

Appendix A

Noise and vibration impact assessment

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27 June 2022

Bonnie Coxon
Manager Integrated Planning
Cowal Gold Operations

Re: E210776 – CGO Underground Development Modification 1 – Noise and vibration impact assessment

Dear Bonnie

1 Introduction

1.1 Background information

The Cowal Gold Operations (CGO) is an operating gold mine on the edge of Lake Cowal, approximately 38 kilometres (km) north-east of West Wyalong in the Bland Shire local government area. CGO is owned and operated by Evolution Mining (Cowal) Pty Limited (Evolution).

Evolution operates CGO under two Ministerial development consents. DA 14/98, which was granted in 1999, generally allows open-cut mining and ore processing on site, until the end of 2040. SSD 10367, granted in 2021 generally allows underground stope mining, backfilling of stopes and delivery of ore to the processing plant, also until the end of 2040.

Evolution is seeking to modify SSD 10367 (Mod 1) (referred hereafter as the 'proposed modification'), pursuant to Section 4.55(2) of the *Environmental Planning and Assessment Act 1979* (EP&A Act) to change the access points to the underground mine, to change the geometry of access tunnels and to increase the annual production rate.

This letter provides an assessment of the potential noise and vibration impacts from the proposed modification.

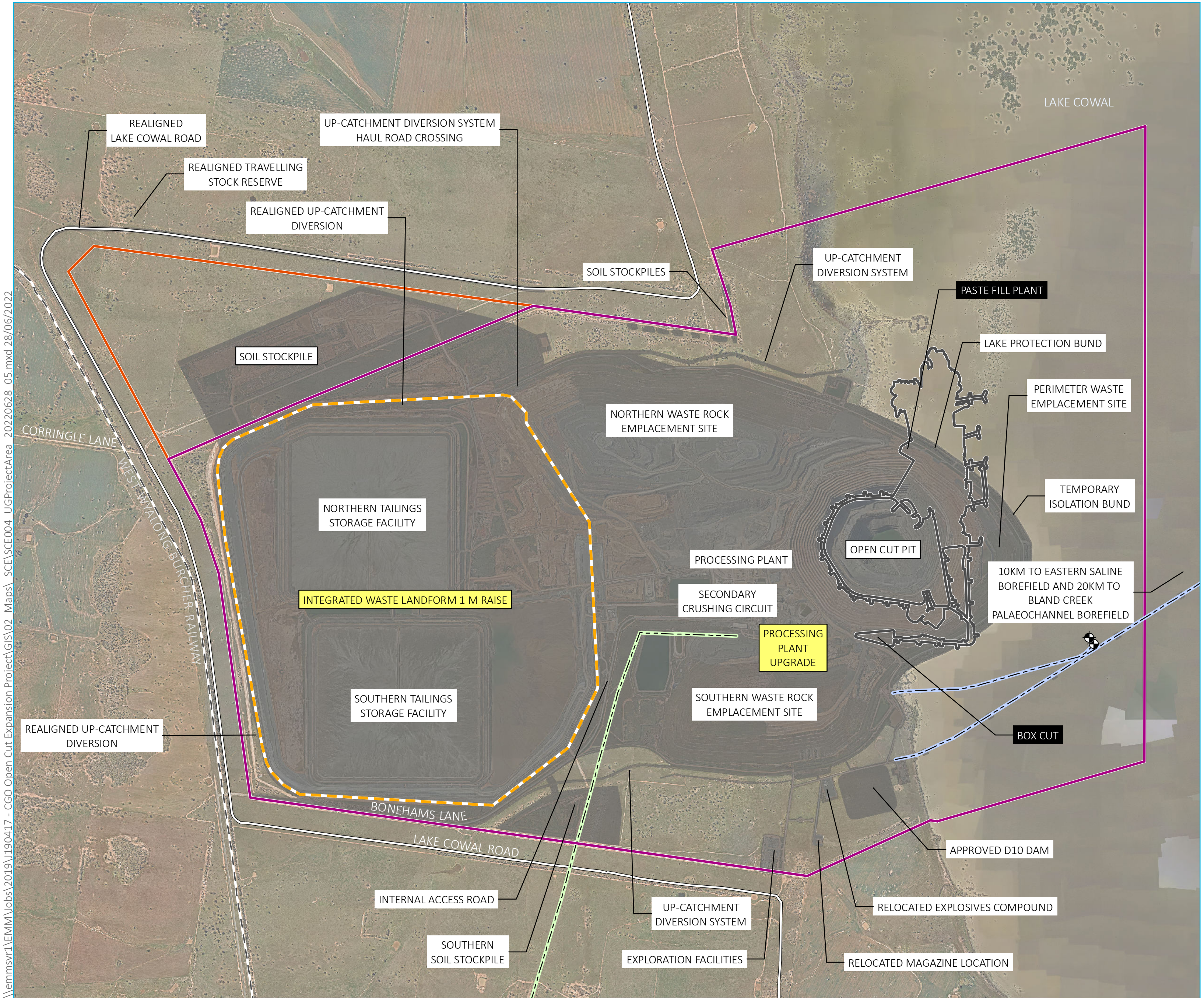
1.2 Approved project

In September 2021, The Minister's delegate approved the CGO Underground Development Project (SSD 10367) (referred hereafter as 'the underground project'). The development consent allows:

- underground stope mining at a rate of up to 1.8 million tonnes per annum (Mtpa), until the end of 2040;
- delivering ore to the CGO processing plant;
- developing a paste fill plant to make cement paste from tailings, and backfilling the stopes with the paste; and

- developing primary and secondary access points to the underground mine, including a box-cut entry, to provide personnel, materials, ore and waste rock haulage and ventilation services.

The layout of the CGO site including the underground project is shown in Figure 1.1. The general design of the underground mine workings, including access points and access tunnels, is shown in Figure 1.2.



KEY
- SSD 10367 approved underground development
- DA14/98 approved surface disturbance
- Mining lease (ML1535)
- Mining lease (ML1791)
- Indicative integrated waste landform perimeter
- Saline groundwater supply bore
- Water supply pipeline
- Electricity transmission line
- Rail line
- Major road
- xxx Approved underground development elements
- xxx Approved Mod 16 surface elements
- xxx Approved surface elements

Source: EMM (2022); Evolution (2021); Nearmap (2021); DFSI (2017)

0 1 2 km
GDA 1994 MGA Zone 55
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Project area - approved underground and surface operations

Cowal Gold Operations
Underground Development Modification 1
Noise and Vibration impact assessment
Figure 1.1

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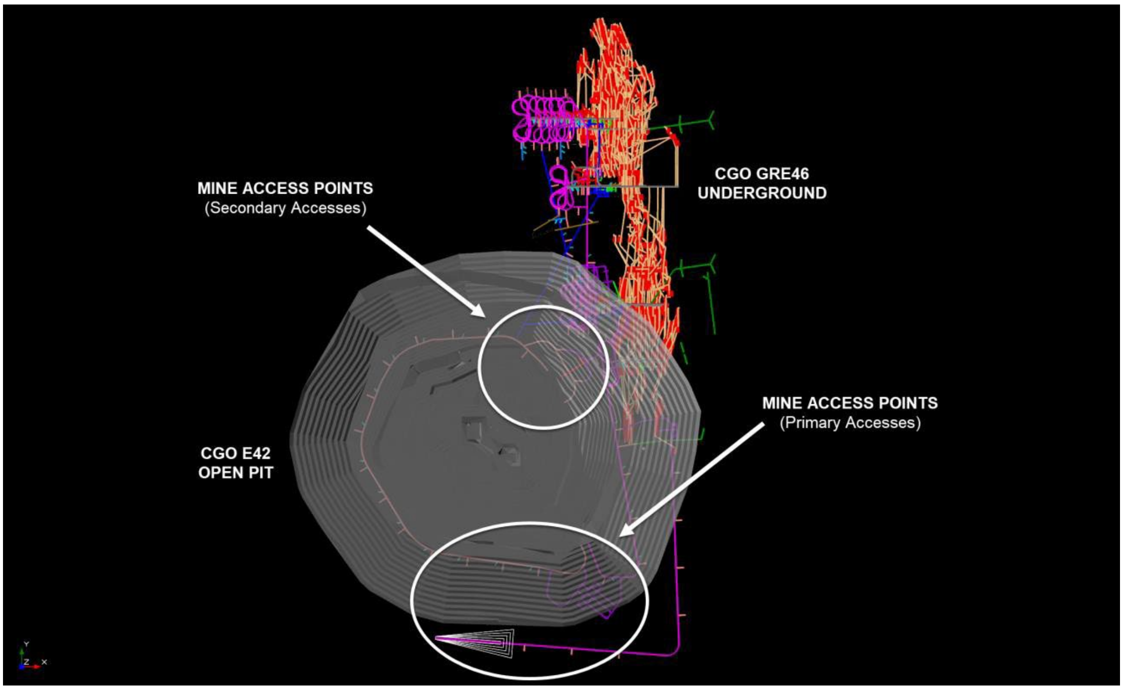


Figure 1.2 Approved underground workings and access points

1.3 Proposed modification

1.3.1 Overview

Since the approval of the underground project, Evolution has undertaken further technical design work and mine scheduling in order to optimise the development and ongoing operation of the underground mine.

This has resulted in changes to the underground mine design, including changes to the access points and taking the access tunnel development from the hanging wall (ie located predominantly on the western side of the orebody) to the footwall on the eastern side of the orebody. Further design iterations have resulted in proposed changes to the way access to the underground mine is accessed.

A review of the mining schedule identified that the underground project can be undertaken to achieve a higher annual ore production rate than the approved rate of production.

The proposed modification seeks to:

- change the way in which the underground workings are accessed from the surface (ie no box-cut);
- change the geometry of the access tunnels to the underground stopes; and
- increase the production rate from 1.8 Mtpa to 2.6 Mtpa.

No changes are proposed to the maximum amount of ore that would be extracted over the life of the project, the approved mining method or stoping areas or the operation of the pastefill plant.

It should be noted that the above changes will not require a modification to the approved existing open-cut mining, processing plant and integrated waste landform operations. These supporting activities will continue to be regulated under DA 14/98.

The proposed changes to the underground project are shown in Figure 1.3.

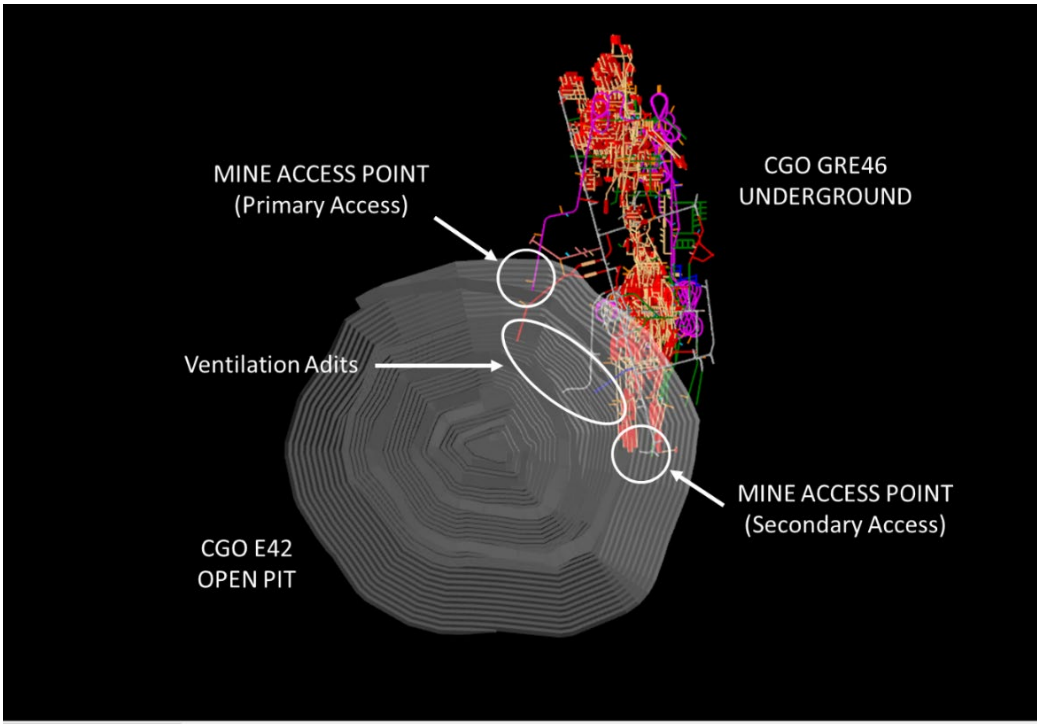


Figure 1.3 Proposed changes to the underground project

1.3.2 Underground access

The proposed modification seeks to change the access points to the underground mine as summarised in Table 1.1.

Table 1.1 Summary of changes to underground mine access points

Access point	Purpose as approved (SSD 10367)	Proposed changes
Main Portal (primary access)	The main service entry for the underground mine for personnel and vehicles	Replaced by a new primary access portal in the north of the E42 pit which will provide worker access, ventilation, maintenance access, ore haulage and waste haulage.
Box-cut	Provides personnel and material access to the mine and provide access for maintenance light vehicles	
Fresh Air Intake/Haulage Decline Portal	Provides a fresh air connection for lower working areas, an emergency egress route from underground workings and an alternate haulage route	
Fresh Air Intake Adit 1	Provides a fresh air ventilation for the lower stope working areas	No change – precise locations to be determined during detailed design
Fresh Air Intake Adit 2	Provides a fresh air ventilation for the material transfer points and for atmospheric dust control.	

Table 1.1 Summary of changes to underground mine access points

Access point	Purpose as approved (SSD 10367)	Proposed changes
Exhaust Adit	Provides exhaust air connection for material transfer points and for atmospheric (dust and air quality) control	
Warraga Decline Portal (secondary access)	Access to the exploration decline and provision of services and ventilation	Proposed for worker access, maintenance access, ore haulage and waste haulage. This will allow separation of vehicles that are transporting ore in the north and south of the mine respectively.

1.3.3 Change to access tunnel geometry

The proposed modification seeks to modify the location of access tunnels between the development declines and the stoping areas. This will require the relocation of the access tunnels to the east and further below Lake Cowal in comparison to the approved layout.

This change will allow for greater safety due to improved stability of the mine workings during production activities and greater efficiency in orebody extraction, ultimately reducing the amount of development required.

1.3.4 Ore production rate

The approved ore production rate is 1.8 Mtpa. The proposed modification seeks to increase this rate to 2.6 Mtpa. Despite this increase in the annual production rate, there would be no change to the total resource that will be extracted for the underground project (ie 27 Mt).

2 Approved SSD 10367 and proposed Mod 1 summary

The proposed changes to the underground project (Mod 1) are compared to the approved activities (SSD 10367) in Table 2.1.

Table 2.1 Mod 1 summary of changes

Aspect	Approved SSD 10367	Proposed Mod 1
Life of mine	To 31 December 2040	No change
Resource	Approximately 27 Mt	No change
Ore production rate	1.8 Mtpa	Up to 2.6 Mtpa
Waste rock production	5.74 Mt	No change
Gold production	1.8 Moz	No change
Mining method	Production of ore via mechanised long hole open stoping	No change

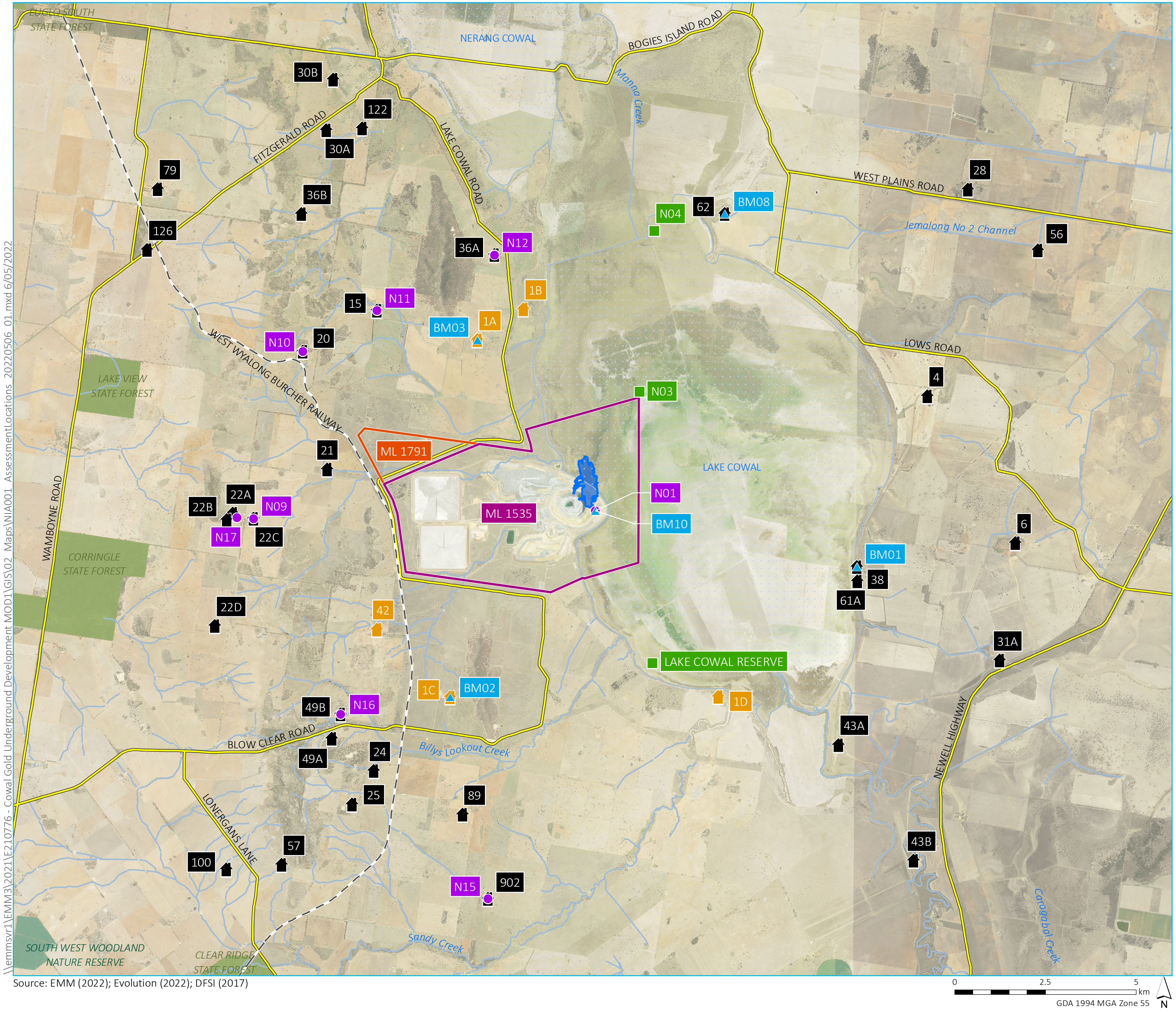
Table 2.1 Mod 1 summary of changes

Aspect	Approved SSD 10367	Proposed Mod 1
Box-cut	Development of a box-cut entry adjacent to the open-cut pit, which will be the main access for personnel and materials to the underground mine and will be used to transport ore to the surface for processing.	Removal of the box-cut from the underground project and its replacement with the primary access portal in the north of the E42 pit.
Decline	Excavation of two declines (in addition to the existing Warraga Decline) to provide underground access and ventilation: one decline via a portal on the existing open pit and the other via a box-cut. The declines will be approximately 6 m wide by 6m high and will extend approximately 1.5 km to the point at which the first production drive commences.	Excavation of one decline (in addition to the existing Warraga Decline) via a portal in the north of E42 pit that will extend approximately 2 km to the point at which the first production drive commences.
Declines access	Six access points to the main decline for access, ore haulage, ventilation circuit, underground services and emergency egress.	Change to the locations of access points to the declines. Removal of Fresh Air Intake/Haulage Decline portal and ventilation drive. Use of the Warraga Decline portal for access and ore and waste rock haulage.
Mining extent	Development of the underground mine will be in stages, as main decline is progressively extended at depth. The underground footprint is estimated to be approximately 135 ha and final depth of approximately -850 m AHD.	No change
Paste backfill	Development of a paste fill plant, and backfilling excavated stopes with cemented paste fill made from cement and tailings.	No change
Workforce	Construction: estimated peak workforce of approximately 225 FTE employees and contractors, which will be used to develop the underground project and the supporting surface infrastructure. Operations: an average of around 160 FTE employees working over two shifts.	No change

3 Original noise and vibration assessment summary

The original noise and vibration impact assessment (NVIA) for the approved underground project (SSD 10367) was completed by EMM in August 2020. The 2020 NVIA assessed the underground project and some minor changes to surface operations (DA 14/98 Mod 16) required to facilitate the underground project. The underground project cannot proceed without the changes to surface operations and hence noise impacts were assessed cumulatively. For this reason, the noise predictions for the underground project presented in the 2020 NVIA included noise emissions associated with the approved CGO open cut operations (DA 14/98). The blasting assessment did not include an assessment with the approved CGO open cut operations (DA 14/98) and was undertaken for the underground project only.

The findings from the 2020 NVIA are summarised in Section 3.1 for noise and Section 3.2 for blasting. For illustration purposes, the assessment locations are shown in Figure 3.1.



Source: EMM (2022); Evolution (2022); DFSI (2017)

\\emmsvr1\EMM3\2021\E210776 - Cowal Gold Underground Development MOD1\GIS\02_Maps\NIA001_AssessmentLocations_20220506_01.mxd 6/05/2022

KEY
- Proposed underground development
- Mining lease (ML1535)
- Mining lease (ML1791)
- Rail line
- Main road
- Watercourse/drainage line
- Waterbody
- NPWS reserve
- State forest

Monitoring locations
- Noise monitoring
- Blast monitoring

Assessment locations
- Residential (Evolution-owned)
- Residential (privately-owned)
- Non-residential

Monitoring and assessment locations

Cowal Gold Operations
Underground Development Modification 1
Noise and vibration impact assessment
Figure 3.1

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3.1 2020 NVIA summary for noise

A noise assessment was completed for the approved underground project (SSD 10367) and findings were presented in the NVIA (EMM 2020).

Noise from construction and operational activities were modelled at all assessment locations during noise-enhancing meteorological conditions. Modelled CGO operational activities included approved surface operations (DA 14/98), the underground project (SSD 10367) and the additional surface operations (DA 14/98 Mod 16) required to facilitate the underground project. Modelled construction activities included the box-cut construction.

Findings of the noise assessment were as follows:

- Noise levels during the box-cut construction were assessed for the day, evening and night periods during noise-enhancing meteorological conditions. CGO noise levels during the box-cut construction (including noise from surface operations) were predicted to satisfy the relevant limits at all assessment locations.
- Operational noise levels were assessed for the day, evening and night periods during noise-enhancing meteorological conditions. CGO operational noise levels (surface and underground operations combined) were predicted to satisfy the relevant limits at all assessment locations.
- Night-time maximum $L_{Aeq,15min}$ and L_{Amax} noise levels were predicted to satisfy the relevant sleep disturbance screening criteria at all residential assessment locations.
- Road traffic noise at nearest residential facades was predicted to satisfy relevant criteria during both the day and night periods. Hence, noise impacts from road traffic noise associated with the underground project was shown to be unlikely.

3.2 2020 NVIA summary for blasting

A blasting assessment was completed for the approved underground project (SSD 10367) and findings were presented in the NVIA (EMM 2020).

There were no significant restrictions to the maximum instantaneous charge (MIC) for blasts proposed to occur during the early stages of the underground access decline development during the day and evening periods Monday to Saturday.

For Sundays and public holidays and the night period Monday to Saturday, a 520 kg MIC limit was recommended to achieve the relevant 95% airblast overpressure and ground vibration limits at the nearest residential receiver during the early stages of the underground access decline development.

During the operational stage of the underground project, no strict control of MIC values was needed to achieve the relevant 95% ground vibration limits at the nearest residential receivers.

4 Assessment for the proposed modification

The underground project as proposed under Mod 1 will be substantially the same as the project for which consent (SSD 10367) was originally granted in 2021. Therefore, the potential for overall mine noise levels to increase as a result of the proposed modification is relatively low. To assess the potential change to the noise and vibration impacts, an assessment (this assessment) was undertaken in accordance with the NSW Environmental Protection Authority (EPA) Noise Policy for Industry (NPfI) and Australian and New Zealand Environment Council (ANZEC) blast guidelines.

This assessment will:

- assess the potential change in overall operational noise levels between approved and proposed operations; and
- assess the potential change in airblast overpressure and ground vibration levels.

4.1 Proposed changes and potential noise and vibration impacts

A summary of what the proposed changes mean in terms of potential noise and vibration impacts (if any) are presented in Table 4.1.

Table 4.1 Proposed changes and potential noise and vibration impacts <Title text>

Proposed Mod 1 changes	Assessed activity	Potential impacts
Production rate increase (1.8 to 2.6 Mtpa)	Hauling movements to the surface	The increase in annual production rate would result in an increase in underground truck movements to the surface. Any increase would however be temporary and limited in the context of overall life of mine haulage. The potential impact from this change is discussed further in Section 4.2.
	Unloading (waste) at dump areas	The modelled source locations for this activity do not change.
	Unloading (ore) at the processing area	The modelled source locations for this activity do not change.
Box-cut removal	Construction of the box-cut	The construction of the approved box-cut is no longer required, and noise emissions associated with this activity would no longer occur.
Declines and underground access	Construction of the box cut	The construction of the new primary access portal to the north of the E42 pit will replace the approved box-cut. Noise emissions during the construction of the approved box-cut were predicted to satisfy the relevant limits at all assessment locations in the 2020 NVIA. Noise emissions form the construction of the new primary access portal are expected to be much lower than those predicted for the approved box cut construction. Therefore, this change is inconsequential to the proposed modification.
	Hauling (waste) from new underground mine access points to dump areas	Waste material would be hauled from the new mine access points located within the E42 pit. The haul distance from the new mine access points would be less than or similar to that from the approved box-cut, resulting in similar or potentially minor reduction in noise emissions from underground truck movements on the surface. Furthermore, the majority of this haul route would be confined to greater depth (lower elevation within the E42 pit), resulting in similar or potentially minor reduction in noise emissions from underground truck movements on the surface. However, this would be offset by the increase in production rate (1.8 to 2.6 Mtpa), which as noted above, would result in an increase in underground truck movements to the surface. The potential impact from this change is discussed further in Section 4.2.

Table 4.1 Proposed changes and potential noise and vibration impacts <Title text>

Proposed Mod 1 changes	Assessed activity	Potential impacts
Declines and underground access	Hauling (ore) from new underground mine access points to processing area	Ore material would be hauled from the new mine access points located within the E42 pit. The haul distance from the new mine access points would be greater than from the approved box-cut, resulting in similar or potentially minor increase in noise emissions from underground truck movements on the surface. However, parts of this haul route would be confined to greater depth (lower elevation within the E42 pit), negating any potential increase in noise emissions from the underground truck movements on the surface. Any potential decrease in noise emissions from the above changes would be offset by the increase in production rate (1.8 to 2.6 Mtpa), which as noted above, would result in an increase in underground truck movements to the surface. The potential impact from this change is discussed further in Section 4.2
Change to access tunnel geometry	Underground blasting	The relocation of the access tunnels between the development declines and the stoping areas would not materially change the distance between blast locations and receivers. Hence, there is no change to the MIC restrictions for blasts recommended in the 2020 NVIA (where relevant) to achieve the airblast overpressure and ground vibration limits at the nearest residential receivers
Transport of material offsite	Offsite transport vehicle movements	No increase in overall transport movements is proposed and hence road traffic noise at nearest residential facades is anticipated to be consistent with findings in the 2020 NVIA, ie satisfy relevant criteria during both the day and night periods.

4.2 Production rate increase and potential noise impacts

The increase in production rate from 1.8 to 2.6 Mtpa is likely to result in an increase in underground truck movements. This potentially represents a 44% increase in material hauled from the underground workings to the surface. When assessed in the context of a worst-case combined scenario, that is inclusive of approved CGO open cut operations (DA 14/98) as assessed in the 2020 NVIA, the increase from 1.8 to 2.6 Mtpa represents a 1% increase in total material hauled (ore and waste) across the CGO site. A 1% increase in total material handled would represent an insignificant change to the modelled hauling activity. Therefore, the proposed modification is unlikely to result in a change to the predicted operational noise levels presented in the 2020 NVIA.

For completeness, the combined operational noise levels for the proposed modification (Mod 1) and the approved CGO open cut operations (DA 14/98) were predicted for the day, evening and night periods. A worst-case scenario including a maximum of 14 underground haul trucks operating at the same time was adopted in the noise model. All other noise modelling assumptions (ie assessment locations, modelled meteorological conditions and sound power levels) are consistent with those adopted in the 2020 NVIA.

The predicted operational noise levels were compared to the relevant CGO noise limits (DA 14/98) as presented in Table 4.2. The noise predictions remain consistent with those presented in the 2020 NVIA.

Table 4.2 Predicted noise levels for CGO approved (DA 14/98) and Mod 1 operations combined

Assessment location	Predicted operational $L_{Aeq,15min}$ noise levels[1], dB			Existing limits (DA 14/98) $L_{Aeq,15min}$, dB			Exceedance of the existing limits (DA 14/98), dB		
	Day	Evening	Night	Day	Evening	Night	Day	Evening	Night
4	<35	<35	<35	35	35	35	Nil	Nil	Nil
6	<35	<35	<35	35	35	35	Nil	Nil	Nil
15[2]	<35	<35	35	N/A	N/A	N/A	N/A	N/A	N/A
20	<35	35	35	35	35	35	Nil	Nil	Nil
21[3]	<35	44	44	N/A	N/A	N/A	N/A	N/A	N/A
22a	<36	<36	36	36	36	36	Nil	Nil	Nil
22b	<35	35	35	35	35	35	Nil	Nil	Nil
22c[4]	<38	38	38	38	38	38	Nil	Nil	Nil
22d	<35	<35	<35	35	35	35	Nil	Nil	Nil
24	<35	<35	<35	35	35	35	Nil	Nil	Nil
25	<35	<35	<35	35	35	35	Nil	Nil	Nil
28	<35	<35	<35	35	35	35	Nil	Nil	Nil
30a	<35	<35	<35	35	35	35	Nil	Nil	Nil
30b	<35	<35	<35	35	35	35	Nil	Nil	Nil
31a	<35	<35	<35	35	35	35	Nil	Nil	Nil
36a	<37	<37	<37	37	37	37	Nil	Nil	Nil
36b	<35	<35	<35	35	35	35	Nil	Nil	Nil
38	<35	<35	<35	35	35	35	Nil	Nil	Nil
42[5]	<35	46	46	N/A	N/A	N/A	N/A	N/A	N/A
43a	<35	<35	<35	35	35	35	Nil	Nil	Nil
43b	<35	<35	<35	35	35	35	Nil	Nil	Nil
49a	<35	<35	<35	35	35	35	Nil	Nil	Nil
49b	<36	<36	36	36	36	36	Nil	Nil	Nil
56	<35	<35	<35	35	35	35	Nil	Nil	Nil
57	<35	<35	<35	35	35	35	Nil	Nil	Nil
61a	<35	<35	<35	35	35	35	Nil	Nil	Nil
62	<35	<35	<35	35	35	35	Nil	Nil	Nil
79	<35	<35	<35	35	35	35	Nil	Nil	Nil
89	<35	<35	<35	35	35	35	Nil	Nil	Nil
90	<35	<35	<35	35	35	35	Nil	Nil	Nil

Table 4.2 Predicted noise levels for CGO approved (DA 14/98) and Mod 1 operations combined

Assessment location	Predicted operational $L_{Aeq,15min}$ noise levels[1], dB			Existing limits (DA 14/98) $L_{Aeq,15min}$, dB			Exceedance of the existing limits (DA 14/98), dB		
	Day	Evening	Night	Day	Evening	Night	Day	Evening	Night
100	<35	<35	<35	35	35	35	Nil	Nil	Nil
122	<35	<35	<35	35	35	35	Nil	Nil	Nil
126	<35	<35	<35	35	35	35	Nil	Nil	Nil
LCR	<53	<53	<53	N/A	N/A	N/A	N/A	N/A	N/A
NO3	<40	45	45	N/A	N/A	N/A	N/A	N/A	N/A
N04	<40	<35	<35	N/A	N/A	N/A	N/A	N/A	N/A

Notes:
1. Combined noise levels for CGO approved (DA 14/98) and Mod 1 operations.
2. Evolution has a noise agreement in place with the landowner of this privately-owned property.
3. Subject to acquisition upon request in accordance with the development consent.
4. Subject to mitigation upon request in accordance with the development consent.
5. Owned by Evolution.
6. Day: 7:00 am to 6:00 pm Monday to Saturday; 8:00 am to 6:00 pm Sundays and public holidays; evening: 6:00 pm to 10:00 pm; night: remaining periods.
7. N/A = not applicable.

5 Management and mitigation

5.1 Noise

The noise management and mitigation measures currently implemented at CGO are described in the Noise Management Plan (NMP). It is noted that the NMP was recently updated (approved in March 2022) following the approval of the most recent CGO open cut operations modification (DA 14/98 Mod 16) and the approval of the underground project (SSD 10367).

The NMP describes the noise monitoring program, protocols for identification and notification of noise incidents, existing implementation of noise mitigation measures, noise complaints management system, community consultation and independent environmental audit processes in place at CGO.

Current noise management and mitigation measures will continue to be implemented at CGO in accordance with DA 14/98, SSD 10367 and the NMP to ensure that potential noise impacts from the CGO open cut operations and underground development project are minimised. No additional noise management and mitigation measures are required for the proposed modification.

5.2 Blasting

The blast management and mitigation measures currently implemented at CGO are described in the Blast Management Plan (BMP).

The BMP describes the blast design and controls, blast management and mitigation measures, blast safety and infrastructure protection measures, blast monitoring program, protocols for identification and notification of blast incidents, blast complaints management system, community consultation and independent environmental audit processes in place at CGO.

Current blast management and mitigation measures will continue to be implemented at CGO in accordance with DA 14/98, SSD 10367 and the BMP to ensure that potential blast impacts from the CGO open cut operations and underground development project are minimised. No additional blast management and mitigation measures are required for the proposed modification.

6 Conclusion

An assessment of potential noise and vibration impacts from the proposed modification (Mod 1) for the underground project was completed. The assessment showed that the proposed changes to the CGO underground development project will not result in any additional noise and vibration (blasting) impacts and would effectively be the same as those presented in the NVIA (EMM 2020) for the approved (SSD 10367) underground project.

Noise and blast management and mitigation measures currently in place at CGO will continue to be implemented in accordance with relevant consent conditions and management plans to ensure that potential noise impacts from the CGO open cut operations and underground development project are minimised.

Yours sincerely

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Reviewed by Najah Ishac on 6 May 2022



Appendix B

Air quality impact assessment

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18 July 2022

Bonnie Coxon
Manager Integrated Planning
Cowal Gold Operations

Re: CGO Underground Development Project Optimisation Mod 1 - Air Quality Review

Dear Bonnie

The following report provides a review of the air quality impacts associated with the Underground Project Optimisation Modification (Mod 1).

1 Introduction

The Cowal Gold Operations (CGO) Underground Development Project was approved in September 2021. CGO's Integrated Planning team has completed an underground mine optimisation study which reviewed design options to assess operational execution risks and efficiencies for the Underground Development Project. This process has led to a change in design which requires a modification application, referred to as the Underground Project Optimisation Modification.

The proposed modification seeks to:

- change the way in which the underground workings are accessed from the surface;
- change the geometry of the access tunnels to the underground stopes; and
- increase the annual production rate from 1.8 million tonnes per annum (Mtpa) to 2.6 Mtpa.

No changes are proposed to the maximum amount of ore that would be extracted over the life of the project, the approved mining method, stoping areas or operation of the pastefill plant.

EMM Consulting Pty Limited (EMM) has been commissioned to undertake a review of air quality impacts associated with the modification.

2 Air quality assessment

An air quality and greenhouse gas assessment (AQGHGA) was prepared to support the CGO Underground Development Project (EMM, 2021), assessing the underground development and associated surface changes. The identified dust emission sources in the AQGHGA are summarised in Table 2.1.

The proposed modification would result in a change to dust emissions (from the increase in annual production rate) and a change in the dust emission source locations (due to the new mine access points changing how material is hauled from underground to surface infrastructure).

A summary of what the modification changes, in terms of the identified dust emission sources, is summarised in Table 2.1.

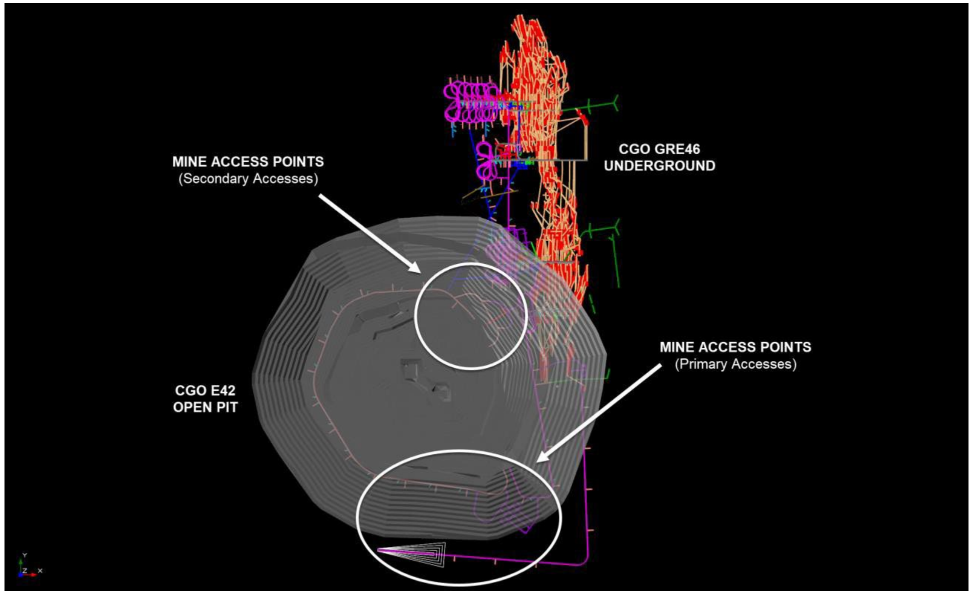
Table 2.1 Dust emission sources assessed in AQGHGA and proposed change due to modification

Project component	AQGHGA dust emission source	Modification change
Surface changes	Development (construction) of the box cut for mine access	Development of the box cut is no longer required, and dust emissions associated with material handling from this activity would no longer occur. It is noted that the development of the box cut was not part of the modelled scenario in the AQGHGA, therefore there would be no change to assess for the modification.
	Waste – hauling from box cut portal to northern waste dump	Waste material would be hauled from the new mine access points. The haul distance from the new mine access point would be less than from the box cut, resulting in a reduction in wheel generated dust per trip. However, this would be offset by an increase in the tonnes per annum (tpa) hauled, which would increase the number of haul trips. The changed haul route would also change the modelled source locations for this activity.
	Waste – unloading at northern waste dump	Increase in the tpa of waste unloaded, resulting in an increase in dust emissions. The modelled source locations for this activity do not change.
	Ore – hauling from box cut to temporary stockpile	Ore material would be hauled from the new mine access point. There would be a minor increase in the haul distance from the new mine access point to the run-of-mine (ROM) pad. The changed haul route would also change the modelled source locations for this activity.
	Ore – unloading ore to temporary stockpile	Increase in the tpa of ore unloaded, resulting in an increase in dust emissions, however it will not increase the total amount of ore handled on site for the life of the project. It is noted that the revised production schedule may result in the underground mine ceasing operations approximately three years earlier than the schedule that was considered in the AQGHGA. This has the potential for lower cumulative impacts later in the project life as the open-pit operations are reduced. The modelled source locations for this activity do not change.
	Ore – rehandle at crusher/ROM pad	
	Ore – crushing	
	Ore – screening	
	Ore – loading to coarse ore stockpile	

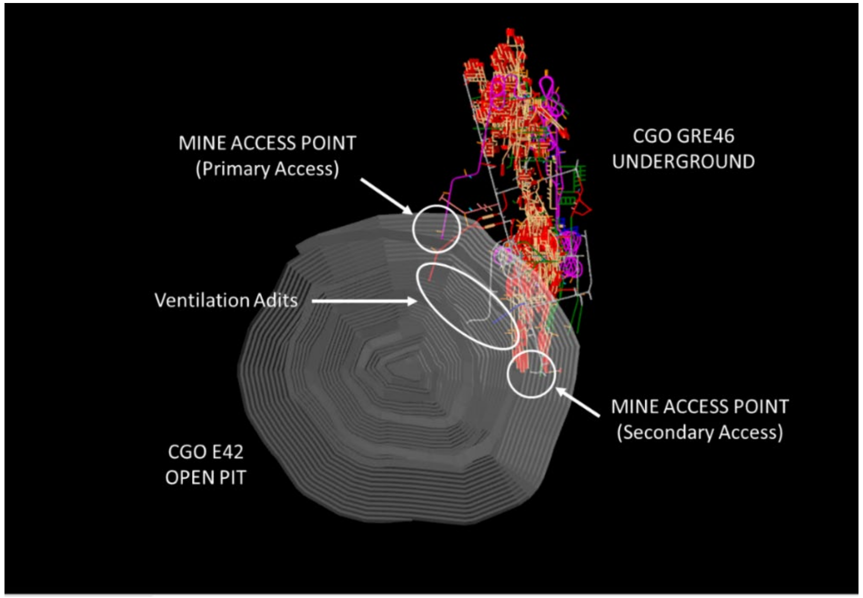
Table 2.1 Dust emission sources assessed in AQGHGA and proposed change due to modification

Project component	AQGHGA dust emission source	Modification change
Underground workings	Blasting for UG development	There would be an increase in the amount of dust generated underground, from the increase in the tpa removed.
	Mining of material (underground)	
	Waste and ore – trucking to surface	Dust emissions generated underground would be released from the exhaust adit, the location of which does not materially change from the approved location.

A comparison of the approved underground workings and access points with the proposed modification is shown in Figure 2.1. The precise locations for the ventilation adits are to be determined during detailed design; however the exhaust adit in the AQGHGA was modelled at the location shown by the mine secondary access points in Figure2.1 which is consistent with the locations for the ventilation adits for the proposed modification (ie no significant change from the AQGHGA).



Approved underground workings and access points



Proposed modification

Figure 2.1 Comparison between the approved UG workings and access with the proposed modification

2.1 Assessment of change

The AQGHGA (EMM, 2021) assessed both the Underground Development Project and the Mod 16 changes to surface operations that were required for the Underground Development. Impacts were assessed concurrently as the project components were linked (the underground development could not proceed without the surface changes modification) and therefore could not be separated for an assessment of cumulative impacts. The cumulative assessment in the AQGHGA also included emissions associated with the approved open cut operations.

Therefore, our assessment of change from the modification needs to be considered in the context of operations across the CGO site. Although there is no increase in total ore production over the life of the mine, the increase in annual ore production rate would result in more material being handled, which would have an associated increase in dust emissions. Although the increase in the maximum annual ore production rate from 1.8 Mtpa to 2.6 Mtpa represents a 44% increase, as noted above, the modelling presented in the AQGHGA also includes total material handling (ore and waste) associated with the approved open cut operations, for a scenario which considered a worst-case combined throughput.

When assessed in the context of a worst-case combined scenario (modelled as 2022 in the AQGHGA), the increase from 1.8 Mtpa to 2.6 Mtpa represents just a 1% increase in total material handled (ore and waste) across the entire CGO site.

The modelled scenario presented in the AQGHGA is still a worst-case combined throughput scenario for the modification. It is noted that the 1% increase in total material handled would represent less than a 1% increase in emissions. Therefore, the modification would result in an insignificant change to the predicted ground level concentrations presented in the AQGHGA.

3 Greenhouse gas emissions

Estimates of greenhouse gas (GHG) emissions for the Underground Development Project were presented in the AQGHGA. The increase in the annual ore production rate for the modification would result in additional diesel combustion and, to a lesser extent, an increase in electricity use, relative to what was assessed in the AQGHGA. Revised estimates of GHG emissions for the modification are presented in Table 3.1, expressed as an annual average.

The revised estimates are made by scaling the AQGHGA emissions by the relative increase in annual ore production for the modification. This approach assumes that the increase in diesel and electricity would result in a linear increase in GHG emissions, which is a conservative assumption, particularly for electricity emissions. The GHG emission estimates presented in the AQGHGA also included the first year of box cut development, which was the highest estimated year for diesel consumption. To provide a like for like comparison between the estimates presented in AQGHGA and the modification, the AQGHGA estimate is revised to exclude the first year of box cut development.

Table 3.1 Revised average annual GHG emission estimates for modification (t CO_2-e/year)

GHG emission scope	AQGHGA estimate	Modification estimate	FY2019 NGERs data
Scope 1 - Diesel	13,275	13,152	70,741
Scope 1 - Explosives	201	236	
Scope 2 - Electricity	42,134	39,814	202,168
Scope 3 - Diesel	681	674	NA
Scope 3 - Electricity	4,682	4,424	

As shown in Table 3.1, the annual average GHG emissions for the modification are effectively the same (slightly lower) as those presented in the AQGHGA (when the first year of box cut development is excluded).

This is expected as, although the annual ore production rate increases, the total ore production over the life of the project does not increase, therefore the average across all years remains the same. When compared against CGO reported NGERs data for existing open cut operations in FY2019, the proposed modification does not change what was reported in the AQGHGA.

4 Recommended mitigation

The existing air quality and greenhouse gas mitigation measures for CGO are described in the Air Quality Management Plan (AQMP) and are consistent with best practice[1]. Measures most relevant to the Underground Development Project are shown in Table 4.1. The AQMP also describes the air quality monitoring network, consisting of a meteorological monitoring station, 12 dust deposition gauges, High Volume Air Sampler (HVAS) and continuous monitoring for PM_{10} at three locations.

No additions to the mitigation measures or air quality monitoring network are required for the proposed modification.

Table 4.1 Air quality management measures

Source	Management measure
Haul road	• All roads and trafficked areas are watered or treated with dust suppressant. • Routes to be clearly marked. • Obsolete roads will be ripped and re-vegetated.
Minor roads	• Minor road development will be limited, and the locations will be defined and within approved surface disturbance areas. • Obsolete roads will be ripped and re-vegetated.
Materials handling	• Water sprays on crusher bin. • Prevention of truck overloading to reduce spillage during ore loading/unloading and hauling. • Freefall height during ore/waste stockpiling will be limited.
Drilling	• Dust aprons will be lowered during drilling for collection of fine dust.

[1] Reference is made to Katestone (2011) for consideration of best practice controls for extractive industries.

Table 4.1 Air quality management measures

Source	Management measure
Blasting	• Fine material collected during drilling will not be used for blast stemming. • Adequate stemming will be used at all times. • Blasting will only occur following an assessment of weather conditions by the Environmental Manager to ensure that wind speed and direction will not result in excess dust emissions from the site towards adjacent residences (see the blasting Management Plan for further details).
Equipment maintenance	• Emissions from mobile equipment exhausts will be minimised by the implementation of a maintenance programme to service equipment in accordance with the equipment manufacturer specifications.
General exposed areas	• Increased watering of exposed surfaces via water trucks or other methods as required.
Gold room doré melt furnace	• Use of a baghouse and associated collection hood/ducting to remove dust particles.

5 Conclusion

A review of the Underground Project Optimisation Modification indicates that the air quality and GHG predictions would be effectively the same as those presented in the AQGHGA for the approved project. The proposed modification would therefore not result in any additional exceedances of the impact assessment criteria. Furthermore, the removal of the box cut from the project and its replacement with a portal in the north of E42 would significantly reduce the material handling, and associated dust emissions, during the first year of development. The revised production schedule also means that underground mining could cease approximately three years earlier than the schedule that was considered in the AQGHGA, which has the potential for lower cumulative impacts later in the project life as the open-pit operations are reduced.

Yours sincerely

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Appendix C

Surface water impact assessment

EMM
creating opportunities

Evolution MINING

REPORT

EVOLUTION MINING

Cowal Gold Operations Underground Development Project Modification 1 Surface Water Review

121155.16, Final, Rev 2
July 2022

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EXECUTIVE SUMMARY

In September 2021, the CGO Underground Development Project (UDP), State Significant Development (SSD) 10367, was approved under Section 4.38 of the NSW *Environmental Planning and Assessment Act 1979*. Evolution Mining (Cowal) Pty Limited proposes to modify SSD 10367 under Section 4.55(2) of the *Environmental Planning and Assessment Act 1979*. The proposed Modification seeks to:

- change the way in which the underground workings are accessed from the surface;
- change the geometry of the access tunnels to the underground stopes; and
- increase the maximum annual production rate from 1.8 million tonnes per annum (Mtpa) to 2.6 Mtpa.

A Surface Water Review has been conducted to assess the potential surface water-related impacts associated with the proposed Modification. Specifically, the following has been assessed:

- potential impacts to the site water balance and water supply security associated with the proposed change in the maximum annual production rate;
- underground inrush risk associated with the proposed in-pit access; and
- potential surface water impacts, specifically relating to Lake Cowal, associated with the proposed Modification.

The results of the updated site water balance modelling indicate that the demand from external sources (the eastern saline borefield, the Bland Creek Palaeochannel borefield and licensed extraction from Lachlan River water entitlements) is predicted to average 2,524 ML/year over the Modification life. Based on the 90th percentile model results, the annual demand from the Bland Creek Palaeochannel borefield is predicted to peak at 3,171 ML in 2024, which is less than the approved annual extraction rate of 3,650 ML.

The maximum predicted annual demand from the Lachlan River is approximately 2,639 ML based on the 90th percentile model results. Based on Department of Environment and Planning - Water trading records, there has been adequate allocation assignment water available on the market from this source in previous years to meet this predicted demand requirement.

No supply shortfalls were predicted for any of the 133 water balance model climatic scenarios.

The predicted maximum pit water volume of 2,396 ML corresponds with a maximum pit water level of 806 m Mine Datum (MD) which is notably lower than the elevation of the lowest proposed access point (fresh air intake adit 1: 957 m MD). As such, there is negligible risk of underground inrush associated with the proposed in-pit access points.

The proposed in-pit access points are expected to involve substantially less surface disturbance and movement of material for construction than that of the approved box-cut and decline. Any surface disturbance would be contained within the current approved disturbance area. As such, no additional impact on inflows to Lake Cowal is expected to occur as a result of the Modification.

Overall, it is concluded that there would be a low risk of more than a negligible hydrological impact on Lake Cowal due to the proposed Modification.

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TABLES

DIAGRAMS

GRAPHS

MAPS

1 INTRODUCTION

1.1 Background

Evolution Mining (Cowal) Pty Limited (Evolution) is the owner and operator of the Cowal Gold Operations (CGO) located approximately 38 kilometres (km) north-east of West Wyalong in New South Wales (NSW) (Map 1).

Open-cut mining operations at the CGO are approved to 2040 and are carried out in accordance with Development Consent DA 14/98 (as modified). In September 2021, the CGO Underground Development Project (UDP), State Significant Development (SSD) 10367, was approved under Section 4.38 of the NSW *Environmental Planning and Assessment Act 1979*.

Evolution proposes to modify SSD 10367 under Section 4.55(2) of the *Environmental Planning and Assessment Act 1979* (herein referred to as the Modification). The proposed Modification area is shown in Map 2.

1.2 Modification Description

The proposed Modification seeks to:

- change the way in which the underground workings are accessed from the surface;
- change the geometry of the access tunnels to the underground stopes; and
- increase the maximum annual production rate from 1.8 million tonnes per annum (Mtpa) to 2.6 Mtpa.

No changes are proposed to:

- the approved mining method, stoping areas or operation of the pastefill plant;
- the approved existing open-cut mining, processing plant and integrated waste landform operations undertaken in accordance with DA 14/98; or
- the approved SSD 10367 site water management system.

The proposed Modification changes as compared to the approved UDP are summarised in Table 1 and illustrated in Map 3.

TABLE 1 MODIFICATION 1 SUMMARY

Aspect	Approved SSD 10367	Proposed SSD 10367 MOD 1
Life of mine	To 31 December 2040	No change
Resource	Approximately 27 Mt	No change
Annual production rate (maximum)	1.8 Mtpa	2.6 Mtpa
Waste rock production	5.74 Mt	No change
Gold production	1.8 Moz	No change
Mining method	Production of ore via mechanised long hole open stoping.	No change

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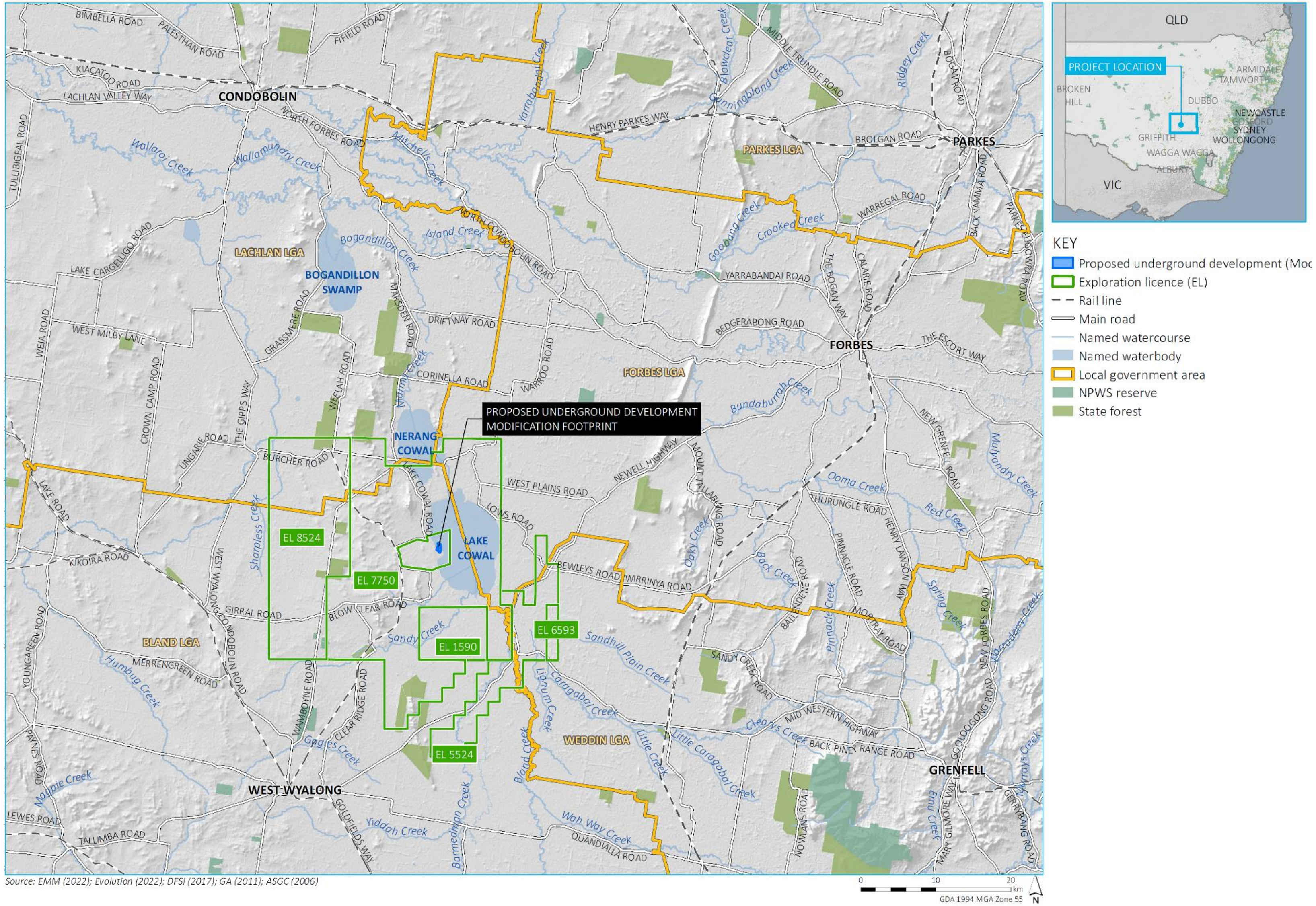
TABLE 1 (CONT.) MODIFICATION 1 SUMMARY

Aspect	Approved SSD 10367	Proposed SSD 10367 MOD 1
Declines	Excavation of two declines (in addition to the existing Warraga Decline) to provide underground access and ventilation: one decline via a portal on the existing open-cut pit and the other via a box-cut. The declines will be approximately 6 m wide by 6 m high and will extend approximately 1.5 km to the point at which the first production drive commences.	Excavation of one decline (in addition to the existing Warraga Decline) via a portal in the north of the open-cut pit that will extend approximately 2 km to the point at which the first production drive commences.
Decline access	Six access points to the main decline for access, ore haulage, ventilation circuit, underground services and emergency egress.	Change to the locations of access points to the declines. Removal of Fresh Air Intake/Haulage Decline portal and ventilation drive. Use of the Warraga Decline portal for access and ore and waste rock haulage.
Mining extent	Development of the underground mine will be in stages, as the main decline is progressively extended at depth. The underground footprint is estimated to be approximately 135 hectares (ha) and final depth of approximately - 850 m Australian Height Datum (AHD).	No change
Box cut	Development of a box-cut entry adjacent to the open-cut pit, which will be the main access for personnel and materials to the underground mine and will be used to transport ore to the surface for processing.	Removal of the box cut from the project and its replacement with a portal in the north of the open-cut pit.
Paste backfill	Development of a paste fill plant, and backfilling excavated stopes with cemented paste fill made from cement and tailings.	No change
Workforce	Construction: estimated peak workforce of approximately 225 full time equivalent (FTE) employees and contractors, which will be used to develop the underground mine project and the supporting surface infrastructure. Operations: an average of around 160 FTE employees working over 2 shifts.	No change



Source: EMM (2022); Evolution (2022); DFSI (2017); GA (2011); ASGC (2006)

MAP 1: REGIONAL LOCATION

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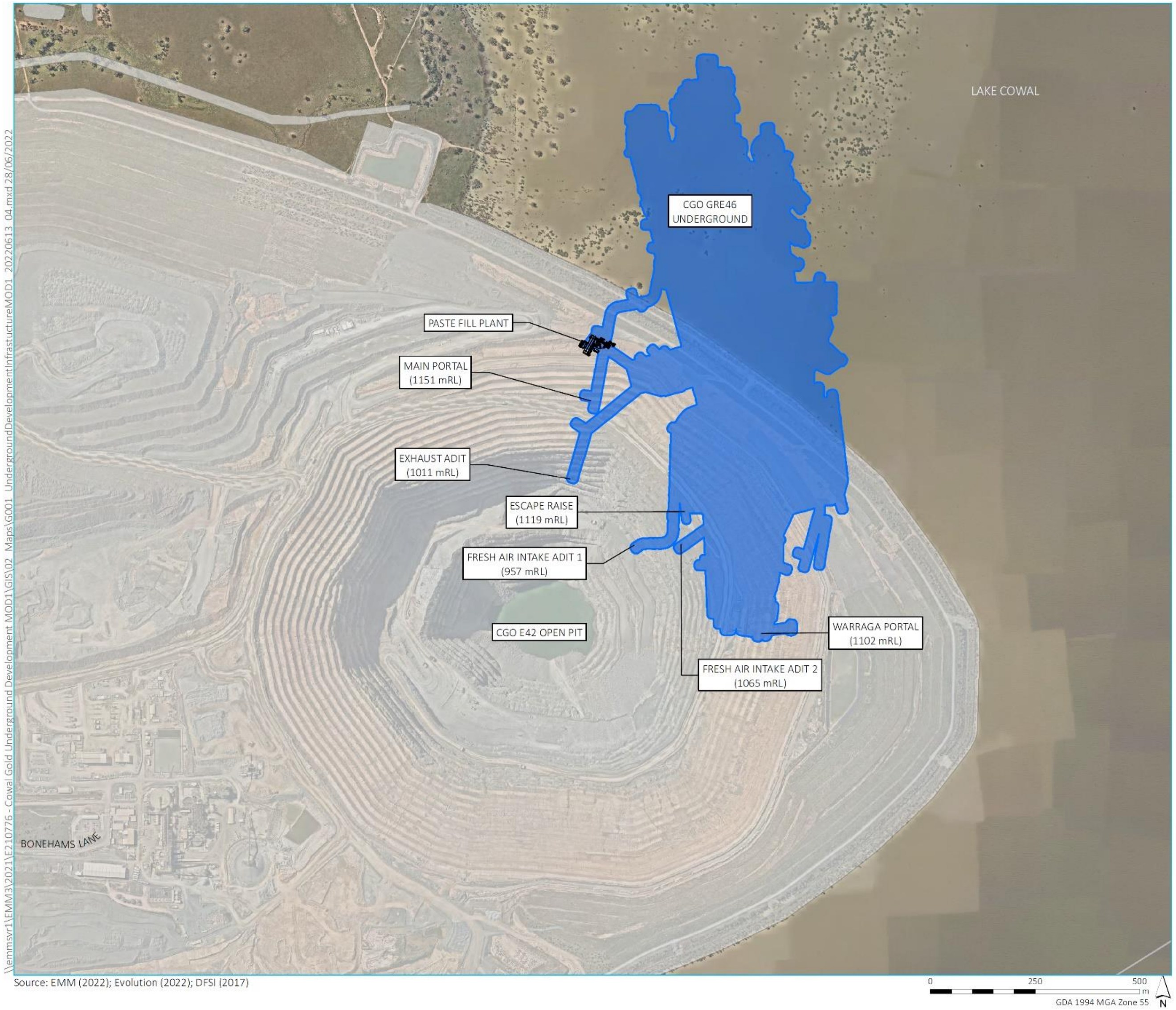
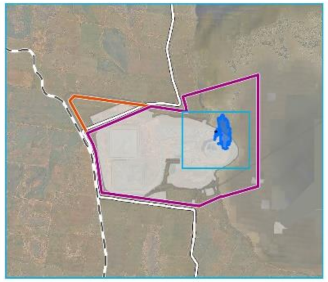
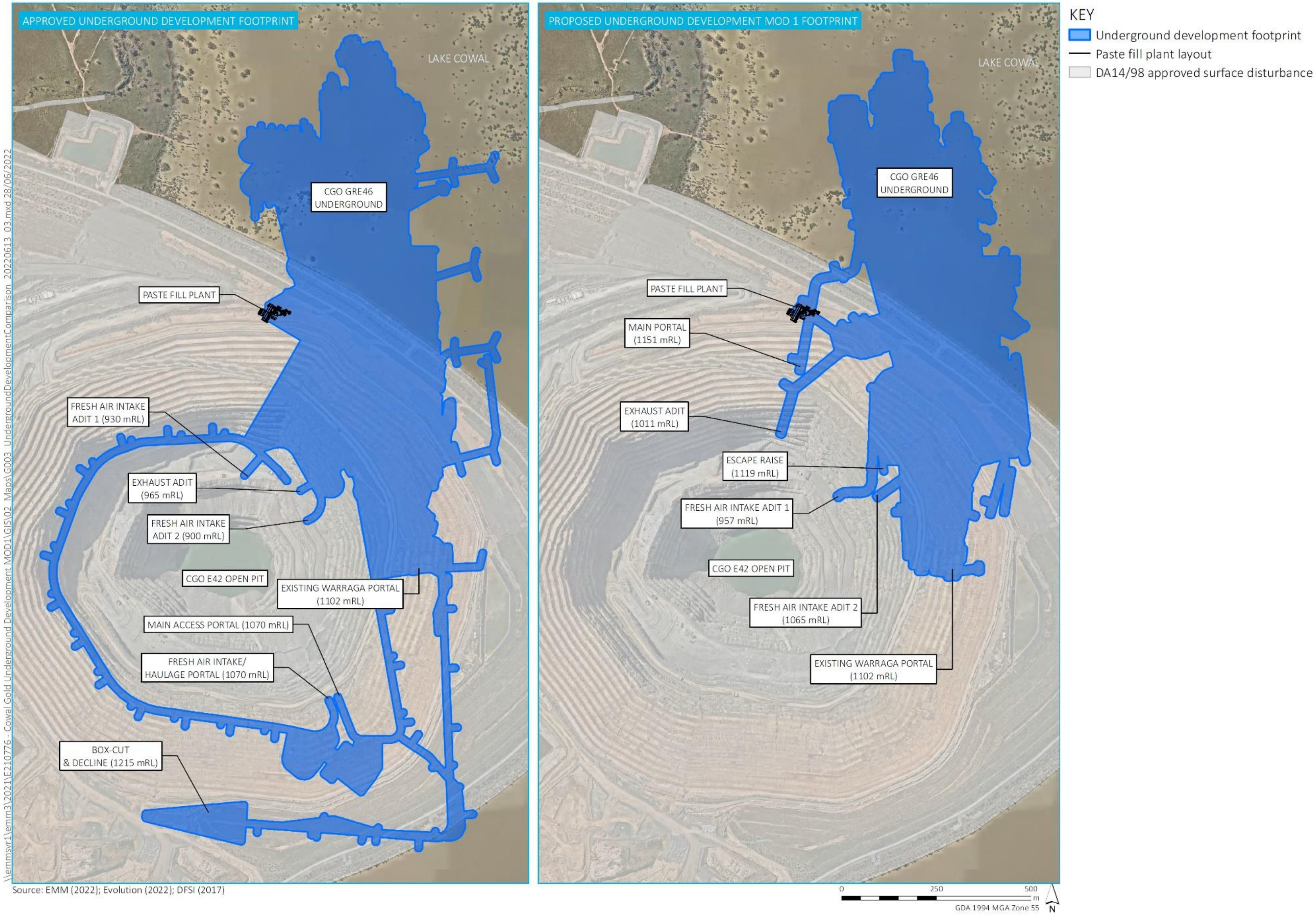




KEY
Proposed underground development (Mo
Paste fill plant layout
Mining lease (ML1535)
Mining lease (ML1791)
DA14/98 approved surface disturbance
Rail line
Main road

MAP 2: MODIFICATION AREA



Source: EMM (2022); Evolution (2022); DFSI (2017)

MAP 3: APPROVED AND PROPOSED UNDERGROUND DEVELOPMENT PROJECT

1.3 Modification Water Management

Water management for the Modification would be undertaken as described in Appendix G (HEC, 2020) of the approved *Cowal Gold Operations Underground Development Environmental Impact Statement* (EMM, 2020).

1.4 Scope of Work

This Surface Water Review report has been prepared by ATC Williams Pty Ltd (ATCW) in support of the Modification Environmental Assessment (EA). The Surface Water Review has assessed the potential surface water-related impacts associated with the proposed Modification. Specifically, the following has been assessed:

- potential impacts to the site water balance and water supply security associated with the proposed change in the maximum annual production rate;
- underground inrush risk associated with the proposed in-pit access; and
- potential surface water impacts, specifically relating to Lake Cowal, associated with the proposed Modification.

1.5 Relevant Planning Instruments

1.5.1 Water Management Act 2000

The objects of the NSW *Water Management Act 2000* which is the principal statute governing management of water resources in NSW, were considered during the assessment.

1.5.2 Water Sharing Plans

Under the *Water Management Act 2000*, the *Water Sharing Plan for the Lachlan Regulated River Water Source 2003* commenced on 1 July 2004 and was replaced on 1 July 2016. The *Water Sharing Plan for the Lachlan Regulated River Water Source 2016* covers licensed surface water accessed from the Lachlan River.

External make-up water supply at CGO is provided to the site via the mine borefield pipeline which draws water from the eastern saline borefield, the Bland Creek Palaeochannel Borefield and water extracted from the Lachlan River via the Jemalong Irrigation Channel. Water is currently extracted from the Lachlan River using regulated flow licences purchased by Evolution on the open market under the *Water Sharing Plan for the Lachlan Regulated River Water Source 2016.* Between approximately 4,000 and 274,000 megalitres (ML) of allocation assignment has been traded annually since records began in the 2004/2005 water year to the 2021/2022 water year[1].

Under the *Water Management Act 2000*, the *Water Sharing Plan for the Lachlan Unregulated and Alluvial Water Sources 2012* commenced on 14 September 2012. The *Water Sharing Plan for the Lachlan Unregulated and Alluvial Water Sources 2012* applies to all unregulated water sources in the Lachlan catchment which occurs naturally on the surface of the ground, and in rivers, lakes and wetlands.

As of March 2022, available water determinations (AWDs) for general security accounts were at 121%, with high security licences at 100% as of July 2021. The NSW Department of Planning and Environment (DPE) - Water closely monitor rainfall and river inflows as well as usage in the Lachlan Valley to

[1] https://waterregister.waternsw.com.au/ accessed 18 June 2022.

determine when subsequent changes to AWDs are made. As at 20 June 2022, Wyangala Dam reservoir was at 95.6% of capacity[2].

Within the *Water Sharing Plan for the Lachlan Unregulated and Alluvial Water Sources 2012,* CGO is located within the Western Bland Creek Water Source, which has a total surface water entitlement of 2,168 megalitres per year (ML/year) divided between 32 surface water licences[1].

2 MODIFICATION SITE WATER BALANCE

Diagram 1, which forms the basis of the site water balance model, conceptually illustrates the various CGO water management system components and their linkages (via system transfers). HEC (2020) provides a detailed description of the CGO water management system and site water balance model. The following sections summarise the site water balance model and the model inputs which have been updated to reflect the Modification.

2.1 Model Description

2.1.1 General

The water balance model developed for the CGO simulates all the inflows, outflows, transfers and changes in storage of water on-site at each model time step (i.e. 6-hourly basis). The model simulates changes in stored volumes of water in all site storages (contained water storages, TSFs, the IWL, open-cut pit and underground) in response to inflows (rainfall runoff, groundwater inflow, tailings water, groundwater bore extraction and licensed extraction from the Lachlan River) and outflows (evaporation, process plant use and dust suppression use).

For each storage, the model simulates:

> *Change in Storage = Inflow – Outflow*
>
> Where:
>
> > *Inflow* includes rainfall runoff, groundwater inflows to the open-cut pit and underground, water liberated from settling tailings ('bleed' water – for the TSFs and IWL) and all pumped inflows from other storages, groundwater bores or the Lachlan River (via the Jemalong irrigation channel).
> >
> > *Outflow* includes evaporation and all pumped outflows to other storages or to a water use[3].

The model was simulated for the period 1 July 2022 to 31 July 2035 (14 years, 1 month). The model simulates 133 "realizations" derived using the historical daily climatic record from 1892 to 2021 (refer Section 2.1.2). Realization 1 uses climatic data from 1892 to 1905, realization 2 uses data from 1893 to 1906, realization 3 uses data from 1894 to 1907 and so on. The results from all realizations are used to generate estimates of supply reliability, spill and open-cut pit water inventory. This method covers the full range of historical climatic variation – including high and low rainfall periods.

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[2] http://realtimedata.water.nsw.gov.au/water.stm accessed 20 June 2022.
[3] The model also provides for and tracks spill if the simulated storage capacity of a water storage is ever exceeded.

