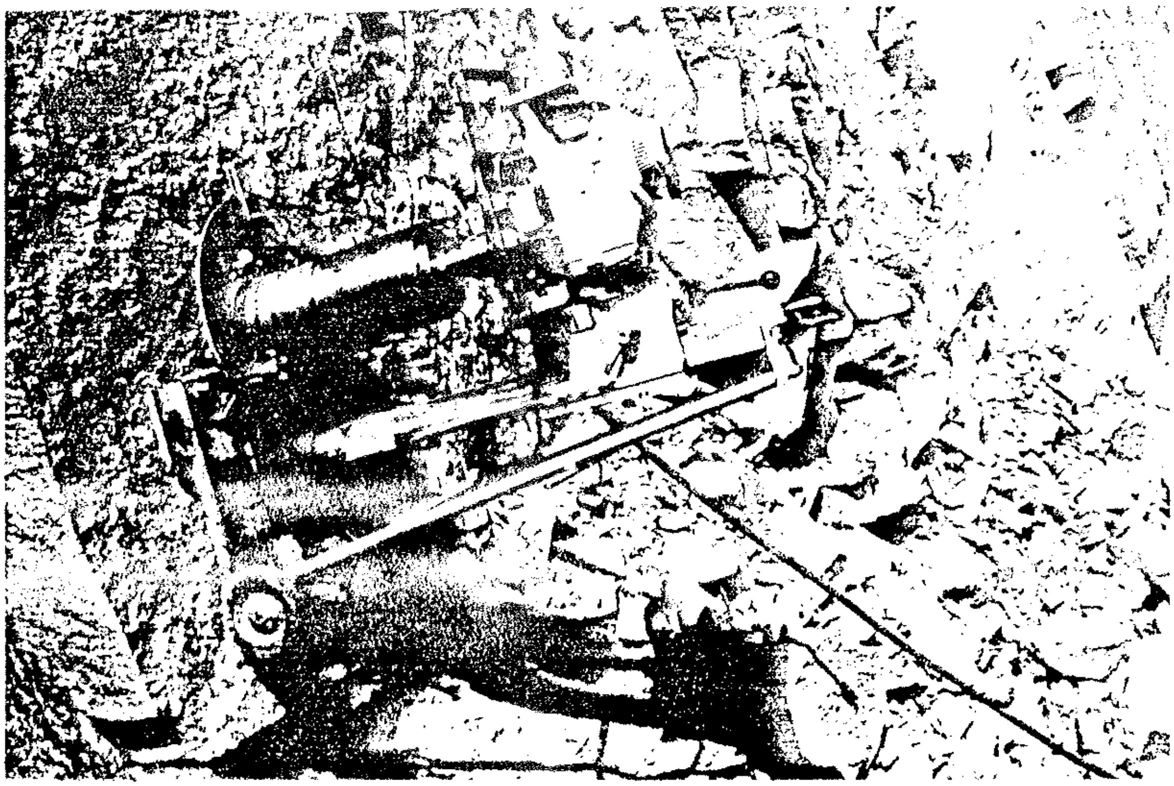

Fig. 7 Photograph of drilling unit used. (from Stewart 1974)

Although coring is very versatile in many senses, the method does have severe problems associated with anchoring and core recovery when highly fractured sample sites are encountered. Furthermore, penetration of the drill into hard igneous rocks sometimes presents a problem. Rate of advance as low as 1 inch/hour has been reported (Stewart 1974). At this rate, it implies that only one sample can be recovered per day. Since time and money are the controlling factors in any project, block sampling is always preferred over coring.

Whether a core or a block, the samples would fall apart readily. Every precaution must be taken to preserve the specimen and the discontinuity intact. It was found that by wrapping the samples in rubber sheet foam, the breakage due to handling and transportation back to the laboratory was virtually eliminated.

For the particular mine under investigation, Gaspe Copper Mines, since the rock was highly weathered and fractured, only the block sampling method was employed. The largest specimen obtained was about 4" x 6".

Test Apparatus and Sample Preparation

Based on experience, the direct shear method was chosen. Testing was to be performed on the small and large shearing machine with a constant strain rate (0.048 inches per minute) under various normal pressures ranging from 0.4 to 21 kg./cm.^{2}. The sizes of the specimens selected for the experiment were to be 2" x 2", 4" x 5", and 12" x 12".

The small shearing machine, Fig. 8, was designed and built at the University of Alberta. It consists of (1) a dead weight lever, (2) a shear box, (3) an electric motor, (4) a gear box, and (5) two LVDT's (linear variable differential transducer) and a three hundred pound lead cell. The machine handles sample sizes 2" x 2" and 4" x 5" and is capable of delivering up to a maximum of 2 tons shear force. The shear load is applied to the lower half of the shear box through a gear box chain driven assembly powered by an electric motor, while the upper half is held in place with a load cell attached to it. This produces a relative motion between the two halves of the shear box along a preconceived plane. The resistance of the upper half to the motion is measured by the load cell in terms of voltage and is later converted to an equivalent shear force. Teflon strips are used to separate the two halves and along the guides so as to minimize the machine friction generated during testing. The normal load is applied to the sample by a dead weight arrangement. The horizontal and vertical movements and the shear load were continually monitored by a digital printer which gave the three readings at about 8 second intervals. In addition, a continuous plot of the horizontal movement versus the shear load was recorded on an x-y plotter.

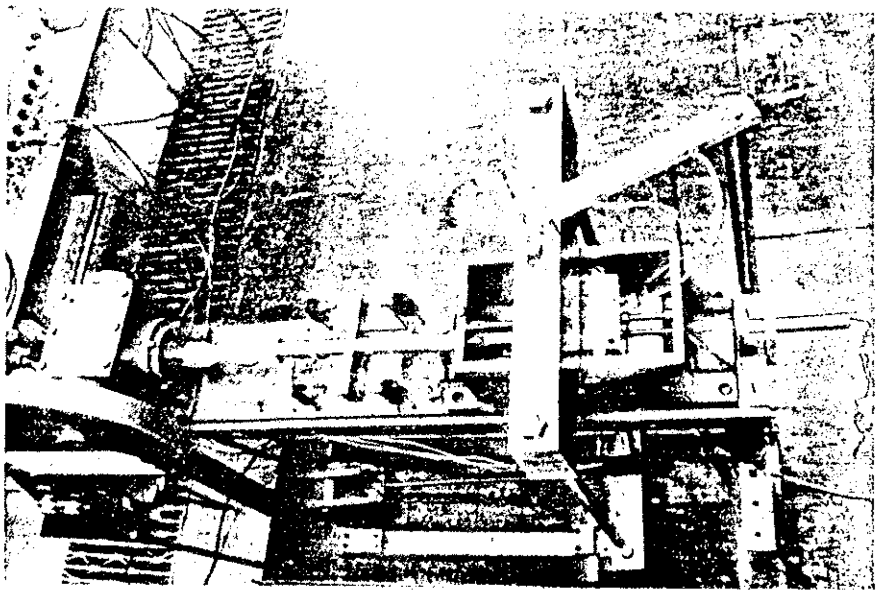

Fig. 8 The modified shear box for the two-inch samples. (from Krahn 1974)

The large machine shown in Fig. 9 is a 10-ton Wykeham-Farrance direct shear machine. It has basically the same set-up except that, (1) the normal load is applied through a load yoke by a hydraulic ram, (2) the vertical displacements are measured at 4 corners instead of just one LVDT at the center of the sample, (3) it can handle bigger samples (up to 12" x 12" in size), and (4) the shear load is measured by strain gauges rather than by load cell.

Having decided the method and the apparatus to be used, one then continued the prepartion of the samples. Each specimen was first roughly cut to 1.8" x 1.8" or 3.8" x 3.8" with a diamond saw, followed by casting it in molds with F-190 grout fast immersible cement, left for curing for about 24 hours, ground to the exact size of the shear box, and finally the sample was ready for testing. The above procedure gives the sample a snug fit in the shear box with the plane of discontinuity seated in the horizontal position and ensures that only horizontal forces are being generated during shearing (Herget et al 1973).

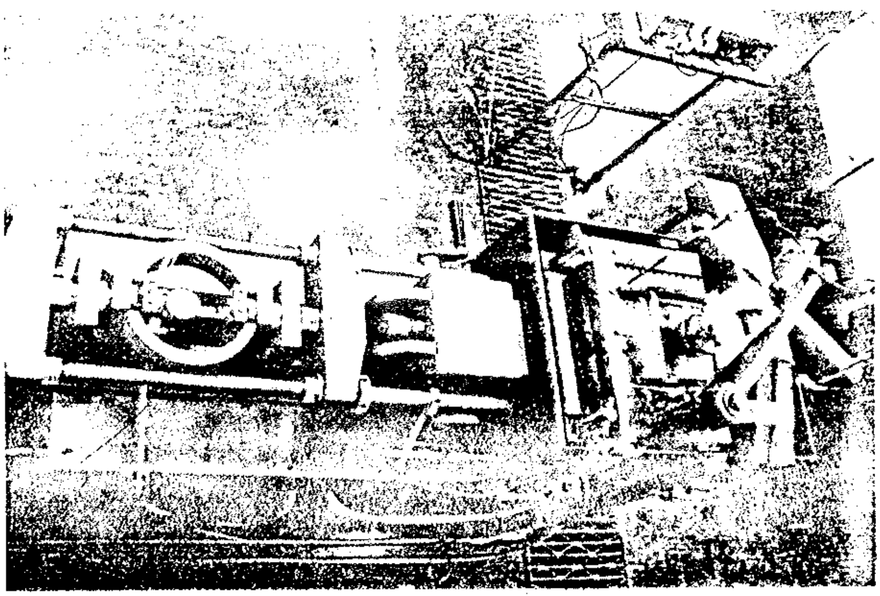

Fig. 9 The Wykeham-Farrance shear box. (from Krahn 1974)

Testing Procedure

The general procedure was to (1) assemble the machine, (2) place the sample in the shear box, (3) apply the intended normal load, and (4) position the horizontal and vertical LVDT's. Once the set-up was completed and the initial readings recorded, the shearing was initiated. With the center of the sample as the reference position, the horizontal deformation continued until a displacement of 0.25" was reached. At this point, the machine was stopped and reversed. Shearing continued in the other direction until it had reached 0.25" past its reference point. The motion was then stopped again and brought back to the original position. This constitutes one complete testing cycle under one normal load. The procedure was repeated on the same specimen under various increasing normal loads.

Analysis of Test Results

Each specimen was tested under 4 or 5 different normal loads. With each normal load, a shear load versus horizontal displacement curve and a vertical versus horizontal displacement plot were obtained. From the information furnished by these curves and with proper friction correction, a third graph - shear stress versus normal stress - was constructed for each specimen. In this third graph, the angle of inclination of the line represents the friction angle of the matter and the y-intercept represents the apparent cohesion of the material.

The construction of the shear stress versus normal stress goes as follows: Given the typical results, Fig. 10 and 11, of a sample GBXX with a surface area A, shearing under a normal load N, the maximum shear force is taken as the peak value of the first 0.25" displacement curve. Dividing the maximum shear force by the corresponding corrected area Ac and the normal load by the corrected area Ac gives the maximum shear stress, τ_{max}, with the corresponding normal stress, σ_n.

$$A_1 = L * W \quad \text{.................. original area ....} \quad (1)$$

$$Ac = (L - \Delta X) * W \quad \text{.......... corrected area ....} \quad (2)$$

$$\tau_{max} = \frac{Fmax}{Ac} \quad \text{..} \quad (3)$$

$$\sigma_n = \frac{N}{Ac} \quad \text{..} \quad (4)$$

By repeating the same procedure over the entire range of normal load and plotting τ_{max} versus σ_n, one ends up with a graph similar to Fig. 12. The friction angle determined is the maximum friction angle designated as ϕmax.

Similarly the same can be done with the residual force, Fres, taken as the average value at zero deformation, $\Delta X = 0$. The friction angle obtained in this case is the residual friction angle designated as ϕres.

$$Fres = \frac{F_1 + F_2}{2} \quad \ldots\ldots\ldots\ldots \quad (5)$$

$$\phi max = \tan^{-1} \frac{\Delta\tau_{max}}{\Delta\sigma_n} \quad \ldots\ldots\ldots\ldots \quad (6)$$

$$\phi res = \tan^{-1} \frac{\Delta\tau_{res}}{\Delta\sigma_{n1}} \quad \ldots\ldots\ldots\ldots \quad (7)$$

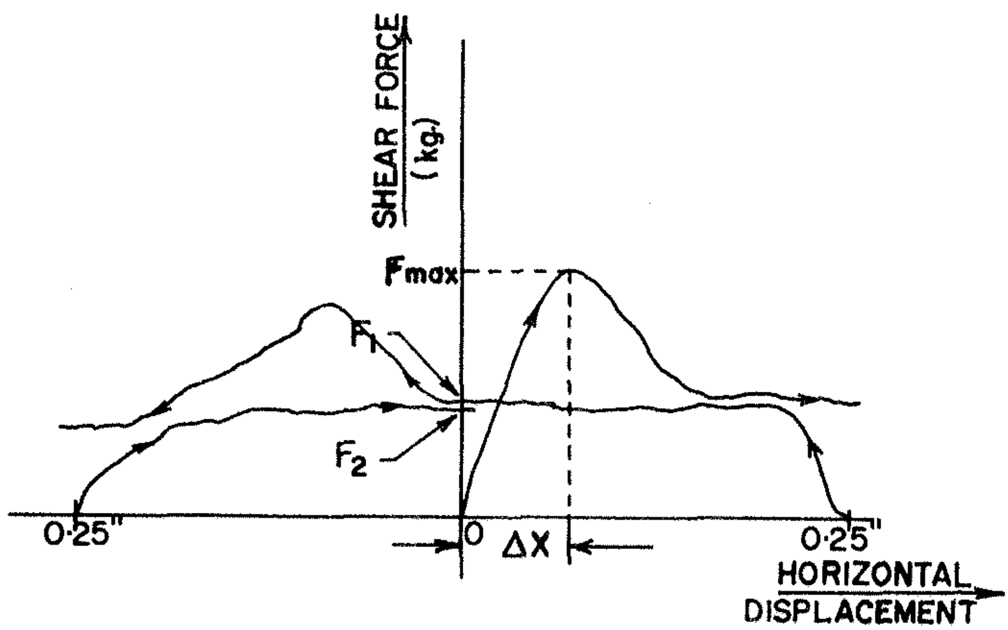

Fig. 10 Typical results of a test recorded in the x - y plotter.

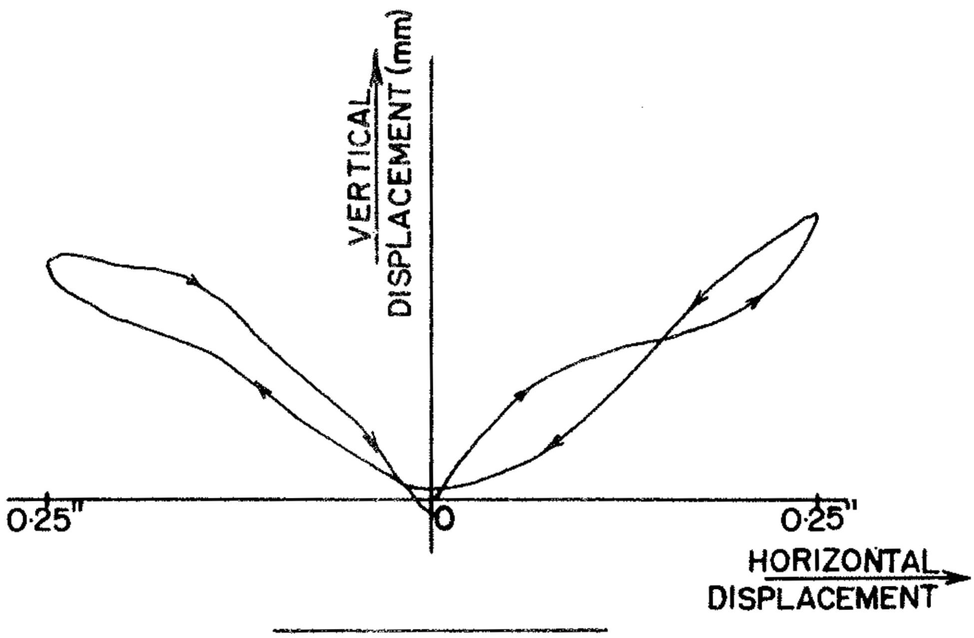

Fig. 11 Typical example of a displacement curve.

