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Damage mapping and monitoring in sublevel caving crosscuts at the Malmberget mine

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Abstract

The LKAB's Malmberget mine in Sweden is one of the largest sublevel caving mines in the world, with an annual production rate averaging 18 million tons. This high rate of production at depth (>1,000 m) creates significant mining-induced stress redistribution on a global scale. At a production-level scale, this redistribution results in undesirable amounts of deformation in the entries and typically leads to general degradation in the footwall contact zones. This is exacerbated by highly varied geological and geotechnical characteristics of the lithology often found in the contact zone. To better understand the impact of mining-induced stress on production level entries, a study was conducted to measure stress changes and associated deformation over a two year period, as mining progressed in the vicinity of the instrumentation. Three-dimensional relative stress measurements using digital hollow inclusion stress cells and multiple-point borehole extensometer measurements were combined with convergence and floor heave measurements and regular damage mapping throughout the contact zone to better understand the evolution of damage in these areas. A site-specific Entry Condition Rating (ECR) system was developed to help geomechanics better track and understand the expected performance of the crosscut given the current state of mining. The result of the work is a better understanding of where and when damage is expected to occur, and the ability to properly time the installation of secondary support in a pre-emptive manner.

Keywords: *stress, deformation, instrumentation, condition ratings*

1 Introduction

Sweden's LKAB mining company was founded in 1890, though mining in the region goes back to 1660s when the first ore sample was collected. Today LKAB operates two underground iron mines, Kiirunavaara and Malmberget, where sublevel caving is the primary mining method. The Malmberget mine comprises some 20 orebodies, of which about half of them are in production today and has an annual production around 18 Mtpa. The mine is in the municipality of Gällivare, Norrbotten county, in northern Sweden. Today's mining is conducted at 800–1,200 m depth, distributed over an area of approximately 2.5 km × 5 km (NS × EW). The dip of the orebodies varies between 15° and 75°, with an average dip of 45°–50°.

Mining is carried out using the sublevel caving method which unavoidably results in redistributions of the stresses moving through the rock. As development progresses, these stresses can concentrate in sensitive areas leading to rockfalls and high deformation. When excavation and extraction proceed to deeper levels stresses accumulate and can overcome the strength of the rock mass hosting the excavations.

In Malmberget, difficulty is encountered when excavating through geological units with highly varied properties, such as that encountered when sedimentary material is replaced by igneous intrusives or magma bodies. Stresses in these areas are concentrated, not only from the effect of multiple excavations, but by the natural capabilities of the different geological units to withstand and/or transmit that stress. It is an area where rock support can be challenging at best, and nearly impossible at worst. In this type of scenario, it is potentially unsafe, and certainly inefficient, to design based solely on 'what was done before'.

To better design entries and required support systems for these areas, a better understanding of how mining-induced stresses and geologically driven stress concentrations interact to induce rock failure. This project helps to address that problem by investigating the interactions between the rock material and redistributed stress (Jones & Saiang 2022a).

2 Site description

2.1 Geology

The ore in Malmberget is approximately 90% magnetite with areas of hematite and apatite. The orebody includes a vein system with impregnations of several minerals and is strongly affected by metamorphic recrystallisation. The main gangue minerals are amphibole, pyroxene and biotite and granitic intrusions often transect the ore (Bergman et al. 2001). The thickness of the orebodies ranges between 20 and 100 m. The volcanic host rock consists of grey, grey-red, red-grey, and red leptites with increasing strength (Jones et al. 2019).

A biotite schist is also found in the mine, occurring in a layered structure along the strike and dip of the orebody footwall contact and is the weakest unit compared to the other rock types in the mine. Its purest forms tend to occur in direct contact with the ore, though it also exists in veins and inclusions of varying proportions in the other major geological units. The rock mass strength and crosscut stability in any particular location are generally directly influenced by the presence of the schist. The different lithologies vary in grain size and mineralogy, though all of them tend to contain an amount of biotite. The higher the rock mass strength, the lower the biotite content.

The geology of the Malmberget mine provides an excellent opportunity to investigate the interplay of stresses flowing through materials of varying geotechnical quality near one another. The best location for this exists along the footwall contact with the orebody. At these locations there is often a zone where the leptite host rock (typically grey or red in the areas under consideration) and the magnetite ore are separated by a thin (5–10 m) layer of biotite schist. These three geological units have very different strengths and deformation characteristics. Their dimensions and proximity to one another are such that all three materials can be instrumented in one crosscut and thus it is reasonable to assume that each of the instruments are subjected to the same general stress redistribution process yet may produce different results due to their differing rock properties.

