STABILITY PROBLEMS AND ROCK SUPPORT AT THE TELLNES OPEN PIT MINE, NORWAY

Bjørn Nilsen, Professor
Norwegian University of Science and Technology (NTNU) /
SINTEF Rock and Soil Mechanics
and
Brian J. Ballou, P.E. Mine Manager
Titania A/S

ABSTRACT

The Tellnes open pit mine in Southwestern Norway has been operating since 1960. The maximum height of pit walls is currently about 150 meters with the current mine plan extending this distance to 400 meters with slope stability being a major controlling factor. As a result of unfavorable combinations of directions of major joints and faults, local stability problems have occurred several places in the pits walls. Stability problems have affected the reliability of main transport roads on several occasions. Most of such incidents have occurred during heavy rainstorms and during periods of repeated freezing/thawing. This paper reviews some of the stability problems and the measures taken to deal with such stability problems.

1 INTRODUCTION

The Tellnes open pit mine is located within the Precambrian Egersund province in Southwestern Norway. Titania A/S has been operating the mine since 1960. In the orebody, about one third of the rock consists of ilmenite ($FeTi0_3$). In recent years the total mined material has been about 8 million tonnes/year and the ore production (ilmenite; $FeTiO_3$) about 3 million tonnes/year.

As shown in Figure 1 the Tellnes ore is an intrusion of ilmenite-norite in the surrounding anorthosite. The orebody is about 2.7 km long, and the width is up to about 400 meters, but as shown in the figure it varies considerably. As also shown in the figure, two dikes of diabase intersect the orebody, and fracture zones of various ages and orientations also intersect the area. Tectonic movement along such zones has caused fracturing and alteration that have great influence on the stability of the pit walls.

Geological features, including fractures and joints, have been mapped continuously as the mine has been excavated. Based on stereographic plotting of more than 3000 strike and dip measurements it has been found that the orientations of main joint sets vary considerably in the area, which may be divided into several structural regions. Concerning stability it is, however, particularly the following distinct sets of continuous joints that affect the situation:

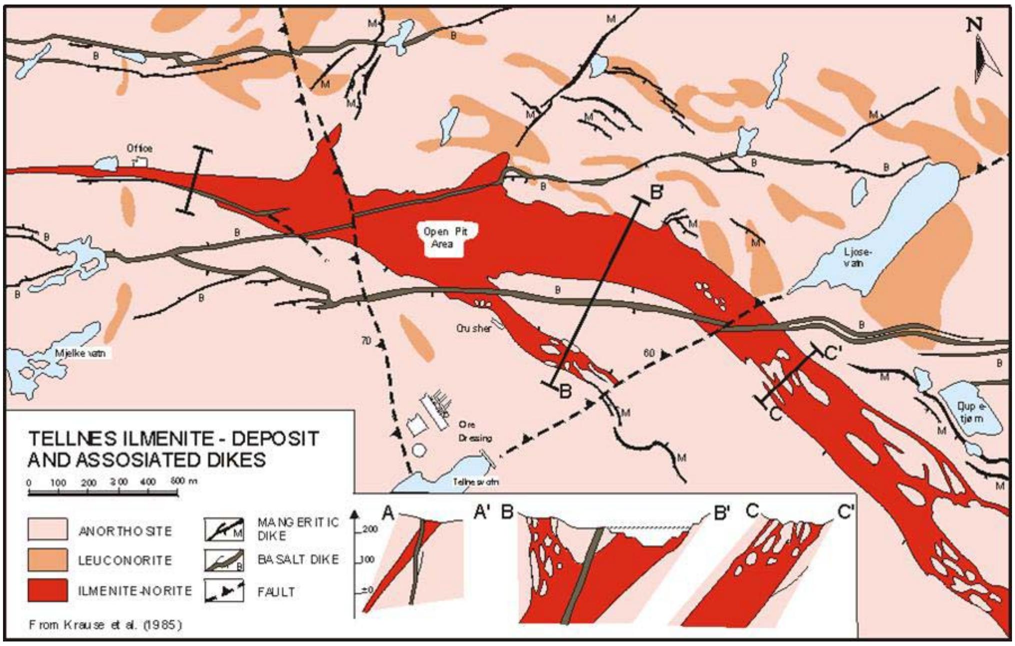

Figure 1. Geology of the Tellnes open pit mine area.