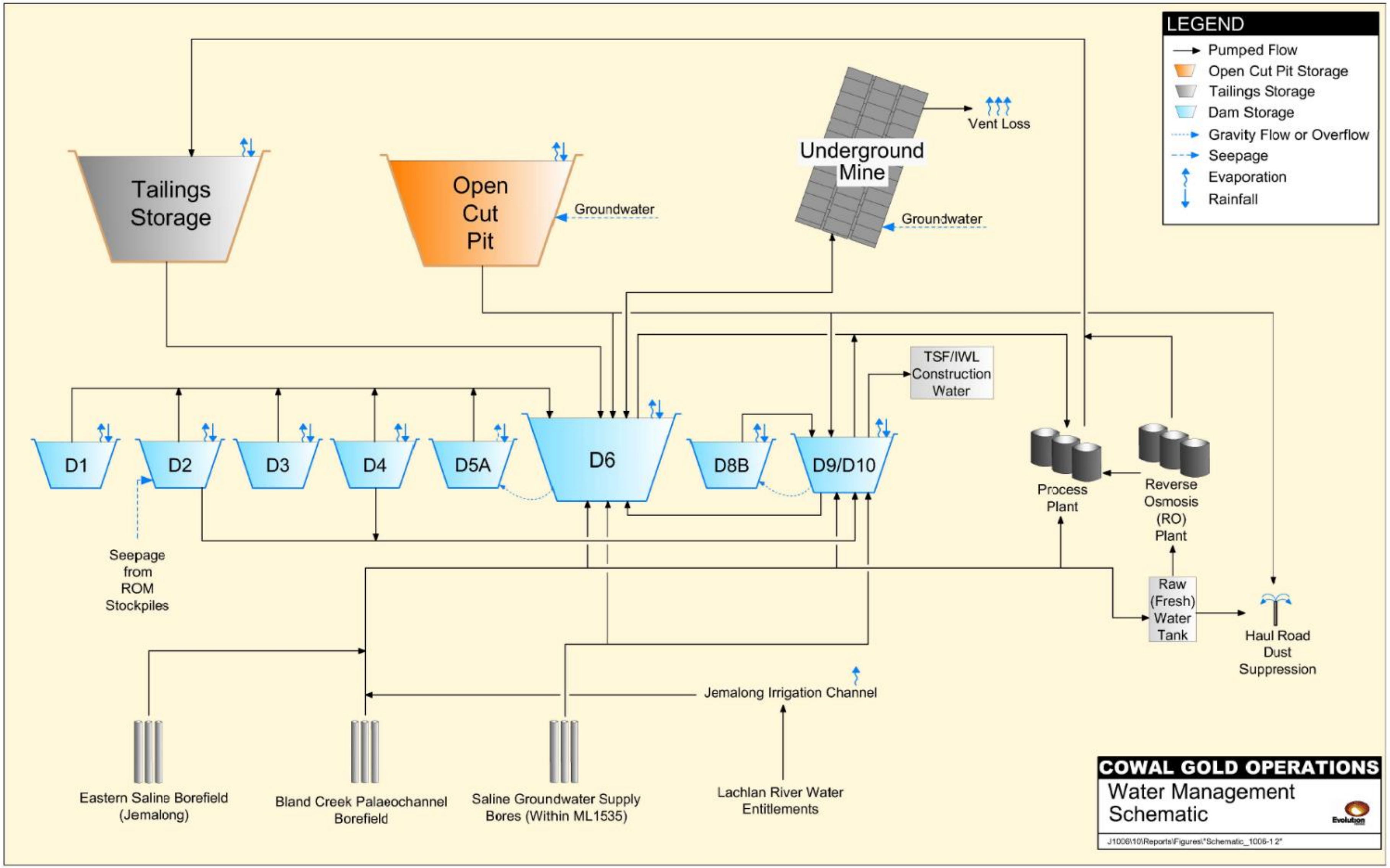

DIAGRAM 1: CGO WATER MANAGEMENT SYSTEM SCHEMATIC

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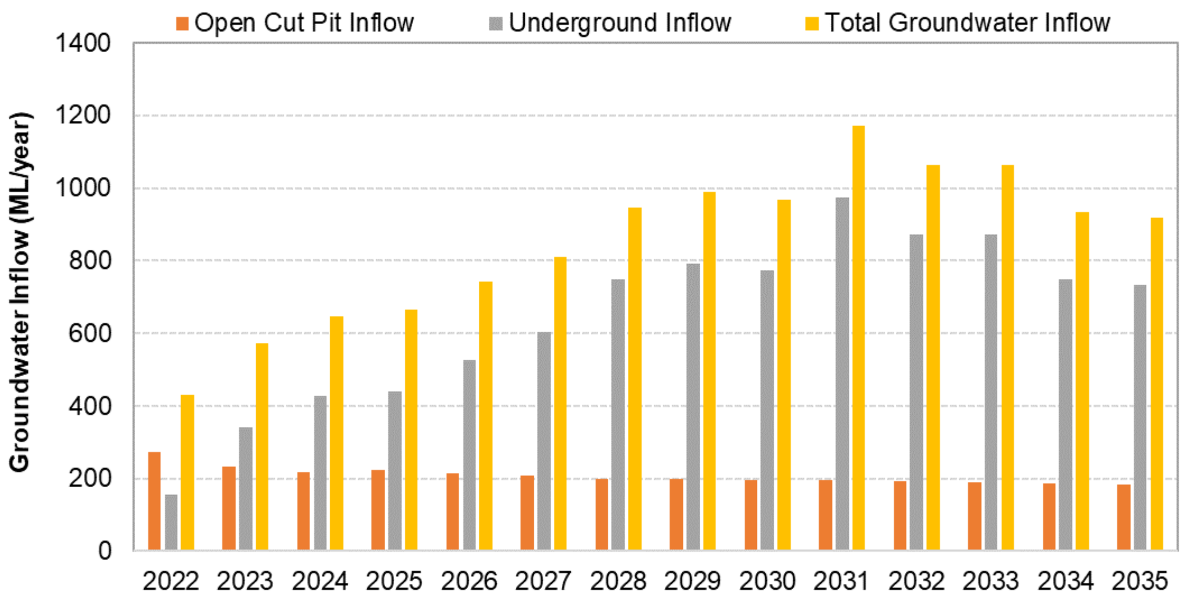
2.1.2 Climatic Data

A total of 133 years of daily rainfall and pan evaporation data (from 1889 to 2021) was sourced from the SILO Point Data[4] and input to the site water balance model. The SILO Point Data was compared with the CGO rainfall data record (for the period from 2002 to December 2021) and found to be similar in magnitude.

2.1.3 Groundwater Inflow

Groundwater inflow to the open-cut pit and underground mine were set to a time-varying rate as predicted by groundwater modelling (EMM, 2022). Graph 1 presents the predicted annual inflow volume for the open-cut pit, underground mine and combined total inflow rate.



GRAPH 1 PREDICTED MINE GROUNDWATER INFLOW RATE

It is noted that the groundwater model has been updated since the UDP EIS submission and, as such, the groundwater inflow predictions vary slightly to that presented in the UDP EIS. The total groundwater inflow was predicted to peak at 1,027 ML/year in 2034 based on the UDP EIS groundwater modelling predictions (Coffey, 2020a) while the total groundwater inflow for the Modification is predicted to peak at 1,173 ML/year in 2031 based on the revised groundwater modelling predictions (EMM, 2022).

2.1.4 Process Plant Water Demand

The process plant water demand (total) was estimated based on projected future processing tonnages, tailings paste backfill volume and assumed conventional tailings and paste backfill solids content. The total tailings tonnage, tailings paste backfill tonnage and conventional tailings tonnage to the TSFs and IWL, as provided by Evolution, were based on a maximum annual production rate of 2.6 Mtpa and total production of 27 Mt over the life of the project.

---

[4] The SILO Point Data is a system which provides synthetic data sets for a specified point by interpolation between surrounding point records held by the Bureau of Meteorology (BoM). Refer https://www.longpaddock.qld.gov.au/silo/point-data/.

For primary ore, a solids concentration of 52% was applied based on the average tailings solids concentration monitored for 2 years to December 2010[5] during a period of processing of primary ore alone (note that recent data provided by Evolution is consistent with this assumed tailings solids concentration). The tailings solids concentration during a previous period (2006/2007) of processing of oxide ore alone (i.e. no primary ore) averaged 37%.

A portion of tailings would be processed and used to produce a backfill to support the excavated stopes. The processed tailings for paste backfill was simulated with a solids content of 74.5%, as advised by Evolution.

The average process plant demand (total) at the proposed processing rates is estimated at 22 ML/day between 2022 and 2030 when both primary and oxide ore are to be processed. The maximum water demand to accommodate processing of primary and oxide ore from the proposed underground mine and open cut operations is estimated at 25 ML/d in 2024. Between 2031 and 2035 when oxide ore processing will have ceased, the average water demand (total) is estimated at 13 ML/day.

2.2 Site Water Balance Results

2.2.1 Site Water Balance Summary

Table 2 summarises the water balance model results of average system inflows and outflows for all model realizations (averaged over the Modification life).

TABLE 2 SUMMARY SITE WATER BALANCE

Water Balance Component	Average Rate (ML/year)
Inflows	
Catchment Runoff	1,335
Tailings Bleed	2,415
Open Pit and Underground Mine Groundwater	761
Saline Groundwater Supply Bores (within ML 1535)	41
Bland Creek Palaeochannel Bores	1,511
Eastern Saline Bores	421
Lachlan River Licensed Extraction*	592
Total Inflow	**7,076**
Outflows	
Evaporation	1,005
Haul Road Dust Suppression	223
Construction Water	92
Process Plant Supply	5,525
Overflow	0
Underground Mine Vent Loss	169
Total Outflow	**7,014**

ML/year = megalitres per year
* Modelled volume of water actually reaching CGO – excludes irrigation channel losses.

[5] Data provided for the Modification 11 Surface Water Assessment (Gilbert & Associates, 2013).

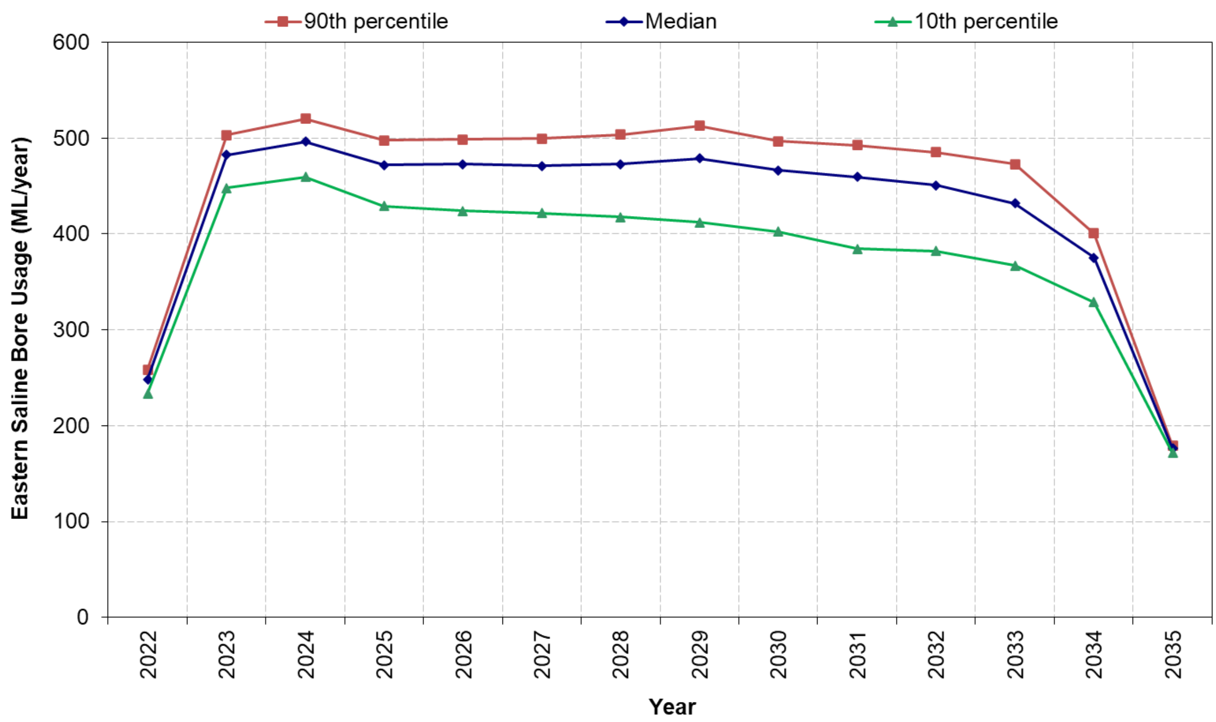
The results summarised in Table 2 show that the predicted total inflows average 7,076 ML/year while total outflows average 7,014 ML/year. Model results indicate that an average of 1,511 ML/year would be required to be sourced from the Bland Creek Palaeochannel Bores - equivalent to 4.1 ML/day. This is slightly above the long-term average of 4 ML/day predicted in the Hydrogeological Assessment (Coffey, 2020b), however, if restrictions were placed on this value, the result would be a slight increase in the simulated volume of water sourced from the Lachlan River.

The demand from external sources (the eastern saline borefield, the Bland Creek Palaeochannel borefield and licensed extraction from Lachlan River water entitlements) is predicted to average 2,524 ML/year. This compares with an average of 2,744 ML/year predicted based on the UDP EIS water balance modelling. The reduction in predicted demand from external sources relates predominately to changes in the proposed processing rates and predicted groundwater inflow rates.

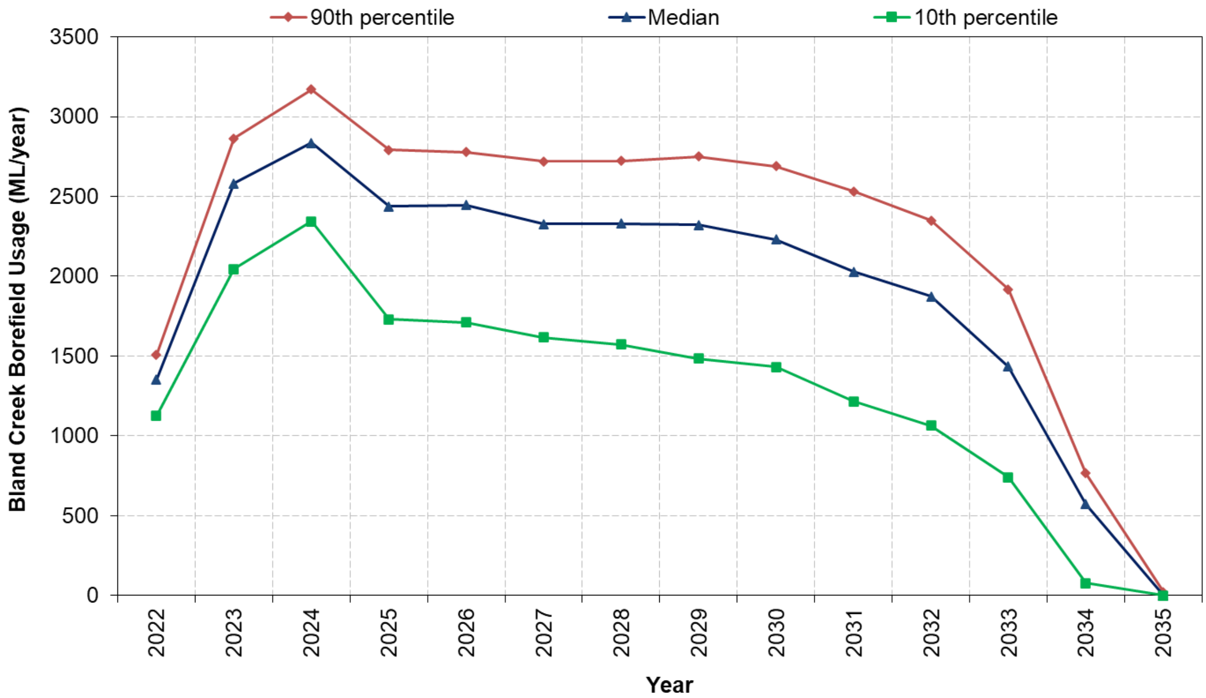
2.2.2 CGO External Water Demand

Graph 2 to Graph 4 show predicted annual water demands from external sources – eastern saline borefield, Bland Creek paleochannel bores and Lachlan River. Graph 2 to Graph 4 plot the median annual water demands, the 90th percentile demand (i.e. the demand that was predicted not to be exceeded in 90% of the simulated 133 climatic sequences) and the 10th percentile demand (i.e. the demand that was predicted not to be exceeded in 10% of the simulated 133 climatic sequences). These percentile plots indicate ranges within which the predicted annual volumes could vary, within these risk or confidence limits/levels.

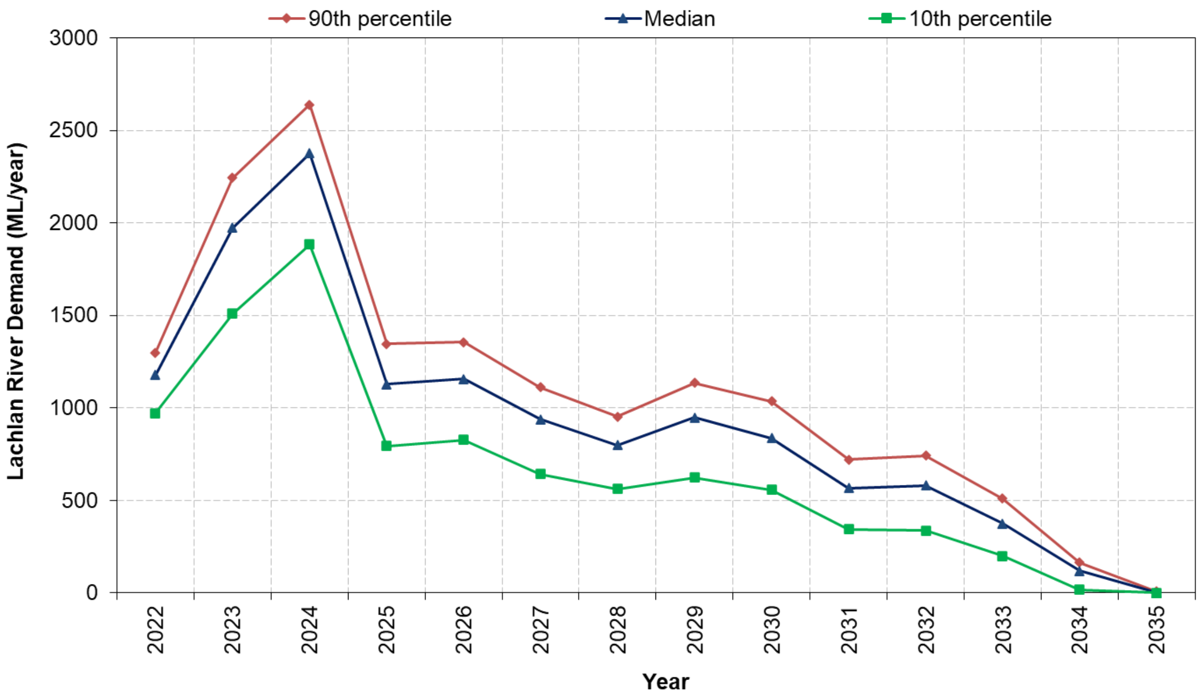
Note that the 2022 annual demand is for the period 1 July to 31 December 2022 and the 2035 annual demand is for the period 1 January to 31 July 2035.



GRAPH 2 PREDICTED ANNUAL EASTERN SALINE BOREFIELD USAGE



GRAPH 3 PREDICTED ANNUAL BLAND CREEK PALAEOCHANNEL BOREFIELD USAGE



GRAPH 4 PREDICTED ANNUAL DEMAND FROM LACHLAN RIVER ENTITLEMENTS

Graph 2 shows that the median annual demand from the eastern saline borefield is predicted to peak in 2024 at approximately 496 ML and to decline to approximately 375 ML by 2034. The median annual demand from the Bland Creek Palaeochannel borefield is predicted to peak in 2034 at approximately

2,835 ML/year and decline to approximately 571 ML by 2034 (refer Graph 3). Based on the 90th percentile model results, the annual demand from the Bland Creek Palaeochannel borefield is predicted to peak at 3,171 ML in 2024, which is less than the approved annual extraction rate of 3,650 ML.

Graph 4 shows that the predicted annual demand from Lachlan River licensed extraction is higher during the early years of the operation of the IWL due to the reduced reclaim associated with early operation of the IWL and higher ore production rates. The median annual demand from the Lachlan River is predicted to peak in 2024 at approximately 2,377 ML and decline from 2025 to approximately 119 ML in 2034.

The maximum predicted annual demand from the Lachlan River is approximately 2,639 ML based on the 90th percentile model results. Based on DPE-Water trading records (refer Section 1.5.2), there has been adequate allocation assignment water available on the market from this source in previous years to meet this predicted demand requirement.

The maximum predicted annual demand from the Lachlan River based on the 90th percentile model results (2,639 ML) is slightly less than that predicted based on the UDP EIS water balance modelling (2,850 ML) due to changes in the proposed production rates and predicted groundwater inflow rates.

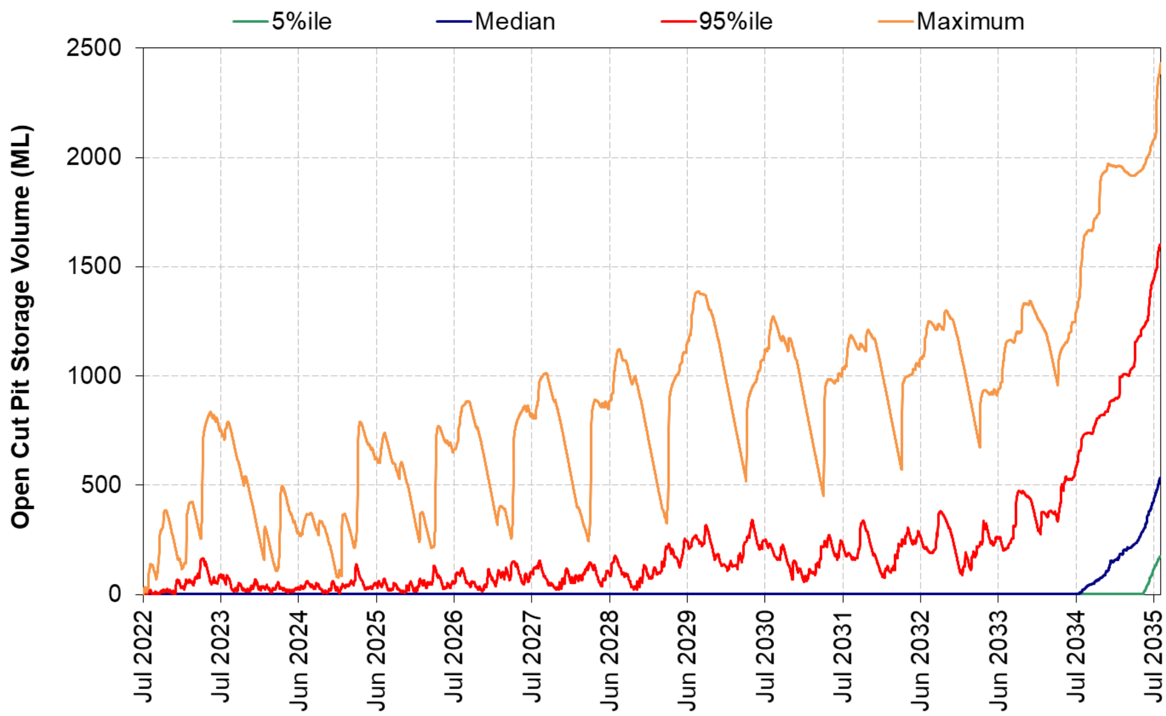
2.2.3 Supply Shortfall

No supply shortfalls were predicted for any of the 133 water balance model simulations.

2.2.4 Maximum Pit Water Volume

The maximum water volume predicted in the open-cut pit and for all 133 model simulations was 2,396 ML. However, the risk of such a large water volume is low. The model results shown in

Graph 5 indicate that, to the end of 2033, there is less than 5% chance that a pit water volume of 500 ML would be exceeded. The stored water volume in the open-cut pit is predicted to increase towards the end of the Modification life when the ore production rate and associated water demand decreases.



GRAPH 5 PREDICTED OPEN-CUT PIT STORAGE VOLUME

3 IN-PIT ACCESS AND UNDERGROUND INRUSH RISK

As described in Section 1, the Modification proposes to change the access points to the underground mine. Rather than accessing the underground mine via the approved box-cut and decline, access is proposed via the open-cut pit. Secondary access points to the underground mine are also proposed in addition to fresh air intake and exhaust adits. Table 3 presents the proposed elevation of the underground access points.

TABLE 3 PROPOSED UNDERGROUND ACCESS POINT ELEVATIONS

Component	Elevation (m Mine Datum)
Main portal	1,151
Warraga portal	1,102
Fresh air intake adit 1	957
Fresh air intake adit 2	1,065
Exhaust adit	1,011
Escape raise	1,119

The predicted maximum pit water volume of 2,396 ML (refer Section 2.2.4) corresponds with a maximum pit water level of 806 m Mine Datum (MD) which is significantly lower than the elevation of the lowest proposed access point (fresh air intake adit 1: 957 m MD). As such, there is negligible risk of underground inrush associated with the proposed in-pit access points.

4 POTENTIAL SURFACE WATER IMPACTS

4.1.1 Lake Cowal

The proposed in-pit access points are expected to involve substantially less surface disturbance and movement of material for construction than that of the approved box-cut and decline. Any surface disturbance would be contained within the current approved disturbance area. As such, no additional impact on inflows to Lake Cowal is expected to occur as a result of the Modification. Additionally, although the underground development will extend beneath Lake Cowal, groundwater impacts to Lake Cowal are predicted to be negligible (EMM, 2022).

No overflows were predicted in the water balance model from either of the contained water storages (D1 and D4) that could overflow to Lake Cowal in any of the 133 model simulations. This outcome is contingent upon pumped dewatering of these storages in between rainfall events. Pump extraction rates of 200 L/s and 105 L/s for storages D1 and D4 were assumed respectively.

4.1.2 Site Water Demand and Supply

Future water demand would be met (in part) by sourcing water from Lachlan River regulated flows (licensed extraction purchased on the open market). Given the provisions inherent in the *Water Management Act, 2000* regarding environmental flows, the impact of sourcing additional regulated flow from the Lachlan River would be neutral because, if not extracted by Evolution for use at CGO, the licences could be either purchased and the same water extracted by others or the water could be used by the existing licence holders if they were unable to sell the water on the open market.

It is recommended that sourcing water from the Bland Creek Palaeochannel Borefield continue in a similar manner as occurs currently, by alternating between this source and the Lachlan River to manage groundwater levels and provide flexibility with respect to extraction rates and the availability of allocation assignments in the Lachlan River.

Overall, it is concluded that there would be a low risk of more than a negligible hydrological impact on Lake Cowal due to the proposed Modification.

5 SUMMARY AND CONCLUSIONS

The Modification to SSD 10367 for the CGO UDP proposes to:

- change the way in which the underground workings are accessed from the surface;
- change the geometry of the access tunnels to the underground stopes; and
- increase the maximum annual production rate from 1.8 Mtpa to 2.6 Mtpa.

The proposed Modification has been reviewed in relation to potential surface water-related impacts with the key review findings summarised as follows:

- the demand from external sources (the eastern saline borefield, the Bland Creek Palaeochannel borefield and licensed extraction from Lachlan River water entitlements) is predicted to average 2,524 ML/year over the Modification life;
- based on the 90th percentile model results (i.e. a 10% assessed chance of being exceeded), the annual demand from the Bland Creek Palaeochannel borefield is predicted to peak at 3,171 ML in 2024, which is less than the approved annual extraction rate of 3,650 ML;
- the maximum predicted annual demand from the Lachlan River is approximately 2,639 ML based on the 90th percentile model results;
- there has been adequate allocation assignment water available on the market from the Lachlan River in previous years to meet this predicted demand requirement;
- no supply shortfalls were predicted for any of the 133 water balance model climatic scenarios;
- based on the model predictions, there is negligible risk of underground inrush associated with the proposed in-pit access points; and
- no additional impact on inflows to Lake Cowal is expected to occur as a result of the Modification.

Overall, it is concluded that there would be a low risk of more than a negligible hydrological impact on Lake Cowal due to the proposed Modification.

REFERENCES

Coffey Services Australia Pty Ltd (2020a). *Cowal Underground Development EIS Mine Site Hydrogeological Assessment*. August 2020.

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Appendix D

Groundwater impact assessment

EMM creating opportunities

Evolution MINING

CGO Underground Development Project Modification 1

Groundwater impact assessment

Prepared for Evolution Mining Pty Limited

June 2022

CGO Underground Development Project Modification 1

Groundwater impact assessment

Evolution Mining Pty Limited

E210776 RP5

June 2022

Version	Date	Prepared by	Approved by	Comments
1	21 June 2022	Katharine Bond	Jeff Whitter	
2	29 June 2022	Katharine Bond	Jeff Whitter	Final

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1 Introduction

1.1 Background information

The Cowal Gold Operations (CGO) is an operating gold mine on the edge of Lake Cowal, approximately 38 kilometres (km) north-east of West Wyalong in the Bland Shire local government area. CGO is owned and operated by Evolution Mining Pty Limited (Evolution). It is located within mining leases (ML) ML 1535 and ML 1791.

Evolution operates CGO under two Ministerial development consents. DA 14/98, which was granted in 1999, generally allows open-cut mining and ore processing on site, until the end of 2040. SSD 10367, granted in 2021 generally allows underground stope mining, backfilling of stopes and delivery of ore to the processing plant, also until the end of 2040.

Evolution is seeking to modify SSD 10367 (Mod 1) (referred hereafter as the 'proposed modification'), pursuant to Section 4.55(2) of the *Environmental Planning and Assessment Act 1979* (EP&A Act) to change the access points to the underground mine, to change the geometry of access tunnels and to increase the annual production rate.

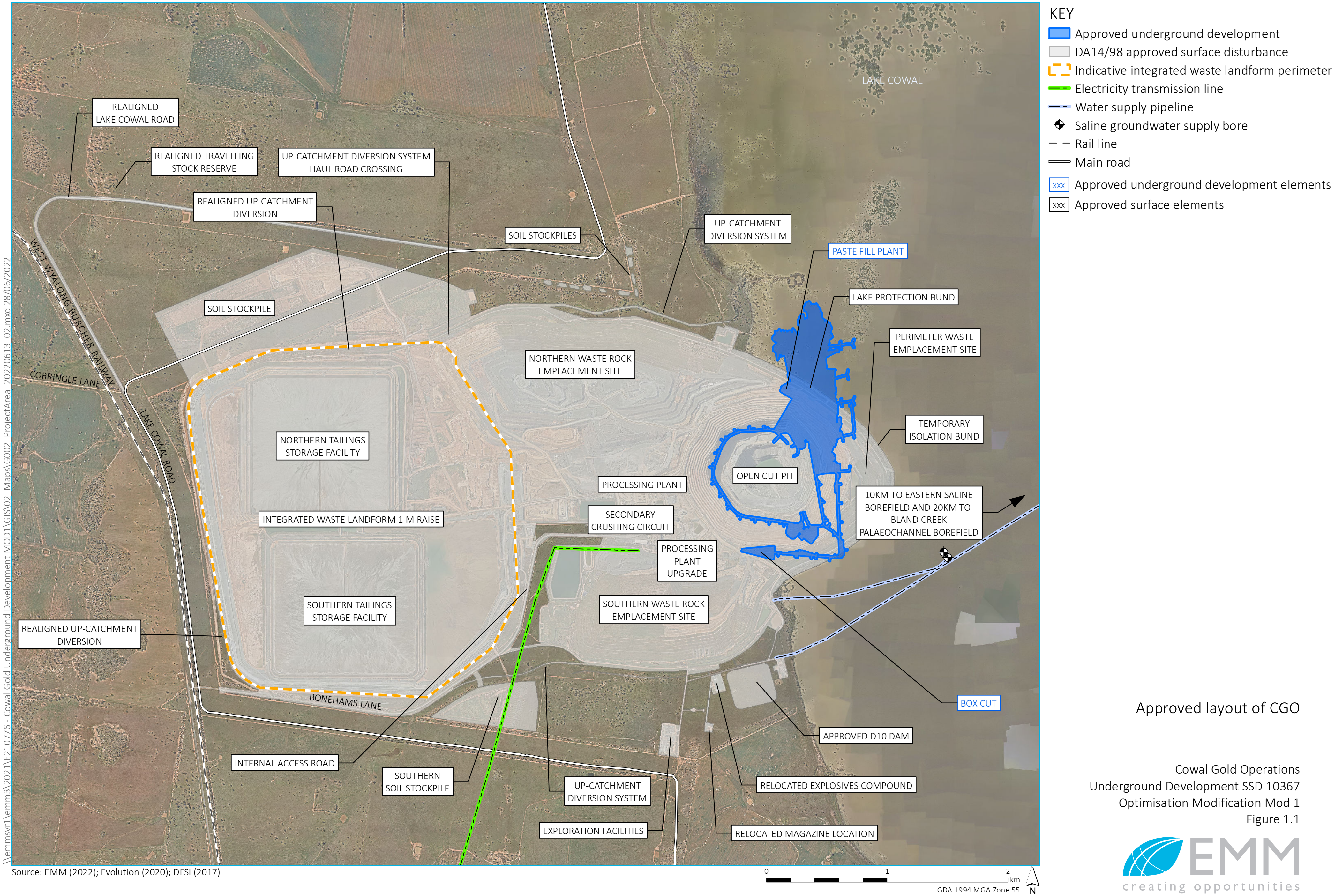
This report provides an assessment of the potential groundwater impacts from the proposed modification.

1.2 Approved project

In September 2021, The Minister's delegate approved the CGO Underground Development Project (SSD 10367) (referred hereafter as 'the underground project'). The development consent allows:

- underground stope mining at a rate of up to 1.8 million tonnes per annum (Mtpa), until the end of 2040;
- delivering ore to the CGO processing plant;
- developing a paste fill plant to make cement paste from tailings, and backfilling the stopes with the paste; and
- developing primary and secondary access points to the underground mine, including a box-cut entry, to provide personnel, materials, ore and waste rock haulage and ventilation services.

The layout of the CGO site including the underground mine workings, access points and access tunnels, is shown in Figure 1.1.



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Source: EMM (2022); Evolution (2020); DFSI (2017)

Approved layout of CGO

Cowal Gold Operations
Underground Development SSD 10367
Optimisation Modification Mod 1
Figure 1.1

EMM
creating opportunities

1.3 Proposed modification

1.3.1 Overview

Since the approval of the underground project, Evolution has undertaken further technical design work and mine scheduling in order to optimise the development and ongoing operation of the underground mine.

This has resulted in changes to the underground mine design, including changes to the access points and taking the access tunnel development from the hanging wall (ie located predominantly on the western side of the orebody) to the footwall on the eastern side of the orebody. Further design iterations have resulted in proposed changes to the way access to the underground mine is accessed.

A review of the mining schedule identified that the underground project can be undertaken to achieve a higher annual ore production rate than the approved rate of production.

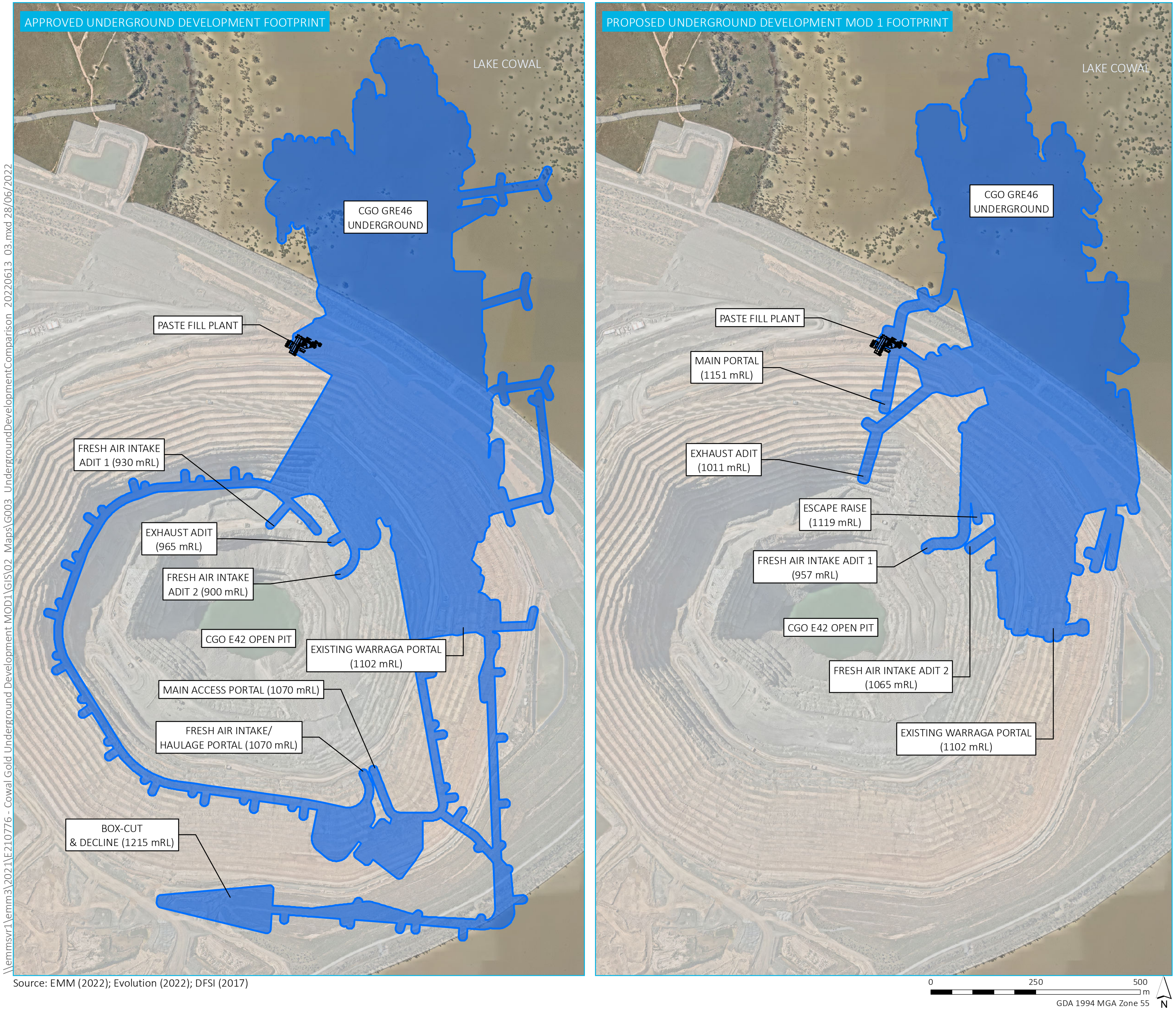
The proposed modification seeks to:

- change the way in which the underground workings are accessed from the surface (ie no box-cut);
- change the geometry of the access tunnels to the underground stopes; and
- increase the production rate from 1.8 Mtpa to 2.6 Mtpa.

No changes are proposed to the maximum amount of ore that would be extracted over the life of the project, the approved mining method or stoping areas or the operation of the pastefill plant.

It should be noted that the above changes will not require a modification to the approved existing open-cut mining, processing plant and integrated waste landform operations. These supporting activities will continue to be regulated under DA 14/98.

A comparison of the proposed modification and approved underground project footprints is shown in Figure 1.2.



KEY

- Underground development footprint
- Paste fill plant layout
- DA14/98 approved surface disturbance

Source: EMM (2022); Evolution (2022); DFSI (2017)

Comparison of the approved underground development and modification footprints

Cowal Gold Operations
Underground Development SSD 10367
Optimisation Modification Mod 1
Figure 1.2

EMM creating opportunities

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1.3.2 Underground access

The proposed modification seeks to change the access points to the underground mine as summarised in Table 1.1.

Table 1.1 Summary of changes to underground mine access points

Access point	Purpose as approved (SSD 10367)	Proposed changes
Main Portal (primary access)	The main service entry for the underground mine for personnel and vehicles	Replaced by a new primary access portal in the north of the E42 pit which will provide worker access, ventilation, maintenance access, ore haulage and waste haulage.
Box-cut	Provides personnel and material access to the mine and provide access for maintenance light vehicles	
Fresh Air Intake/Haulage Decline Portal	Provides a fresh air connection for lower working areas, an emergency egress route from underground workings and an alternate haulage route	
Fresh Air Intake Adit 1	Provides a fresh air ventilation for the lower stope working areas	No change – precise locations to be determined during detailed design
Fresh Air Intake Adit 2	Provides a fresh air ventilation for the material transfer points and for atmospheric dust control.	
Exhaust Adit	Provides exhaust air connection for material transfer points and for atmospheric (dust and air quality) control	
Warraga Decline Portal (secondary access)	Access to the exploration decline and provision of services and ventilation	Proposed for worker access, maintenance access, ore haulage and waste haulage. This will allow separation of vehicles that are transporting ore in the north and south of the mine respectively.

1.3.3 Change to access tunnel geometry

The proposed modification seeks to modify the location of access tunnels between the development declines and the stoping areas. This will require the relocation of the access tunnels to the east and further below Lake Cowal in comparison to the approved layout.

This change will allow for greater safety due to improved stability of the mine workings during production activities and greater efficiency in orebody extraction, ultimately reducing the amount of development required.

1.3.4 Ore production rate

The approved ore production rate is 1.8 Mtpa. The proposed modification seeks to increase this rate to 2.6 Mtpa. However, it should be noted that the ore production rate from the underground project is expected to be below 2 Mtpa for most years, except for five years when it is expected to reach between 2 and 2.5 Mtpa.

Despite this increase in the annual production rate, there would be no change to the total resource that will be extracted for the underground project (ie 27 Mt).

1.4 Approved SSD 10367 and proposed Mod 1 summary

The proposed changes to the underground project (Mod 1) are compared to the approved activities (SSD 10367) in Table 1.2.

Table 1.2 Mod 1 summary of changes

Aspect	Approved SSD 10367	Proposed Mod 1
Life of mine	To 31 December 2040	No change
Resource	Approximately 27 Mt	No change
Ore production rate	1.8 Mtpa	Up to 2.6 Mtpa
Waste rock production	5.74 Mt	No change
Gold production	1.8 Moz	No change
Mining method	Production of ore via mechanised long hole open stoping	No change
Box-cut	Development of a box-cut entry adjacent to the open-cut pit, which will be the main access for personnel and materials to the underground mine and will be used to transport ore to the surface for processing.	Removal of the box-cut from the underground project and its replacement with the primary access portal in the north of the E42 pit.
Declines	Excavation of two declines (in addition to the existing Warraga Decline) to provide underground access and ventilation: one decline via a portal on the existing open-cut pit and the other via a box-cut. The declines will be approximately 6 m wide by 6m high and will extend approximately 1.5 km to the point at which the first production drive commences.	A new primary access portal (in addition to the existing Warraga Decline) in the north of the E42 pit will replace the box-cut. The new portal will be approximately 6 m wide by 6m high and will extend approximately 2 km to the point at which the first production drive commences.
Decline access	Six access points to the main decline for access, ore haulage, ventilation circuit, underground services and emergency egress.	Change to the locations of access points to the declines. Removal of the Fresh Air Intake/Haulage Decline portal and ventilation drive. Use of the Warraga Decline portal for access and ore and waste rock haulage
Mining extent	Development of the underground mine will be in stages, as main decline is progressively extended at depth. The underground footprint is estimated to be approximately 135 ha and final depth of approximately -850 m AHD.	No change
Paste backfill	Development of a paste fill plant, and backfilling excavated stopes with cemented paste fill made from cement and tailings.	No change

Table 1.2 Mod 1 summary of changes

Aspect	Approved SSD 10367	Proposed Mod 1
Groundwater-related water supply sources	The saline groundwater supply bores within ML 1535. The Eastern Saline Borefield located approximately 10 km east of Lake Cowal's eastern shoreline. The Bland Creek Palaeochannel Borefield, which is pumped from four production bores located approximately 20 km to the east-northeast of the CGO in accordance with approved extraction limits.	No change
Workforce	Construction: estimated peak workforce of approximately 225 FTE employees and contractors, which will be used to develop the underground project and the supporting surface infrastructure. Operations: an average of around 160 FTE employees working over two shifts.	No change

2 Original groundwater impact assessment summary

The original groundwater impact assessment (GIA) for the approved underground project (SSD 10367) was completed by Coffey Services Australia Pty Ltd (Coffey) in September 2020. The 2020 GIA used results from the predictive three-dimensional numerical modelling based on an existing mine site numerical groundwater flow model (built using FEFLOW, Version 7.2 software) which was completely re-worked, considering the proposed underground mining to the north of the existing open pit.

Two further hydrogeological reports were also prepared as part of the assessment of the approved project:

1. Coffey (2021a) CGO Underground Development EIS - *Addendum 1 of the hydrogeological assessment*, prepared for EMM Pty Ltd.

2. Coffey (2021b) *CGO Underground Development EIS - Addendum 2 of the hydrogeological assessment*, prepared for EMM Pty Ltd.

The original GIA also used a numerical groundwater model for the palaeochannel borefield for mine water supply. This model was not used in this assessment due to this modification proposing no changes to mine water supply.

2.1 Groundwater level impacts

During underground mining, impacts to groundwater levels were predicted to be minor. Groundwater drawdown resulting from stopes, access tunnels and the existing open-cut pit were predicted to be mostly contained within ML 1535 and ML 1791, apart from small areas to the north and south where the 1 m drawdown contour was marginally outside of ML 1535. No external water bores or users were predicted to be affected by the drawdown.

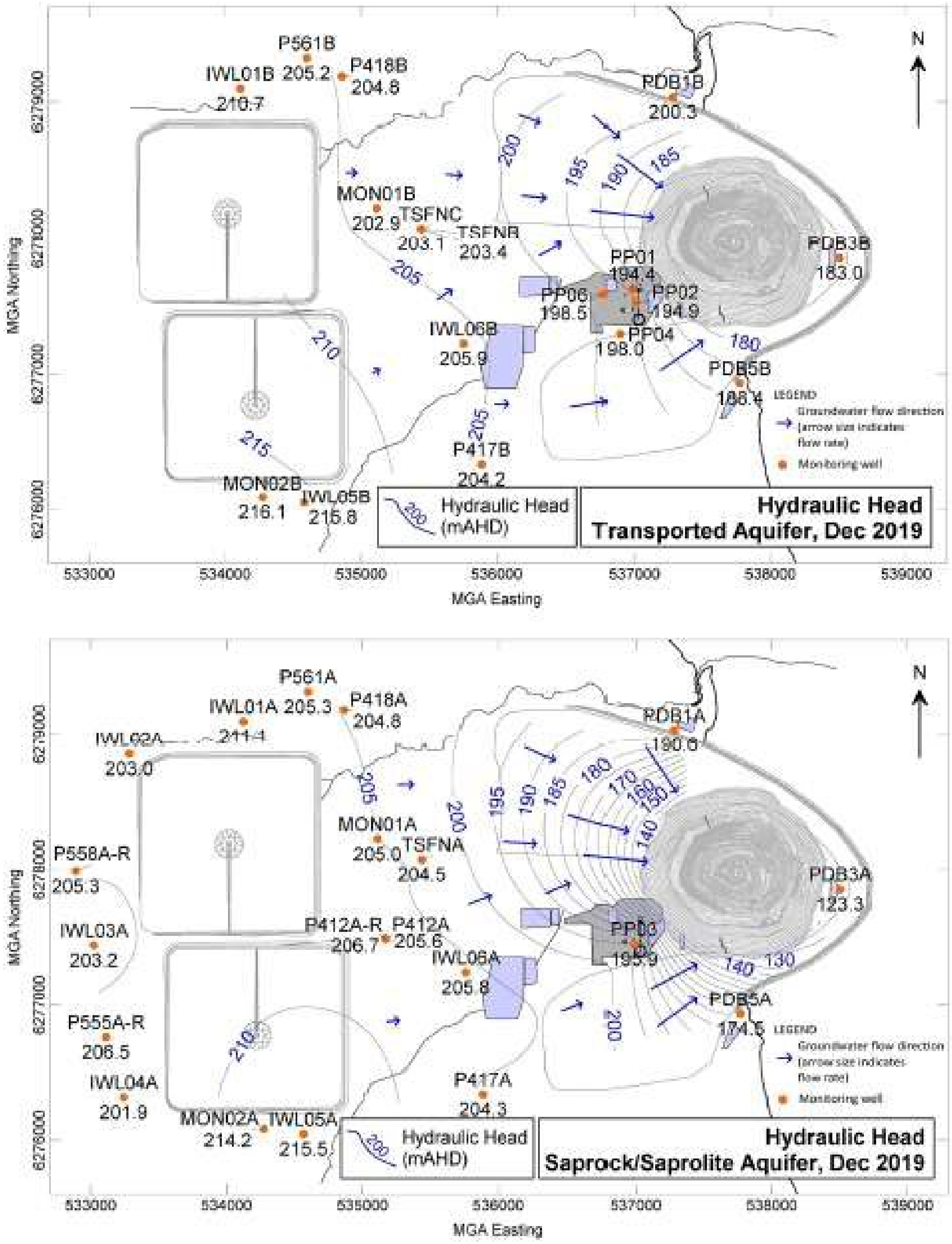
Mounding of the groundwater table caused by seepage from the tailings storage facilities (TSF) and the Integrated Waste Landform (IWL) was predicted, but the groundwater head draws that leakage towards the open-cut pit (Figure 2.1). The recharge from the IWL was not expected to result in any material impacts to local groundwater resources and continues to be closely monitored during existing approved operations.

Following mine closure, groundwater inflow and surface water run-off to the open-cut pit was expected to result in a lake forming in the open-cut pit, with the pit lake level rising to a level where groundwater inflow and surface water run-off was balanced by evaporation from the pit lake. There was predicted to be a slight recovery in groundwater heads around the open-cut pit in the Transported, Saprolite and Saprock units[1] of around 5 m between 2038 and 2058 and then a negligible change between 2058 and 2138. Predicted impacts on groundwater levels after mine closure was therefore considered to be minor.

[1] Locally, at the CGO site, four hydrogeological units have been identified:

1. The Transported unit: comprising alluvium (thick clay sequences and more permeable zones of gravel within a sandy clay matrix) of the Quaternary-aged Cowra Formation. The Cowra Formation is laterally equivalent to the Transported unit (Barrick 2010).
2. The Saprolite unit: underlies the Transported unit and is of relatively low hydraulic conductivity. The unit comprises extremely weathered rock, often weathered to clay.
3. The Saprock unit: underlies the Saprolite unit and occurs in the weathered fractured surface of the Lake Cowal Volcanics. The unit comprises highly to moderately weathered rock with some zones of clay.
4. The Primary Rock unit: consisting of slightly weathered to fresh rock underlying the Saprock unit. This unit is generally considered to be less fractured and less permeable than the Saprock.

Following mine closure, once the tailings emplacement is complete, it was predicted that recharge would taper off and mounding would dissipate over time with the diminishing hydraulic head in the tailings mass. The hydraulic pull from the resulting pit lake as compared to the regional hydraulic head would remain in perpetuity due to open water evaporation being greater than the environmental recharge.



Source: Figure 6-9 of the Cowal Underground Development EIS Mine Site Hydrogeological Assessment (August 2020)

Figure 2.1 Observed hydraulic head in December 2019 for the Transported and Saprolite units (Coffey 2020)

2.2 Groundwater inflows

The CGO open pit, in operation since 2005, is a significant groundwater sink. Groundwater is drawn toward the open pit, mainly from the fractured rock aquifer in which most of the open pit is excavated (Coffey, 2021b).

Combined groundwater inflows into the open-cut pit, proposed stopes and access tunnels were predicted to increase from 1 ML/day in 2020 to a peak of 2.8 ML/day between 2031 and 2038. Inflow to the open-cut pit on its own was predicted to fall from 1 ML/day in 2020 to 0.5 ML/day between 2031 and 2038.

Following mine closure, groundwater inflow to the open-cut pit was expected to rise from approximately 0.5 ML/day in 2038 to 0.9 ML/day by 2066. From 2066 to 2100 the inflow rate to the open-cut pit was predicted to gradually fall to approximately 0.65 ML/day then remain at around that rate as the inflow to the pit is almost balanced by evaporation from the pit lake surface. From 2040 to 2066 it was expected that the access tunnel voids and the paste backfill in the stopes would gradually fill with groundwater. Inflow into these areas was predicted to fall from 1.65 ML/day to less than 0.1 ML/day during this time.

2.3 Impacts to Lake Cowal

Lake Cowal is a surface water fed water body, originating from Bland Creek and occasional flooding of the Lachlan River. It is separated from the proposed underground development by a 120 m combined thickness of lake sediments and extremely weathered to fresh rock, with vertical permeabilities of less than 1 x 10^{-3} m^3/day. As a result of the low vertical permeabilities, it was estimated that most of the groundwater inflow (up to 1.8 ML/day) would be from deep groundwater originating in the rock surrounding the underground development and not from Lake Cowal.

When Lake Cowal is full it occupies an area of 13,000 hectares and would therefore lose on average 534,000 m^3/day to evaporation (assuming 1.5 m net pan evaporation). This means that the average rate of evaporation from the surface of Lake Cowal is approximately 300 times the predicted maximum rate of groundwater inflow due to the approved project alone (1,800 m^3/day). As such, the impact of mine groundwater inflow on the water levels of Lake Cowal was considered to be negligible (Coffey, 2021a).

The water in the completed mine workings beneath Lake Cowal is predicted to remain below 85 mAHD, while the water level within the open pit void would be below 80 mAHD 200 years after the end of mining. This is well below the level of the bed of Lake Cowal (201.5 mAHD) and so it was considered that there was no prospect of seepage from the mine entering Lake Cowal (Coffey, 2021b).

2.4 Impacts to private water supply works

Modelled groundwater head drawdown around the mine site due to the open pit and underground development increased with depth below ground (Coffey, 2021b).

Figure 2.2 shows that for public bores around the mine site with an elevation above 150 m AHD, the combined groundwater head drawdown from the approved open pit development and the proposed underground development would be less than 2 m in January 2038 when compared to the groundwater head since 2004. The date is representative of the period immediately before the end of underground mining. Figure 2.3 shows that for public bores around the mine site with an elevation of less than 150 m AHD, the combined groundwater head drawdown from approved open pit development and the proposed underground development would be less than 2 m in January 2038 (Coffey, 2021b).

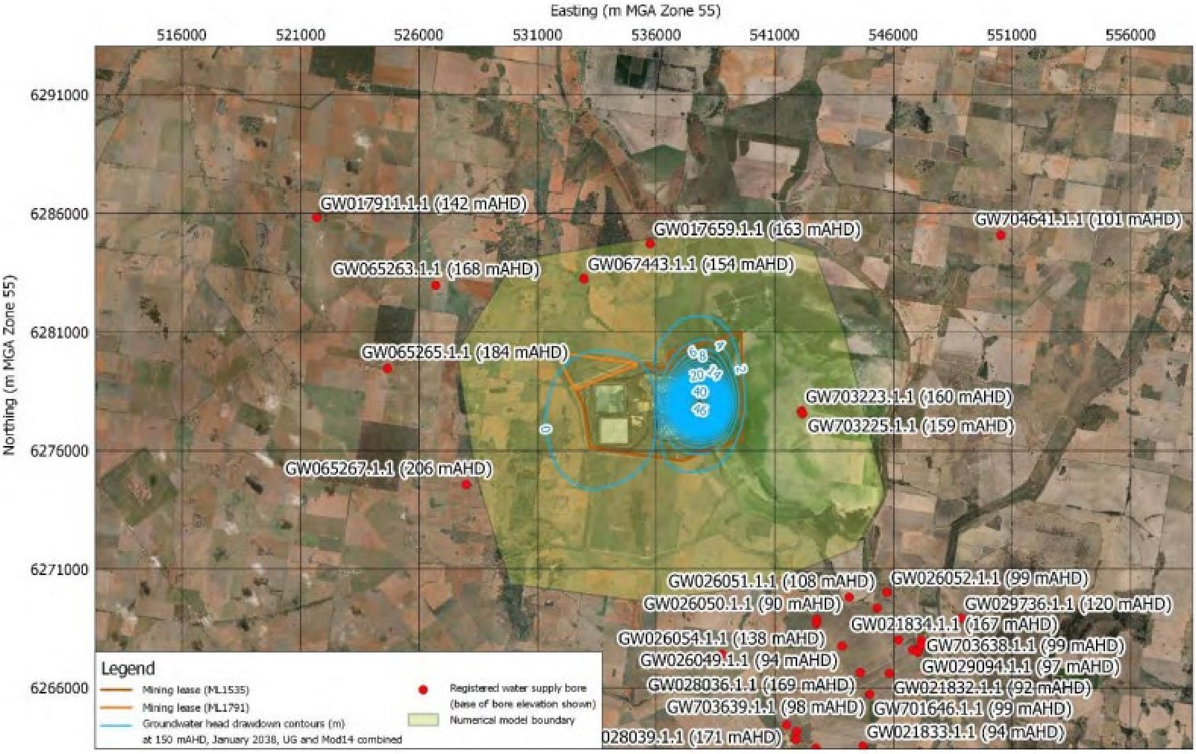


Figure 2.2 Combined (Mod 14 and proposed underground development) drawdown at 150 mAHD, January 2038 (Coffey 2021b)

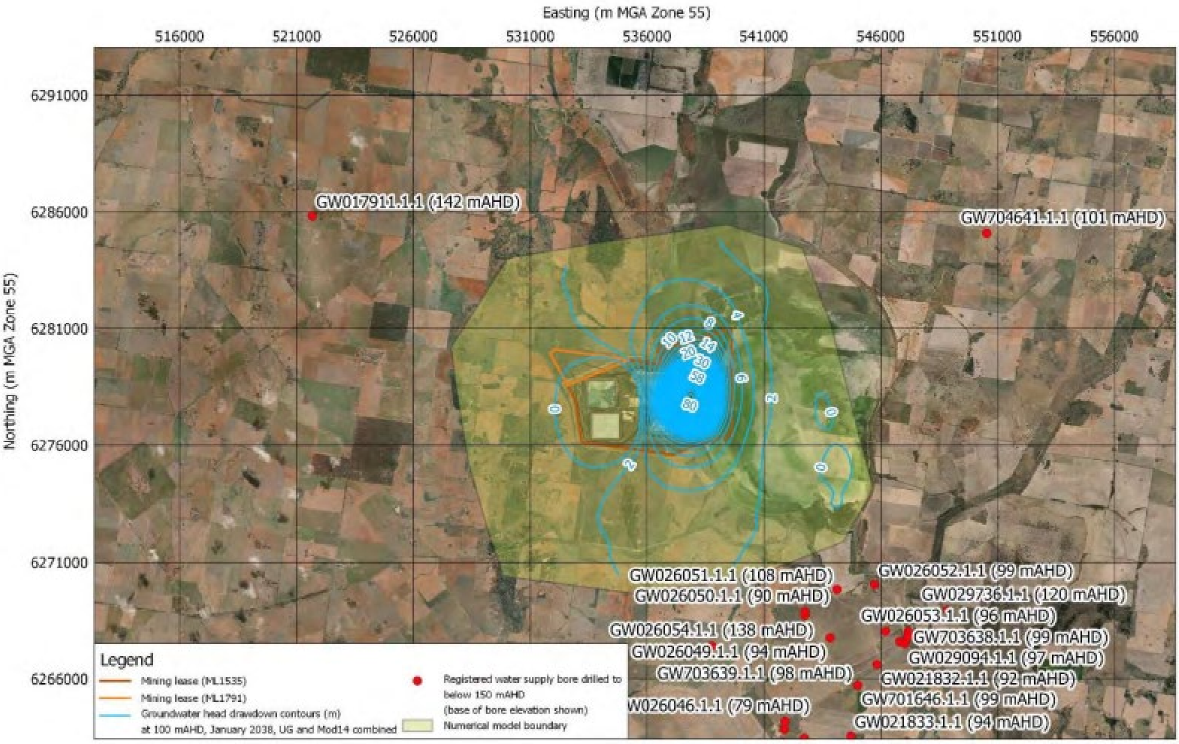


Figure 2.3 Combined (Mod 14 and proposed underground development) drawdown at 100 mAHD, January 2038 (Coffey 2021b)

2.5 Groundwater quality impacts

An assessment of contaminant migration, based on a conservative assessment of the movement of contaminants originating from the IWL, was undertaken. Contaminants identified as having the potential to be released from the IWL included cyanide, arsenic, zinc and other heavy metals (Coffey 2018). Of these, cyanide is the only substance introduced by the mining operation as the metals and arsenic are derived from the mine ore.

The assessment predicted that after 100 years the potential for groundwater quality changes due to seepage from the IWL stored water would extend up to approximately 2 km from the IWL walls (there were no registered water supply bores within this distance). Consideration of cyanide decay times indicated that cyanide concentrations were predicted to fall well below detectable limits prior to seeping outside the CGO mine area.

Further assessment in Coffey (2020b) showed there was no evidence of any risk to existing registered groundwater users from TSF/IWL groundwater seepage in the 200 years post-mining (Figure 2.4).

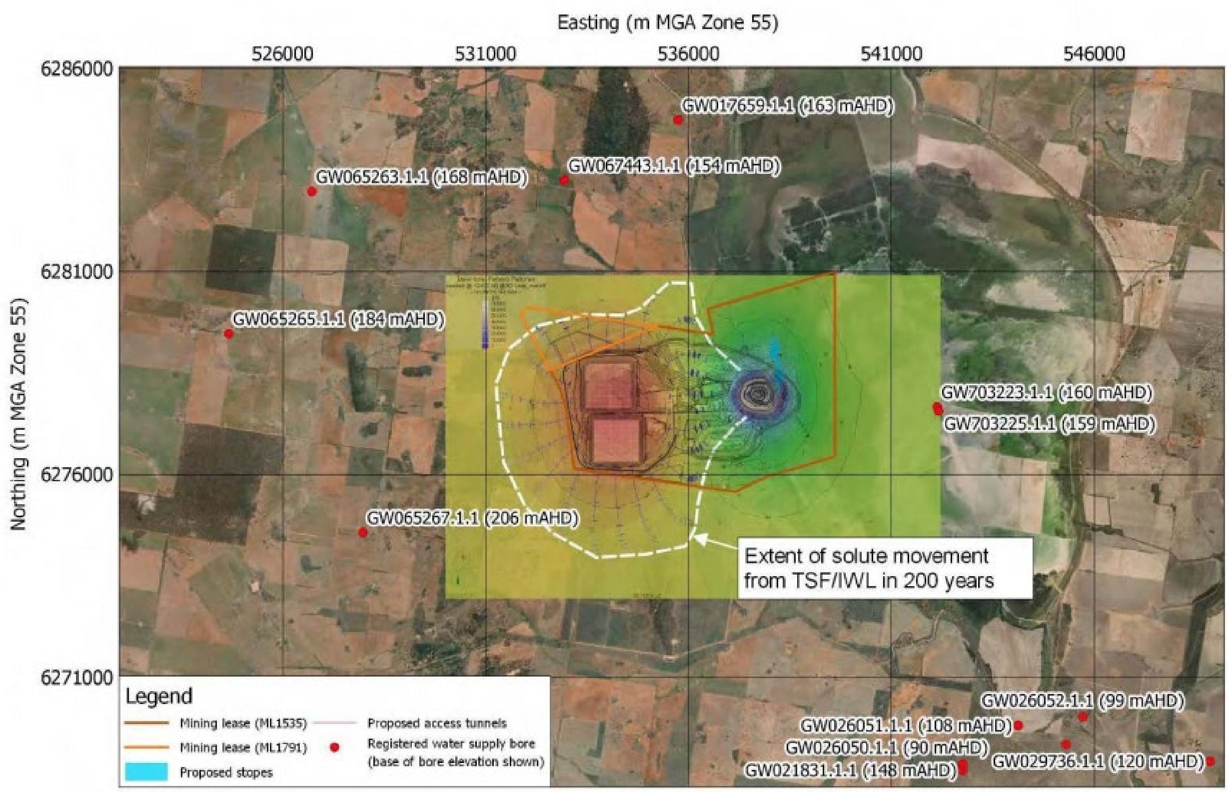


Figure 2.4 Predicted extent of solute movement in 200 years (from Coffey, 2021b)

2.6 NSW Aquifer Interference Policy

The Aquifer Interference Policy (AIP) was released by the NSW government in September 2012 to address water licensing and the potential impacts of aquifer interference activities within NSW. It provides a framework for assessing the impacts of aquifer interference activities on water resources. The AIP is relevant to CGO as it applies to mining activities such as open cut voids and the disposal of water taken from aquifers.

According to Coffey (2020), groundwater quality within ML1535 has EC generally in the range of 30,000 microsiemens per centimetre (µS/cm) to 55,000 µS/cm for the Transported, Saprolite and Saprock units. Data was not available for the Primary Rock, but the EC in the Primary Rock is expected to be similar (or higher due to the presence of salts in the rock). This equates to a total dissolved solids concentration of between 19,200 mg/L and 35,200 mg/L. The groundwater source at CGO is, therefore, defined by the AIP as a "less productive groundwater source".

The AIP states that a proposed development must address minimal impact considerations for impacts on water table, water pressure and water quality. It requires planning for measures if the actual impacts are greater than predicted, including making sure that there is sufficient monitoring in place.

An assessment of the approved project against the minimal impact considerations of the AIP is set out in Coffey (2020) and summarised in Table 2.1.

Table 2.1 Assessment of minimal impact considerations detailed in Aquifer Interference Policy (from Coffey 2020)

Minimal impact consideration	Assessment provided
(i) No more than a specified cumulative variation in the water table within 40 m from a high priority groundwater dependent ecosystem (GDEs) or a high priority culturally significant site.	The model-predicted groundwater drawdown up to 20 years post-mine closure remained largely within ML1535. As there are no high priority GDEs, priority culturally significant sites or supply works within ML1535 (or within 40 m of the boundary of ML1535), minimal impact considerations (i) to (iii) were met.
(ii) No more than a specified limit in the water table decline at any water supply work.	
(iii) No more than a specified cumulative pressure head decline at any supply work.	
(iv) Any change in groundwater quality that lowers the beneficial use category of the groundwater source beyond 40 m from the activity.	During the life of the CGO, dewatering from the open pit, stopes and access tunnels would only have a small and localised (i.e. within ML1535) impact on groundwater quality. Over the longer term, groundwater would flow towards the open pit, ultimately terminating there. The groundwater quality in the region surrounding the open pit void was not expected to change significantly due to this process, though the quality of the water within the open pit was expected to change (e.g. salinity will increase). The beneficial use of groundwater was not expected to change due to dewatering or the presence of the open pit. Thus, minimal impact consideration was met.
(v) No increase of more than 1% per activity in long-term average salinity in a highly connected surface water source at the nearest point of activity.	As the equilibrium surface water level in the open pit (the pit lake) following the end of mining would be well below the ground surface, water from the pit lake would not be released. Thus, it was not classified as a highly connected surface water source, meeting minimal impact consideration.
(vi) No mining activity below the natural ground surface within 200 m laterally from the top of the high bank and 100 m vertically beneath of a highly connected surface water source that is defined as a "reliable water supply".	There were no known "reliable water supplies" within 200 m laterally from the top of the high bank. Lake Cowal is an ephemeral lake, and so was not considered to be a "reliable water supply". Thus, minimal impact consideration was met.

2.6.1 Groundwater licensing requirements

With respect to licensing under the *Water Management Act 2000* (WM Act), the AIP states:

> A water licence is required under the *Water Management Act 2000* (unless an exemption applies, or water is being taken under a basic landholder right) where any act by a person carrying out an aquifer interference activity causes:
>
> - the removal of water from a water source; or

- the movement of water from one part of an aquifer to another part of an aquifer; or
- the movement of water from one water source to another water source, such as:
  - from an aquifer to an adjacent aquifer; or
  - from an aquifer to a river/lake; or
  - from a river/lake to an aquifer.

The CGO lies within the following groundwater sources:

- Upper Lachlan Alluvial Zone 7 Management Zone of the *Water Sharing Plan for the Lachlan Alluvial Groundwater Sources 2020.*
- Lachlan Fold Belt groundwater source of the *Water Sharing Plan for the NSW Murray Darling Basin Fractured Rock Groundwater Sources 2020.*

The numerical modelling predicted dewatering rates from the affected groundwater sources due to inflow to the open pit, stopes and tunnels. It was assessed that 90% of groundwater inflow originates from the fractured rock aquifer with the remaining 10% from the overlying Upper Lachlan Alluvium (Coffey, 2020).

The predicted annual groundwater volumes required to be licensed within each Water Sharing Plan for the approved project are summarised in Table 2.2.

Table 2.2 Groundwater licensing requirement summary (from Coffey 2020)

Water sharing plan	Management zone / Groundwater source	Predicted groundwater inflow/extraction volume requiring licensing (ML/year)		Currently licensed unit shares (February 2018)
		Existing	During modification	
Lachlan Alluvial Groundwater Sources 2020	Upper Lachlan Alluvial Zone 7 Management Zone	282 (maximum)	293 (maximum)[1]	366
NSW Murray Darling Basin Fractured Rock Groundwater Sources 2020	Lachlan Fold Belt groundwater source	212 (average) 277 (maximum)	759 (average)[2] 1,004 (maximum)[3]	3,294

1. Includes 256 ML/year extraction associated with the saline supply bores within ML1535 based on peak usage of 0.7 ML/d, plus 10% of modelled maximum inflow from the Upper Lachlan Alluvial Zone.
2. Modelled average total inflow (796 ML/year) minus average open pit inflow from Upper Lachlan Alluvial Zone (37 ML/year)
3. Modelled maximum total inflow (1022 ML/year in 2031-2039) minus open pit inflow from Upper Lachlan Alluvial Zone (18 ML/year in 2021-2039)

Post mining the long-term inflow rate was assessed to be 230 ML/year from the fractured rock groundwater source and less than 7.3 ML/year from the Upper Lachlan Alluvial Zone. These volumes would continue to require licensing.

3 Assessment for the proposed modification

The underground project as proposed under Mod 1 will be substantially the same as the project for which consent (SSD 10367) was originally granted in 2021. Therefore, the potential for groundwater impacts to increase because of the proposed modification is relatively low. To assess the potential change to the groundwater impacts, an assessment (this assessment) was undertaken using an updated mine site groundwater model. The model has been used to:

- assess the differences in drawdown associated with the new optimised underground mine plan; and
- assess any potential changes to the mine inflow volumes.

3.1 Methodology

As previously mentioned, Coffey developed a numerical groundwater mine site FEFLOW model which was used to inform the EIS and RTS approval process for the approved project.

The mine site groundwater model was updated in the undertaking of this proposed modification GIA. For example, there was some refinement of the model mesh required to model the new location of the proposed stopes and access tunnels. Further details regarding the numerical model development is provided in Appendix A.

The calibration of the groundwater model was not reviewed as there were no changes to any applied parameters in the groundwater model.

3.2 Proposed changes and potential impacts

3.2.1 Groundwater level impacts

The modelled groundwater table drawdown for the proposed modification at 2038 (towards the end of mining), 2058 and 2138 are shown in Figure 3.1 to Figure 3.3.

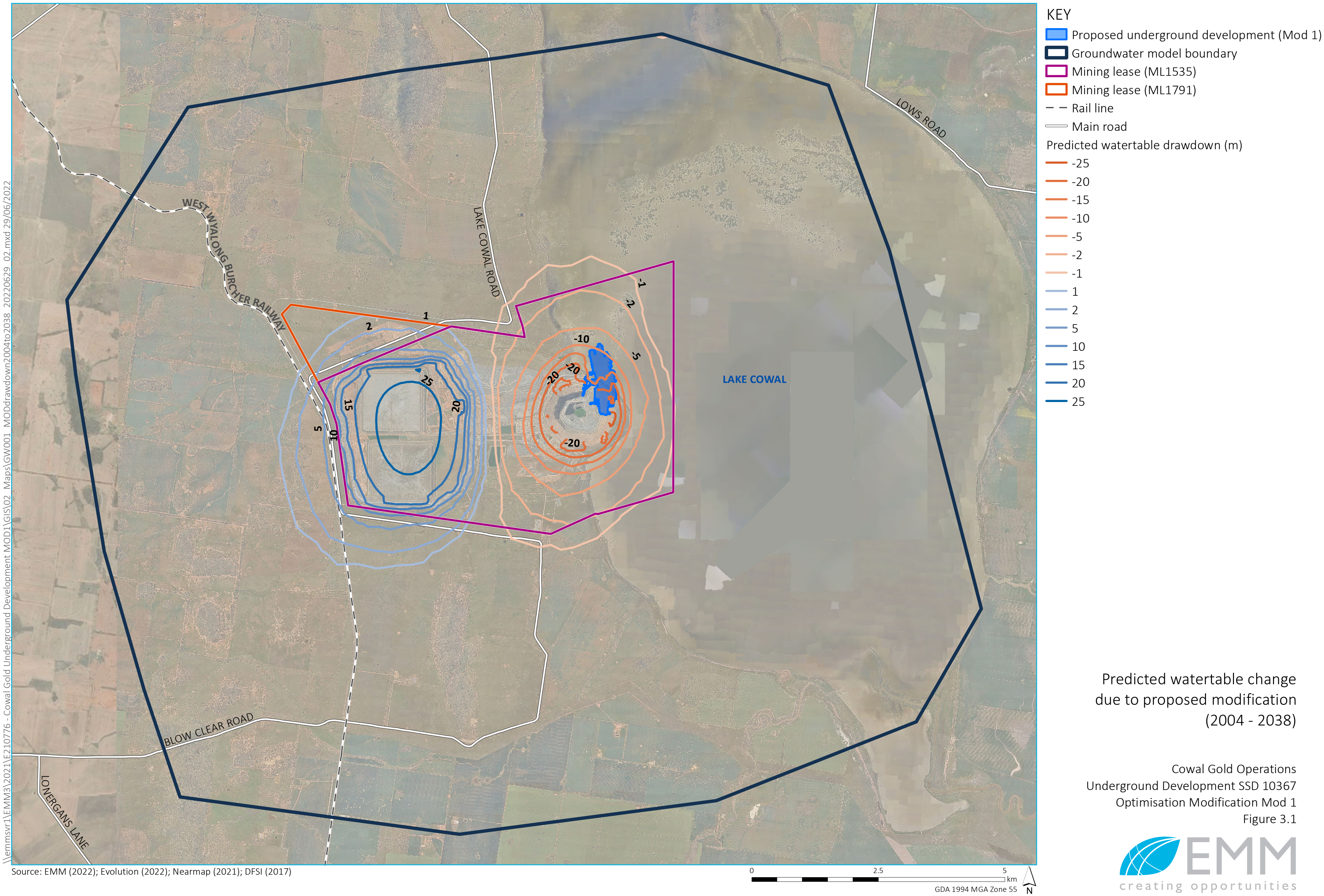
During underground mining, impacts to groundwater levels are still predicted to be minor. Groundwater drawdown resulting from stopes, access tunnels and the existing open-cut pit are still anticipated to be mostly contained within ML 1535 and ML 1791, apart from small areas to the north and south where the 1 m drawdown contour are marginally outside of ML 1535 (Figure 3.1).