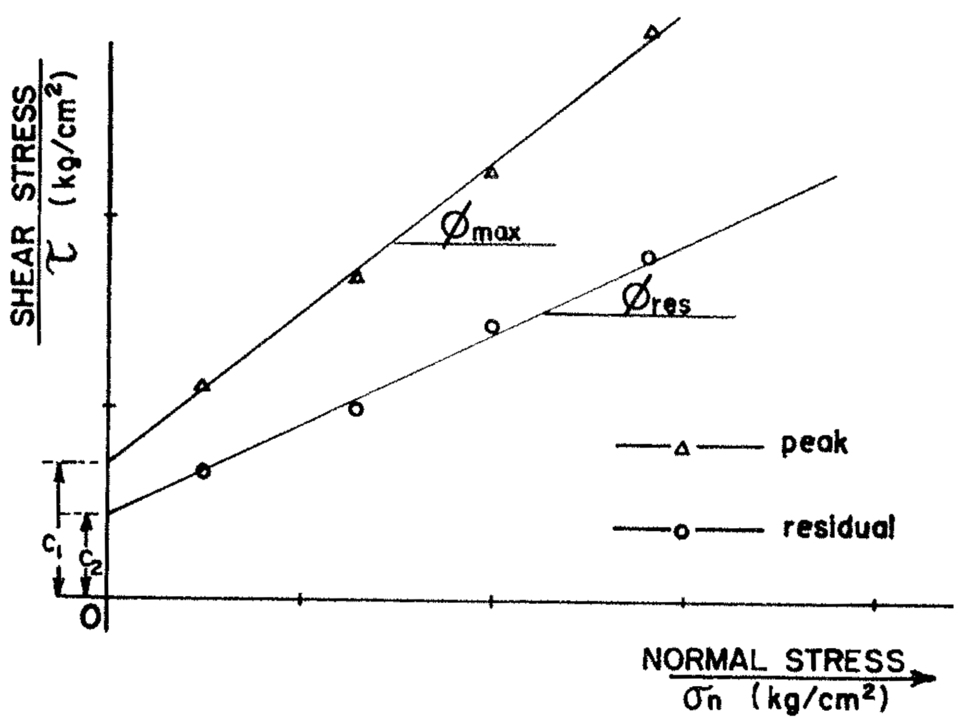

Fig. 12 Graph showing how the strength parameters are determined.

CHAPTER V

CASE HISTORY - GASPE COPPER MINES

Regional Geology

The Gaspe Peninsula, situated in the eastern part of Quebec province, is about 150 miles long (in the east-west direction) and 75 miles wide. It is bounded northeast by the Gulf of St. Lawrence and on the south by the Bay of Chaleurs. Physiographically and geologically, Gaspe is part of the northeastern trending Appalachian Mountain system, which extends from Alabama to Newfoundland.

The geology of the area is comprised of three belts, all of which run parallel to the longer axis of the peninsula, Fig. 13. The northern belt, consisting of easily eroded shale and limestone, is underlain by rocks of both the Ordovician and Cambrian age. The central belt is a strip of 20 - 35 miles wide -- Central Gaspe Basin -- underlain for the most part by Silurian and Devonian sedimentary rocks. Some granite intrusions and volcanic rocks occur in the north-central part of this belt. The southern belt is characterized by the overlapping flat-lying Carboniferous conglomerate, the Bonaventure formation. In all the three belts, there is evidence of at least one volcanic activity occuring during the geologic ages. All the rocks, except those of the Carboniferous age, have been folded and faulted by the mountain-building movements which took place at the close of the Ordovician time (Taconic revolution), at the end of the Lower Devonian (Shickshockian or Acadian revolution), and again after the end of Devonian sedimentation, Jones (1942).

Mineral deposits are found in several parts of the Gaspe Peninsula. A common feature that has been observed in these deposits is that they occur along planes of bedding, jointing, cleavage, contact, and along fractures,

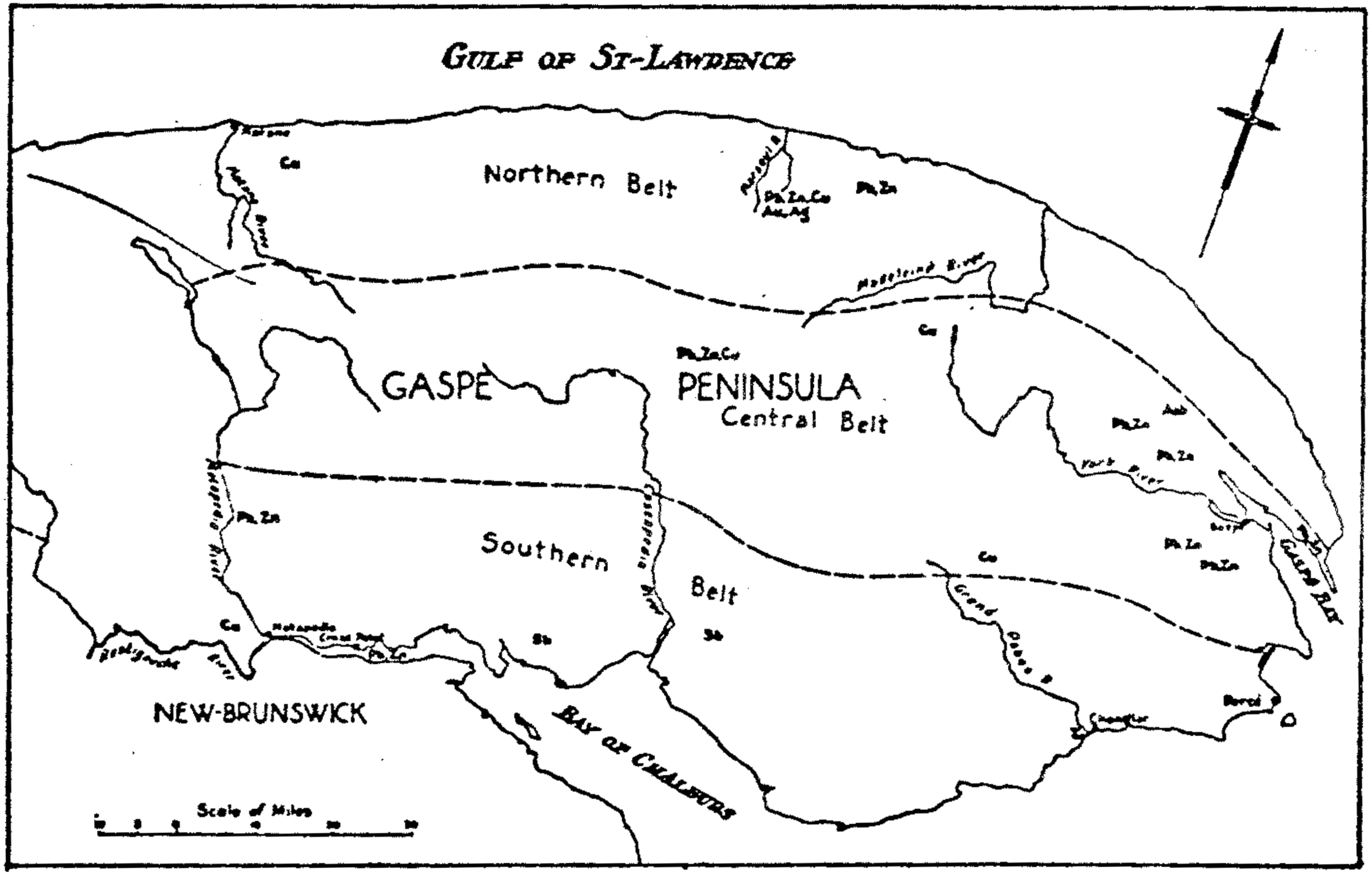

Fig. 13 Sketch map of Gaspe Peninsula, Quebec taken from the paper by Jones.

forming a network of closely spaced discontinuous veinlets. These networks have been traced for lengths varying up to more than half a mile, and widths up to more than 20 feet. Depths extending well over 500 feet have been discovered in regions of high relief.

The chief metals being mined in this peninsula are lead, zinc, and copper. Other minerals such as gold, silver, iron, molybdenum, antimony, chromite and asbestos are also recovered as bi-products.

General Information

The Gaspe Copper Mine, a wholly owned subsidiary of Noranda Mines Ltd., is located in Murdochville, Quebec, some 20 miles southeast of Mont-Louis and 20 miles west of Bald Mountain (see Fig. 14). The mine came into production in April 1955, producing mainly copper with molybdenum, gold, and silver as bi-products. Estimated ore reserves as of December 31, 1972 were 285,018,000 tons, grading 0.49% copper per ton. Present production capacity is set as 34,000 tons of sulphide ore per day. Current developments on the pit have advanced to the 250 foot level of the proposed 1000 foot deep ultimate pit.

Local Geology and Discontinuity

The geology of the particular section of the Gaspe pit under investigation, as deduced from the information supplied by the mine office, consists of the Grande Greve Formation overlying the Cape Bon Ami Formation (see Fig. 15). A dyke, 900 feet wide, cuts vertically across the Formation causing alteration zones to both sides of the dyke. Seated to the right of the dyke and immediately above the altered rock is a zone of weathered material, generally called oxides. For simplicity the two Formations, the Grande Greve and the Cape Bon Ami, and the altered rock are referred to as limey quartzite and skarn respectively. Structurally, there are two major

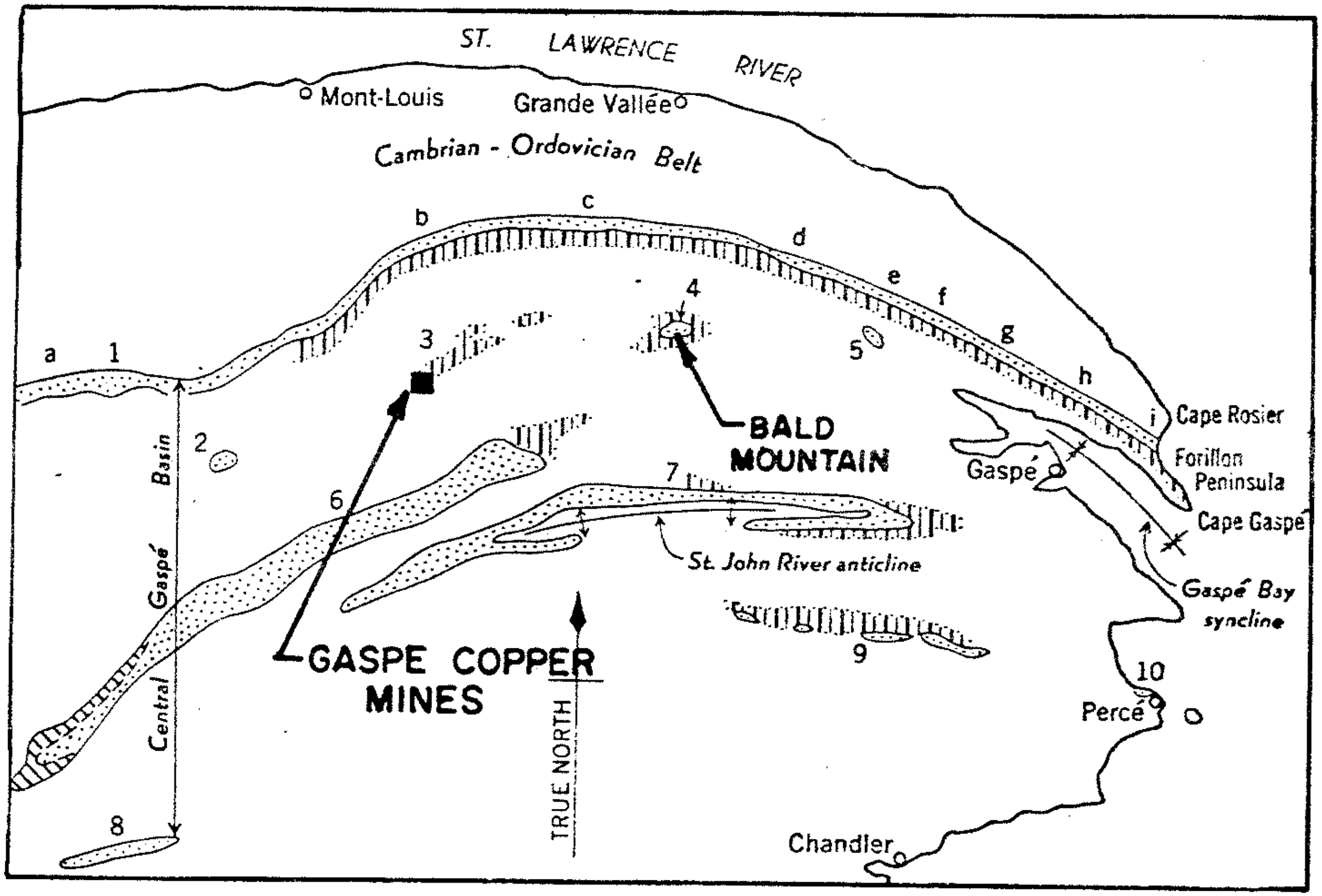

Fig. 14 Sketch of the location of the Gaspe Copper Mines.

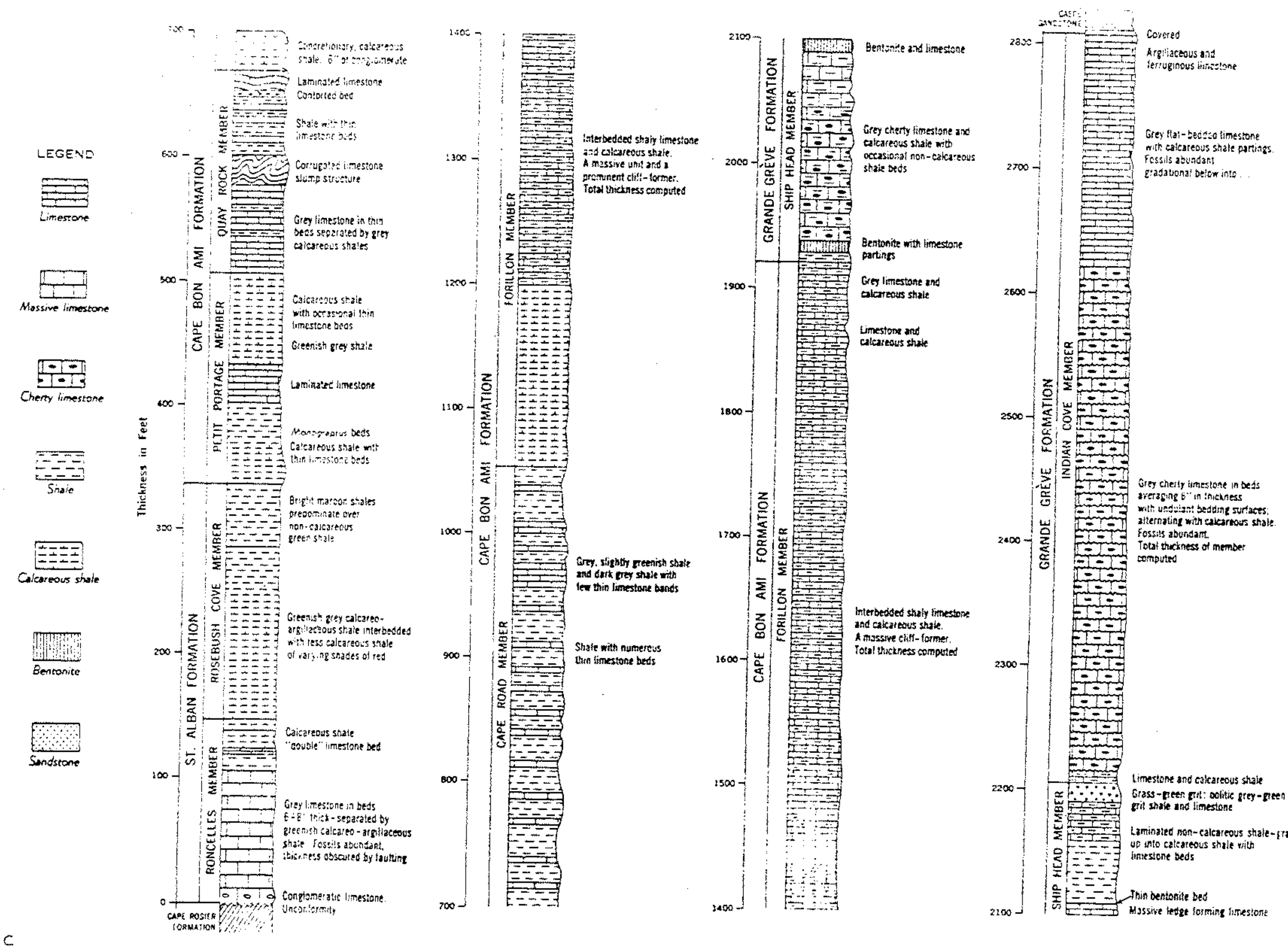

Fig. 15 Profile of the two Formations taken from G.S.C.

sets of joints, dipping at 22^{o} and 86^{o}, which affect the overall stability performance of the pit wall. A radial cross-section along S 13^{o} W of the pit illustrating both the rock types and the structural discontinuities is shown in Fig. 16a and 16b. The corresponding shear strength parameters of these rocks are summarized in Table II.