- In the hanging (SW) wall: very continuous joints with intermediate dip to the NE, representing risk for plane failure.
- In the foot(NE)wall: continuous joints with intermediate dip to the SE and W-SW, respectively, representing risk for wedge and/or plane failure.

Gouge material consisting of chlorite, calcite, illite and smectite, is often found in the most distinct joints. For the smectite free swell of up to 230% has been measured, indicating very high swelling capacity and low frictional resistance against sliding. This further increases the risk of sliding.

2 PIT WALL DESIGN

The final pit slope at Tellnes has a bench height of 15 meters and a bench for every 30 meters giving an overall slope angle of approximately 55^o. The maximum height of pit walls is currently about 150 meters with the current mine plan extending this distance to 400 meters. Figure 2 gives an overview of the mine as seen from the West in June 2005.

For contour blasting, pre-splitting with 32 mm pipe charges in 2.5” holes is used. As can be seen in Figure 2 and 5, this gives good results when the geological conditions are favorable (mainly in the footwall). In the hanging wall, as can be seen in Figure 4, smooth walls are considerably more difficult to obtain.

As a result of unfavorable combinations of directions of major joints and faults, local stability problems have occurred several places in the high pit walls. Most of such incidents have occurred during heavy rainstorms, during periods of repeated freezing/thawing and, most likely, as result of previous, unfavorable blasting practices. In the following, some of these stability problems, and the stabilizing measures, which have been used to cope with such problems, will be discussed. Main emphasis will be placed on:

Figure 2. Tellnes open pit mine seen from West. Hanging wall (SW) to the right, footwall (NE) to the left. The top of the footwall is here at level ~250 m and the bottom is at level ~80 m.

1) The stability problems in the hanging wall caused by the very distinct, undercutting discontinuities.
2) An incident above the main transport road in the footwall, where a minor slide occurred, and a large wedge in the remaining wall was considered potentially unstable due to unfavorably oriented discontinuities.
3) Stability problems in the footwall threatening the foundation of the main transport road.

3 STABILITY PROBLEMS IN THE HANGING WALL

The hanging wall area (southern side) is the area of the mine, which has experienced the most frequent and the most serious stability problems. The general problem here, as previously mentioned, is that very distinct, continuous joints, often containing significant amounts of clay gouge, are oriented with strike approximately parallel to the pit wall and with intermediate dip towards the bottom of the pit, thus representing a risk for plane failure between the benches as illustrated in Figure 3. Typical strike of such joints is N115- 160^{o}E, and the dip is typically 40-50^{o}NE (with around 45^{o} as the most common).

The dip angle is very close to critical angle for sliding, and since cross joints releasing potential sliding volumes at the sides are quite common, bench stability problems due to sliding has been a recurring problem in the hanging wall. Installation of rock bolts as spot bolting therefore was introduced early in particularly difficult areas. Based on some unexpected incidents of sliding, systematic bolting in difficult areas has been introduced more recently.