2.2 Mining method

The mining method employed in the study areas is transverse sublevel caving. For this method, rings of 8–14 production blastholes are drilled upwards into in a trapezoidal fan pattern in the roof at 3.5 m intervals. The first three rings in each crosscut are spaced closer together to create a larger initial opening designed to initiate mining. The rings are long enough to extend beyond the undisturbed rock above the entry up to the previous sublevel, meaning that the centre holes can be up to 60 m long, while the side holes are typically around 30–32 m long. After the crosscut is opened for mining, the remaining blast rings are loaded and detonated consecutively towards the footwall drift. The large size of the blast rings results in noticeable stress redistribution throughout the mine, and it is this stress redistribution that drives much of the damage that occurs in the crosscuts.

Rock support in the area routinely comprises 3.05 m long dynamic bolts on a 1 m × 1 m spacing, 10 cm thick fibre-reinforced shotcrete, and dynamic welded-wire mesh.

The work undertaken in this project extends through the Alliansen (AL), Hoppet (HO) and Printzsköld (PR) orebodies. The Alliansen orebody is furthest east of the three and is one of the oldest orebodies in the mine. It was first mined as an open pit beginning in 1905 and then converted to full-scale sublevel caving below the 300 m level. Level numbers correspond to a local coordinate system where depth and level numbers increase downwards, relative to the top of the mountain. The PR orebody furthest west and first began production on level 780, proceeding downwards from there. The two orebodies begin to merge around level 1000, with the HO orebody becoming a distinct entity beginning on level 1023 in between them. From that depth downwards the three orebodies form a combined ore unit along with the Gunilla (GN) orebody, sitting west of the PR orebody. The combined footwall drive for these four orebodies is approximately 2,000 m long.

3 Instrumentation and data collection

3.1 Instrumentation sites and input data

The premise of the instrumentation plan was that the mining activities redistribute the pre-existing stresses in the area and that these stress redistributions create increased stress concentrations which are channelled through and around the development entries in the mine. Depending on the rock quality parameters and entry geometries, this can result in stress changes, seismicity, and/or deformation. The instrumentation and data collection plan were devised to capture both the driving stress changes and the resulting damage and deformation in the footwall contact zone and its varying geological units. Proximity to active mining varied from 30 m up to over 750 m at different times of the project. Every production blast regardless of location was considered to determine its individual effect on the instrumented locations.

Because the timing and location of each individual production blast is exactly known throughout the mine, and because each production blast produces its own stress redistribution, it becomes possible to identify how different production blasting activities impact both the stress changes and the deformations and damage. Stress measurements were made using 3D digital hollow inclusion stress cells (HID-cells), which acquired stress data relative to the beginning of the measurement period. Deformation was collected using a combination of multiple-point borehole extensometers, wall-to-wall convergence stations (using a laser distance metre) and floor heave measurements. Entry damage was tracked by establishing 5 m long damage mapping zones throughout the instrumentation areas (Figure 1). Convergence and floor heave measurements were taken at the start and finish of each damage mapping zone. Additional input data included geologic mapping and Geologic Strength Index (GSI) values collected by the mine geologists upon entry development.

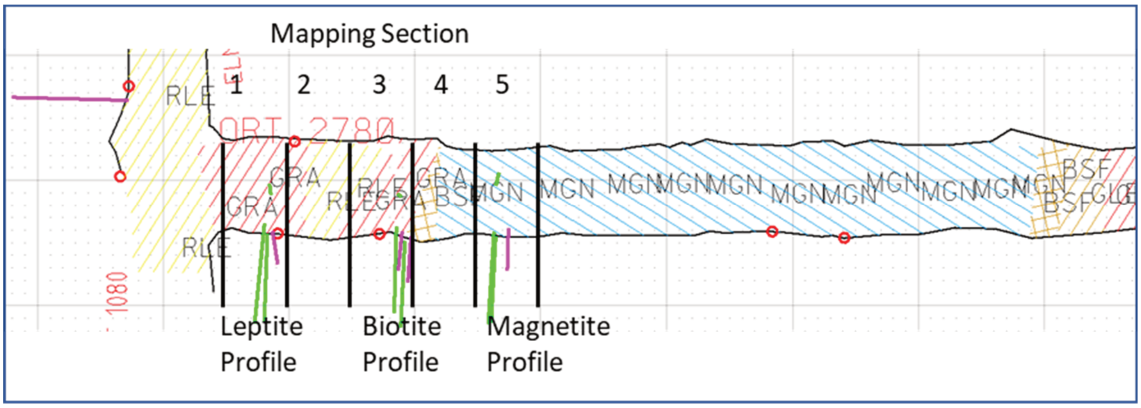

Figure 1 Instrumentation profiles combined with damage mapping sections in a geologic map of the AL1081o2780 instrumentation site. Extensometers shown as green lines and stress cells as pink lines

Data was collected from five primary sites located in the PR, HO and AL orebodies, on levels 996 and 1023 in PR, and on level 1080 in HO and AL. The different sites are depicted in Figure 2, with the individual instrument locations circled in red. The five site locations and the number of instrumentation profiles installed in each are detailed in Table 1. Additional floor heave measurements were taken in crosscuts near the instrumented sites to expand the dataset.