Stability Analysis

Two methods were used in the stability analysis of the slope, (1) the Morgenstern-Price Method, which was described in Chapter III, and (2) the wedge analysis.

The wedge is based on the following simplifying assumptions:

(1) The sliding block behaves as a rigid body.

(2) Sliding occurs only along the subhorizontal joint set (inclined at 22^{o}).

(3) Water pressure is distributed along the failure surface as shown in Fig. 16c.

(4) Where more than one rock type is encountered, a weighted average of the friction angle is used.

Hence, for a unit thickness, the safety factor derived is:

$$F.S. = \frac{S}{D}$$

where $S = \left[N - U_2 - U_1 \cos (86 - \alpha^{o})\right] \tan\phi + c \times l$

$D = T + U_2 \sin (86^{o} - \alpha^{o})$

S is the sum total of the resisting shear force

D is the sum total of the driving force

N is the component of weight (W) of the rock mass acting normal to the slip surface ($= W\cos\alpha$)

T is the component of weight of the rock mass acting parallel to the slip surface ($= W\sin\alpha$)

U_2 and U_1 are the resultant forces exerted by water

C is the cohesion

l is the length of the sliding surface

α is the inclination of the sliding plane ø is the friction angle (where two materials are involved, a weighted average is used)

F.S is the factor of safety

The results of the wedge and Morgenstern-Price analyses are presented in Table III and IV respectively. Table V summarizes the safety factors derived from the two methods. Calculations and other pertinent information are given in the Appendix.

Table II Rock strength parameters

ROCK TYPE	COHESION C	FRICTION ANGLE ϕ	COHESION C	FRICTION ANGLE ϕ	BULK DENSITY (PCF)		REMARKS
	ACROSS JOINTS (PSF)	ACROSS JOINTS (DEGREE)	ALONG JOINTS (PSF)	ALONG JOINTS (DEGREE)	γ_{SAT}	γ_{WET}	
(1)			4300	39°	170	165	(1) represents the skarn.
(2,3)	327	51°	409	32°	170	165	the dyke and weathered rock (2,3) are taken as having the same properties with two strengths.
(4)			4300	39°	170	165	(4) represents the skarn.
(5,6)	5440	39°	3960	39°	170	165	(5,6) represent the limey quartzites with two strength parameters.
(7)			0	0°	117	117	(7) is the equivalent wt. of the blgds in terms of ten ft. of rock.

Table III Results from wedge analysis

SLIP SURFACE	α	W (WT. OF THE SLIDING BLOCK)	U_2	U_1	FRICTION ANGLE ϕ	L (LENGTH OF SLIDING SURFACE)	COHESION C	$S=[N-U_2-U_1\cos(86^\circ-\alpha)]*\tan\phi+C*L$	$D=[W\sin(\alpha)+U_1\sin(86^\circ-\alpha)]$	F.S
	DEGREE	TON	TON	TON	DEGREE	FT.	TON/FT.^{2}	TON	TON	FACTOR OF SAFETY
(FIG·17) A-B-C	22°	5186	2260	1224	33·1°	768	0·232	1488	3044	0·489
(FIG·18) D-E-F	22°	11432	3761	519	34·3°	1076	0·380	4916	4753	1·034
(FIG·19) G-H-I	22°	10133	2132	22	34·2°	1042	0·367	5310	3820	1·390
(FIG·20) J-K-L	22°	17342	4821	171	35·3°	1390	0·515	8632	6657	1·297
(FIG·21) M-N-O	22°	15645	5239	0	36·1°	1759	0·604	7816	5867	1·332
(FIG·22) P-Q	29°	12017	2102	0	49·9°	1907	0·157	10289	5826	1·766
(FIG·23), R-R	35°	7060	1492	0	50·3°	1759	0·126	5391	4049	1·331

ϕ, C given above are the weighted average values

U_2 and U_1 are the resultant forces exerted by water

Table IV Results from Morgenstern Price method

<table>
<tr><th colspan="2" rowspan="2">FAILURE SURFACE</th><th>FACTOR OF SAFETY</th><th>FACTOR OF SAFETY</th></tr>
<tr><th>GROUNDWATER TABLE AT 425 FEET BELOW THE SURFACE.</th><th>GROUNDWATER TABLE LOWERED TO 1005 FEET.</th></tr>
<tr><td rowspan="2">SEE FIG·25</td><td>A – B – C</td><td>0·54</td><td>1·18</td></tr>
<tr><td>M – N – O</td><td>1·41</td><td>1·81</td></tr>
<tr><td rowspan="2">SEE FIG·26</td><td>P – Q</td><td>1·77</td><td>—</td></tr>
<tr><td>R – R'</td><td>1·35</td><td>—</td></tr>
<tr><td rowspan="2">SEE FIG·27</td><td>$C_1 - C_2 - C_3 - C_4$</td><td>3·90</td><td>—</td></tr>
<tr><td>$C_1 - C_2 - C_3 - C_5$</td><td>3·34</td><td>—</td></tr>
</table>

Table V Summary of the results from the two methods

FAILURE SURFACE	Factor of safety using wedge analysis	Factor of safety using Morgenstern Price method	
		groundwater table at 425 feet	groundwater table lowered to 1005 ft.
A-B-C (SEE FIG·17,25)	0·49	0·54	1·18
D-E-F (SEE FIG·18)	1·03	—	—
G-H-I (SEE FIG·19)	1·39	—	—
J-K-L (SEE FIG·20)	1·30	—	—
M N O (SEE FIG·21,25)	1·33	1·41	1·81
P-Q (SEE FIG·22,26)	1·77	1·77	—
R-R′ (SEE FIG·23,26)	1·33	1·35	—
C_1-C_2-C_3-C_4 (SEE FIG·27)	—	3·90	—
C_1-C_2-C_3-C_5 (SEE FIG·27)	—	3·34	—

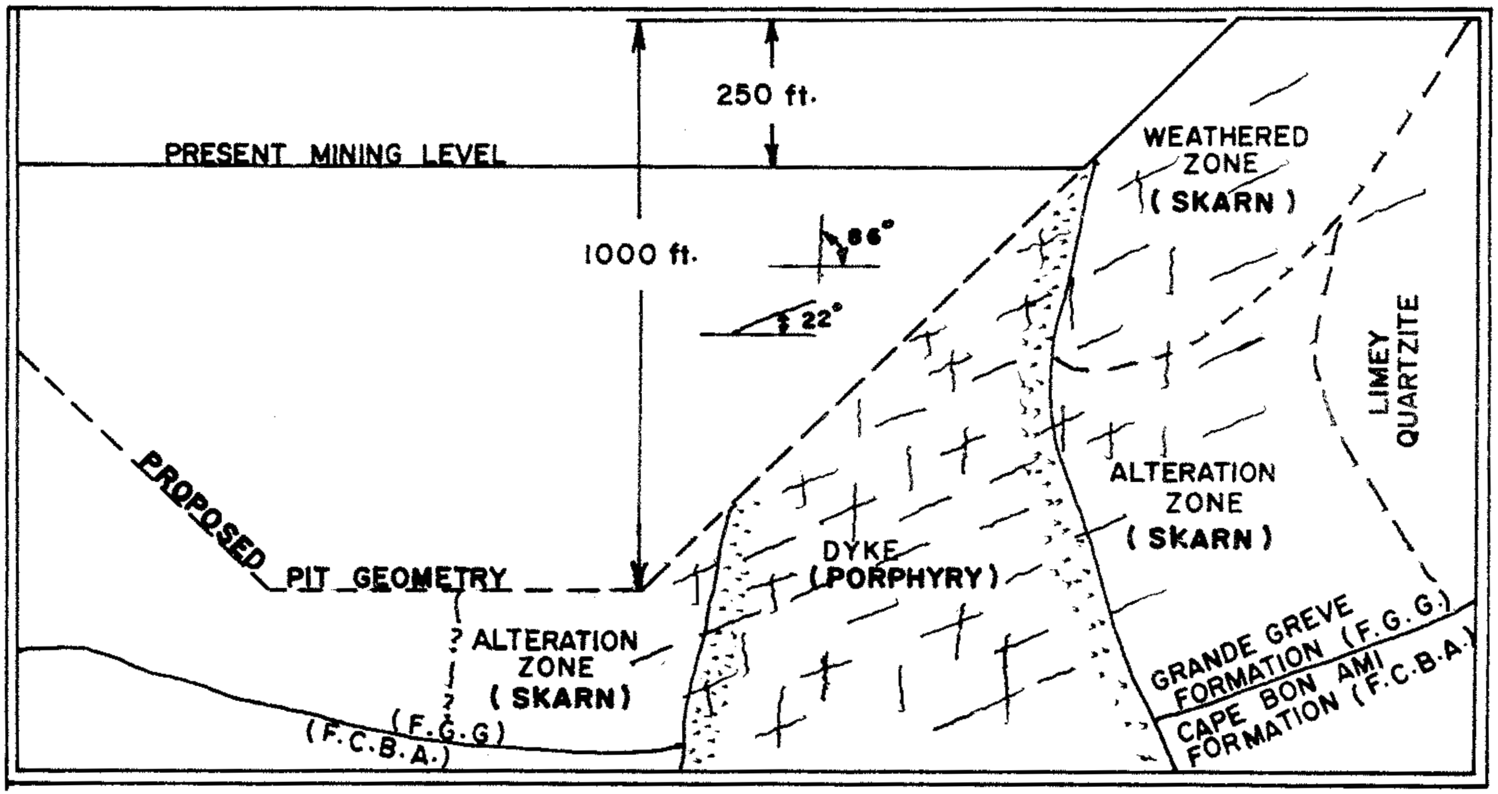

Fig. 16a Profile of the pit showing the major rock types and the structure discontinuities.

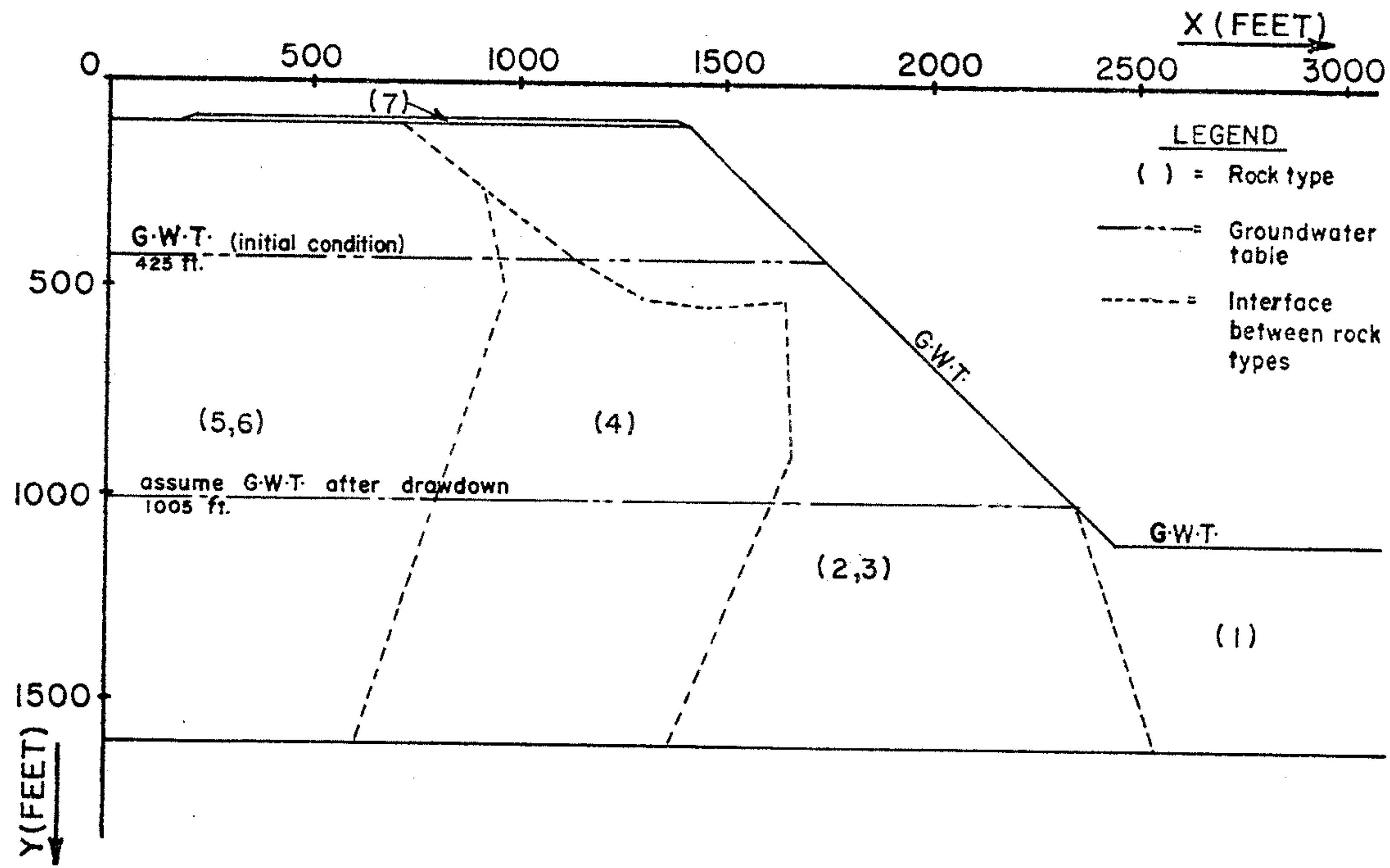

Fig. 16b Profile of the pit showing groundwater condition.