Following mine closure, groundwater inflow and surface water run-off to the open-cut pit is still expected to result in a lake forming in the open-cut pit. There is still predicted to be a slight recovery in groundwater heads around the open-cut pit in the Transported, Saprolite and Saprock units of around 5 m between 2038 and 2058 and then a negligible change between 2058 and 2138. Predicted impacts on groundwater levels after mine closure is therefore still considered to be minor.

Mounding of the groundwater table caused by seepage from the IWL is still predicted during mine operation, but the groundwater head will still draw that leakage towards the open-cut pit. Following mine closure, once the tailings emplacement is complete, it is still predicted that recharge will taper off and mounding will dissipate over time with the diminishing hydraulic head in the tailings mass. The hydraulic pull from the end pit lake relative to the regional groundwater levels will continue forever due to evaporation rates being higher than recharge.

The difference between the approved project and proposed modification modelled groundwater table drawdown in 2038, 2058 and 2138 are shown in Figure 3.4 to Figure 3.6. During underground mining, Figure 3.4 indicates that drawdown isn't as pronounced in the south-east section of ML 1535 with the proposed modification. Proposed changes to the mine plan (removal of box cut and underground development access to the east of the open pit) are the main factors in the predicted watertable drawdown in the south-east and western areas of the open pit. During post-mining, most of the differences in the modelled groundwater table drawdown are shown to occur around the open cut and underground mines themselves, extending to the eastern section of ML 1535.

Figure 3.4 to Figure 3.6 show that during both mining and post-mining, differences in drawdown depth between the approved project and proposed modification are mostly contained within ML 1535 and ML 1791. All drawdown variations are within plus or minus 5 m. This again indicates that predicted impacts on groundwater levels during and after mine closure is still considered to be minor under the proposed modification.



\\emmsvr1\EMM3\2021\E210776 - Cowal Gold Underground Development MOD1\GIS\02_Maps\GW001_MODdrawdown2004to2038_20220629_02.mxd 29/06/2022

Source: EMM (2022); Evolution (2022); Nearmap (2021); DFSI (2017)

Predicted watertable change
due to proposed modification
(2004 - 2038)

Cowal Gold Operations
Underground Development SSD 10367
Optimisation Modification Mod 1
Figure 3.1

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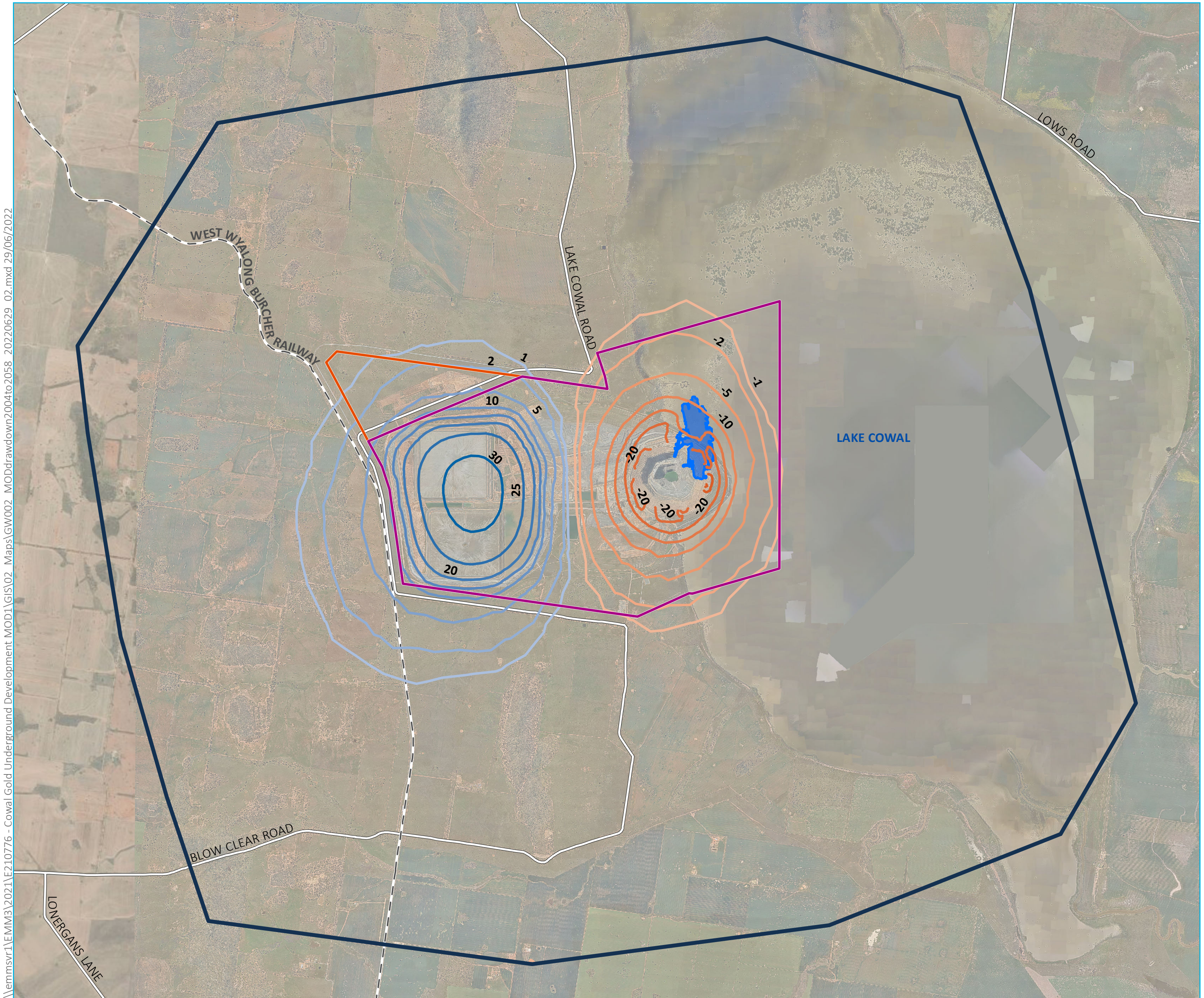

\\emmsvr1\EMM3\2021\E210776 - Cowal Gold Underground Development MOD1\GIS\02 Maps\GW002 MODdrawdown2004to2058 20220629 02.mxd 29/06/2022

KEY

- Proposed underground development (Mod 1)
- Groundwater model boundary
- Mining lease (ML1535)
- Mining lease (ML1791)
- Rail line
- Main road

Predicted watertable drawdown (m)

- -25
- -20
- -15
- -10
- -5
- -2
- -1
- 1
- 2
- 5
- 10
- 15
- 20
- 25
- 30

Predicted watertable change due to proposed modification (2004 - 2058)

Cowal Gold Operations
Underground Development SSD 10367
Optimisation Modification Mod 1
Figure 3.2

Source: EMM (2022); Evolution (2022); Nearmap (2021); DFSI (2017)

0 2.5 5 km
GDA 1994 MGA Zone 55
N

EMM
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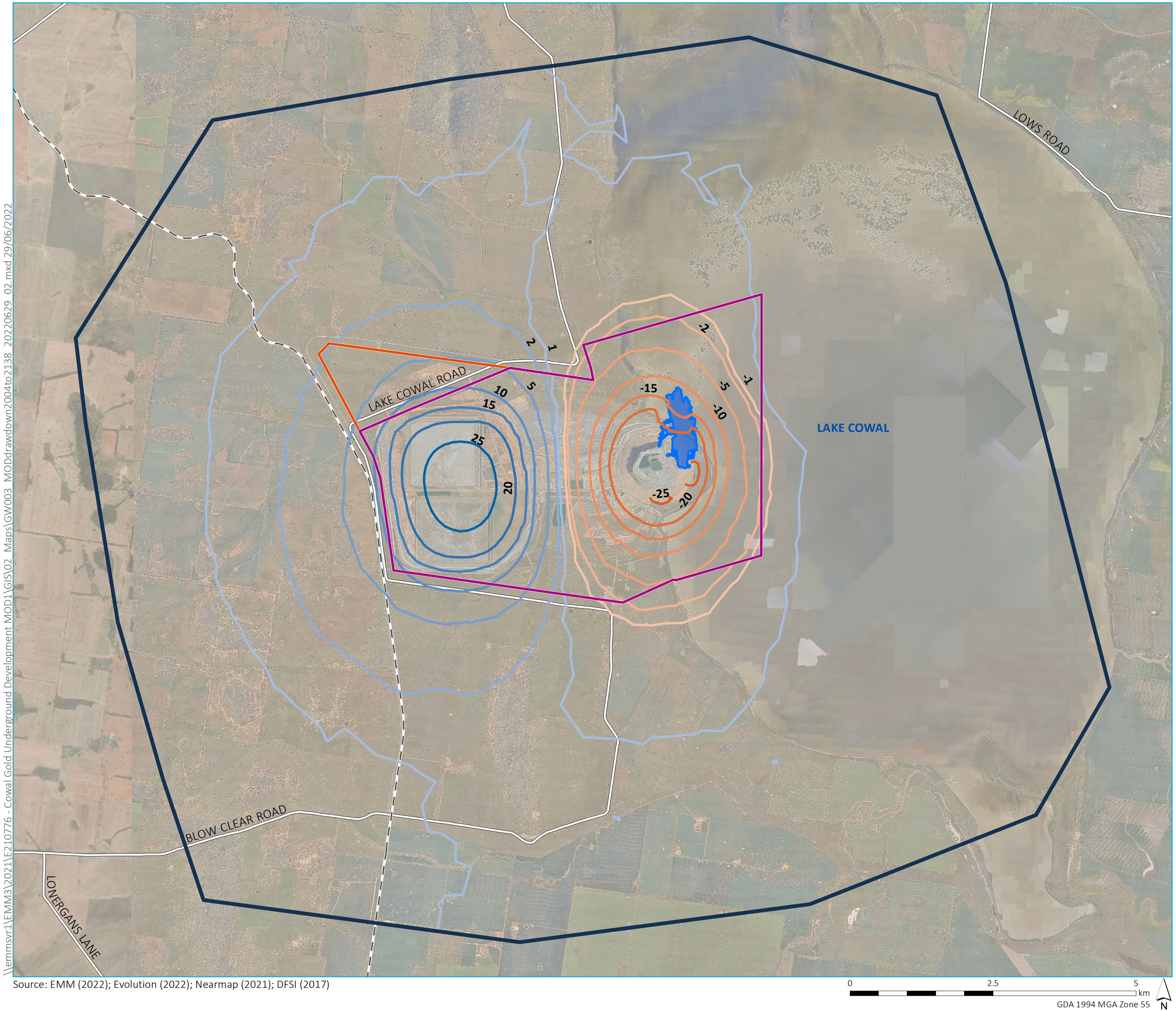

Source: EMM (2022); Evolution (2022); Nearmap (2021); DFSI (2017)

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Predicted watertable change due to proposed modification (2004 - 2138)

Cowal Gold Operations
Underground Development SSD 10367
Optimisation Modification Mod 1
Figure 3.3

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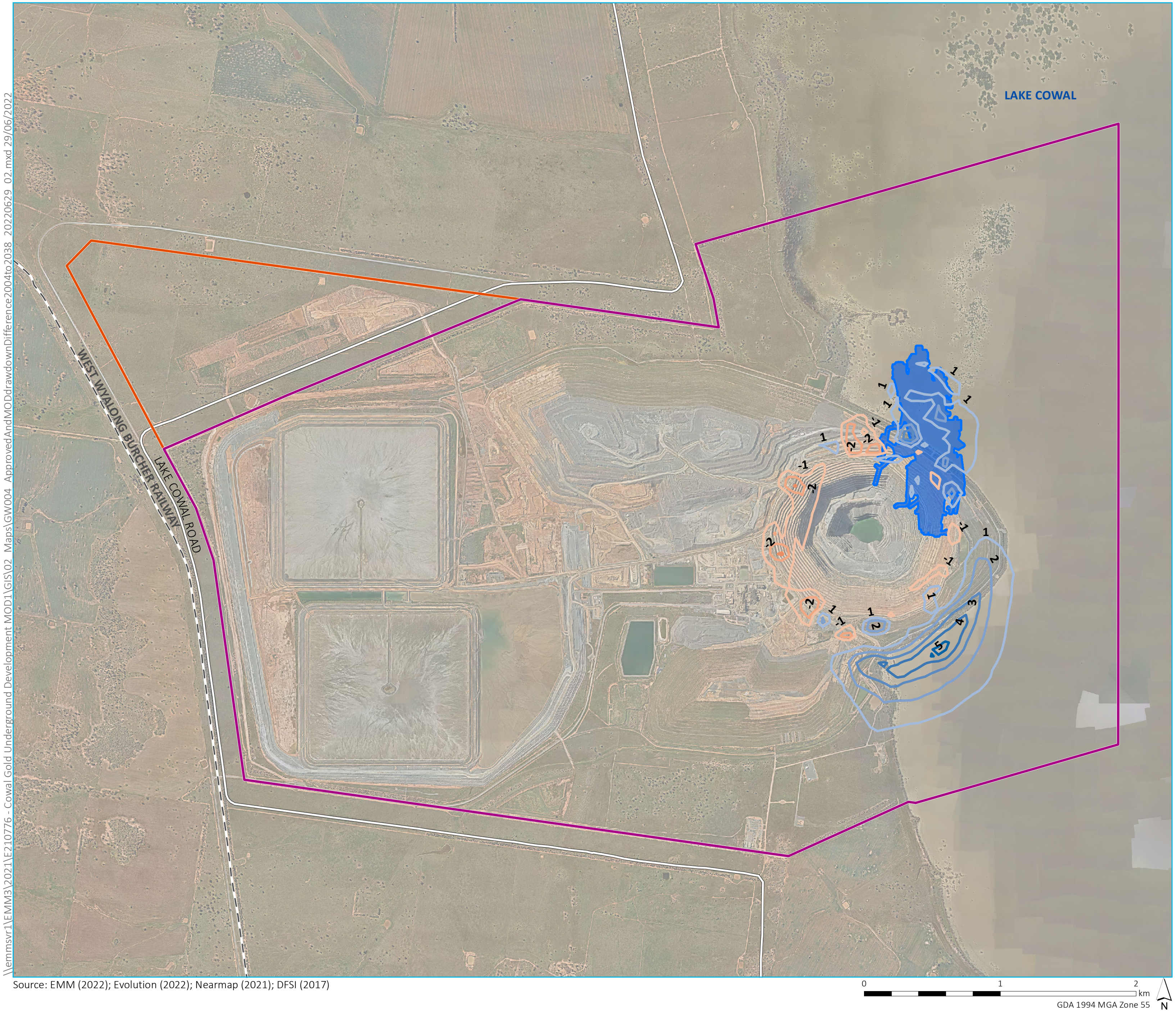

KEY
- Proposed underground development (Mod 1)
- Mining lease (ML1535)
- Mining lease (ML1791)
- Rail line
- Main road

Predicted watertable drawdown difference (m)
- -3
- -2
- -1
- 1
- 2
- 3
- 4
- 5

Source: EMM (2022); Evolution (2022); Nearmap (2021); DFSI (2017)

\\emmsvr1\EMM3\2021\E210776 - Cowal Gold Underground Development MOD1\GIS\02_Maps\GW004_ApprovedAndMODdrawdownDifference2004to2038_20220629_02.mxd 29/06/2022

Predicted watertable change between approved project and proposed modification (2004 - 2038)

Cowal Gold Operations
Underground Development SSD 10367
Optimisation Modification Mod 1
Figure 3.4

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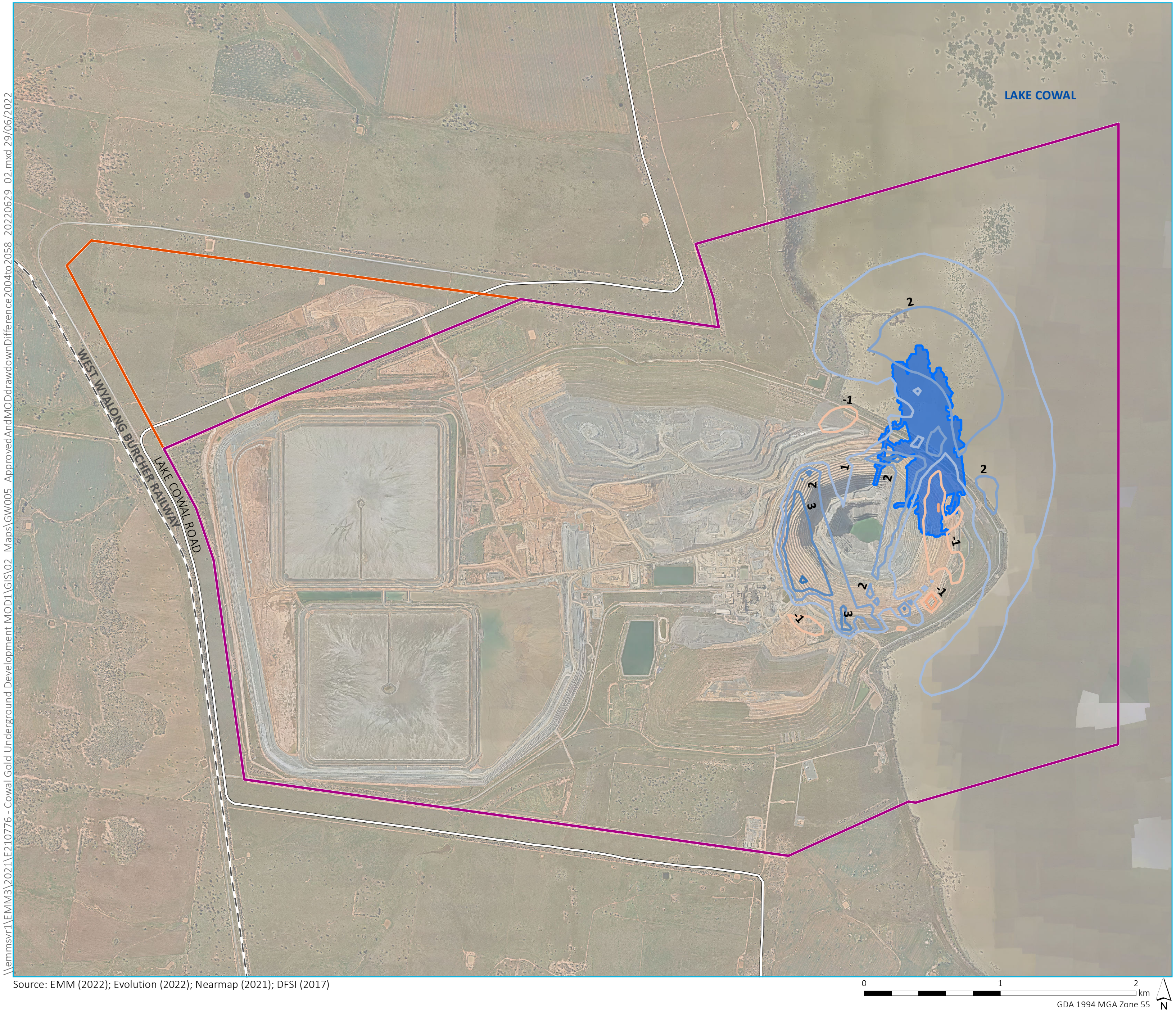

KEY

Proposed underground development (Mod 1)
Mining lease (ML1535)
Mining lease (ML1791)
Rail line
Main road

Predicted watertable drawdown difference (m)
-2
-1
1
2
3
4

Source: EMM (2022); Evolution (2022); Nearmap (2021); DFSI (2017)

\\emmsvr1\EMM3\2021\E210776 - Cowal Gold Underground Development MOD1\GIS\02_Maps\GW005_ApprovedAndMODdrawdownDifference2004to2058_20220629_02.mxd 29/06/2022

Predicted watertable change between approved project and proposed modification (2004 - 2058)

Cowal Gold Operations
Underground Development SSD 10367
Optimisation Modification Mod 1
Figure 3.5

EMM
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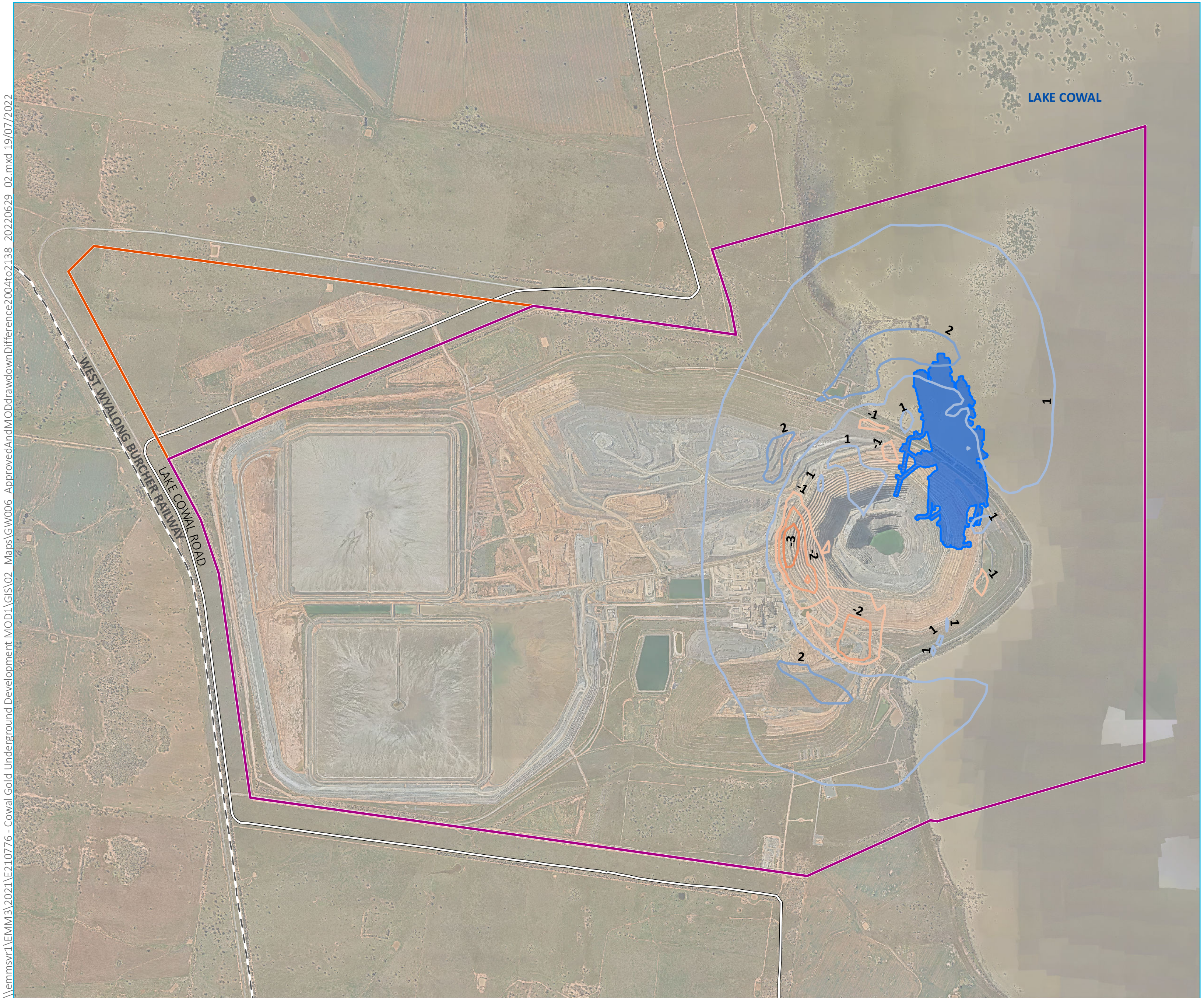

KEY

- Proposed underground development (Mod 1)
- Mining lease (ML1535)
- Mining lease (ML1791)
- Rail line
- Main road

Modelled groundwater table drawdown difference (m)

- -3
- -2
- -1
- 1
- 2

Source: EMM (2022); Evolution (2022); Nearmap (2021); DFSI (2017)

0 1 2 km
GDA 1994 MGA Zone 55
N

Predicted watertable change between approved project and proposed modification (2004 - 2138)

Cowal Gold Operations
Underground Development SSD 10367
Optimisation Modification Mod 1
Figure 3.6

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3.2.2 Groundwater inflows

Combined groundwater inflows into the open-cut pit, proposed stopes and access tunnels have been predicted to increase from 1.1 ML/day in 2020 to a peak of 3.2 ML/day in 2031 (Figure 3.7). This is an increase of 0.4 ML/day on the peak predicted for the approved project (2.8 ML/day). Inflow to the open-cut pit on its own has been predicted to fall from 0.8 ML/day in 2020 to 0.5 ML/day in 2035, which was similar to what was modelled for the approved project.

Following mine closure, groundwater inflow to the open-cut pit is still expected to rise from approximately 0.5 ML/day in the mid-2030s to 0.9 ML/day by the mid-2060s. From the mid-2060s to around 2120, the inflow rate to the open-cut pit is predicted to gradually fall to approximately 0.65 ML/day then remain at around that rate. This gradual decrease is a little slower than previously modelled.

Under the proposed stope backfill plan, stopes will only be open for two months, and are then backfilled and returned to the natural groundwater environment. Stopes were modelled open for one year to be conservative. By the mid-2060s inflow into these areas is still predicted to fall to less than 0.1 ML/day.

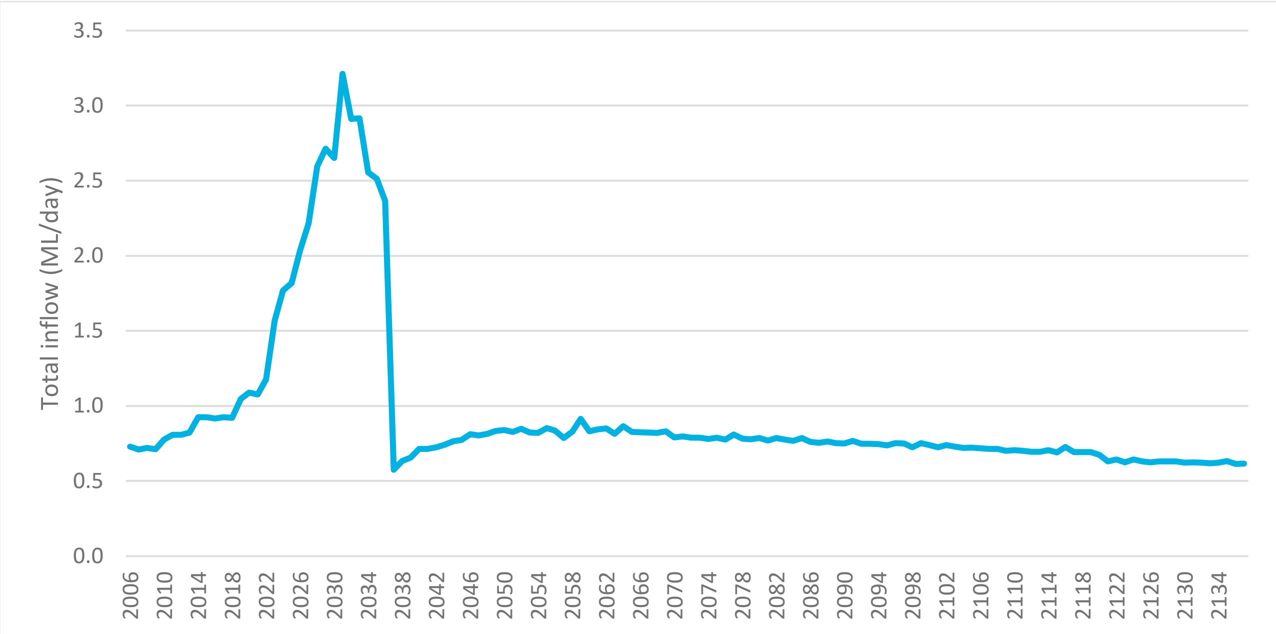


Figure 3.7 Predicted updated groundwater inflow to the open-cut pit, stopes and access tunnels (2006–2138)

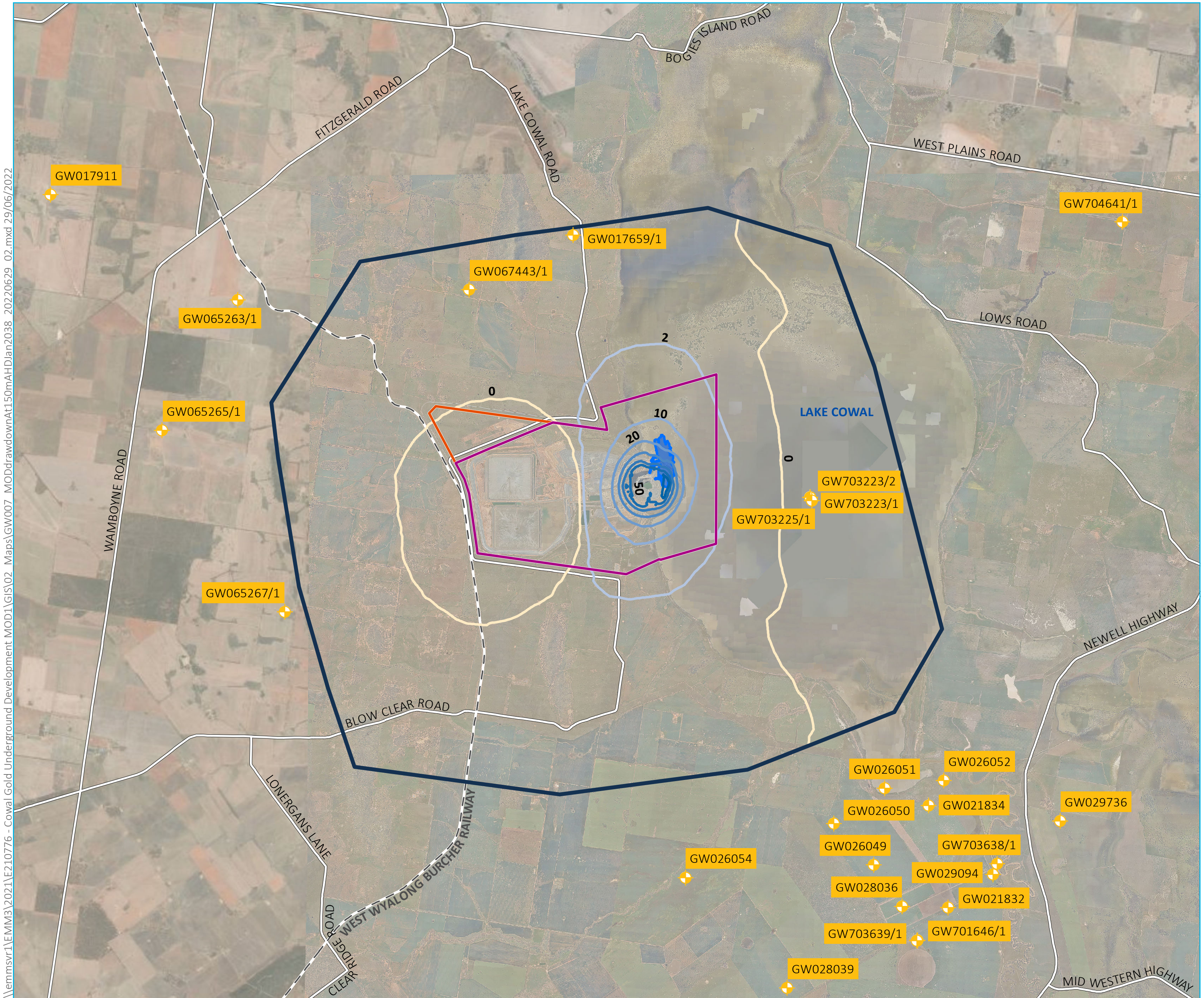
3.2.3 Impacts to Lake Cowal

The average rate of evaporation from the surface of Lake Cowal (534,000 m^3/day) is approximately 165 times the predicted maximum rate of groundwater inflow due to the proposed modification (3,212 m^3/day). As such, the impact of mine groundwater inflow on the water levels of Lake Cowal is still considered to be negligible.

The water in the completed mine workings beneath Lake Cowal and within the open pit void are predicted to remain at similar levels to those predicted for the approved project 200 years after the end of mining. The levels are therefore still below the level of the bed of Lake Cowal and so seepage from the mine to Lake Cowal is considered unlikely. It would appear that by this time the underground mine workings may no longer be acting as a sink and may be returning to a more natural state of groundwater flow.

3.2.4 Impacts to private water supply works

Figure 3.8 shows that for the third-party bores around the mine site with an elevation above 150 m AHD, the groundwater head drawdown from the proposed modification would still be less than two metres in January 2038 when compared to the groundwater head since 2004. The predicted drawdown for the proposed modification shows that the two metre drawdown contour interval (Figure 3.8) to be very similar to the one for the approved project (Figure 2.2), with no third-party bores to be within 2,100 m of the contour. Figure 3.9 shows that for third-party bores around the mine site with an elevation of less than 150 m AHD, the groundwater head drawdown from proposed modification would also still be less than two metres in January 2038. The closest third-party bore to the two-metre drawdown is approximately 4,500 m away from the contour to the south of the proposed project. The two-metre drawdown contour of the proposed modification project is closer to the mine site due to the removal of the boxcut, therefore reducing the potential to have impacts to third-party users.



Source: EMM (2022); Evolution (2022); NGIS (2022); Nearmap (2021); DFSI (2017)

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KEY

- Proposed underground development (Mod 1)
- Groundwater model boundary
- Mining lease (ML1535)
- Mining lease (ML1791)
- Rail line
- Main road
- Bores above 150 m

Modelled groundwater table drawdown (m)

- 0
- 2
- 10
- 20
- 30
- 40
- 50
- 60

Proposed modification drawdown at 150 mAHD, January 2038

Cowal Gold Operations
Underground Development SSD 10367
Optimisation Modification Mod 1
Figure 3.8

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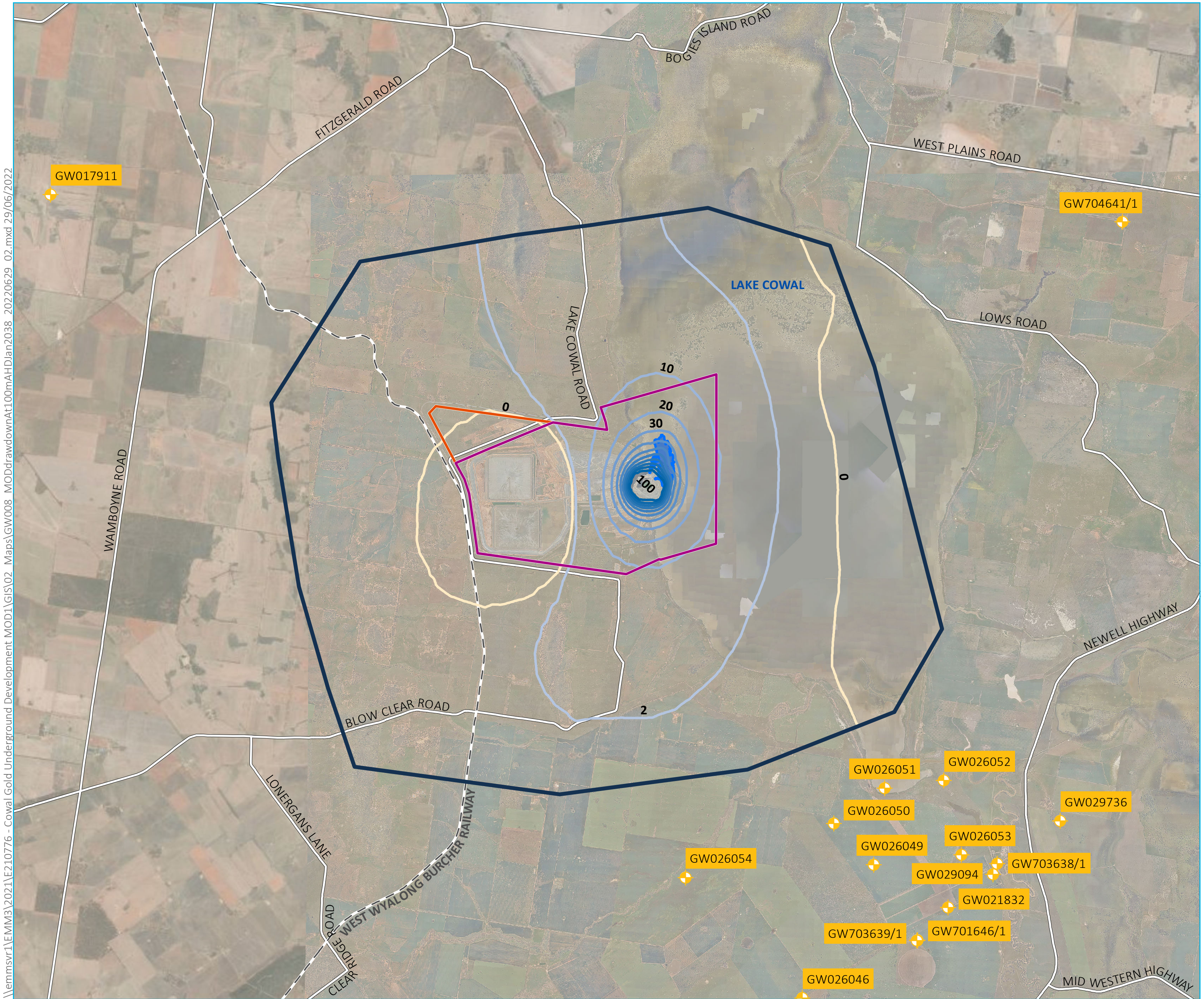

Source: EMM (2022); Evolution (2022); NGIS (2022); Nearmap (2021); DFSI (2017)

KEY
- Proposed underground development (Mod 1)
- Groundwater model boundary
- Mining lease (ML1535)
- Mining lease (ML1791)
- Rail line
- Main road
- Bores above 150 m

Modelled groundwater table drawdown (m)
- 0
- 2
- 10
- 20
- 30
- 40
- 50
- 60
- 70
- 80
- 90
- 100

Proposed modification drawdown at 100 mAHD, January 2038

Cowal Gold Operations
Underground Development SSD 10367
Optimisation Modification Mod 1
Figure 3.9

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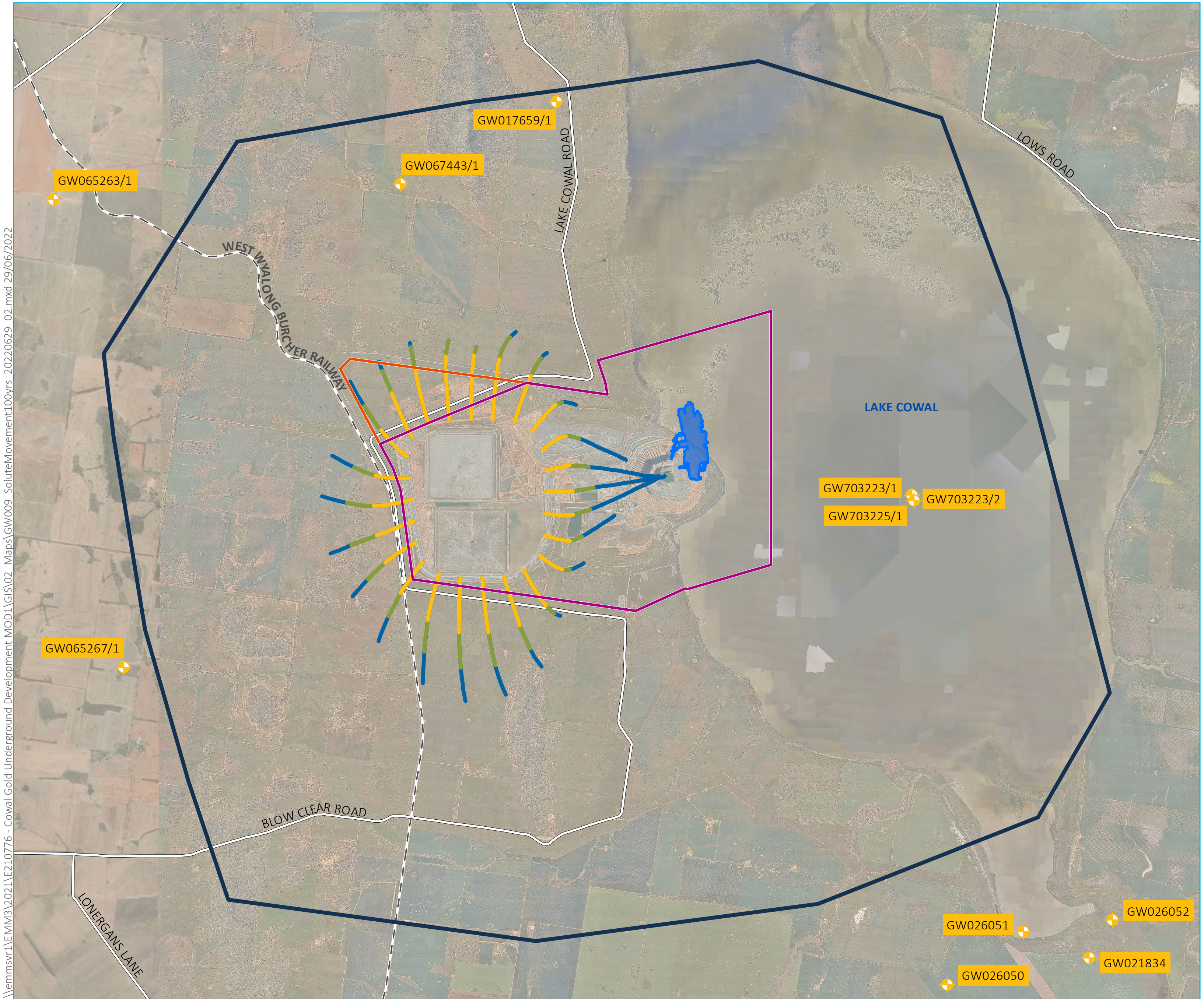
3.2.5 Groundwater quality impacts

An updated assessment of contaminant migration, based on a conservative assessment of the movement of contaminants originating from the IWL, has been undertaken. While the proposed modification does not change the operation of the IWL, the groundwater modelling simulated whether the proposed changes to the underground mine would change the predictions made in the original EIS in relation to the potential long-term groundwater movement from it.

The assessment predicted that after 100 years the potential for groundwater quality changes due to seepage from the IWL stored water may extend up to approximately 2.3km from the IWL walls (Figure 3.10), which is 300 m further than the extent modelled under the approved project. There are still no registered water supply bores within this distance.

Figure 3.11 also shows that there continues to be no evidence of any risk to existing registered groundwater users from TSF/IWL groundwater seepage in the 200 years post-mining.



Predicted extent of solute movement in 100 years - proposed modification

Cowal Gold Operations
Underground Development SSD 10367
Optimisation Modification Mod 1
Figure 3.10

Source: EMM (2022); Evolution (2022); NGIS (2022); Nearmap (2021); DFSI (2017)

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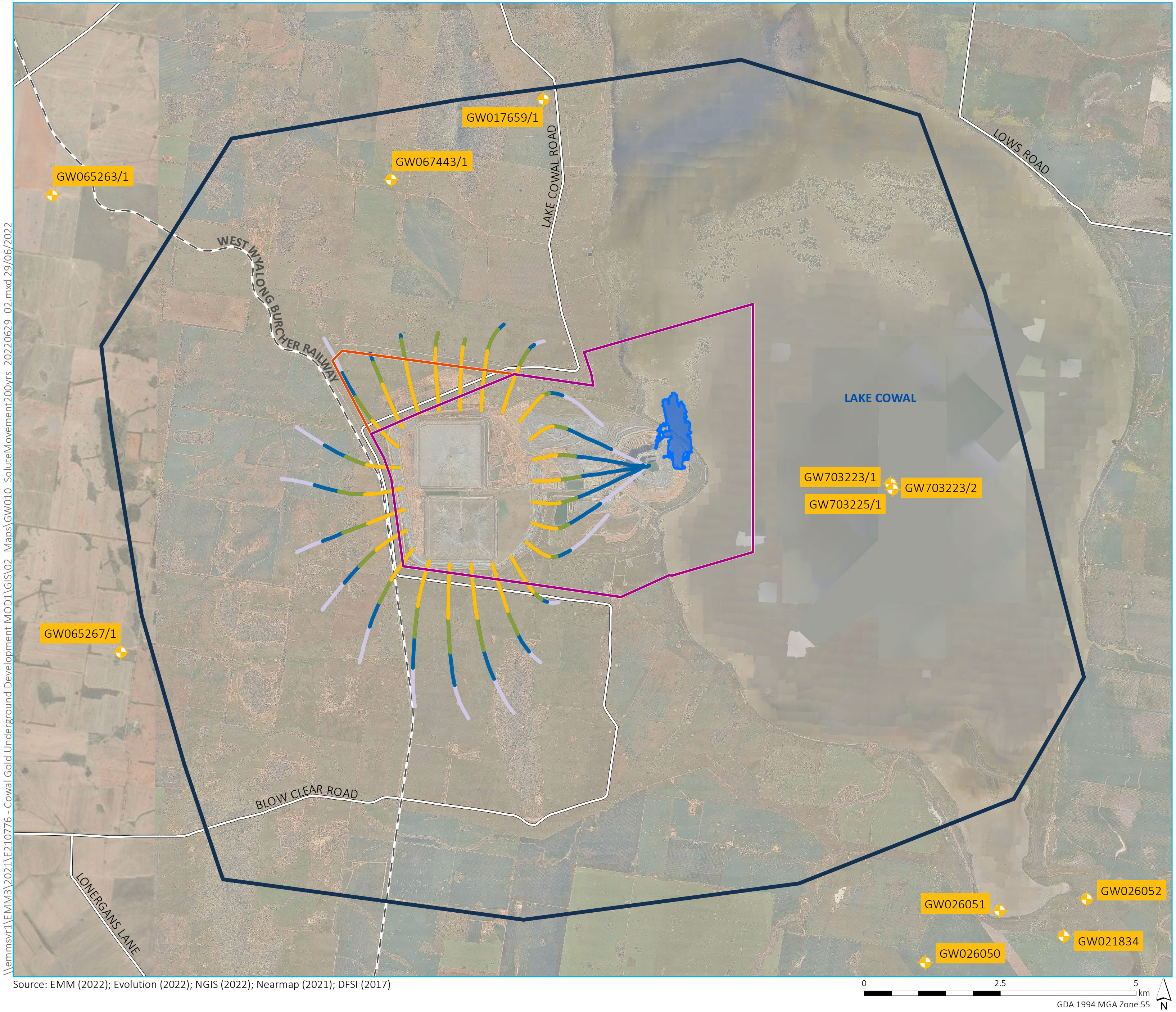

Source: EMM (2022); Evolution (2022); NGIS (2022); Nearmap (2021); DFSI (2017)

KEY
Proposed underground development (Mod 1)
Groundwater model boundary
Mining lease (ML1535)
Mining lease (ML1791)
Rail line
Main road
Bores above 150 m
Predicted solute movement over 200 years
2038 - 2058
2058 - 2088
2088 - 2138
2138 - 2238

Predicted extent of solute movement in 200 years - proposed modification

Cowal Gold Operations
Underground Development SSD 10367
Optimisation Modification Mod 1
Figure 3.11

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3.2.6 NSW Aquifer Interference Policy

An assessment of the proposed modification against the minimal impact considerations of the AIP for a less productive groundwater source is summarised in Table 3.1.

Table 3.1 Assessment of minimal impact considerations detailed in Aquifer Interference Policy

Minimal impact consideration	Assessment provided
(i) No more than a specified cumulative variation in the water table within 40 m from a high priority groundwater dependent ecosystem (GDEs) or a high priority culturally significant site.	The model-predicted groundwater drawdown up to 20 years post-mine closure remains largely within ML1535. As there are no high priority GDEs, priority culturally significant sites or supply works within ML1535 (or within 40 m of the boundary of ML1535), minimal impact considerations (i) to (iii) are still met.
(ii) No more than a specified limit in the water table decline at any water supply work.	
(iii) No more than a specified cumulative pressure head decline at any supply work.	
(iv) Any change in groundwater quality that lowers the beneficial use category of the groundwater source beyond 40 m from the activity.	During the life of the CGO, dewatering from the open pit, stopes and access tunnels will still only have a small and localised (ie within ML1535) impact on groundwater quality. Over the longer term, groundwater will still flow towards the open pit, ultimately terminating there. The groundwater quality in the region surrounding the open pit void is not expected to change significantly due to this process, though the quality of the water within the open pit is still expected to change (eg salinity will increase). The beneficial use of groundwater is not expected to change due to dewatering or the presence of the open pit. Thus, minimal impact consideration is still met.
(v) No increase of more than 1% per activity in long-term average salinity in a highly connected surface water source at the nearest point of activity.	As the equilibrium surface water level in the open pit (the pit lake) following the end of mining will still be well below the ground surface, water from the pit lake will not be released. Thus, it is not classified as a highly connected surface water source, meeting minimal impact consideration.
(vi) No mining activity below the natural ground surface within 200 m laterally from the top of the high bank and 100 m vertically beneath of a highly connected surface water source that is defined as a "reliable water supply".	There are still no known "reliable water supplies" within 200 m laterally from the top of the high bank. Lake Cowal is an ephemeral lake, and so is not considered to be a "reliable water supply". Thus, minimal impact consideration is still met.

i Groundwater licensing requirements

The predicted annual groundwater volumes required to be licensed within each Water Sharing Plan for the proposed modification are summarised in Table 3.2.

Table 3.2 Groundwater licensing requirement summary for the proposed modification

Water sharing plan	Management zone / Groundwater source	Predicted groundwater inflow / extraction volume requiring licensing (ML/year)	
		Approved project (Coffey 2020)	Proposed modification
Lachlan Alluvial Groundwater Sources 2020	Upper Lachlan Alluvial Zone 7 Management Zone	293 (maximum)[1]	285 (maximum)[4]
NSW Murray Darling Basin Fractured Rock Groundwater Sources 2020	Lachlan Fold Belt groundwater source	759 (average)[2] 1,004 (maximum)[3]	802 (average)[5] 1,152 (maximum)[6]

1. From Coffey 2020: Includes 256 ML/year extraction associated with the saline supply bores within ML1535 based on peak usage of 0.7 ML/d, plus 10% of modelled maximum inflow from the Upper Lachlan Alluvial Zone.
2. From Coffey 2020: Modelled average total inflow (796 ML/year) minus average open pit inflow from Upper Lachlan Alluvial Zone (37 ML/year)
3. From Coffey 2020: Modelled maximum total inflow (1022 ML/year in 2031-2039) minus open pit inflow from Upper Lachlan Alluvial Zone (18 ML/year in 2021- 2039)
4. Includes 256 ML/year extraction associated with the saline supply bores within ML1535 based on peak usage of 0.7 ML/d, plus maximum inflow from the Upper Lachlan Alluvial Zone (10% of modelled maximum open pit inflow) (29 ML/year).
5. Modelled average total inflow (823 ML/year) minus average open pit inflow from Upper Lachlan Alluvial Zone (21 ML/year)
6. Modelled maximum total inflow (1,172 ML/year in 2031) minus open pit inflow from Upper Lachlan Alluvial Zone in 2031 (20 ML/year)

Post mining the long-term inflow rate was assessed to be around 243 ML/year from the fractured rock groundwater source and less than 27 ML/year from the Upper Lachlan Alluvial Zone (based on the continued assumption that 90% of groundwater inflow originates from the fractured rock aquifer with the remaining 10% from the overlying Upper Lachlan Alluvium). These volumes will continue to require licensing.

4 Management and mitigation

As stated in the approved project EIS (EMM, 2020), the existing CGO Surface Water, Groundwater, Meteorological and Biological Monitoring Programme (SWGMBMP) will continue to guide the ongoing management of the quality and quantity of surface and groundwater within and around the site.

Additionally, the following management and mitigation measures were developed as part of the approved project and are still of relevance to the proposed modification:

- continuation of monitoring of piezometers in the vicinity of the TSFs;
- installation of new monitoring piezometers to replace those that would be lost during the construction of the IWL;
- review of groundwater levels on an annual basis; and
- develop a groundwater control plan and design control measures to address water level rise at the IWL.

The following monitoring activities, specified in the approved project EIS (EMM, 2020), Coffey (2021a) and development consent (SSD-10367) are also still recommended:

- continued groundwater level monitoring at the existing monitoring piezometers around the CGO site to validate the predictive modelling, particularly in the vicinity of the open-cut pit, TSFs, stopes and access tunnels and ML1535 saline groundwater supply borefield;
- establishment of new monitoring bores to replace those that would be displaced by the IWL;
- install and establish new monitoring bores to comply with Condition B9 of the Development Consent (SSD-10367) specifying that groundwater to be monitored between Lake Cowal and the underground development;
- expansion of the CGO annual groundwater monitoring review to include groundwater level monitoring at fully grouted piezometers;
- use the groundwater model to verify the project inflows every three years as per the conditions of approval;
- the CGO annual groundwater monitoring review should report groundwater inflow volumes into the underground development, according to each underground area of the stopes and access tunnels, in a similar way that open pit dewatering volumes are currently reported; and
- Lake Cowal water levels should be continuously monitored.

No additional groundwater management and mitigation measures are recommended as an outcome of the assessment of the proposed modification.

5 Conclusion

An assessment of potential groundwater impacts from the proposed modification (Mod 1) for the underground project was completed. The assessment showed that the proposed changes to the CGO underground development project will not result in any groundwater impacts and would effectively be the same as those presented in the GIA (Coffey 2020), as well as the supplementary report (Coffey 2021a and Coffey 2021b) for the approved (SSD 10367) underground project.

Groundwater management and mitigation measures currently in place at CGO will continue to be implemented in accordance with DA 14/98, SSD 10367 and the management plans to ensure that potential groundwater impacts from the CGO open cut operations and underground development project are minimised.

References

Barrick 2010, *Saline Groundwater Assessment – Cowra Aquifer*, Cowal Gold Mine, prepared by Barrick Australia Limited for Evolution Mining (Cowal) Pty Ltd.

Coffey 2018, *Cowal Gold Operations Processing Rate Modification (MOD 14), Bland Creek Palaeochannel Borefield and Eastern Saline Borefield Groundwater Assessment*, prepared by Coffey Services Australia Pty Ltd for Evolution Mining (Cowal) Pty Ltd.

Coffey 2020, *Cowal Underground Development EIS – Mine Site Hydrogeological Assessment*, prepared for EMM Consulting Pty Ltd.

Coffey (2021a) CGO Underground Development EIS - Addendum 1 of the hydrogeological assessment, prepared for EMM Pty Ltd.

Coffey (2021b) *CGO Underground Development EIS - Addendum 2 of the hydrogeological assessment*, prepared for EMM Pty Ltd.

EMM 2020, *Cowal Gold Operations Underground Development – Environmental Impact Assessment*, prepared for Evolution Mining (Cowal) Pty Ltd.

Appendix A

Technical memo - Cowal Gold Operations underground update

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Memorandum

29 June 2022

To: Bonnie Coxon
Manager Approvals, Evolution Mining (Cowal) Pty Limited

From: Jeff Whitter

Subject: Cowal Gold Underground Mine Plan Modification groundwater modelling

Dear Bonnie,

1 Introduction

Evolution Mining Pty Ltd (Evolution) has re-designed underground access development geometry for the Cowal Gold Operations (CGO). The method of access to the underground mine and the ore production schedule have also been reviewed. EMM understands the proposed modification involves:

- changing the geometry of the underground decline from the footwall of the open-cut pit to the hanging wall;
- changing the main access to the underground mine from the approved box-cut to an in-pit access; and
- increasing the ore production rate from 1.8 million tonnes per annum (Mtpa) to 2.6 Mtpa.

As these matters were not assessed in the original SSD application, they need to be assessed and approved under a separate modification application.

As part of the environmental impact statement (EIS) and subsequent responses to submissions (RTS) a numerical groundwater model was developed and used to simulate potential groundwater impacts at CGO.

Evolution is proposing an optimised underground mine plan, which has been simulated using the EIS groundwater model to support assessments of the potential changes resulting from the modification.

1.1 Modelling objectives

The objectives of the numerical groundwater modelling are to predict, relative to modelling of the approved mine plan:

- potential changes to mine inflows; and
- differences in drawdown associated with the optimised underground mine plan.

The modelling completed for this project was in accordance with the Australian Groundwater Modelling Guidelines (Barnett et al 2012), and generally followed the approach taken for the previous modelling associated with the mine site modelling at Lake Cowal.

2 Model design

Coffey Services Australia (Coffey) developed a numerical groundwater model using the FEFLOW modelling code (Coffey 2020). Predictions from this model were used to support the groundwater impact assessment component of the EIS (EMM 2020) and RTS (Coffey 2021a, 2021b). The existing model was slightly modified in this project to be suitable for simulating the proposed modification. Some refinement of the model mesh was required to represent locations of the proposed new stopes and access tunnels.

2.1 Software and numerical solution

The updated CG4 model was constructed and run using FEFLOW (v7.403). Numerical solver settings were consistent with the EIS model.

2.2 Domain and spatial discretisation

2.2.1 Model domain

The new CG4 model uses the same model domain as the groundwater model developed for the EIS and RTS.

2.2.2 Mesh

A new mesh was created using an unstructured triangular meshing (Delaunay Triangulation) method, in a manner similar to that used to develop the existing EIS groundwater model mesh. To avoid automated refinement on intersecting or curved input lines or very close points, input points were specified to control the mesh around the open pit, tailings storage facilities (TSFs) and integrated waste landform (IWL) and the proposed stopes and access tunnels.

The points defining the underground mine were generated from a 10 x 10 m grid within the footprint of the access tunnels and stopes. These points were used with the points used in the Coffey EIS model to update the mesh.

The CG4 model consists of 282,294 elements and 149,739 nodes. Figure 2.1 shows a plan view of the mesh.

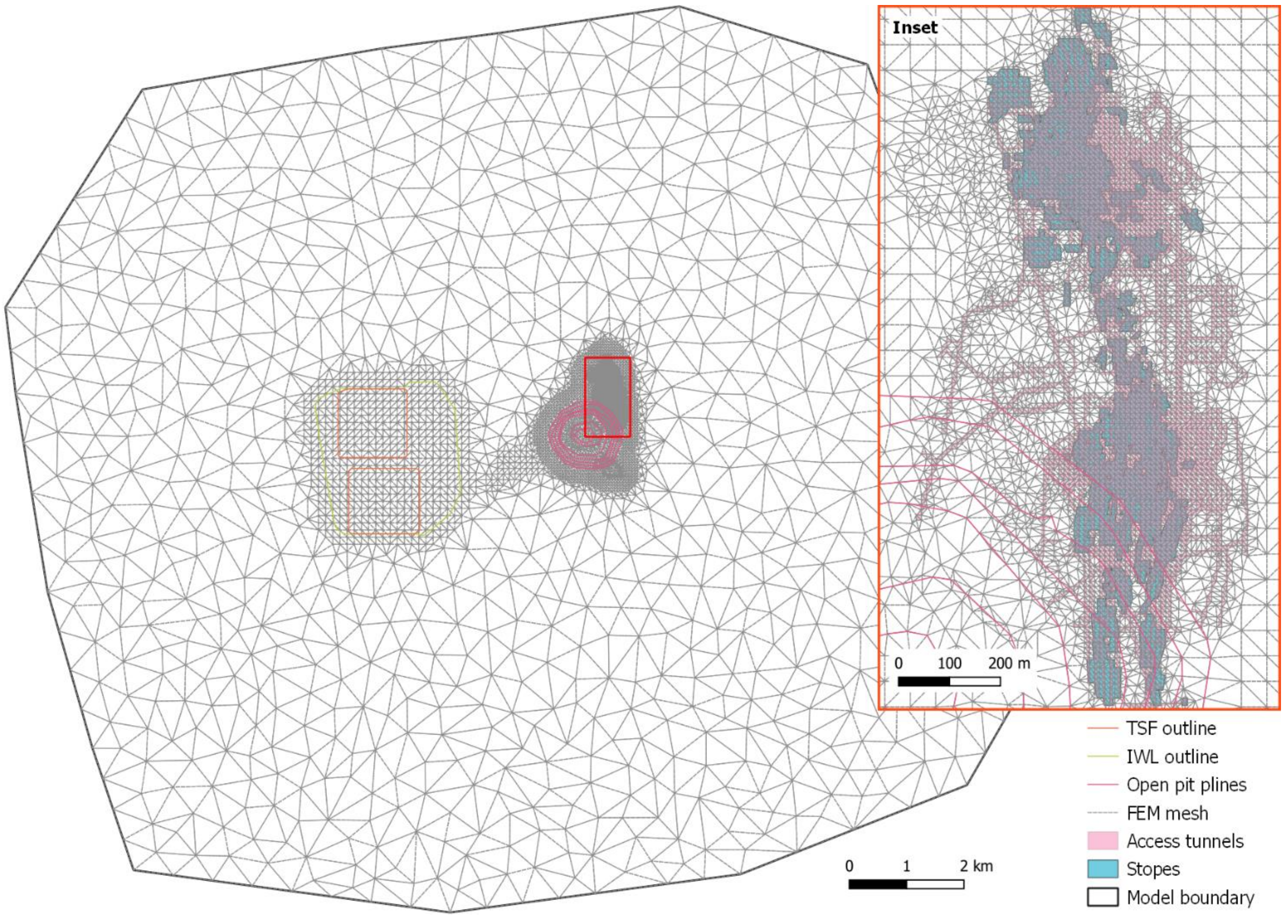

Figure 2.1 CG4 model domain and mesh showing input features used to generate elements in the vicinity of the mine infrastructure

2.2.3 Layers

The EIS and CG4 models are each discretised vertically into 19 layers. The slice elevations in the CG4 model were taken from the existing EIS model. Model layers represent the hydrostratigraphic units (HSUs) as follows:

- Transported unit (variable thickness): Model layers 1 and 2, excluding the areas in layer 1 under the TSFs and IWL;
- Saprolite unit (variable thickness): Model layers 3 and 4;
- Saprock unit (variable thickness): Model layer 5;
- Primary Rock unit (constant thickness): Model layers 6 to 19; and
- TSF and IWL: Model layer 1 under the TSF and IWL footprints only.

Elevations assigned to the tops and bottoms of the model layers are summarised in Table 2.1.

Table 2.1 Model layers and elevations

Layer	Layer top (mAHD)	Layer bottom (mAHD)	HSU
1	203.0 – 345.8	180.7 – 260.4	Transported unit
2	180.7 – 260.4	158.5 – 187.3	Transported unit
3	158.5 – 187.3	146.5 – 180.8	Saprolite unit
4	146.5 – 180.8	130.9 – 176.4	Saprolite unit
5	130.9 – 176.4	105.4 – 148.3	Saprock unit
6	105.4 – 148.3	100	Primary rock unit
7	100	50	Primary rock unit
8	50	0	Primary rock unit
9	0	-100	Primary rock unit
10	-100	-200	Primary rock unit
11	-200	-300	Primary rock unit
12	-300	-400	Primary rock unit
13	-400	-500	Primary rock unit
14	-500	-600	Primary rock unit
15	-600	-700	Primary rock unit
16	-700	-800	Primary rock unit
17	-800	-1000	Primary rock unit
18	-1000	-1250	Primary rock unit
19	-1250	-1600	Primary rock unit

2.3 Temporal discretisation

The simulation period covers the life of mine (LoM) and 200 years post-closure, ranging from 1 January 2004 to 1 January 2238. This period corresponds to the Excel serial days ranging from 37987 d to 123454 d (equivalent to 85467 d in the EIS model).

The relationships between LoM years, corresponding EIS groundwater model times and the current CG4 model times are presented in Table 2.2.

Table 2.2 LoM dates and corresponding model times

LoM Date	EIS model time [d]	CG4 model time [d]
1/1/2004	0	37987
1/1/2005	366	38353
1/1/2006	731	38718

Table 2.2 LoM dates and corresponding model times

LoM Date	EIS model time [d]	CG4 model time [d]
1/1/2007	1096	39083
1/1/2008	1461	39448
1/1/2009	1827	39814
1/1/2010	2192	40179
1/1/2011	2557	40544
1/1/2012	2922	40909
1/1/2013	3288	41275
1/1/2014	3653	41640
1/1/2015	4018	42005
1/1/2016	4383	42370
1/1/2017	4749	42736
1/1/2018	5114	43101
1/1/2019	5479	43466
1/1/2020	5844	43831
1/1/2021	6210	44197
1/1/2022	6575	44562
1/1/2023	6940	44927
1/1/2024	7305	45292
1/1/2025	7671	45658
1/1/2026	8036	46023
1/1/2027	8401	46388
1/1/2028	8766	46753
1/1/2029	9132	47119
1/1/2030	9497	47484
1/1/2031	9862	47849
1/1/2032	10227	48214
1/1/2033	10593	48580
1/1/2034	10958	48945
1/1/2035	11323	49310
1/1/2036	11688	49675
1/1/2037	12054	50041

Table 2.2 LoM dates and corresponding model times

LoM Date	EIS model time [d]	CG4 model time [d]
1/1/2038	12419	50406
1/1/2238	85467	123454

2.4 Boundary Conditions

The boundary conditions (BCs) employed in the CG4 groundwater model are consistent with the EIS model. The main difference is the locations and timing of the seepage face BCs used to represent the updated underground mine access tunnels and stopes.

Constant, fixed head or no flow boundary conditions were applied to all nodes on the edge of the model. These fixed heads were selected based on a steady state calibration against 2004 monitoring data near the open pit, prior to open pit mining. The steady state model was used for the purpose of obtaining starting heads over the model domain for 1 January 2004. Rainfall recharge was applied uniformly to the top layer of the model at a calibrated rate of 6.9×10^{-6} m/d (1.4 mm/yr). This equates to 0.6% of average annual rainfall.

The following fixed heads were applied to the model boundaries:

- 198 mAHD on the eastern edge;
- 205 mAHD along the western edge;
- time-varying fixed head for Lake Cowal;
- no flow along the southern edge; and
- no flow along the northern edge.

3 History-matching

History-matching performance of the existing EIS groundwater model was not reviewed as there were no changes to the applied parameter values.

3.1 Calibrated hydraulic properties

The CG4 base case groundwater model employed the "Set 1" material properties reported in the EIS groundwater modelling report (see Coffey 2020). Table 3.1 presents the adopted base case material properties from the EIS.

Table 3.1 EIS calibrated material properties (after Coffey 2020)

Parameter	Unit	Set 1 parameter value	Unit
Kxx1	Transported	0.021682	m/d
Kxx2	Saprolite	0.011259	m/d
Kxx3	Saprock	0.009151	m/d

Table 3.1 EIS calibrated material properties (after Coffey 2020)

Parameter	Unit	Set 1 parameter value	Unit
Kxx4	Primary rock	0.001009	m/d
Kyy1	Transported	0.021682	m/d
Kyy2	Saprolite	0.022049	m/d
Kyy3	Saprock	0.018101	m/d
Kyy4	Primary rock	0.002109	m/d
Kzz1	Transported	0.000995	m/d
Kzz2	Saprolite	0.000342	m/d
Kzz3	Saprock	0.000915	m/d
Kzz4	Primary rock	0.000101	m/d
Ss1	Transported	0.000477	1/m
Ss2	Saprolite	8.06E-07	1/m
Ss3	Saprock	2.21E-05	1/m
Ss4	Primary rock	1.23E-07	1/m

4 Predictive modelling

Representation of future mining activities included:

- TSF and IWL;
- open pit;
- access tunnels; and
- stopes.

4.1 Access tunnels and stopes

The 3-D geometry of the proposed access and haulage tunnels and stopes were converted to suitable file formats for input into the numerical model.

Development of the access tunnels was supplied at annual steps from 2019 to 2031 and stopes were supplied at annual steps from 2023 to 2035.

4.1.1 Access tunnels

Assignment of the access tunnel nodal selections / seepage face BCs for each slice was undertaken using the tunnel plan footprints generated for the closest slice. The tunnel plan footprint at each slice was generated by projecting the tunnel plan onto the closest slice, effectively clipping the plan at an elevation equal to half the layer thickness above and below the specified slice. An example of the tunnel plan footprint for LoM year 2023 for slice 10 (elevation = -200m) is presented in Figure 4.1.

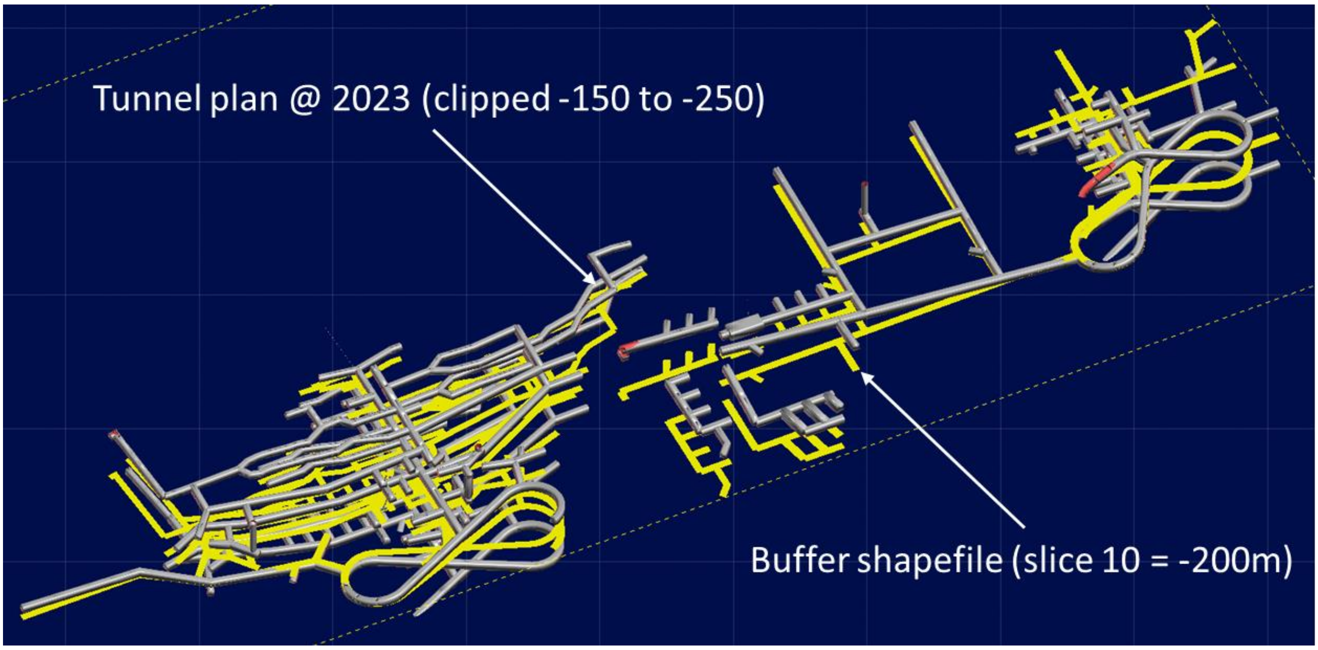

Figure 4.1 Example of shapefile polygon used to select nodes to assign seepage face boundary conditions. Footprint generated from the 3-D access tunnel plan projected on to the closest slice, at 2023 on slice 10

4.1.2 Stopes

Assignment of the stope selections/elemental properties was undertaken using a buffer of the stope plan footprints located within each layer. The stope plan footprint for each layer was generated by projecting the stope plan, clipped at an elevation equal to the top and bottom of each layer. The seepage face BCs representing the stopes were assigned by converting the elemental stope selections into nodal selections and assigning seepage face BCs.

Dewatering of the stopes was represented according to the annual stope schedule provided. Seepage face BCs were turned on around the active stope elements. It was assumed that the stopes were active for a twelve month period and then backfilled using a paste with storage properties consistent with the EIS model. The altered properties representing the stopes persisted for the remainder of the simulation.

4.2 Model verification

Modelled hydraulic head contours from the EIS model and the CG4 groundwater model were compared to verify that the model results are consistent. LoM (2020 and 2038) results are presented as an appendix to this technical memo (Appendix A).

The contours presented in Appendix A indicate that the predicted changes in groundwater levels are consistent between the EIS and CG4 models, with similar changes in heads around mine features that were not modified between the two scenarios simulated with the different model versions (ie TSF and IWL). There are some differences in groundwater levels around the UG mine where the geometry of the access tunnels was modified.

The LoM inflows to the open pit calculated using the EIS and CG4 models are presented in Appendix B. The inflows to the open pit calculated by the updated model are consistent with the EIS model.

