Sublevel caving a narrow vein mine – Pajingo's case study

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ABSTRACT

The Pajingo gold operation in Queensland is approximately 55 km south of Charters Towers and 134 km south-west of Townsville. The gold deposit was discovered on the northern end of the Drummond Basin in 1983 and first open pit production began in 1986.

Mining progressed underground in the 1990's and gold production peaked in the early 2000's as the main Vera/Nancy lodes were extracted, producing 330k oz in 2003 (a record at the time). More recently, as Ore Reserves and grade have diminished, the mine has had to develop innovative mining practices to maximise recovery of the remaining Mineral Resource. This has included a modified sublevel caving method against previously mined/backfilled zones. The method has been coined modified sublevel shrinkage (MSLS). Full scale trials of the new mining method have been successfully completed. This has allowed the site to revaluate the Mineral Resource and include areas into the Ore Reserve that were once deemed sterilised, adding significant value to the operation and extending its life-of-mine.

This paper describes the first of two large scale MSLS trials and key operational learnings from it.

INTRODUCTION

Minjar Gold's Pajingo operations in Queensland are approximately 55 km south of Charters Towers. It consists of the two separate underground mines and a conventional crush-grind carbon-in-pulp processing plant that produces gold-silver dorê. The Pajingo deposit is part of a series of deposits in a precious metal epithermal field that covers 150 square kilometres of sporadically outcropping auriferous quartz veins hosted by volcanics (Parks, 2003). Battle Mountain Gold Company discovered the deposit in 1983 and first gold production began in 1986 from open pit mines. Underground operations began in 1996 and used twin decline access. They reached full-scale production in August 1997. Since 1986, the Pajingo gold operation has produced more than 3.3 million ounces.

Pajingo traditionally employs mechanised narrow-vein stoping methods and more commonly modified avoca with a bottom up sequence. Benches are backfilled in panels ranging from 10 m to 30 m in strike length with run-of-the mine waste backfill. Benches are typically spaced at 15 m to 25 m intervals depending on grade distribution and ground conditions. The majority of the main lodes were mined in the early to mid-2000's with a number of gold bearing quartz domains periphery to the main lodes left behind. The horizontal offset between main and periphery domains is up to 10 m. A significant change in mining method and economic strategy would have been needed to mine these at the time.

In recent years the rate of mining at Pajingo has exceeded the generation of Mineable Resource from near-mine exploration activity. To sustain production from the underground mine, engineers examined an alternate mining method that would allow extraction of the periphery domains that were once deemed "sterilised". Several methods were considered with the only practicable solution a style of sublevel caving (SLC). This involved mining against historic backfilled stopes either longitudinally or transversely while retreating to or along the periphery domain boundaries, allowing the backfill to cave in front of the cave. This has been labelled modified sublevel shrinkage (MSLS) based on its similarity with the method described by Mackay (2014) at the nearby Mt Wright mine.

MODIFIED SUBLEVEL SHRINKAGE (MSLS)

Published examples of bulk mining against fill in a narrow vein setting similar to MSLS could not be found. Preliminary calculations indicated that while possibly viable, the feasibility of the method was sensitive to mining assumptions and inputs such as recovery and dilution factors. For this reason, two large scale trials of MSLS were conducted. Two areas of the mine were identified as suitable locations for the trials due to the minimal development and rehabilitation required for access, as well as their mineralisation characteristics such as geometry and thickness mimicking those of other areas of the mine where MSLS could be viable.

The first and largest trial was located in the hanging wall of the Jandam lode and accessed from the 703/715 levels (Figure 1). These remnant levels were in production at the time and had a strike drive mined in the hanging wall of the orebody extracting a parallel lode with a combination of conventional down and up-hole stopes. Early ore drive development of these levels cross-cut numerous discrete splay veins that connected the main (old) and secondary (current) domains. The splays were further tested with sludge holes drilled from the access drive and subsequent modelling indicated an in-situ Resource of approximately 10 000 t at an average au grade of 5.4 g/t. Based on an assumed 70% metal recovery and 30% unplanned dilution this equated to 13 000 t at 4.2 g/t, above the current underground cut-off grade of 3.3 g/t.