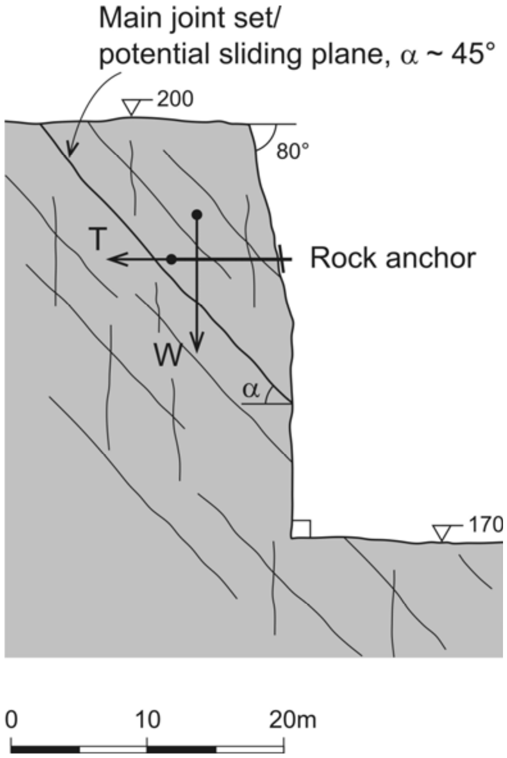

Figure 3. Principle sketch of bench stability problem due to undercutting joints in the hanging wall.

The basis for design of bolt pattern is systematical, continuous joint mapping and estimation of required anchoring force (T) based on the assumption that the volume prior to support is in the state of unstable equilibrium, i.e. the factor of safety FS~1.0. The estimation is based on standard pit wall geometry; i.e. 30 m between benches, 80^o wall angle the first 15 m and 90^o wall the next 15 m. The volume undercut by the joint is calculated for alternative locations of potential sliding plane, including the worst case scenario; i.e. undercutting joint starting from just above the foot of the wall, and for alternative dip angles.

The weight (W) of the undercut volume is estimated according to geometry, and the force tending to induce sliding (S), represented by the weight component parallel to the sliding direction (and in the state of unstable equilibrium equal to the stabilizing force, R), thus will be:

$S = R = W \cdot \sin \alpha$

Rock anchoring has a stabilizing effect by reducing the driving as well as by increasing the stabilizing forces. The required anchoring force (T) for a factor of safety FS and active (total) friction angle φ_a thus may be estimated based on:

$(R + T \cdot \sin \alpha \cdot \tan \varphi_a) / (S - T \cdot \cos \alpha) = FS$

As active friction angle, φ_a~60^o is considered relevant for the joints in question here.

The decision concerning rock anchoring is based on continuous mapping of joint locations and orientations in the actual area of the pit wall. In addition to rock anchoring, where required, drainage by drilling holes inclined slightly upwards is carried out. This is believed to have a considerable stabilizing effect, and in the equation above, a FS of 1.1-1.2 without drainage taken into account is believed to be well on the safe side relatively to the FS of 1.2-1.4, which is commonly used in open pit mining (Wyllie & Mah, 2004).

The extent of rock anchoring having been carried out in the hanging wall varies with geology from no bolting in some areas to systematical bolting with relatively close spacing in other areas. Mainly, 12 m long, diameter 32 mm rebar bolts have been applied. In some areas, like between A-B and to the right of B in Figure 4, bundles of 3 bolts have been installed in each borehole, with average borehole-spacing of ~1.5 m (zigzag pattern).