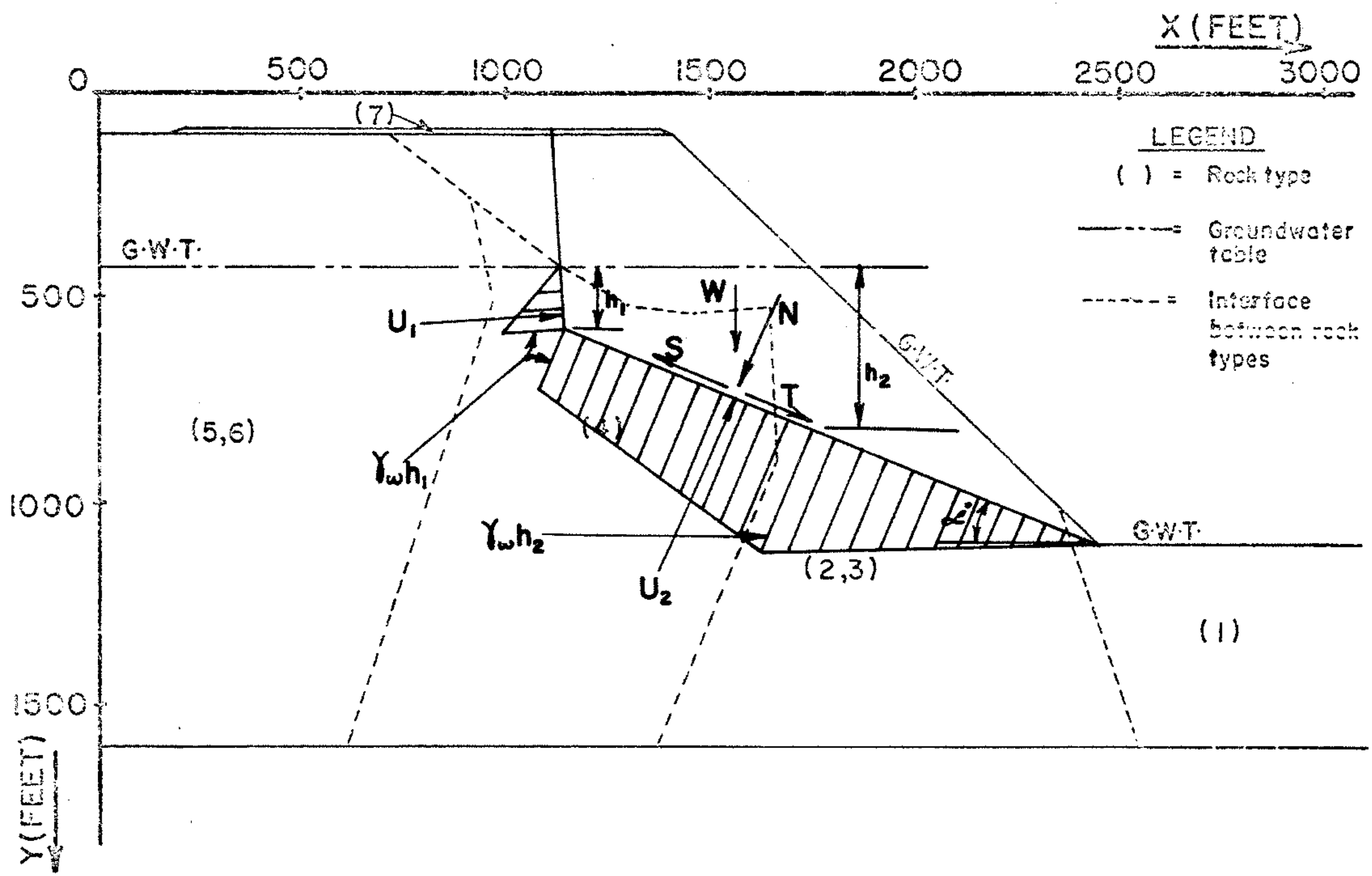

Fig. 16c Force diagram for wedge analysis.