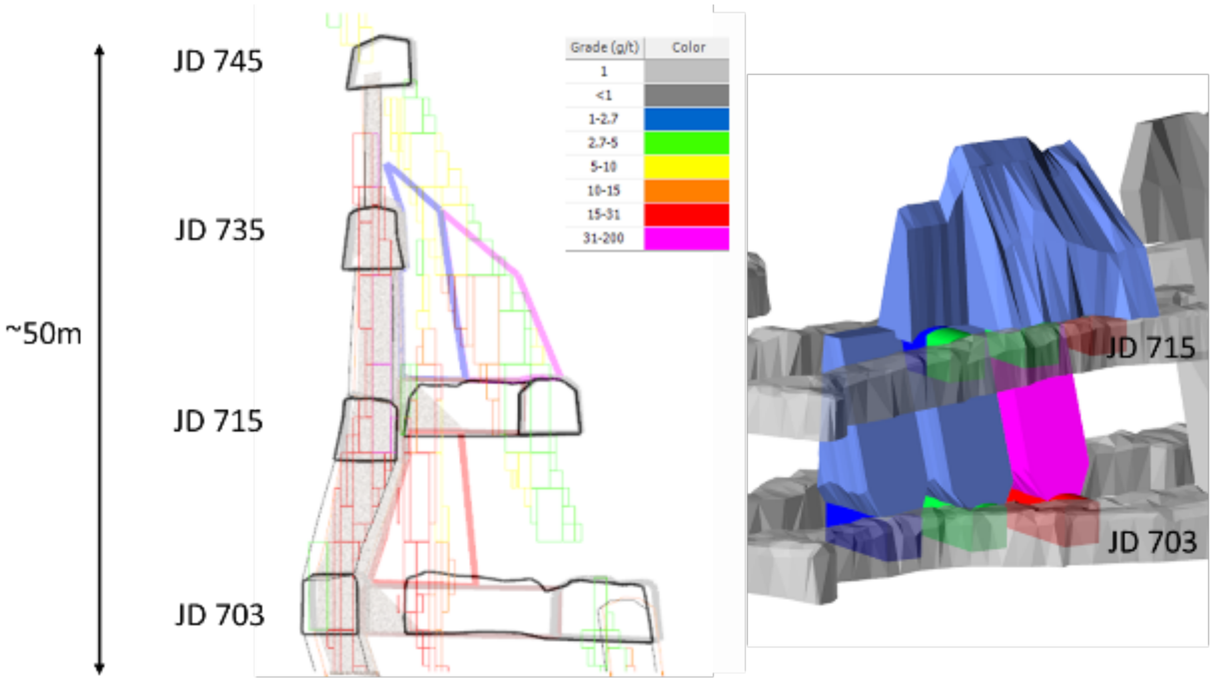

FIG 1 – First MSLS trial area accessed from pre-existing hanging wall development

A risk assessment for the proposed mining method was completed which identified the key risks:

- Unknown behaviour of the fill with potential for it to hang up
- Losing access to the brow or needing to access and open brow
- Poor drill and blast performance leading to poor recoveries
- Reactive ground (confirmed by testing)
- Grade control model performance/call factor

The aim of the trials was to better understand these risks and to test mining assumptions and determine inputs for larger case studies in other areas of the mine.

Development of the 703/715 Level MSLS

Three cross-cut drives were mined from each level. Each cross-cut was mined with the standard site ore drive profile of 4.0 m (wide) x 4.5 m (high) with a 5 m pillar between each cross-cut (wall to wall). Cross-cuts were spaced as close as possible under geotechnical guidelines and aimed to undercut the higher-grade areas. To promote stability of the pillars the cross-cuts were mined on survey control and only the lead in pillars were chamfered. Cross-cuts were named DP1 – DP3 (“DP”- draw point) with the order in which they would be reached by a loader during production. This would then allow the operators to easily remember which draw point was which when bogging to a prescribed sequence.

Ground conditions in the area were fair to good with rock quality designation (RQD) in excess of 70 with typically one joint set present plus a random. Stress conditions were benign due to the area falling within the stress shadow of the mined main Jandam domain. Numerical modelling of the planned MSLS was conducted with minimal changes to localised stresses predicted. Despite this the cross-cuts were supported with a relatively stiff ground support regime of 2.4 m stiff bolts (split sets with cement cartridges) and weld mesh down to grade line. Cable bolts were installed at each intersection prior to turning out each cross-cut. Cable bolts were also installed within the cross-cuts prior to production to support localised faulting.

Initially the cross-cuts were designed to break through into the fill to determine the fill behaviour prior to production drill design. Once the fill was exposed for the first time it became evident that the fill was prone to hanging up, exhibiting cemented rockfill-like properties. It is likely that the fill had consolidated over time as benching progressed upwards and water ingress transported fines into the fill mass. Managing cross-cuts with hung-up fill became a significant challenge with numerous failed attempts at slashing the backfill to “rill”. To best manage this, bunds were built as large as possible with mesh pinned to the backs and draped over the bund to prevent falling rocks from ejecting out of the open draw point. All remaining cross-cuts were subsequently stood-off 2.5 m from breakthrough to avoid the hazard.

Drill and Blast

Drill Design

Upon completion of the cross-cut development the backfill interface ahead of the face was probed at a 5 m x 5 m nominal spacing across what was expected to be the full exposure of fill during MSLS operations. This was done to test the accuracy of the void model (most of which was comprised of design shapes only) and allow better placement and stand-off of production holes. The probe holes were then used to install markers as part of a marker trial discussed later.

Drill design parameters were based on principles established by larger SLC operations presented by Trout (2002) and te Kloot (2017) as well as site-based experience with larger scale stoping involving fanned rings. The drill design aimed to fragment the rock finer than the exposed backfill material exhibited during development to allow preferential flow of the broken ore. Initial drill pattern parameters chosen are presented in Table 1.

TABLE 1 - Drill design parameters used for the MSLS

Drill Design Parameter	Design	Comment
Ring spacing/burden	1.8 m	Maximum bog depth of Caterpillar 1700 loader
Toe spacing	1.8 m	Based on ring spacing and previous site experience with bulk stoping (fanned rings)
Holes per ring	7-10	Max 2 holes within apex (between cross-cut pillars)
Ring dump	10°	
Hole diameter	76-89 mm	Dependent on primer type/size
Shoulder angle	70°	Steepest angle expected to recover as primary draw
Max hole length	22 m	Limit of drill fleet (note 22 m requires hand loading 3 drill rods with current drill setup)

Breakthrough cuts were planned for the jumbo with stand-up rings to be drilled with the production drill rig back to conventional ring patters. Production holes against the fill were stood off 0.5 m at the toe perpendicular to the fill boundary (as determined from probing) to confine the charge column.

Adjacent rings were drilled parallel with each other and not offset. This was done to avoid committing to a ring firing sequence and allow flexibility in the firing schedule. Toes of interacting holes between cross-cuts were stood off from each other by 1 m. This was practically achieved by simply shortening the hole that was in the ring suspected to be fired first in the schedule as it would have the greatest radial burden to the adjacent ring (Figure 2).