Instrumentation was installed in profiles so that the multipoint borehole extensometers and HID-cells were aligned in a particular geology as much as was possible. Anchor positions and stress cells were installed close to the excavation walls in order to better reflect the stress conditions in the actual deformation zone. Each site had variations in the amount of instrumentation for practical reasons, however a fully instrumented site comprises of the instrumentation as depicted in Figure 3.

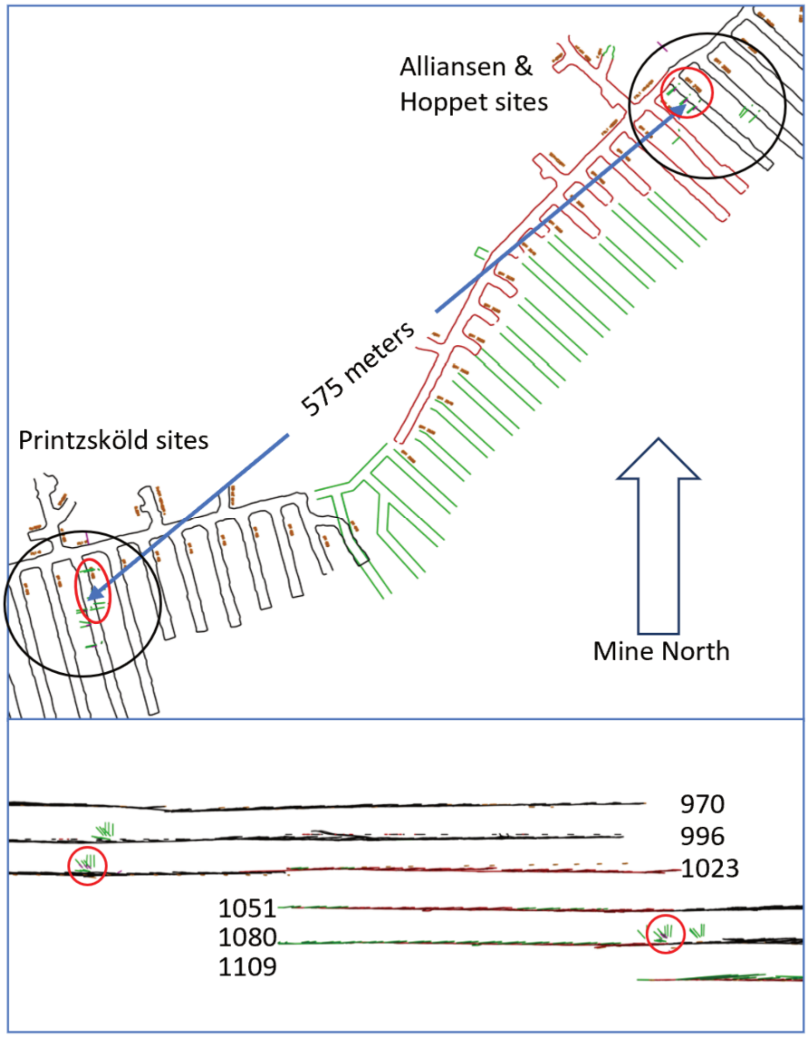

Figure 2 Mine layout and instrumentation sites viewed from above (top) and from the side (bottom)

Table 1 Instrumentation sites, levels, crosscuts and profiles

Orebody	Level	Crosscut	ID	Instrument profiles (#)
PR	996	4090	PR996o4090	3
PR	1023	4080	PR1023o4080	3
HO	1080	2760	HO1080o2760	1
AL	1081	2780	AL1081o2780	3
AL	1081	2800	AL1081o2800	2

Far greater detail has been documented in the full report of this project (Jones & Saiang 2022a), however, Table 2 provides an overview of the total amount of instrumentation installed, it's accuracy and resolution, as well as the collection time intervals.

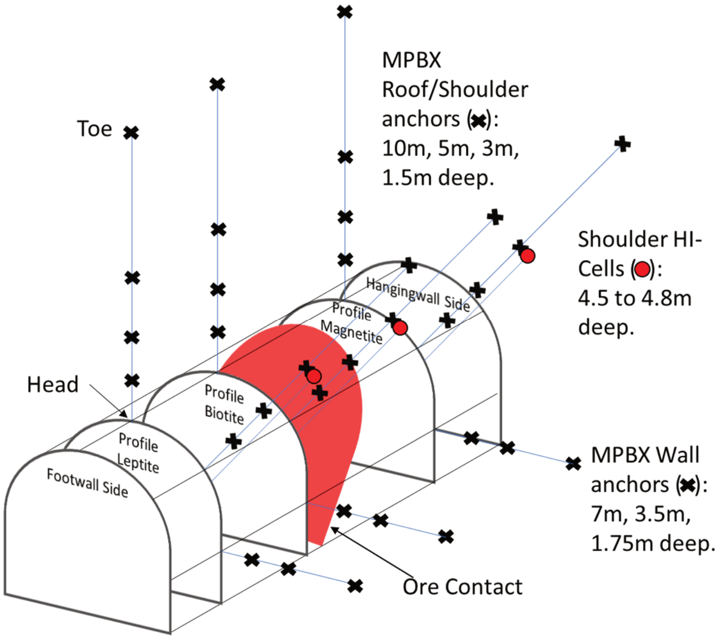

Figure 3 Layout of a typical fully instrumented site, including anchor positions and instrument lengths