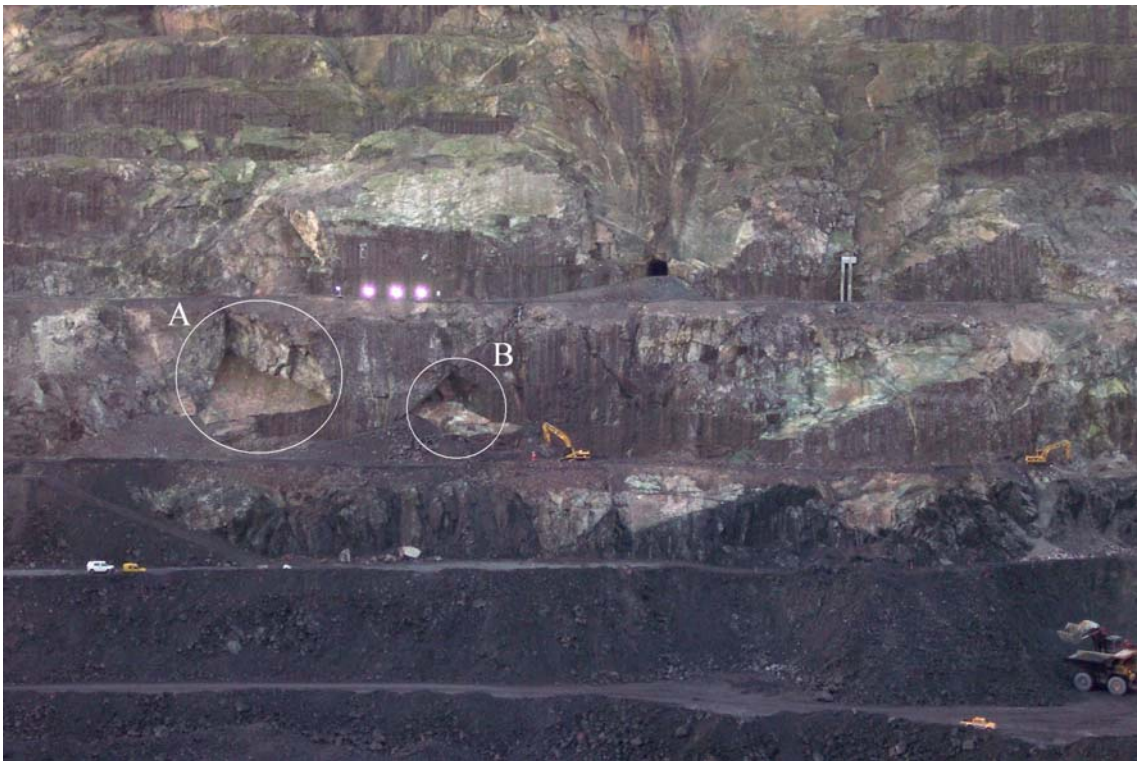

Figure 4. Overview of the central part of the hanging wall, characterized by distinct, undercutting joints.

As shown on Figure 4, the very distinct, undercutting joints have caused sliding of considerable tonnage (A~1000 tonnes and B~200 tonnes) before it was possible to install rock support. The incidents at A and B on the photo occurred during production blasting, and before the final wall was established. Still, incidents like these illustrate the importance of continuous joint mapping and installing the required support in time. In practice, this is often difficult since the daylighting of critical joints will not be visible before the final wall has been established. Alternative support methodology by installing bundles of 6-9 rebar bolts in boreholes drilled from above and parallel to the pit wall therefore has been evaluated (Nilsen & Hagen, 1990), but so far has not been used in the hanging wall (although as of January 2006 is being designed). As will be discussed in the next chapter, this approach has been used with good results for stabilizing an unstable area above the main transport road in the footwall.

4 STABILITY PROBLEMS IN THE FOOTWALL

4.1 Wedge failure above main transport road

One of the most serious incidents in the footwall occurred during a period of very heavy rain in August 1997, when wedge failure of about 5,000 tonnes took place. As shown in Figure 5, the failure was located just above the main transport road (level 155), and was caused by unfavorably oriented, distinct joints filled with clay minerals and chlorite (N70^{o}E/64^{o}SE and N170^{o}E/50^{o}W, respectively, measured as orientations for Western and Eastern, undercutting joint).