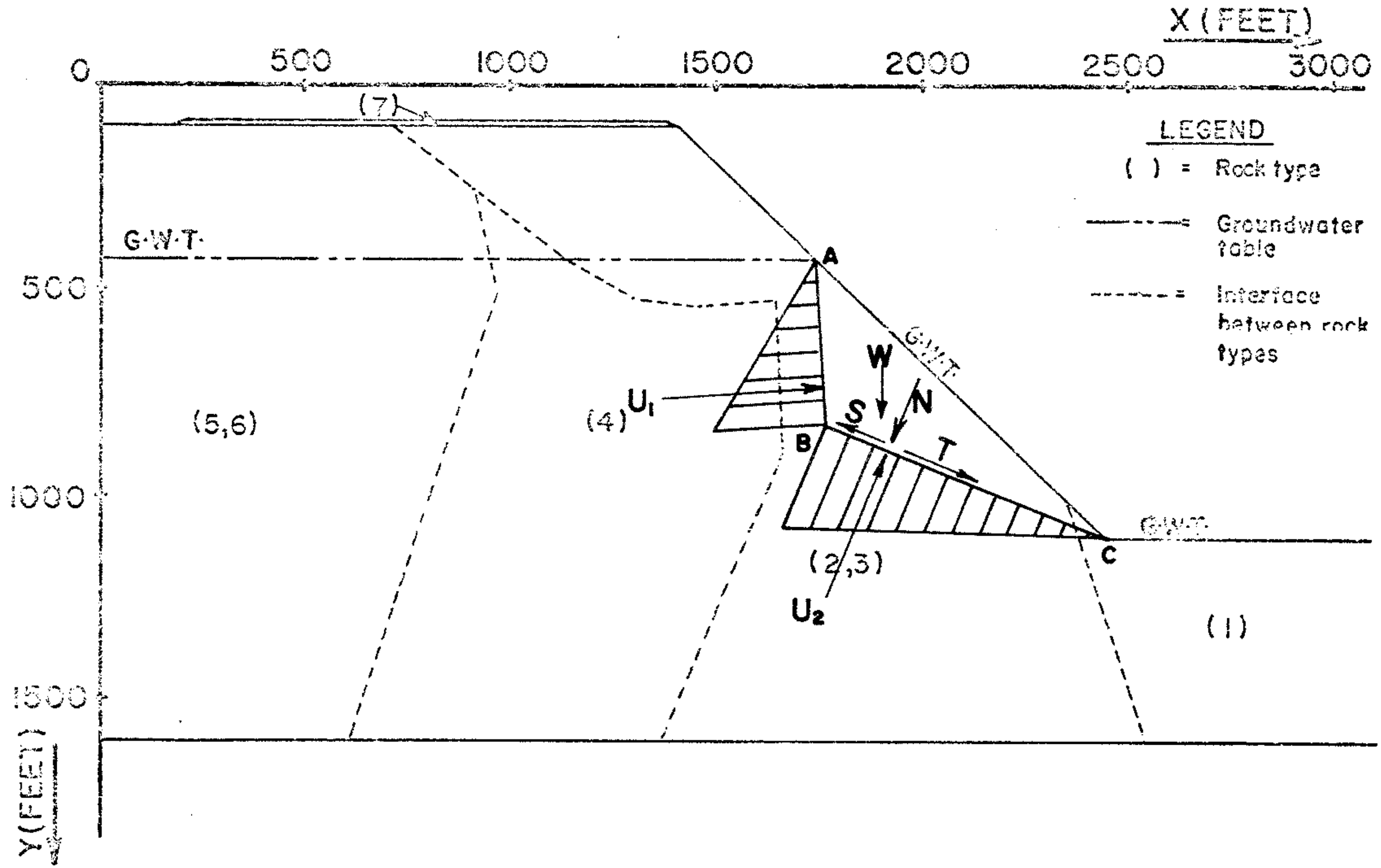

Fig. 17 Diagram showing the forces acting on wedge A-B-C

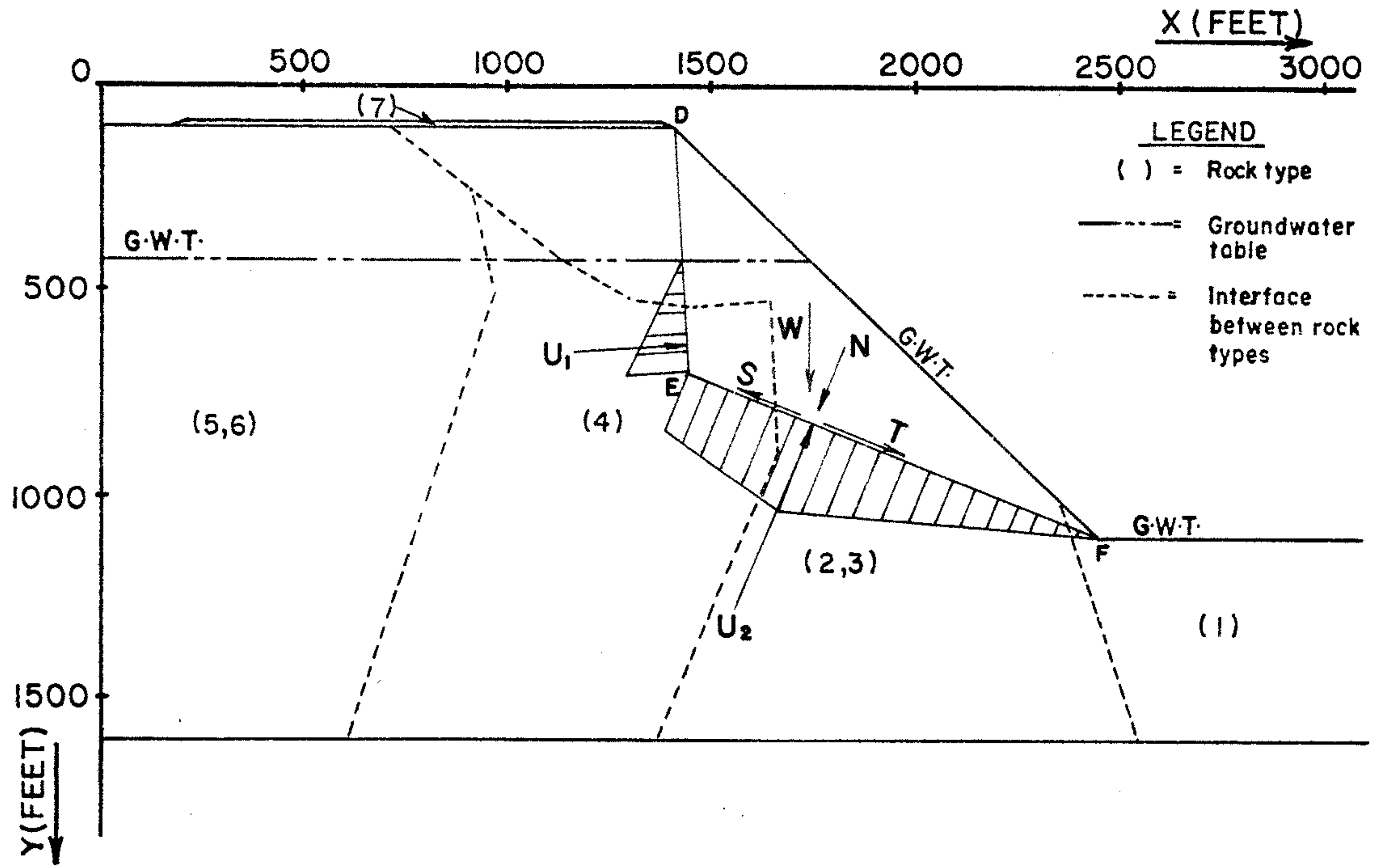

Fig. 18 Diagram showing the forces acting on wedge D-E-F

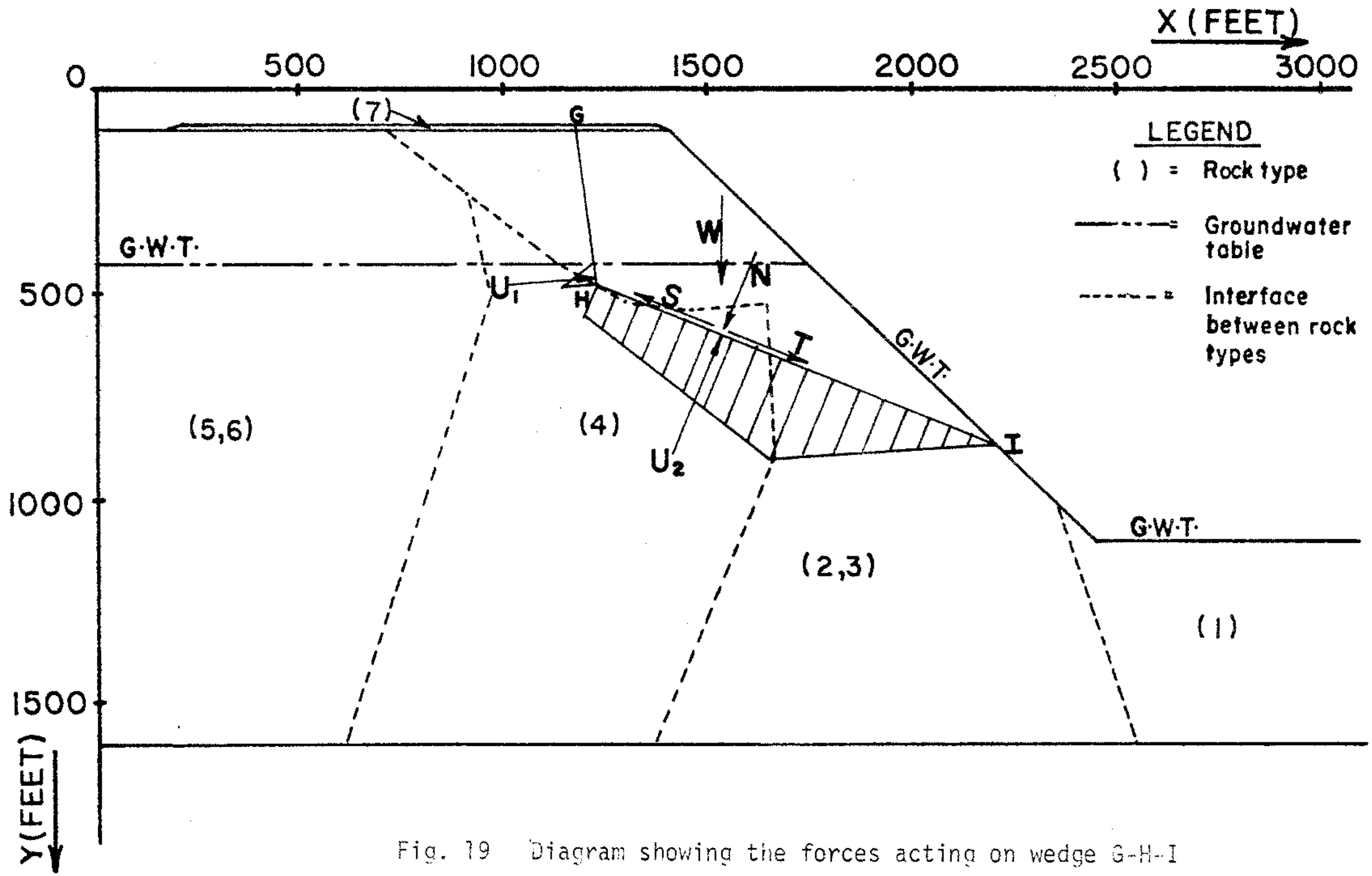

Fig. 19 Diagram showing the forces acting on wedge G-H-I

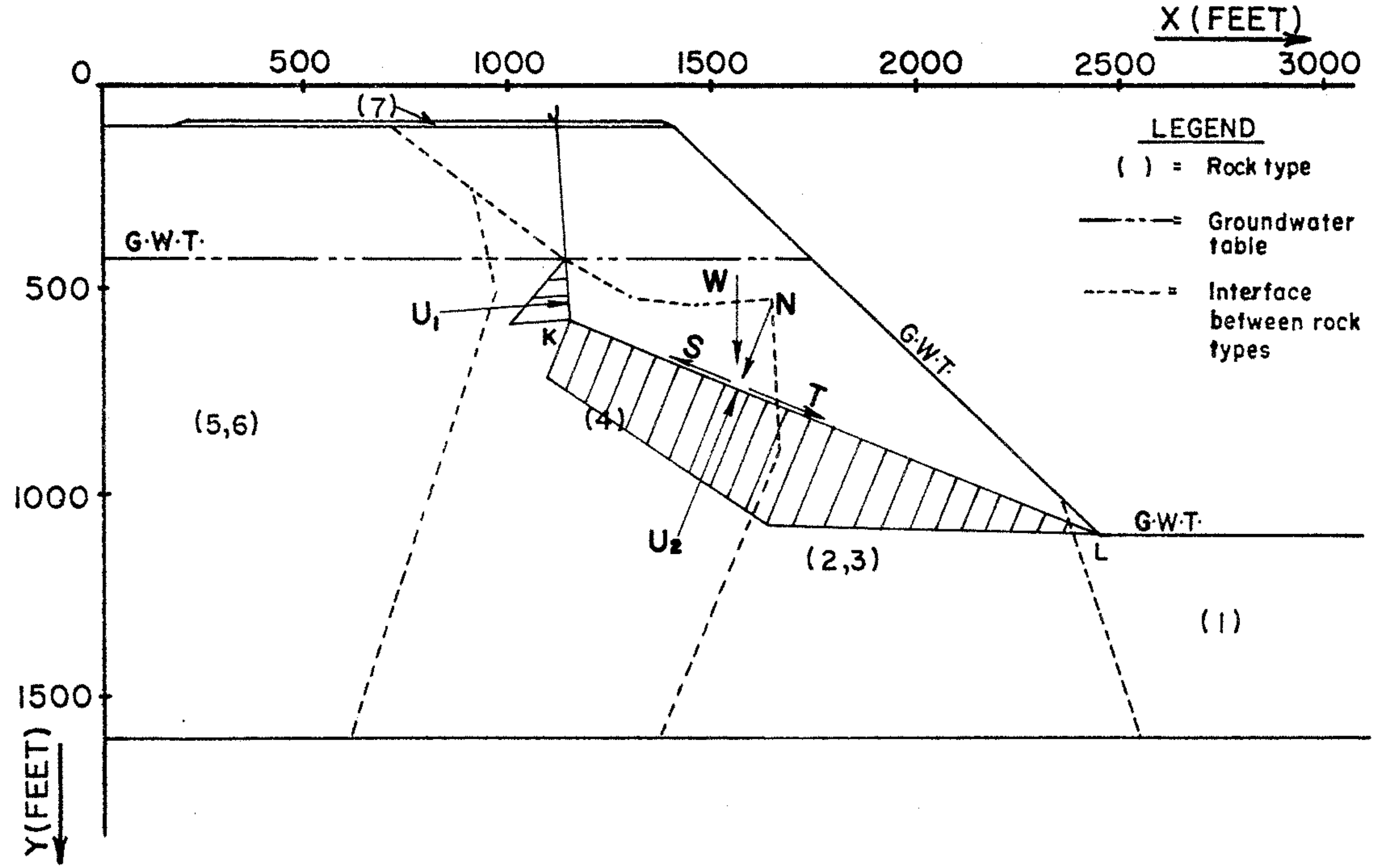

Fig. 20 Diagram showing forces acting on wedge J-K-L

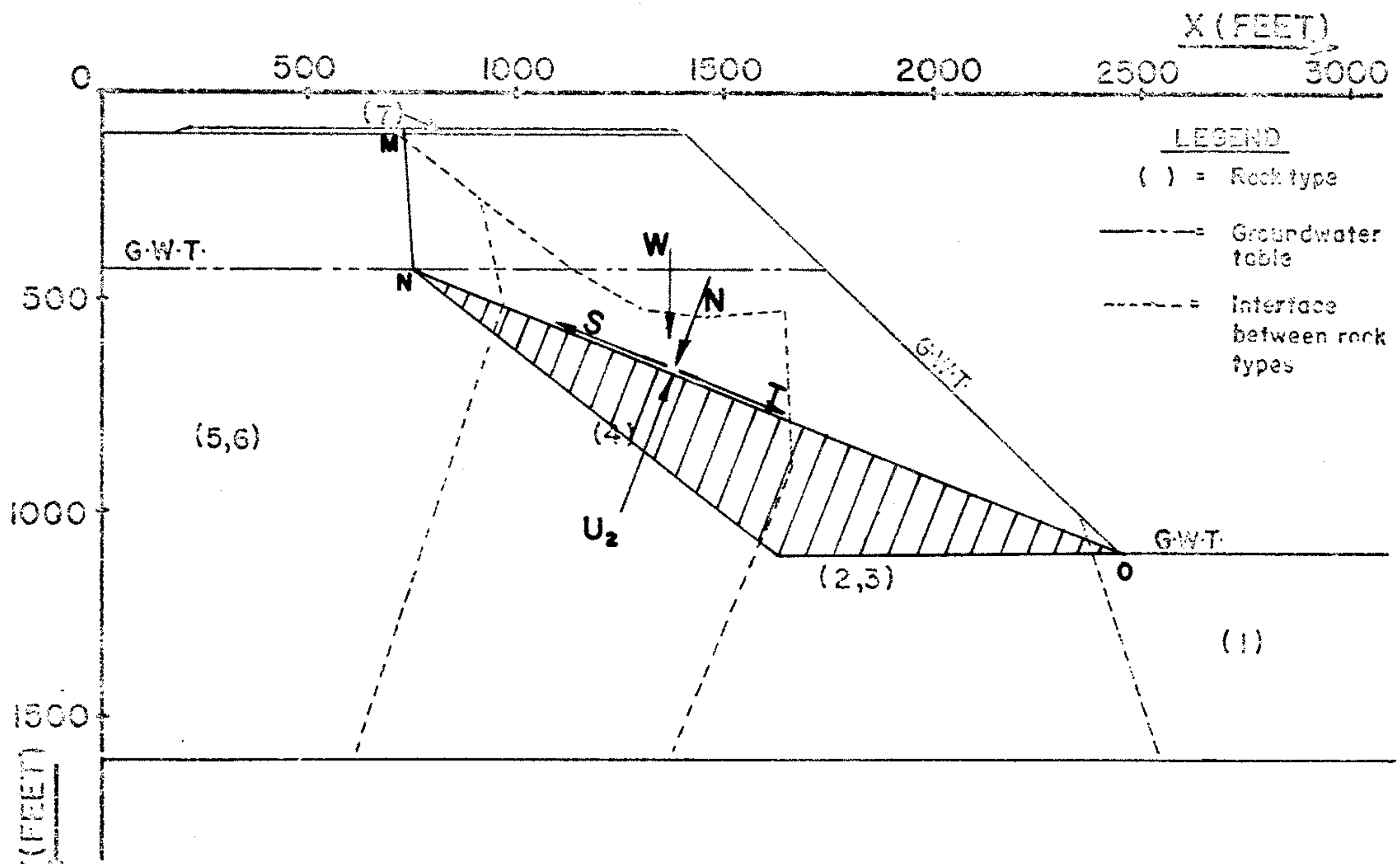

Fig. 21 Diagram showing the forces acting on wedge M-N-O

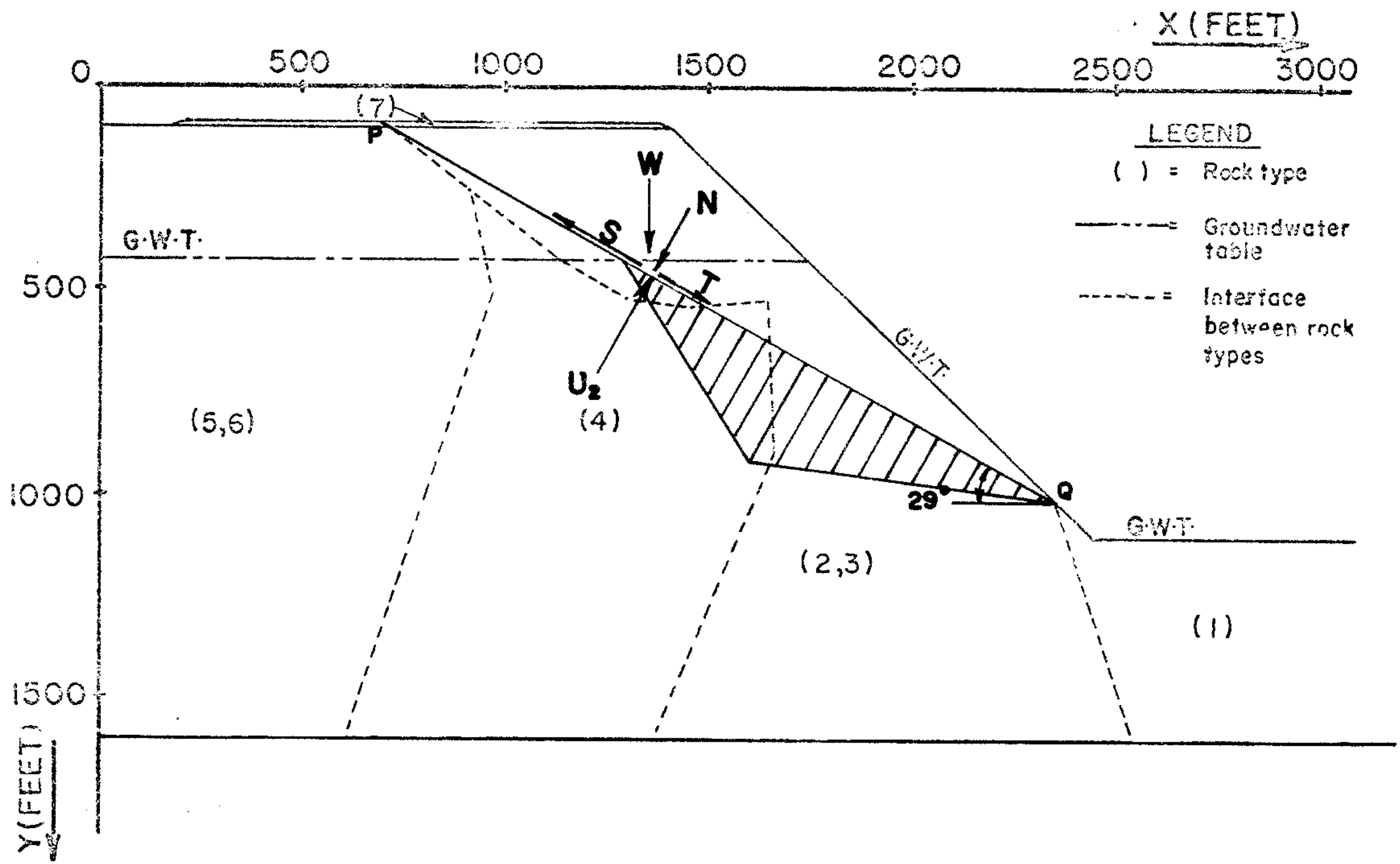

Fig. 22 Diagram showing the forces acting on wedge P - Q'

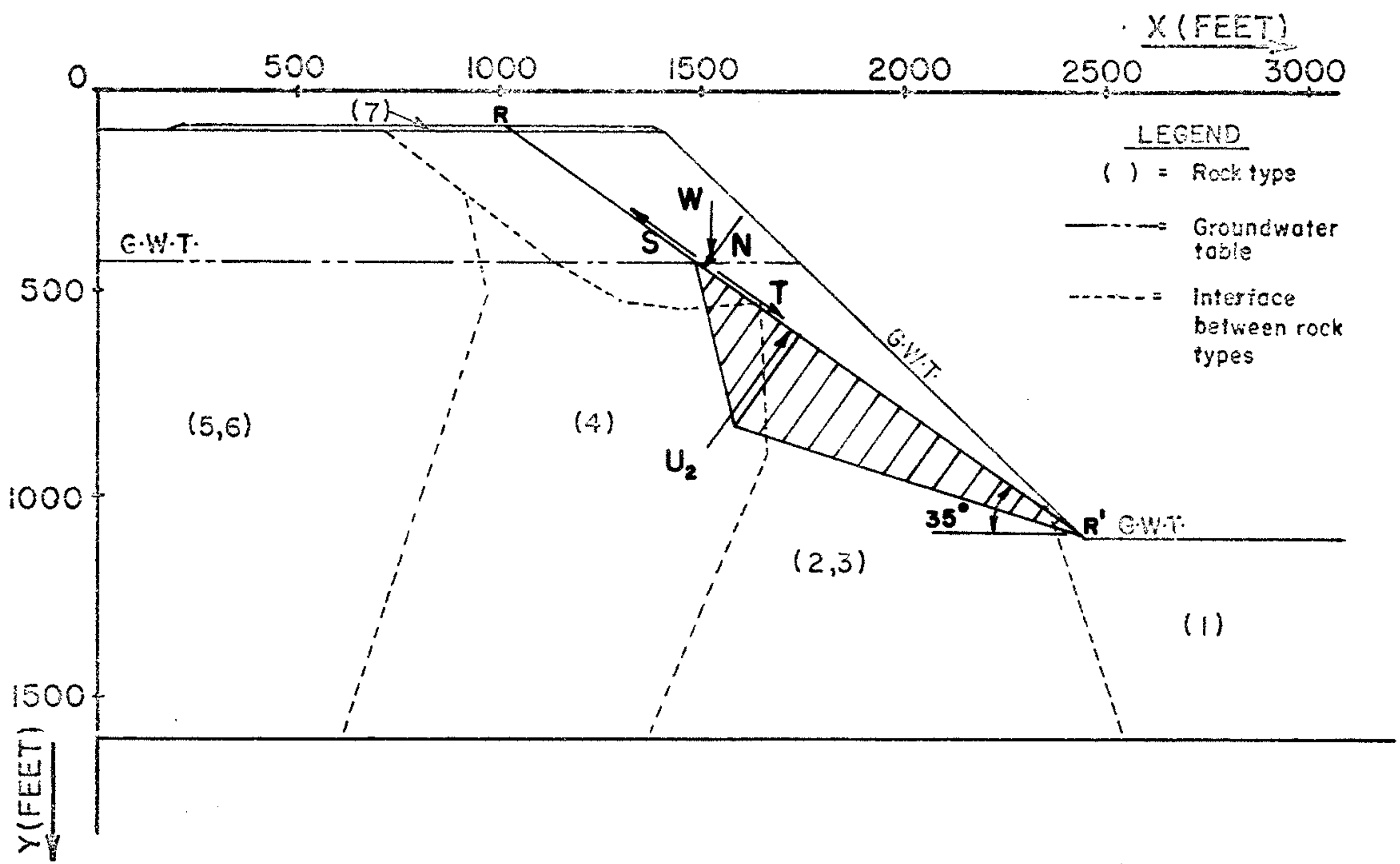

Fig. 23 Diagram showing the forces acting on wedge R - R'