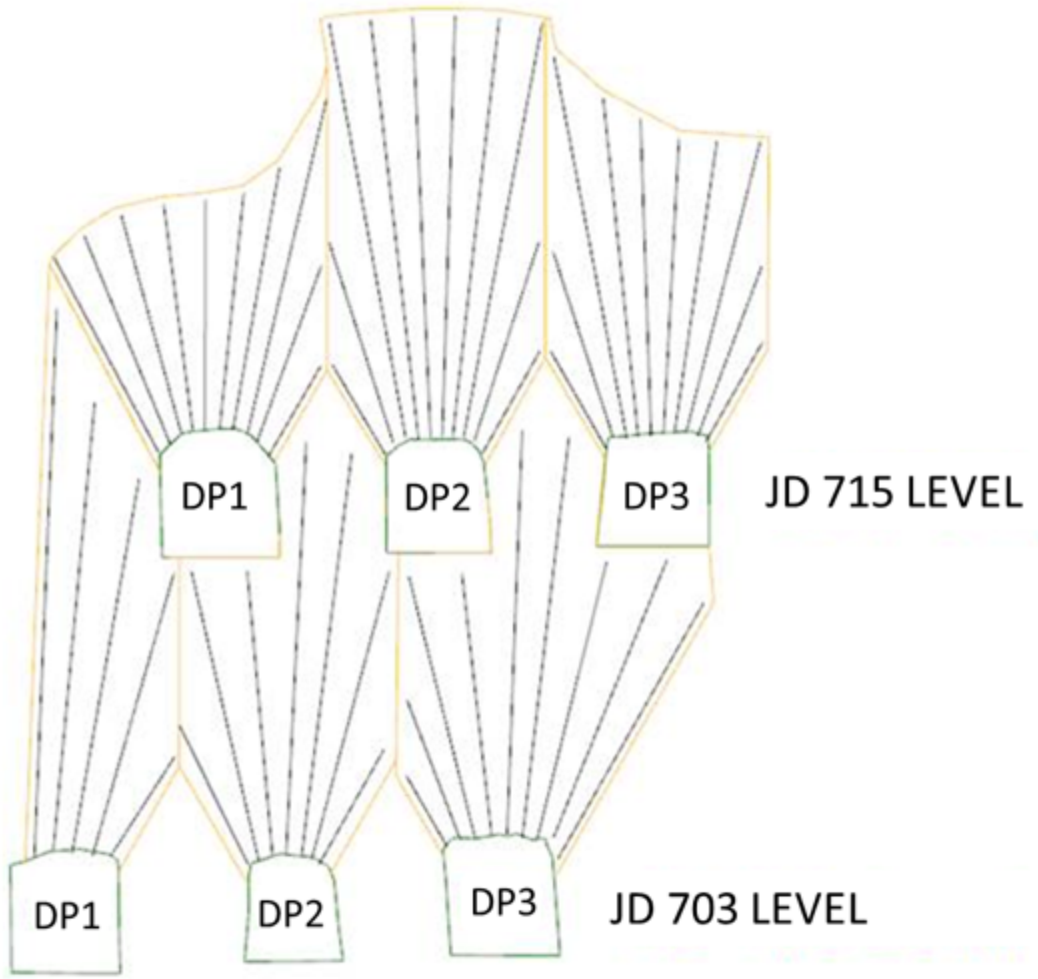

FIG 2 – Final ring design across both levels showing the first full height ring of each DP.

Following the risk assessment, a decision was made by site personnel to mitigate the risk of injury posed by open brows and hung-up fill. To manage this each level of the MSLS was to be completely pre-drilled and pre-charged with Orica's wireless blasting technology WebGen™. This necessitated the need to drill 89 mm diameter holes instead of the site standard 76 mm for all holes that were to be pre-charged and "slept" (the minimum hole diameter to fit WebGen™ primers).

Drilling

Drilling was completed by the production drilling contractor with an Atlas Copco Simba H1257 drill rig with T45 running gear and Minnovare's Production Optimiser™ installed. The Production Optimiser™ has shown to reduce average drill deviation at Pajingo from 3.8% to 1.6% (measured as percentage of hole length) based on surveyed down hole breakthroughs. The engineers considered drill accuracy to be critical to success of the MSLS and ensured that drilling was only done with experienced drillers on the drill rig with the Production Optimiser™ installed. Survey checks of the drill rig setup (picking up points on the rail) on hole collars were completed sporadically.

Holes drilled prior to production drilling including probe holes, sludge holes etc. were grouted prior to commencement of production drilling. Any production holes that inadvertently broke through into fill were redrilled 0.5 m shorter. Production holes that intersected unknown open drill holes were also redrilled to reduce the risk of interaction between fired/pre-charged rings.

Drill pressures required adjusting to suit the larger 89 mm holes. This was adjusted in the split feed that would otherwise be used for adjusting reaming pressures. This allowed the driller to quickly adjust between drilling 76 mm and 89 mm holes. Despite the larger hole sizes drill rates exceeded scheduled rates, on numerous occasions exceeding 300 m per shift, reducing the drilling cost unit rate by 5%. This was facilitated by drilling up-holes in good ground, minimal rig movements and the use of the Production Optimiser which has shown to more than halve the time between holes due to ease of setting up on each collar. This was particularly beneficial in this case as often the rig was heavily articulated around the DP pillars and would have otherwise needed the offsets strung from wall pins across the access drive.

Following production drilling the jumbo drilled each break through cut, adjusting the backs holes to suit the lowest/flattest drilled production holes. This typically meant the back three rows of the development cut would be progressively angled up to match the first dumped production ring. The face plan was then given to the engineers to include in the charge plans.

Charge Design

Following drilling of each level all production holes were prepped carefully with all feedback regarding hole quality and potential interactions between holes/rings noted. Poor or unclear prep results resulted in the charge crew returning and re-prepping the holes.