4.3 Predictive uncertainty analysis

To facilitate an assessment of uncertainty in groundwater model predictions, a deterministic analysis with subjective probability assessment was performed on the CG4 model. This is consistent with a Type 1 uncertainty analysis as described by Middlemis & Peeters (2018). The model was run with a limited number of different parameters as was completed for the EIS by Coffey in 2020. The EIS and RTS groundwater modelling was presented with an adopted set of calibrated hydraulic parameters (Set 1) and three sets of alternatively calibrated parameters (Sets 2 to 4, see Table 4.1). The same parameter sets were used in the current project to provide a consistent approach to uncertainty analysis when simulating the proposed mine plan modification. For any additional information the reader should refer to the EIS and RTS documentation.

Table 4.1 Alternative sets of calibrated model parameters

Parameter	Unit	Set 1	Set 2	Set 3	Set 4
Kxx1	Transported	0.021682	0.016553	0.014206	0.019974
Kxx2	Saprolite	0.011259	0.005342	0.010743	0.00888
Kxx3	Saprock	0.009151	0.007651	0.015006	0.007038
Kxx4	Primary rock	0.001009	0.001149	0.000378	0.00113
Kyy1	Transported	0.021682	0.016553	0.014206	0.019974
Kyy2	Saprolite	0.022049	0.010461	0.021038	0.017391
Kyy3	Saprock	0.018101	0.015134	0.029682	0.013922
Kyy4	Primary rock	0.002109	0.002403	0.000791	0.002362
Kzz1	Transported	0.000995	0.00115	0.001507	0.000896
Kzz2	Saprolite	0.000342	0.000287	0.000594	0.000278
Kzz3	Saprock	0.000915	0.000765	0.000879	0.000704
Kzz4	Primary rock	0.000101	0.000115	0.000595	0.000113
Ss1	Transported	0.000477	0.000428	0.000387	0.000404
Ss2	Saprolite	8.06E-07	4.64E-07	2.24E-06	6.54E-07
Ss3	Saprock	2.21E-05	1.03E-05	2.21E-05	1.93E-05
Ss4	Primary rock	1.23E-07	4.26E-08	7.29E-08	1.06E-07

5 Predictive modelling results

5.1 Groundwater levels during mining and post-closure

Figure 5.1 to Figure 5.16 provide modelled groundwater head contours at January 2020, January 2038, January 2058 and January 2138. These dates represent approximately the time prior to underground development, just prior to the end of underground mining, and approximately 20 years and 100 years post mining, respectively.

Figure 5.17 provides modelled 5 m drawdown and mounding contours for each of the four parameter sets in the uncertainty analysis described in Section 4.3.

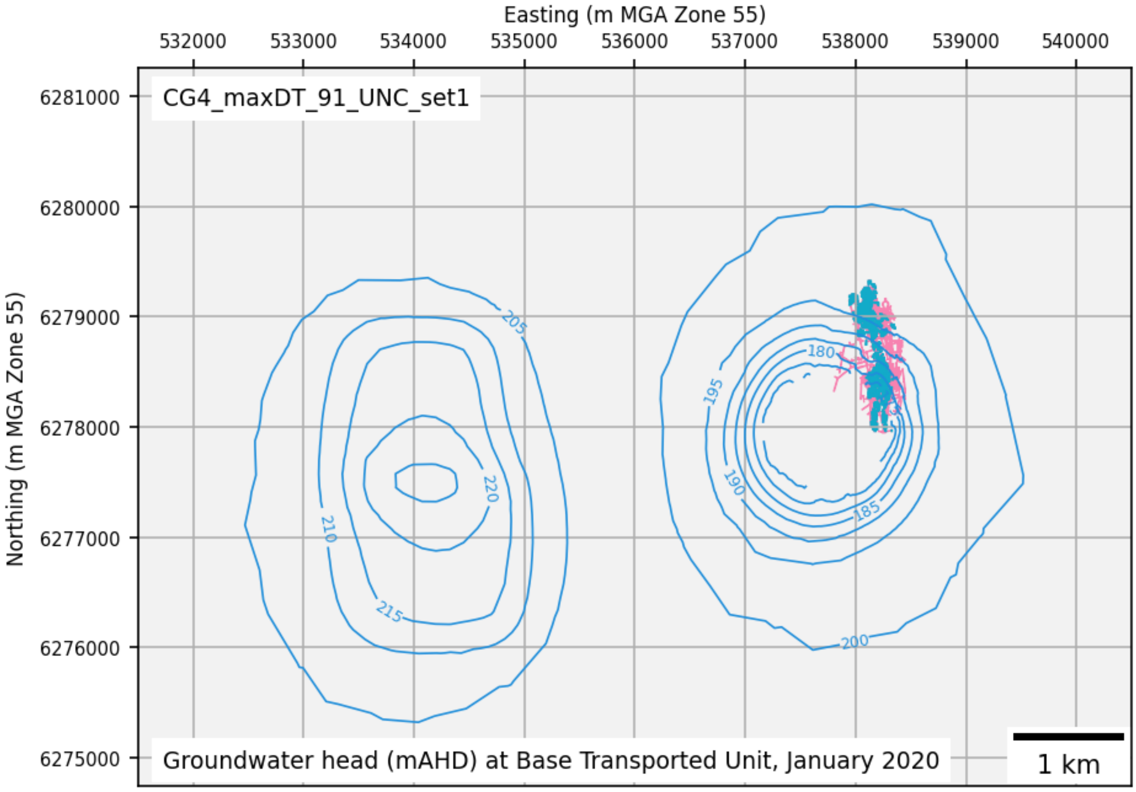

Figure 5.1 Groundwater head (mAHD) at the base of the transported unit for January 2020

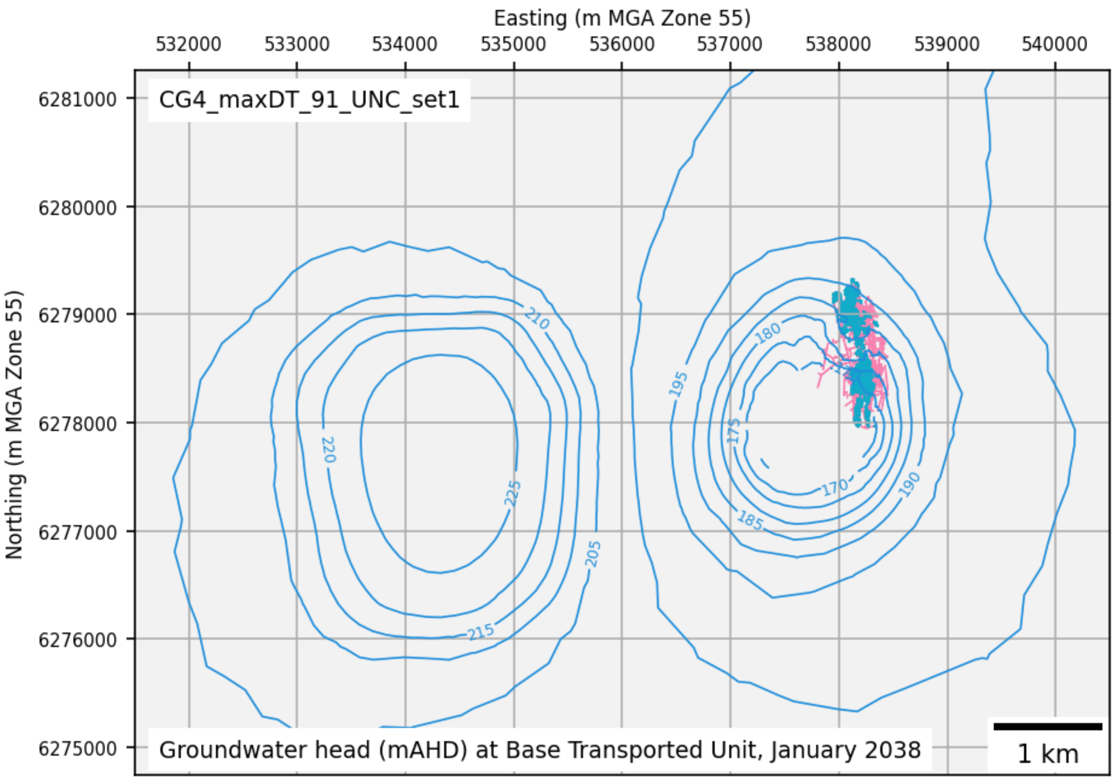

Figure 5.2 Groundwater head (mAHD) at the base of the transported unit for January 2038

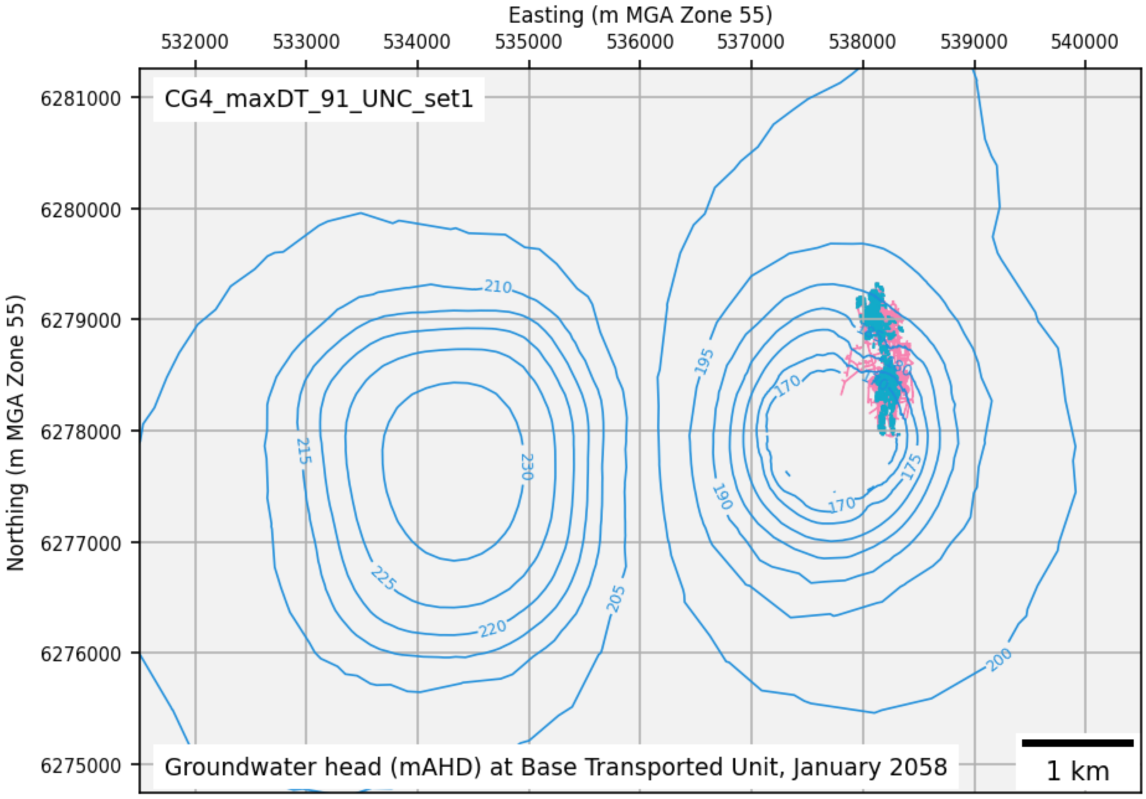

Figure 5.3 **Groundwater head (mAHD) at the base of the transported unit for January 2058**

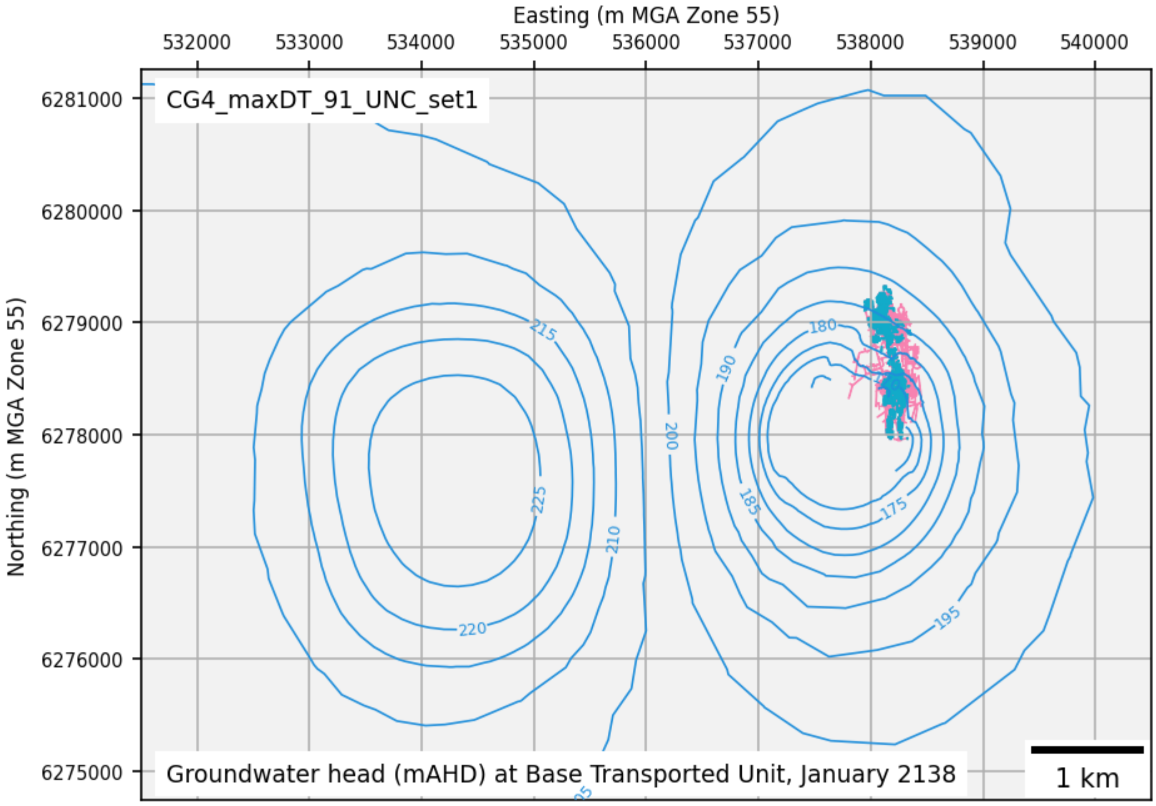

Figure 5.4 **Groundwater head (mAHD) at the base of the transported unit for January 2138**

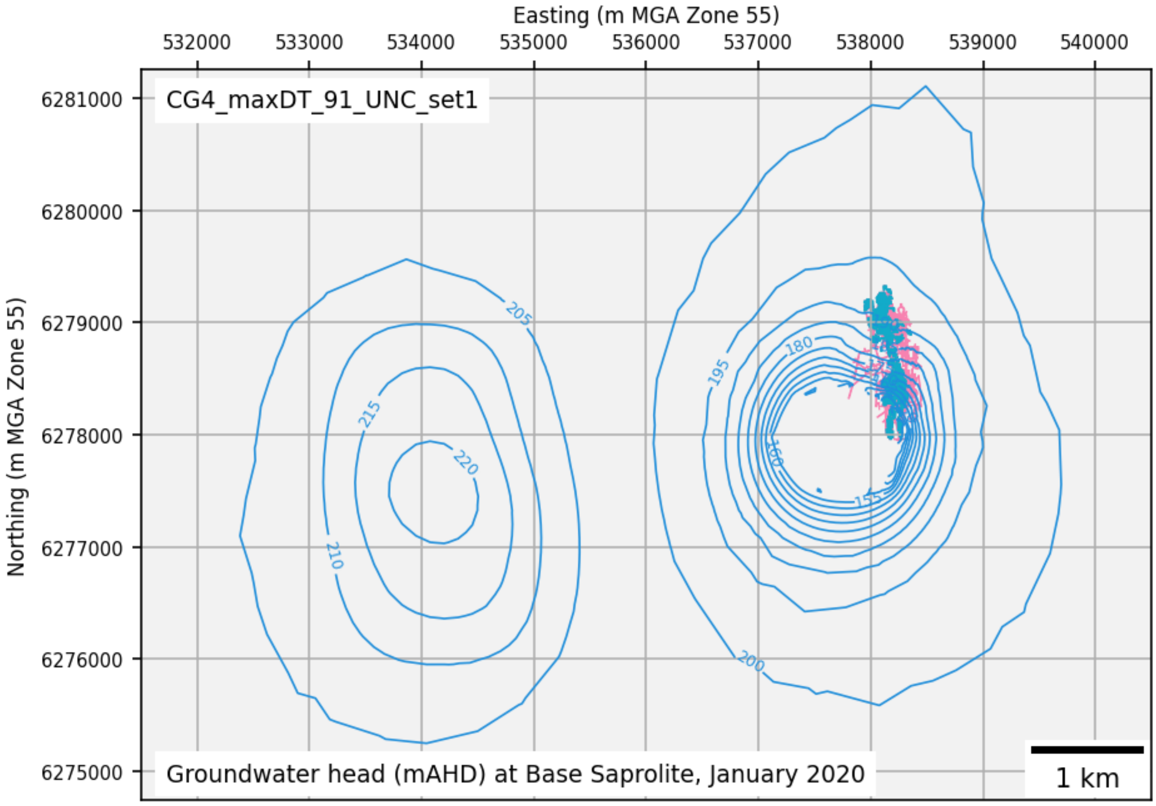

Figure 5.5 Groundwater head (mAHD) at the base of the saprolite unit for January 2020

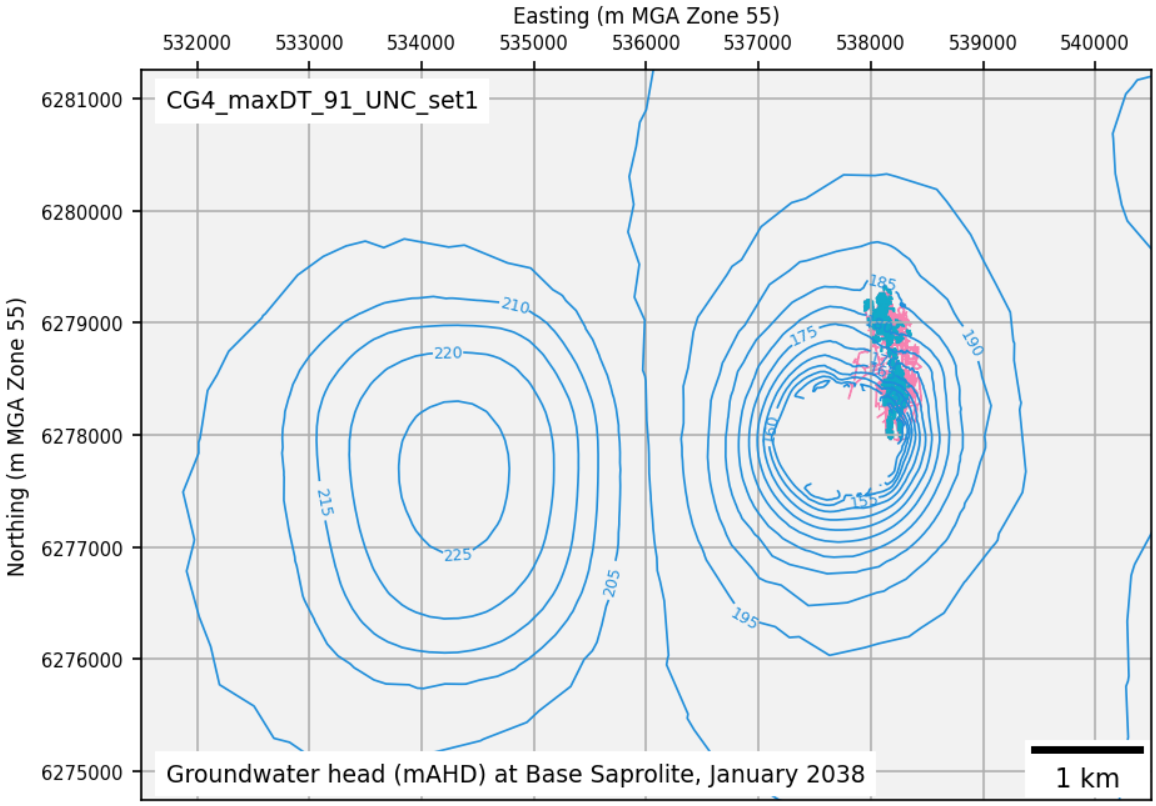

Figure 5.6 Groundwater head (mAHD) at the base of the saprolite unit for January 2038

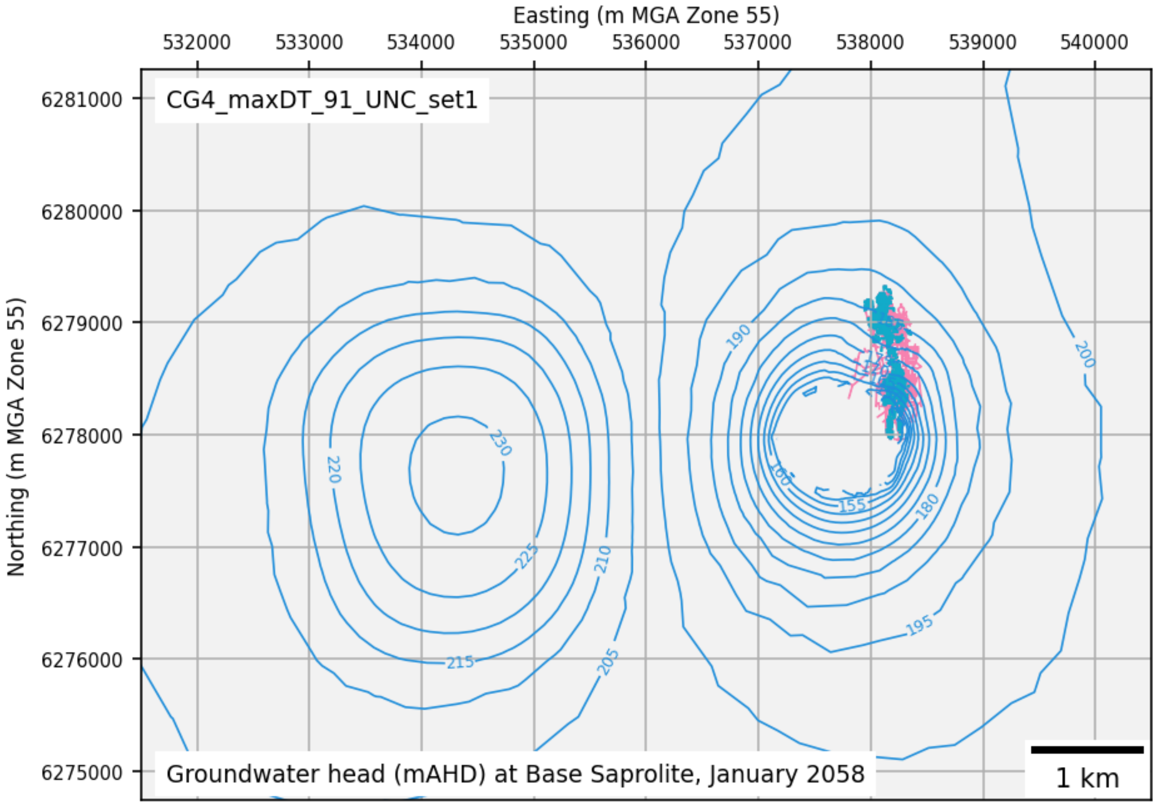

Figure 5.7 Groundwater head (mAHD) at the base of the saprolite unit for January 2058

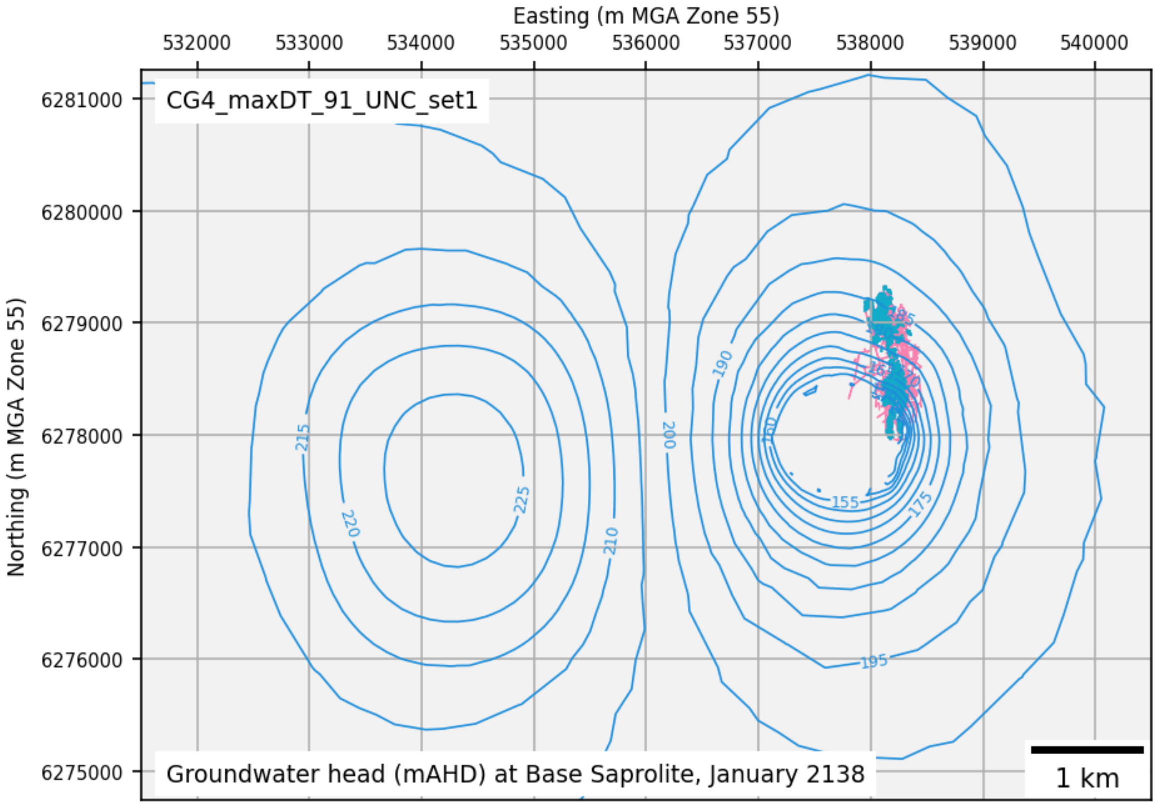

Figure 5.8 Groundwater head (mAHD) at the base of the saprolite unit for January 2138

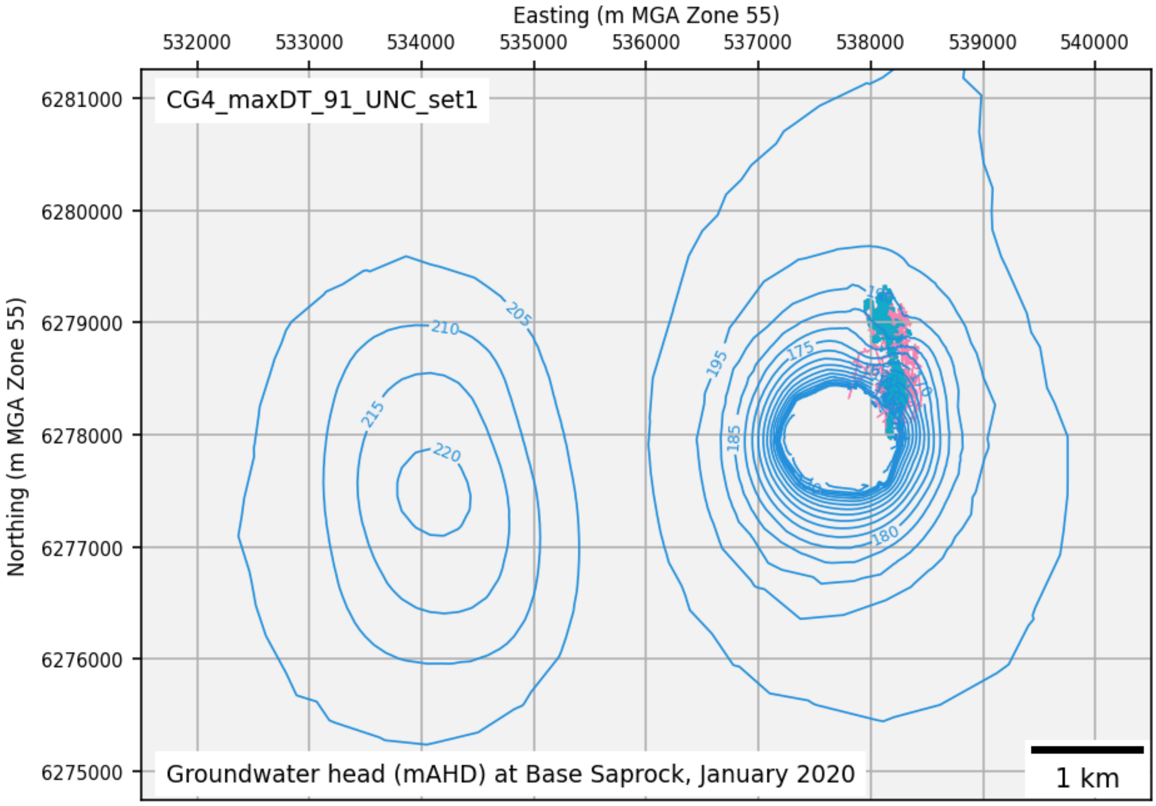

Figure 5.9 Groundwater head (mAHD) at the base of the saprock unit for January 2020

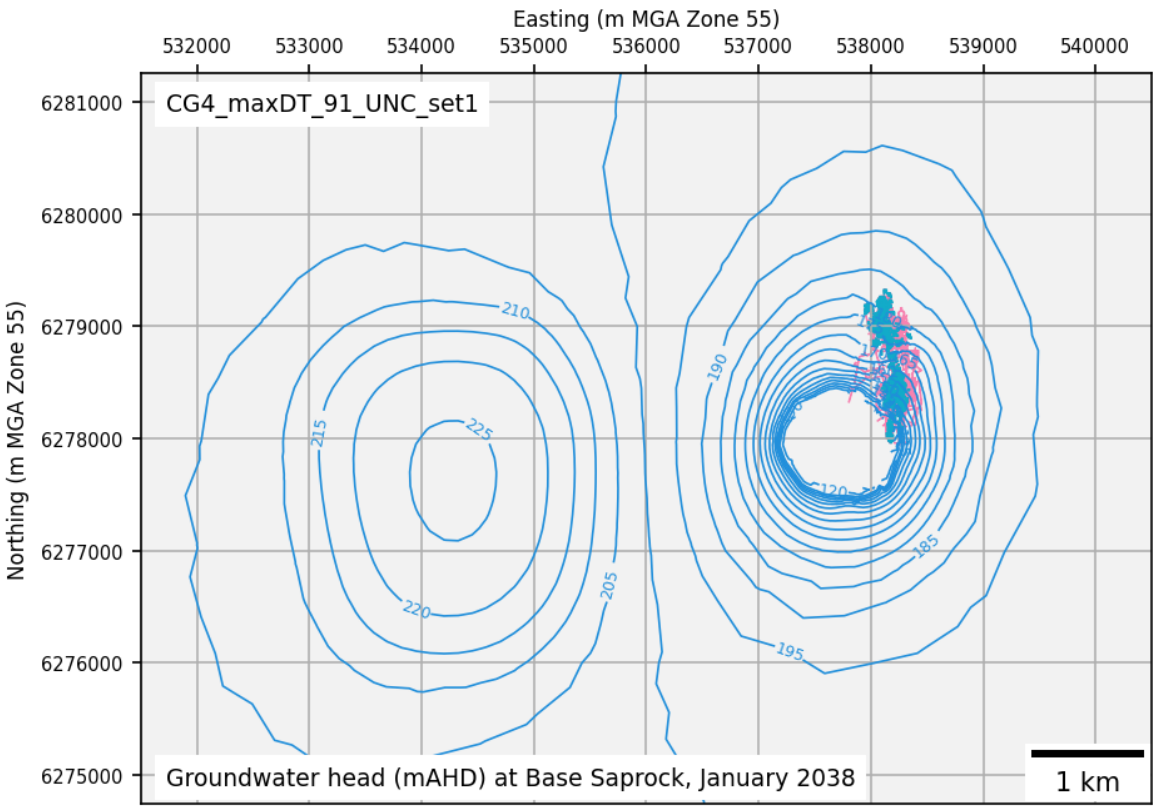

Figure 5.10 Groundwater head (mAHD) at the base of the saprock unit for January 2038

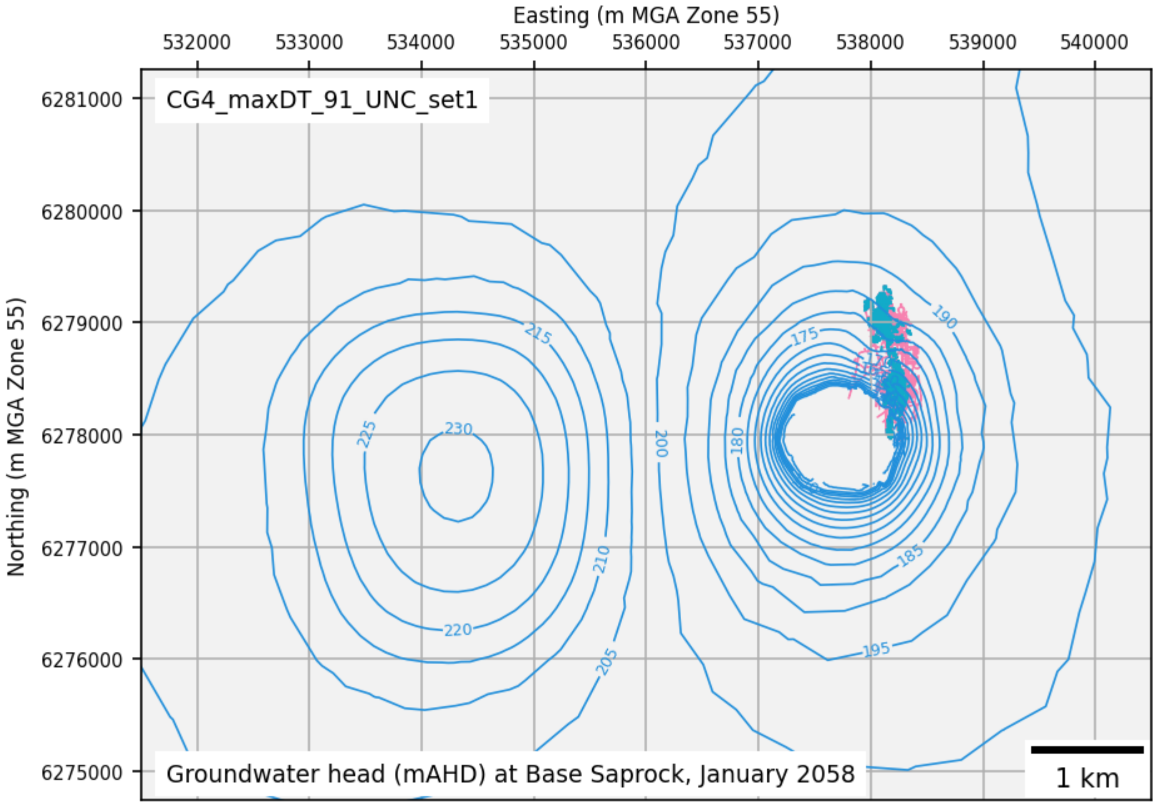

Figure 5.11 Groundwater head (mAHD) at the base of the saprock unit for January 2058

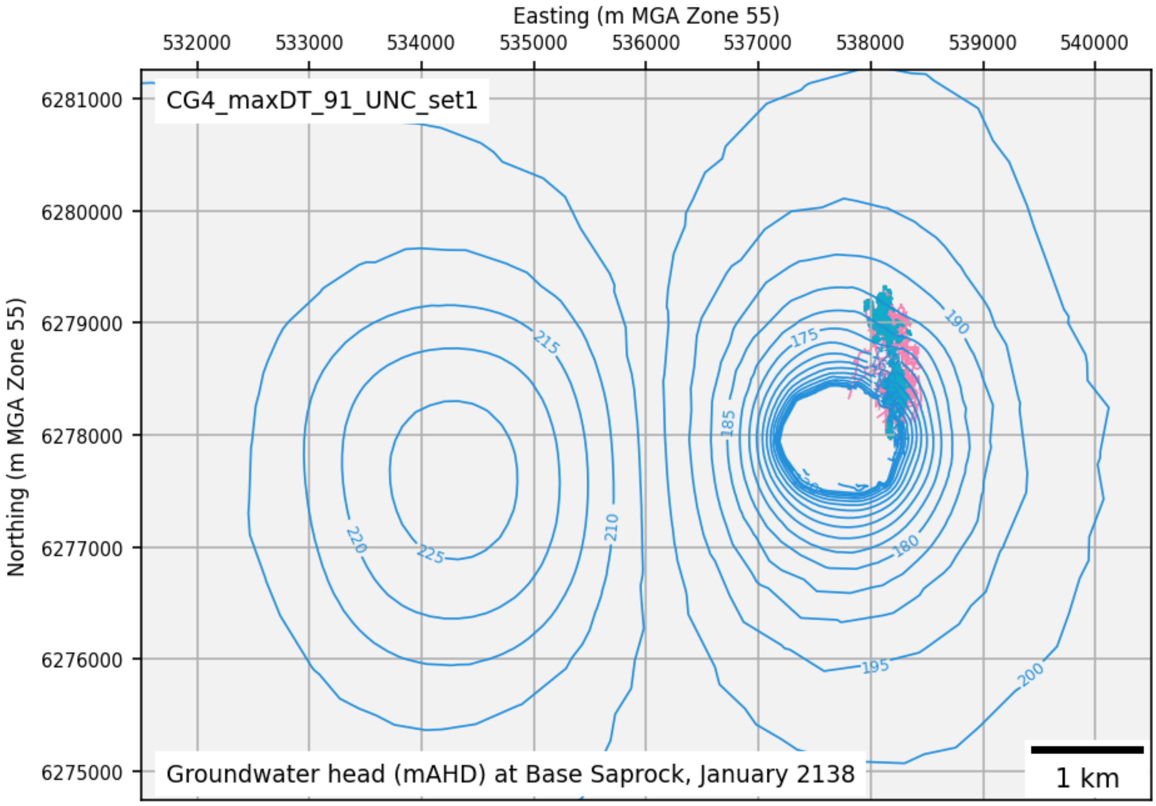

Figure 5.12 Groundwater head (mAHD) at 0 mAHD of the primary rock unit for January 2138

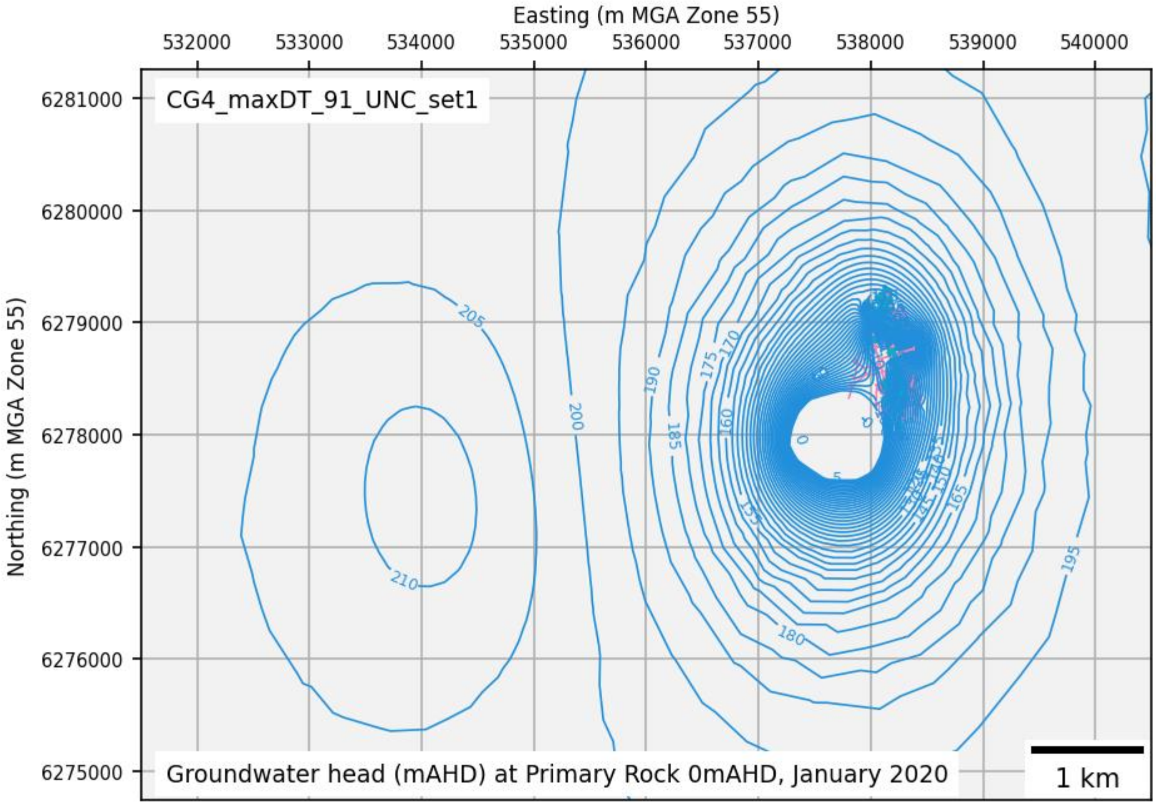

Figure 5.13 **Groundwater head (mAHD) at 0 mAHD of the primary rock unit for January 2020**

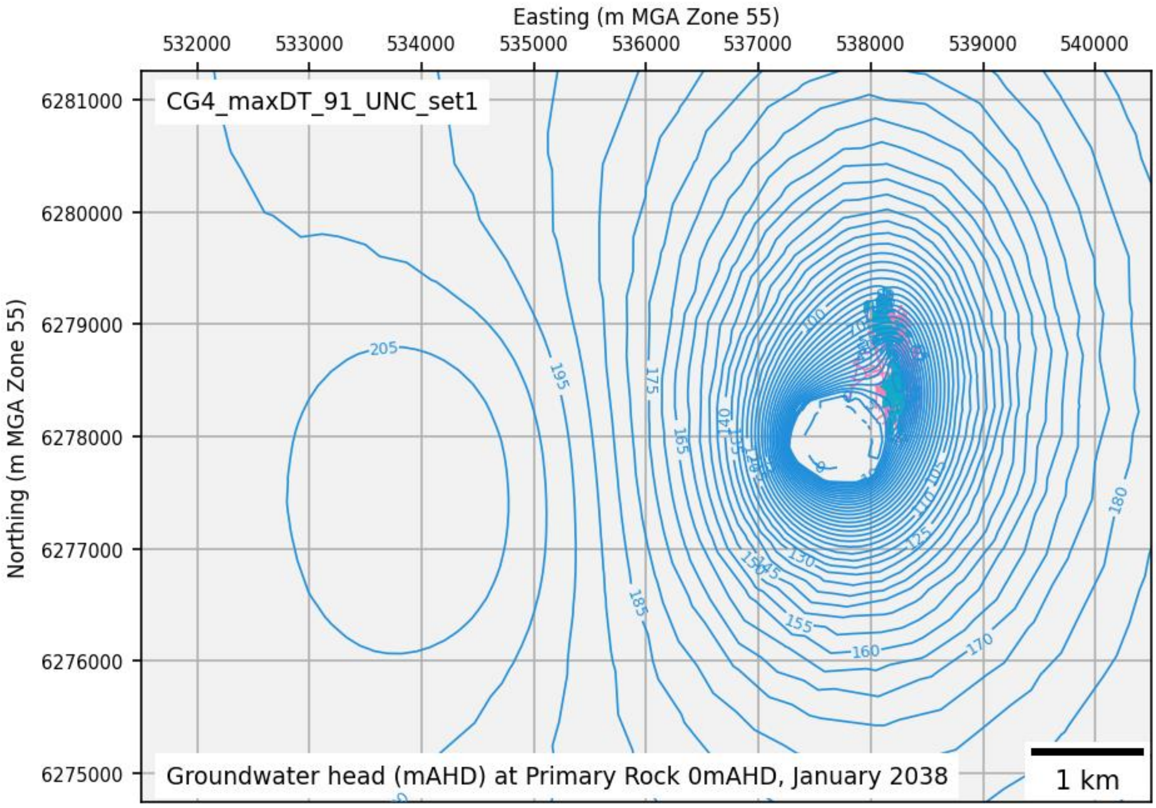

Figure 5.14 **Groundwater head (mAHD) at 0 mAHD of the primary rock unit for January 2038**

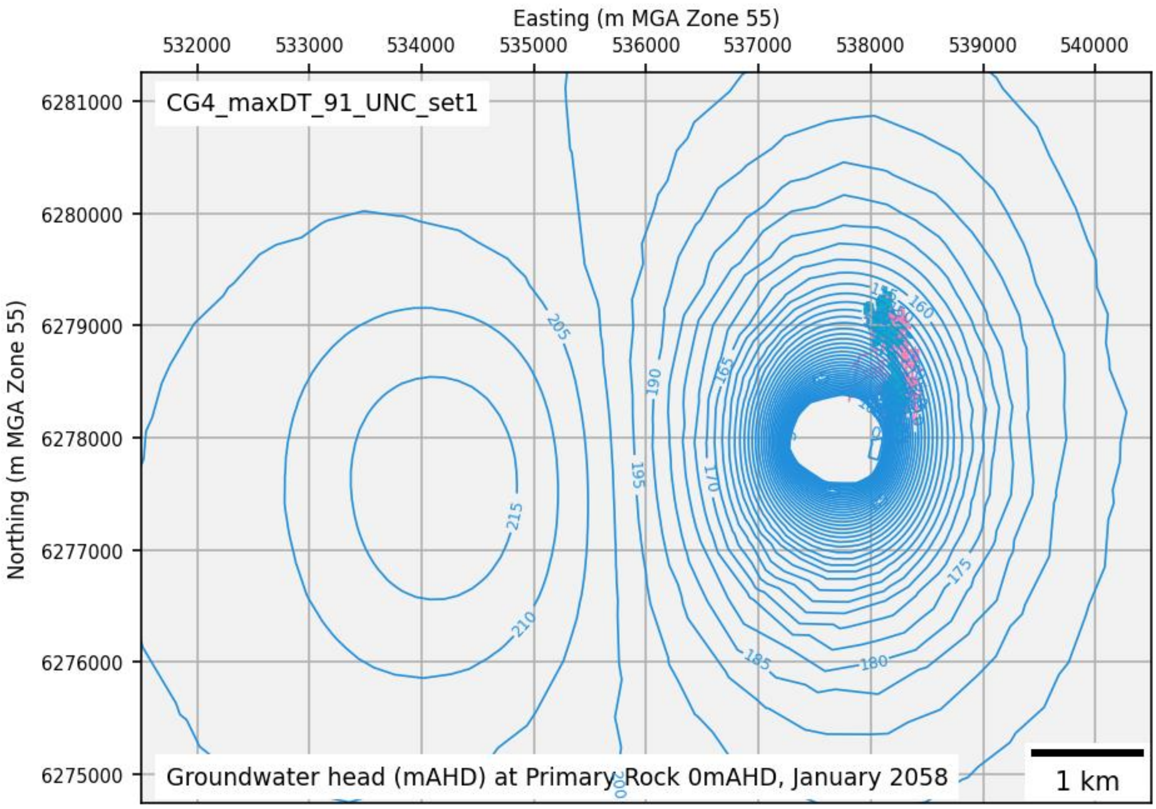

Figure 5.15 Groundwater head (mAHD) at 0 mAHD of the primary rock unit for January 2058

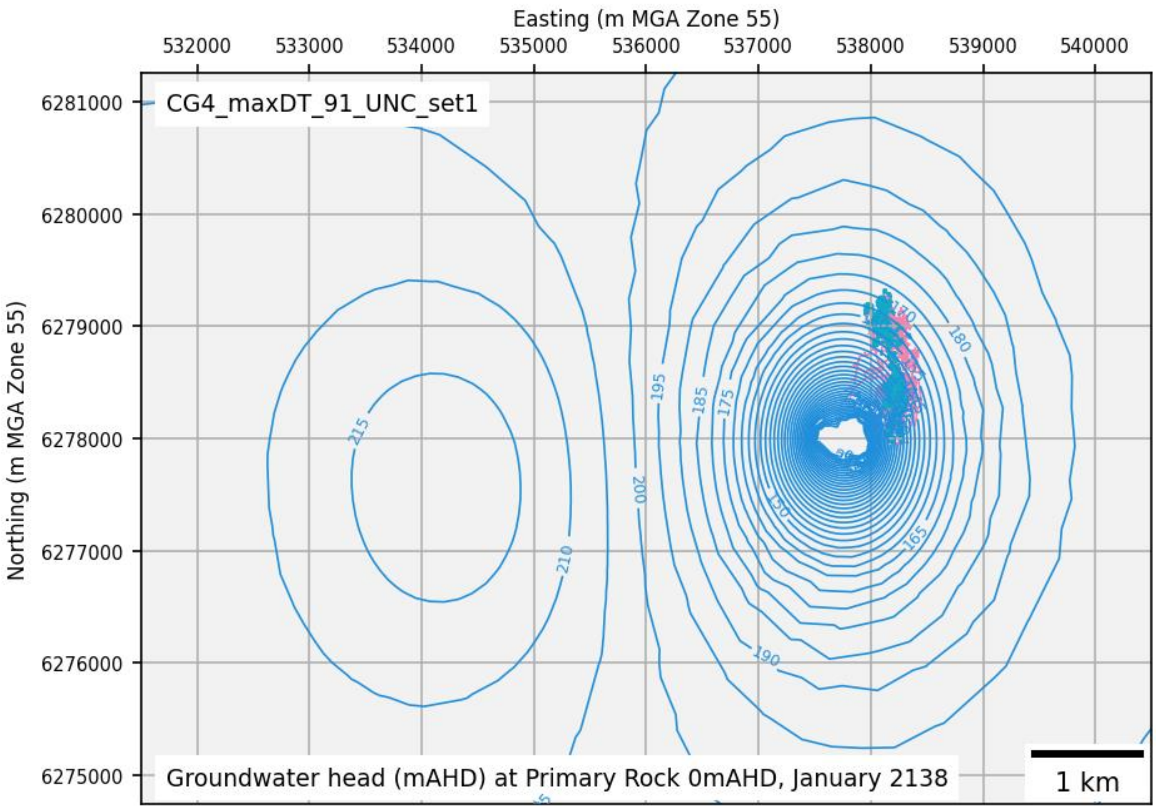

Figure 5.16 Groundwater head (mAHD) at 0 mAHD of the primary rock unit for January 2138

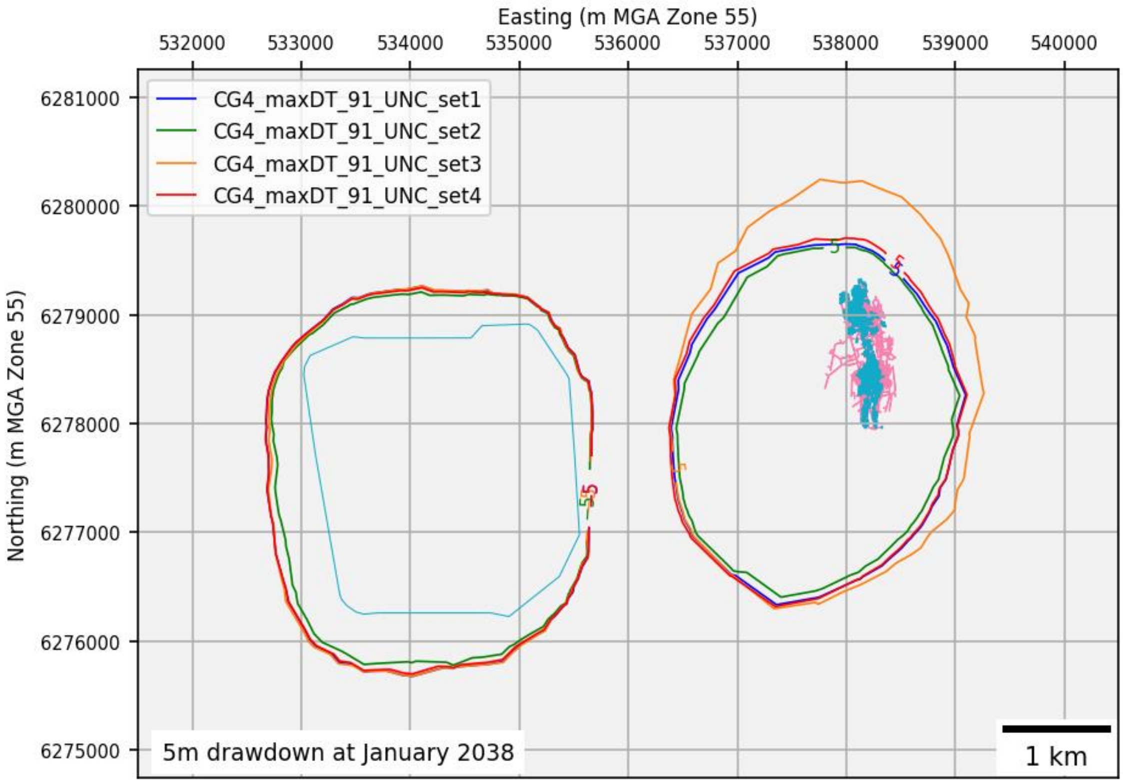

Figure 5.17 Uncertainty in modelled groundwater drawdown at the watertable (slice 2) for January 2038

5.2 Annual groundwater inflows during mining

Predicted annualised average inflows to the open pit, access tunnels, stopes and total mine inflows are presented in Figure 5.18 to Figure 5.21 for the base case and alternative parameter sets. Tabulated annual averaged inflows are presented in Table 4.1. Tabulated mine inflow predictions for parameter sets 2 to 4 are included in Appendix C.

Uncertainty in the observed rate of groundwater inflow to the open pit was incorporated to the inflow results by allowing for a possible range in the observed rate of groundwater inflow to the open pit of between 750 m^3/d and 1,100 m^3/d for the period January 2018 to January 2020.

To account for the uncertainty in the observed rate of groundwater inflow to the open pit, the maximum and minimum of the four predicted inflows were factored up or down. The scale factor was assessed based on the adopted rate of groundwater inflow to the open pit of 950 m^3/d on 1 January 2020 which was used for model calibration. At each time point in Figure 5.21, the minimum of the four predicted inflows was scaled by 0.79 and the maximum by 1.16 to obtain the minimum and maximum inflows shown in the figure, consistent with the EIS and RTS approach.

Despite the changes to the model, the results from the updated model (CG4) are consistent with the EIS model, with similar changes to the groundwater inflows to the mine features that did not change in the model (ie open pit).

The updated model predicts inflows to the access tunnels and stopes are the same order as those predicted by the EIS model, with different timings related to the altered scheduling of the UG mine plan.

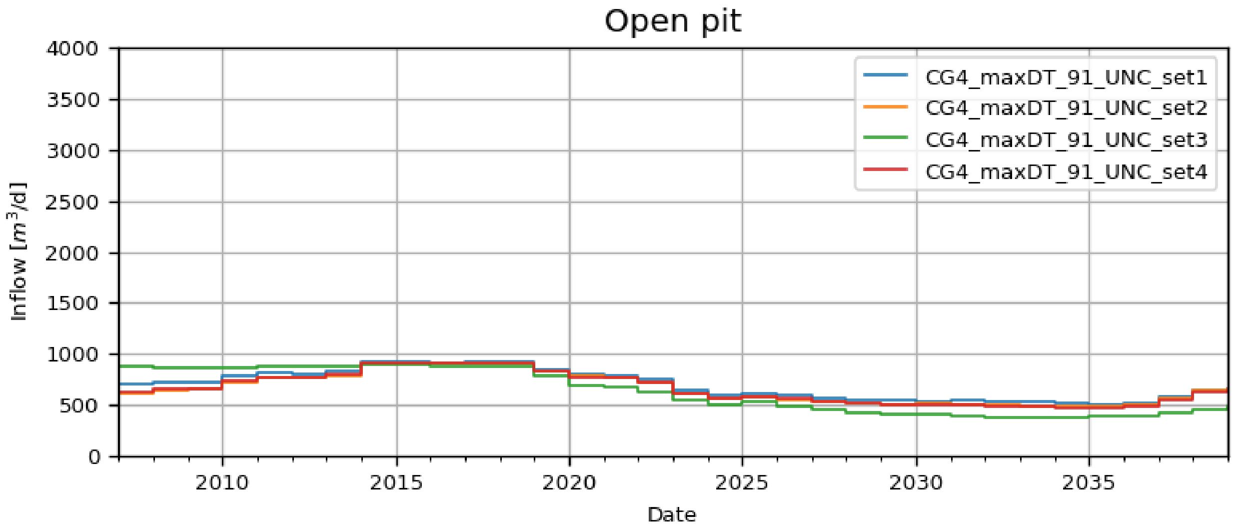

Figure 5.18 Predicted annual groundwater inflow to open pit (2007 to 2039)

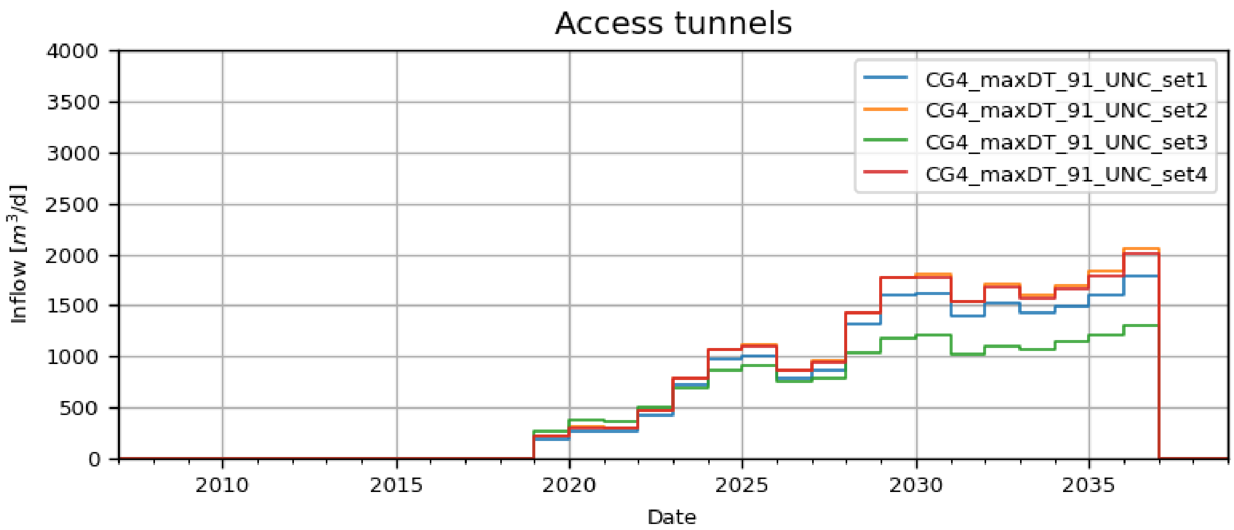

Figure 5.19 Predicted groundwater inflow to access tunnels (2007 to 2039)

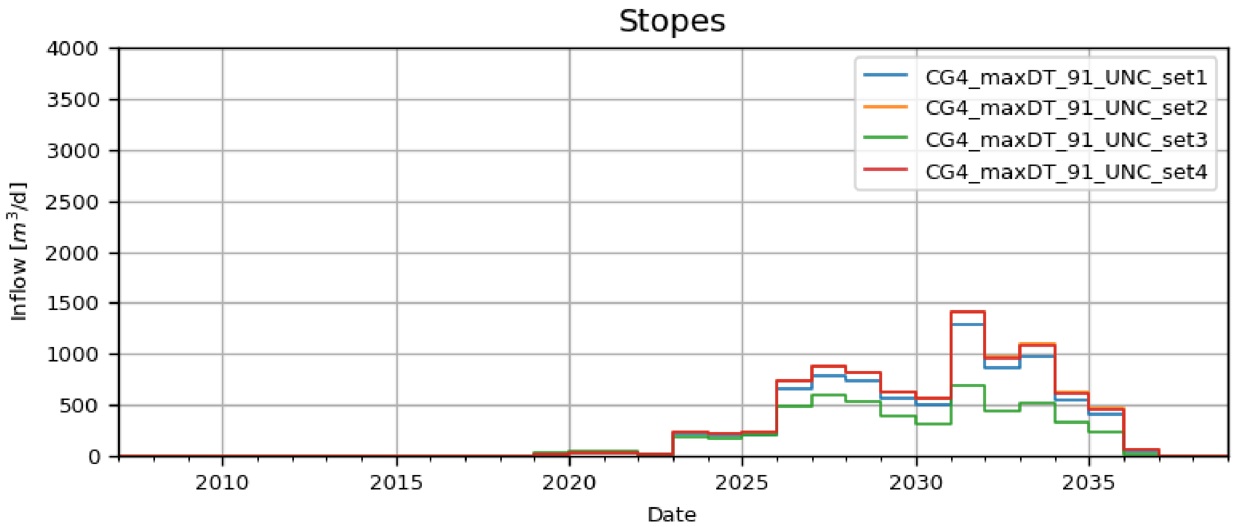

Figure 5.20 Predicted groundwater inflow to stopes (2007 to 2039)

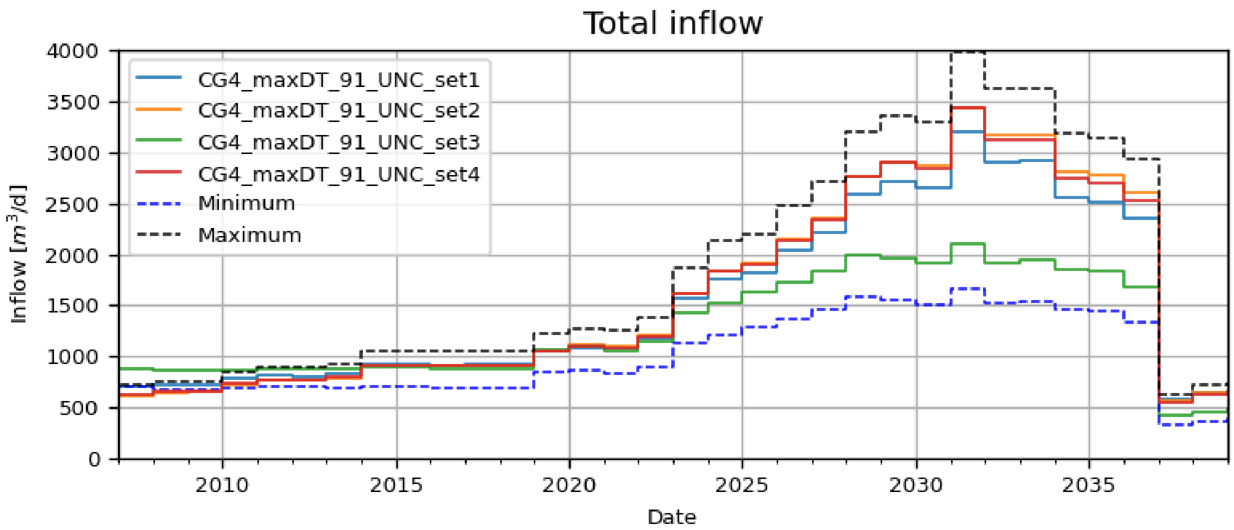

Figure 5.21 Predicted annual groundwater inflow allowing for observational uncertainty (2007 to 2039)

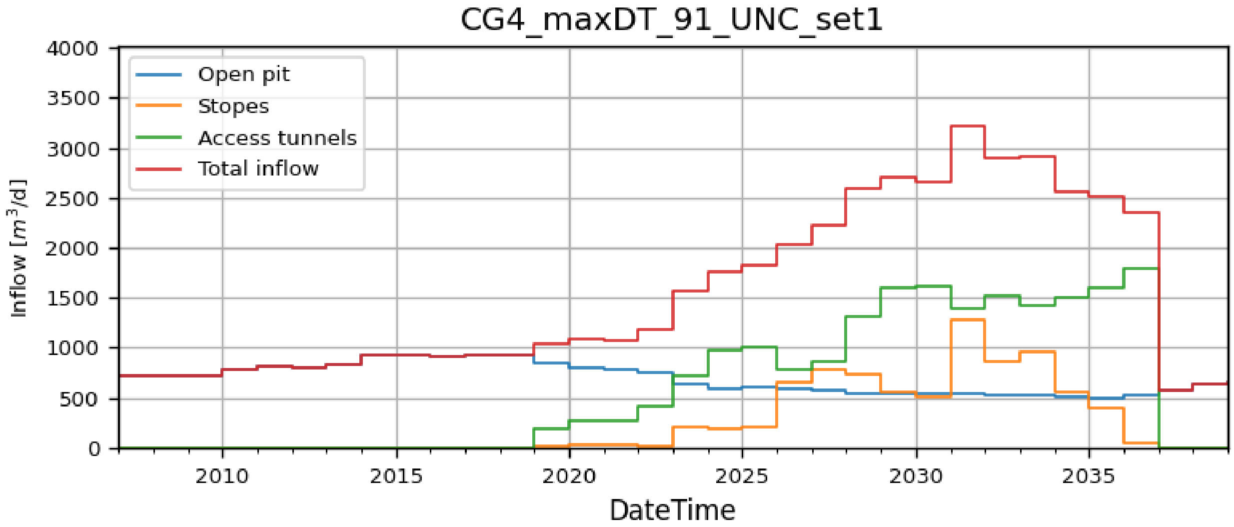

Table 5.1 CG4 base case parameter Set 1 predicted mine inflows (m^3/d)

Year	Open Pit	Access Tunnels	Stopes	Total inflow
2004	0	0	0	0
2005	12731	0	0	12731
2006	729	0	0	729
2007	711	0	0	711
2008	721	0	0	721
2009	712	0	0	712
2010	775	0	0	775
2011	808	0	0	808
2012	807	0	0	807
2013	823	0	0	823
2014	925	0	0	925
2015	924	0	0	924
2016	916	0	0	916
2017	925	0	0	925
2018	920	0	0	920
2019	847	188	11	1046
2020	794	269	26	1089
2021	784	266	26	1076
2022	748	421	7	1176

Table 5.1 CG4 base case parameter Set 1 predicted mine inflows (m^3/d)

Year	Open Pit	Access Tunnels	Stopes	Total inflow
2023	639	722	208	1569
2024	597	976	195	1768
2025	613	999	206	1818
2026	589	785	659	2033
2027	568	865	784	2217
2028	549	1314	733	2595
2029	541	1607	564	2712
2030	537	1611	503	2651
2031	541	1394	1277	3212
2032	524	1528	860	2912
2033	523	1428	964	2915
2034	508	1496	551	2555
2035	502	1605	406	2513
2036	521	1799	47	2366
2037	575	0	0	575
2038	633	0	0	633

5.3 Migration of solutes from the TSF / IWL

5.3.1 Pathline analysis

Pathline analysis, where the pathline follows a particle in a transient flow field, was undertaken to indicate the fate of contaminants from the TSF / IWL. The method used to generate the pathlines was consistent with the EIS modelling. The effective porosity was set to 0.001, consistent with EIS and RTS modelling. Pathlines were seeded from the periphery of the IWL at the end of mining (corresponding to 2038). The final time of the streamline calculations are presented for travel times of 20, 50, 100, 200 years post-closure, corresponding to the years 2058, 2088, 2138 and 2238.

Results of the pathline analysis at the specified travel times are presented in plan view and section views in Figure 5.22 to Figure 5.29. Plan view comparisons of the pathlines predicted by the EIS and CG4 models are presented in Appendix D.

Visual inspection of the CG4 pathline analysis shows that the post-closure migration of solutes from the TSF and IWL are consistent with the EIS model results, with the differences observed attributed to the changed location and development schedule of the UG mine access tunnels.