Table 2 Instrumentation details

Data collection method	Collection time interval	Accuracy	Resolution	Number
MPBX	1 min to 24 hr	+/– 2%	0.254 mm	33
HID-cell	1 min to 8 hr	+/– 10 ppm	0.1 με	8
Wall-wall convergence	8–12 weeks	+/– 5 mm	1 mm	34 stations
Floor heave measurement	6 months	+/– 30 mm	1 mm	50 stations
Damage mapping	10 days – 3 weeks	n/a	n/a	31 sections

3.2 Damage mapping

Damage mapping was undertaken on a regular basis during the project. Five-metre-wide damage mapping sections were measured and marked on the walls underground with vertical white numbered lines (Figure 4). These lines corresponded to the floor heave mapping locations and the convergence monitoring stations. Photographic records were kept and proved very useful for back-analysis and quality control.

Figure 4 Damage mapping of wall (left) and floor (right)

Crack painting and coloured numbering of major cracks were used to help track new damage as it occurred. Complete sets of records were kept regarding the state of the crosscut, shotcrete damage, rock damage, falls of rock, chips, and/or dust, water ingress and support corrosion, bolt and mesh damage, floor heave measurement, etc.

To the greatest extent possible, damage mapping was undertaken approximately every 2–3 weeks throughout the project and always by the same person. The interval varied depending on time availability, mine area closures, etc. From this information, it was clear which entries were changing most rapidly and as such these received greater attention. Damage mapping was also completed before and/or after notable events that were expected to impact entry condition such as nearby production blasting, reinforcement activities, large seismic events, pre-planned breaks (summer holiday, for example), etc. The cumulative mapping occasions are shown in Figure 5.