All MSLS charging was done with high density (1.0 g/cc) inhibited emulsion to maximise the allowable sleep time in the reactive ground. Based on reactive ground testing from samples taken in the 703/715 levels a maximum sleep time of 7 days was permissible with the use of inhibited product. This would ultimately dictate the firing and bogging schedule to ensure all pre-charged rings were fired within 7 days of being charged.

Stand-up rings against the fill were charged with standard iKon electronic detonators. Break through cuts were charged with long period (LP) detonators with an iKon detonator timed zero attached to the detonator cord. All remaining holes to be pre-charged and slept were charged with WebGen detonators. Primer placement was staggered between holes and rings following standard site standards for priming rules (single prime for charge lengths less than 7 m, double prime for charge lengths between 7 and 12 m and triple priming for charge lengths greater than 12 m).

All three draw points were planned to be fired together in the first blast to unconfine the backfill simultaneously and reduce the risk of hole interaction between draw points. This included the break through cuts up to the first full height production ring. This exceeded the sites standard tolerance for swell factor of 25 – 30% however it was expected the backfill would compress up to 0.5 m (based on previous tight firing experience) and a portion of the first few rings would eject back into the access drive allowing additional void for swell (Figure 5).

Holes and rings were timed such that the entire level could be fired in one firing should there be any significant issues during the charging process that would delay the first firing to 7 days after the first rings were charged. Delays between holes were generally 21 milliseconds (ms) with 50 ms additional delay between subsequent rings in each draw point. A 7 ms offset between holes in adjacent rings meant that the maximum instantaneous charge (MIC) was kept to a minimum. The only holes timed to detonate at the same time were interacting holes between rings which were typically the shortest holes.

Minimum uncharged collar lengths of 1 m were initially designed for the 715 level. This was increased to 1.5 m for the 703 level following evidence of brow overbreak. Holes that were suspected to have intersected unknown drill holes were charged so that that section of the hole was left un-charged with either an extended uncharged collar or decked charge. Holes that intersected void or vuggy ground were sleeved to avoid decoupling of the emulsion column. Critical radius of 0.5 m was applied and adjusted visually in Deswik (mining software used on site) ring by ring. Shoulder holes were designed with the minimum un-charged collar by default.

Individual charge plans for each ring were issued to easily allow individual rings to be fired separately or grouped together during the WebGen firings.

Charging

Charging was a collaborative effort between site personnel and Orica personnel under the conditions of use of the WebGen detonators. Charging was done in a sequence that maximised the effective sleep time by charging what was to be fired last as late as possible. All production charging was done on day shift to manage the charging quality. Charging was carried out over 3-4 days (per level) and progressed slower than scheduled due to multiple pass charging with WebGen detonators. The charge hose could not fit beside a WebGen primer inside an 89 mm hole meaning triple primed holes required 4 passes of charging. It was worth noting that the actual hole diameter could have been 82 – 89 mm due to the use of re-sharped drill bits. The speed of charging was also inhibited by the limited carrying capacity of the charge vehicle (1.3 t) that required multiple trips to the magazine to fill up with emulsion.

All production holes were bagged off with MTI gas bags at the design un-charged collar length. Red/green caps were also used as an added defence to hole ejection. Each WebGen primer was tied to a tether tape that was left protruding from the collar of each hole (Figure 3). To assist with identification of potential hole slumping all tether tapes were trimmed to 0.5 m from collar with a knot tied at the hole collar. Each WebGen primer was uniquely labelled to identify them during charging as well aid in identification and investigation of misfires.

FIG 3 – 703 DP3 after charging face and rings showing tether tapes of pre-charged holes.

The date and time each ring was charged were recorded and displayed on the muster room wall. This was used to track the progress of firings and to highlight dates when each ring would reach its maximum 7 days sleep time and need to be fired.

Firing and Bogging

Due to the limited standoff between the cave and the access drive it was expected that following the first firing access past DP1 would be blocked by the rill profile (Figure 4). To manage this the floor of the access drive and cross-cuts was filled in with 1 m of waste mullock immediately prior to charging. This reduced the horizontal reach of the rill towards the cross-cut and allowed access past DP1 to access DP2 and DP3 post firing 1. Filling the floors also minimised the cross-sectional area of the draw point so that in the case of a sudden rill from an open brow the loader was less likely to be engulfed (more likely to be pushed out) as well positioning the cab of the loader closer to the backs away from the line of fire (Figure 4).