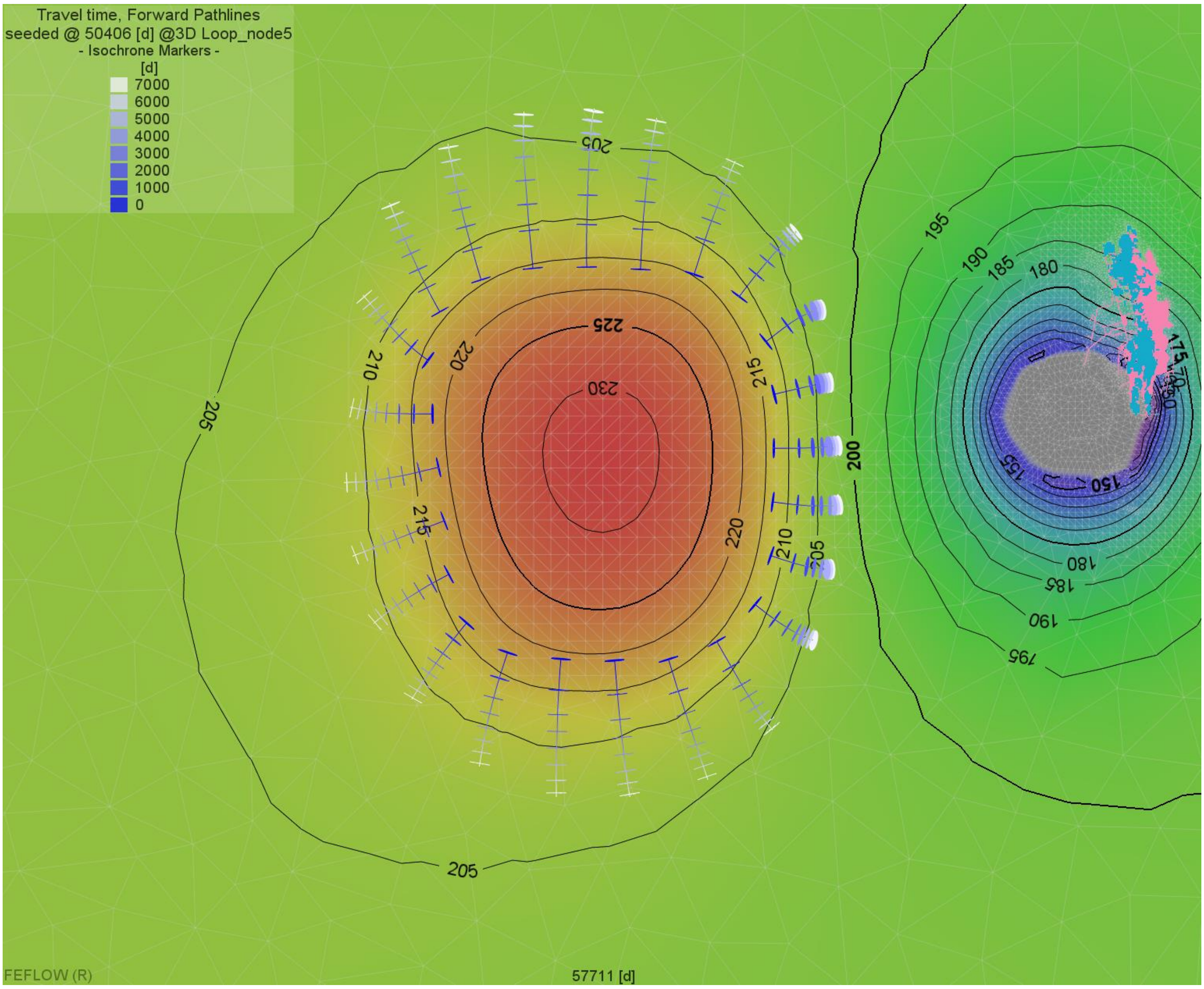

Figure 5.22 **Plan view showing predicted movement of contaminants from the IWL for the period 2038 to 2058**

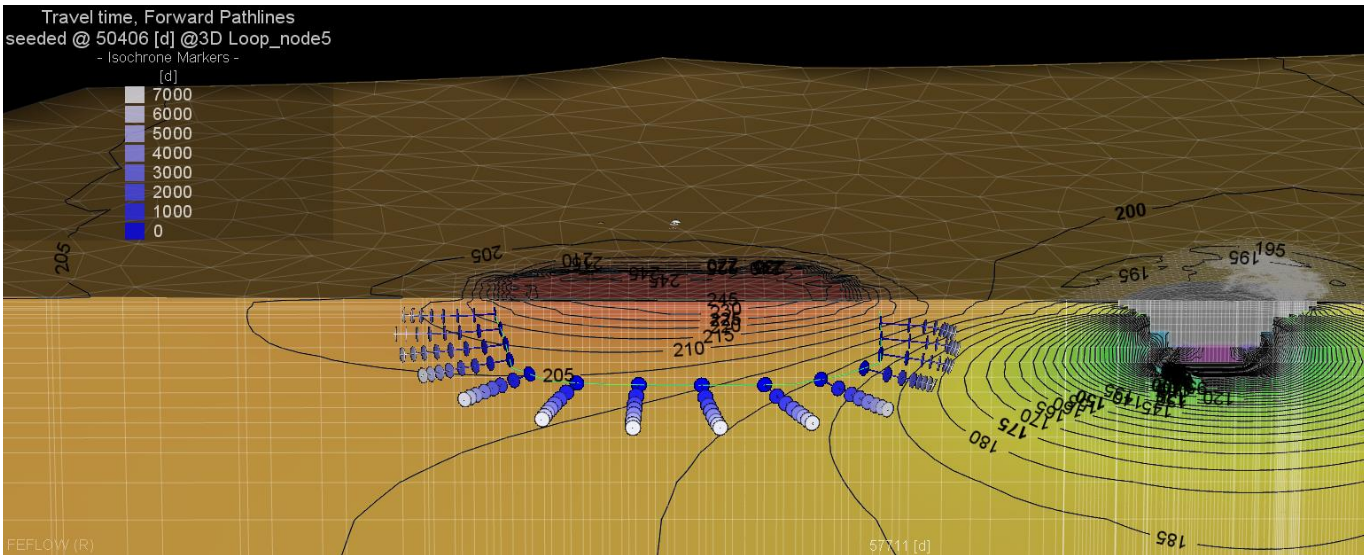

Figure 5.23 **Section view looking north showing predicted movement of contaminants from 2038 to 2058**

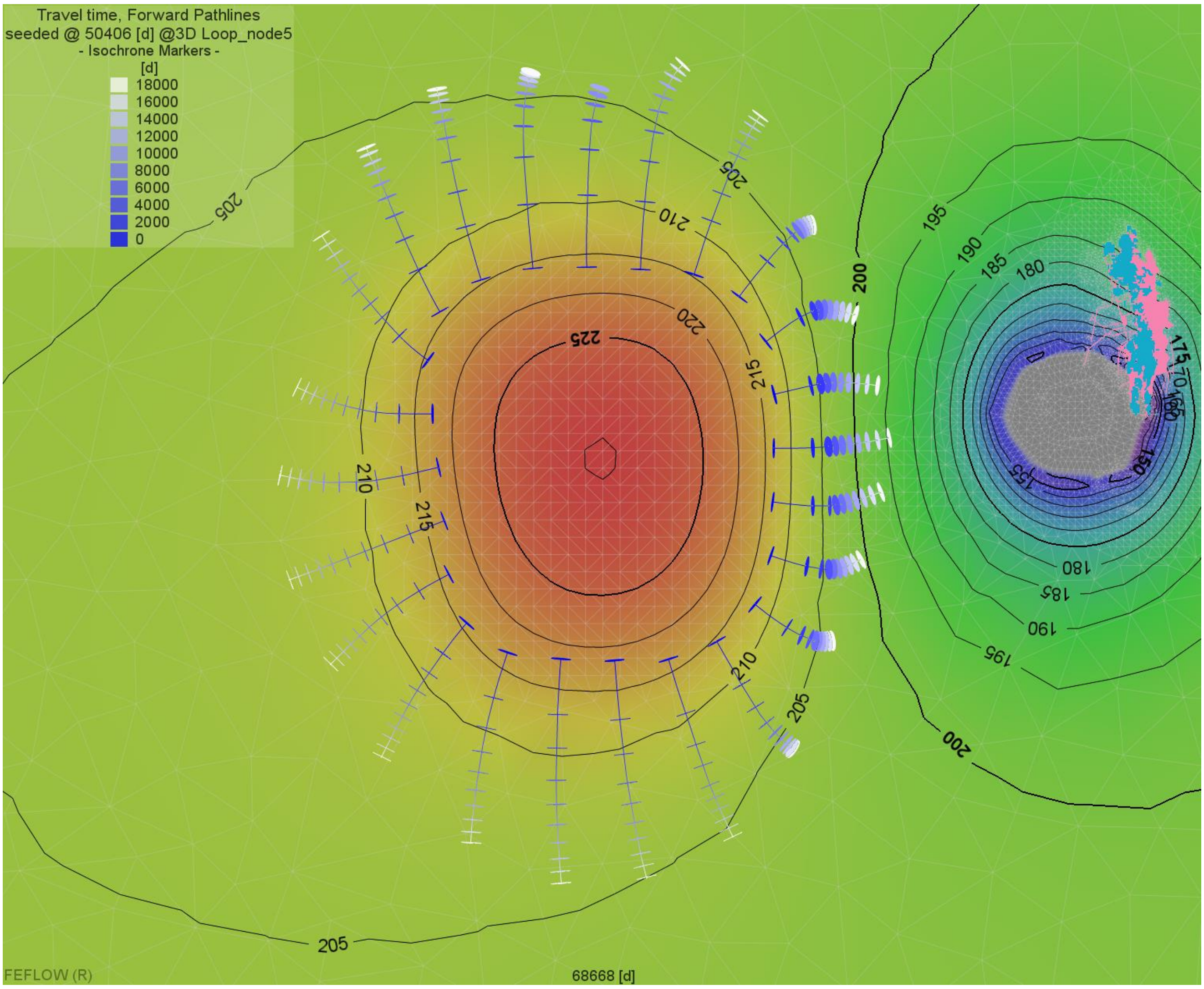

Figure 5.24 Plan view showing predicted movement of contaminants from the IWL for the period 2038 to 2088

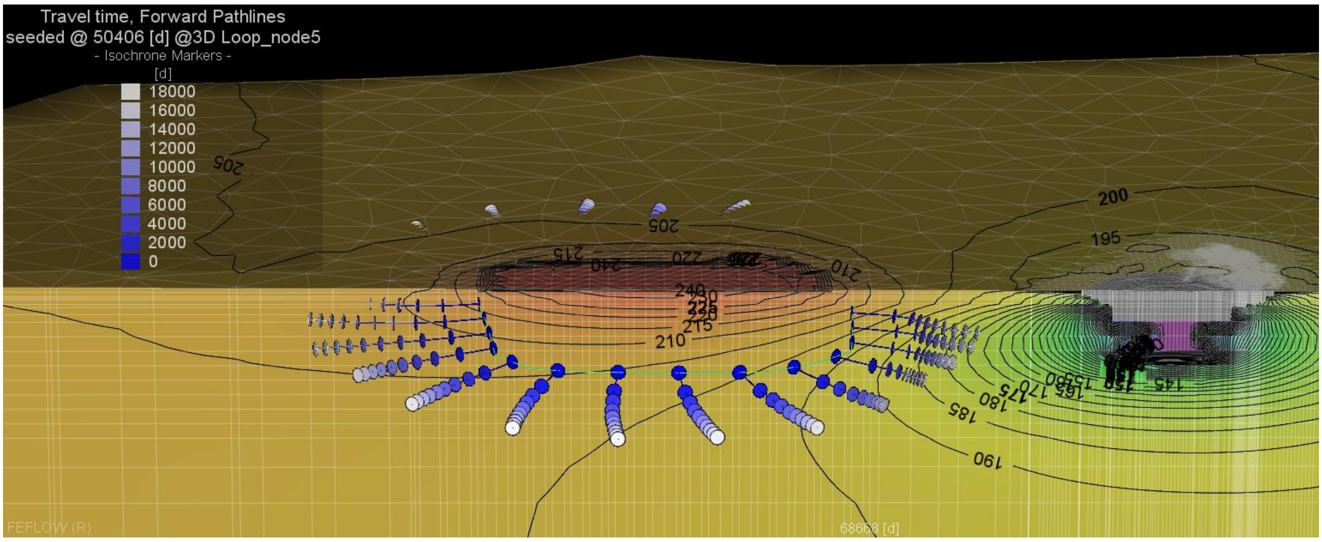

Figure 5.25 Section view looking north showing predicted movement of contaminants from 2038 to 2088

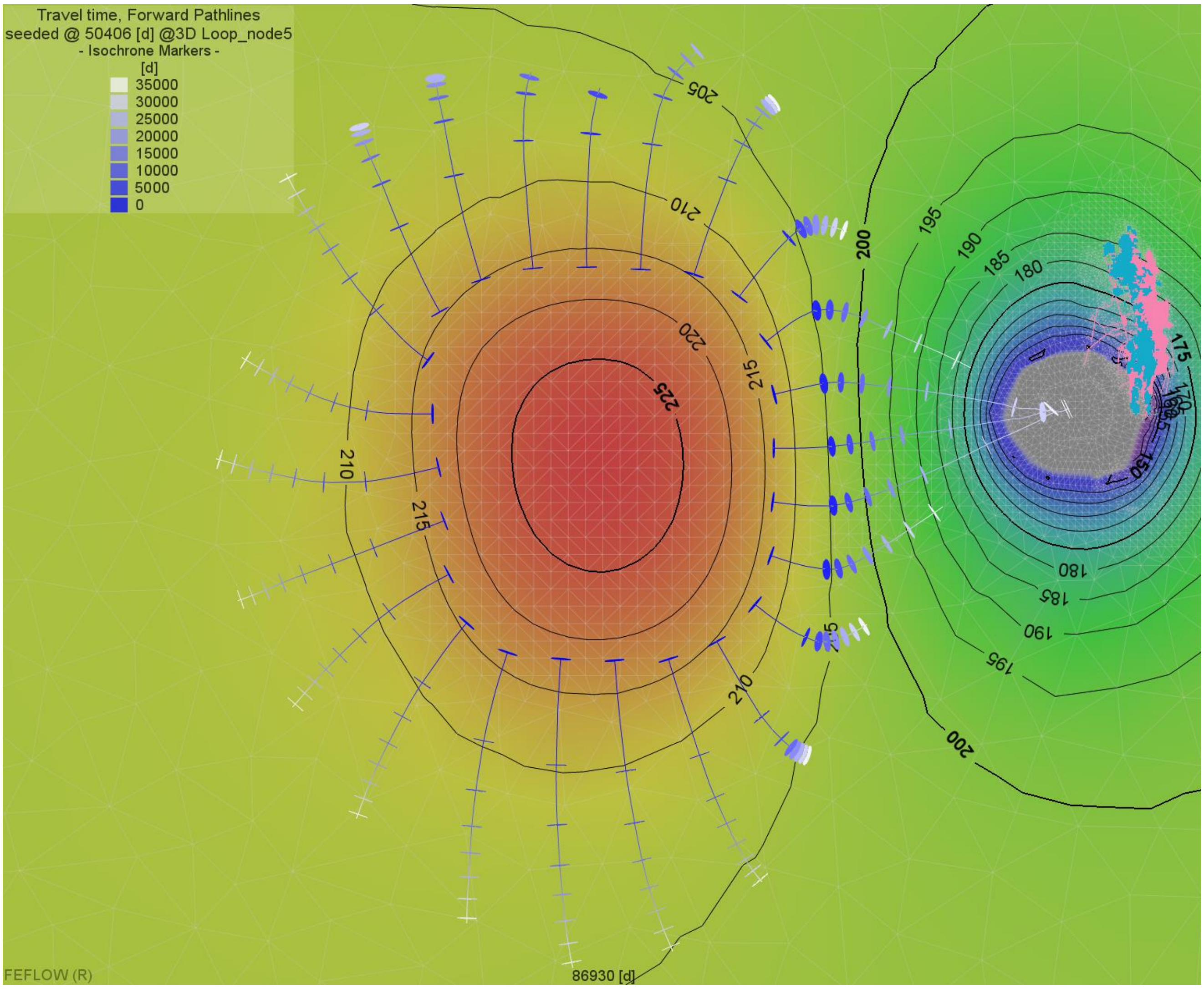

Figure 5.26 **Plan view showing predicted movement of contaminants from the IWL for the period 2038 to 2138**

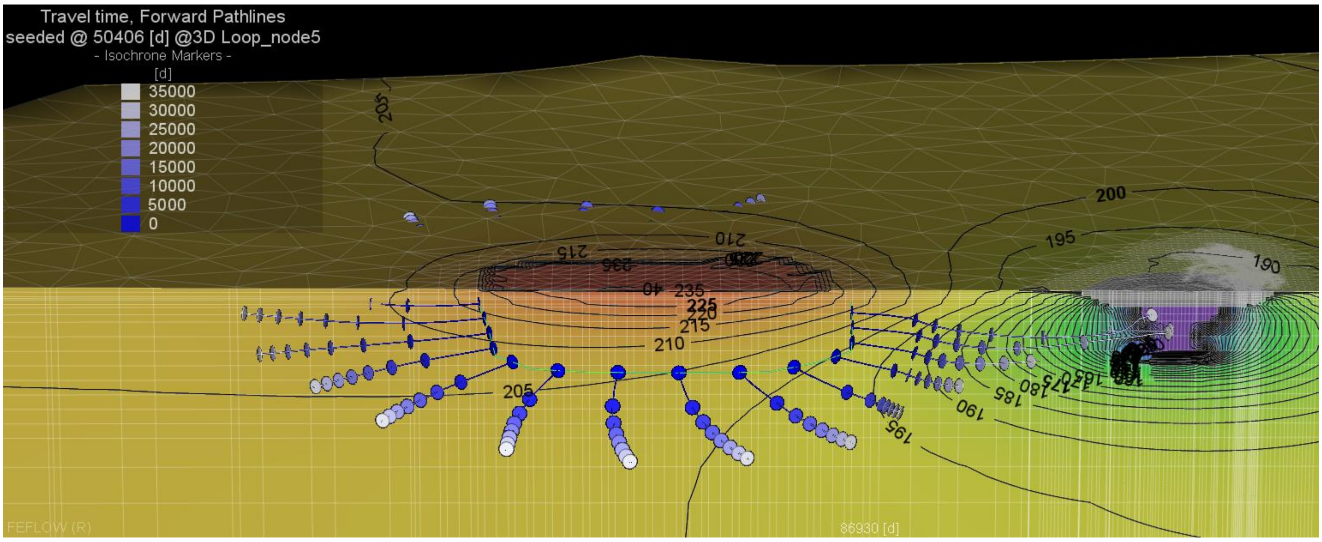

Figure 5.27 **Section view looking north showing predicted movement of contaminants from 2038 to 2138**

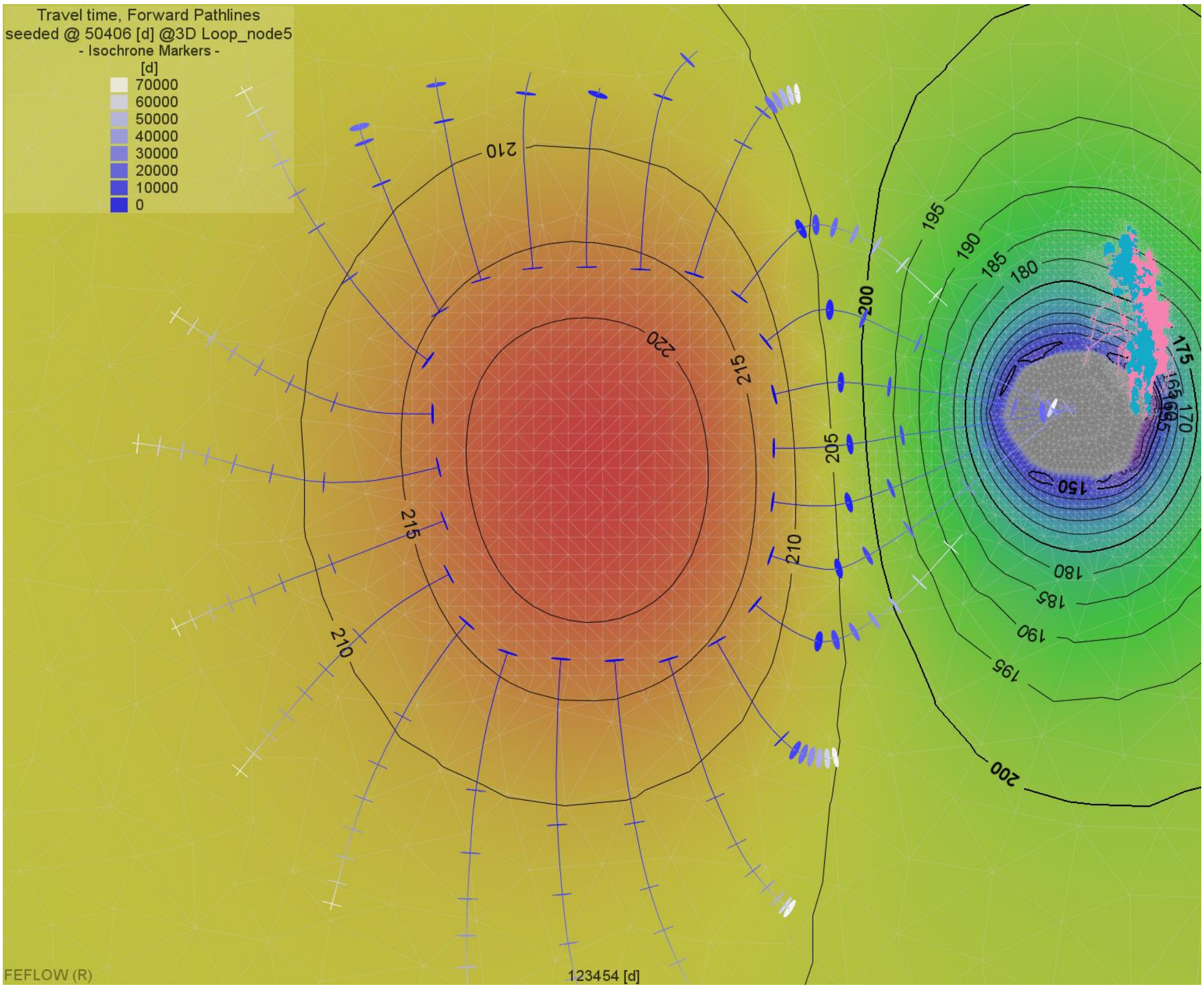

Figure 5.28 Plan view showing predicted movement of contaminants from the IWL for the period 2038 to 2238

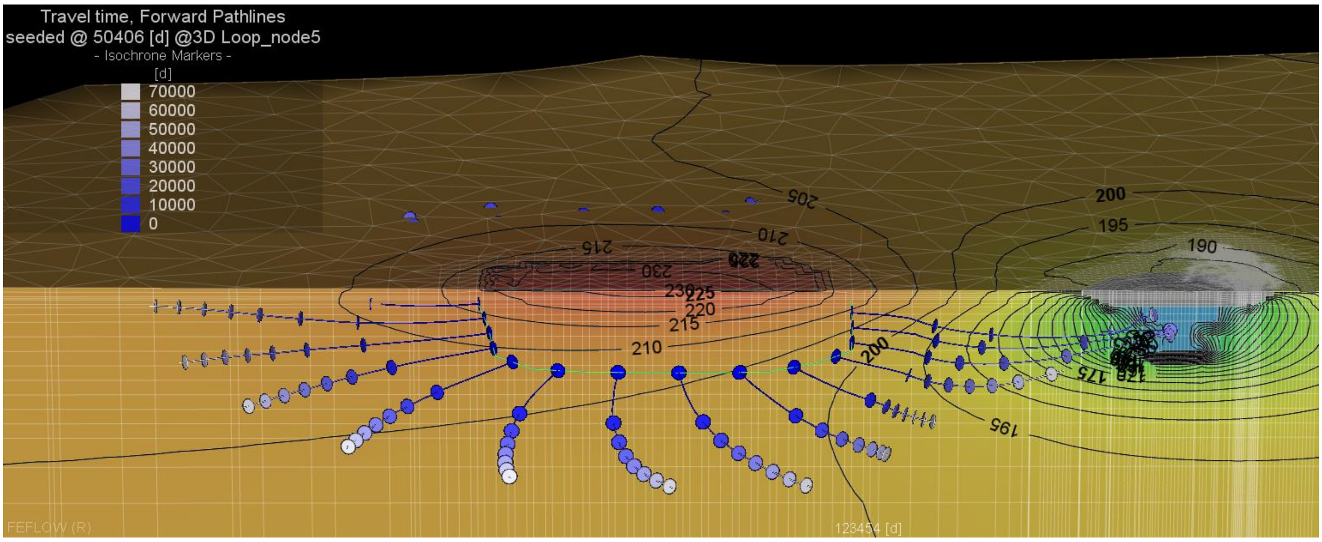

Figure 5.29 Section view looking north showing predicted movement of contaminants from 2038 to 2238

6 Summary

The groundwater flow model developed for the CGO EIS (EMM 2020) has been updated to simulate proposed modification of the underground mine plan.

Despite the changes to the model, the results from the updated model (CG4) are consistent with the EIS model, with similar predicted changes to hydraulic heads and inflows to the mine features that did not change in the model (ie open pit, TSF and IWL).

The updated model predicts inflows to the access tunnels and stopes are the same order as those predicted by the EIS model, with different timings related to the altered scheduling of the UG mine plan.

Particle-tracking using pathline analysis shows that the LoM and post-closure migration of solutes from the TSF and IWL are consistent with the EIS modelling predictions, with differences attributed to the changed location and development schedule of the UG mine access tunnels.

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References

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Appendix A
Verification of CG4 model against EIS model heads

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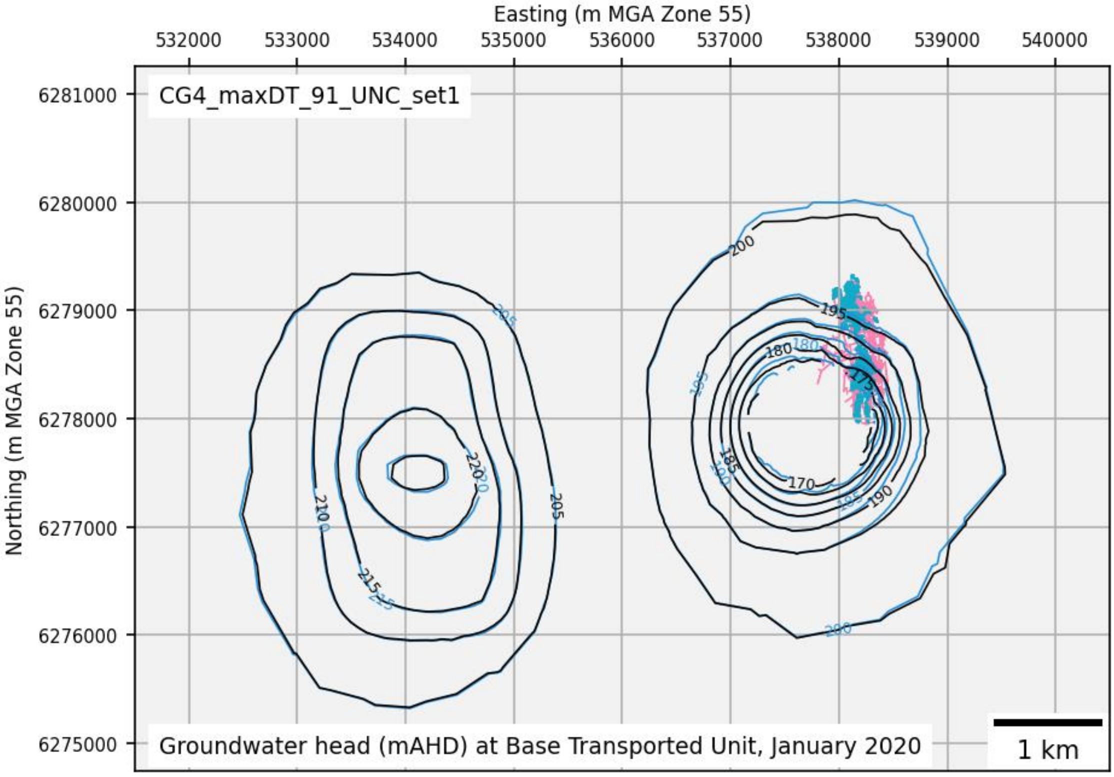

Figure A.1 Comparison of EIS (black) and CG4 (blue) groundwater head contours at base transported unit January 2020

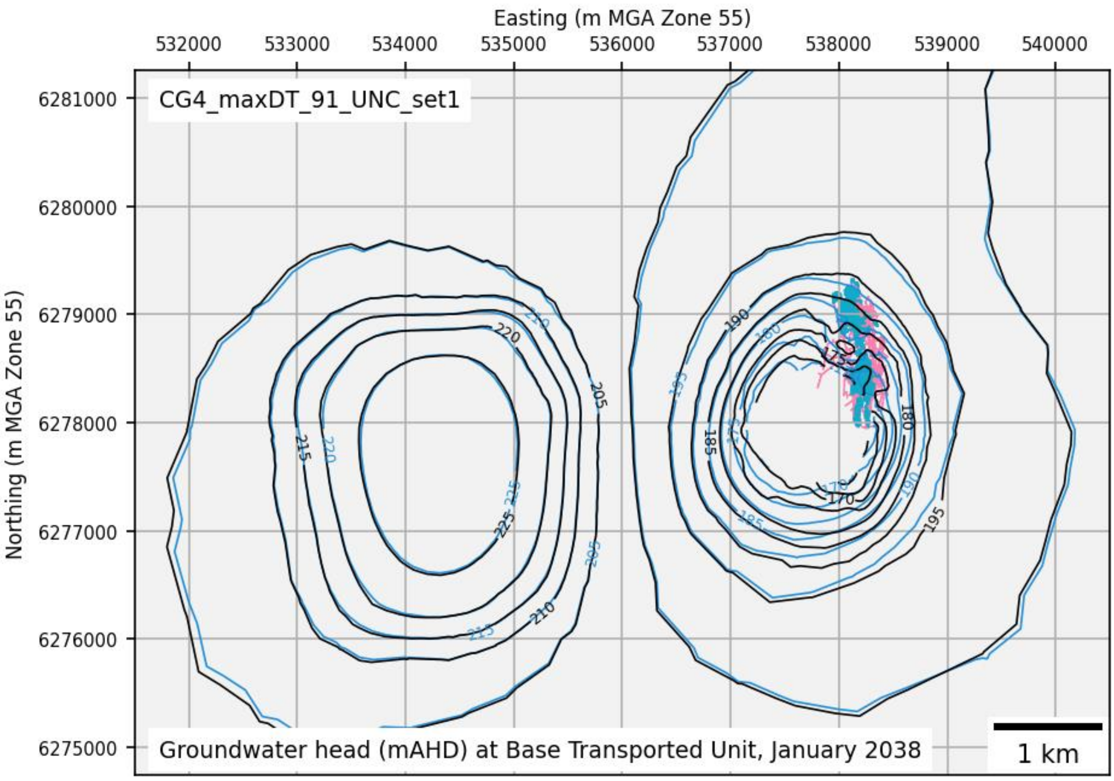

Figure A.2 Comparison of EIS (black) and CG4 (blue) groundwater head contours at base transported unit January 2038

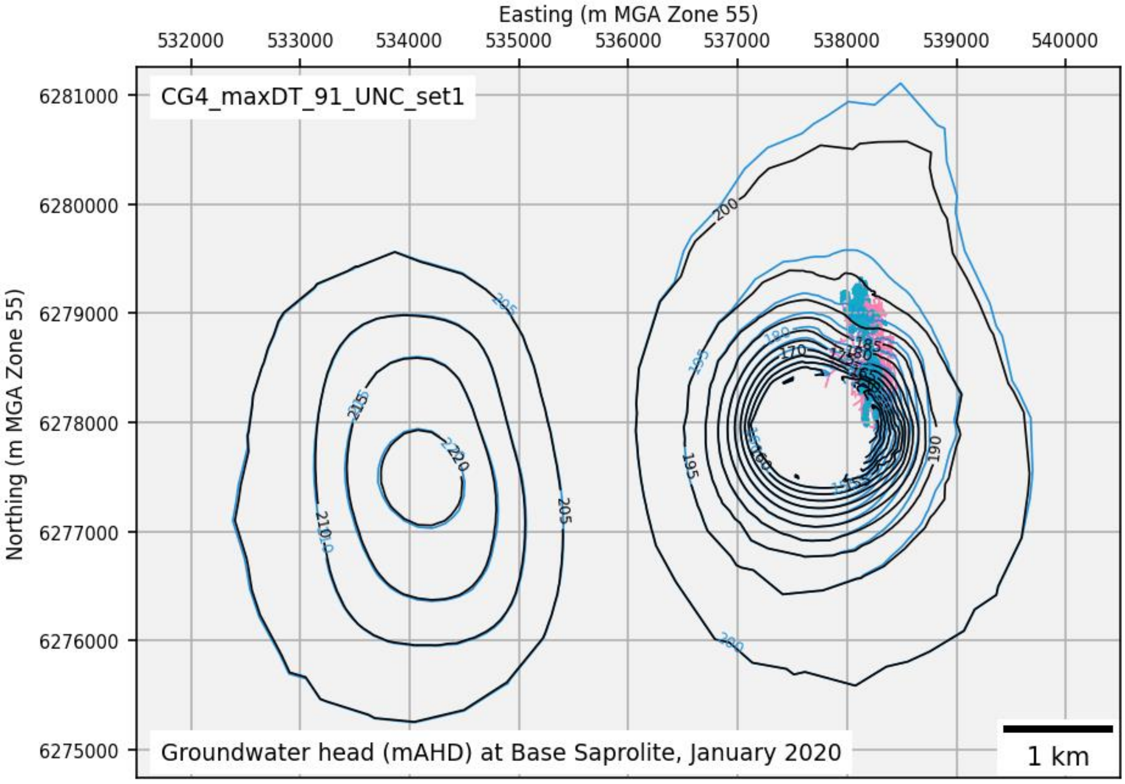

Figure A.3 **Comparison of EIS (black) and CG4 (blue) groundwater head contours at base saprolite unit January 2020**

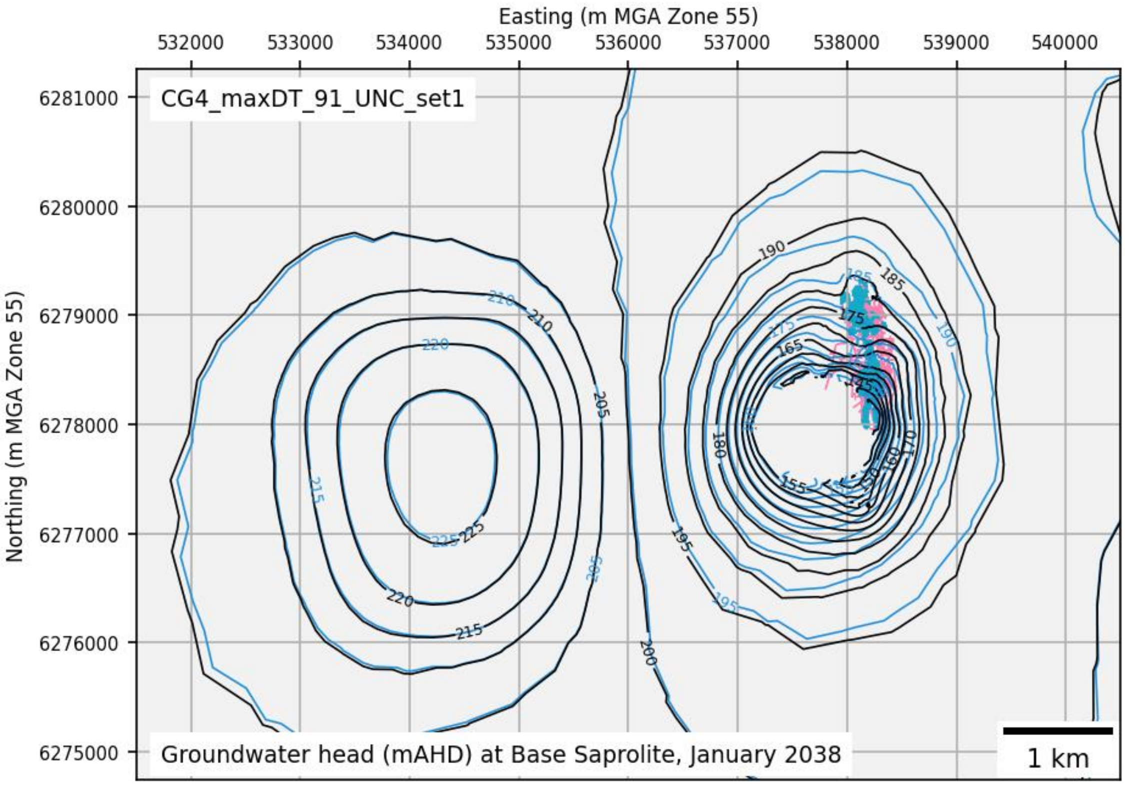

Figure A.4 **Comparison of EIS (black) and CG4 (blue) groundwater head contours at base saprolite unit January 2038**

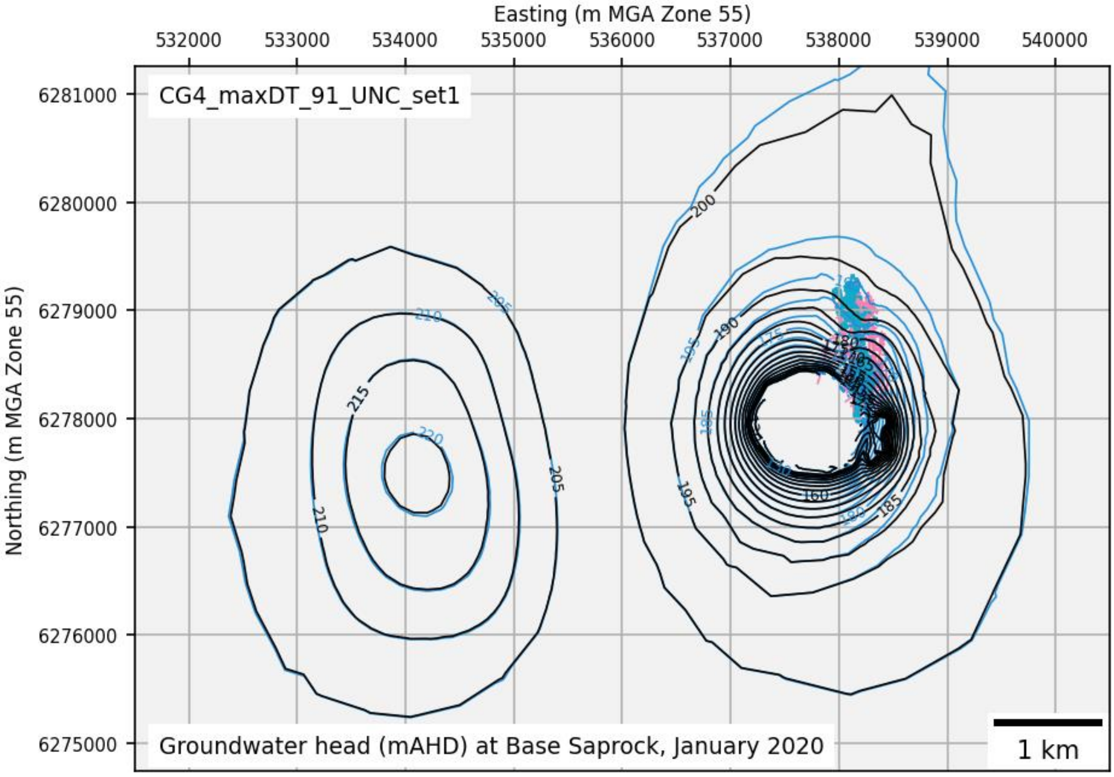

Figure A.5 **Comparison of EIS (black) and CG4 (blue) groundwater head contours at base saprock unit January 2020**

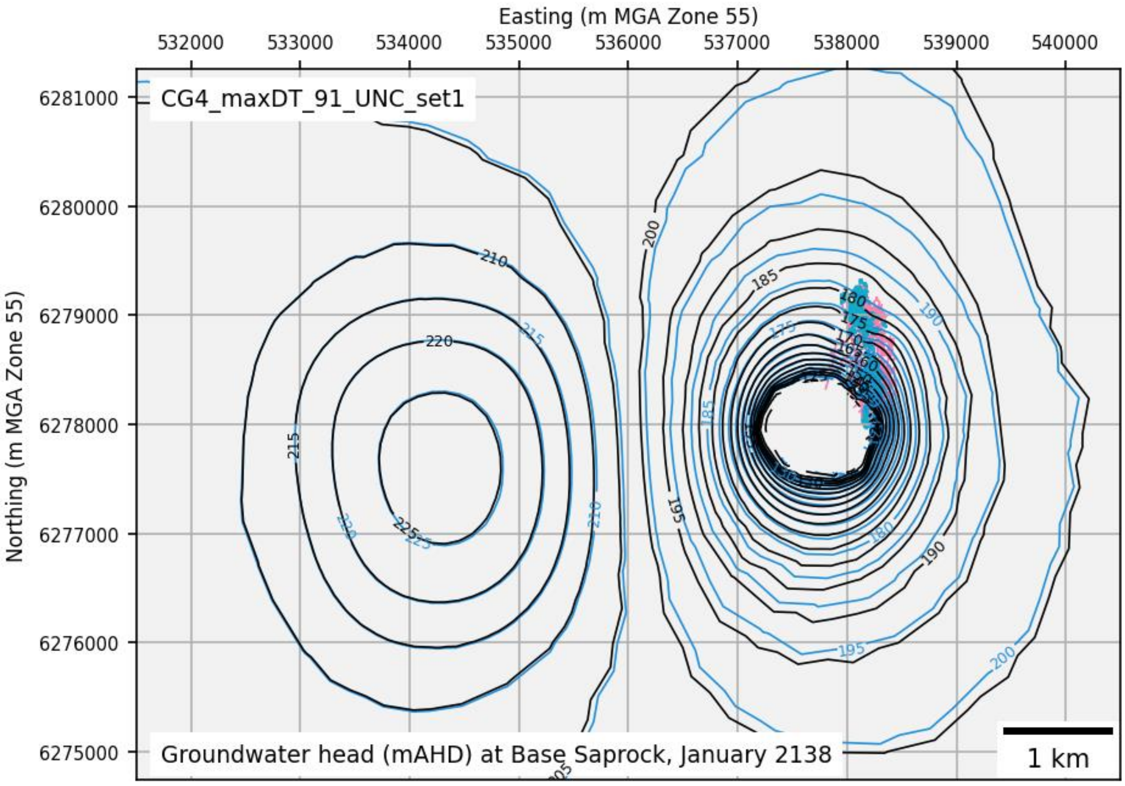

Figure A.6 **Comparison of EIS (black) and CG4 (blue) groundwater head contours at base saprock unit January 2038**

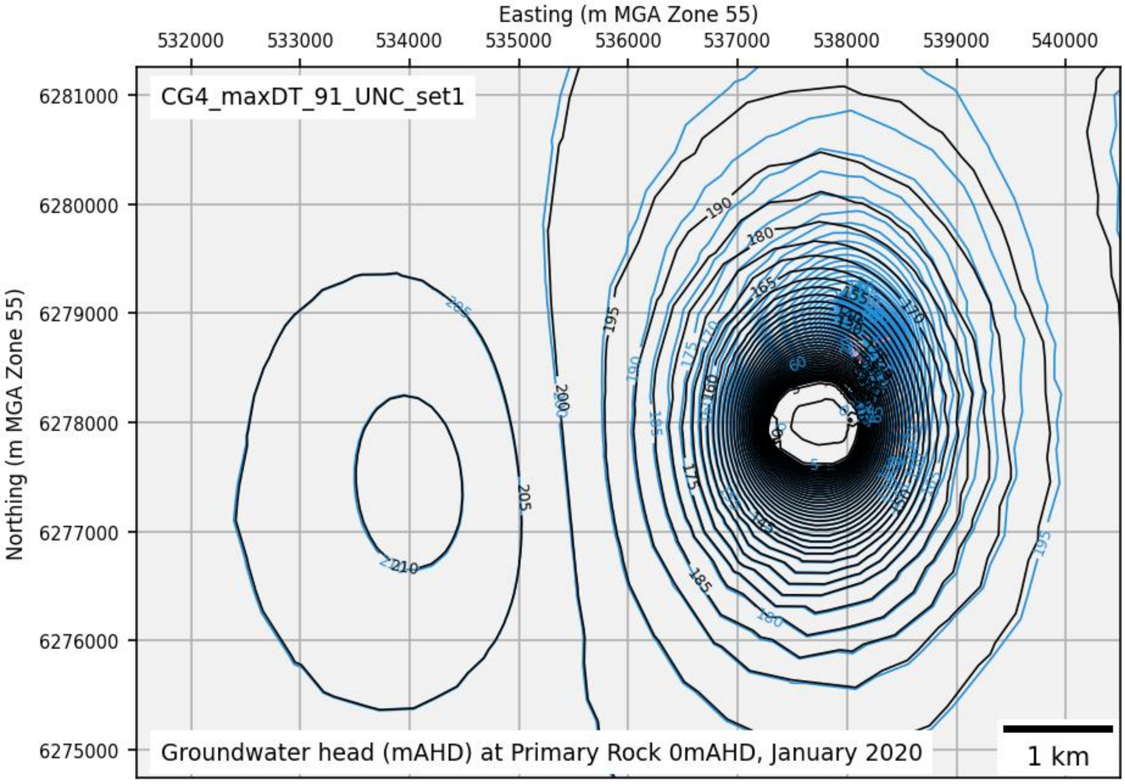

Figure A.7 Comparison of EIS (black) and CG4 (blue) groundwater head contours for the Primary rock unit at 0 mAHD, January 2020

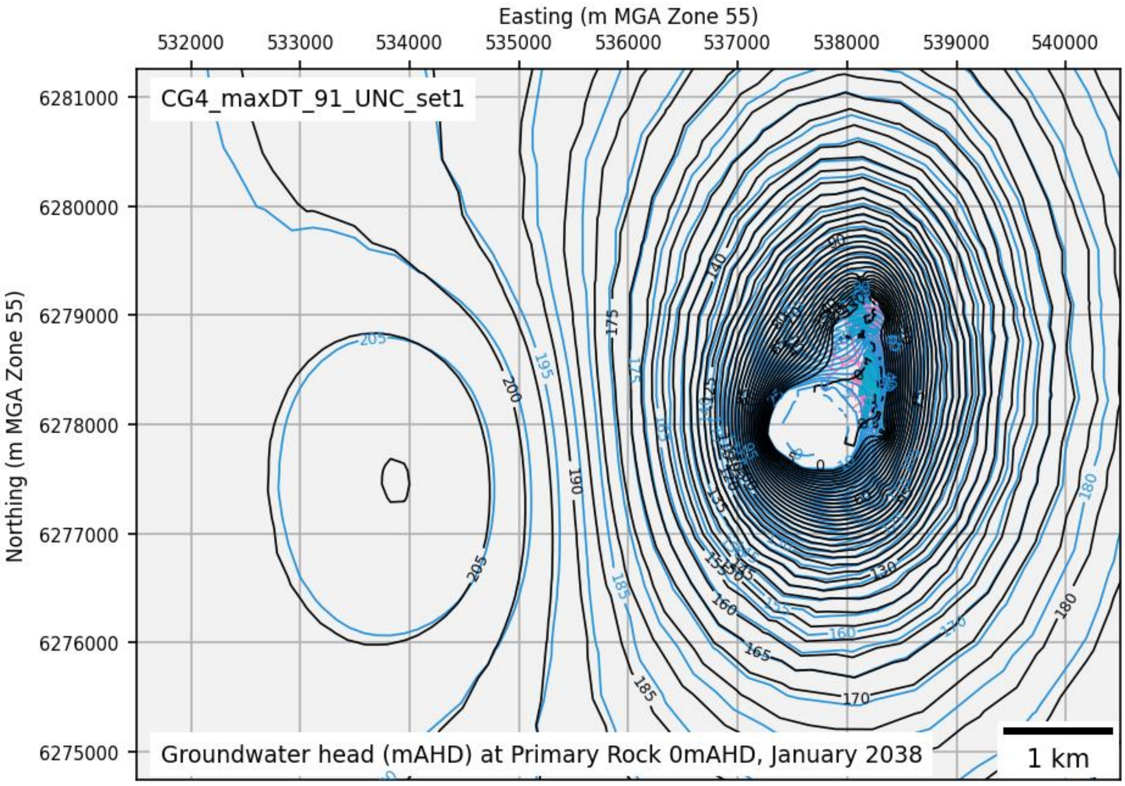

Figure A.8 Comparison of EIS (black) and CG4 (blue) groundwater head contours for the Primary rock unit at 0 mAHD, January 2038

Appendix B

Verification of CG4 model against EIS model open pit inflows

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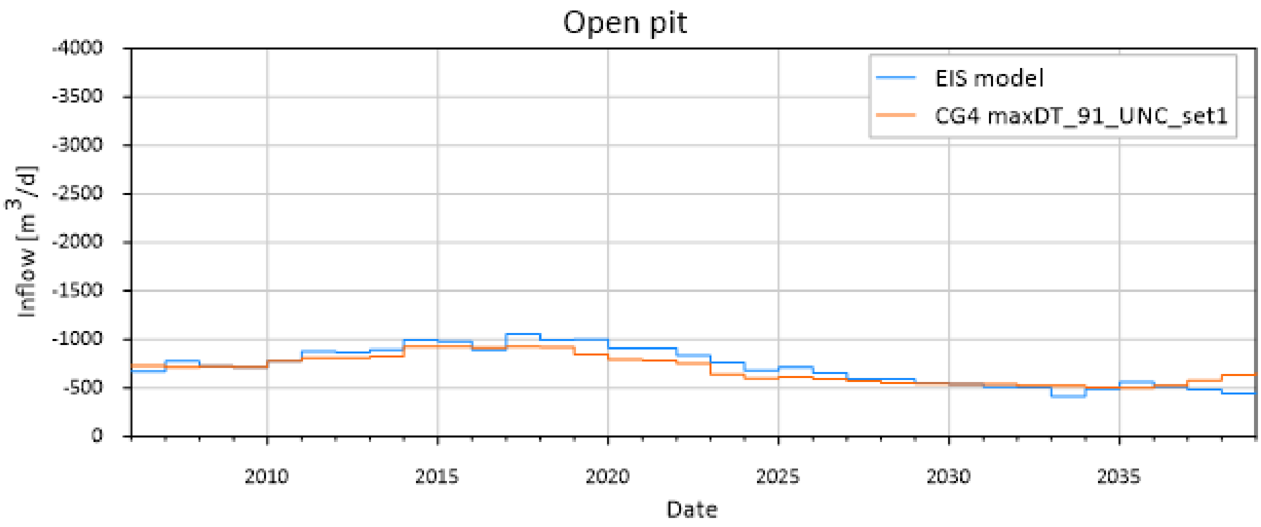

Figure B.1 Comparison of inflows to the open pit for LoM calculated using the EIS model and the CG4 model

Appendix C

Mine inflows for alternative parameter sets

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Table C.1 Mine inflow results for parameter sets 2 to 4

Year	Set 2				Set 3				Set 4			
	Open pit	Access tunnels	Stopes	Total	Open pit	Access tunnels	Stopes	Total	Open pit	Access tunnels	Stopes	Total
2004	0	0	0	0	0	0	0	0	0	0	0	0
2005	12530	0	0	12530	12588	0	0	12588	12586	0	0	12586
2006	577	0	0	577	852	0	0	852	627	0	0	627
2007	609	0	0	609	883	0	0	883	624	0	0	624
2008	644	0	0	644	858	0	0	858	652	0	0	652
2009	650	0	0	650	859	0	0	859	648	0	0	648
2010	717	0	0	717	866	0	0	866	730	0	0	730
2011	766	0	0	766	881	0	0	881	768	0	0	768
2012	768	0	0	768	882	0	0	882	770	0	0	770
2013	787	0	0	787	876	0	0	876	790	0	0	790
2014	908	0	0	908	897	0	0	897	907	0	0	907
2015	904	0	0	904	891	0	0	891	905	0	0	905
2016	904	0	0	904	880	0	0	880	904	0	0	904
2017	910	0	0	910	876	0	0	876	910	0	0	910
2018	910	0	0	910	871	0	0	871	909	0	0	909
2019	833	211	12	1056	778	265	24	1067	830	208	12	1051
2020	776	303	29	1109	687	366	45	1097	769	298	29	1096
2021	771	301	29	1101	664	349	44	1056	763	295	28	1087
2022	723	473	8	1204	626	495	17	1137	721	466	8	1195
2023	603	779	229	1611	544	691	191	1426	605	777	229	1611
2024	562	1069	217	1848	496	866	165	1527	563	1063	216	1842
2025	582	1109	231	1921	533	901	197	1632	579	1092	228	1899
2026	553	862	734	2149	487	755	477	1720	554	854	731	2139
2027	535	948	873	2356	457	786	593	1837	533	940	870	2343
2028	518	1438	811	2767	426	1038	538	2001	514	1438	812	2764
2029	500	1773	631	2903	403	1178	384	1965	500	1769	629	2898
2030	508	1799	568	2875	399	1198	311	1909	500	1780	562	2842
2031	499	1535	1408	3442	383	1024	691	2099	493	1531	1417	3440
2032	492	1708	976	3176	378	1103	443	1924	486	1687	964	3137
2033	486	1597	1094	3177	378	1064	508	1950	477	1574	1078	3129

Table C.1 Mine inflow results for parameter sets 2 to 4

Year	Set 2				Set 3				Set 4			
	Open pit	Access tunnels	Stopes	Total	Open pit	Access tunnels	Stopes	Total	Open pit	Access tunnels	Stopes	Total
2034	487	1697	626	2810	379	1143	328	1849	468	1660	614	2743
2035	481	1828	468	2777	381	1211	236	1829	469	1782	456	2707
2036	497	2057	53	2608	386	1297	7	1689	482	2007	52	2541
2037	560	0	0	560	416	0	0	416	538	0	0	538
2038	639	0	0	639	458	0	0	458	617	0	0	617

Appendix D

Comparison of EIS and CG4 pathlines

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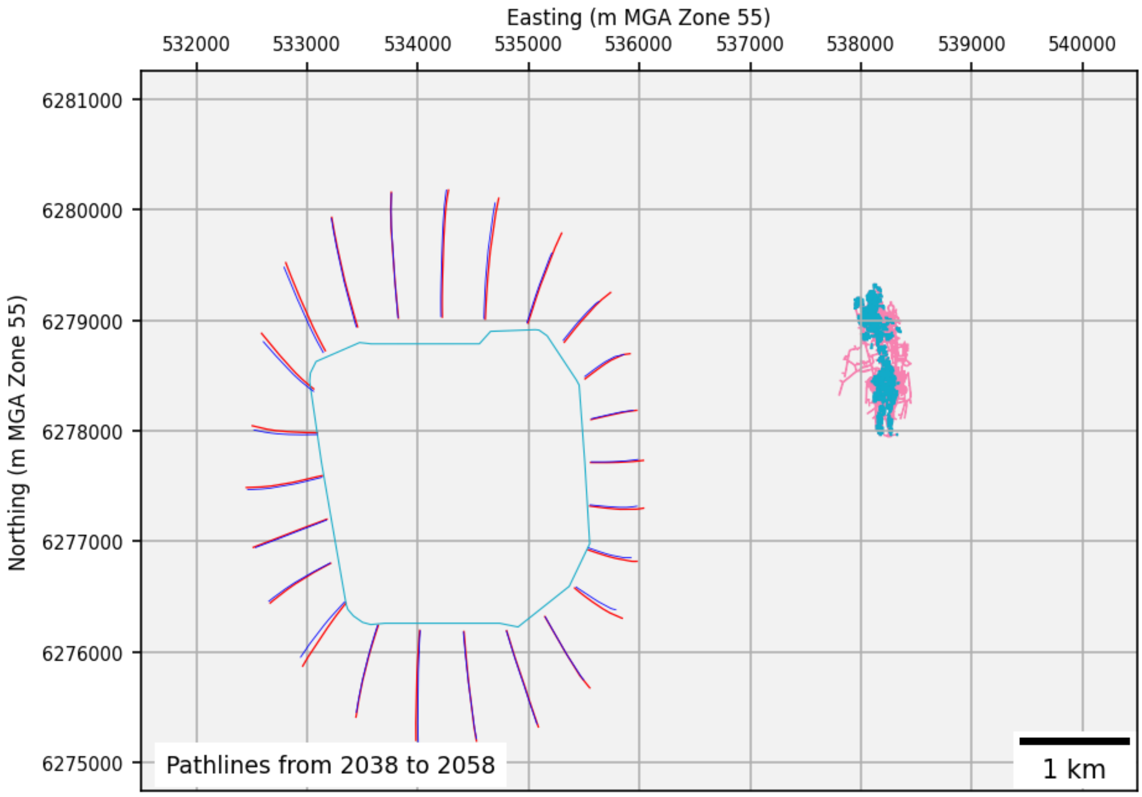

Figure D.1 **Plan view of pathlines predicted by the EIS (blue) and CG4 (red) models for the 20 year period 2038 to 2058**

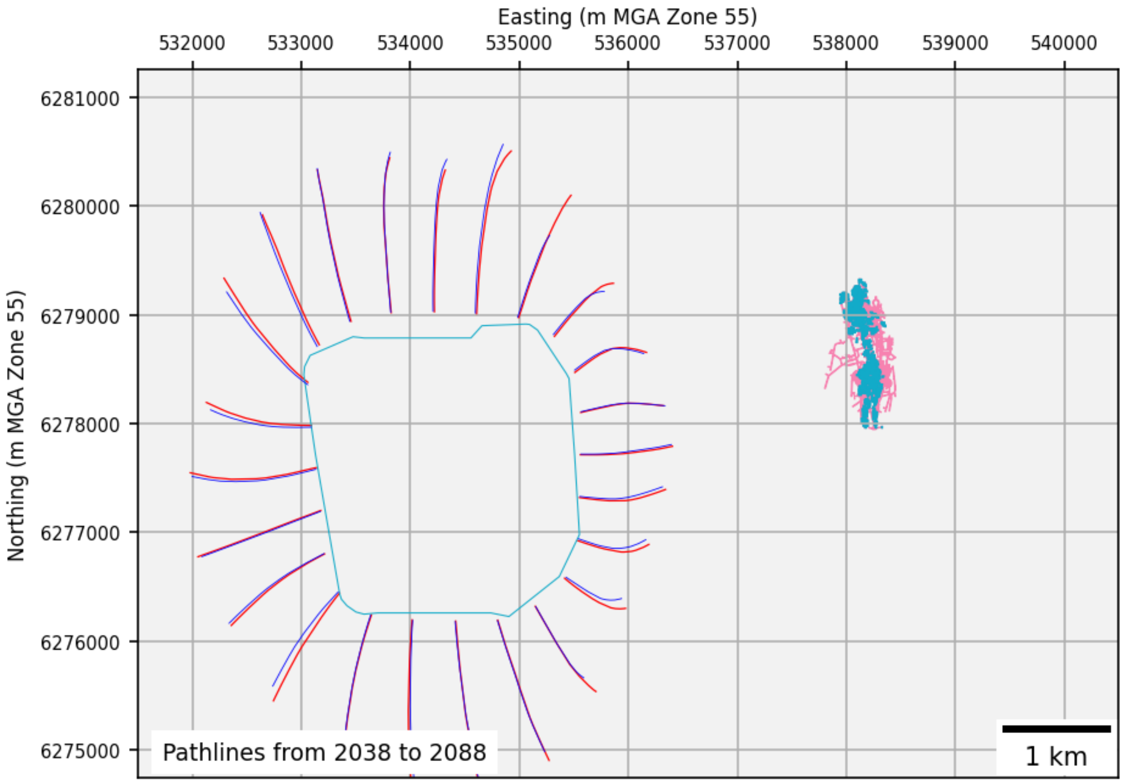

Figure D.2 **Plan view of pathlines predicted by the EIS (blue) and CG4 (red) models for the 50 year period 2038 to 2088**

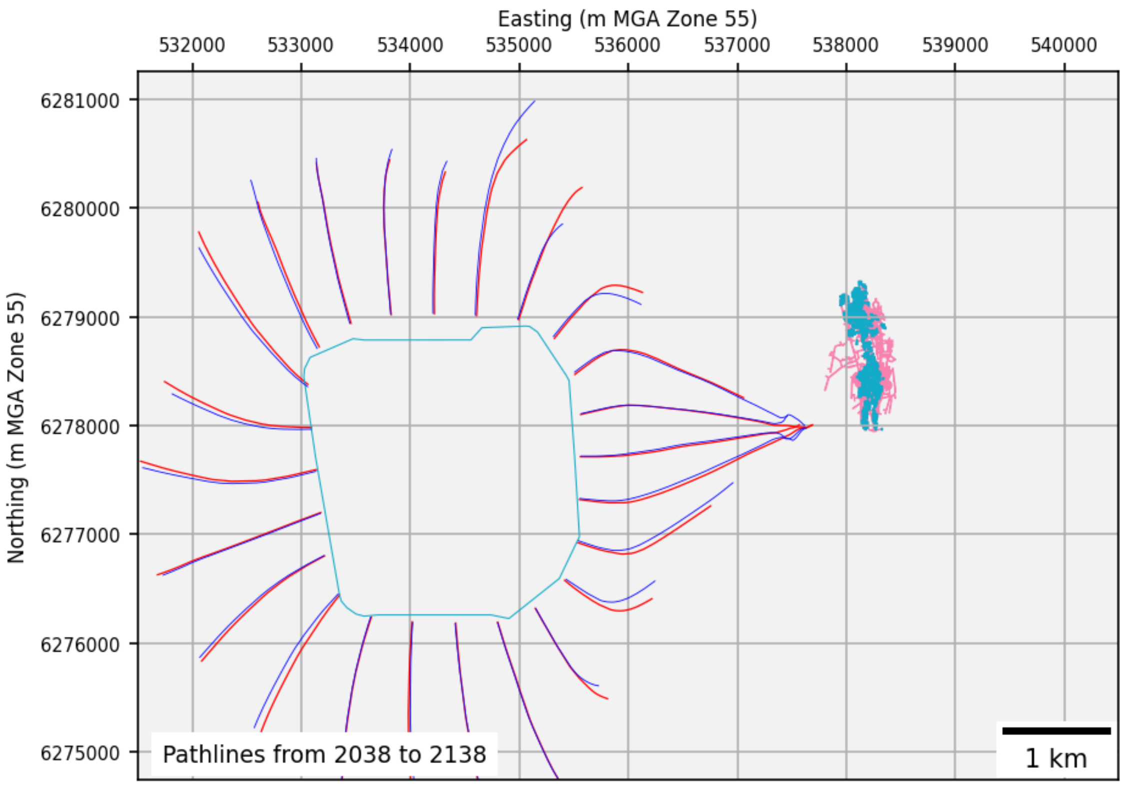

Figure D.3 **Plan view of pathlines predicted by the EIS (blue) and CG4 (red) models for the 100 year period 2038 to 2138**

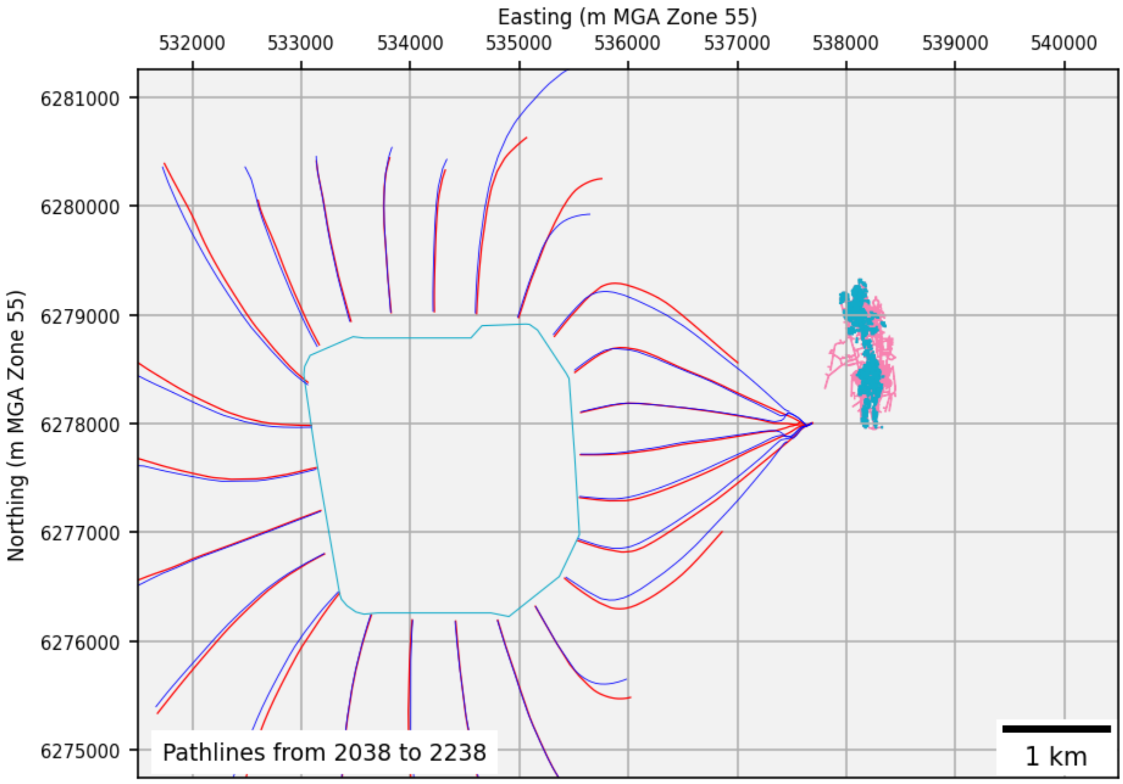

Figure D.4 **Plan view of pathlines predicted by the EIS (blue) and CG4 (red) models for the 200 year period 2038 to 2238**

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Appendix E

Geotechnical review

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UNDERGROUND MINING SEQUENCE AND DECLINE LOCATION REVIEW FOR LAKE COWAL

Assessment of design scenario 4

PREPARED FOR
EVOLUTION MINING PTY LTD

DOCUMENT CONTROL

Date	Version	Comments	Signed
2021JAN22	DRAFT01	Initial draft for review & comments.	

EXECUTIVE SUMMARY

Beck Engineering (BE) has conducted a global stability assessment for three conceptual mine plans for underground mining at Lake Cowal, located 45 km north east of the town of West Wyalong in central NSW. The mine is currently an operating pit with early stage underground mine development currently being completed.

The aim of this project was to:

1. Assess LOM stability and deformation for the underground mine, including underground infrastructure and ventilation shafts using the existing LOM numerical model to provide guidance of major vulnerabilities and opportunities in previous mine the mine plan

2. Simulate 2-3 alternate mine infrastructure layouts and/or mining sequences in a numerical model to forecast the behaviour of the proposed mining sequences including the impact in ore drives, level accesses, declines, pillars and infrastructure.

3. Provide guidance on the mine design and stoping sequence and recommend changes as required.

Our assessment is based on numerical modelling using finite element (FE) methods. An overview of the assessment, including the main findings, risks and recommendations is summarised below. More extensive details are provided in Section 3 of this report.

This report documents our assessment of a fourth mining scenario, in addition to the previous assessments of the original three mine plans provided.

Main findings

Infrastructure and decline layout:

1. The stoping sequence, level layout, decline and infrastructure position vary significantly in each mine design option assessed in this project. This means direct trade-offs and comparisons are difficult as there are multiple variables that change when comparison options. Each design aspect of each option has been assessed individually for the three mine design options provided.
2. Each design option and mining sequence has advantages and disadvantages over its counterparts. The recommended adjustments to the mine plan draws on components from each of the three mine plans which are summarised here and described in detail in the body of the report.
   - The main difference for Design option 4 from the other designs assessed (including the relatively similar design in FW S01) is the additional footwall drive connections in the longitudinal panels. This has potential productivity benefits, however the additional accesses and planned central retreat stoping sequence forms diminishing pillars. This mostly occurs in the South panel. Although this is generally considered unfavourable from a geotechnical perspective, the rockmass conditions are favourable and depth is relatively shallow (i.e. low stress conditions). This results in limited stability and ground control problems are forecasts in these areas.
   - Central declines (and central diminishing pillars) are generally not recommended below a depth of ~500m due to stress concentration causing ground control problems, particularly in the final stopes and near the level accesses. Both the north and south production panels have large barren areas which form waste pillars. This enables a central decline to be positioned with a large waste pillar near the level access without significant ground control problems developing due to the large waste pillars and low extraction ratio. i.e. a diminishing pillar is only formed in isolated parts of the panel due to the barren regions. However, infill drilling may (or may not) identify additional mineralisation and the orebody, and planned stoping may become more continuous than currently known and planned. This would not adversely impact the mine plan above a depth of 400-500m due to the favourable rockmass conditions. However, below a depth of ~500-600m, diminishing central pillars would inevitably be subject to increasing ground control problems such as deformation in drives, elevated stope over break, production delays for rehabilitation and redrilling and potential seismicity.
3. Tight spiral declines and stacked level accesses (i.e. FW S01 design) are generally not recommended due to the high local excavation ratio. However, no adverse stress concentration is forecast in the model for the tight spiral

declines or stacked level accesses in the FW S01 option due to the favourable rockmass conditions and relative shallow mining depth. However, a figure 8 type decline such as the layout in Design option 4 is considered to be more (geotechnically) favourable.

Transverse vs longitudinal mining:

4. Each of the four mine designs evaluated in this project include both longitudinal and transverse stope development and mining sequences.
   - Design option #4 and the FW S01 option are predominately longitudinal access drives, with transverse stope development for thicker sections of the orebody in both the north and south panels.
   - Design option 4 has additional longitudinal mining fronts that are formed by more centrally located access drives. These form a central retreat sequence in places of the South panel, including diminishing pillars.
   - Design option 4 can be summarised by:
     - Decline and infrastructure layout is very similar to FW option S01
     - Longitudinal stoping with transverse access in thicker sections of the orebody
     - Semi-continuous overhand stoping in the south panels
     - Overhand stoping in the thicker sections north panel is primary-secondary overhand stoping, and is less continuous compared to the south panel
     - Thicker areas of mineralisation (sometimes multiple lenses) are mined with a mix of longitudinal and transverse drives in the mid section of the north panel
     - Diminishing sill pillars within each panel, either the sill pillars between stoping blocks, and where longitudinal stoping retreats to a central access (in some places)
     - Six stopes in the current design have crown pillars <10-20m with the top of fresh rock boundary.
5. Multiple Galway splays are present in the FW of the south panel. This causes a number of adverse fault interactions, including:
   - Longitudinal ore drives for the FW stopes are aligned with faults on some levels. These faults are, in areas, subparallel and run along the longitudinal ore drives.
   - Footwall drives run subparallel/parallel to Galway faults in places
   - Galway splay faults bound the FW of the orebody. This causes the final drawpoint brow position to be located at the same location of stopes with transverse access
   - Fault interaction is variable on each level and through the decline in Design option 4. Galway splay faults intersect level accesses, FW infrastructure and intersect part of the decline, particularly for the south mining panel
6. Faults in the north panel are more widely distributed. Generally, interaction with faults, namely the Galway splays, is more adverse in the south panel compared to the north panel.
7. Transverse stoping in thicker sections of the north and south panels provides more active mining areas and is likely to be more favourable for productivity. Long lead-lags between primary and secondary stopes should be avoided (target 2 stope maximum where possible) and a semi-continuous chevron/pyramid mining front should be targeted to minimise extensive diminishing pillars with previously mined panels above as the overhand stoping panels converge with previously mined stoping blocks.

Design option #4

8. The mine layout and sequence is somewhat similar to the FW S01 mine plan
9. Generally isolated areas of minor to moderate rockmass damage (only) for early and mid stages of mining in both the north and south panels

10. Moderate stress concentration occurs in the diminishing central pillar (~30-40 MPa) in the South panel. Stress concentration is limited due to the orientation of the major principal stress and depth of mining. The increase in stress in the diminishing pillar is not sufficiently high to cause significant damage in production areas during extraction of the stopes in the diminishing pillar.
11. Stress concentration of ~50-60 MPa is forecast in the central diminishing pillar in the mid section of the North panel. The stress is not sufficiently high to cause significant or extensive rockmass damage throughout the central pillar. However, this level of stress is sufficiently high (>0.4 x UCS) to cause low levels of seismicity in diminishing pillars.
12. Moderate damage occurs in the secondary stopes in the thicker mid section of the North panel (i.e. the pillars between the primary stopes). These stopes have elevated potential for crown instability, which would impact the ability to re-access the overcut drives and cause production delays during rehabilitation.
13. Primary-secondary stoping with long (vertical) lead-lags not recommended in the lower North panel due to rockmass damage caused in the secondaries. Altering the sequence in the thicker (overhand) stoping blocks to be a more chevron shaped front will help to mitigate ground control problems associated with the close out (sill) pillars. Most problems are forecast to occur in the secondary stopes in the late stages of each stoping block.
14. Transverse layout and sequence has advantages in the thicker section of the North panel. However, the current sequence in design #4 is unfavourable and forms diminishing sill pillars with a flat mining front (including secondary stopes in the diminishing sill). The upper transverse mining panel is currently split into 2 x two level panels, rather than a single 4 level panel.
15. Transverse stoping and accesses may not be warranted in the narrow sections where single stopes across the orebody width are planned. Galway splay faults adversely impact final stope brows (along the footwall), the footwall drives and some sections of the longitudinal ore drives. We note that characterisation and definition of the major faults in the underground mining precinct is ongoing.
16. A section of the lower North panel is mined using multiple north south ore drives (longitudinal layout) for the multiple lenses. The mineralisation and stope layout is separated for each lens which differs from the more continuous mineralisation and stoping in the (transverse) panel above. Some stopes are close to the main access cause minor to moderate damage locally to the access drive. Recommend to delay these stopes and mine them as production retreats back to the access and FW drive.
17. There are stopes mined out of sequence in both north and south panels. These out of sequence stopes mine out areas along the level access prior to other stopes being mined. A more continuous / progressive retreat sequence towards the level access is recommended. It is likely these out of sequence stopes are minor scheduling errors only, and would be corrected in future design iterations.
18. The barren pillars in both the south and north panel prevents true diminishing pillars from being formed by planned stoping. This of course is not applicable if infill drilling determines the mineralisation is more continuous than currently known.

Recommendations

South panel

1. Six stopes in the current design have crown pillars <10-20m with the top of fresh rock boundary. Recommend to flag these stopes for potential redesign, pending ongoing updates to the geological interpretation of the cover sequence boundaries. These stopes are mined late in the mine life (as per previous recommendations) and this assessment can be done in years to come.

North panel

2. Transverse layout and sequence has advantages in the thicker section of the North panel. However the current sequence in design #4 is unfavourable and forms diminishing sill pillars with a flat mining front (including secondary stopes in the diminishing sill). The upper transverse mining panel is currently split into 2 x two level panels, rather than a single 4 level panel. These should be joined to a single four level panel.

3. The stoping sequence in the North panel should be adjusted to be more continuous in the thicker transverse stoping areas. Primary-secondary stoping in the diminishing sill pillars should be avoided as far as practical.
4. For the lower North panel is mined using multiple north south ore drives (longitudinal layout). We recommend to delay mining of stopes close to the main level accesses prior to mining of the main stoping areas on each level. This will help to delay potential damage to the accesses, as well as delays for bogging, filling and curing of stopes near the access, rather than mining the larger groups of stopes on the levels.

General

5. Confirmation of the pre-mining stress state. Some stress measurements completed are impacted by mining induced effects of the pit and (potentially) the Glenfiddich fault and Cowal shear. The current interpretation of the insitu stress regime provided for this assessment is not aligned with the regional stress field or measured stress at nearby mines. Additional analysis, field measurements and field observation to further confirm the pre-mining stress is required.
6. Compilation and review of rock strength and characterisation data for the open pit and underground datasets. There is no single dataset available at present.
7. Characterisation of known faults, including strength classification, thickness, infill material and general geological description.
8. Fault continuity should be confirmed by Evolution from ongoing drilling and underground mapping.
9. Review faults with updated mapping and ID of faults during mine development. Plan to adjust the mine design for the location of drives, intersections, stope brows etc as required.
10. Adjust the location of intersections to avoid faults during development, where practical. Additional cablebolting will be required where major faults interact with intersections.
11. Do not mine 4-way intersections, as far as practical. Use two 3-way intersections instead.
12. Review mining sequence on each level to ensure level access is not lost due to stopes mined out of sequence.

Limitations

In addition to the normal resolution limits associated with the current finite element model, the main limitations of this assessment are:

1. The most significant limitation of this assessment include:
    a. Confidence in the insitu stress regime. A sensitivity model run was completed in addition to the project scope to assess the effects of a pre-mining stress state more closely aligned with the regional stress field.
    b. Fault properties used in the model are aligned with recent underground modelling projects. It is noted these properties are stronger than previous assessments from 2014 to 2019, and are considered to be high given the geological description of the faults
    c. Fault continuously is unknown and must be confirmed by Evolution from ongoing drilling and underground mapping
2. There is no intermediate scale fault model available and a DFN has not been included in the model
3. Variability in the rock strength properties provided for each lithology domain, including differences in the open pit and underground datasets

Enquiries

Please direct further enquiries to the undersigned.

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Principal Engineer, Mining & Rock Mechanics

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1 INTRODUCTION

Lake Cowal is located 45 km north east of the town of West Wyalong in central NSW. The mine is currently an operating pit with early underground mine development currently occurring. The mine is owned by Evolution Mining.