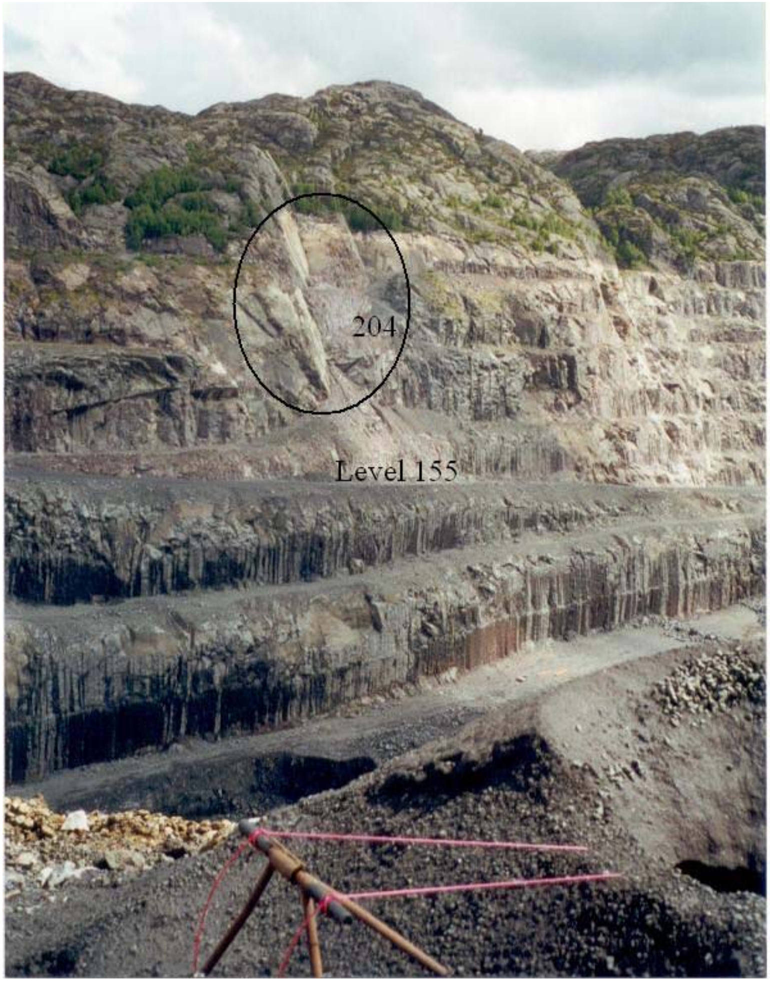

Figure 5. Slide area above the main transport road in the footwall, profile 1150 (photo taken from SW).

The main transport road in this area is at level approx. 155. The upper limit of the slide was at level 184, and the material from the estimated 5,000 tonne slide completely covered the bench at level 170. As can be seen on Figure 5, the very distinct joint to the West can be followed all the way from the top to the bottom of the pit, and the stability of this part of the mine, the area above the main transport road included, therefore was considered to be highly questionable.

In principle, two alternatives were considered possible for solving this problem; rock support or removal of the unstable volume by blasting. Removal of the entire unstable volume was considered to represent a very comprehensive and risky operation, and even if successfully carried out, would give an unfavorable geometry with protruding corners in parts of the wall. It was therefore concluded that a combination of rock support and minor trimming would be the best alternative.

Based on the site investigations and calculation of required support, the following approach was finally concluded to be the best for ensuring satisfactory stability in this area (see also Figure 6):

1) Anchoring of the estimated 14,000 tonne wedge above level 204 with 150 tonne, tensioned rock anchors installed in slightly upwards-inclined drillholes. Based upon similar design principle as described in Chapter 3 and a safety factor of 1.3, a total number of 10 rock anchors, 10-25 m in length were installed.
2) Support of the estimated 3,600 tonne wedge below level 204 was carried out with bundles of rebar steel in vertical boreholes as described at the end of Chapter 3. Based on shear capacity of rebar estimated to be approximately equal with the tensile capacity, 6 bundles of 10-20 m long rebar steel, each consisting of 5 diameter 32 mm rebar and representing a capacity of 150 tonne, were installed.
3) Drilling of drainage holes, inclined slightly upwards.
4) Careful trimming of the outer/protruding part of the volume below level 204.