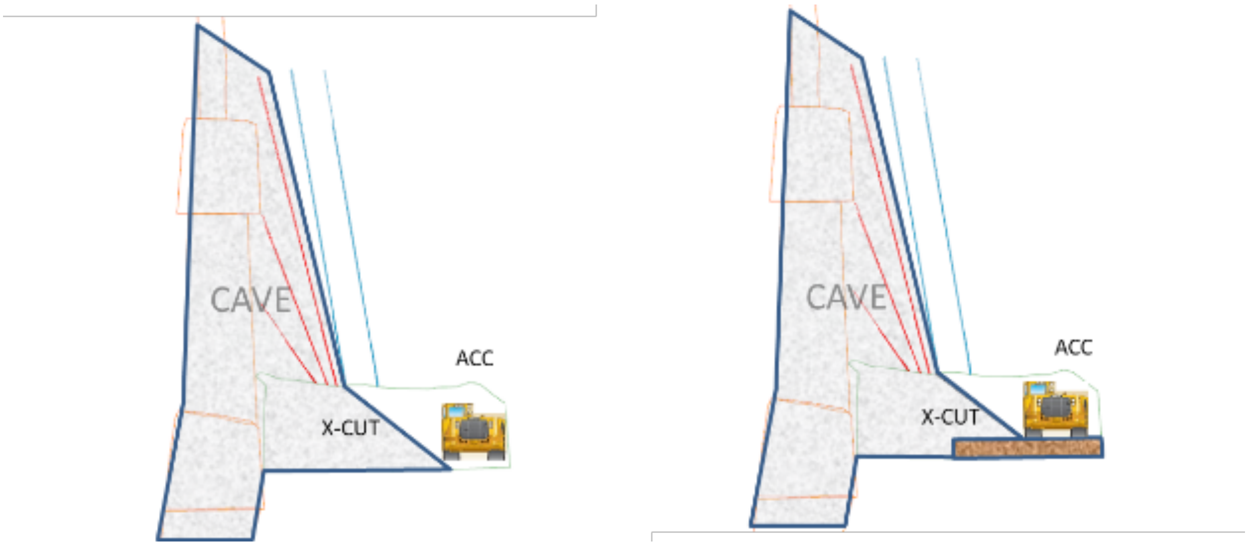

FIG 4 – Unfilled floor (left) and 1 m filled floor (right), red rings 1st firing, blue rings are pre-charged

Generally, the firing and bogging schedule was constrained by the maximum sleep time of the pre-charged rings. Engineers aimed to minimise the risk of needing to fire out of sequence or earlier than the draw point status would warrant by aiming for as much primary recovery as quickly as possible following each firing.

All three draw points were fired together in the first blast to unconfine the backfill simultaneously to promote caving and choked brows as well as reduce the risk of hole interaction between draw points. This appeared to work well with all three draw points on the 715 level found to be choked with a mix of broken ore and waste following the first firing (Figure 5).

FIG 5 – JD 715 looing along the access past the three cross cuts after the 1st firing

Sound engineering judgment was applied when deciding which rings would be fired each day based on the following principles:

- Grade performance of each draw point and quantity of quartz in the rill (all quartz was assumed to be grade bearing based on sludge results and since sample assays could not be returned quickly enough to feed into daily decision making)
- Rolling schedule of remaining rings to be fired and their respective maximum sleep times
- Expected grades of each firing (higher grade rings targeted to fire as early as possible)
- Brow location and expected rill toe location with respect to accessing past DP1 to DP2 and DP3

Following each firing the expected tonnes for each draw point was updated on the back of the shift sheet. Bogging quantities and cycles were prescribed following visual inspection by either an engineer or geologist. Grades of the recovered material lagged bogging by 24 hours so often the decision to continue bogging a draw point was based on visual observation of the material and an estimate of the quartz content relative to waste material, compared to previous days material and returned assays. Grades from 715 level were generally far better than planned encouraging continued bogging above 100% recovery delaying subsequent firings. This then condensed the remaining firings however this risk was offset by the above planned metal recovery from initial firings.

Bogging was done conventionally while all draw points were fully choked off. Bogging was done on remotes only as soon as one of the draw points opened. This occurred approximately halfway through extraction of 715 and almost immediately in 703. Since the remaining rings were pre-charged with WebGen wireless detonators there was no need for personnel to access the brows which allowed bogging to continue after the brows were cracked. The risk of not recovering material left behind was deemed greater than the risk of losing a remote loader within the cave. Grades from 715 level warranted continued bogging well after the final firing until shut-off grade, ultimately bogging 150% of total planned tonnes for the level. Upon completion of bogging, the 715 level resembled a large open stope rather than a SLC.

Both the 715 and 703 levels were serviced with an ore pass system allowing rapid extraction of the MSLS as bogging from the draw points and loading trucks could occur simultaneously. This allowed in excess of 1000 t per shift to be moved from the cave when needed and reduced the need to fire rings when ore remained in the rill. This also permitted haulage from other areas of the mine while bogging of the MSLS continued, de-risking the scheduled delivery of ore.

RESULTS

Generally, the 703/715 MSLS trial performed exceptionally well, recovering 3472 oz compared to planned 1876 oz. Overall achieved recoveries were higher than planned and were estimated from the recovery of markers and a cavity monitoring scan (CMS) obtained in 715 DP1 once bogging was complete. Fill dilution was far higher than planned and was determined from bogged tonnages and estimated recoveries of fired material. The results are summarised in Table 2.

Surprisingly the backfill caved immediately following the first firing in 715 level. Visual observation of the material indicated that the mix of backfill and ore ranged between 20%/80% and 70%/30% respectively throughout the bogging cycles. Despite the higher than planned fill dilution, grades performed exceptionally well. Assuming 90% of the designed shapes were recovered, this leaves 1050 oz excess that originated from either the fill or grade call error. From fill alone this would represent an average fill grade of 4.5 g/t. Despite historical cut-off grades higher than this (meaning material of this grade could have been used as fill) this was considered unreasonable. Commentary from site geologists indicated that the grade call factor could be ±30% which would account for approximately 700 oz of the 1050 oz if the grade was under called, leaving 350 oz attributed to the fill/external dilution (15 000 t @ 0.72 g/t).

TABLE 2 - Design and actual tonnes/grade for the first MSLS trial (*includes planned dilution and recovery, #estimated based on 715 CMS and marker recovery)

		Planned*	Actual
715	Tonnes	7501	11475
	Grade (g/t)	4.8	4.2
	Ounces	1164	1557
703	Tonnes	5763	12805
	Grade (g/t)	3.8	4.7
	Ounces	712	1915
Total	Tonnes	13264	24280
	Grade (g/t)	4.4	4.4
	Ounces	1876	3472
	Dilution	30%	90%
	Recovery	70%	90%#

The "fill" dilution is likely to consist of a small amount of high grade (>20 g/t) remnant stope material that was left unrecovered within the stope for a variety of reasons. Only a few hundred tonnes of this material would be needed to yield 0.72 g/t across 15 000 t of pure waste rock. For this reason, it is likely that the true fill grade was closer to 1 g/t, possibly higher.