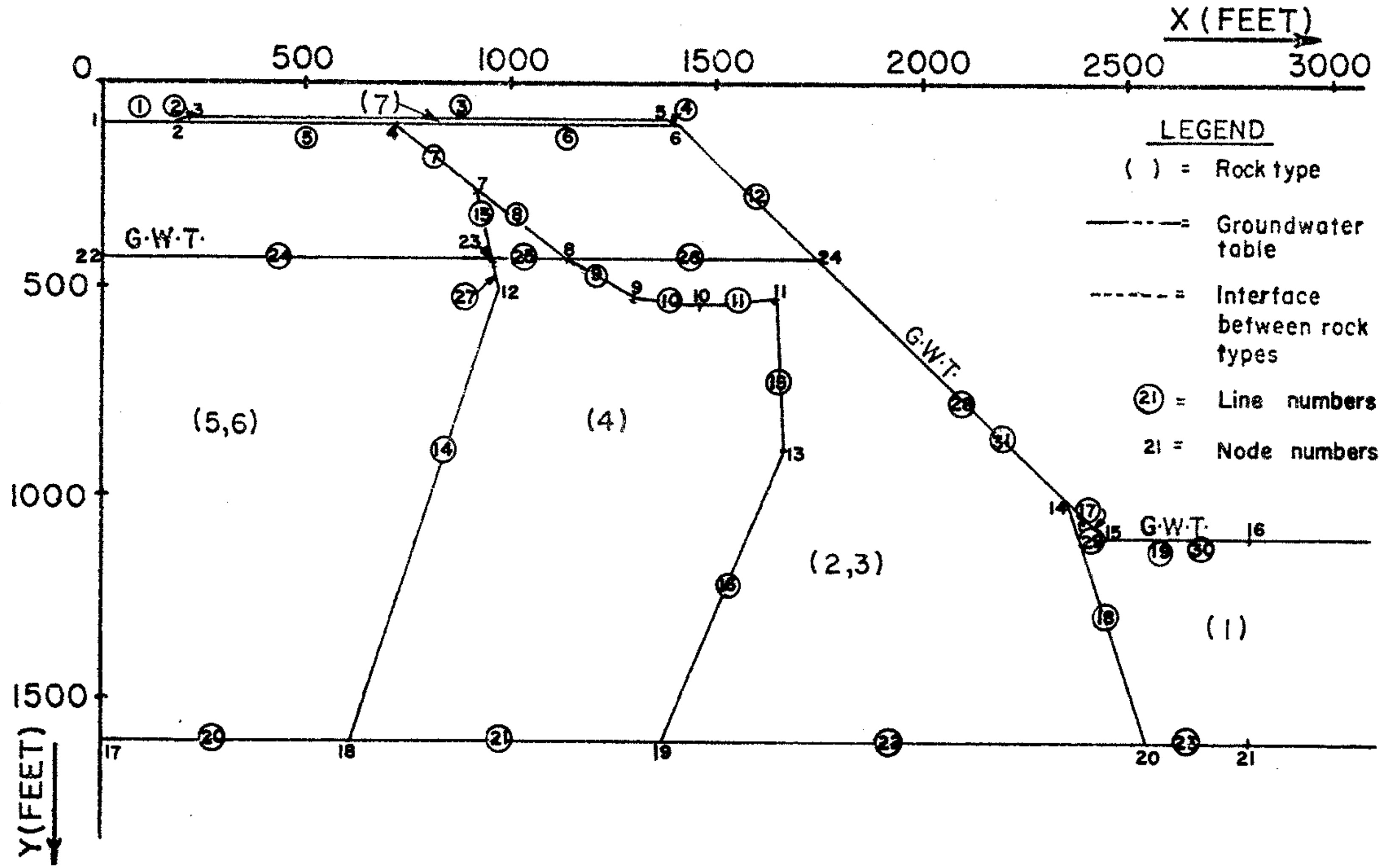

Fig. 24 Input data for Morgenstern-Price Method

LINE DATA		
POINT- I	POINT- J	SOIL TYPE
1	2	6
2	3	7
3	5	7
5	6	7
2	4	6
4	6	3
4	7	6
7	8	4
8	9	4
9	10	4
10	11	4
6	24	3
7	23	6
12	18	4
11	13	4
13	19	3
14	15	1
14	20	3
15	16	1
17	18	6
18	19	4
19	20	3
20	21	1
22	23	0

LINE DATA (continued)

1	6	11
23	8	0
8	24	0
23	12	6
24	14	3
14	15	0
15	16	0
24	14	0
374	423	106

SOIL DATA						
TYPE	γ (WET) PCF	γ (SAT) PCF	C' PSF	ϕ' DEGREE	R_u	EQ
1	165.0	170.0	4300.0	39.0	-1.0	0.0
2	165.	170.	409.	32.	-1.	0.
3	165.	170.	327.	51.	-1.	0.
4	165.	170.	4300.	39.	-1.	0.
5	165.	170.	3960.	39.	-1.	0.
6	165.	170.	5440.	39.	-1.	0.
7	117.	117.	0.	0.	-1.	0.
28	1107.	1137.	18736.	239.	-7.	0.

N	PROJECT NAME
9	GASPE PIT STABILITY ANALYSIS IA

N Is A Multiple Of 4 Of The Number Of Characters Under "PROJECT".

EARTHQUAKE ANALYSIS	0 MEANS NO	1 MEANS YES
0		

NO. OF POINTS	NO. OF LINES	NO. OF SOILS
24	31	7

NOTE- Decimals Points Need Not Be Punched.

WATER LEVEL (LOW X)	WATER LEVEL (HIGH X)	X CO-ORD OF ℄
425.00	1100.00	0.0

POINT DATA

POINT	NO. OF LINE	X CO-ORD	Y CO-ORD
1	1	0.0	100.0
2	3	177.5	100.
3	2	202.5	90.
4	3	715	100.
5	2	1375.	90.
6	3	1400.	100.
7	3	910.	255.
8	4	1136	425.
9	2	1300.	520.
10	2	1460.	535.
11	2	1645.	520.
12	2	970.	505.
13	2	1670.	890.

POINT DATA (continued)

POINT	NO. OF LINE	X CO ORD	Y CO ORD
14	5	2350.0	1005.0
15	4	2450.	1100.
16	2	2800.	1100.
17	1	0.	1600.
18	3	610.	1600.
19	3	1370.	1600.
20	3	2545.	1600.
21	1	2800.	1600.
22	1	0.	425.
23	4	954.18	425.
24	4	1741.16	425.
300	62	30581.34	16710.

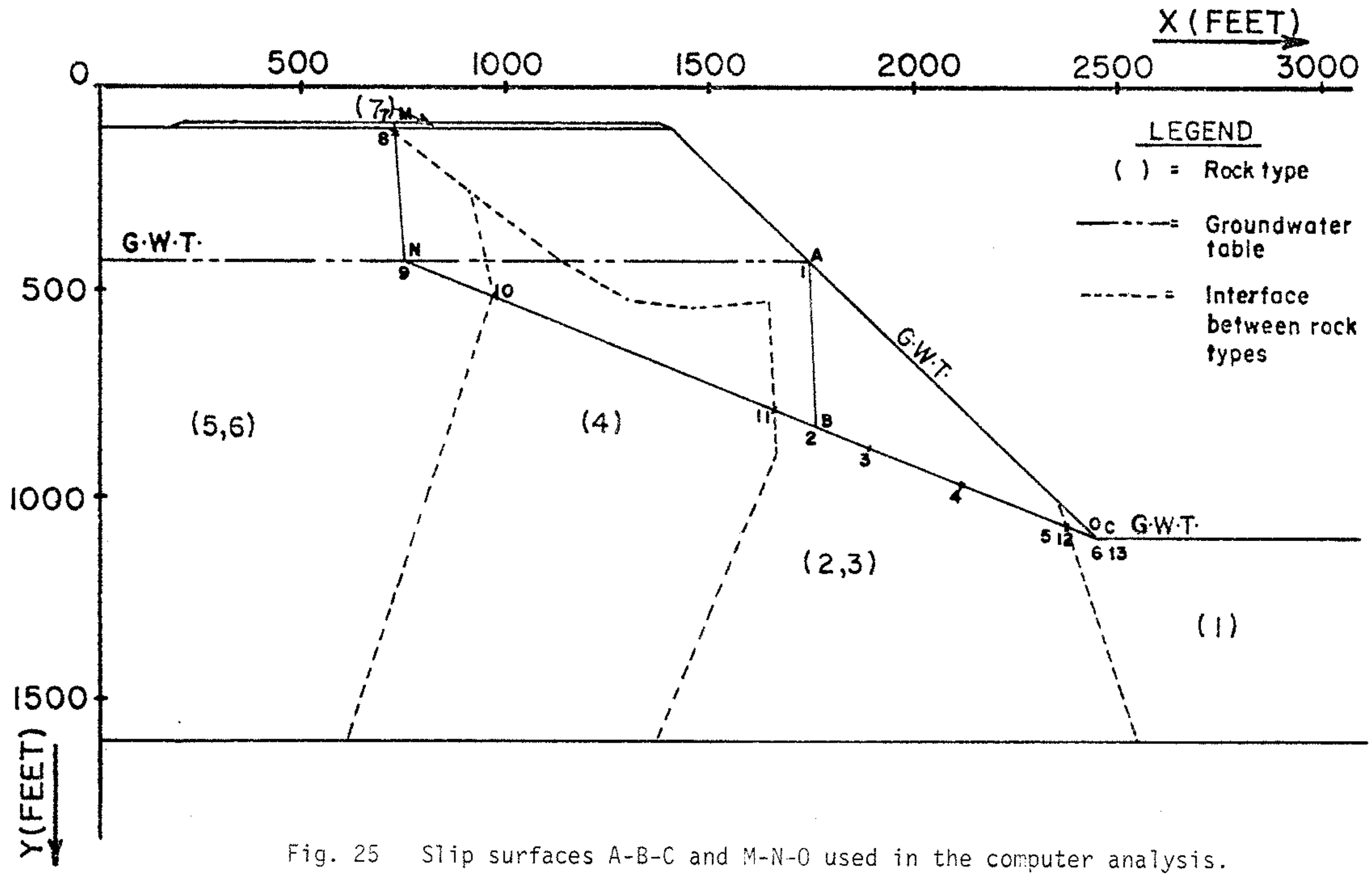

Fig. 25 Slip surfaces A-B-C and M-N-O used in the computer analysis.

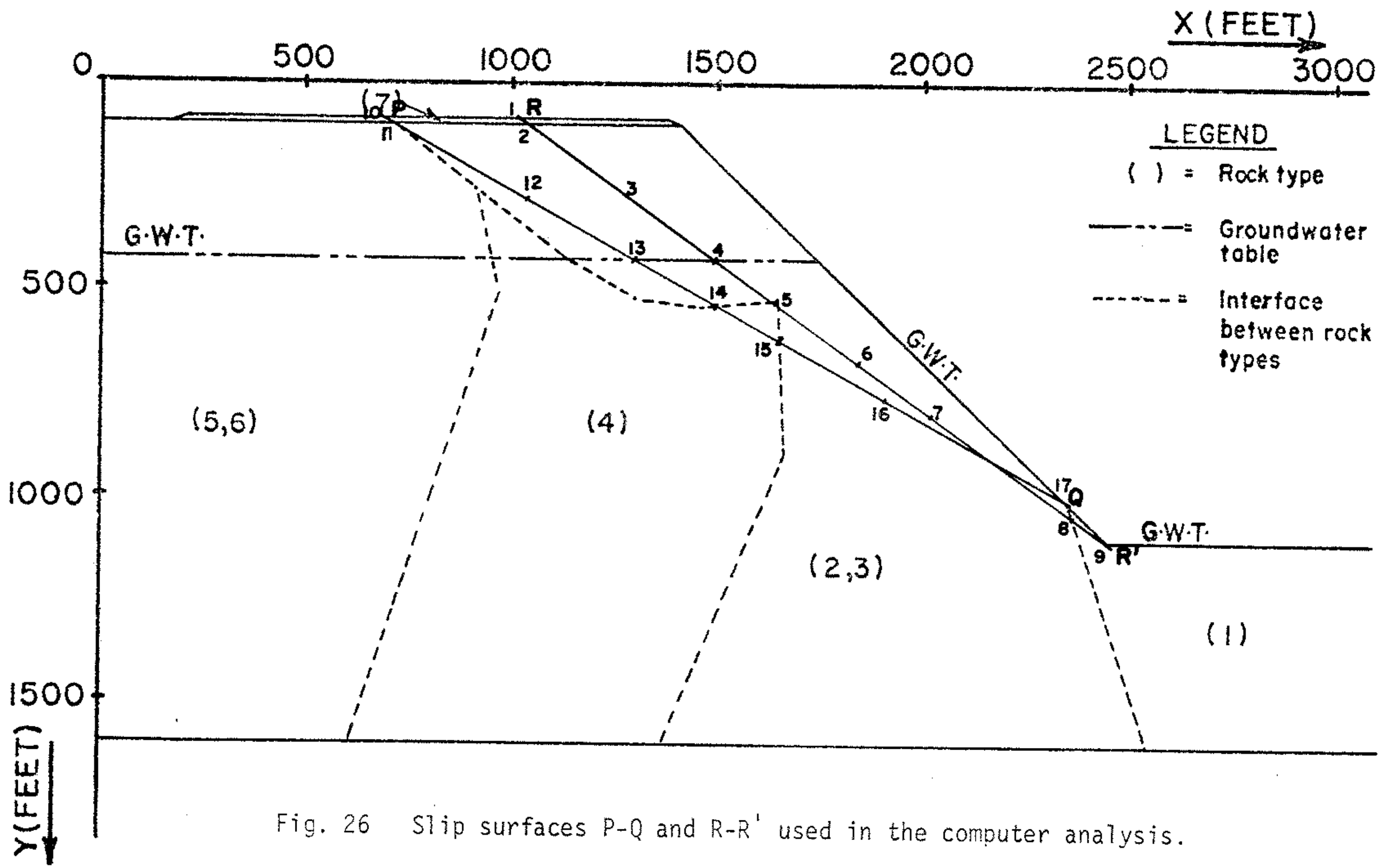

Fig. 26 Slip surfaces P-Q and R-R' used in the computer analysis.

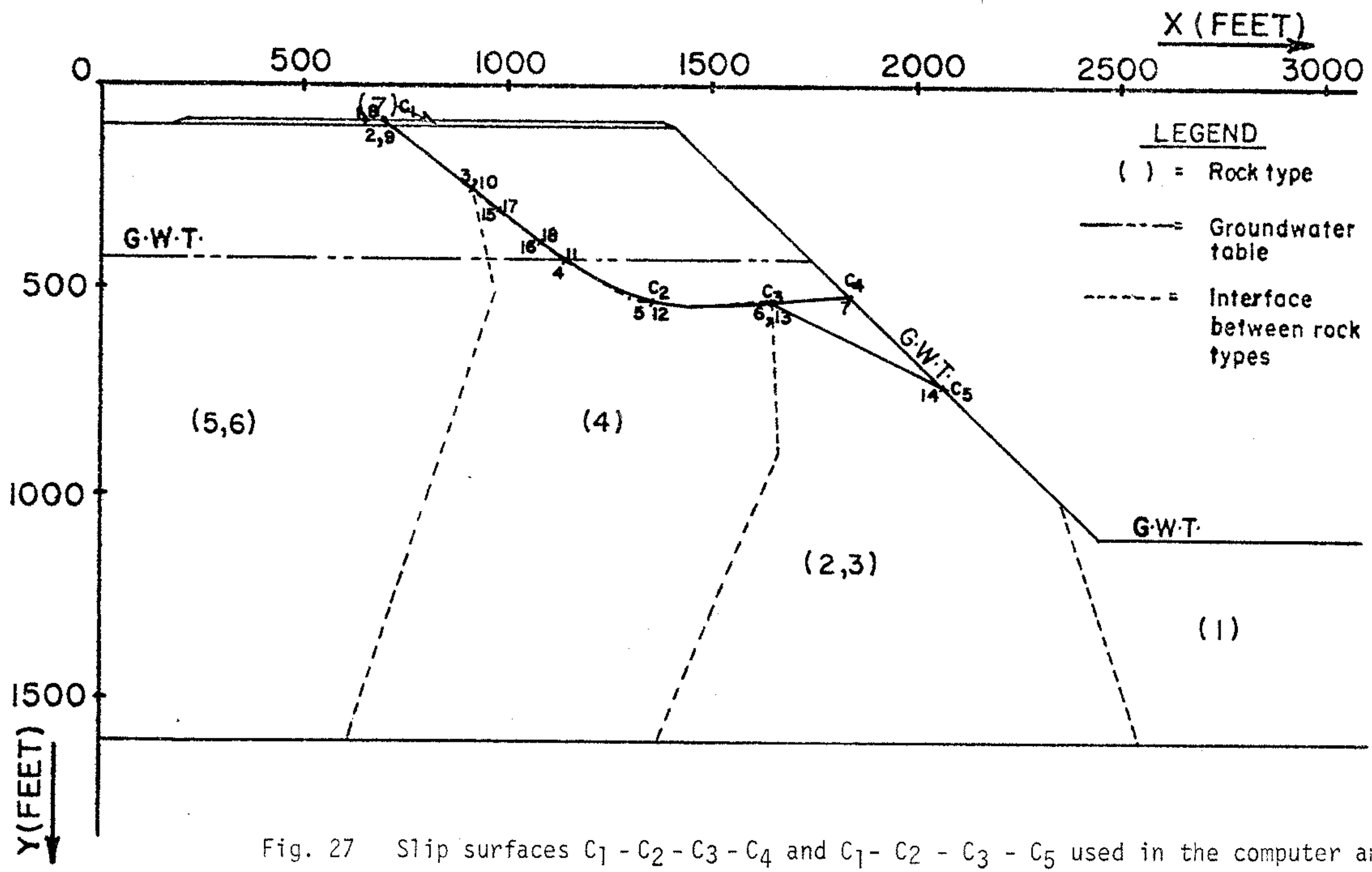

Fig. 27 Slip surfaces $C_1 - C_2 - C_3 - C_4$ and $C_1 - C_2 - C_3 - C_5$ used in the computer analysis

CHAPTER VI

DISCUSSION

Nine possible slip surfaces were investigated using either the wedge analysis or Morgenstern-Price method or both. The factors of safety estimated from the two methods are in reasonable agreement. As was expected, the results obtained from the wedge analysis are lower; i.e., on the safer side. Hence, implications are, for rock slopes where discontinuities play an important role and where the entire rock mass moves as one rigid body along a planar surface as is usually the case, a simple wedge analysis will suffice provided that the pore water pressures are properly accounted for.