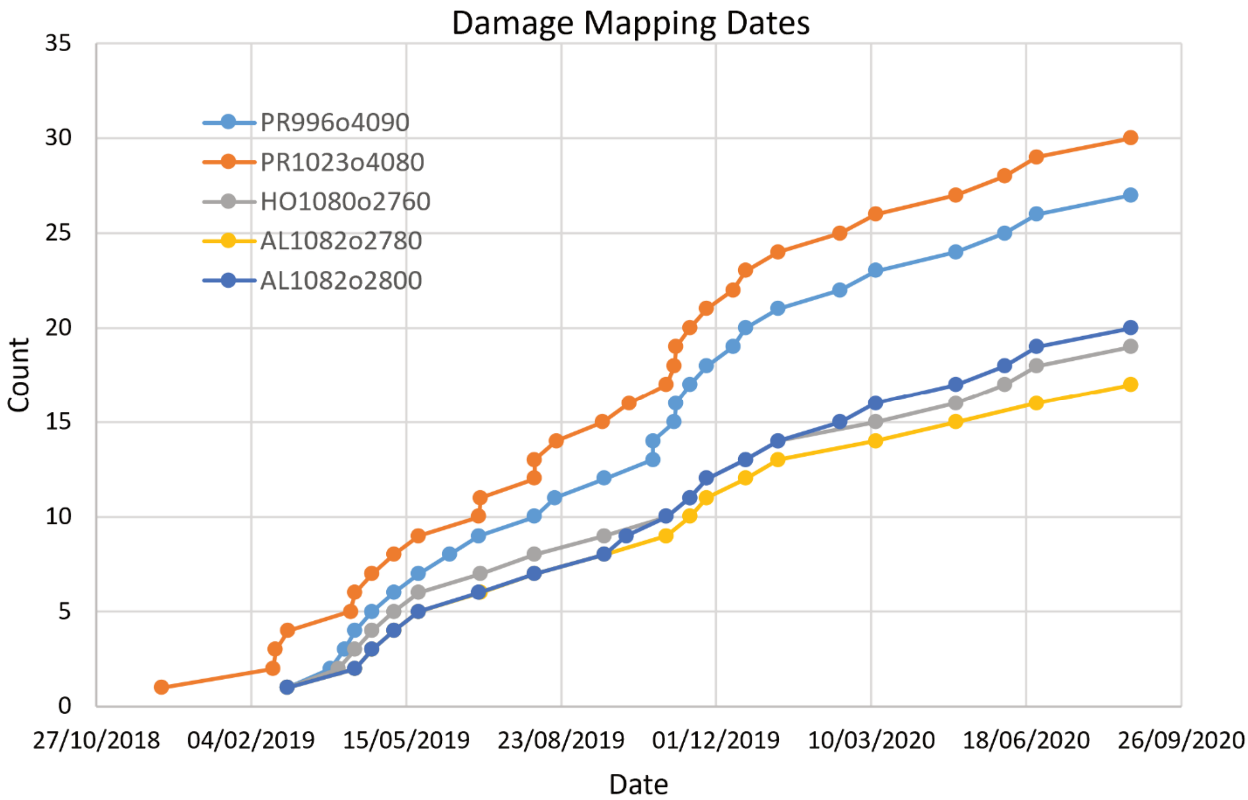

Figure 5 Progression of damage mapping in each instrument site

4 Entry condition rating system

Many damage mapping systems have been developed for rockbursts, deformation, and support damage (Duan et al. 2015; Kaiser et al. 1992; Lawson & Zahl 2012; Mikula & Gebremedhin 2017). In general, a damage mapping system is a method of simplifying damage assessment and allowing the user to link that damage more easily with other information. Damage can be linked to hazard level, to location, to mining progression, etc. It can help to better understand when, how and why the damage occurs and can help to better respond to the damage by providing thresholds for response. As such, it is an important part of many ground control management plans, and seismic hazard management plans.

Based on the data collected in this study, a database was created from the notes and observations from the damage mapping activities. These observations were analysed for trends which were used to create an empirical damage mapping system to help classify the condition of the entries found underground. This was used to better understand the occurrence of damage and relate it to the stress conditions and the mining activities that drive stress redistribution.

The developed system begins with the assumption that immediately following excavation and initial support installation, the rock retains the largest portion of its own self-supporting capability that will ever exist during the lifetime of the crosscut. From that point onwards, there is a constant degradation through further fracturing, crack propagation, deformation, and heave, all of which degrade the crosscut's stability.

At the same time, immediately after installation, the initial reinforcement systems that are installed have their greatest support capacity remaining, and apply compressive forces to the rock, providing confinement and reinforcing the rock's self-supporting capability. Over time, the rock tends to degrade, losing its ability to self-support, and consequently more of the load is transferred to the reinforcement system.

As the load on the reinforcement begins to approach its ultimate strength, the amount of support provided decreases. At this point the rock itself has lost some of its self-supporting capability, and the supporting capacity of the reinforcement approaches zero. From a ground-reaction point of view, this is when the deformation rate, and entry damage rate, tend to increase. Eventually reinforcement support capacity and/or self-supporting capability reaches zero and a rockfall occurs.

The question of how the damage mapping system accounts for rehabilitation/re-support is addressed as follows. Typically, rehabilitation includes installation of new support members and/or scaling of the entry. The entry behaves differently in each case. If new support is installed, but no scaling or removal of old support is undertaken, then the entry benefits from both the new added confinement and support provided to the existing support capacity, while some smaller, unknown residual support capacity remains from the initial support members. Correspondingly, there exists some amount of self-supporting capability in the rock mass, though it is significantly lower than immediately after development.

If scaling is completed prior to installing new support, the rock mass loses the residual strength provided by the already damaged, less-supporting material, and any residual strength remaining in the old support. This leaves more competent rock, better able to support itself, but this remaining rock mass still suffers from a reduction in self-supporting capability relative to a freshly developed entry. The scaled entry is wider than the original, reducing stability. The existing rock support is removed, eliminating any of its remaining capacity, and is replaced with new. Thus, the newly supported entry begins from a 'worse off' position on its path to collapse than after it was newly developed.

While many damage classification systems look at rock condition and support condition as separate issues, such as Kaiser's original Rock Damage Level and Support Damage Level (Kaiser et al. 1992), this doesn't provide adequate consideration for what happens following re-support or rehabilitation as described above. For this reason, the Entry Condition Rating (ECR) considers both rock and support conditions together prior to rehabilitation or re-support, and then looks exclusively at support condition following rehabilitation.

The system uses a simple 0–7 rating prior to any rehabilitation activities, with 0 being freshly developed and supported, and 7 being a rockfall of any type that is not prevented by the existing support (thus rock caught behind the mesh doesn't count as a fall). Following rehabilitation, the entry condition ratings have a different scale numbered from I-VI, expressed in roman numerals to help with clarity. The full descriptions and scales developed can be found in Tables 3 and 4.

Once the ECR system was developed, the database was reviewed again, this time applying the developed ECR system based on the records for each time the entry was visited. Each individual 5 m mapping section was rated independently for each crosscut. Since the system is site-specific, minimal calibration was required to ensure that it properly applies the correct rating to a given location. However, it is valuable to check the system to ensure that it captures the changes in entry conditions over time. To do so the condition ratings for each damage mapping section in each crosscut were graphed for visual verification of function. The ECR progression for PR996o4080 is shown in Figure 6.

In the graphs solid lines are used for the rating before any rehabilitation (left vertical axis) and dashed lines are used after rehabilitation (right axis). The biotite sections are known to deteriorate fastest, and typically first, when compared to other damage mapped sections.