Drill and Blast Performance

Drill and blast performance on 715 level was exceptional with no bridges or hang-ups reported. Fragmentation appeared good in general with the majority of the oversize material exhibiting oxidised surfaces indicating it originated from directly adjacent to the backfill. Following the initial firing there was evidence across all three draw points of brow break back from the closely spaced stand-up collars despite 1 m uncharged collars. There was no evidence that pre-charged rings were jeopardised as a result however without the use of wireless detonators the leads of the pre-charged holes would have been cut and access to those holes lost. Minimum un-charged collars were increased from 1 m to 1.5 m for the 703 level to assist preserving brow positions. The longer un-charged collars worked well in preserving brows.

Initial recoveries from 703 level were poor (less than 50% of fired tonnes in 703 DP1). It quickly became apparent that the breakthrough cut in at least one draw point had not broken through into the fill. This was observed by the remote loader reaching what appeared in the cameras as a hard rock wall approximately 1.5 m past the last row of installed bolts in DP1, 1 m short of the fill, exhausting the draw point of all broken material. The other two draw points appeared to leave small skins against the fill with good recovery of planned tonnes. The exact cause of this is unknown however the breakthrough cuts in 703 were bored by a different jumbo operator and only had two easer holes whereas 715 breakthrough cuts were bored with a more traditional 4 easer pattern. Exact hole standoff distances to the fill and charging techniques between the two levels were not monitored closely enough to determine their respective impacts. Subsequent rings (pre-charged) were fired in the hope the ground in front was sufficiently broken to achieve full height and break into the 715 level. This was successful and recoveries of subsequent ring firings in DP1 improved significantly, breaking through to the 715 level as planned. Secondary recovery of 715 material from the 703 level was measured from the marker trial.

Maximum toe spacing was increased from 1.8 m to 2.0 m for the 703 level which dropped one hole from each ring. This was done to test the sensitivity of the design to toe spacing, aiming to reduce the collar density (an issue due to the smaller drive profiles). Based on the total recovered tonnes and marker trial results it appeared the increased toe spacing had a negligible impact on performance. The resultant increased collar spacing is believed to have contributed to the improved brow performance at 703 level.

During bogging from 715 level two WebGen primers were recovered in the rill. One was fully assembled while the other comprised of the battery pack only with evidence of booster detonation. Neither of these primers could have been within a charged column when it was initiated (holes confirmed as initiated from blast monitoring results). The planned placement locations of these primers were investigated based on their unique label. The fully assembled unit originated from the first pre-charged ring in 715 DP1. Its exact location was towards the toe of the hole against the backfilled stope with 0.8 m true burden (Figure 6). It is likely that this primer became dislodged following the first firing before reporting to the draw point. Subsequently, the 703 level initial stand-up firings were adjusted to include an additional ring so that the first pre-charged ring had the full 1.8 m burden to reduce the risk of this reoccurring.

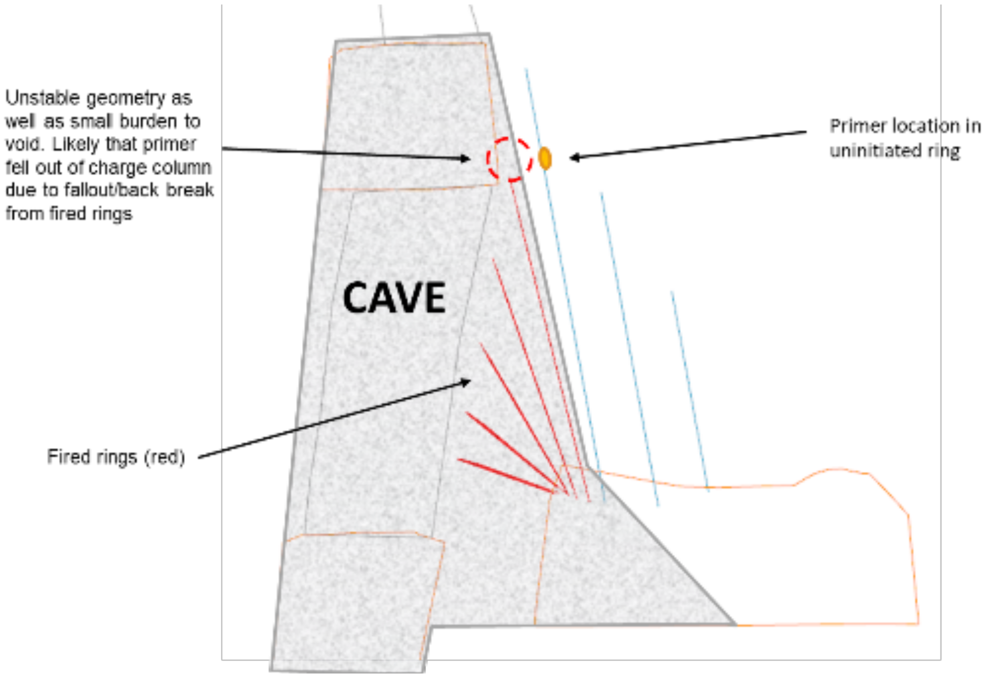

FIG 6 – Origin of assembled WebGen unit

The second WebGen unit discovered originated from the second pre-charge ring in DP1 4 m from the collar. There was a full 1.8 m burden between this ring and the ring in front. The previous ring had more than 100% of its fired tonnes bogged prior to firing the ring from which the unit was found. Based on this it was surmised that the unit was likely dislodged from the hole during the ring firing by an adjacent hole and detonated away from its charged column, not as a result of ring back break from the previous ring fired.

Marker Trial

A simple marker trial was conducted to gain an understanding of recovery of the higher-grade material within 3-4 m of the fill boundary. The recovery of this material was considered critical to the success and continued use of the MSLS method.