CONCLUSION

As seen in Table V, the pit wall is unstable unless the water level would be sufficiently drawn down. Otherwise, rock slide will occur possibly along a plane between the A-B-C and D-E-F surfaces. It is also shown that by lowering the groundwater table from its original level at 425 feet below the surface to 1,005 feet, the stability condition of the slope is considerably improved. Hence, it is therefore advisable that drain holes be drilled all around the pit wall as the development progresses to its final depth.

APPENDIX

SPECIFICATION OF FAILURE SURFACES
****

PROJECT --GASPÉ PIT STABILITY ANALYSIS 1A
(WITHOUT EARTHQUAKE ANALYSIS)

NUMBER OF POINTS --------- 13

NUMBER OF SURFACES ------ 2

NUMBER OF Q.S ----------- 13

POINTS ON FAILURE SURFACE

Q	X CO-ORD	Y CO-ORD	F
1	1745.00	424.00	1.0
2	1765.00	750.00	1.0
3	1895.00	895.00	1.1
4	2110.00	980.00	1.4
5	2370.00	1070.00	1.0
6	2450.00	1099.00	0.5
7	728.00	89.00	1.0
8	730.00	100.00	1.0
9	755.00	320.00	5.0
10	970.00	520.00	8.0
11	1665.00	790.00	1.3
12	2330.00	1085.00	1.4
13	2450.00	1099.00	0.5
14	21963.00	9221.00	24.2

FAILURE SURFACE DATA

SURFACE		Q1	Q2	Q3	Q4	Q5	Q6	Q7	Q8					
1	**A-B-C**	1	-2	2	-2	3	-2	4	-2	5	-1	6	0	0
2	**M-N-O**	7	-7	8	-5	9	-5	10	-4	11	-2	12	-1	13
3		8	-9	10	-7	12	-7	14	-6	16	-3	18	-1	13

SURFACE NUMBER 1. 1 A-B-C

X COORD	E		X FORCE	Y-YT		Y-YT/Y-Z		VERT F		SMALL F	
1765.00	130013.	130013.	120618.	192.683	192.683	0.637	0.637	2.151	2.151	1.000	1.000
1895.00	2421247.	2421247.	2470919.	175.233	175.233	0.542	0.542	1.253	1.253	1.100	1.100
2110.00	3730780.	3730780.	4845675.	100.146	100.146	0.492	0.492	0.965	0.965	1.400	1.400
2350.00	1611854.	1611854.	1541394.	21.594	21.594	0.372	0.372	1.304	1.304	1.031	1.031
2370.00	1523674.	1523674.	1413575.	11.642	11.642	0.253	0.253	1.041	1.041	1.000	1.000
2371.48	1464196.	1464196.	1345845.	11.331	11.331	0.251	0.251	1.025	1.025	0.991	0.991
2381.13	1127179.	1127179.	972957.	9.324	9.324	0.236	0.236	1.113	1.113	0.930	0.930
2448.30	-1743.	-1743.	-826.	0.0	0.0	0.0	0.0	0.0	0.0	0.511	0.511

F = 0.544 LAMBDA = 0.928(11 14 0)

SURFACE NUMBER 2, 1 M-N-O

X COORD	E		FORCE	Y-YT		Y-YT/Y-Z		VERT F		SMALL F	
730.00	4125.	4125.	314.	3.287	3.287	0.329	0.329	0.0	0.0	1.000	1.000
731.49	-39971.	-39971.	-3763.	5.638	5.638	0.244	0.244	6.301	6.301	1.238	1.238
755.00	-163904.	-163904.	-62312.	218.503	218.503	0.950	0.950	-14.406	-14.406	5.000	5.000
774.38	-63324.	-63324.	-25376.	584.544	584.544	2.357	2.357	-39.376	-39.376	5.270	5.270
867.87	578188.	578188.	289052.	-46.552	-46.552	-0.139	-0.139	5.847	5.847	6.575	6.575
910.00	939267.	939267.	511543.	-14.880	-14.880	-0.040	-0.040	4.094	4.094	7.163	7.163
954.18	1349837.	1349837.	798417.	3.129	3.129	0.008	0.008	3.023	3.023	7.779	7.779
966.22	1467299.	1467299.	866643.	6.720	6.720	0.016	0.016	2.809	2.809	7.947	7.947
970.00	1504614.	1504614.	915220.	7.761	7.761	0.018	0.018	2.730	2.730	8.000	8.000
1136.00	-902177.	-902177.	-438998.	0.585	0.585	0.001	0.001	0.400	0.400	6.400	6.400
1300.00	-3369304.	-3369304.	-1234472.	-2.578	-2.578	-0.005	-0.005	2.486	2.486	4.819	4.819
1370.00	-4448120.	-4448120.	-1401505.	1.057	1.057	0.002	0.002	2.953	2.953	4.144	4.144
1375.00	-4525752.	-4525752.	-1409379.	1.411	1.411	0.002	0.002	2.991	2.991	4.096	4.096
1400.00	-4912722.	-4912722.	-1439962.	3.374	3.374	0.006	0.006	3.255	3.255	3.855	3.855
1460.00	-5798983.	-5798983.	-1444577.	9.405	9.405	0.017	0.017	3.809	3.809	3.276	3.276
1581.27	-7355986.	-7355986.	-1178577.	27.616	27.616	0.057	0.057	5.518	5.518	2.107	2.107
1645.00	-8052114.	-8052114.	-913953.	40.607	40.607	0.090	0.090	7.391	7.391	1.493	1.493
1663.20	-8235771.	-8235771.	-824954.	44.770	44.770	0.102	0.102	12.155	12.155	1.317	1.317
1665.00	-8253619.	-8253619.	-815628.	45.194	45.194	0.103	0.103	12.318	12.318	1.300	1.300
1670.00	-8254935.	-8254935.	-816430.	46.910	46.910	0.108	0.108	12.312	12.312	1.301	1.301
1741.16	-8273822.	-8273822.	-825029.	71.275	71.275	0.179	0.179	12.226	12.226	1.311	1.311
2330.00	276163.	276163.	29397.	-12.566	-12.566	-0.127	-0.127	12.703	12.703	1.400	1.400
2350.00	239559.	239559.	22768.	6.322	6.322	0.077	0.077	14.175	14.175	1.250	1.250
2373.06	180744.	180744.	14287.	32.506	32.506	0.551	0.551	27.988	27.988	1.040	1.040
2381.13	173798.	173798.	13433.	35.673	35.673	0.633	0.633	28.528	25.528	1.016	1.016
2445.80	-2868.	-2868.	-111.	0.0	0.0	0.0	0.0	0.0	0.0	0.509	0.509

F = 1.411 LAMEDA = 0.076(10 16 0)

18:29.38 2.862 RC=0

SPECIFICATION OF FAILURE SURFACES
****

PROJECT --GASPE PIT STABILITY ANALYSIS 1A
(WITHOUT EARTHQUAKE ANALYSIS)

NUMBER OF POINTS -------- 1[illegible]

NUMBER OF SURFACES ------ 2

NUMBER OF Q,S ----------- 13

PCINIS ON FAILURE SUFFACE

Q	X CO-ORD	Y CO-ORD	F
1	1745.[illegible]	424.[illegible]	1.[illegible]
2	1765.[illegible]	75[illegible].[illegible]	1.[illegible]
3	1895.[illegible]	895.[illegible]	1.[illegible]
4	2[illegible]1[illegible].[illegible]	98[illegible].[illegible]	1.4
5	237[illegible].[illegible]	1[illegible].[illegible]	1.2
6	245[illegible].[illegible]	[illegible]99.[illegible]	1.[illegible]
7	728.[illegible]	[illegible].[illegible]	[illegible].5
8	73[illegible].[illegible]	1[illegible].[illegible]	[illegible].5
9	755.[illegible]	32[illegible].[illegible]	1.2
1[illegible]	97[illegible].[illegible]	52[illegible].[illegible]	1.3
11	1665.[illegible]	79[illegible].[illegible]	1.[illegible]
12	233[illegible].[illegible]	[illegible]85.[illegible]	1.3
13	245[illegible].[illegible]	1[illegible]9[illegible].[illegible]	2.1
14	21963.[illegible]	9221.[illegible]	1[illegible].6

BY LOWERING GROUNDWATER TABLE

FAILURE SURFACE DATA

SURFACE		Q1	Q2	Q3	Q4	Q5	Q6	Q7	Q8					
1	**A-B-C**	1	-2	2	-[illegible]	3	-2	4	-2	5	-1	6	[illegible]	[illegible]
2	**M-N-O**	7	-7	8	-5	9	-5	1[illegible]	-4	11	-2	12	-1	13
3		8	-8	1[illegible]	-7	12	-7	14	-6	16	-3	18	-1	13

LOWER S.W.T.

SURFACE NUMBEF 1, 1 **A-B-C**

X COORD	E		X FORCE	Y-YI		Y-YI/Y-Z		VERT F		SHELL F	
1765.00	335809.	335809.	169893.	87.712	87.712	0.290	0.290	3.023	3.023	1.100	1.100
1895.00	2323582.	2323582.	1293113.	55.455	55.455	0.171	0.171	2.301	2.301	1.193	1.193
2110.00	1151475.	1151475.	815577.	39.499	39.499	0.194	0.194	1.825	1.825	1.430	1.430
2182.22	741210.	741210.	504159.	29.639	29.639	0.194	0.194	1.919	1.919	1.344	1.344
2350.00	199515.	199515.	122680.	0.320	0.320	0.006	0.006	2.163	2.163	1.215	1.215
2370.00	240835.	240835.	146012.	2.199	2.199	0.046	0.046	2.639	2.639	1.200	1.200
2371.48	237390.	237390.	143677.	2.151	2.151	0.049	0.049	2.659	2.659	1.196	1.196
2381.13	213810.	213810.	126794.	2.372	2.372	0.160	0.160	2.714	2.714	1.172	1.172
2448.30	-1.	-1.	-1.	0.0	0.0	0.0	0.0	0.0	0.0	1.004	1.004

F = 1.177 LAMBDA = 0.506 (5 7 0)

LOWER G-W-T-

SURFACE NUMBER 2, 1 **M-N-O**

X COORD	B	B	X FORCE	Y-YI	Y-YI	Y-YI/Y-Z	Y-YI/Y-Z	VERT F	VERT F	SMALL F	SMALL F
730.00	2554.	2554.	599.	3.191	3.191	0.319	0.319	0.0	0.0	0.533	0.533
731.49	-29735.	-29735.	-7559.	5.739	5.739	0.248	0.248	2.199	2.198	0.542	0.542
755.00	-24572.	-24572.	-13637.	763.163	763.163	3.319	3.319	-73.264	-73.264	1.200	1.200
773.97	139749.	139749.	79272.	-131.4[illegible]	-131.4[illegible]	-0.531	-0.531	14.795	14.795	1.209	1.209
8[illegible]5.[illegible]	444165.	444165.	254969.	-34.182	-34.182	-0.123	-0.123	5.873	5.873	1.223	1.223
91[illegible].[illegible]	1773131.	1773131.	1058425.	13.155	13.155	0.135	0.135	2.738	2.738	1.272	1.272
966.21	2673786.	2673786.	1628943.	23.921	23.921	0.156	0.156	2.258	2.258	1.298	1.298
971.0[illegible]	2739174.	2739174.	1671[illegible].	24.549	24.549	0.157	0.157	2.226	2.226	1.301	1.301
1136.[illegible]	1487972.	1487972.	872693.	-5.171	-5.171	-0.[illegible]	-0.[illegible]	2.675	2.675	1.228	1.228
13[illegible].[illegible]	563762.	563762.	316241.	-72.856	-72.856	-0.131	-0.131	4.311	4.311	1.158	1.158
137[illegible].[illegible]	58195.	58195.	30734.	-763.235	-763.235	-1.314	-1.314	27.452	27.452	1.127	1.127
1375.[illegible]	21397.	21397.	11293.	-2174.571	-2174.571	-3.532	-3.532	71.693	71.693	1.125	1.125
14[illegible].[illegible]	-152593.	-152593.	-85[illegible]23.	271.421	271.421	0.462	0.462	-7.519	-7.519	1.114	1.114
146[illegible].[illegible]	-595782.	-595782.	-314324.	69.191	69.191	0.125	0.125	-1.731	-1.731	1.088	1.088
1581.68	-1409530.	-1409530.	-685698.	19.922	19.922	0.041	0.041	0.547	0.547	1.037	1.037
162[illegible].[illegible]	-1664328.	-1664328.	-796134.	13.412	13.412	0.029	0.029	0.661	0.661	1.019	1.019
1645.30	-1819933.	-1819933.	-867417.	11.175	11.175	0.023	0.023	0.714	0.714	1.019	1.019
1663.2[illegible]	-1931261.	-1931261.	-916993.	8.122	8.122	0.019	0.019	2.471	2.471	1.0[illegible]1	1.001
1665.[illegible]	-1942311.	-1942311.	-911425.	7.931	7.931	0.018	0.018	2.475	2.475	1.0[illegible]	1.0[illegible]
1671.[illegible]	-1913611.	-1913611.	-9[illegible]13.	7.917	7.917	0.018	0.019	2.469	2.469	1.012	1.012
2149.66	12151.	12151.	6892.	612.644	612.644	3.211	3.211	11.215	11.215	1.219	1.219
233[illegible].[illegible]	233395.	233395.	173339.	8.973	8.973	0.091	0.091	2.319	2.2[illegible]	1.3[illegible]	1.3[illegible]
235[illegible].[illegible]	159465.	159465.	1[illegible]569.	1.866	1.866	0.122	0.122	2.[illegible]93	2.[illegible]93	1.437	1.433
2378.36	143112.	143112.	114914.	13.512	13.512	0.223	0.223	3.395	3.395	1.621	1.621
2381.13	147364.	147364.	119172.	13.923	13.923	0.247	0.247	3.295	3.295	1.641	1.641
2448.80	272.	272.	147.	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	2.092	2.092

F = 1.5[illegible] LAMBDA = .448 ([illegible])

20:57.32 2.278 SC=

SPECIFICATION OF FAILURE SURFACES
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PROJECT --GASPE PII STABILITY ANALYSIS 1A
(WITHOUT EARTHQUAKE ANALYSIS)