The aim of Beck Engineering's geotechnical and mining assessment was to:

- Assess LOM stability and deformation for the underground mine, including underground infrastructure and ventilation shafts using the existing LOM numerical model to provide guidance of major vulnerabilities and opportunities in previous mine the mine plan
- Simulate 2-3 alternate mine infrastructure layouts and/or mining sequences in a numerical model to forecast the behaviour of the proposed mining sequences including the impact in ore drives, level accesses, declines, pillars and infrastructure.
- Provide guidance on the mine design and stoping sequence and recommend changes as required.

Our assessment is based on numerical modelling using finite element (FE) methods.

This assessment did not include:

- Modelling of ground support,
- Detailed seismic forecasting,
- Detailed stability forecasts for all individual stopes, drives or benches,
- Hydrogeological modelling.
- Forecasts of stope backfill behaviour.
- A site visit. This was not required for the scope of the project.

This report documents our analysis method, results, associated interpretation, conclusions and our recommendations for the consideration of Evolution Mining and all relevant stakeholders.

This report documents our assessment of a fourth mining scenario, in addition to the previous assessments of the original three mine plans provided.

2 PROJECT WORKFLOW, BACKGROUND DATA & MODEL COMPOSITION

This section summarises the modelling project workflow, the available background data and assumptions relevant to the project and describes how these data and assumptions have been incorporated into the workflow.

2.1 Project workflow & simulation framework

The modelling workflow for this project was:

1. Initial mining engineering and rock mechanics appreciation of the project including compilation of all relevant geometric data into a 3D CAD database using commercial software.

2. Discontinuum finite element (FE) mesh construction using commercial software and in-house scripting tools. Higher-order finite elements were used for all volume elements.

3. Assignment of the geotechnical domains, material properties, initial conditions, boundary conditions and the mining and fill sequence to the FE mesh.

4. Solution of the stress, strain and displacement fields and released energy for each step in the modelled mining sequence using the Abaqus Explicit FE solver. Abaqus Explicit is a commercial, general purpose, 3D, non-linear, continuum or discontinuum FE analysis package designed specifically for analysing problems with significant plasticity, large strain gradients, high deformation levels and large numbers of material domains. Commercial software and in-house post-processing scripts are used to process the Abaqus output and visualise the results.

5. Forecasting of future behaviour for the current LOM plan. Section 3 documents the model results, our interpretation of the results in a mining context and associated discussion.

There is limited data available to enable quantitative model calibration based on observations and measurements. Consequently, this project does not include calibration, except to the extent that that the results are generally consistent with previous geotechnical reports from the Cobar district and our general experience in stoping mines under similar geotechnical conditions.

The Levkovitch-Reusch 2 (LR2) discontinuum constitutive framework was applied in Abaqus to describe the mechanical behaviour of the rockmass and structures. The LR2 framework includes:

1. Three-dimensional (3D) geometry, with the mine excavations sequenced in a sufficient number of separate excavation steps (called frames) to capture the necessary temporal resolution for the project scope.

2. Strain-softening dilatant constitutive model for the rockmass and structures with a generalised Hoek-Brown yield criterion. Different material properties are assigned to each geotechnical domain.

3. Discontinuum formulation using cohesive finite elements to model discrete structures. Cohesive elements are free to dislocate, dilate and degrade and can realistically capture the behaviour of thin structures which tetrahedral finite elements cannot achieve as effectively. The complete interpreted structural model at the required resolution can be included, and where appropriate, can be supplemented with a discrete fracture network (DFN) to improve the structural resolution.

4. Structures less persistent than those modelled explicitly can be represented by "smearing" the effects of structures within the continuum regions of the modelled rockmass.

5. Hydromechanical coupling, where necessary (but not used in this project), to capture the effects of pore water pressure on the rockmass yield surface, or to estimate water flow rates.

The LR2 modelling framework aims for physical similitude, by making the fewest possible assumptions about the governing physics of the entire mine system within a single physics-based numerical model, at the required scale of the analysis. This results in a realistic but complex model, since complexity is the reality of all mines. Building a realistic mine model by including the governing physics means that realistic rockmass behaviour evolves naturally in the model and is therefore essential for developing a detailed understanding of the likely rockmass response to mining.

2.2 Topography

Topographic data supplied to BE by Evolution Mining was used to build the natural surface profile into the numerical model. The natural ground surface directly above the proposed mining footprint at Lake Cowal is predominantly flat or gently sloping. There is no significant topographical relief within the spatial limits of the model.

2.3 Stress field

Lake Cowal has recently undertaken insitu stress measurements using various methods and service providers. Test data was reviewed by MiningOne to establish the insitu stress regime shown in Figure 2-1 and Table 2-1. Some test locations impacted by stress redistribution effects from the pit due to the proximity of the test locations. This was noted in the MiningOne review and tested using a numerical model to assess the impact at each test location. Other test locations are noted as being close to major faults, including the Glenfiddich faults and Cowal shear. The effects of these structures on the stress measurements is unknown.

The insitu stress field derived by MiningOne has a north-south orientated major principal stress. However nearby mines such as Cadia and Northparkes, as well as the Australian stress map (Lee, et al 2008) have an approximately East-West major principal stress orientation (i.e the major and intermediate stress directions for the measured stress field and "regional stress field" are swapped in the proposed insitu stress regime by MiningOne)

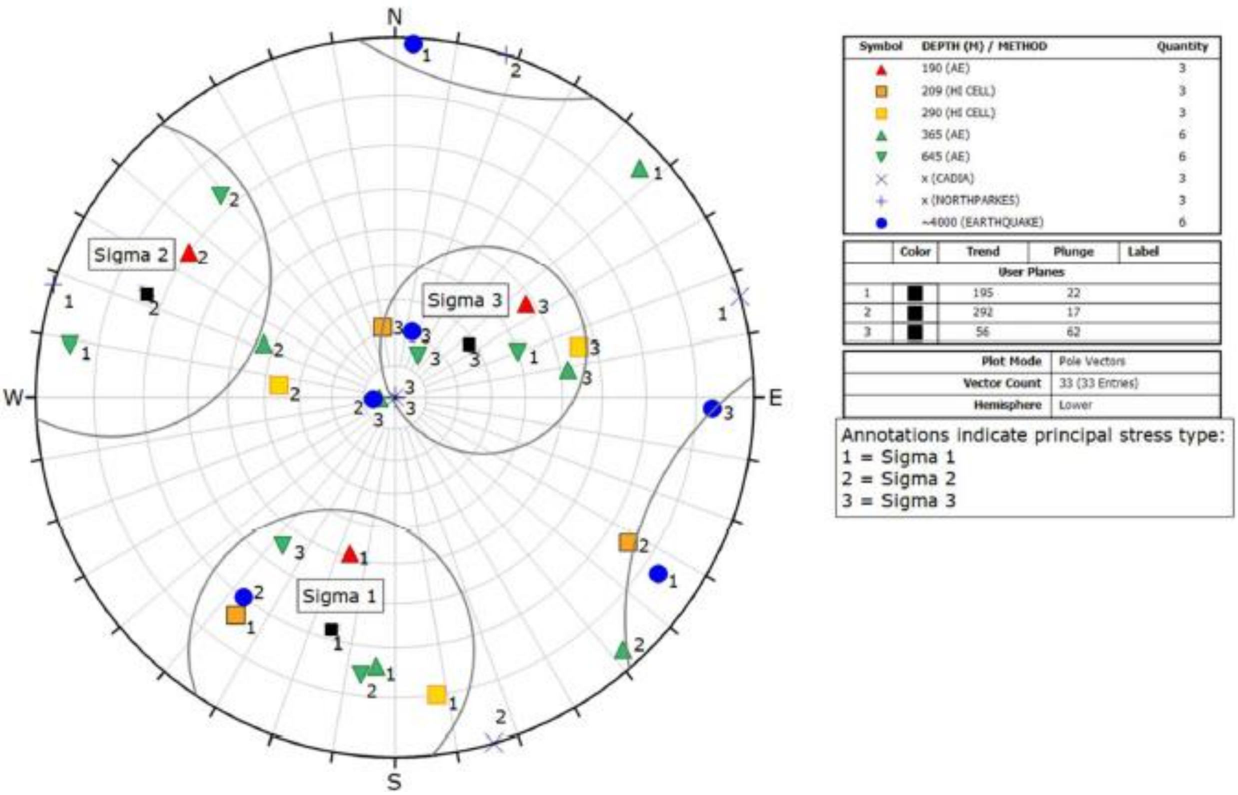


Figure 2-1: Overview of the Lake Cowal stress measurements (after MiningOne, 2021)

Table 2-1: Assumed in-situ stress field S01 for the Lake Cowal mine (after MiningOne, 2021)

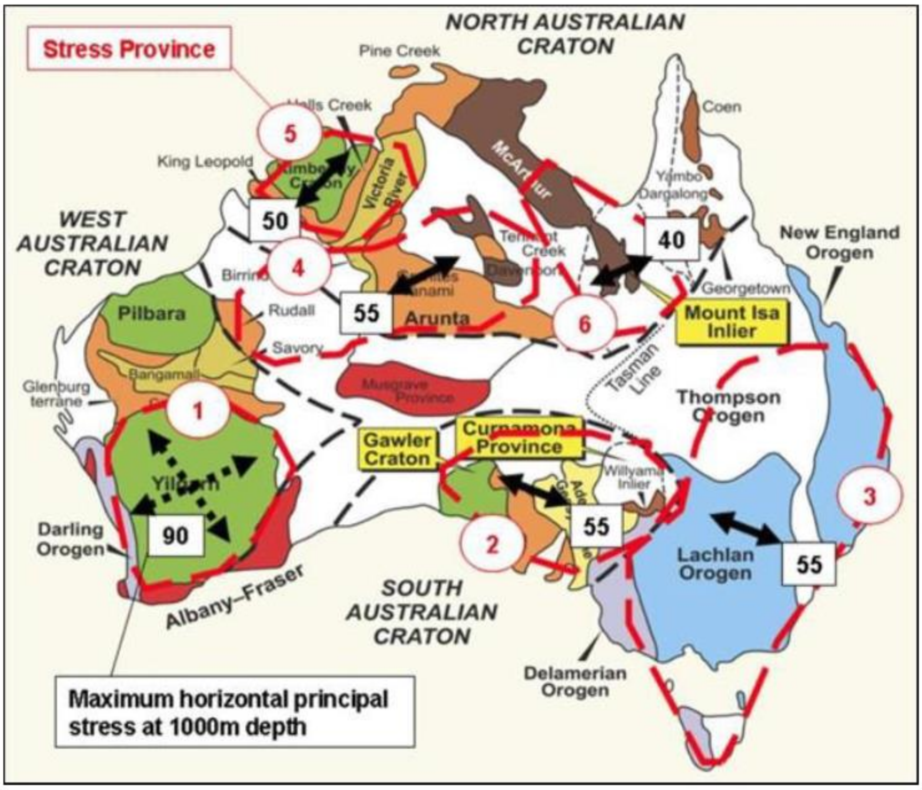
Principal stress component	**Magnitude gradient** (MPa/km)	**Dip** (degrees)	**Bearing** (degrees)
σ_1	53	22	195
σ_2	34	17	292
σ_3	28	62	056

A second stress regime was simulated for the hangingwall decline option (i.e the base case mine design) as a sensitivity evaluation (see Table 2-2). It is noted that a north-south major principal stress direction is favourable for mining (compared to an east-west orientation) as the is lower stress concentration in the stoping abutments and central diminishing pillars.

The alternate stress regime used was matched to the Australian stress map (see Figure 2-2) and the Northparkes stress regime. This stress regime also approximately matches the Cadia stress orientation, and previous open pit assessments by AMC, Itasca and MiningOne, as well as UG assessments by Beck Engineering, at the Lake Cowal mine

The orientation of the major and intermediate stresses are effectively swapped compared to the insitu stress regime derived from site measurements. The major and intermediate stresses are also assumed to be horizontal. This stress regime has a East-West major principal stress and is likely to cause higher stress concentration in pillars in the north-south trending underground mine. The model simulations completed in the assessment include:

- R06 – hangingwall design with East-West regional stress regime
- R07 - hangingwall design with North-South insitu stress
- R08 – S01 footwall design with North-South insitu stress
- R09 – S09 footwall design with North-South insitu stress



	Stress Province	Principal Stresses (at a depth of 1000m, for stress measurements > 500m)	
		σ1 Orientation	Magnitudes (MPa); σ1 : σ2 : σ3
1	Yilgarn	Variable, WSW-ENE (?)	**90** : 50 : 35
2	Gawler-Curnamona	WNW-ESE	**55** : 40 : 30
3	Lachlan	WNW-ESE	**55** : 35 : 30
4	Arunta	WSW-ENE	**55** : 40 : 25
5	Kimberley	SW-NE (?)	**50** : 40 : 25 (?)
6	Mt Isa Inlier	WSW-ENE	**40** : 30 : 20

Figure 2-2: Australian stress regime (after Lee, et al 2008)

Table 2-2: Assumed in-situ stress field S02 for the Lake Cowal mine (used for the sensitivity assessment)

Principal stress component	**Magnitude gradient** (MPa/km)	**Dip** (degrees)	**Bearing** (degrees)
σ_1	55	0	290
σ_2	37	0	200
σ_3	27	90	0

2.4 Geotechnical domain assignment

The material properties have been applied according to the lithology. This domaining approach is a necessary assumption in the absence of a separate detailed geotechnical domain model, but from our general understanding of rock mass conditions at the Lake Cowal mine, this assumption is appropriate.

Geology wireframes were provided for both the underground and open pit precincts and material properties for each domain were applied to each wireframe accordingly. A cross section through the geology model showing proposed underground mining is provided in Figure 2-3.

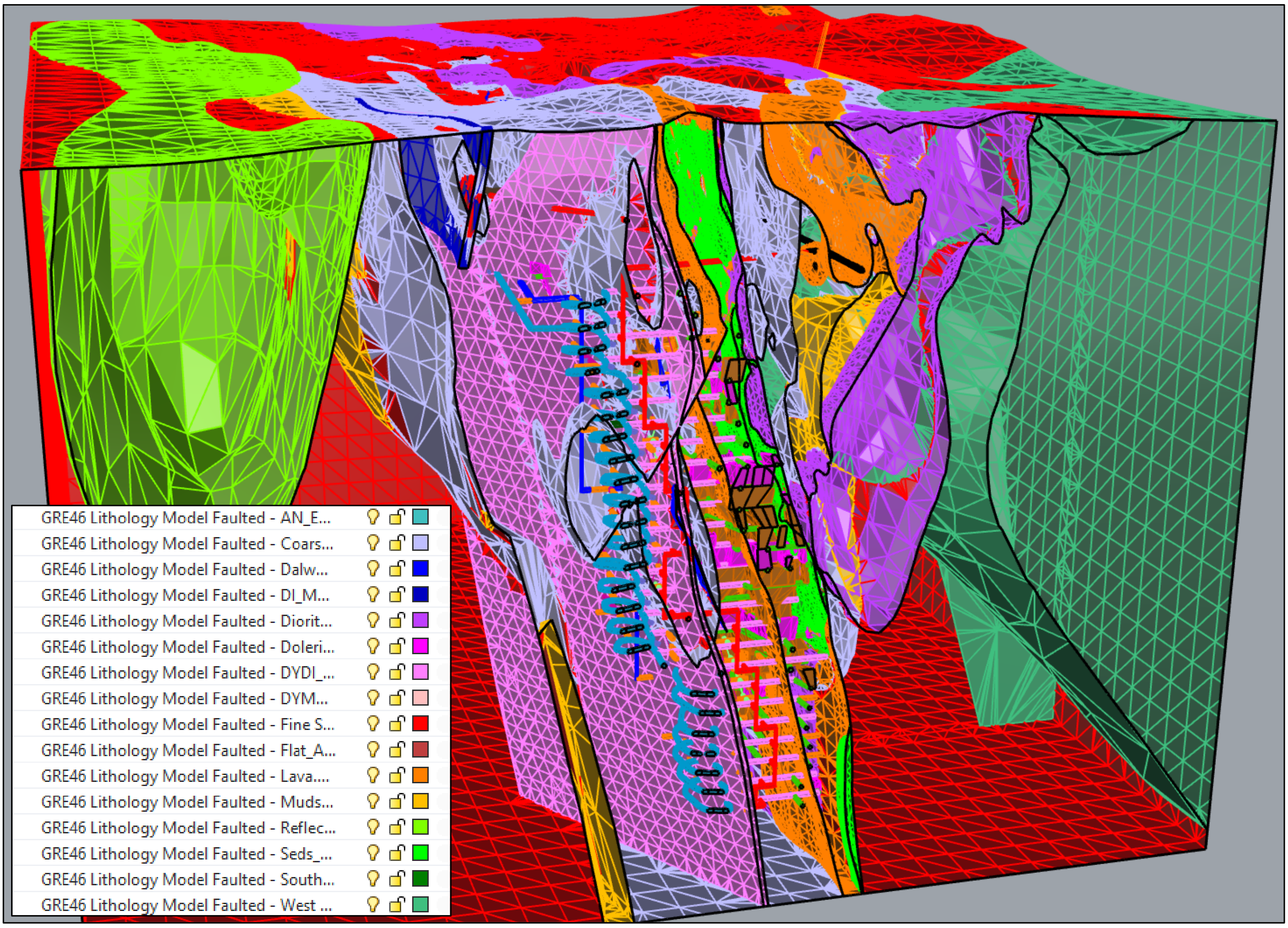


Figure 2-3: Perspective view to the northeast showing the main rockmass domains in the model.

2.4.1 Estimated material properties for modelling

The material properties derived by these methods are given in Table 2-3. Derivation of material properties by a calibration process is preferred over this approach. However, given that suitable data is not yet available for calibration, the estimation process is necessary at this stage of the project. All values were defined in consultation with the Lake Cowal Geotechnical Engineers.

The following nomenclature is used in Table 2-3:

UCS = uniaxial compressive strength.
GSI = geological strength index.
n, s = strength adjustment parameters for anisotropic behaviour.
ϵ_0 = 0 = plastic strain at start of peak strength stage (see Figure 2-4).
ϵ_1 = plastic strain at start of transitional strength stage (see Figure 2-4).
ϵ_2 = plastic strain at start of residual strength stage (see Figure 2-4).
E = Young's modulus for the rockmass.
ν = Poisson's ratio for the rockmass.
s, m, a = generalised HB yield parameters for the rockmass.
d = rockmass dilation parameter.

Table 2-3: Material property set M01 used in the FE model.

LR2-Materials

Name	ρ [kg/m³]	UCSi [MPa]	GSI	n_{aniso}	s_{aniso}	Level	ε_{plast} [%]	E [GPa]	v	s	m_b	a	e	dilation
HOX	2000.00	15	30	1.00	1.00	PEAK	0.00	4.25	0.25	6.34E-5	0.32	0.53	0.60	0.05
						RES	30.84	4.24	0.25	1.00E-5	0.32	0.53	0.60	0.00
AND	2770.00	125	65	1.00	1.00	PEAK	0.00	26.27	0.25	2.10E-3	1.97	0.50	0.60	0.33
						TRANS	0.96	22.53	0.25	5.82E-4	1.34	0.50	0.60	0.22
						RES	7.32	19.38	0.25	1.00E-5	0.67	0.50	0.60	0.00
DIOR	2800.00	132	60	1.00	1.00	PEAK	0.00	25.17	0.25	1.27E-3	1.75	0.51	0.60	0.29
						TRANS	0.52	23.14	0.25	6.12E-4	1.40	0.51	0.60	0.23
						RES	6.56	19.80	0.25	1.00E-5	0.70	0.51	0.60	0.00
DIPP	2770.00	132	60	1.00	1.00	PEAK	0.00	25.17	0.25	1.27E-3	1.75	0.51	0.60	0.29
						TRANS	0.52	23.14	0.25	6.12E-4	1.40	0.51	0.60	0.23
						RES	6.56	19.80	0.25	1.00E-5	0.70	0.51	0.60	0.00
DYDI_E42	2770.00	132	60	1.00	1.00	PEAK	0.00	25.17	0.25	1.27E-3	1.75	0.51	0.60	0.29
						TRANS	0.52	23.14	0.25	6.12E-4	1.40	0.51	0.60	0.23
						RES	6.56	19.80	0.25	1.00E-5	0.70	0.51	0.60	0.00
E_VOL	2770.00	127	66	1.00	1.00	PEAK	0.00	26.79	0.25	2.32E-3	2.04	0.50	0.60	0.34
						TRANS	1.01	22.71	0.25	5.91E-4	1.36	0.50	0.60	0.23
						RES	7.27	19.50	0.25	1.00E-5	0.68	0.50	0.60	0.00
L_VOL	2770.00	127	75	1.00	1.00	PEAK	0.00	30.12	0.25	5.70E-3	2.68	0.50	0.60	0.45
						TRANS	1.67	22.71	0.25	5.91E-4	1.36	0.50	0.60	0.23
						RES	7.94	19.50	0.25	1.00E-5	0.68	0.50	0.60	0.00
U_VOL	2700.00	127	74	1.00	1.00	PEAK	0.00	29.74	0.25	5.16E-3	2.60	0.50	0.60	0.43
						TRANS	1.60	22.71	0.25	5.91E-4	1.36	0.50	0.60	0.23
						RES	7.86	19.50	0.25	1.00E-5	0.68	0.50	0.60	0.00
BRX	2800.00	132	40	1.00	1.00	PEAK	0.00	20.83	0.25	1.72E-4	0.96	0.52	0.60	0.16
						RES	2.77	19.80	0.25	1.00E-5	0.70	0.52	0.60	0.00
LAVA	2760.00	159	70	1.00	1.00	PEAK	0.00	30.57	0.25	3.46E-3	2.66	0.50	0.60	0.44
						TRANS	0.96	25.08	0.25	7.13E-4	1.65	0.50	0.60	0.28
						RES	5.99	21.11	0.25	1.00E-5	0.78	0.50	0.60	0.00
MONZ	2800.00	132	75	1.00	1.00	PEAK	0.00	30.58	0.25	5.70E-3	2.74	0.50	0.60	0.46
						TRANS	1.59	23.14	0.25	6.12E-4	1.40	0.50	0.60	0.23
						RES	7.63	19.80	0.25	1.00E-5	0.70	0.50	0.60	0.00
AND_UG	2770.00	125	65	1.00	1.00	PEAK	0.00	26.27	0.25	2.10E-3	1.97	0.50	0.60	0.33
						TRANS	0.96	22.53	0.25	5.82E-4	1.34	0.50	0.60	0.22
						RES	7.32	19.38	0.25	1.00E-5	0.67	0.50	0.60	0.00
C_SED_UG	2700.00	145	68	1.00	1.00	PEAK	0.00	28.90	0.25	2.83E-3	2.36	0.50	0.60	0.39
						TRANS	0.96	24.15	0.25	6.64E-4	1.53	0.50	0.60	0.25
						RES	6.46	20.49	0.25	1.00E-5	0.74	0.50	0.60	0.00
D_LAVA_UG	2700.00	168	69	1.00	1.00	PEAK	0.00	30.67	0.25	3.13E-3	2.67	0.50	0.60	0.45
						TRANS	0.83	25.61	0.25	7.42E-4	1.74	0.50	0.60	0.29
						RES	5.59	21.46	0.25	1.00E-5	0.81	0.50	0.60	0.00
DIOR_UG	2900.00	155	69	1.00	1.00	PEAK	0.00	29.94	0.25	3.13E-3	2.54	0.50	0.60	0.42
						TRANS	0.93	24.83	0.25	7.00E-4	1.62	0.50	0.60	0.27
						RES	6.09	20.95	0.25	1.00E-5	0.77	0.50	0.60	0.00

Table 2-4: Material property set M01 used in the FE model (continued)

LR2-Materials

Name	ρ [kg/m³]	UCSi [MPa]	GSI	n_{aniso}	s_{aniso}	Level	ε_{plast} [%]	E [GPa]	v	s	m_b	a	e	dilation
DOL_UG	2800.00	132.00	75.00	1.00	1.00	PEAK	0.00	30.58	0.25	5.70E-3	2.74	0.50	0.60	0.46
						TRANS	1.59	23.14	0.25	6.12E-4	1.40	0.50	0.60	0.23
						RES	7.63	19.80	0.25	1.00E-5	0.70	0.50	0.60	0.00
MONZ_UG	2800.00	132.00	75.00	1.00	1.00	PEAK	0.00	30.58	0.25	5.70E-3	2.74	0.50	0.60	0.46
						TRANS	1.59	23.14	0.25	6.12E-4	1.40	0.50	0.60	0.23
						RES	7.63	19.80	0.25	1.00E-5	0.70	0.50	0.60	0.00
F_SED_UG	2800.00	127.00	66.00	1.00	1.00	PEAK	0.00	26.79	0.25	2.32E-3	2.04	0.50	0.60	0.34
						TRANS	1.01	22.71	0.25	5.91E-4	1.36	0.50	0.60	0.23
						RES	7.27	19.50	0.25	1.00E-5	0.68	0.50	0.60	0.00
LAVA_UG	2700.00	168.00	69.00	1.00	1.00	PEAK	0.00	30.67	0.25	3.13E-3	2.67	0.50	0.60	0.45
						TRANS	0.83	25.61	0.25	7.42E-4	1.74	0.50	0.60	0.29
						RES	5.59	21.46	0.25	1.00E-5	0.81	0.50	0.60	0.00
MSTONE_UG	2700.00	70.00	67.00	1.00	1.00	PEAK	0.00	19.99	0.25	2.56E-3	1.54	0.50	0.60	0.26
						TRANS	2.34	15.70	0.25	2.96E-4	0.80	0.50	0.60	0.13
						RES	12.41	14.36	0.25	1.00E-5	0.50	0.50	0.60	0.00
FAULTS_LRX	2200.00	50.00	40.00	1.00	1.00	PEAK	0.00	12.07	0.25	1.72E-4	0.59	0.52	0.60	0.10
						RES	14.92	11.47	0.25	1.00E-5	0.43	0.52	0.60	0.00
HOST	2700.00	130.00	65.00	1.00	1.00	PEAK	0.00	26.68	0.25	2.10E-3	2.01	0.50	0.60	0.34
						TRANS	0.90	22.97	0.25	6.04E-4	1.39	0.50	0.60	0.23
						RES	7.03	19.68	0.25	1.00E-5	0.69	0.50	0.60	0.00

MOHR-COULOMB-like Materials

Name	ρ [kg/m³]	n_{aniso}	s_{aniso}	Level	ε_{plast} [%]	E [GPa]	v	cohesion [kPa]	ϕ [°]	dilation
TRANS	1850.00	1.00	1.00	PEAK	0.00	0.40	0.30	26.00	26.70	0.25
SOX	1950.00	1.00	1.00	PEAK	0.00	0.40	0.30	28.00	24.00	0.25
FAULTS_MC	2200.00	1.00	1.00	PEAK	0.00	7.50	0.25	85.00	38.00	0.25

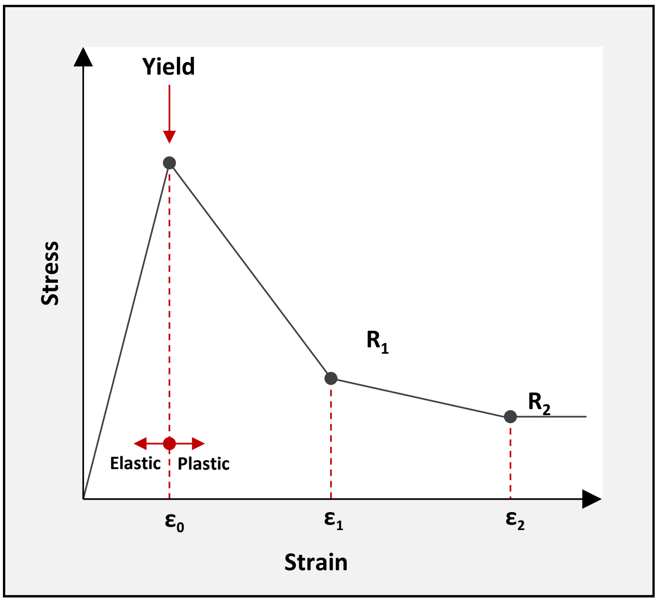


Figure 2-4: Indicative rockmass softening curve demonstrating the plastic strain transition points ϵ_1 and ϵ_2.

2.5 Hydrogeological conditions

No detailed information on the current hydrogeology or planned mine de-watering strategy was available to BE for this project. We have therefore ignored potential groundwater effects for this analysis and applied a fully drained constitutive formulation.

2.6 Mining geometry & sequence

The numerical model included the complete LOM geometry and current geological/structural model for the proposed Lake Cowal Mine, comprising the following:

- Main decline, production level accesses, ore drives and other miscellaneous tunnel development,
- Stopes
- Ventilation shafts,
- Major geotechnical/lithological domains and, major geological structures

All mine development and stope geometry was included within the model following the extraction sequence defined by Evolution Mining. The model steps and corresponding mining dates in that schedule are listed in Table 2-5. A total of 85 individual mining extraction steps were simulated in the model. This level of temporal detail allows the evolution of the stress path, rock mass damage and associated displacements to be modelled to a high level of realism, taking account of the gradual process of void creation and filling as the mineral extraction progresses.

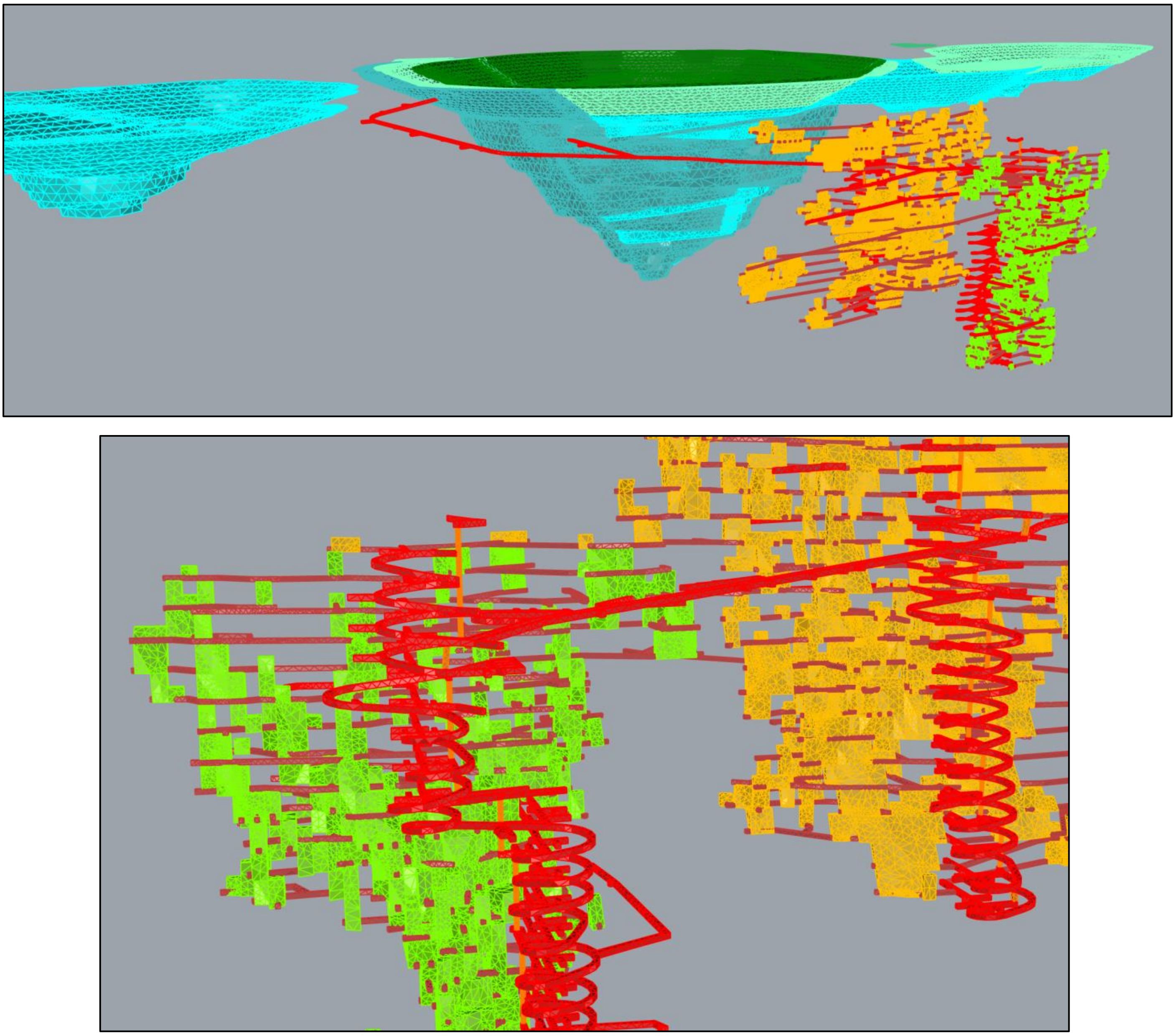


Figure 2-5: Mining geometry built in the model (note the small satellite pits were built in the model, but not simulated as being mined in the planned mining schedule)

Table 2-5. Model frames and corresponding mining dates.

Frame No	Frame Name	Frame No	Frame Name	Frame No	Frame Name
1	Y2007_Q1	31	Y2024_Q2	61	Y2031_Q4
2	Y2008_Q1	32	Y2024_Q3	62	Y2032_Q1
3	Y2009_Q1	33	Y2024_Q4	63	Y2032_Q2
4	Y2010_Q1	34	Y2025_Q1	64	Y2032_Q3
5	Y2011_Q1	35	Y2025_Q2	65	Y2032_Q4
6	Y2012_Q1	36	Y2025_Q3	66	Y2033_Q1
7	Y2013_Q1	37	Y2025_Q4	67	Y2033_Q2
8	Y2014_Q1	38	Y2026_Q1	68	Y2033_Q3
9	Y2015_Q1	39	Y2026_Q2	69	Y2033_Q4
10	Y2016_Q1	40	Y2026_Q3	70	Y2034_Q1
11	Y2017_Q1	41	Y2026_Q4	71	Y2034_Q2
12	Y2018_Q1	42	Y2027_Q1	72	Y2034_Q3
13	Y2019_Q2	43	Y2027_Q2	73	Y2034_Q4
14	Y2020_Q1	44	Y2027_Q3	74	Y2035_Q1
15	Y2020_Q2	45	Y2027_Q4	75	Y2035_Q2
16	Y2020_Q3	46	Y2028_Q1	76	Y2035_Q3
17	Y2020_Q4	47	Y2028_Q2	77	Y2035_Q4
18	Y2021_Q1	48	Y2028_Q3	78	Y2036_Q1
19	Y2021_Q2	49	Y2028_Q4	79	Y2036_Q2
20	Y2021_Q3	50	Y2029_Q1	80	Y2036_Q3
21	Y2021_Q4	51	Y2029_Q2	81	Y2036_Q4
22	Y2022_Q1	52	Y2029_Q3	82	Y2037_Q1
23	Y2022_Q2	53	Y2029_Q4	83	Y2037_Q2
24	Y2022_Q3	54	Y2030_Q1	84	Y2037_Q3
25	Y2022_Q4	55	Y2030_Q2	85	Y2037_Q4
26	Y2023_Q1	56	Y2030_Q3		
27	Y2023_Q2	57	Y2030_Q4		
28	Y2023_Q3	58	Y2031_Q1		
29	Y2023_Q4	59	Y2031_Q2		
30	Y2024_Q1	60	Y2031_Q3		

2.7 Stope filling methodology & fill properties

In the model, stopes to be mined in frame i starting at time t_i are excavated over the period t_i to t_i + 0.1s by ramping down the Young's modulus from the rockmass value to the void value of 100 kPa. Stopes are filled at the end of the frame (at $t_i + 3.0$s) by setting the elastic constants of the stope void to fill properties. In practice, the mine could leave stopes open for longer than modelled and may not always achieve tight filling.

For this project, the following elastic constants were applied for stope fill:

- Young's modulus $E_{\text{fill}} = 200$ MPa.
- Poisson's ratio $\nu_{\text{fill}} = 0.20$.

2.8 Structural resolution of the model

Evolution Mining provided BE with a number of digital 3D CAD files which contained wireframes of several major geological structures that intersect the Lake Cowal underground and adjacent host rock geotechnical domains. These geological structures were built into the model explicitly as discontinuum components using traction-separation based cohesive elements. Cohesive elements allow simulation of the discrete behaviour associated with faults and shears. In BE's LR2 constitutive model, faults and shears are free to dislocate and dilate and the fault surfaces can dilate and degrade. The major geological structures provided to BE for the modelling project are shown in Figure 2-6 and Figure 2-7.

The resolution of the available structural information allows mine-scale interpretations of the model results. This means that average strains across the rockmass between modelled structures can be simulated and interpreted, but local strains due to structures smaller than those modelled explicitly cannot develop in the model. To obtain forecasts of potential peak strains, which may be needed to assess the potential for locally high deformation levels around individual stopes for example, a model incorporating structures with persistence smaller than the scale of the excavations themselves would be needed. Information on intermediate or small-scale structural features is not currently available to this analysis, given the very early stage of the geotechnical characterisation the Lake Cowal deposit.

With the current model, we therefore cannot forecast the stability of individual stopes, because stope stability forecasts depend largely on stope-scale structures. Likewise, we cannot forecast the stability of individual drives because such forecasts depend on drive-scale structures. The model does allow general interpretations of stope and drive stability based on, for example, forecast deformation arising from weaker rockmass conditions, adverse geometric configurations and sequences, but explicit forecasts are not possible without greater detail on the rock mass characteristics.

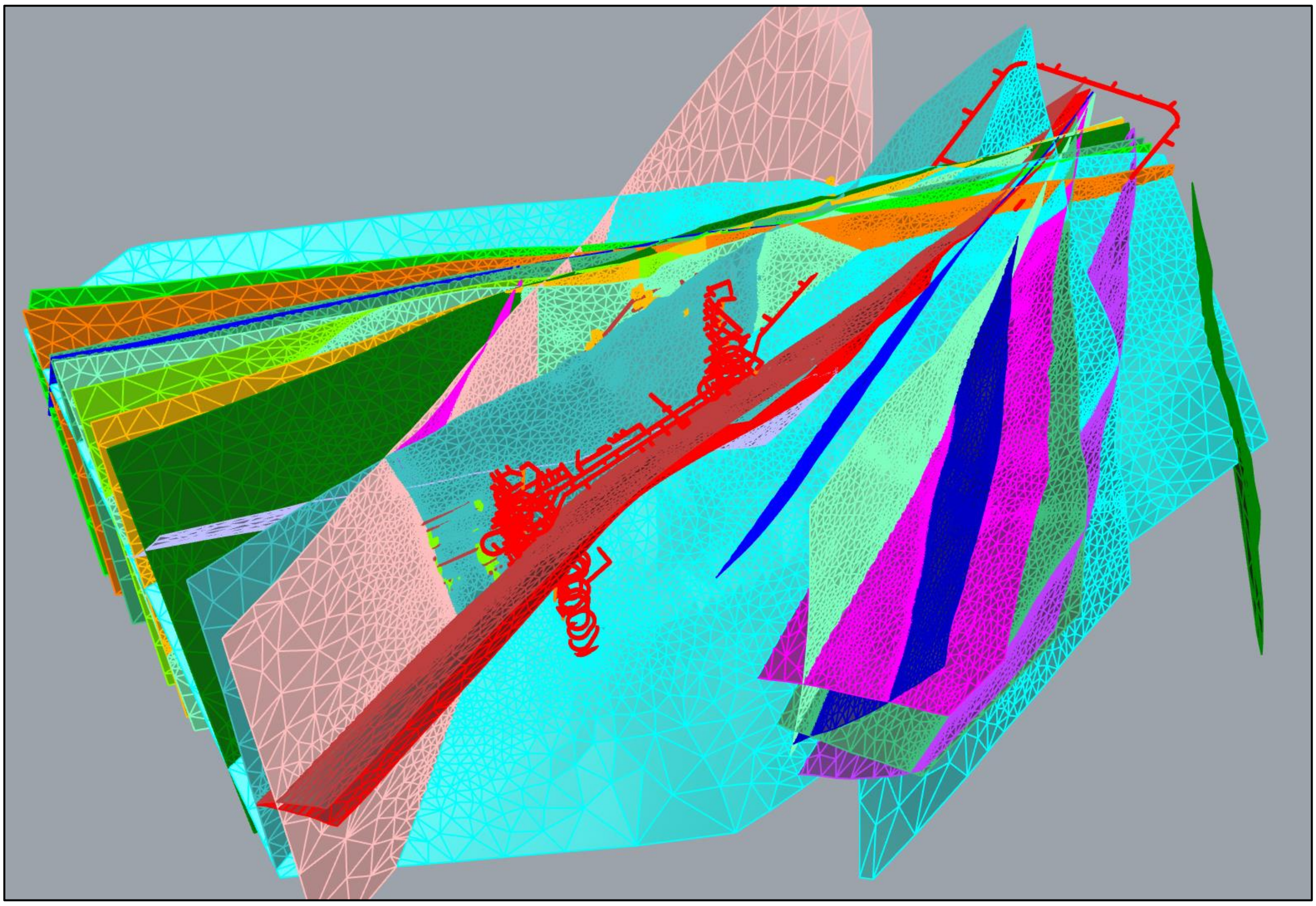


Figure 2-6 Faults included in the numerical model

E42_Lithology_Model_-_Bowmore
E42_Lithology_Model_-_Bulldog
E42_Lithology_Model_-_Central_F
E42_Lithology_Model_-_Cross_Hou
E42_Lithology_Model_-_E41_Cowal
E42_Lithology_Model_-_Wamboyne
E42_Lithology_Model_-_Western_F
E42_Lithology_Model_-_Whisky_Fa
E42_Lithology_Model_-_Wyrra

GRE46 Lithology Model Faulted - Ardbeg.dxf
GRE46 Lithology Model Faulted - Cowal_FW.dxf
GRE46 Lithology Model Faulted - Cowal_HW.dxf
GRE46 Lithology Model Faulted - Galway Splay 1.dxf
GRE46 Lithology Model Faulted - Galway Splay 2.dxf
GRE46 Lithology Model Faulted - Galway Splay 3.dxf
GRE46 Lithology Model Faulted - Galway Splay 4.dxf
GRE46 Lithology Model Faulted - Galway Splay 5.dxf
GRE46 Lithology Model Faulted - Galway Splay 6.dxf
GRE46 Lithology Model Faulted - Galway Splay 7.dxf
GRE46 Lithology Model Faulted - Galway Splay 8.dxf
GRE46 Lithology Model Faulted - Glennfiddich revised.dxf
GRE46 Lithology Model Faulted - Jura.dxf
GRE46 Lithology Model Faulted - Lava Dyke Fault.dxf
GRE46 Lithology Model Faulted - Lower Manna.dxf
GRE46 Lithology Model Faulted - Structural Divide West.dxf
GRE46 Lithology Model Faulted - Upper Manna.dxf

Figure 2-7 Faults included in the numerical model

2.9 Mine design scenarios

The conceptual geometry of the proposed mining options at Lake Cowal are illustrated on the following pages.

The model geometry is colour coded by calendar year according to the preliminary extraction sequence for the mine. Evolution personnel provided BE with a detailed monthly extraction sequence spanning 2021-2035 and this has been used without simplification as the basis for the numerical simulation sequence.

This report documents our assessment of the fourth mining scenario (only), in addition to the previous assessments of the original three mine plans provided. The reader should refer to the original report and model results presentations for the main findings and recommendations for design options 1-3. The mine designs for options 1-3 are provided here fore reference only.

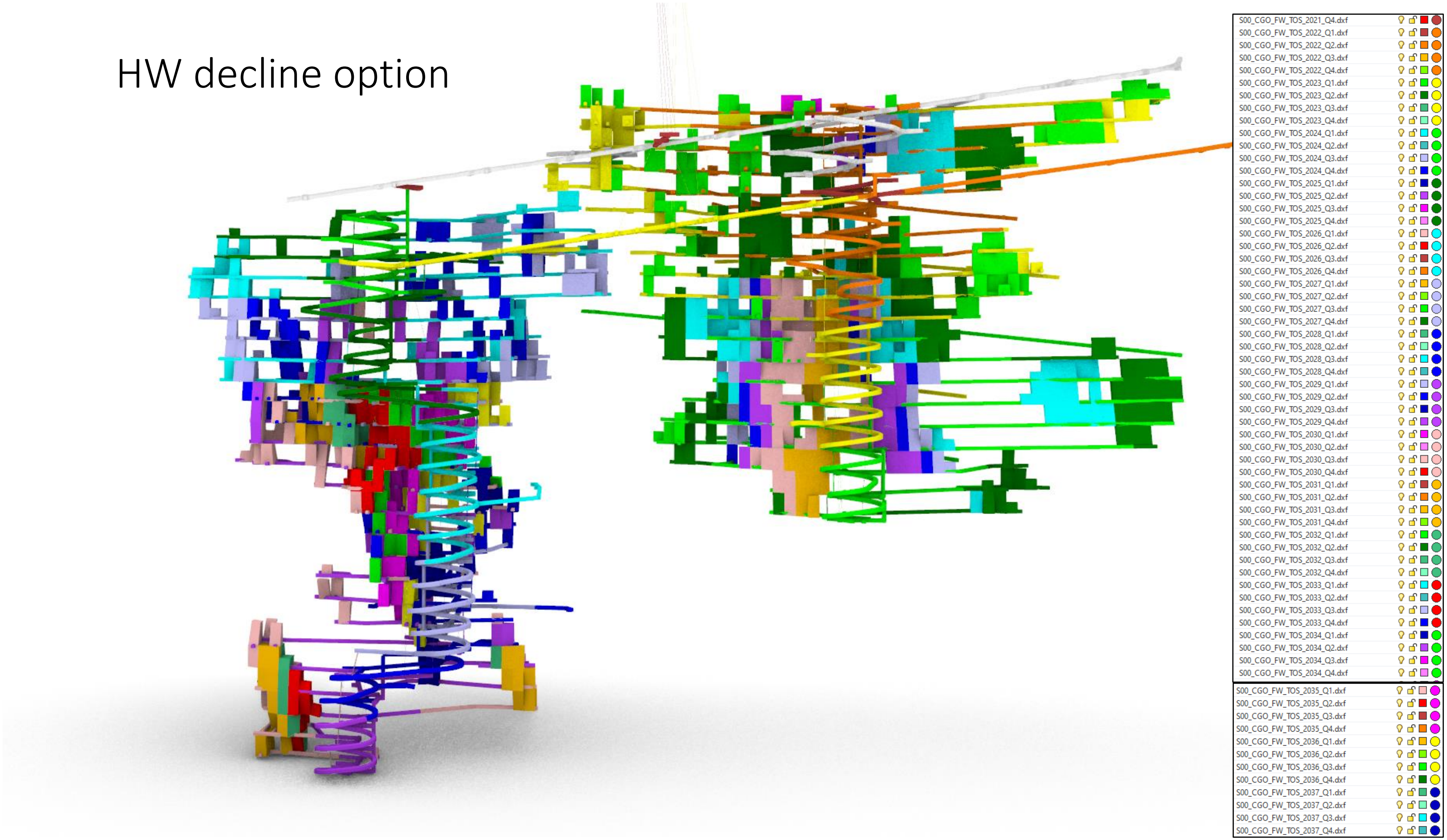


Figure 2-8: Option 1 - Hangingwall decline

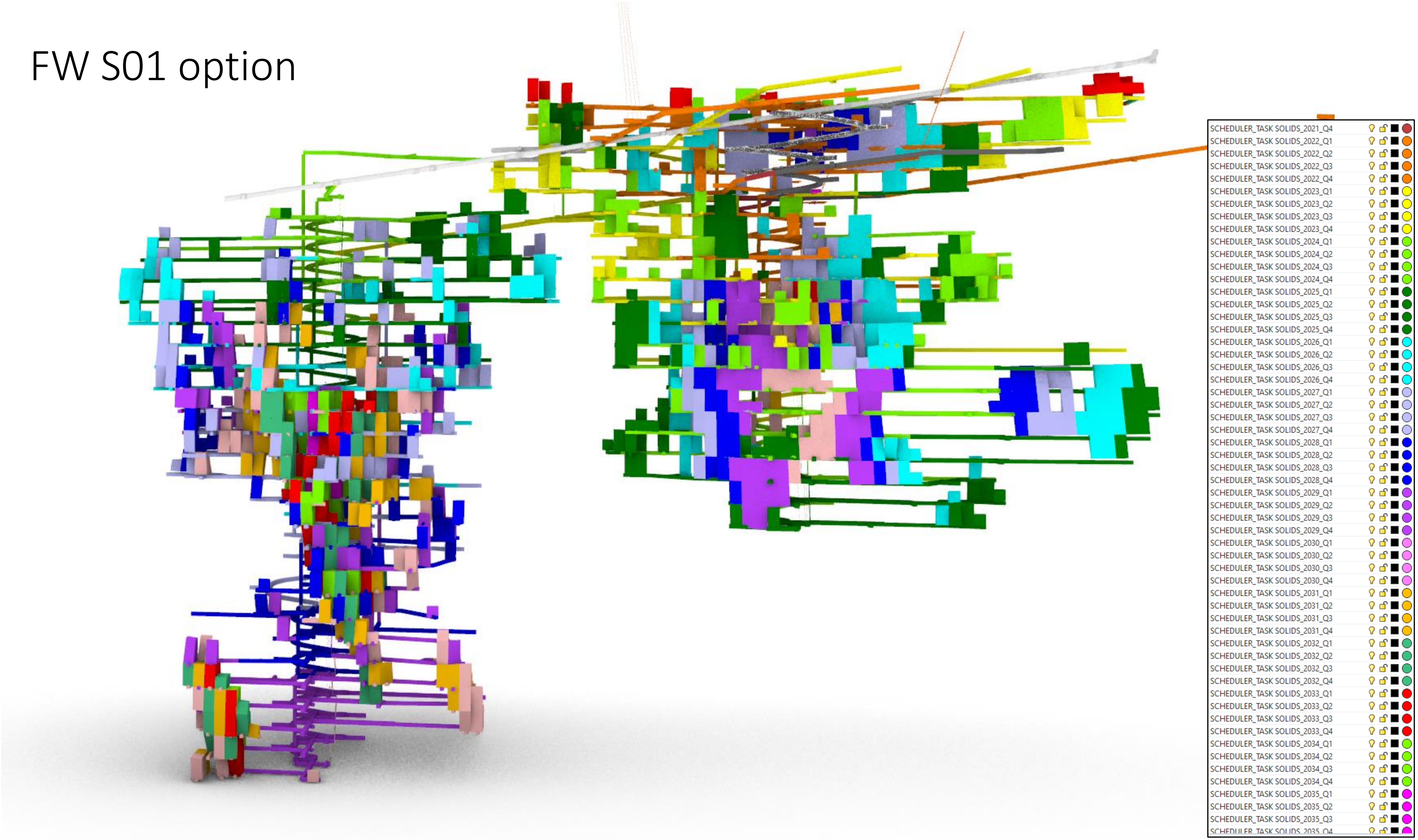


Figure 2-9: Option 2 – Footwall decline S01

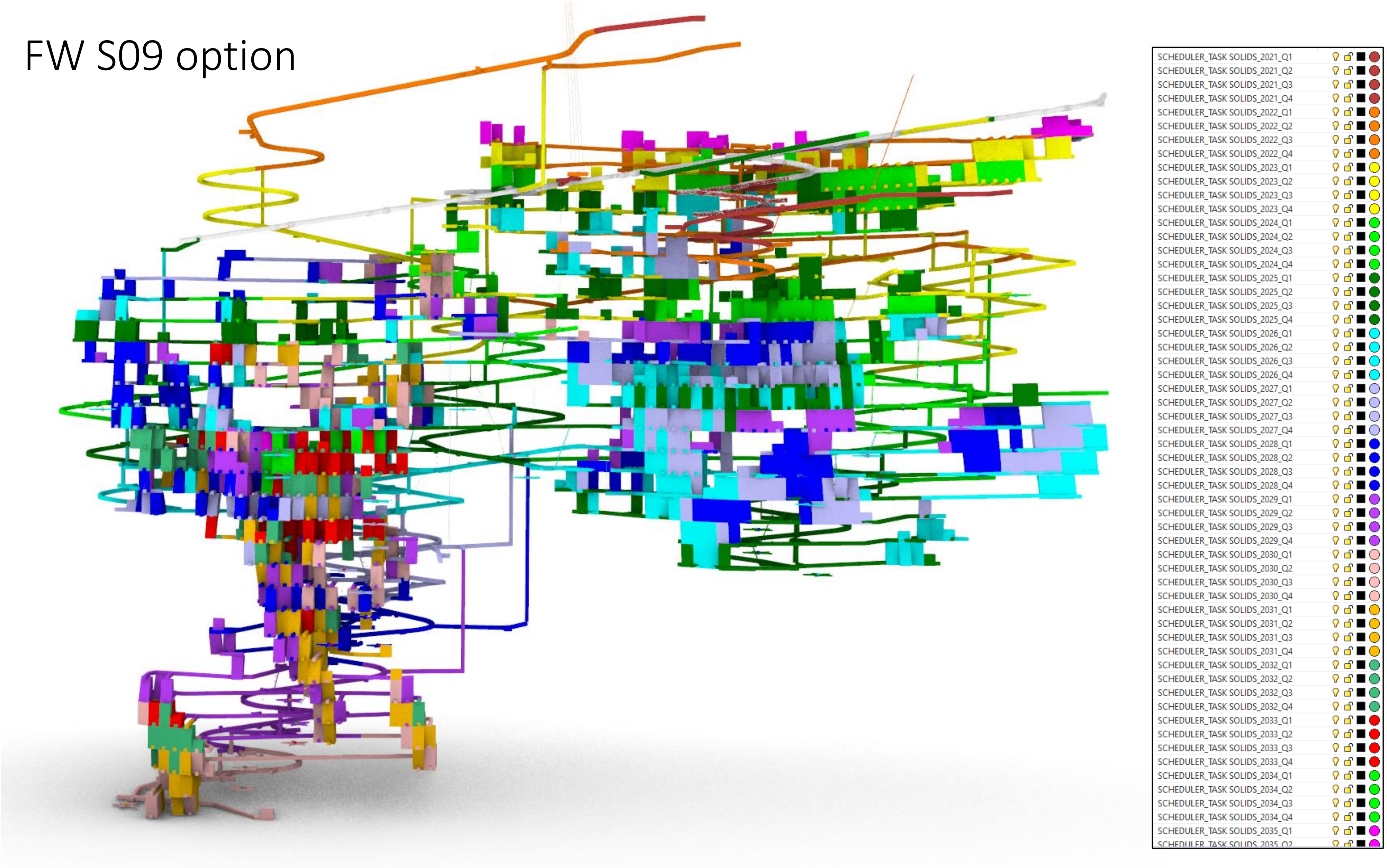


Figure 2-10 Option 3 – Footwall decline S09

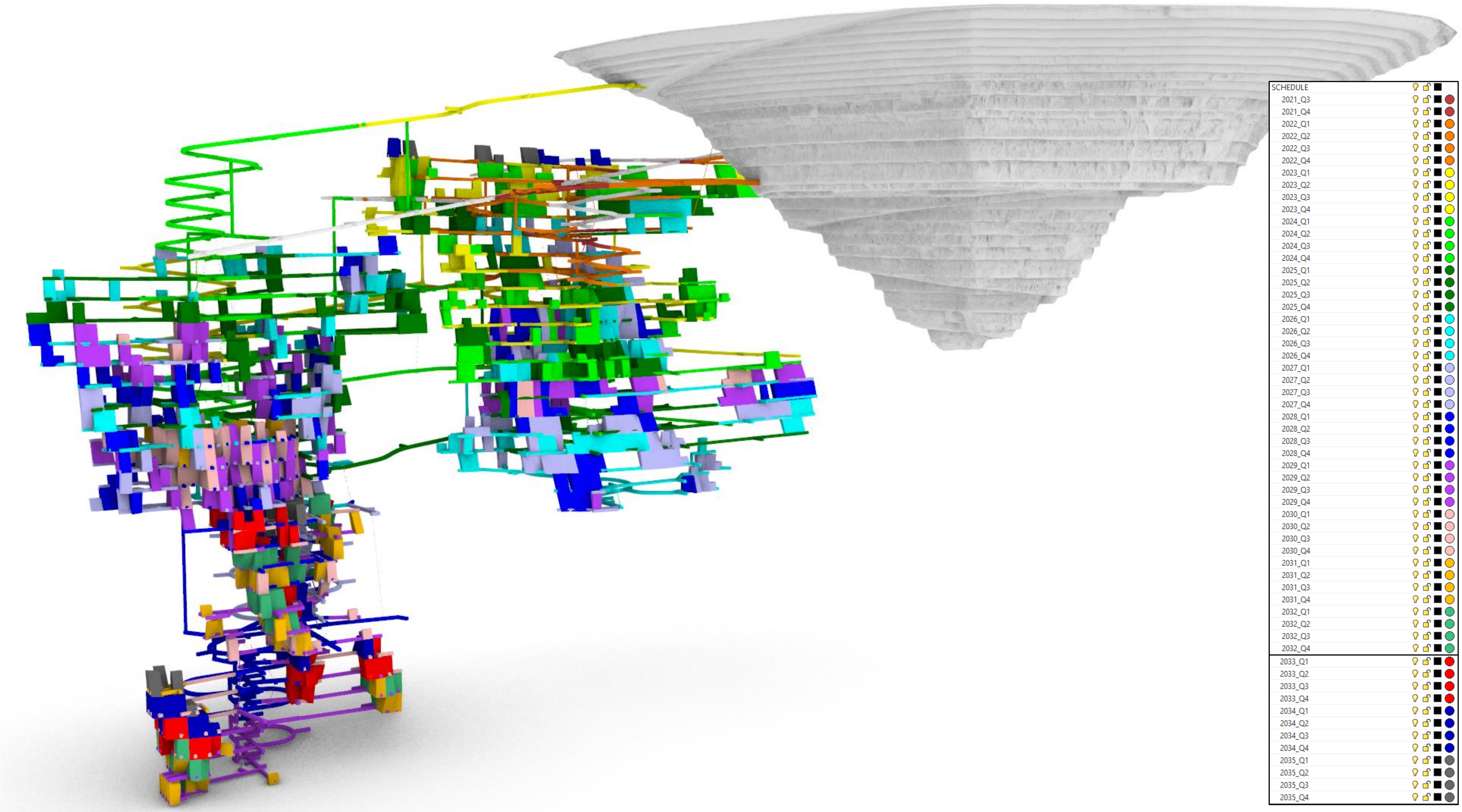


Figure 2-11 Design option 4 (facing east)

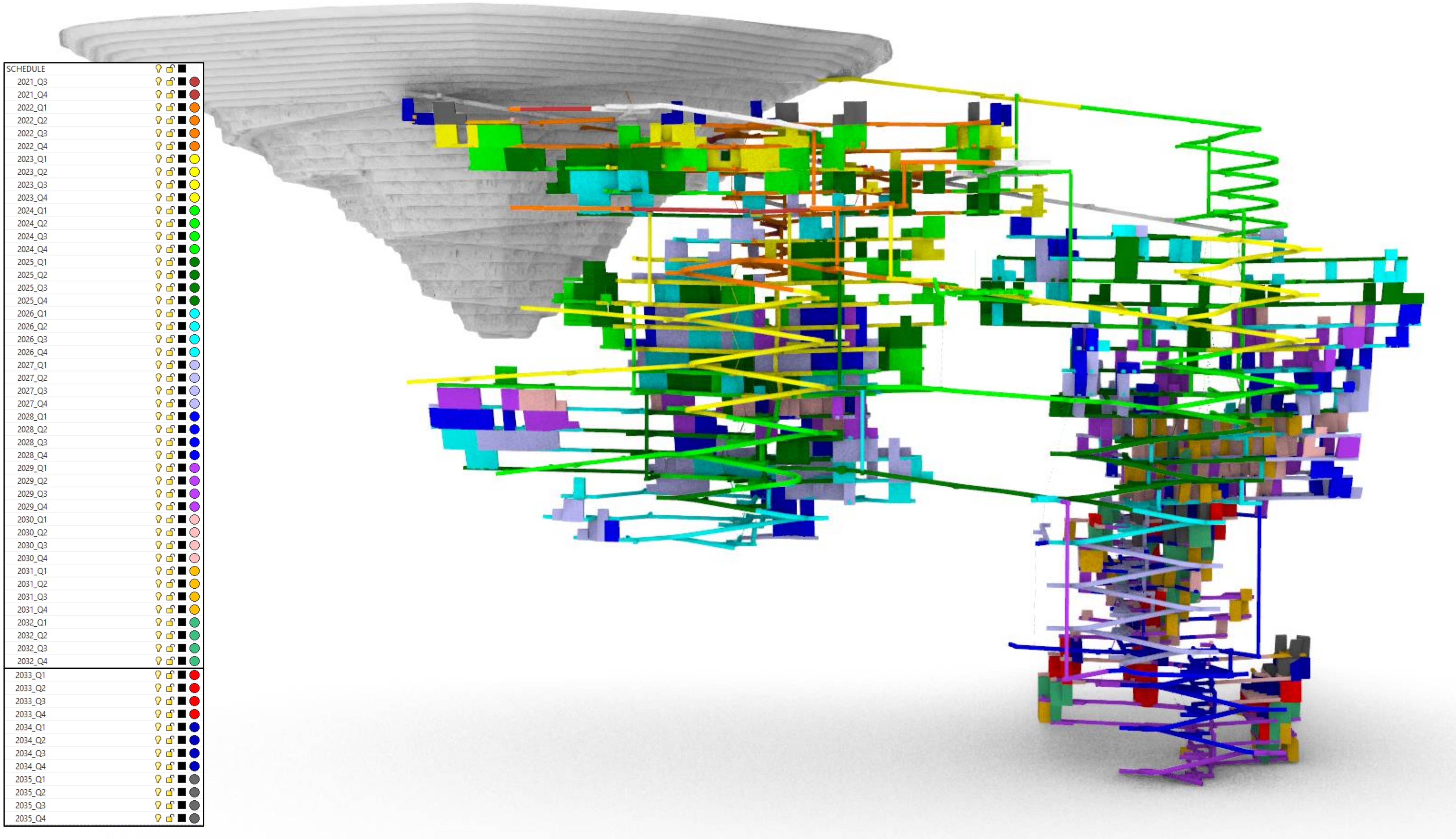


Figure 2-12 Design option 4 (facing west)

(

3 FORECASTS, INTERPRETATION & DISCUSSION

This section provides an assessment of the proposed Lake Cowal design geometry and numerical simulation results. The mining geometry and numerical results are assessed for any indications of global or excavation scale instabilities, as far as the resolution of the model allows, given the available input data at this time. Detailed discussion of the numerical forecasts of mining induced rock mass damage, displacement and stress affecting key mining infrastructure are presented in the following sections, with suggested improvement opportunities relevant to the proposed mine design.

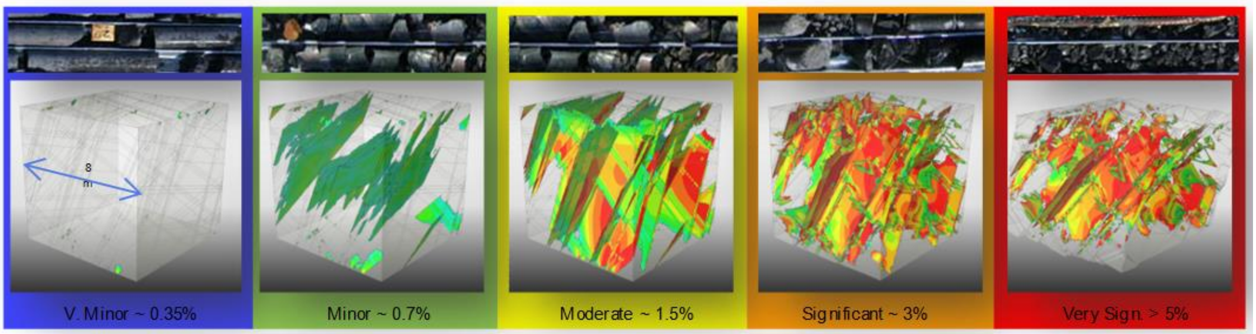
Mechanisms of rock mass instability can occur at various spatial scales. *Global Mine Scale* instabilities can develop over time as a result of progressive creation of large voids, such as the stope mining of large individual lodes or the orebody as a whole. Adverse impacts on mine infrastructure such as high strains and displacements can potentially occur many tens if not hundreds of metres from the production cavities in the extreme case, especially where large scale geological structures are present. The outcome for the excavations depends on the mining geometry, geology and rock mass strength characteristics. Assessment of global instability potential in terms of a quantitative mechanical response of the rock mass (strain, displacement & stress) adjacent the important mine infrastructure is critical, as the adverse consequences of large-scale instabilities can have very significant implications for the continuity of mining activity. An example of global mine scale instability would be the large-scale change in shear stress on major geological structures around a large open stoping block, causing shear failure on the structures extending deep into the region of permanent mine infrastructure. *Excavation Scale* instabilities may also have significant consequences, but they occur on a smaller spatial scale. An example of an excavation scale instability would be a single or small group of stopes causing elevated strains through the main decline, leading to ground support damage, seismicity or convergence of the excavation.

3.1 Rockmass damage scale

BE's rockmass damage scale is shown in Figure 3-1. Rockmass damage is plotted on a logarithmic scale called logP, where $\mathrm{logP} = \log_{10}(1000\epsilon_p + 1)$ and ϵ_p is the deviatoric equivalent plastic strain. This allows a wide range of plastic strain magnitudes to be plotted with a convenient linear colour scale. The damage scale in terms of stress and strain is shown in Figure 3-2. In stoping mines such as Lake Cowal:

1. Minor rockmass damage indicates a low likelihood of instability.
2. Moderate rockmass damage indicates an increased likelihood of instability, particularly in stope hanging walls and crowns.
3. Significant rockmass damage is characterised by relatively high frequency of instability, leading to reduced recovery productivity, higher dilution, increased ground support rehabilitation and associated mining costs.
4. Very significant rockmass damage is characterised by severe stability problems for open stopes and development and this often necessitates alternative mining methods.

It is essential to note that these damage categories are indicative only. Persistent structures present at length scales below the inherent resolution of the model are likely to exist and these would strongly influence the stability of both development and production mining excavations.



logP ≈ 0.65 logP ≈ 0.90 logP ≈ 1.2 logP ≈ 1.5 logP > 1.7

Figure 3-1: Rockmass damage scale.

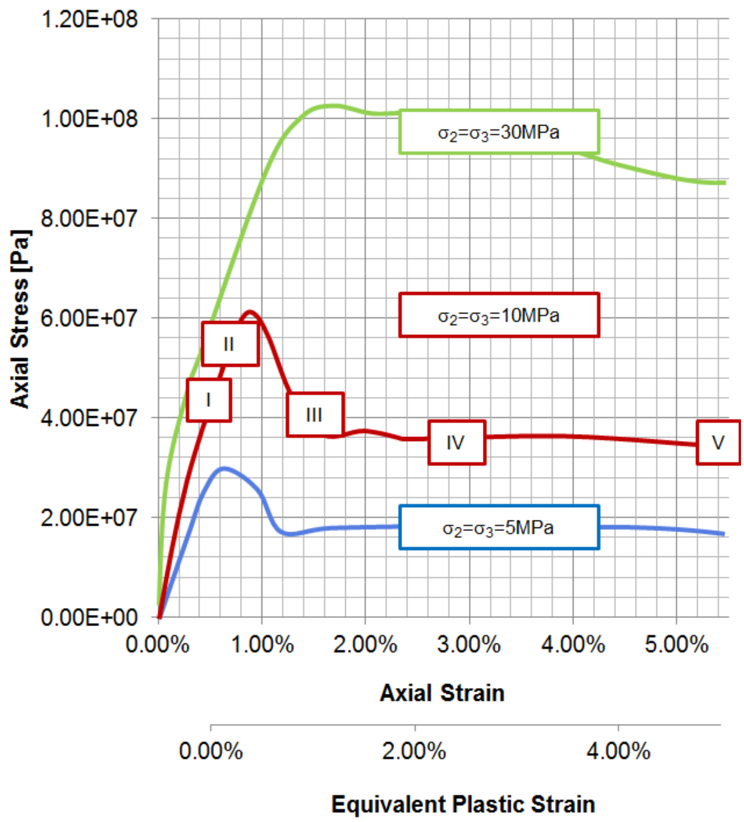


Figure 3-2: Stress vs. Strain chart showing corresponding rock mass damage levels.