Markers were constructed from 40 mm diameter stainless steel pipe, cut at 150 mm lengths (expected mean particle size of the cave material) with a unique number welded on the outside. Reflective tape was wrapped around the two ends of each pipe before the outside of the pipe was lathered in casting resin to give the reflective tape some abrasion resistance. Basic tests completed on site prior to the trial found that without a coat of casting resin the reflective strips would be removed easily by firing/bogging and make the markers difficult to see underground.

Sixty-nine markers were installed across both levels in the probe holes drilled prior to production drilling. This resulted in a ring of markers between each stand-up ring, 3 holes per ring. Markers were spaced evenly in holes at 2 m intervals and fully grouted except for 4 markers that were simply left on the floor of 715 DP2 and DP3. Markers were unable to be installed in 715 DP2 and DP3 probe holes due to unsafe proximity to the open brow where the cross-cut drives had broken through. Based on similar marker trials (Power, 2004) it was expected that approximately 20% of the markers would be found visually and the remainder found by the electromagnet above the crushed ore belt at the mill.

At the time of writing 46 markers (67% of the installed 69) had be found with markers still being recovered by the mill from stockpiled material. Of the 46 markers recovered 11 were found visually at either the draw point, ore pass or surface stockpiles, representing 16% of the installed markers. Preliminary results are shown in Figure 7.

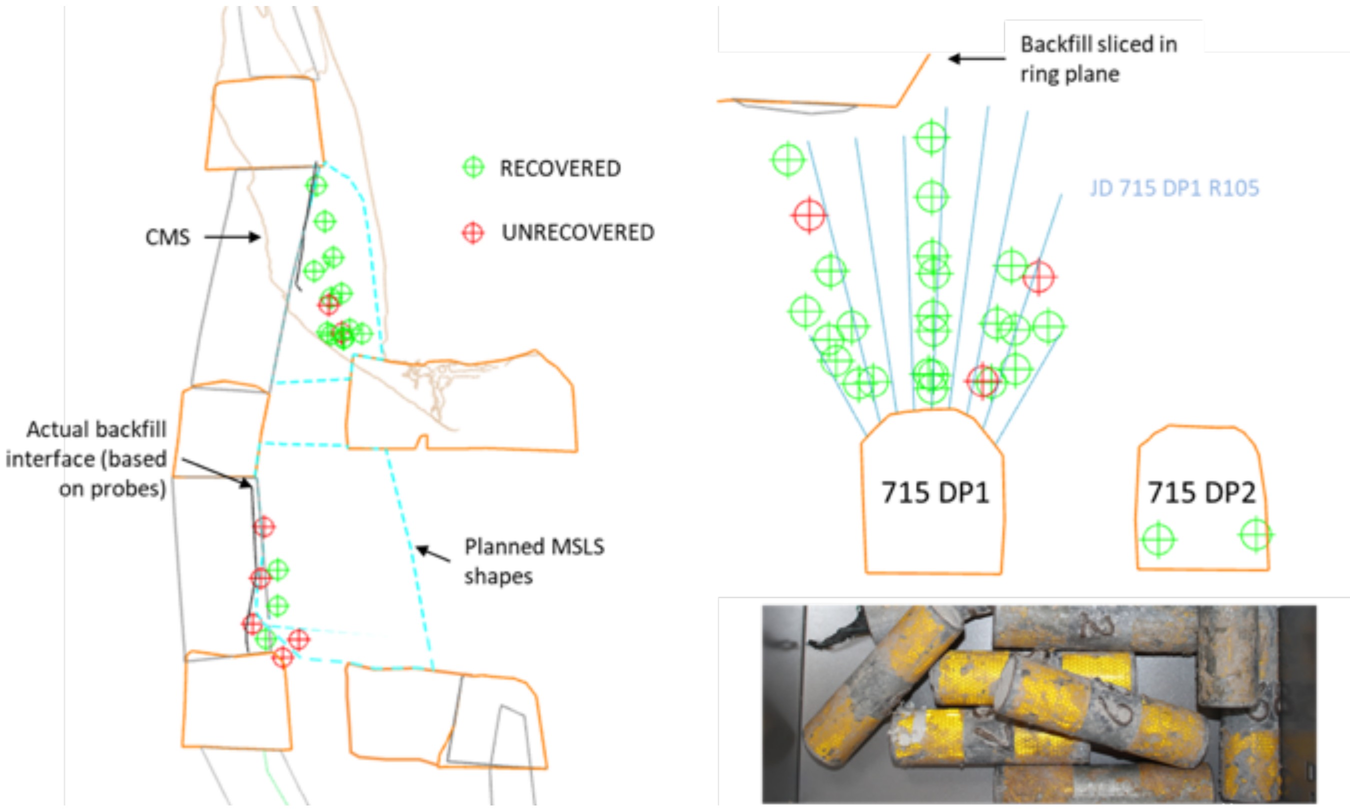

FIG 7 - Preliminary marker results from 715/703 (cross-section coincident with DP1 on both levels) showing recovered makers (green and bottom right) and unrecovered markers (red) at the time of writing.

Recovery of markers installed in 715 confirmed good recovery of the higher-grade ore immediately adjacent to the fill. This was also confirmed with an opportune CMS taken from DP1 (Figure 7). The maximum ring/draw width was achieved (8.2 m) with recovery extending to the 70° shoulder holes. Markers installed in the floor of 715 DP2 were recovered from 703 level confirming secondary recovery and successful breakthrough to the 715 level from 703. Only 3 markers installed in 715 were not recovered at the time of writing, one of which was confirmed as unrecovered by the CMS.