Table 3 Entry condition rating before rehabilitation

Description	Scale
Fresh shotcrete/smooth floor. In effect, a new entry.	0
A few small cracks, generally hairline to 5 mm width. Not usually connecting to one another.	1
Many small hairline to 5 mm cracks, beginning to interconnect, or a few big cracks, 5–10 mm, not interconnecting. Areas around the floor which are not meshed may also begin to show larger cracks or limited deformation.	2
Cracks wider than 10 mm are present (1 crack per 2 m entry length). They may be parallel or subparallel, generally breaking up the surface. Small plates of shotcrete may exist in localised areas. Typically, the shotcrete will be pressing against the support, but the support is not necessarily highly stressed. Bolt plates may be minorly bent or the mesh may be tightening. No broken bolts. The floor may be beginning to show signs of heaving, especially in the biotite contact zone areas along the footwall side of the ore. In areas without mesh there may be shotcrete beginning to fall-off the wall.	3
Cracks wider than 10 mm wide have interconnected and large plates of broken shotcrete are present, generally with side lengths more than 1–2 m. They are typically still adhering to the rock surface, but smaller pieces may also be caught in the mesh support. In an unmeshed area, larger pieces of shotcrete may be found on the floor, though it may also have been large areas of shotcrete that have fallen in many smaller pieces instead of one large piece broken on impact. The floor of the entry may show obvious floor heave, typically less than 30 cm. Signs of wall breakage will be present along the floor/wall intersection at the heave zone.	4
Shotcrete plates are present and may have been caught in the mesh or still be partly adhering to the rock surface. Scaling may be desired in an unmeshed area to prevent material from coming down. In a meshed area the bolt plates are bending, and the mesh is very tight. If there are specific areas bulging outwards, mesh may snap. Static rockbolts may break. Floor heave may be significant, up to 75 cm. At this point there may be signs of shotcrete plates buckling in the walls. (Note: it is possible that entire sections of the wall may appear unbroken yet are actually disconnected from the rock behind. Look for large cracks showing visible gaps behind the shotcrete. In these areas, shotcrete cracking may be minimal. This is more likely to occur in the walls, though typically the roof will still show the expected damage level.)	5
The shotcrete plates have started to breakdown into smaller sized pieces, less than 1 m in dimension. These pieces are likely disconnected and hanging in the mesh. The mesh between bolts will be highly stretched. Bolt plates will likely be significantly bent, and the bolt could be pulling through the mesh and/or shotcrete. Apparently unaffected bolts near other bolts that are highly stressed may be a sign that the bolt has de-coupled from the rock mass and is providing little support. Because of the smaller-sized plates there may be a large quantity of rock or shotcrete chips on the ground where they have broken off the roof or wall plates. Broken bolts and snapped mesh wires are a definite possibility in these zones. In floor-heave zones the heave is likely 75–100 cm or more. The shotcrete in these zones has buckled and broken and the shotcrete from above or below the buckle may be sliding underneath the other side, causing it to push outwards and causing a great deal of strain on the support, and a large amount of real or apparent convergence.	6
Any rockfall has occurred where the support elements were unable to prevent it. These rockfalls need not be massive in size, and might only contain a few cubic metres of material, or may only be the shotcrete falling through broken mesh. An investigation will be required to determine the root cause and severity of the fall.	7

Table 4 Entry condition after rehabilitation

Description	Scale
Second support layer in good condition. No signs of bending, pulling, or stress. Mesh not strained. Fresh new layer of shotcrete without cracking.	I
Mesh tight. Bolts and plates OK. Shotcrete may have begun to form small cracks.	II
Bolt plates showing signs of bending. Bolts stressed. Bolts OK. Larger cracking in new shotcrete, generally larger, separated cracks, buckling possible, especially if not bolted.	III
Bolts showing signs of bending with stress. Plates significantly bent/warped. New shotcrete may begin to form plates. Shotcrete may fall if no mesh in place.	IV
Bolts and/or plates pulling through the new layer of mesh. New shotcrete highly damaged. Bolts may begin to break.	V
Rockfall of any type where the support elements were unable to prevent it.	VI