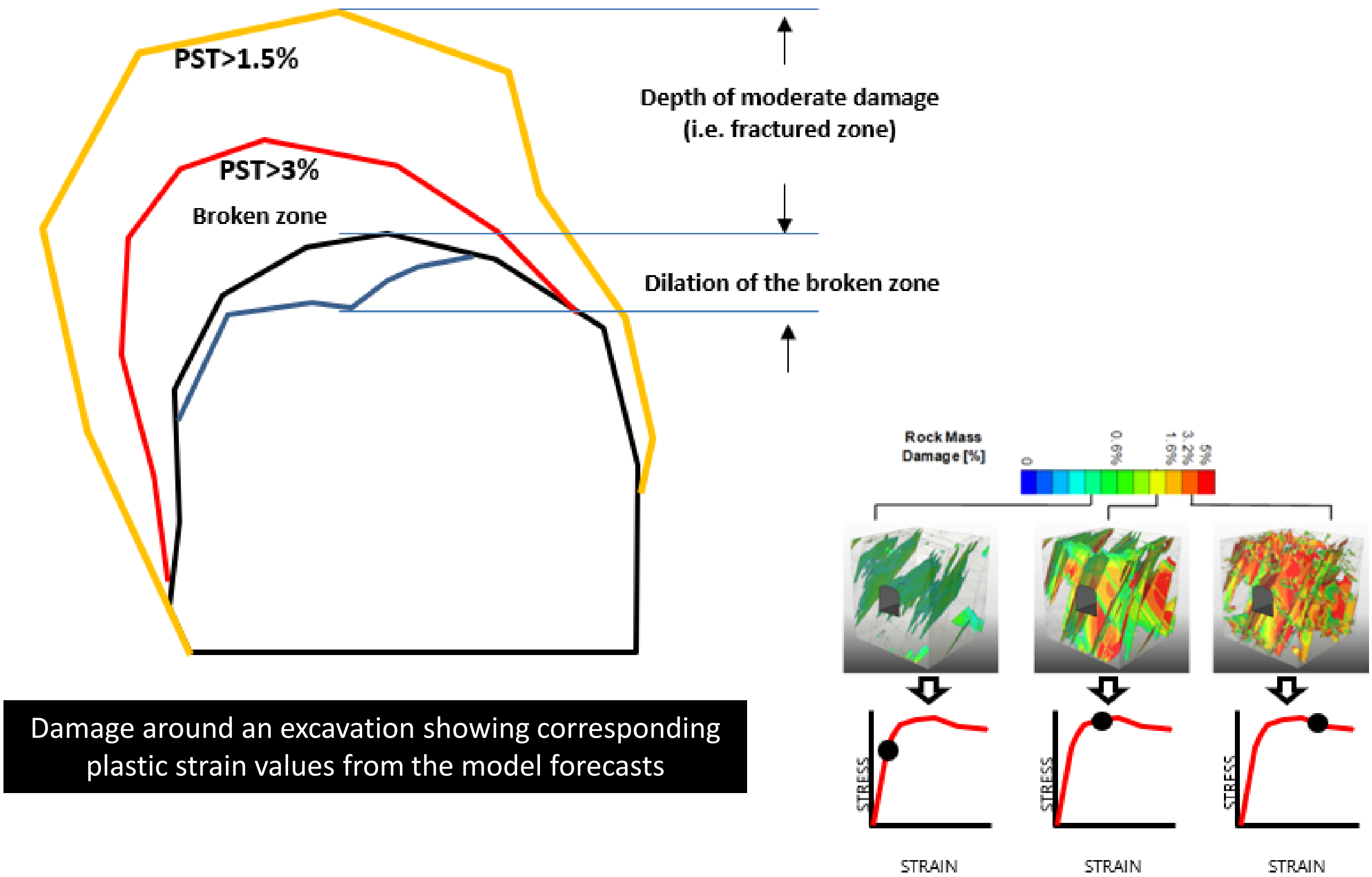


Figure 3-3: Example of rockmass damage and damage to the rockmass in the perimeter of a drive

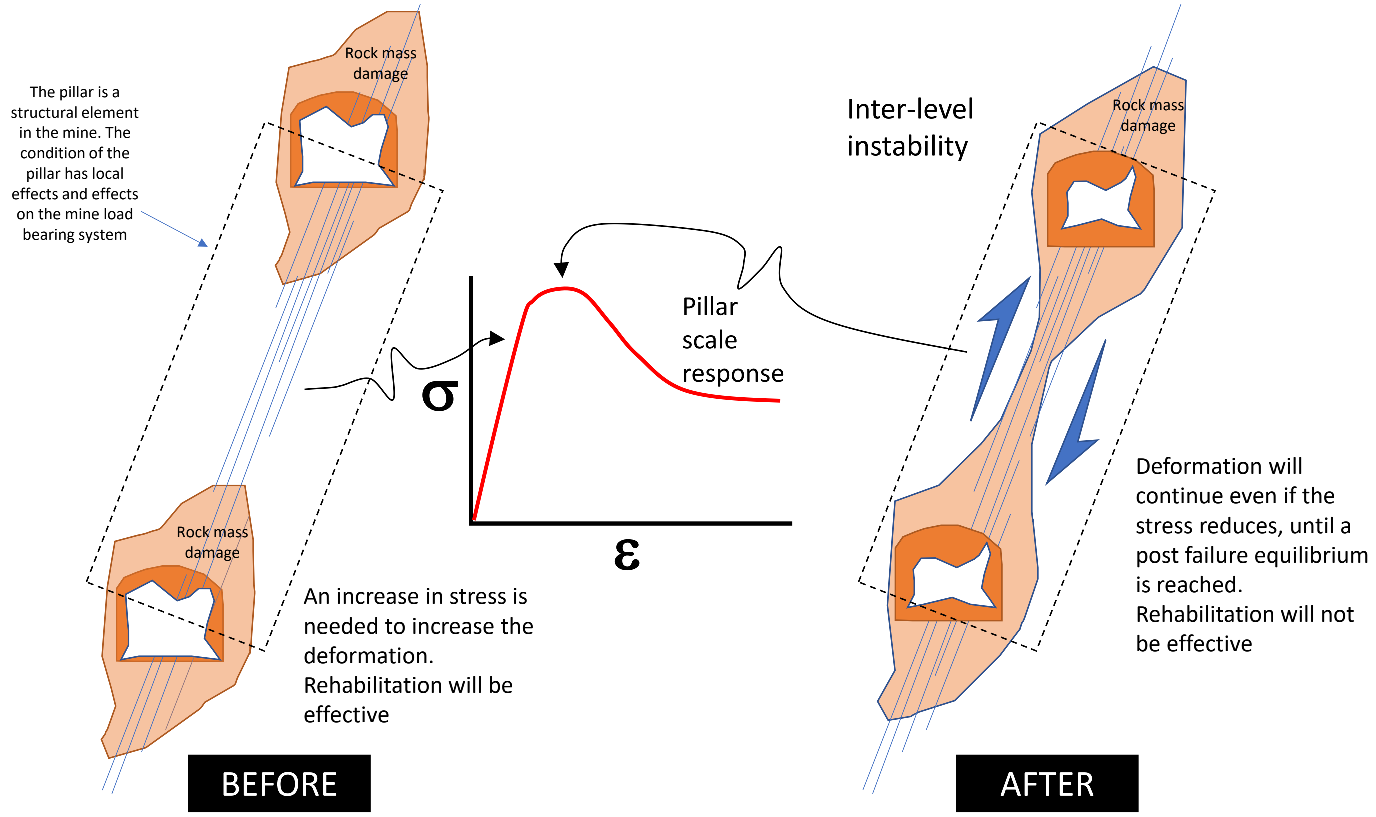


Figure 3-4: Damage mechanism for drives on neighboring levels intersected by a fault

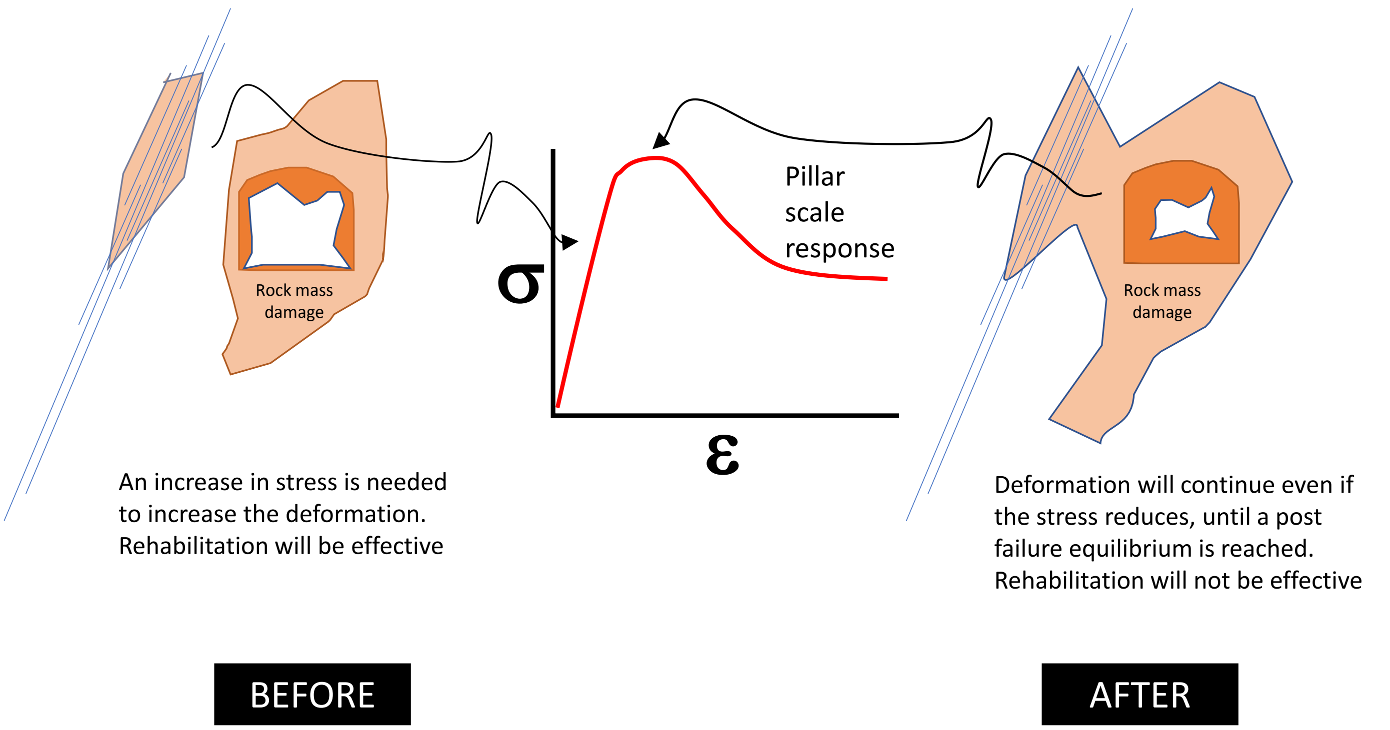


Figure 3-5: Damage mechanism for drives in proximity to a fault or major defect

3.2 Design option #4

A review of the Design Option #4 mine plan identified the design has the following features in the proposed sequence (see Figure 3-6 and Figure 3-22

- Longitudinal stoping with some transverse access in thicker sections of the orebody
- Continuous stoping in multiple overhand panels
- Multiple Galway splay faults intersecting the declines and level accesses. It is noted that declines placed in almost any location with interact with faults. This is also true for the level accesses (Figure 3-7 to Figure 3-12)
- Diminishing sill pillars between each overhand panel. However the size on the levels impacted is relatively small due to the boundaries of each panel and the barren areas between planned stopes and stoping blocks.
- Diminishing central pillar in the mid region of each panel (see Figure 3-14 to Figure 3-16).

Model forecasts for the Design option 4 are illustrated in Figure 3-23 to Figure 3-51. The main findings from the model forecasts for this option were:

- The mine layout and sequence is generally similar to the FW S01 mine plan
- Generally isolated areas of minor to moderate rockmass damage (only) for early and mid stages of mining in both the north and south panels
- Moderate stress concentration occurs in the diminishing central pillar (~30-40 MPa) in the South panel. Stress concentration is limited due to the orientation of the major principal stress and depth of mining. The increase in stress in the diminishing pillar is not sufficiently high to cause significant damage in production areas during extraction of the stopes in the diminishing pillar.
- Stress concentration of ~50-60 MPa is forecast in the central diminishing pillar in the mid section of the North panel. The stress is not sufficiently high to cause significant or extensive rockmass damage throughout the central pillar. However, this level of stress is sufficiently high (>0.4 x UCS) to cause seismicity in the pillar
- Moderate damage occurs in the secondary stopes in the thicker mid section of the North panel (i.e. the pillars between the primary stopes). These stopes have elevated potential for crown instability, which would impact the ability to re-access the overcut drives and cause production delays during rehabilitation. See Figure 3-34 to Figure 3-39.
- Primary-secondary stoping with long (vertical) lead-lags is not recommended in the lower North panel due to rockmass damage caused in the secondaries. A more continuous overhand sequence is recommended (similar to the sequence adopted in the HW decline option
- Some stopes mined in the middle of the north panel are close to the main access cause minor to moderate damage locally to the access drive. Recommend to delay these stopes and mine them as production retreats back to the access and FW drive. See Figure 3-40.
- There are stopes mined out of sequence in both north and south panels. These out of sequence stopes mine out areas along the level access prior to other stopes being mined. A more continuous / progressive retreat sequence towards the level access is recommended. It is likely these out of sequence stopes are minor scheduling errors only, and would be corrected in future design iterations

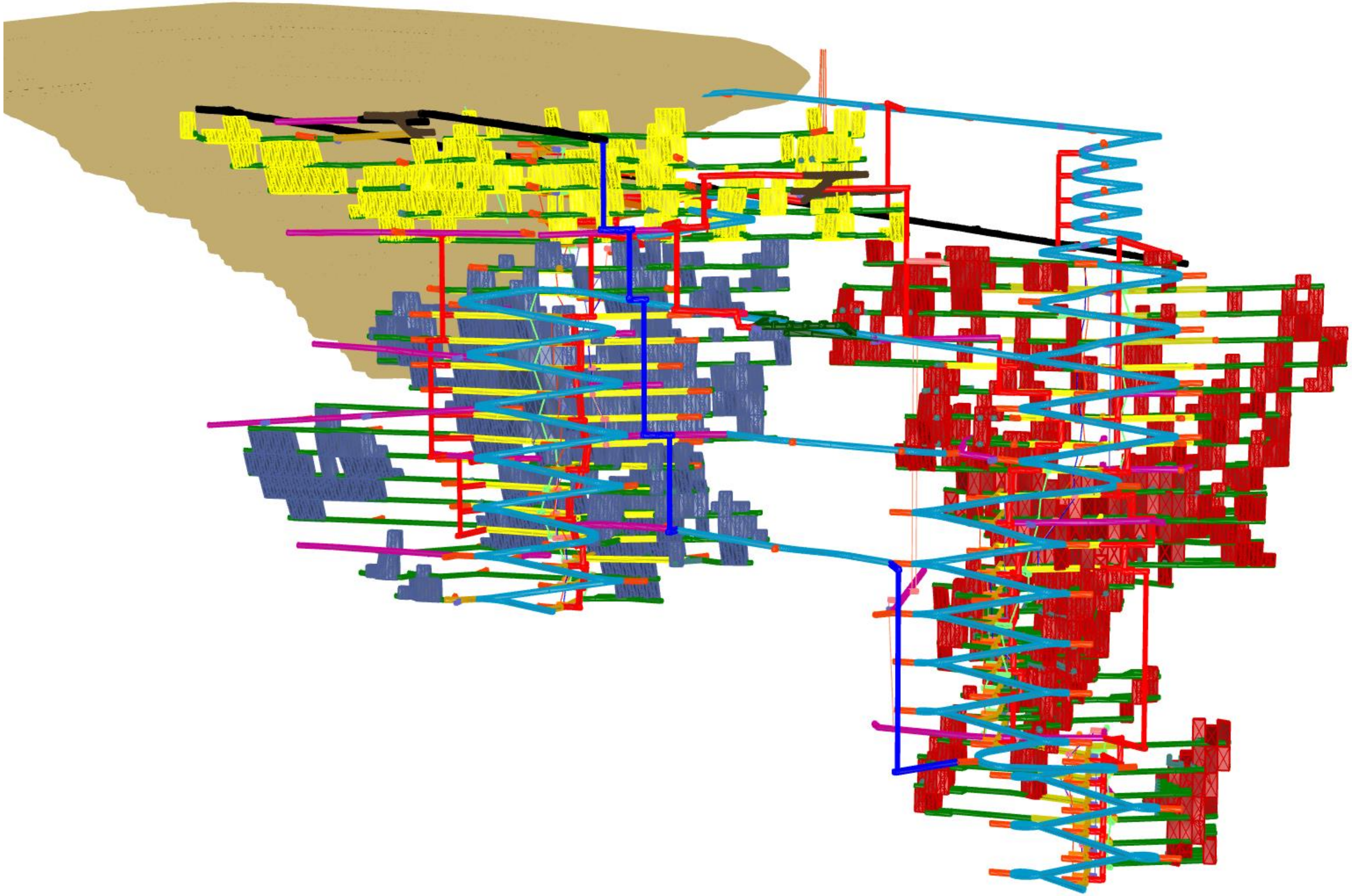
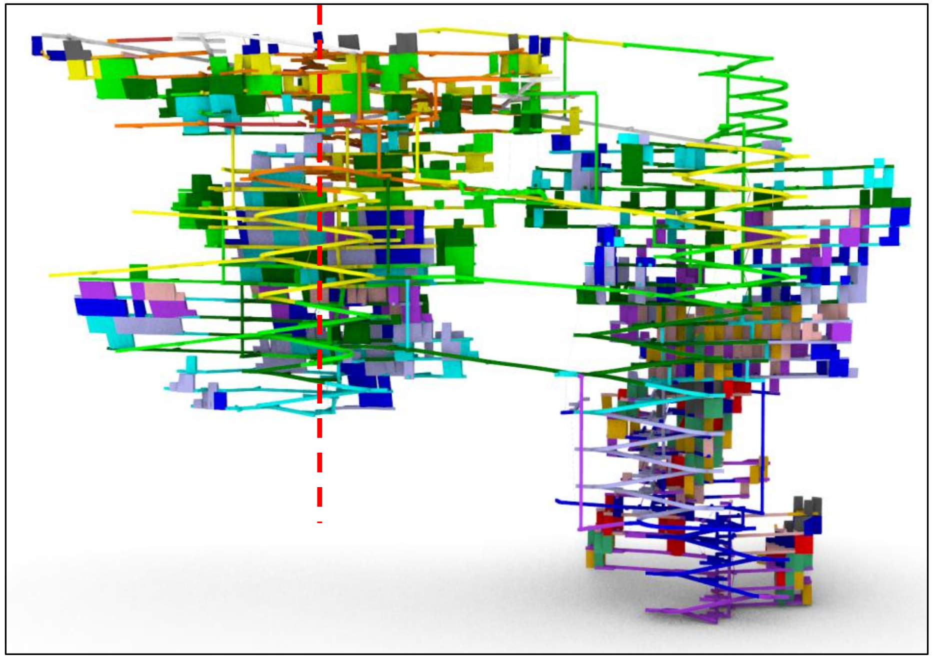


Figure 3-6: Perspective view of the of Design option 4.

South decline fault interaction

- Galway splays intersect some decline loops.
- Level accesses cross the Galway splay faults at an angle that is generally favourable (close to perpendicular)
- Some intersections and north-south orientation infrastructure drives are unfavourably located with respect to the Galway splay faults



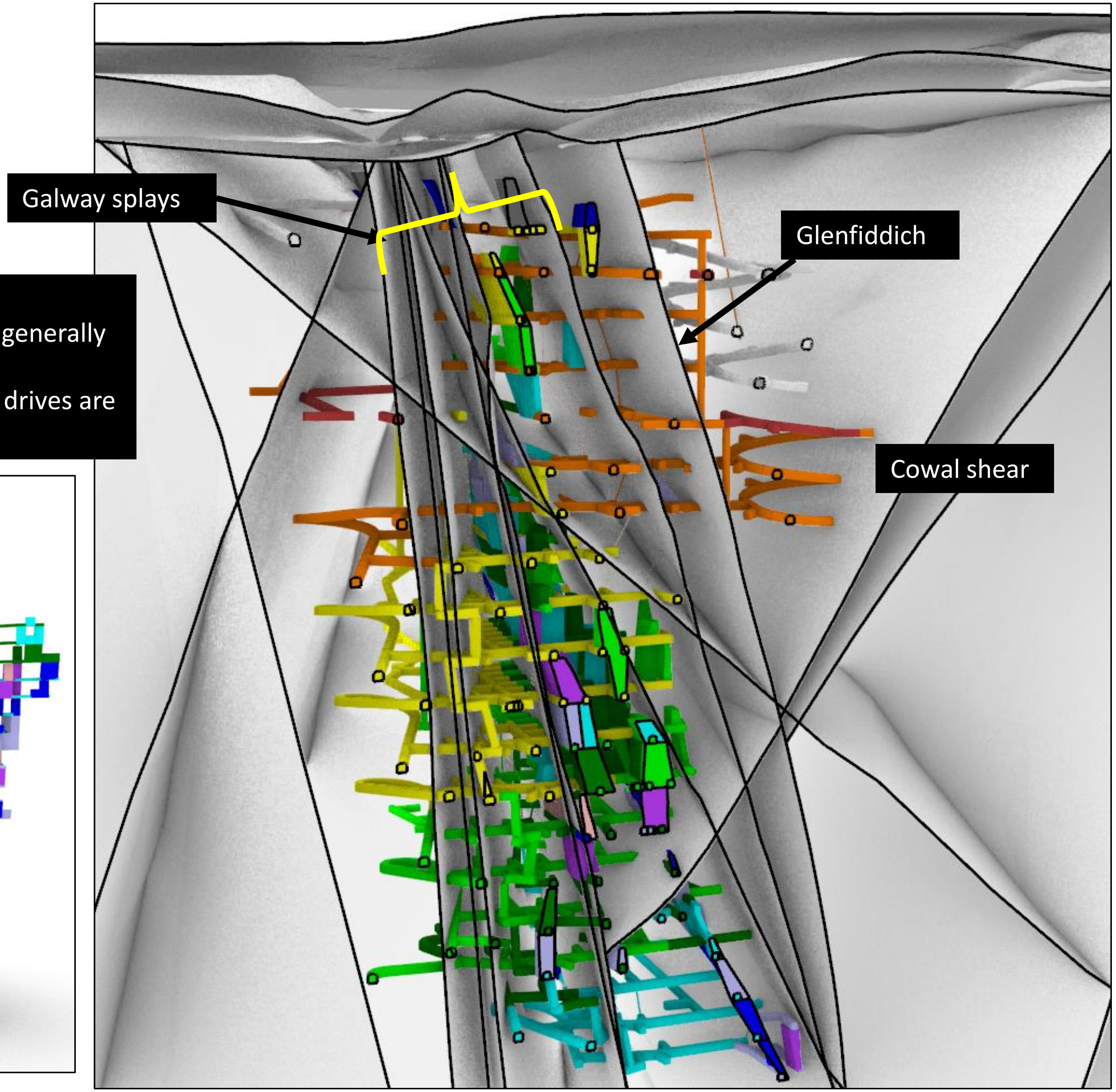


Figure 3-7: Cross section though the south panel decline showing interaction with known major faults

South decline fault interaction

- Galway splays intersect the decline. Fault intercepts are generally in the figure 8 bends and fault crossings in the decline are generally sub-perpendicular to the splay faults
- Ongoing fault characterisation (i.e. discrete faults vs a faulted zone) as well as confirmation of fault properties and persistence is required to further assess potential stability implications for the decline

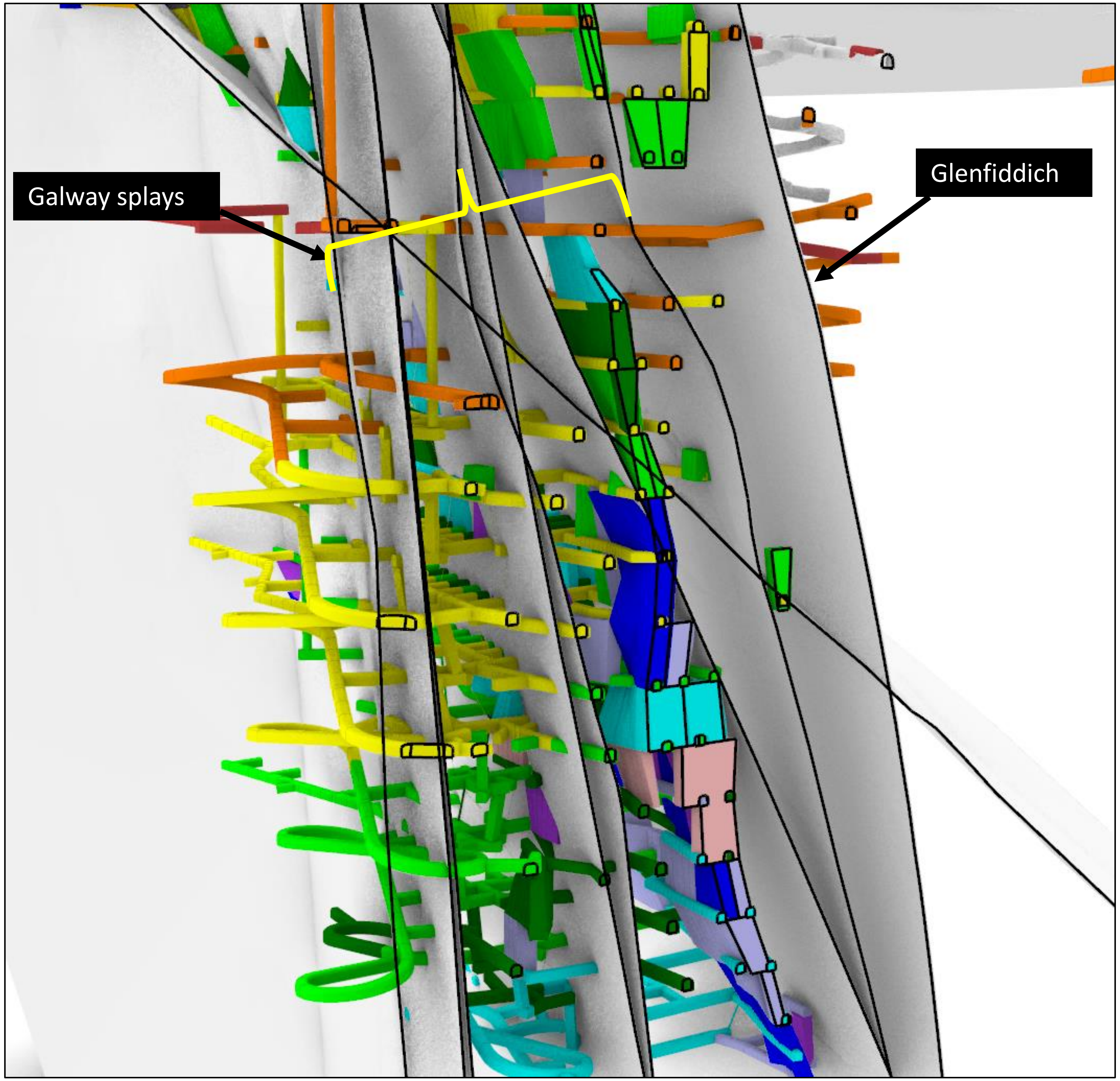
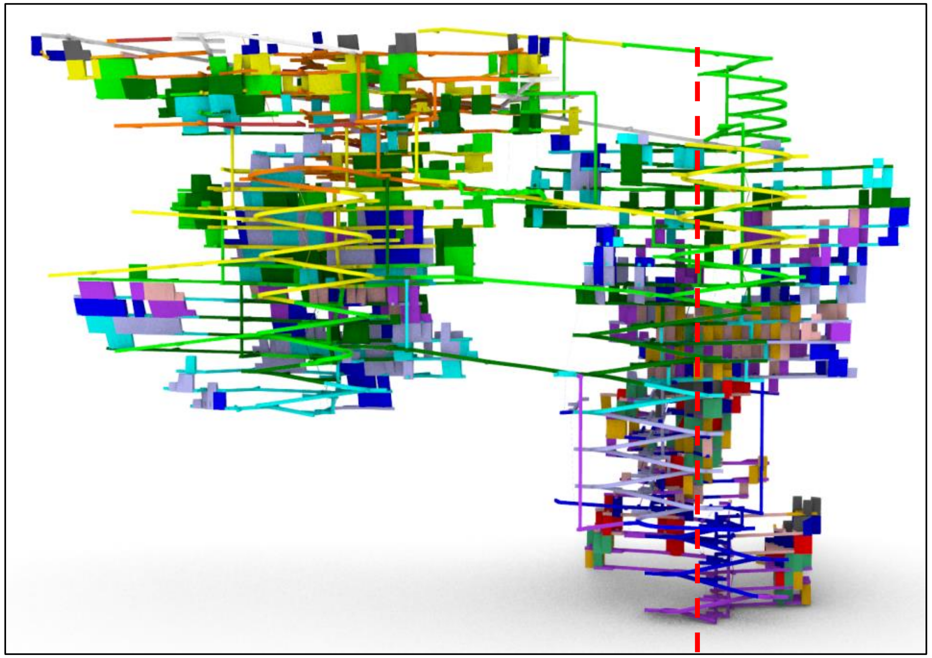


Figure 3-8: Cross section though the south panel decline showing interaction with known major faults

North decline fault interaction

- Galway splays intersect the declines. This cannot be avoided for any FW decline option given the distribution of the Galway splay faults in the current interpretation
- Level accesses cross the Galway splay faults at an angle that is generally favourable (close to perpendicular)
- Some intersections and north-south orientation infrastructure drives are unfavourably located with respect to the Galway splay faults



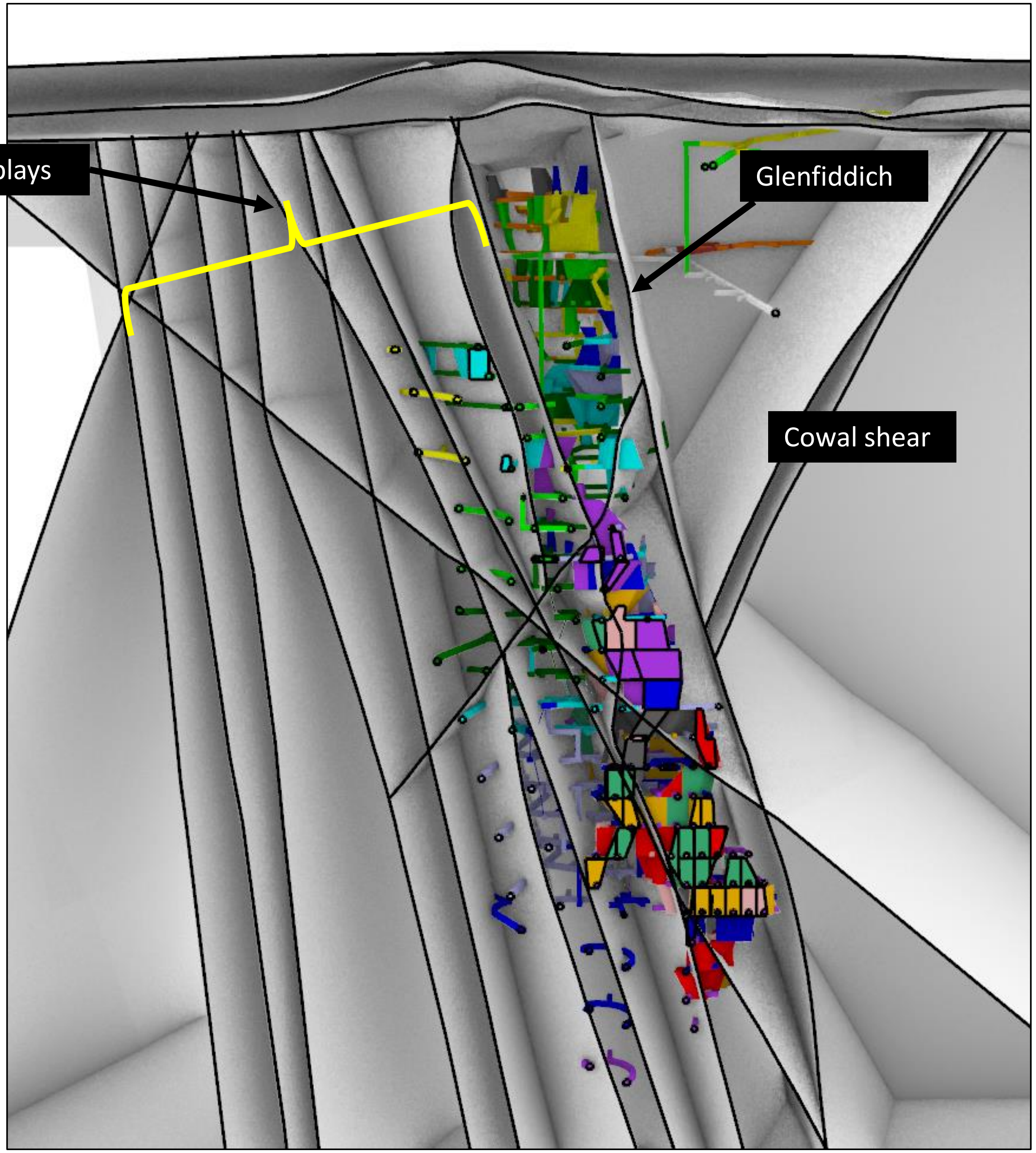


Figure 3-9: Cross section though the south panel decline showing interaction with known major faults

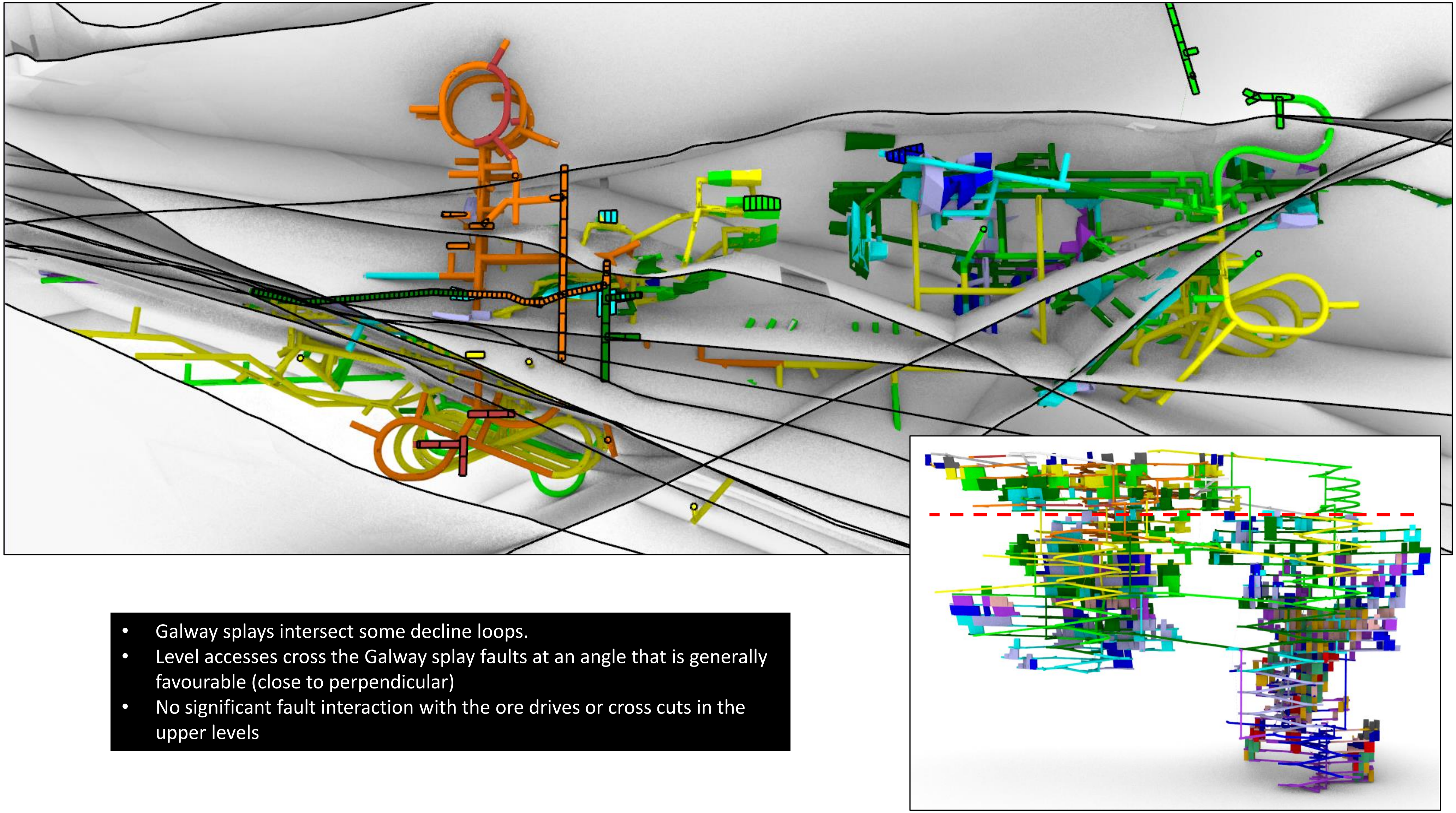


Figure 3-10: Plan view showing interaction between planned development and major faults

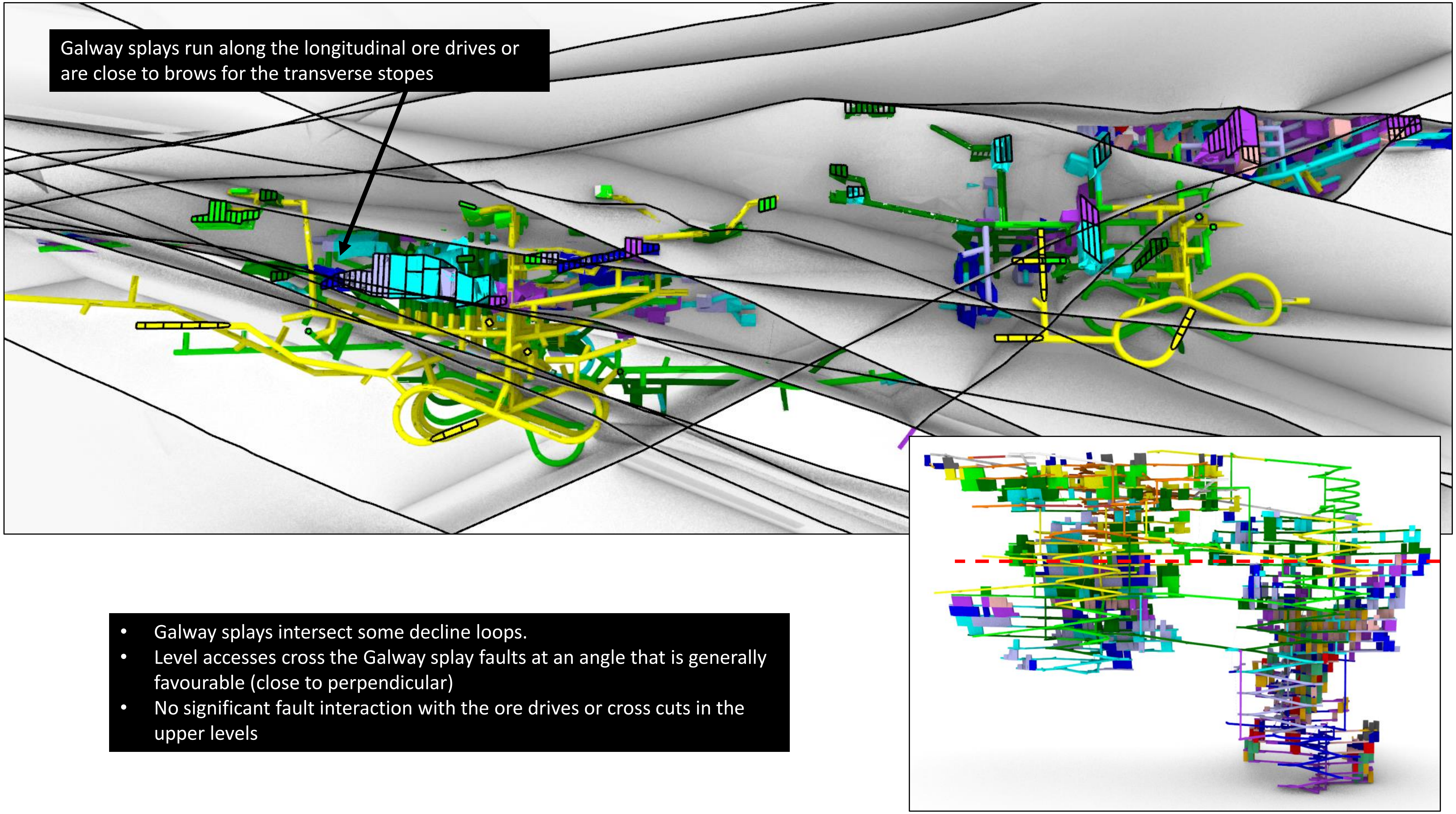
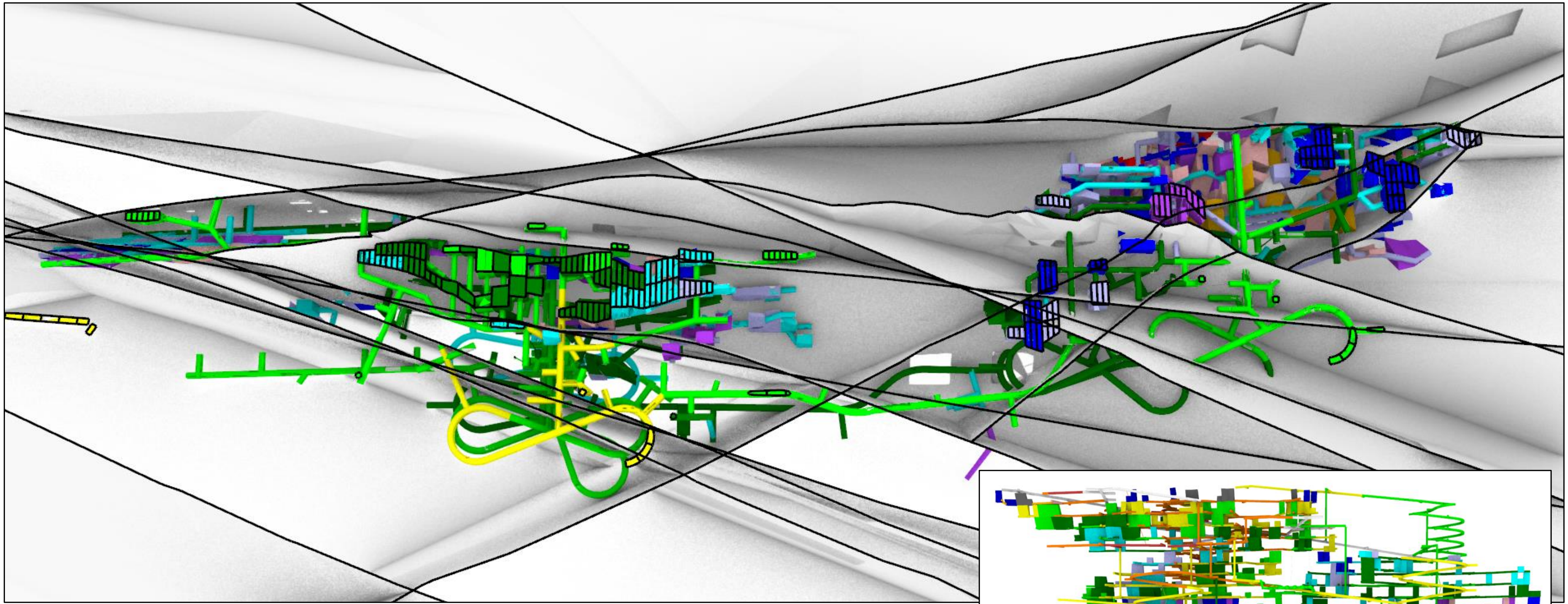


Figure 3-11: Plan view showing interaction between planned development and major faults



- Galway splays intersect some decline loops.
- Level accesses cross the Galway splay faults at an angle that is generally favourable (close to perpendicular)
- No significant fault interaction with the ore drives or cross cuts in the upper levels

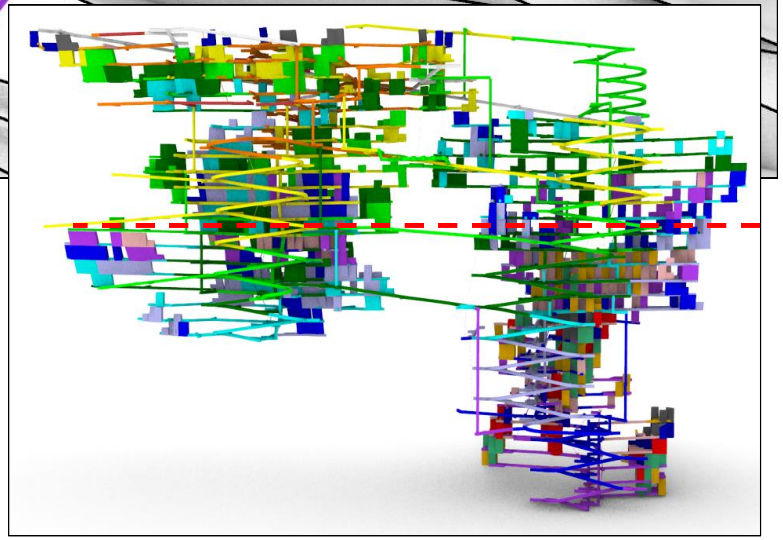


Figure 3-12: Plan view showing interaction between planned development and major faults

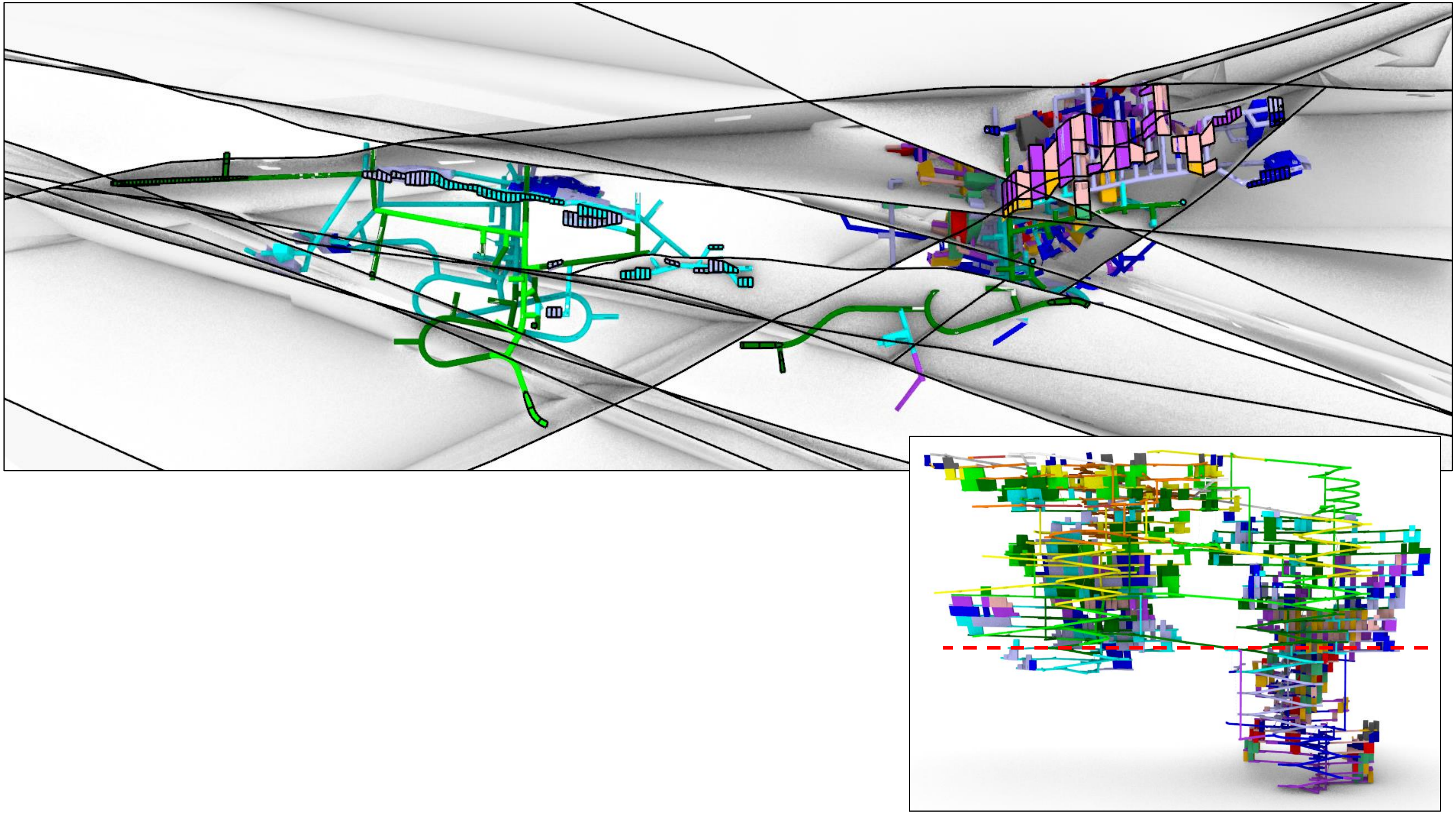


Figure 3-13: Plan view showing interaction between planned development and major faults

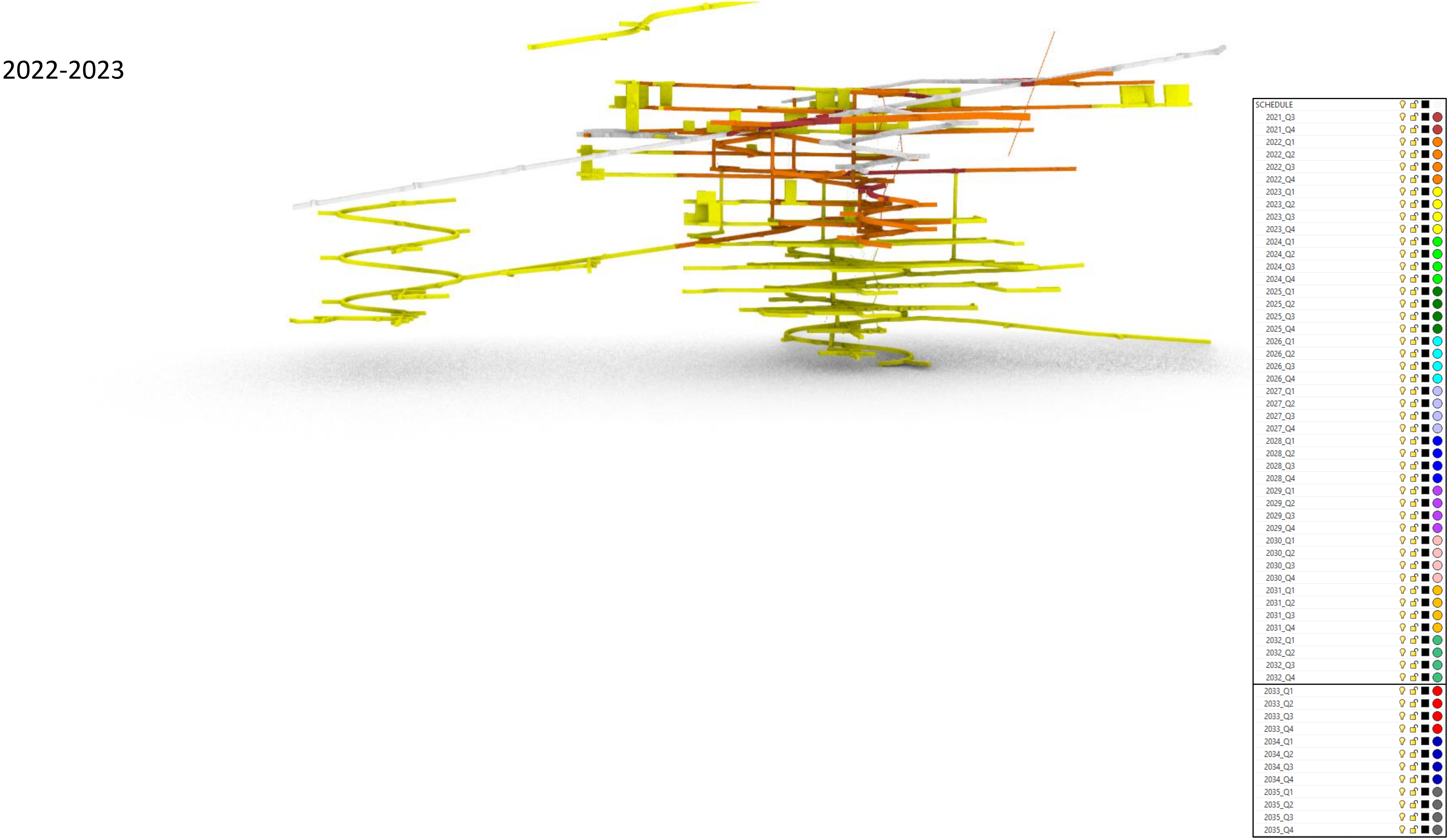


Figure 3-14: Mining sequence for Design option #4

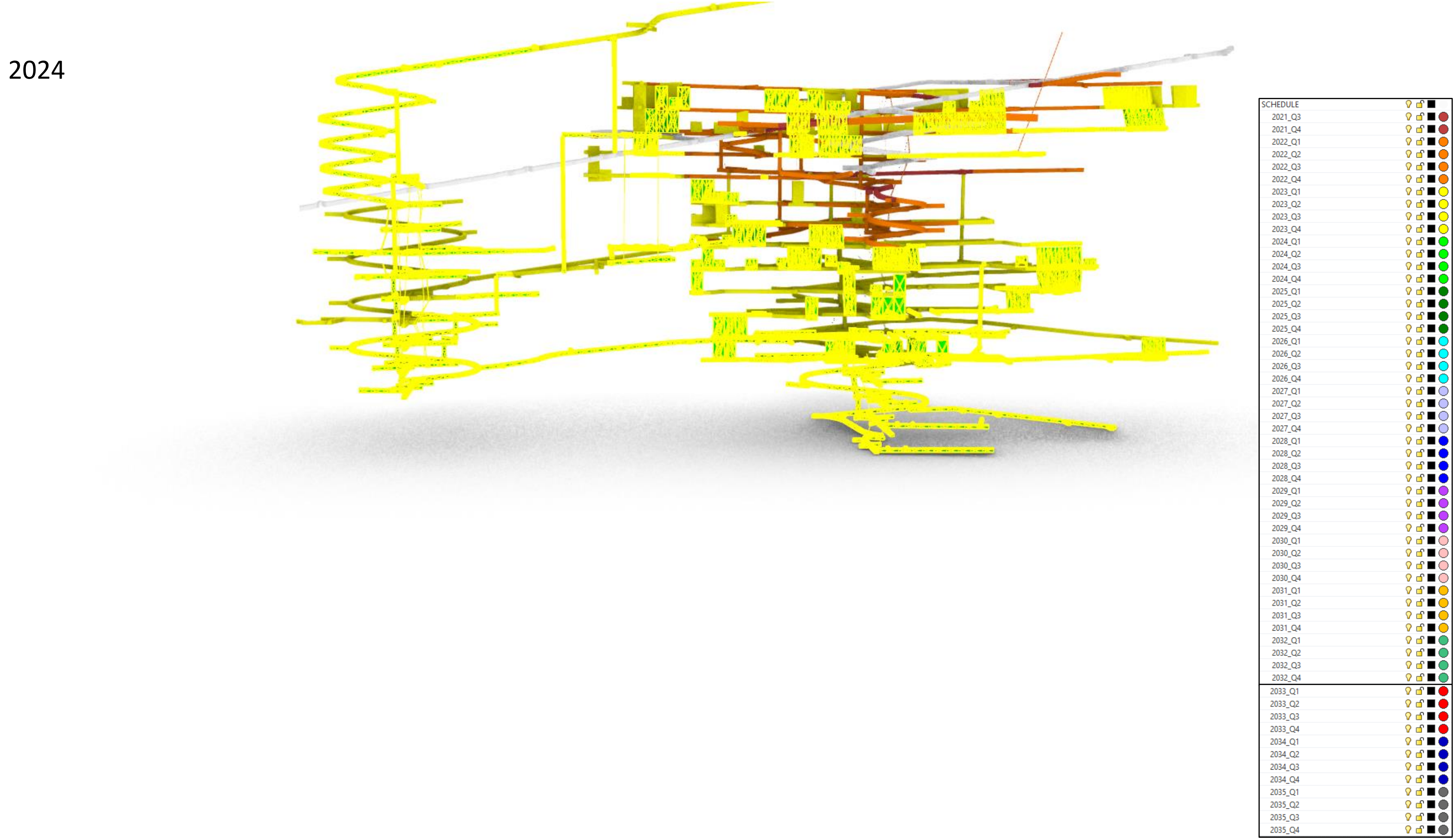


Figure 3-15: Mining sequence for Design option #4

2025

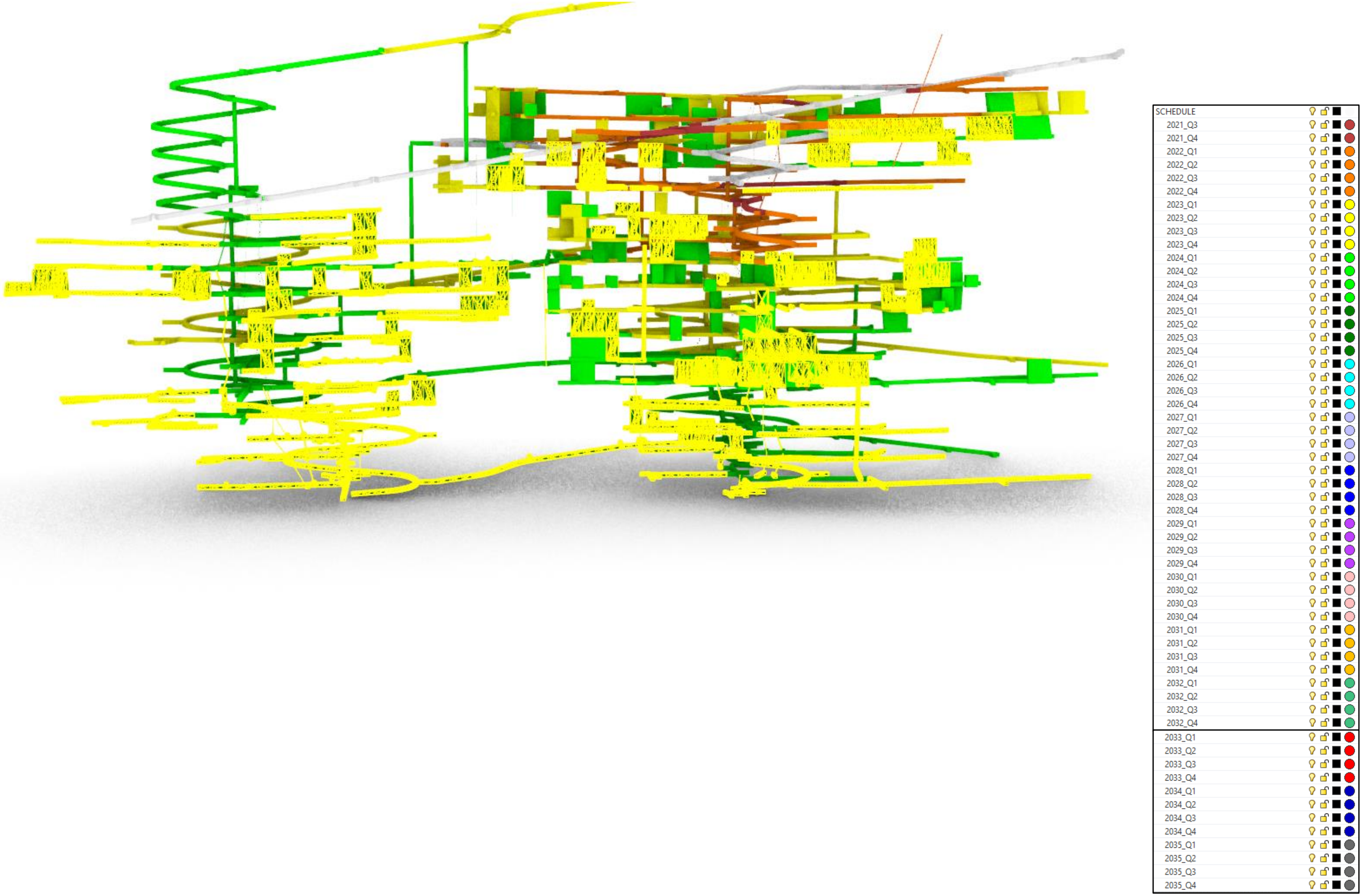


Figure 3-16: Mining sequence for Design option #4

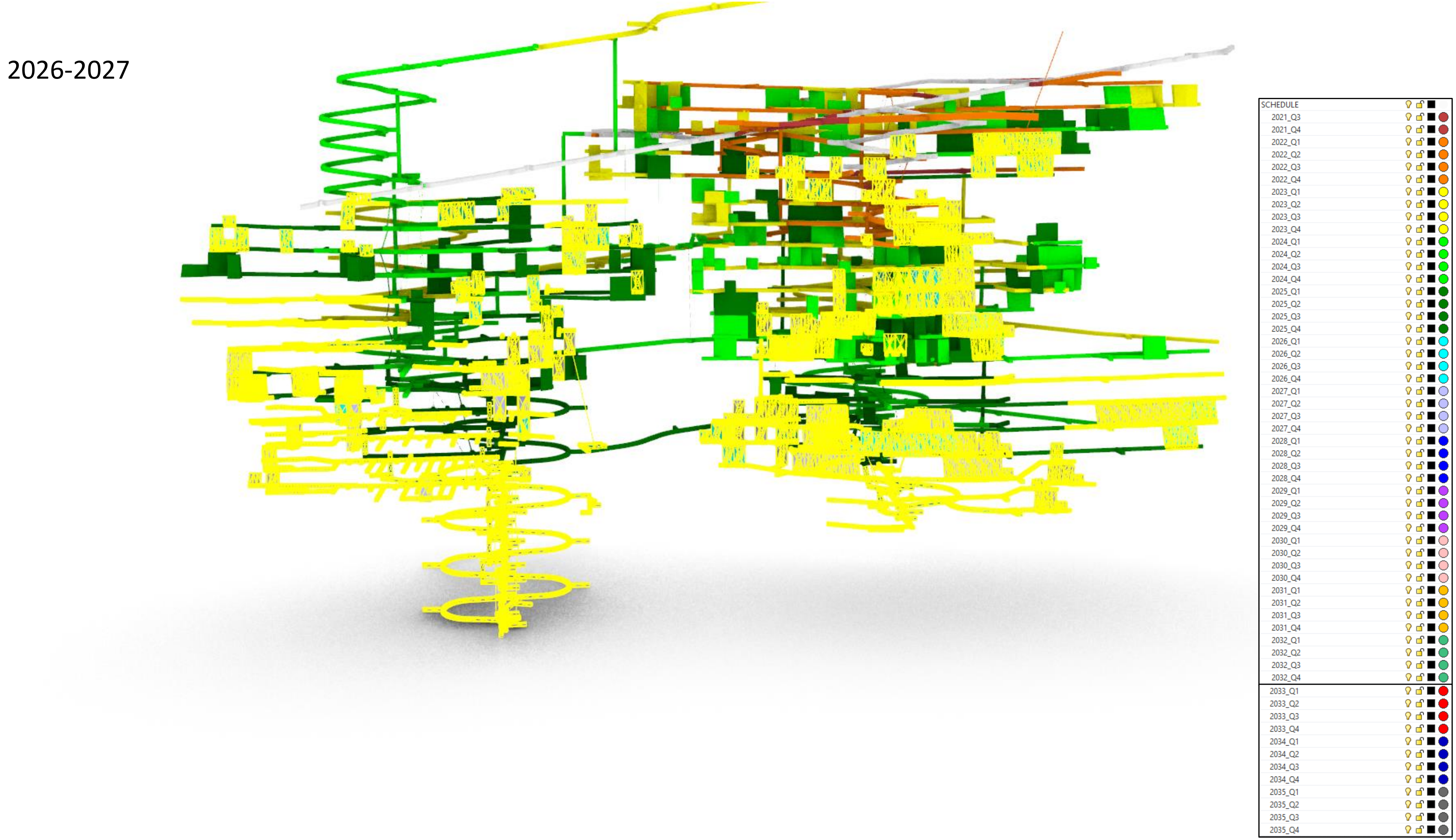


Figure 3-17: Mining sequence for Design option #4

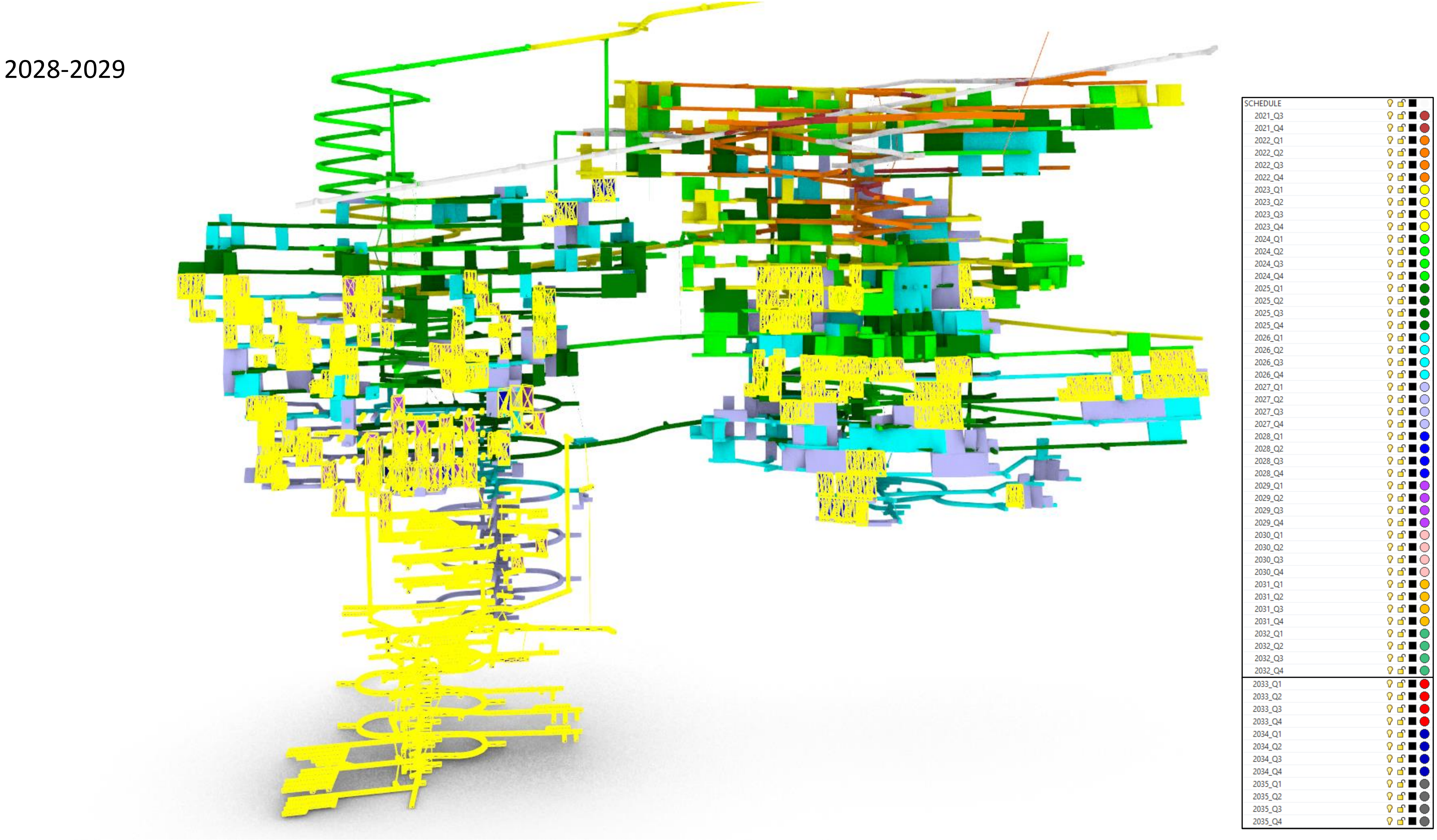


Figure 3-18: Mining sequence for Design option #4

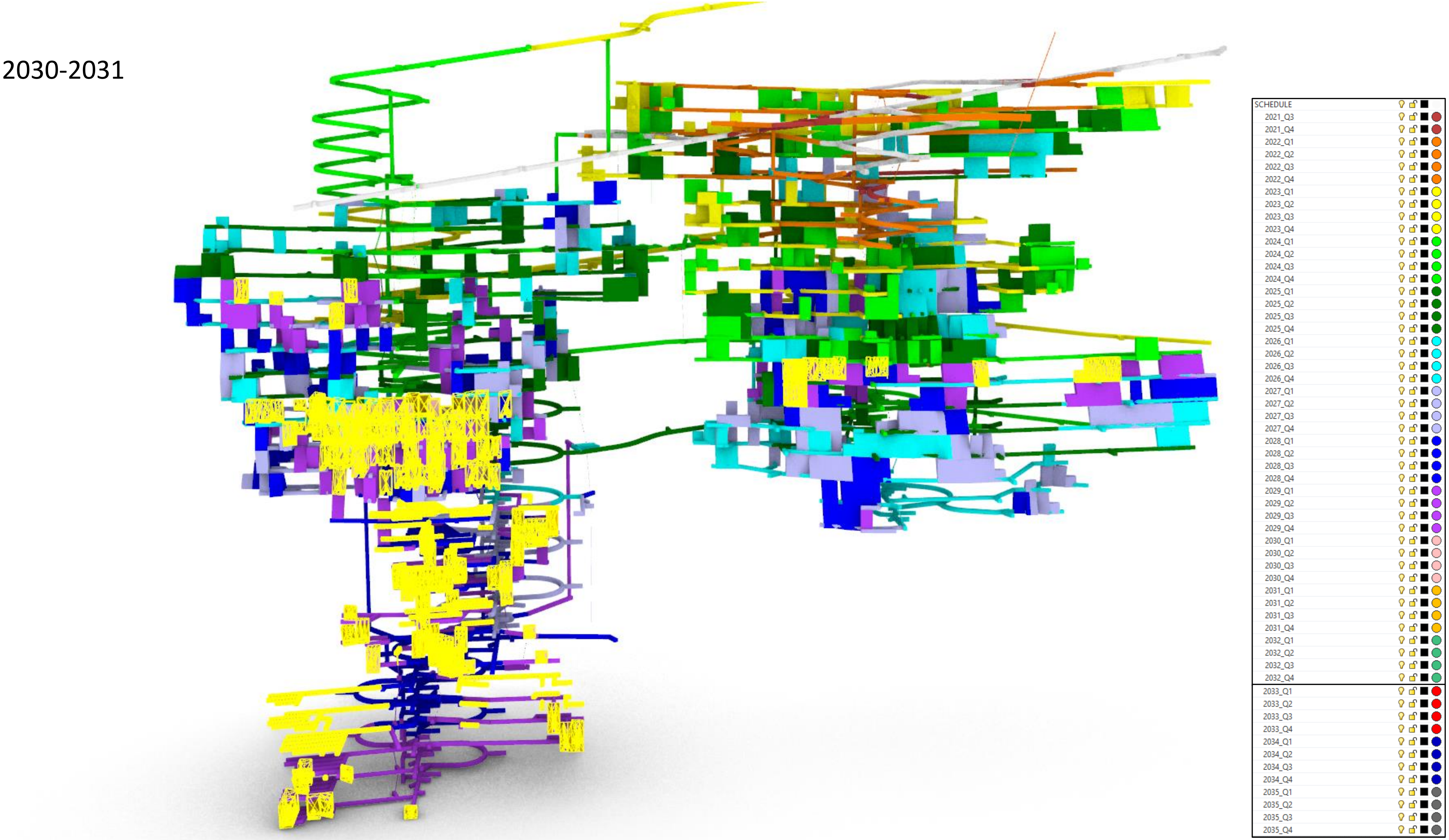


Figure 3-19: Mining sequence for Design option #4

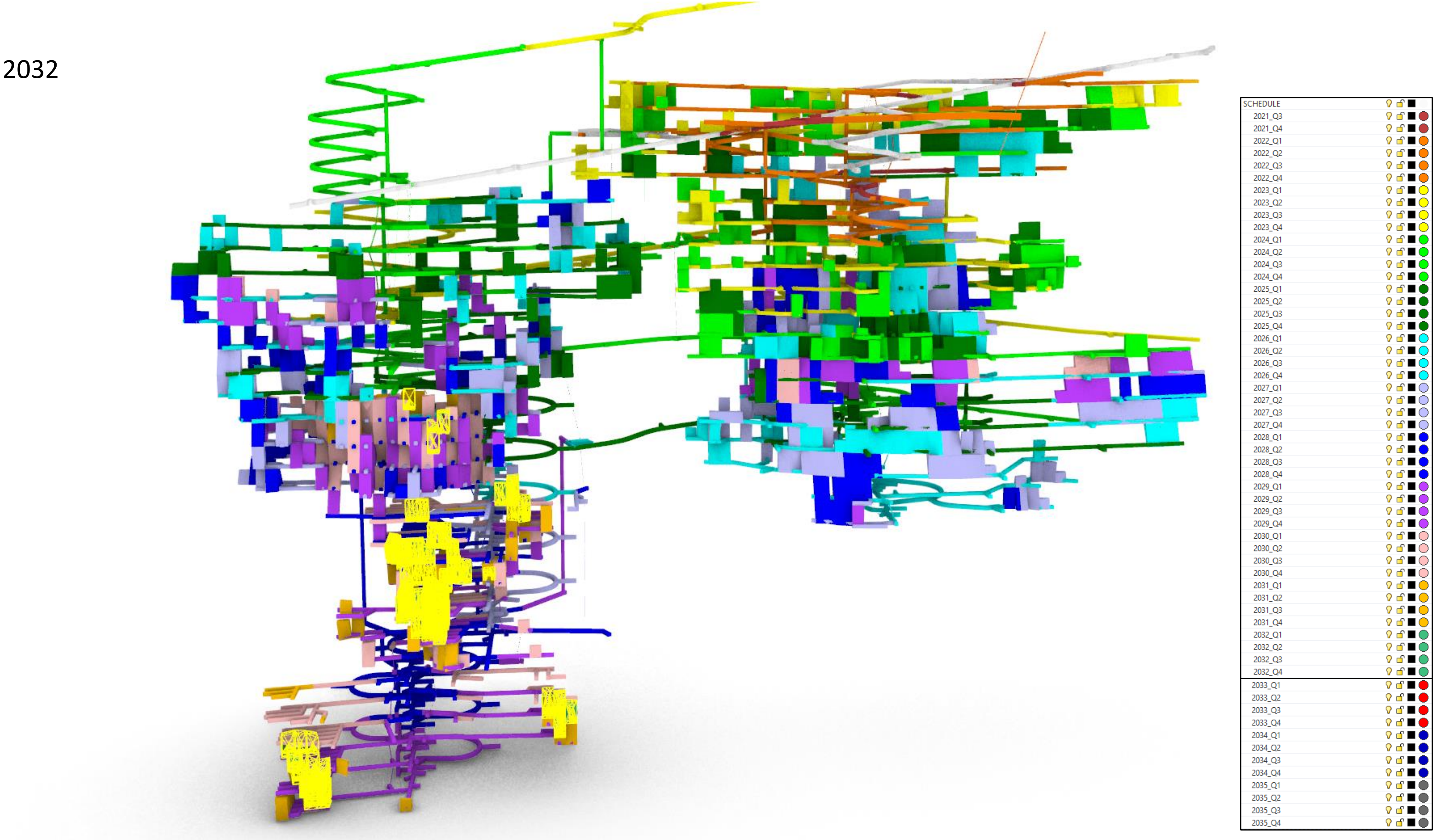


Figure 3-20: Mining sequence for Design option #4

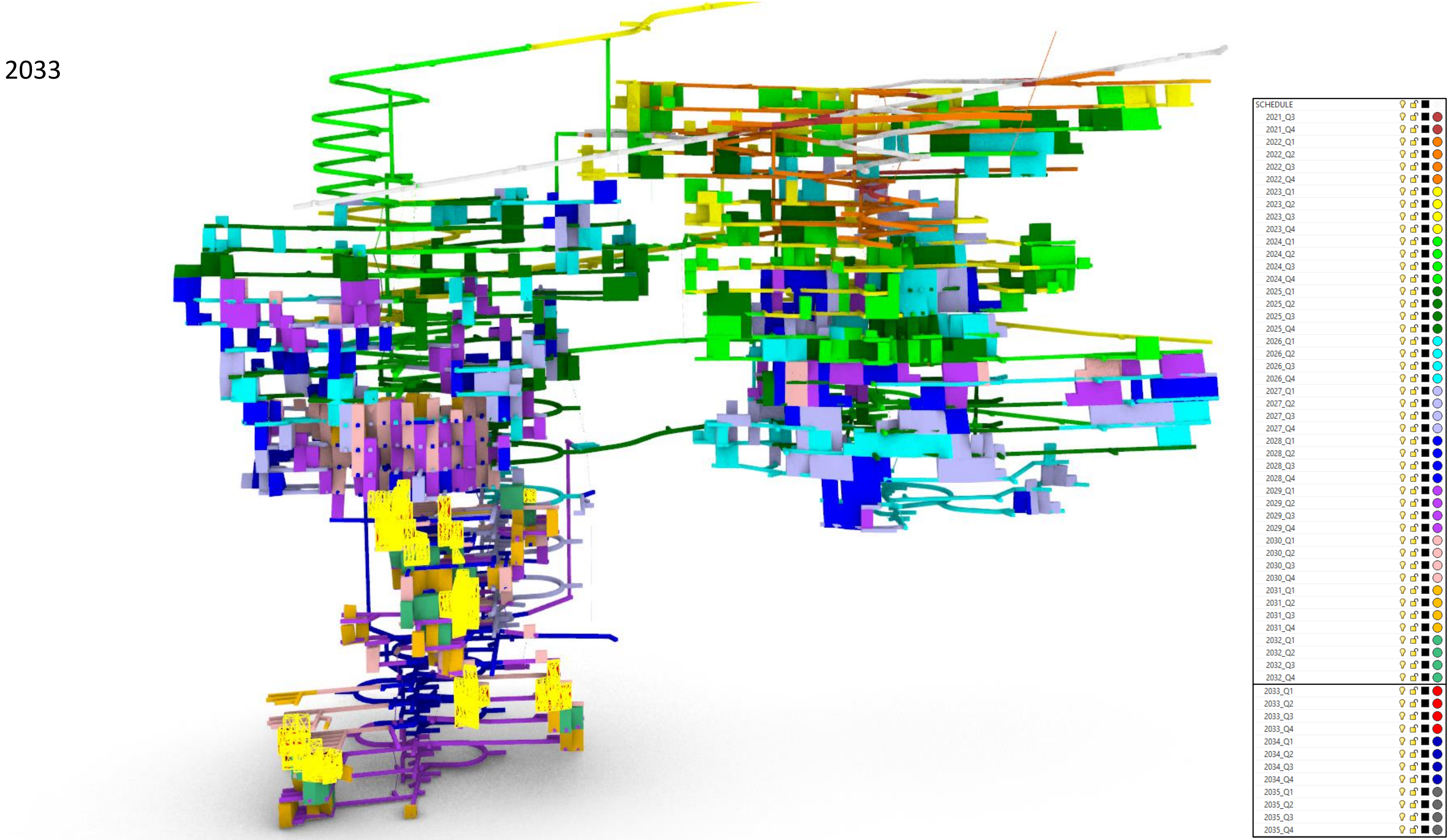


Figure 3-21: Mining sequence for Design option #4

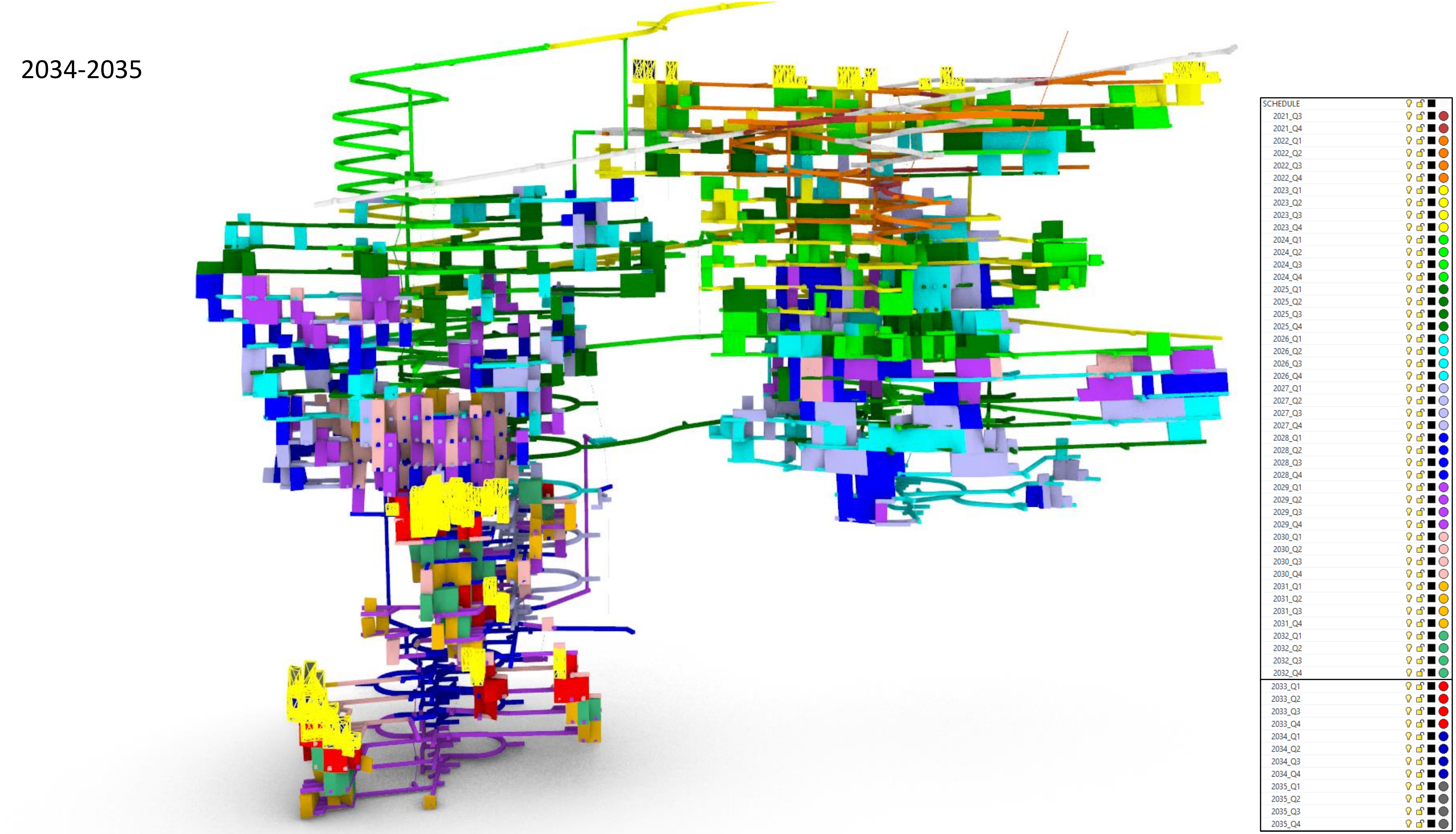


Figure 3-22: Mining sequence for Design option #4

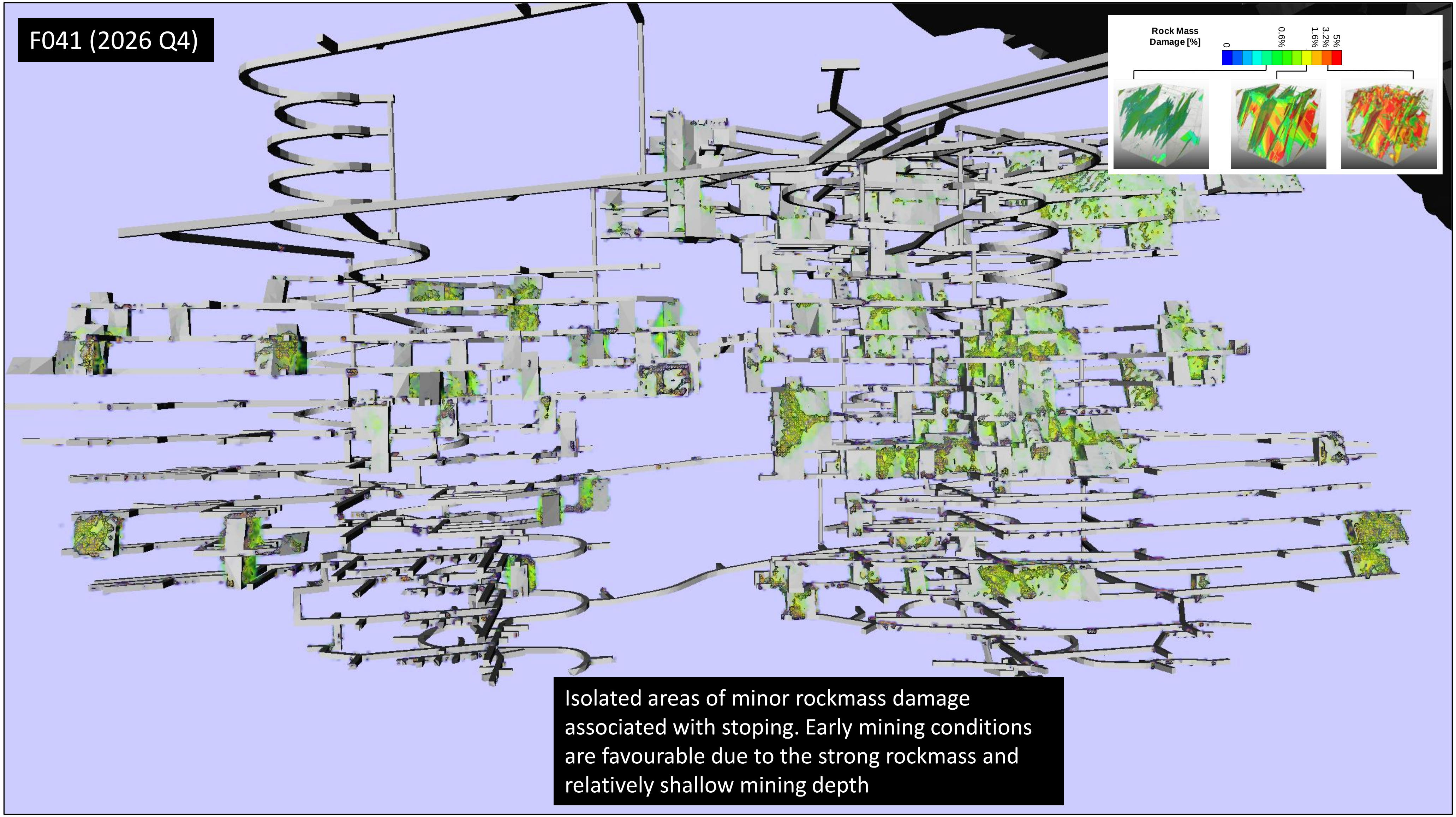


Figure 3-23 Rockmass damage forecasts for mining complete to the end of 2026 (image faces East)