Results from the 703 level confirmed that poor recovery was achieved immediately against the fill (Figure 7), particularly in DP1, despite total tonnage drawn from the level exceeding the plan. Again, the majority of markers installed proximate to the shoulder holes were recovered. Due to the nature of using an ore pass system and surface stockpiles the true timing of marker recovery was difficult to determine (since only 4 of the 46 markers recovered were found at the draw point) preventing any form of flow analysis.

Other Operational Successes/Challenges

The main risk associated with this mining method was considered the uncertainty with regards to the fill behaviour with potential for the draw points to open prematurely, prohibiting access. This was controlled with pre-charging with wireless detonators which was successful in removing the need to access the brows at all once they opened (which occurred regularly during bogging). This was considered extremely successful in the respect that primary recovery could be maximised from over bogging tonnes, particularly early on when grades were higher, and at no stage was personnel access to the brow required, nor was working on draw point rills.

Filling of the floors was in the most part successful in maintaining access past DP1 and DP2 for as long as possible. Poor floor fill material was used to fill the floor in 703 level which created numerous issues during bogging with occasions when the remote loader became stuck, articulated around a draw point pillar after its wheels would sink into the fill. Bogging around the pillars was tight and was limited to the Caterpiller 1600 loader for the most part particularly on remotes. Due to the limited cross-cut length the loader was unable to straighten up before bogging the rill which reduced its ability to dig to full depth but also meant the rill would be bogged from an angle and not draw evenly. The offset between the access drive and the cave was in this case fixed however future MSLS work will include additional cross-cut lengths to manage these issues.

APPLICATIONS OF MSLS AT PAJINGO

The primary purpose of MSLS trials to date has been to determine key mining inputs for areas of the mine with potentially considerable Mineral Resources that require substantial capital investments to access. One of these areas is the Vera MSLS (Figure 8) which contains approximately 320 000 t at 3.2 g/t au for 33 000 oz (including fill dilution at 1 g/t). This potential Resource has been regarded as sterilised in previous JORC compliant reserve statements and has now been included in the 2020 mine plan as a Mineable Resource. It is estimated that there is up to 100 000 oz contained in similar areas of the mine that with further trials and planning will become viable to mine with the new mining method.

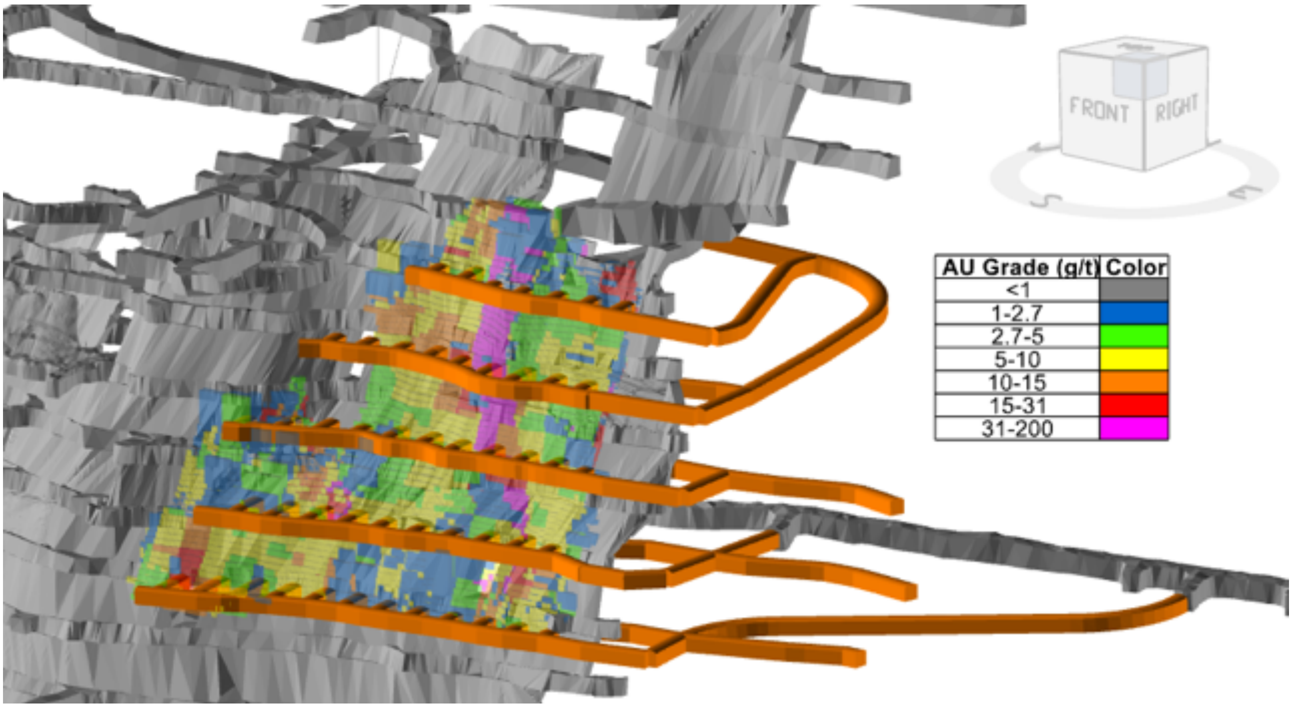

FIG 8 – Planned Vera MSLS showing mined/backfilled areas (grey) and planned extraction levels showing modelled grades.

CONCLUSIONS

By challenging traditional mining methods at Pajingo the technical and operational teams have identified a viable mining method to extract resources proximate to historically mined and backfilled areas. Modified sublevel shrinkage (MSLS) has proven to be a safe and effective method of caving against fill and is planned to be incorporated into future mine plans. This will unlock significant resources left behind and extend the production schedule to overlap with the planned commencement of ore delivery from the new mine, Lynne in 2020/2021.

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