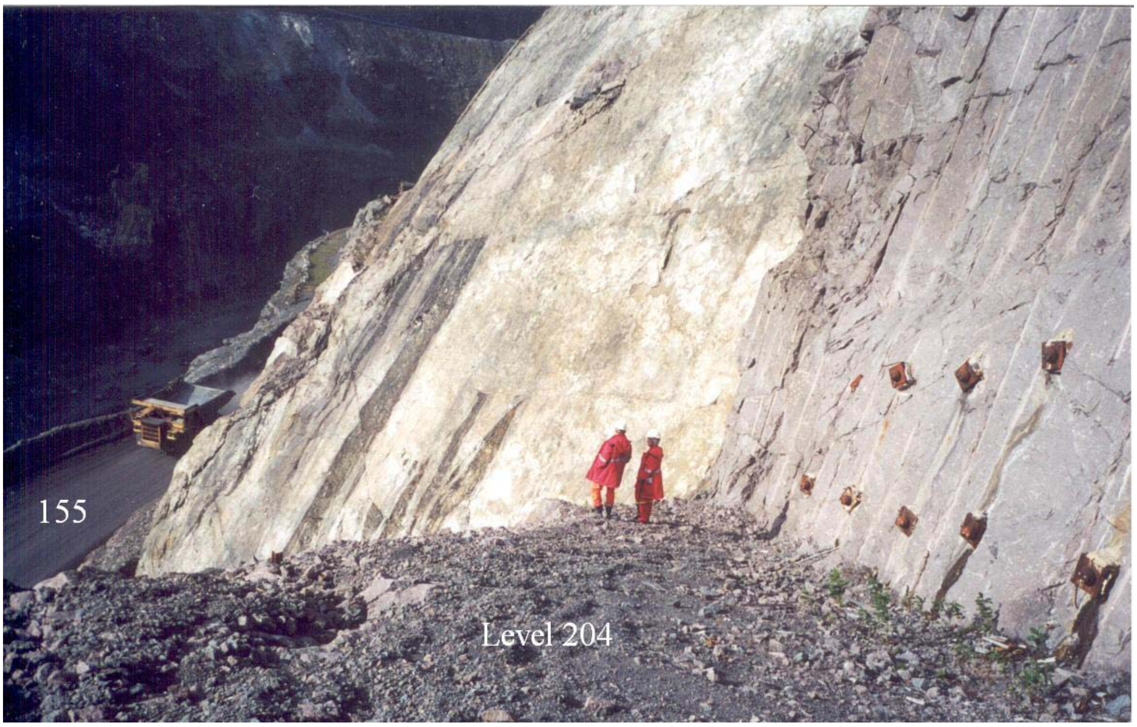

Figure 6. Support of footwall after wedge failure above main transport road.

The reason for the different approaches of supporting the volumes above and below level 204 was that access was possible for installing ~horizontal anchors in the former, but not in the latter area. The support, including the somewhat unconventional approach for the volume below level 204, has functioned well, and no stability problems have been experienced after support was installed.

4.2 Stability problem below the main transport road

Nearby the area discussed above, a little further to the East and below the main transport road, a very distinct clay-filled joint caused problems in 2001-2002. As shown in Figure 7, a very distinct joint is here oriented approximately parallel with the pit wall (strike/dip approximately N135^{o}E/57^{o}SW). The joint has a filling of clay minerals and obviously may represent the sliding plane of a potential large-scale plane failure if undercut. As shown on Figure 7, the pit wall follows this joint from the main transport road at level 140 to level 112, and it is obvious that the joint continues further down.

The undercutting joint caused stability problems resulting in some narrowing of the road as can be seen on Figure 7, and to prevent further sliding that would cause unacceptable narrowing, quite comprehensive rock anchoring was carried out of the wall between level 140 and 112.

Figure 7. The footwall at the Eastern part of Tellnes open pit mine as seen in October 2002. The main transport road is at level 140 and the bottom of the pit at level 80.