NUMBER OF POINTS --------- 17

NUMBER OF SURFACES ------ 2

NUMBER OF Q,S ----------- 17

POINTS ON FAILURE SURFACE

Q	X CO-ORD	Y CO-ORD	F
1	1010.00	89.00	1.0
2	1030.00	100.00	1.0
3	1280.00	280.00	1.0
4	1495.00	425.00	1.0
5	1640.00	530.00	1.0
6	1845.00	680.00	1.0
7	2015.00	805.00	1.0
8	2365.00	1040.00	1.2
9	2450.00	1099.00	1.0
10	690.00	89.00	1.0
11	710.00	100.00	1.3
12	1030.00	290.00	1.5
13	1300.00	425.00	1.0
14	1495.00	545.00	1.0
15	1650.00	630.00	1.0
16	1910.00	765.00	1.0
17	2365.00	1000.00	1.0
18	26280.00	6892.00	16.0

FAILURE SURFACE DATA

SURFACE	Q1	Q2	Q3	Q4	Q5	Q6	Q7	Q8									
1 R-R'	1	-7	2	-3	3	-3	4	-3	5	-3	6	-3	7	-3	8	-1	9
2 P-Q	10	-7	11	-3	12	-3	13	-3	14	-4	15	-3	16	-3	17	0	0
3	11	-14	13	-6	15	-6	17	-6	19	-7	21	-6	23	-6	25	-1	9

SURFACE NUMBER 1, 1 **R-R'**

X COORD	E		X FORCE	Y-YT		Y-YT/Y-Z		VERT F		SMALL F	
1030.00	4485.	4485.	2482.	-0.021	-0.021	-0.002	-0.002	0.0	0.0	1.000	1.000
1030.00	4485.	4485.	2482.	-0.021	-0.021	-0.002	-0.002	1.000	1.000	1.000	1.000
1136.00	-119900.	-119900.	-66365.	6.442	6.442	0.075	0.075	1.597	1.597	1.000	1.000
1280.00	-557046.	-557046.	-308327.	14.846	14.846	0.076	0.076	1.923	1.923	1.000	1.000
1300.00	-665278.	-665278.	-368234.	14.651	14.651	0.072	0.072	1.951	1.951	1.000	1.000
1370.00	-1100800.	-1100800.	-609296.	15.587	15.587	0.062	0.062	2.013	2.013	1.000	1.000
1375.00	-1135284.	-1135284.	-626383.	15.709	15.709	0.062	0.062	2.016	2.016	1.000	1.000
1400.00	-1312090.	-1312090.	-726246.	16.410	16.410	0.063	0.063	2.114	2.114	1.000	1.000
1460.00	-1730879.	-1730879.	-958048.	18.826	18.826	0.077	0.077	2.148	2.148	1.000	1.000
1495.00	-1962827.	-1962827.	-1086431.	20.585	20.585	0.088	0.088	2.160	2.160	1.000	1.000
1495.00	-1962827.	-1962827.	-1086431.	20.585	20.585	0.088	0.088	2.160	2.160	1.000	1.000
1581.27	-2355313.	-2355313.	-1303674.	30.686	30.686	0.143	0.143	2.177	2.177	1.000	1.000
1628.08	-2538122.	-2538122.	-1404859.	36.183	36.183	0.177	0.177	2.184	2.184	1.000	1.000
1640.00	-2581258.	-2581258.	-1428735.	37.594	37.594	0.187	0.187	2.122	2.122	1.000	1.000
1645.00	-2598181.	-2598181.	-1438102.	38.238	38.238	0.191	0.191	2.095	2.095	1.000	1.000
1645.97	-2601441.	-2601441.	-1439906.	38.363	38.363	0.192	0.192	2.186	2.186	1.000	1.000
1670.00	-2679370.	-2679370.	-1483040.	41.467	41.467	0.213	0.213	2.188	2.188	1.000	1.000
1741.16	-2879177.	-2879177.	-1593634.	50.848	50.848	0.284	0.284	2.194	2.194	1.000	1.000
1845.00	-2021063.	-2021063.	-1118665.	51.841	51.841	0.332	0.332	2.185	2.185	1.000	1.000
2015.00	-869827.	-869827.	-481452.	60.209	60.209	0.505	0.505	2.150	2.150	1.000	1.000
2350.00	192251.	192251.	126782.	-6.291	-6.291	-0.252	-0.252	1.937	1.937	1.191	1.191
2360.47	201589.	201589.	133608.	-5.045	-5.045	-0.229	-0.229	1.931	1.931	1.197	1.197
2365.00	205173.	205173.	136277.	-4.622	-4.622	-0.223	-0.223	1.874	1.874	1.200	1.200
2381.13	173116.	173116.	111347.	-3.801	-3.801	-0.229	-0.229	1.901	1.901	1.162	1.162
2446.09	-0.	-0.	-0.	0.0	0.0	0.0	0.0	0.0	0.0	1.009	1.009

F = 1.350 LAMBDA = 0.554(9 11 0)

SURFACE NUMBER 2, 1 P-Q

X COORD	Z		X FORCE	Y-YT		Y-YT/Y-Z		VERT F		SMALL F	
710.00	4617.	4617.	2241.	0.553	0.553	0.155	0.155	0.0	0.0	1.300	1.300
710.00	4617.	4617.	2241.	0.553	0.553	0.155	0.155	[illegible]	[illegible]	1.300	1.300
715.00	3153.	3153.	1534.	1.476	1.476	0.114	0.114	10.907	10.907	1.303	1.303
729.76	-2274.	-2274.	-1114.	-2.453	-2.453	-0.113	-0.113	-2.083	-2.083	1.312	1.312
774.38	-28734.	-28734.	-14380.	1.950	1.950	0.040	0.040	1.087	1.087	1.340	1.340
910.00	-202219.	-202219.	-107597.	5.364	5.364	0.042	0.042	1.780	1.780	1.425	1.425
954.18	-289103.	-289103.	-156807.	5.851	5.851	0.038	0.038	1.828	1.828	1.453	1.453
973.00	-323855.	-323855.	-176852.	5.964	5.964	0.036	0.036	1.838	1.838	1.462	1.462
1030.00	-473152.	-473152.	-265016.	6.103	6.103	0.031	0.031	1.860	1.860	1.500	1.500
1136.00	-1108756.	-1108756.	-539732.	1.228	1.228	0.005	0.005	2.289	2.289	1.304	1.304
1300.00	-2348645.	-2348645.	-876962.	9.713	9.713	0.029	0.029	3.087	3.087	1.000	1.000
1370.00	-2600618.	-2600618.	-971046.	24.880	24.880	0.066	0.066	3.096	3.096	1.000	1.000
1375.00	-2619359.	-2619359.	-978044.	25.908	25.908	0.068	0.068	3.096	3.096	1.000	1.000
1400.00	-2713675.	-2713675.	-1013267.	30.952	30.952	0.105	0.105	3.183	3.183	1.000	1.000
1460.00	-2931106.	-2931106.	-1094447.	42.645	42.645	0.116	0.116	3.198	3.198	1.000	1.000
1476.57	-2987691.	-2987691.	-1119575.	45.309	45.309	0.127	0.127	3.212	3.212	1.000	1.000
1495.00	-3048892.	-3048892.	-1138427.	49.305	49.305	0.139	0.139	3.119	3.119	1.000	1.000
1581.27	-2975375.	-2975375.	-1110977.	65.785	65.785	0.216	0.216	2.734	2.734	1.000	1.000
1645.00	-2953916.	-2953916.	-1112964.	77.448	77.448	0.264	0.264	2.418	2.418	1.000	1.000
1650.00	-2953413.	-2953413.	-1112777.	78.336	78.336	0.268	0.268	2.951	2.951	1.000	1.000
1652.52	-2966557.	-2966557.	-1107664.	78.356	78.356	0.269	0.269	3.221	3.221	1.000	1.000
1670.00	-3055236.	-3055236.	-1140796.	78.594	78.594	0.278	0.278	3.226	3.226	1.000	1.000
1741.16	-3371910.	-3371910.	-1259139.	81.121	81.121	0.321	0.321	3.242	3.242	1.000	1.000
1910.00	-1668983.	-1668983.	-623183.	59.060	59.060	0.331	0.331	3.213	3.213	1.000	1.000
2320.77	5.	5.	2.	0.0	0.0	0.0	0.0	0.0	0.0	1.000	1.000

F = 1.766 LAMBDA = 0.373 (4 6 0)
19:52.54 2.746 RC=0

SPECIFICATION OF FAILURE SURFACES
****

PROJECT --GASPE PIT STABILITY ANALYSIS 1A
(WITHOUT EARTHQUAKE ANALYSIS)

NUMBER OF POINTS -------- 18

NUMBER OF SURFACES ------ 2

NUMBER OF Q,S ----------- 11

POINTS ON FAILURE SURFACE

Q	X CO-ORD	Y CO-ORD	F
1	710,00	89,50	1,0
2	715,00	100,00	1,0
3	910,00	254,50	1,0
4	1136,00	424,50	1,0
5	1360,00	535,00	1,1
6	1645,00	519,50	1,2
7	1845,00	520,00	1,0
8	710,00	89,50	1,0
9	715,00	100,00	1,0
10	910,00	254,50	1,0
11	1136,00	424,50	1,0
12	1360,00	535,00	1,0
13	1645,00	519,50	1,0
14	2055,00	720,00	1,0
15	972,50	300,00	1,0
16	1097,50	382,00	1,0
17	972,50	300,00	1,0
18	1097,50	382,00	1,[illegible]
19	20987,00	6450,00	1[illegible],[illegible]

FAILURE SURFACE DATA

SURFACE		Q1	Q2	Q3	Q4	Q5	Q6	Q7	Q8			
1	$C_1-C_2-C_3-C_4$	1	2	3	15	-4	16	4	5	6	7	0
2	$C_1-C_2-C_3-C_5$	8	9	10	17	-4	18	11	12	13	-2	14
3		9	11	13	32	-8	34	15	17	19	5	14

SURFACE NUMBER 1, 1 $\boxed{C_1-C_2-C_3-C_4}$

X COORD	E		X FORCE	Y-YT		Y-YT/Y-Z		VERT F		SMALL F	
715,00	3378.	3378.	1177.	2,780	2,780	0,278	0,278	0,0	0,0	1,000	1,000
715,00	3378.	3378.	1177.	2,780	2,780	0,278	0,278	0,000	0,0	1,000	1,000
774,38	98267.	98267.	34248.	10,270	10,270	0,180	0,160	3,371	3,371	1,000	1,000
910,00	894758.	894758.	311838.	30,005	30,005	0,182	0,182	3,490	3,490	1,000	1,000
954,18	1277958.	1277958.	445389.	35,183	35,183	0,179	0,179	3,500	3,500	1,000	1,000
970,00	1432224.	1432224.	499153.	37,071	37,071	0,178	0,176	3,503	3,503	1,000	1,000
972,50	1457425.	1457425.	507936.	37,370	37,370	0,178	0,178	3,503	3,503	1,000	1,000
1097,50	3078492.	3078492.	1072905.	45,398	45,398	0,155	0,155	3,508	3,508	1,000	1,000
1136,00	4029079.	4029079.	1404199.	60,260	60,260	0,180	0,180	3,513	3,513	1,000	1,000
1137,01	4036930.	4036930.	1407572.	60,289	60,289	0,180	0,160	3,511	3,511	1,000	1,000
1300,00	5397433.	5397433.	2018816.	63,737	63,737	0,153	0,153	3,286	3,286	1,073	1,073
1336,54	5728920.	5728920.	2175369.	64,178	64,178	0,148	0,148	3,241	3,241	1,090	1,090
1360,00	6062104.	6062104.	2324016.	63,200	63,200	0,142	0,142	3,203	3,203	1,100	1,100
1370,00	5855039.	5855039.	2251794.	60,974	60,974	0,137	0,137	3,196	3,196	1,104	1,104
1375,00	5751515.	5751515.	2215496.	59,856	59,856	0,135	0,135	3,193	3,193	1,105	1,105
1400,00	5238309.	5238309.	2033822.	54,154	54,154	0,125	0,125	3,248	3,248	1,114	1,114
1423,29	4777066.	4777066.	1866344.	48,548	48,548	0,119	0,119	3,229	3,229	1,122	1,122
1460,00	3829365.	3829365.	1514883.	42,127	42,127	0,113	0,113	3,202	3,202	1,135	1,135
1581,27	1370684.	1370684.	562564.	16,243	16,243	0,065	0,065	3,154	3,154	1,176	1,176
1645,00	496908.	496908.	207816.	-9,802	-9,802	-0,053	-0,053	3,246	3,246	1,200	1,200
1670,00	282593.	282593.	115724.	-31,321	-31,321	-0,193	-0,193	3,474	3,474	1,175	1,175
1741,16	-128152.	-128152.	-49301.	81,115	81,115	0,856	0,856	2,582	2,582	1,104	1,104
1840,87	3.	3.	1.	0,0	0,0	0,0	0,0	0,0	0,0	1,004	1,004

F = 3,895 LAMBDA = 0,349(8 9 0)

SURFACE NUMBER 2, 1 $\boxed{C_1-C_2-C_3-C_5}$

X COORD	E		X FORCE	Y-YD		Y-YD/Y-Z		VERT F		SMALL F	
715.00	2433.	2433.	1260.	2.321	2.321	0.232	0.232	0.794	0.794	1.000	1.000
715.00	2433.	2433.	1260.	2.321	2.321	0.232	0.232	0.794	0.794	1.000	1.000
774.38	80103.	80103.	41485.	6.279	6.279	0.117	0.117	2.361	2.361	1.000	1.000
910.00	741082.	741082.	383804.	18.482	18.482	0.112	0.112	2.374	2.374	1.000	1.000
954.18	1055835.	1055835.	546814.	20.829	20.829	0.106	0.106	2.377	2.377	1.000	1.000
970.00	1182597.	1182597.	612464.	21.740	21.740	0.104	0.104	2.377	2.377	1.000	1.000
972.50	1203307.	1203307.	623189.	21.837	21.837	0.104	0.104	2.377	2.377	1.000	1.000
1097.50	2577663.	2577663.	1334964.	22.594	22.594	0.077	0.077	2.373	2.373	1.000	1.000
1136.00	3382779.	3382779.	1751931.	37.031	37.031	0.111	0.111	2.374	2.374	1.000	1.000
1137.01	3388198.	3388198.	1754737.	36.947	36.947	0.110	0.110	2.374	2.374	1.000	1.000
1300.00	4330273.	4330273.	2242636.	25.347	25.347	0.061	0.061	2.387	2.387	1.000	1.000
1336.54	4560691.	4560691.	2361968.	23.191	23.191	0.054	0.054	2.388	2.388	1.000	1.000
1360.00	4841872.	4841872.	2507591.	21.284	21.284	0.049	0.049	2.386	2.386	1.000	1.000
1370.00	4589295.	4589295.	2376782.	16.575	16.575	0.037	0.037	2.390	2.390	1.000	1.000
1375.00	4463085.	4463085.	2311416.	14.142	14.142	0.032	0.032	2.392	2.392	1.000	1.000
1400.00	3838021.	3838021.	1987699.	0.976	0.976	0.002	0.002	2.456	2.456	1.000	1.000
1423.29	3276992.	3276992.	1697144.	-13.313	-13.313	-0.033	-0.033	2.463	2.463	1.000	1.000
1460.00	2076644.	2076644.	1075488.	-47.988	-47.988	-0.129	-0.129	2.498	2.498	1.000	1.000
1581.27	-1017880.	-1017880.	-527157.	126.425	126.425	0.505	0.505	2.229	2.229	1.000	1.000
1645.00	-2108591.	-2108591.	-1092033.	33.448	33.448	0.180	0.180	2.329	2.329	1.000	1.000
1670.00	-1966513.	-1966513.	-1018451.	35.117	35.117	0.201	0.201	2.328	2.328	1.000	1.000
1741.16	-1648715.	-1648715.	-853865.	39.647	39.647	0.280	0.280	2.330	2.330	1.000	1.000
2046.43	-0.	-0.	-0.	0.0	0.0	0.0	0.0	0.0	0.0	1.000	1.000

P = 3.343 LAMBDA = 0.518(14 16 0)
10:32.21 2.59 RC=0

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