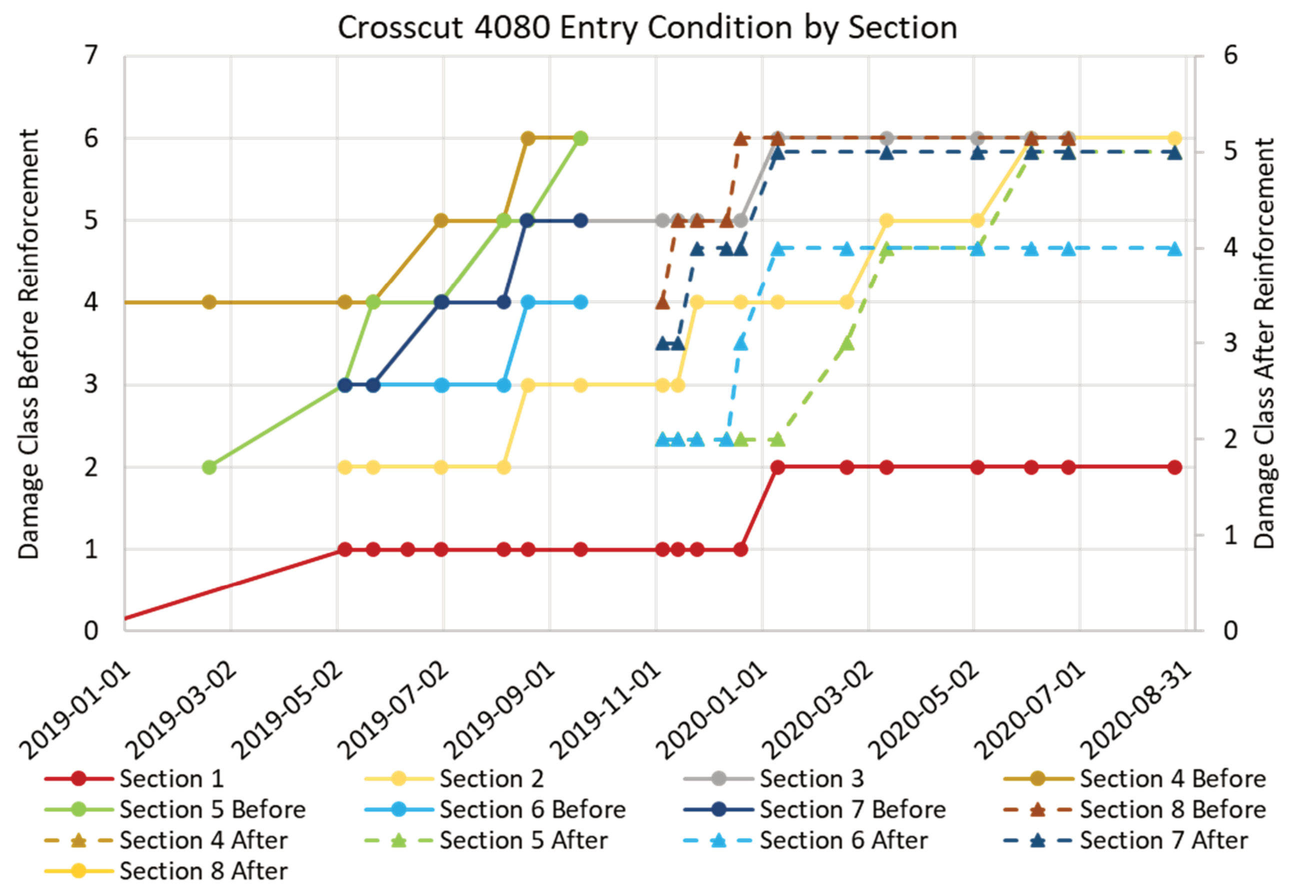

Figure 6 Typical damage progression using the ECR system. Some sections were rehabilitated while others were not

The ECR can also be validated to ensure that it responds to actual stress changes, which is essential for the system to be tied to the actual stress conditions underground. This also ensures that the system can serve as an indicator of deformation magnitude, allowing damage mapping to serve as a method of determining more precisely when the support system is likely to have reached its full capacity, resulting in increased risk, safety hazard and fall likelihood. Thus, it can indicate when preventative maintenance is required.

Figure 7 shows the stress and deformation results from the shoulder of the magnetite profile of the AL1081 2780 crosscut, along with the ECR record from that crosscut. It indicates that the stress cell reached its maximum relative confining stress at the exact same time as there was a large increase in the deformation rate of the extensometer anchor point located closest to the excavation boundary.