Instead of excavating directly from level 112 to level 80, risking damage of the toe of the bench, a 15 m wide bench at level 95 was established, giving a 13 m wide foot as illustrated in Figure 8.

The main question concerning this very important area of the open pit mine therefore was whether:

1) Attempting to support the wall between level 112 and 80 by using rock anchors, giving “ordinary wall design” as shown in Figure 8, top right.

or:
2) Letting the final wall follow the distinct joint all the way from level 140 to level 80, as shown in Figure 8, bottom right.

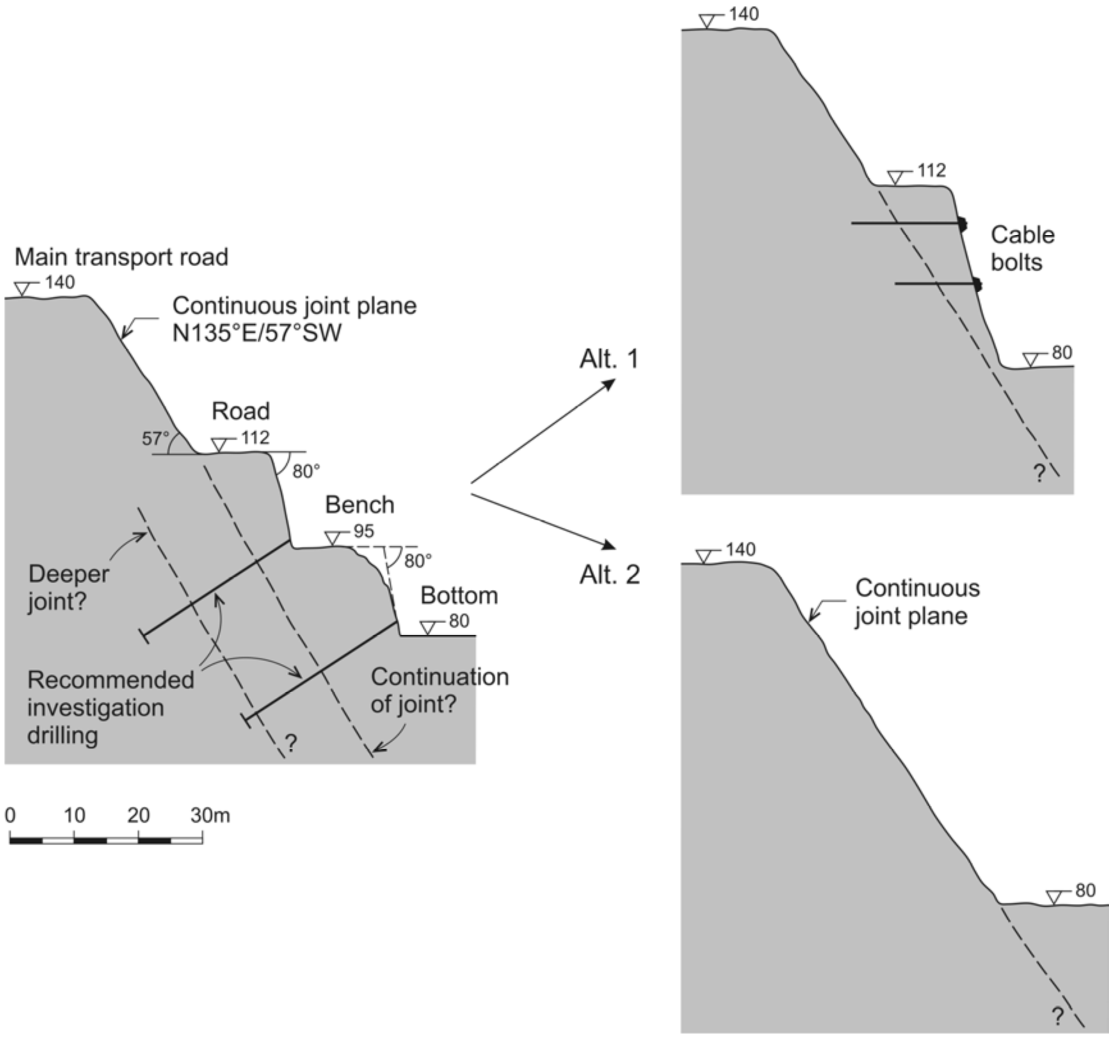

Figure 8. Cross section of the temporary pit wall below the main transport road shown in Figure 7, and the two alternatives for final wall design between level 140 and 80 (right).

Based on overall evaluation of safety, feasibility and cost, alternative 2) was concluded to be the best solution. The main reason for choosing alternative 2) was that the rock mass was considered to be heavily fractured and that considerable displacement had already taken place. Rock anchoring similar to the layout illustrated in Figure 6 (above level 204 was therefore considered difficult in practice.

An additional complicating factor here was the considerable uncertainty related to the exact location and character of the joint plane below level 112. In particular, information concerning potential curving of the structure towards depth was considered important for the planning of further excavation. To investigate the underlying characteristic of the structure the use of borehole camera in drillholes was recommended as indicated in Figure 8. In the final program, 3 such holes were drilled, (to allow an accurate projection of any potential structures onto an orientated plane) and inspected with optic televiewer. This supplementary investigation with borehole camera provided very valuable information concerning the rock mass conditions and confirmed that alternative 2) was the best option for the final wall in this area.

Based on this, the pit was designed by following the distinct joint all the way from level 140 to level 80, which proved to give a stable final wall in this area.

5 CONCLUDING REMARKS

The slope stability conditions at Tellnes open pit mine are governed mainly by the orientations of very continuous joints, which are often filled with smectite, chlorite and other slippery minerals. When these joints are oriented approximately parallel with the final pit wall, and have a dip angle steeper than approximately 40^{o} towards the pit bottom, sliding tends to occur unless the necessary stabilizing measures are taken in time. Most often, problems due to plane failure occur in the hanging wall, but locally problems due to wedge and plane failure occur also in the footwall.

The most severe cases of sliding typically have occurred during periods of heavy rain and/or as result of repeated freezing/thawing. Thus the cases from Tellnes confirm the general experience that groundwater pressure and climate have major impact on the stability.

To be able to control the stability and to have the possibility of taking the necessary stabilizing measures in time, continuous geological mapping, particularly of distinct joints, is crucial. Even when this is done, it is often difficult to get safe access into the areas that require support due to the high and steep walls. Thus, it is very important to use skilled and experienced people for carrying out this type of work. In any case, good quality perimeter blasting is important for the stability, and also for ensuring safer working conditions at lower levels of the mine.

As the Tellnes open pit is expanded to the East and is mined to greater depths, having good control of stability is critical to both to safety and production. Based on experience, Titania feels that early detection and remedy of unstable conditions is important and cost effective and will now evaluate various options for monitoring the high wall situation. A GPS based monitoring system is one of the leading candidates to aide the staff in addressing the long-term stability issues at Titania.

REFERENCES

Elvebakk, H. & Nilsen, B. Mapping of geological structures by using optic televiewer. Proc. Nordic Mineral Symposium, Trondheim, 2004, Geological Society of Norway 1:2004, pp 13-14 (in Norwegian).

Nilsen, B. Tellnes open pit mine, slope stability evaluations. SINTEF Rock and Soil Mechanics, 2005, Report STF50F05121, 8p (in Norwegian).

Nilsen, B. & Hagen, R. Stability problems and proposed unconventional rock support at the Tellnes open pit mine. Proc. Norwegian Rock Blasting Conference, Oslo, 1990, Tapir Publ., pp. 23.1-23.20 (in Norwegian).

Wyllie, D.C. & Mah, C.W. Rock Slope Engineering - Civil and Mining. Spon Press, 2004, 431 p.