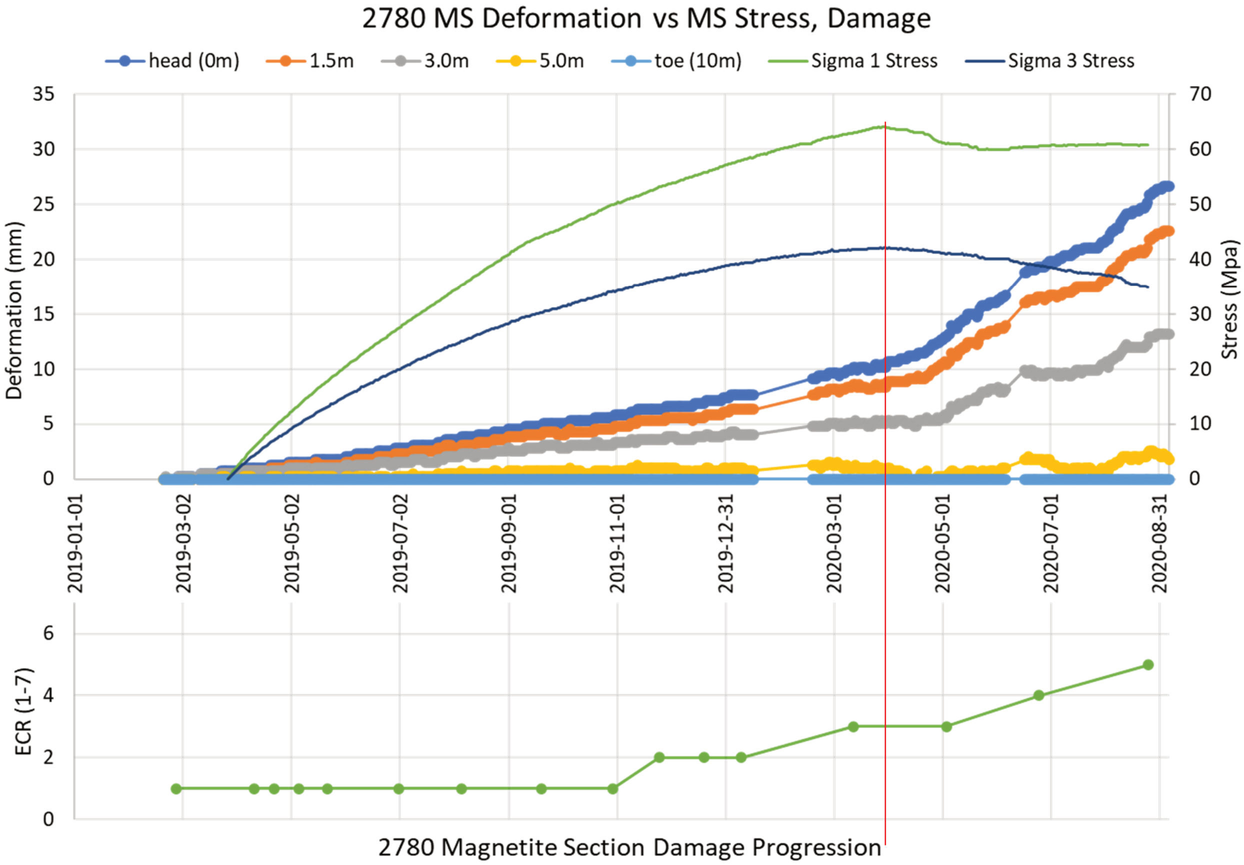

Figure 7 Combination of deformation and stress measurements with the ECR tracking

The peak relative confining stress also occurs at roughly the same time when the ECR has reached level 3. As defined, the level 3 ECR is the highest level ECR the entry can reach before visual damage begins to occur to the support elements. At this level plates begin to show signs of bending, larger cracks begin to interconnect, and the entry begins to show signs of the stress. The correlation between the peak confining stress and the increased deformation rate was something that held true throughout the data.

5 Discussion and conclusion

The correlation between the first visible signs of stress in the crosscut and the maximum relative confining stress is both logical and useful. In most cases, deformation in an underground entry is a natural process that follows from the creation of cracks and fractures in the rock mass. The stresses in the rock induce failures along planes of weakness, resulting in a bulking of the mass, leading directly to deformation. At the same time, the creation of those small failures tends to reduce the ability of the rock mass to transmit stress through it – it becomes weaker and less stiff. Thus, the relationship is sound from a theoretical point of view.

From a practical point of view, the creation of the same fractures and cracks signals the 'beginning of the end' for the rock mass. It indicates a point in time where the material is no longer capable of sustaining itself unassisted, and some of the energy is transferred to the support system which artificially provides confinement and stability for a limited time. Thus, a link between peak relative confining stress, leads to increased deformation rate, which leads to increased energy transferred to the support system, which leads to visible signs of the supports taking load. How long the supports can continue to take additional load depends on the support system design, but, in the face of continued stress being redistributed from ongoing mining, it is only a matter of time until the system fails.

Based on the results of this study, it was recommended that a systematic installation of secondary support should be completed in all high-deformation areas after the damage mapping has indicated the entry has reached a state of ECR 3 to 4. At this point the increased deformation rate tends to lead to rapid degradation of the entry. One of the goals of the study was to provide the mine with a reliable method of easily determining when to install secondary support to prevent problems and hazards, rather than requiring the mine to react to problems and hazards. As such, the secondary support should be installed prior to the total failure of the primary support. Thus, waiting until an ECR of 5 or 6 is unreasonable.

Using observations that indicate a crosscut has reached ECR 4 may allow the mine to install secondary support before it is needed, it is still reacting to the conditions rather than having advance knowledge. The next step is to develop an entry condition prediction method before the excavation is even created. A well-functioning damage mapping system is especially valuable when deformation and stress measurements are coupled to rock quality in addition to damage observations. The system then has the potential to be used in combination with numerical and analytical methods to determine in advance when an underground entry is likely to reach a state of damage requiring response. The ECR system in the Malmberget mine is such a system and these points of research will be the next consideration for the future (Jones & Saiang 2022b).

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