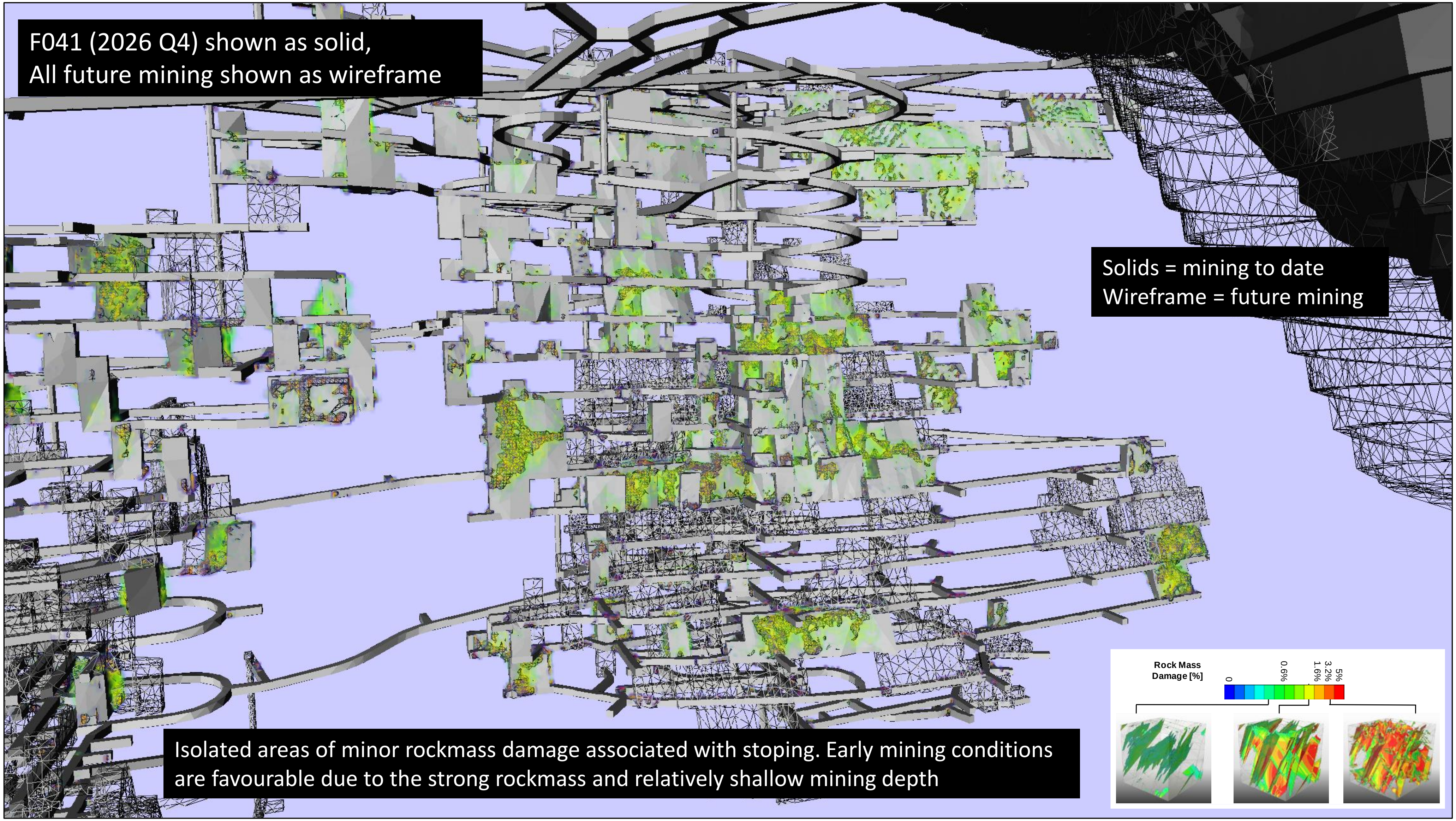


Figure 3-24: Rockmass damage forecasts for mining complete to the end of 2026, with future mining shown as wireframe (image faces East)

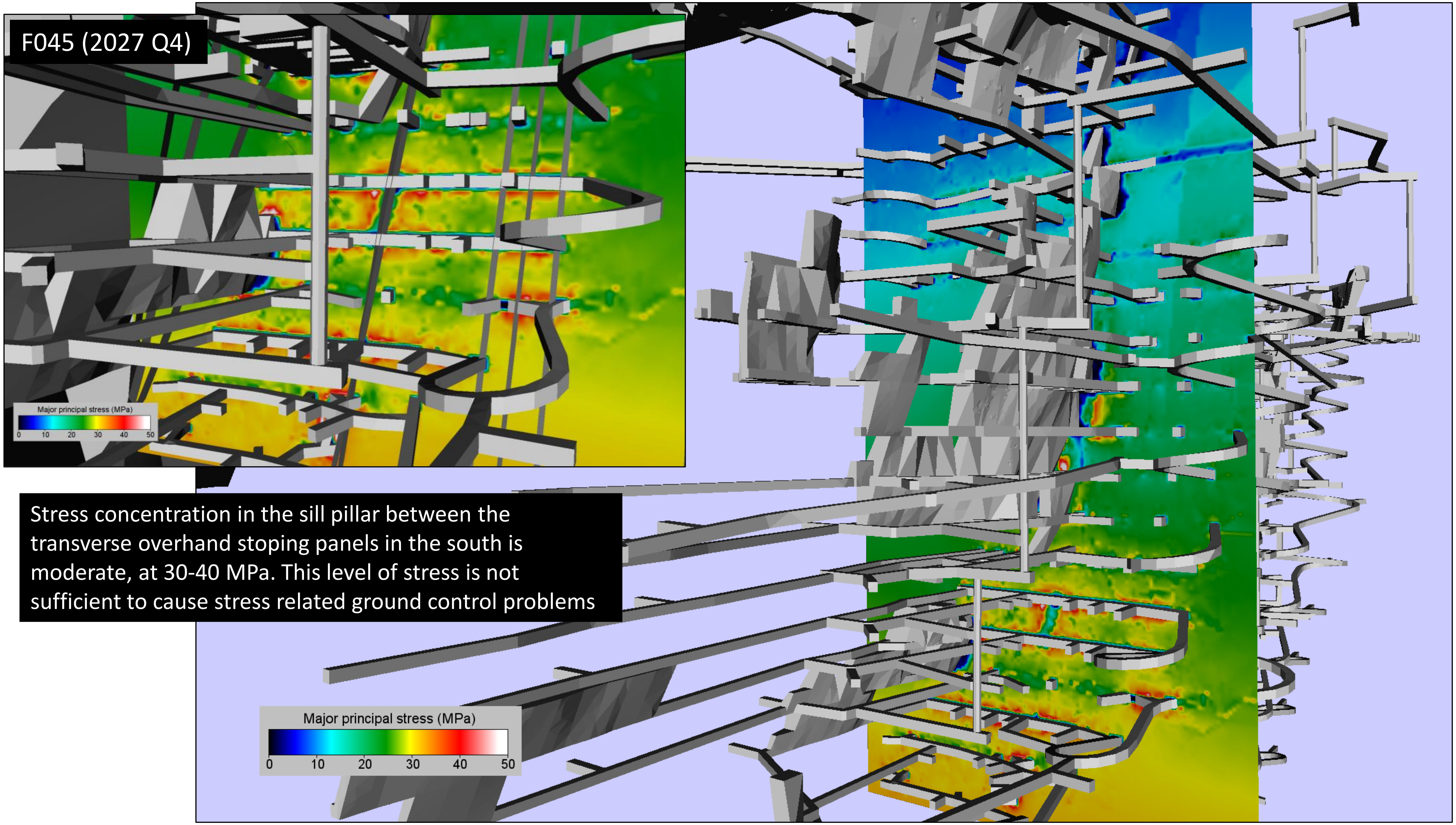


Figure 3-25: Cross sections through the South panel showing major principal stress (images face approximately north)

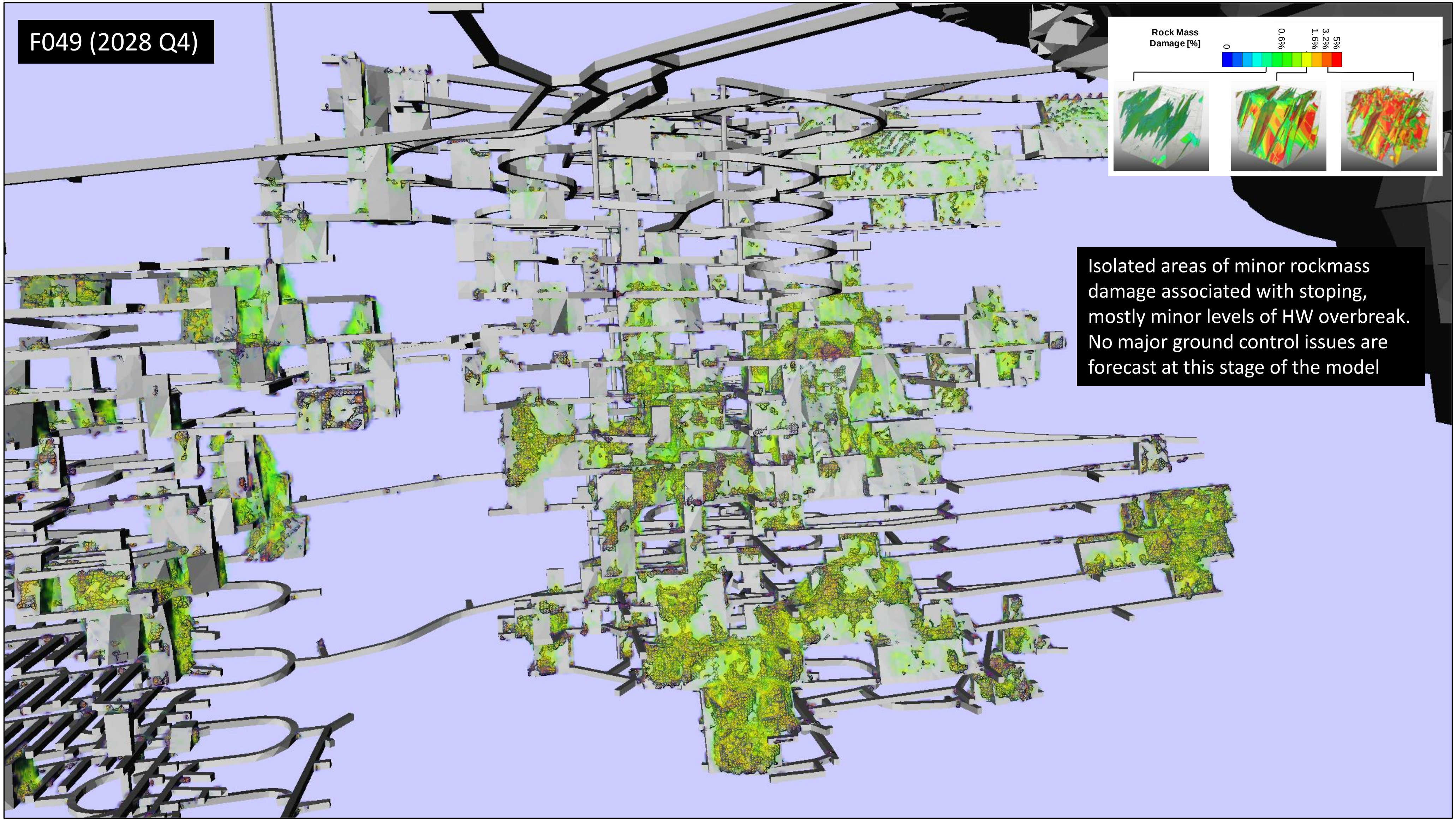


Figure 3-26: Rockmass damage forecasts for mining complete to the end of 2028 (image faces East)

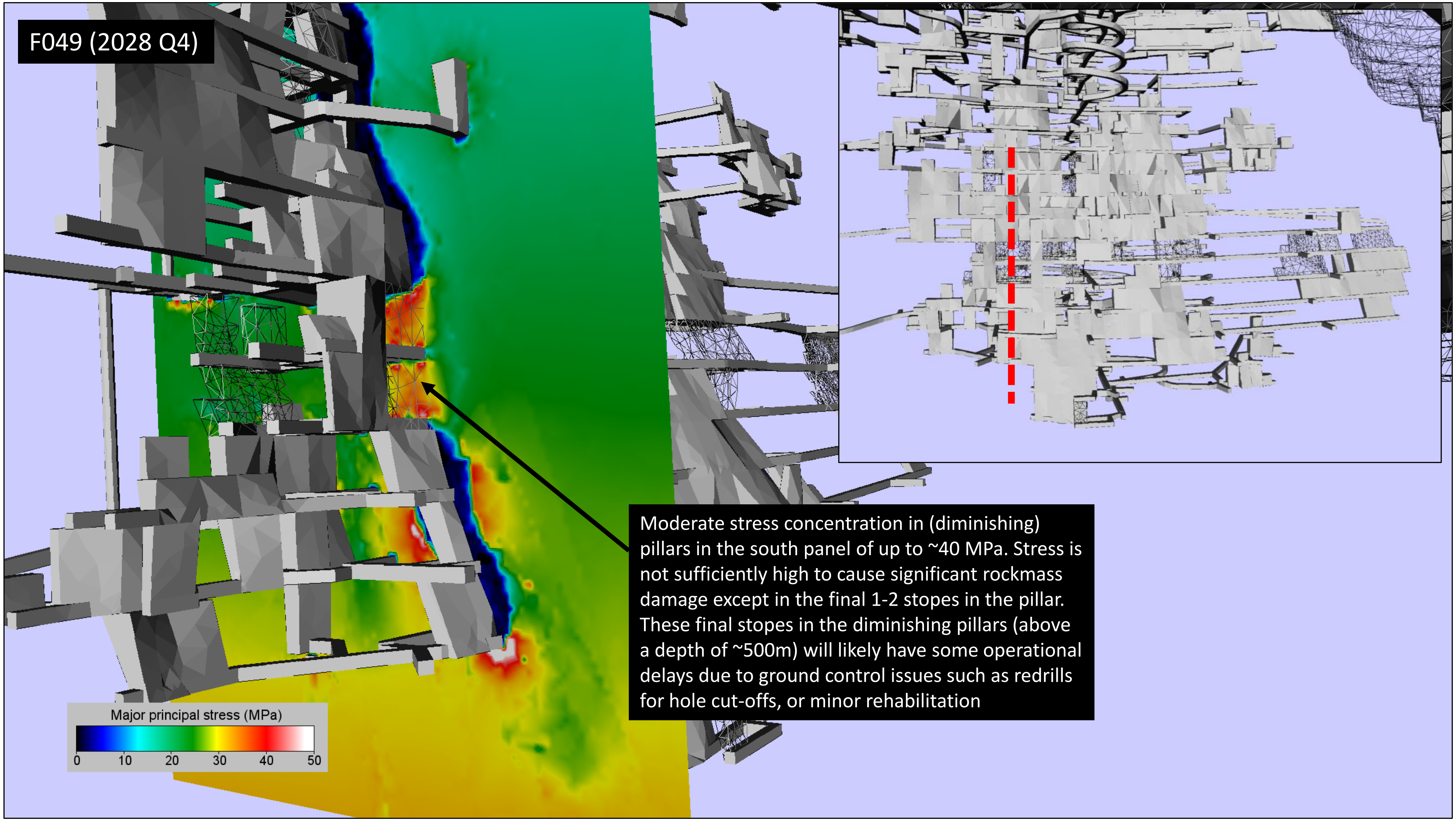


Figure 3-27: Stress concentration forecast in the late stages of mining in the south panel

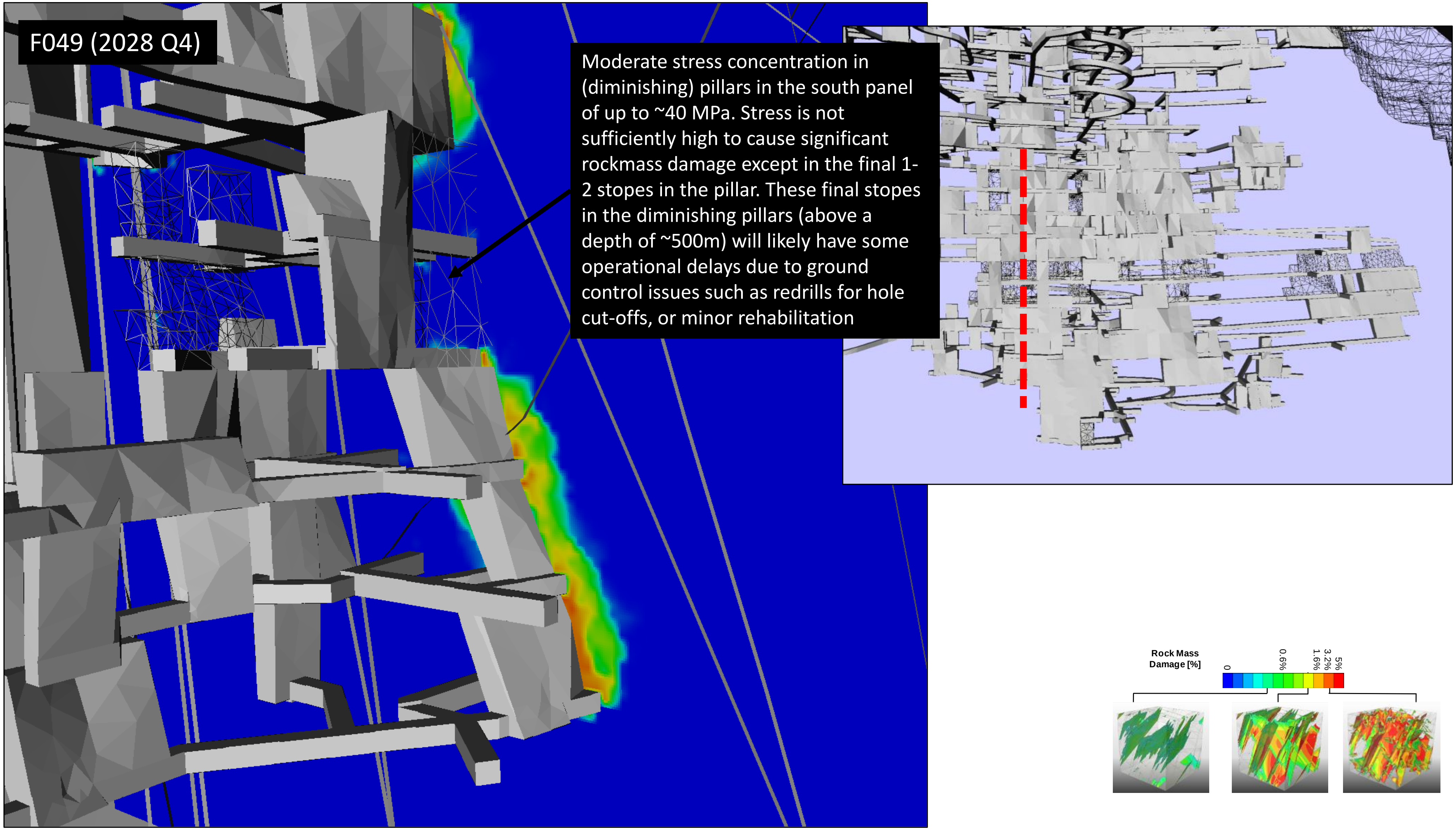


Figure 3-28: Cross section showing rockmass damage forecasts in the lower section of the south panel in the late stages of mining

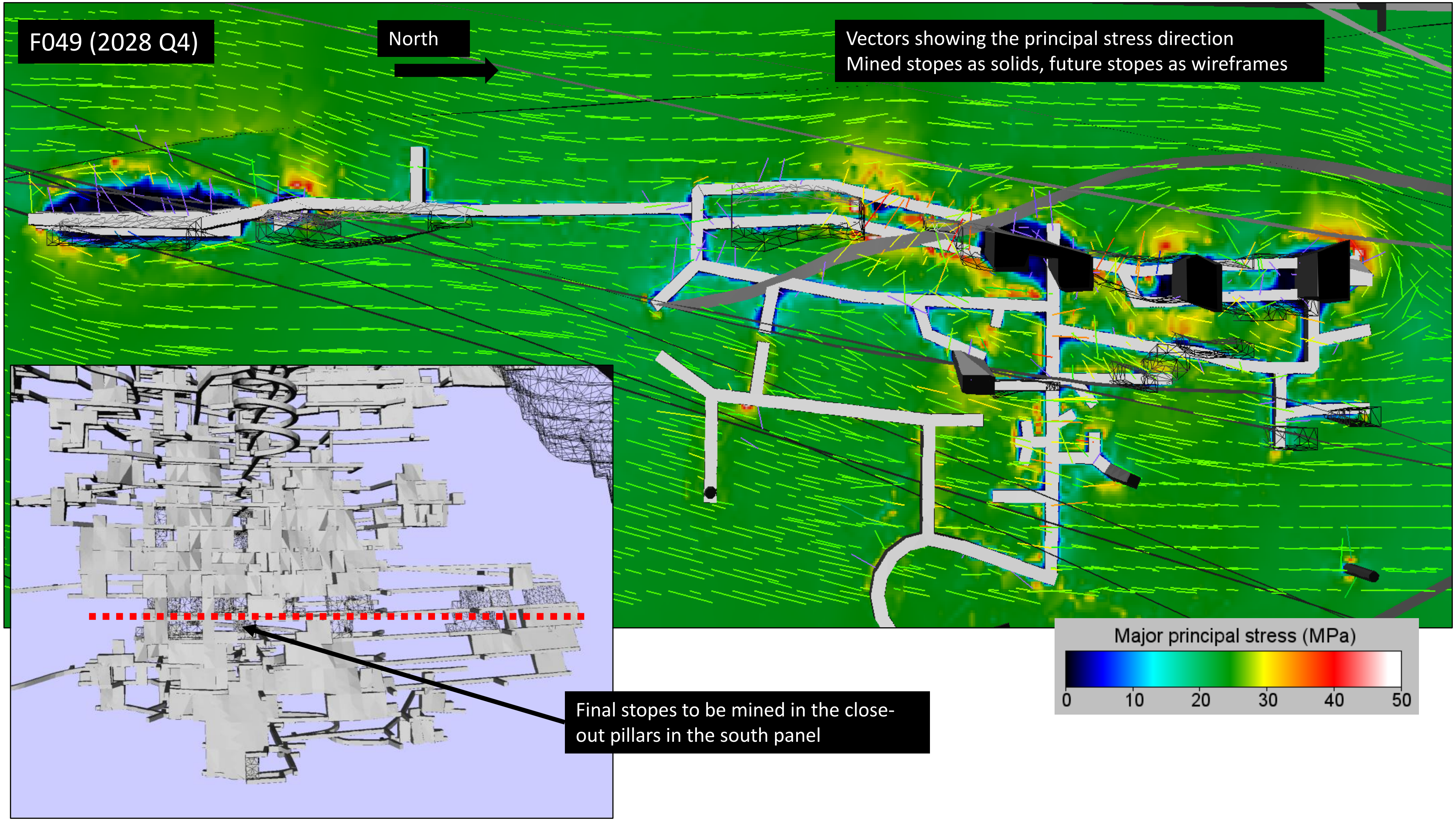


Figure 3-29: Horizontal cross section through the final stoping level in the south panel showing stress concentration and direction vectors

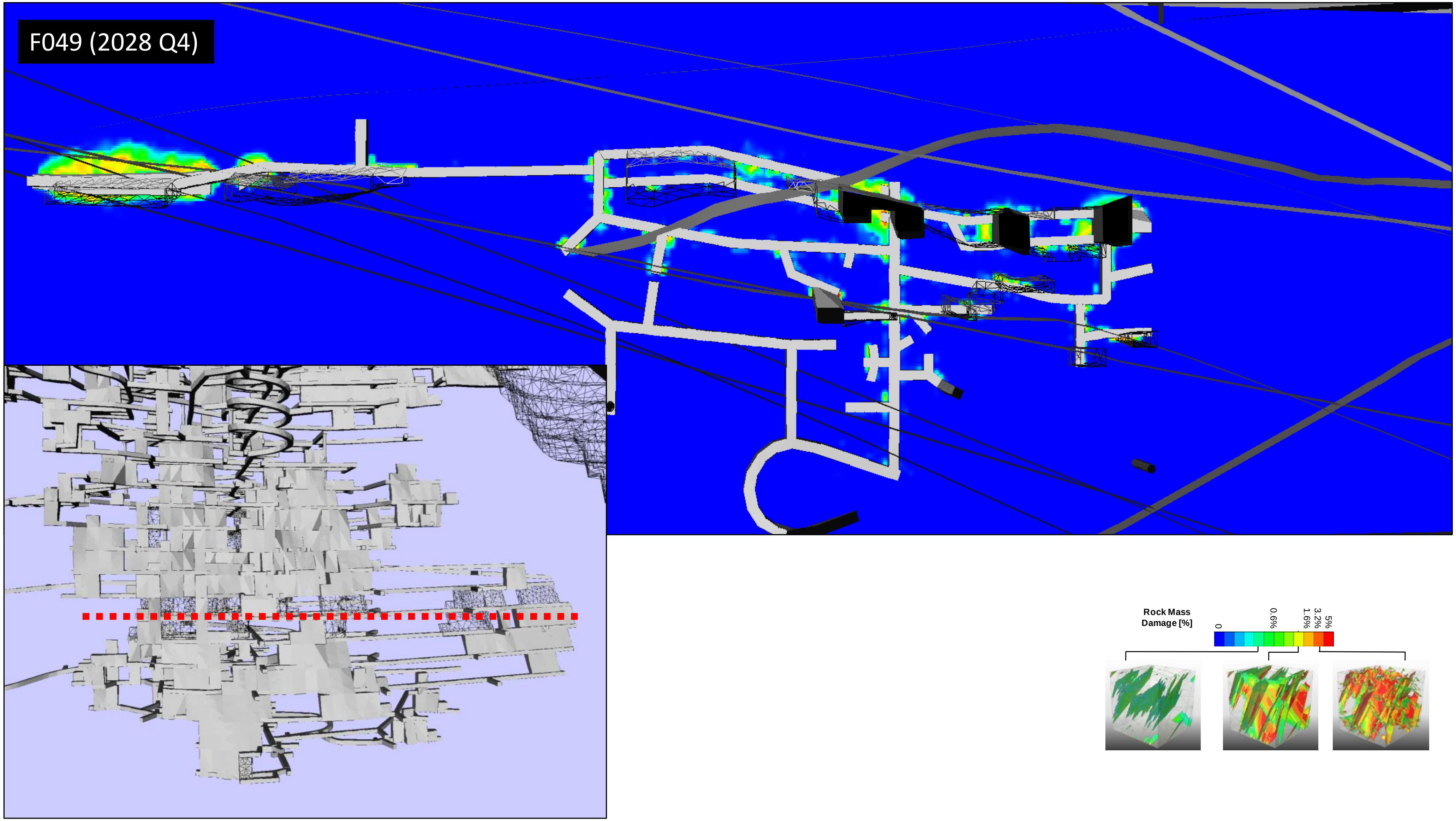


Figure 3-30: Horizontal cross section through the final stoping level in the south panel showing forecast rockmass damage

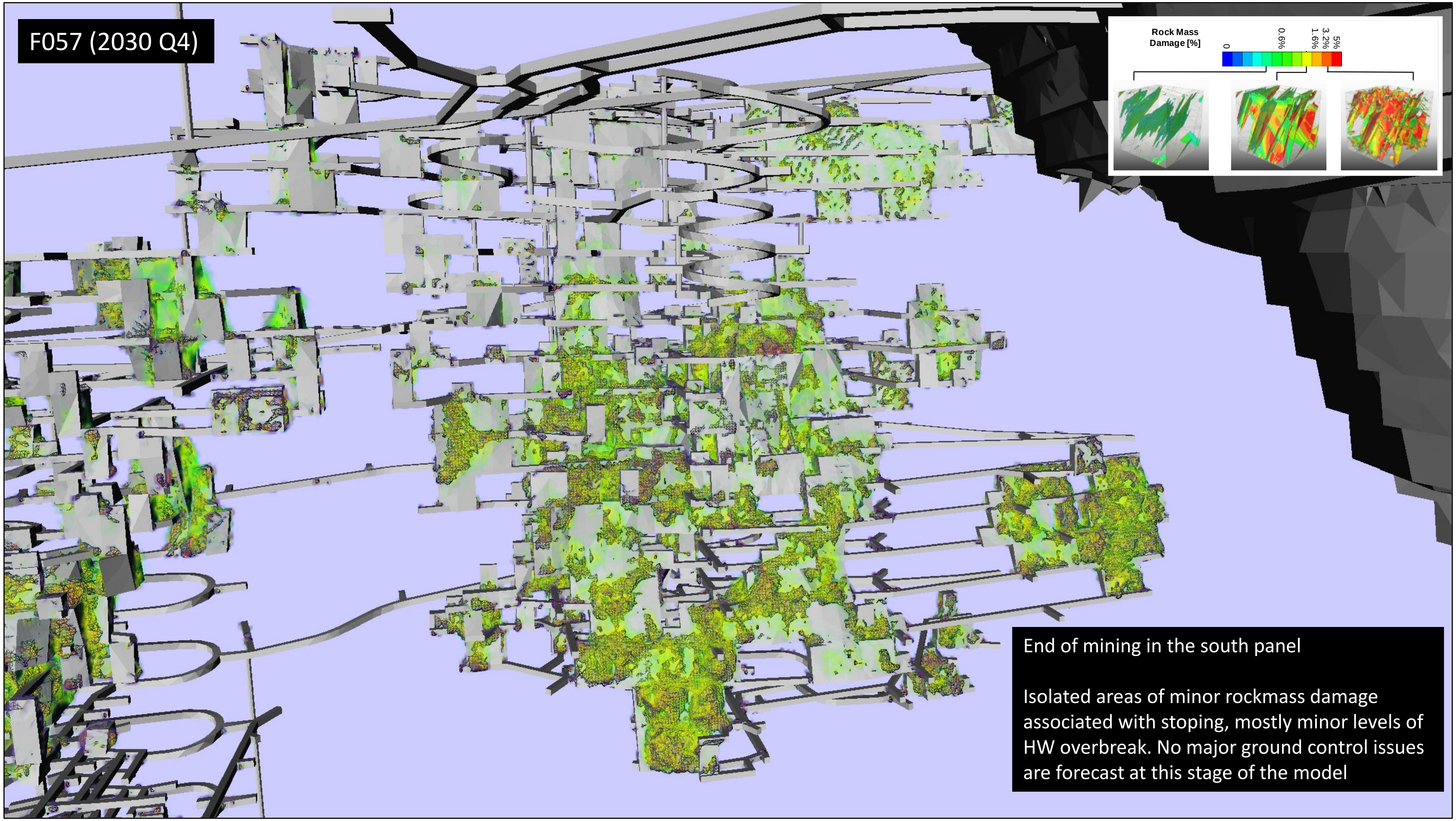


Figure 3-31: Forecast rockmass damage in the south panel in 2030

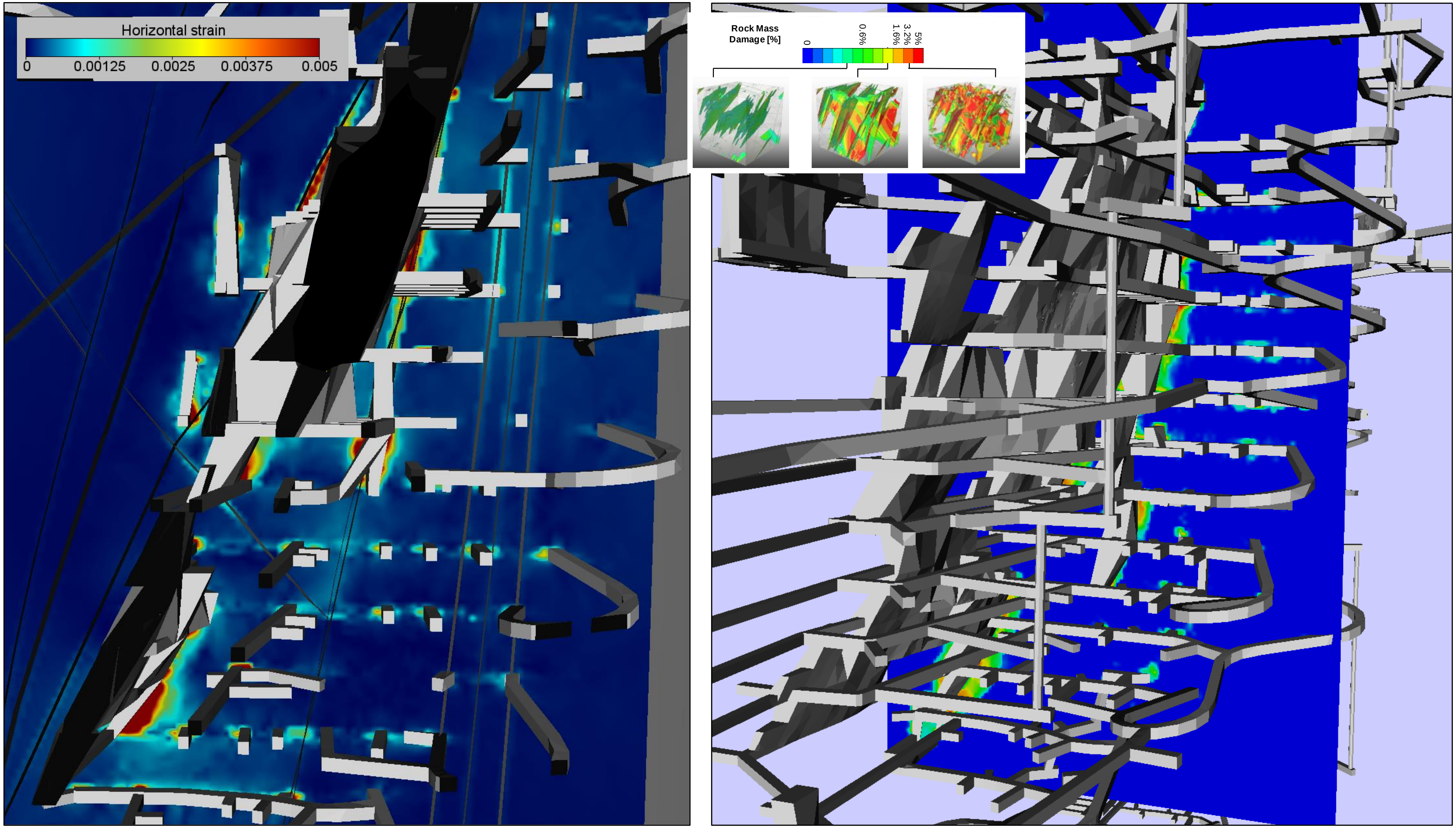


Figure 3-32: Cross sections showing horizontal strain (left) and rockmass damage (right) in the decline region of the south panel at end of mining

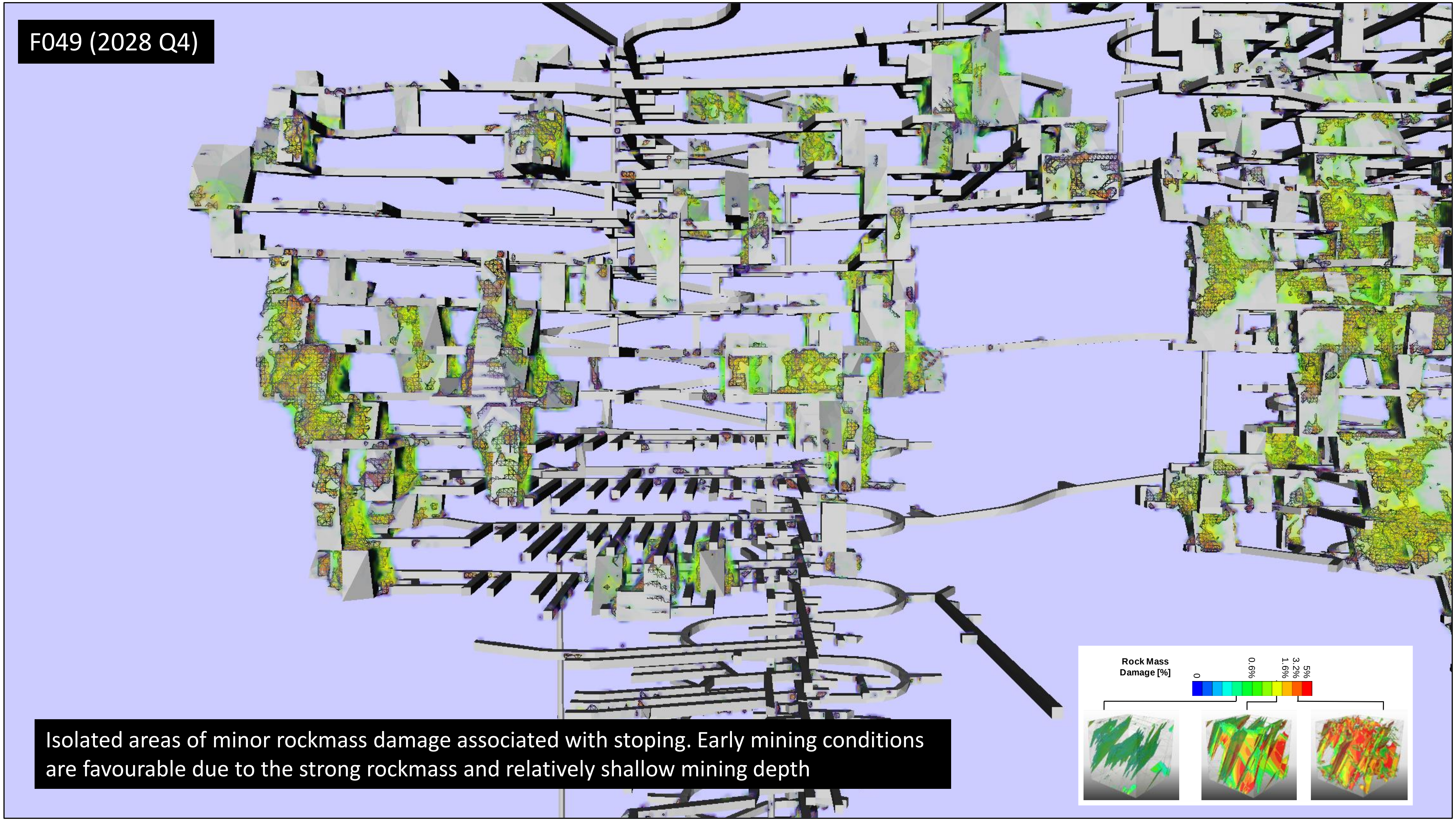


Figure 3-33: Rockmass damage forecasts during early stages of mining in the north panel (end 2028)

Figure 3-34: Rockmass damage forecasts during early stages of mining in the north panel (end 2029), including damage forecast in the secondary stopes

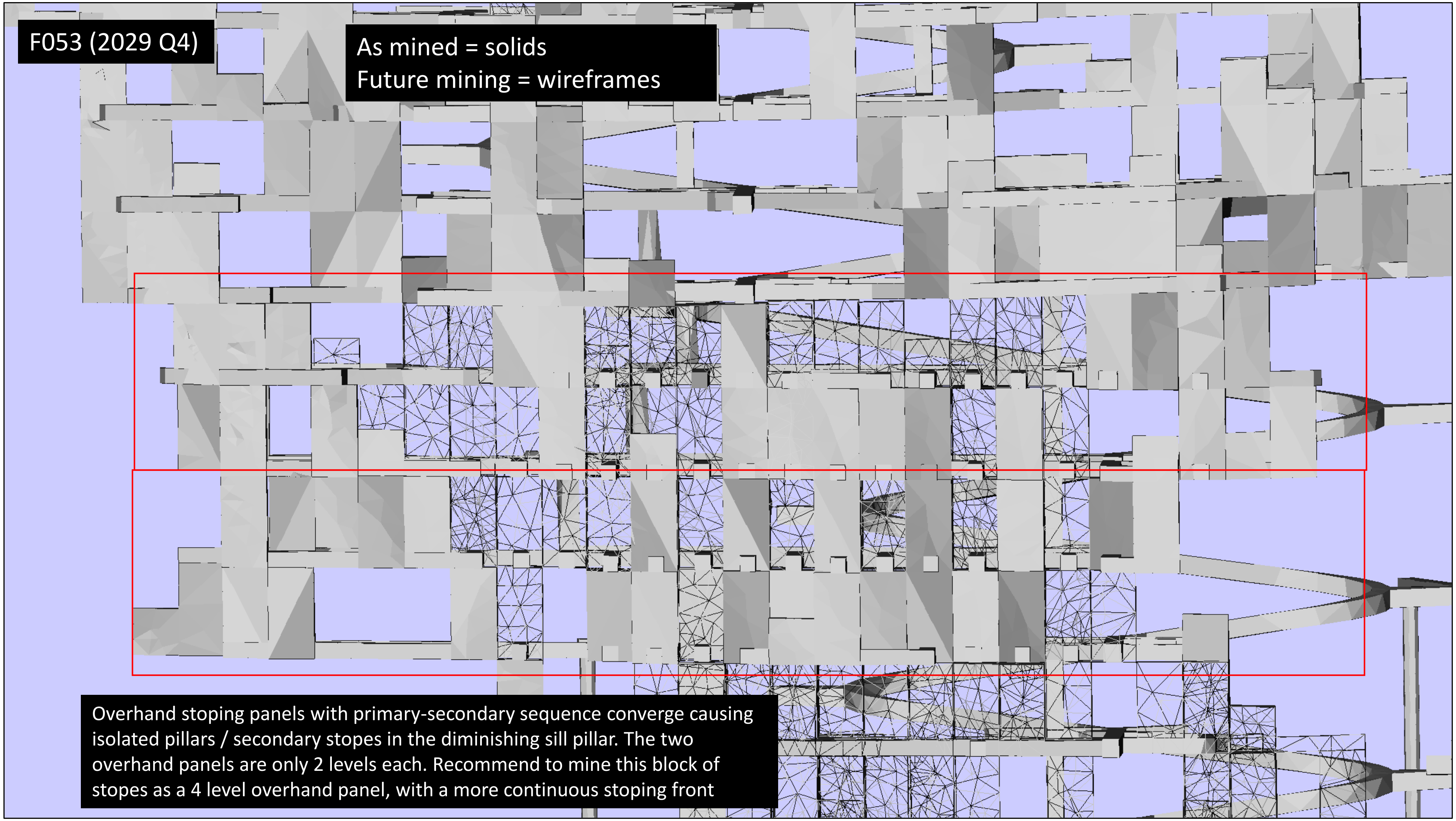


Figure 3-35: Mining to the end of 2029 (solids) and future mining (as wireframes)

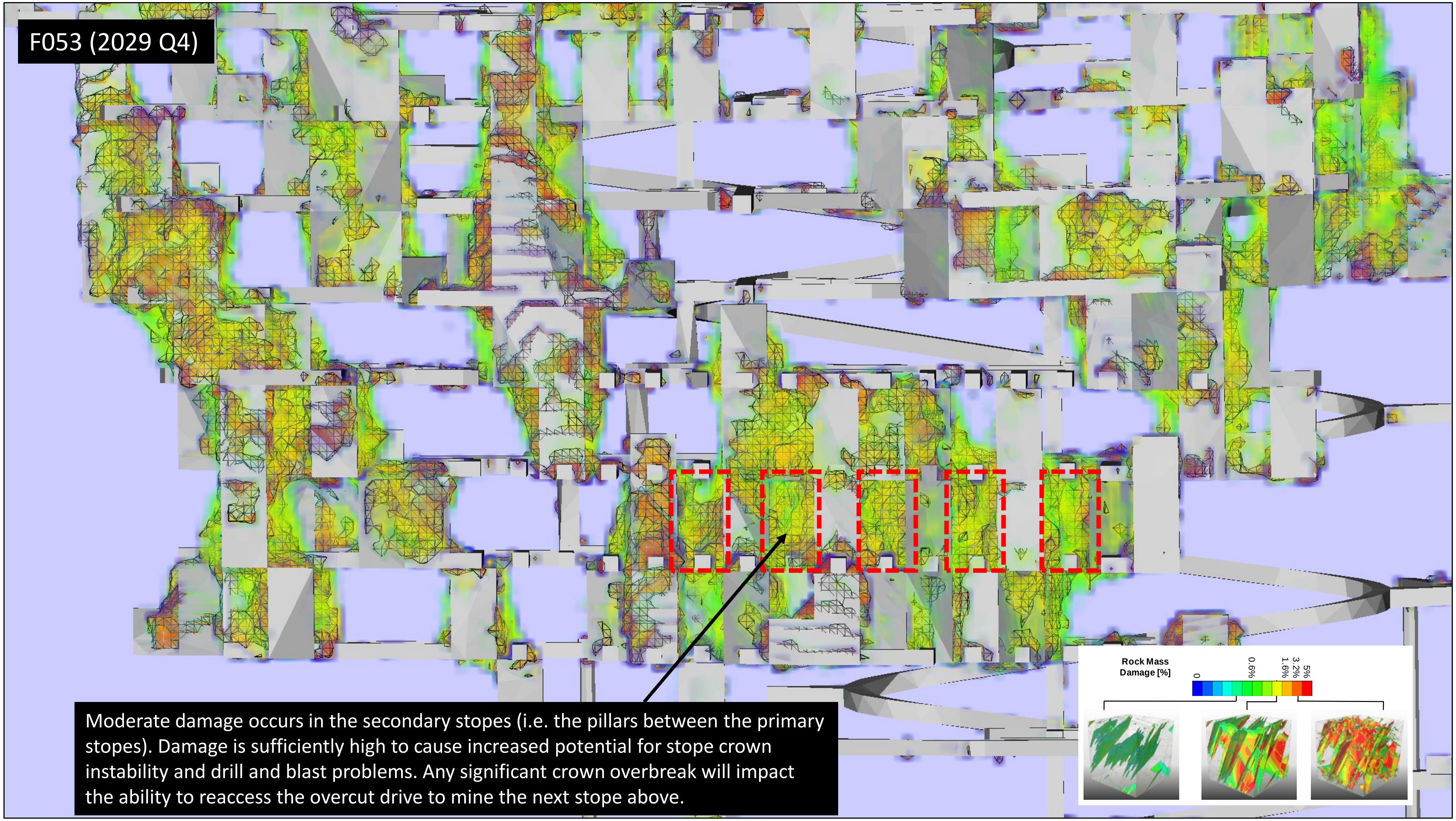


Figure 3-36: Mining to the end of 2029, showing yielding and damage to the rockmass in the secondary stopes

F053 (2029 Q4)

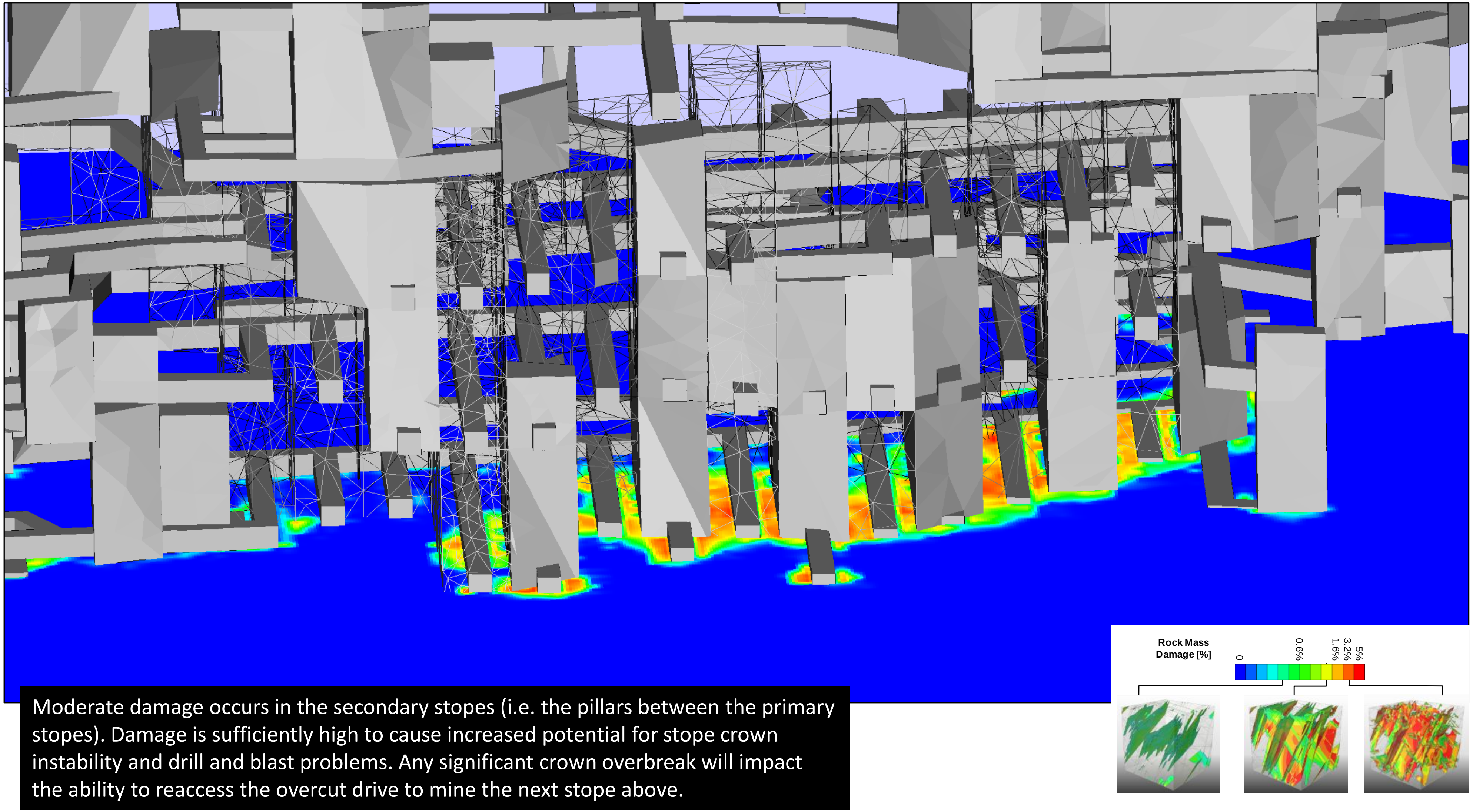


Figure 3-37: Mining to the end of 2029 (solids) and future stoping (wireframes), showing yielding and damage to the rockmass in the secondary stopes

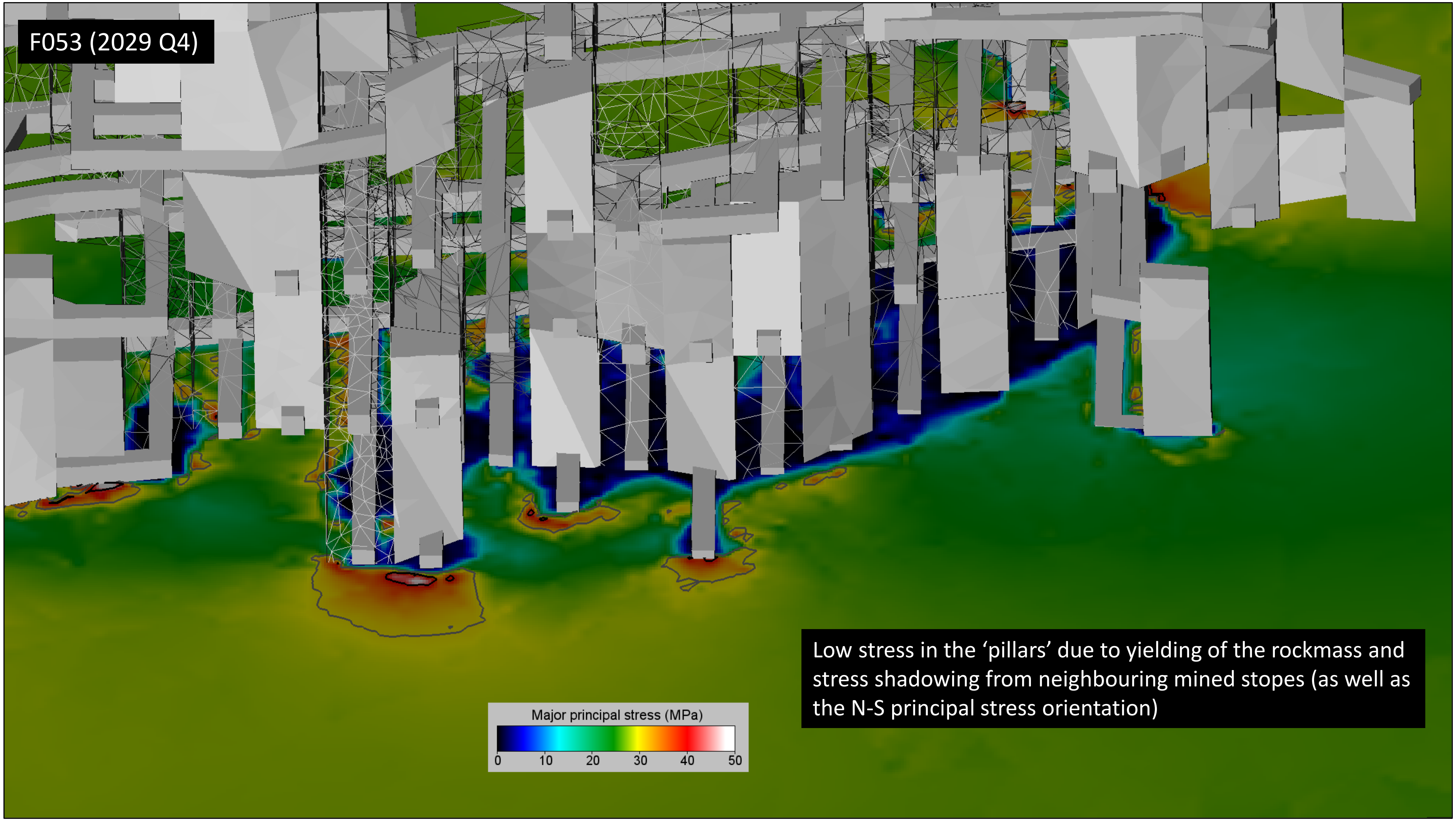


Figure 3-38: Mining to the end of 2029 (solids) and future stoping (wireframes), with a cross section showing major principal stress

F053 (2029 Q4)

Low stress in the 'pillars' due to yielding of the rockmass and stress shadowing from neighbouring mined stopes (as well as the N-S principal stress orientation)

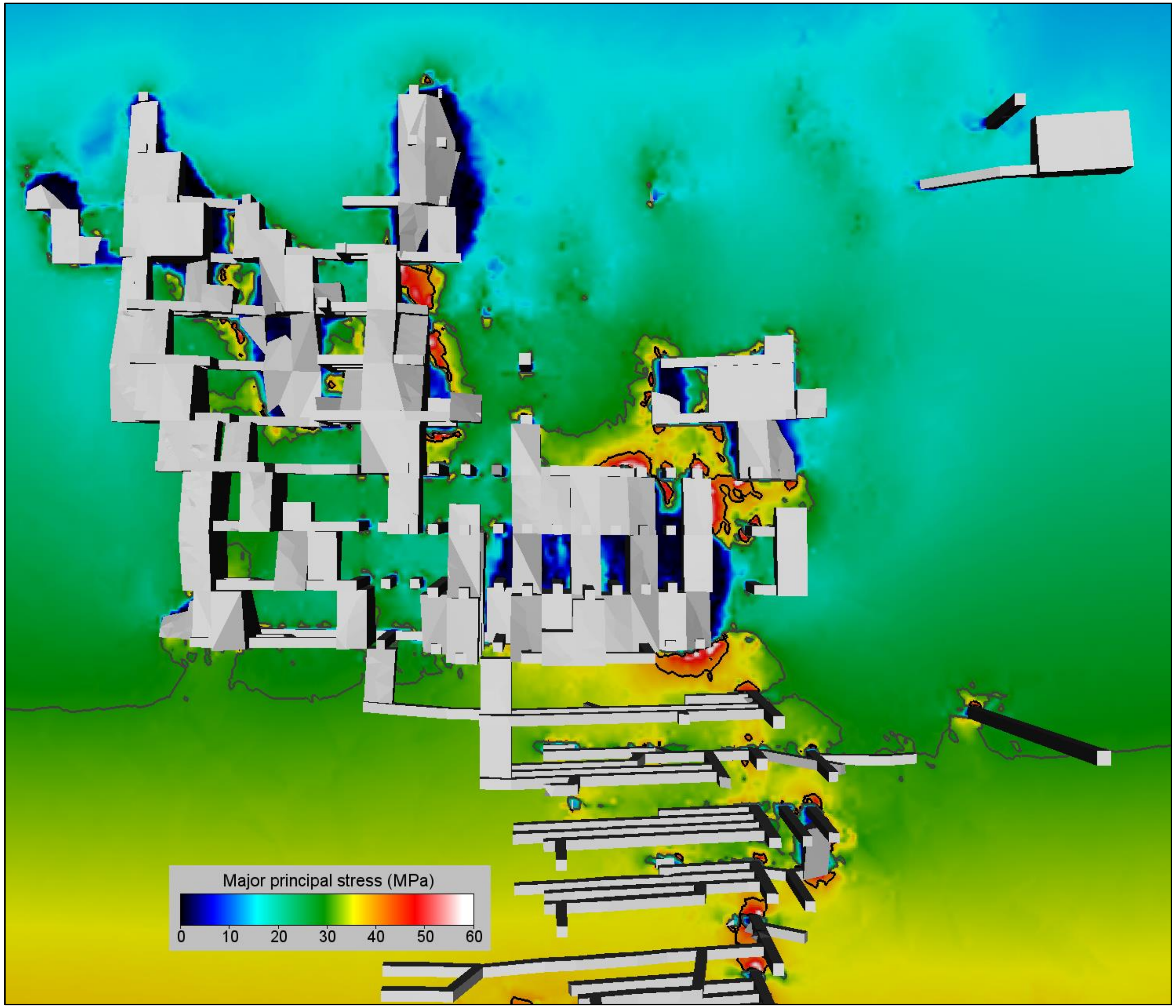


Figure 3-39: Cross section (facing East) showing major principal stress forecasts for mining to the end of 2029 in the transverse stoping blocks in the mid section of the North panel

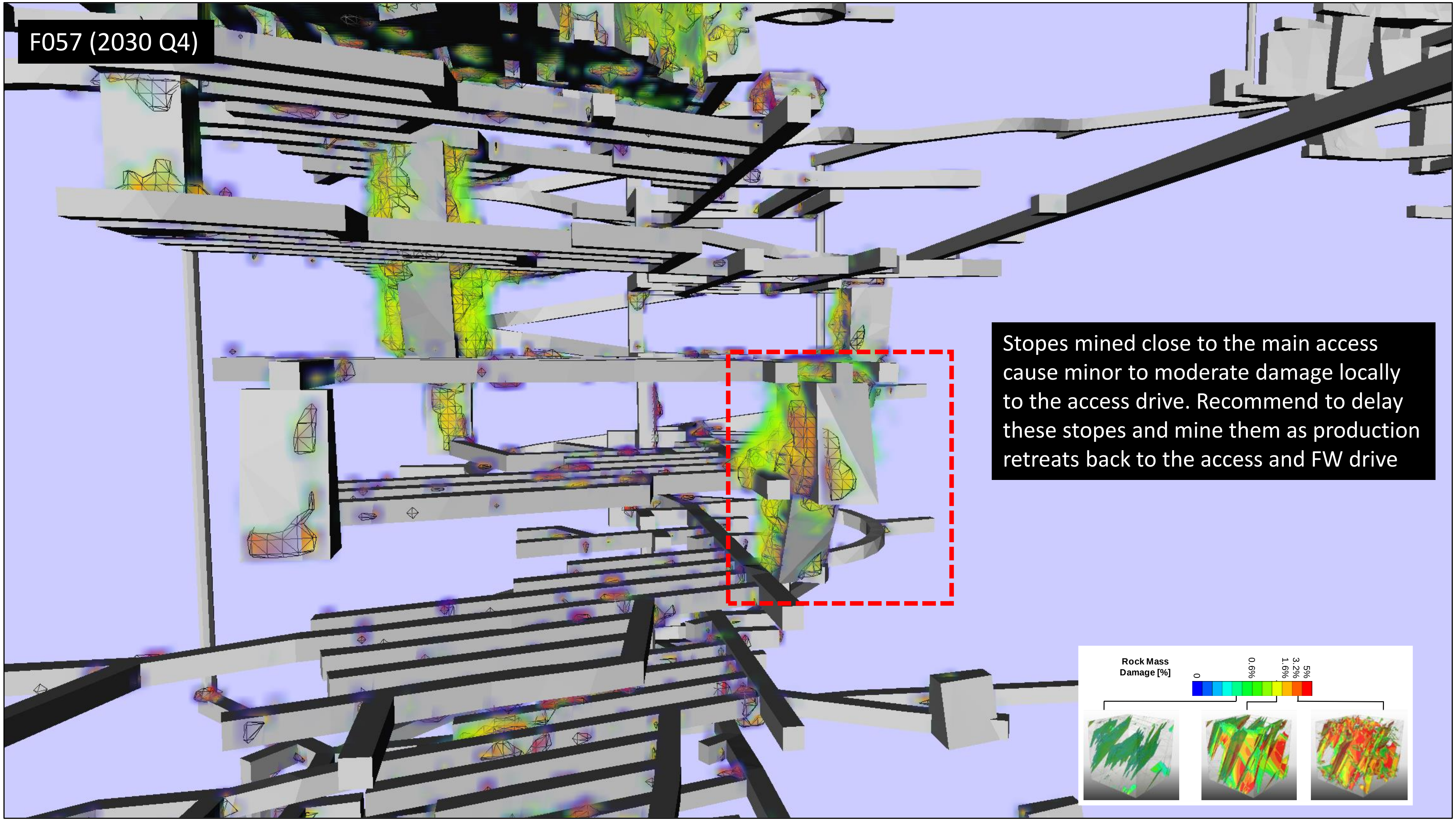


Figure 3-40: Stopes mined close to the level access in the longitudinal section of the low North panel

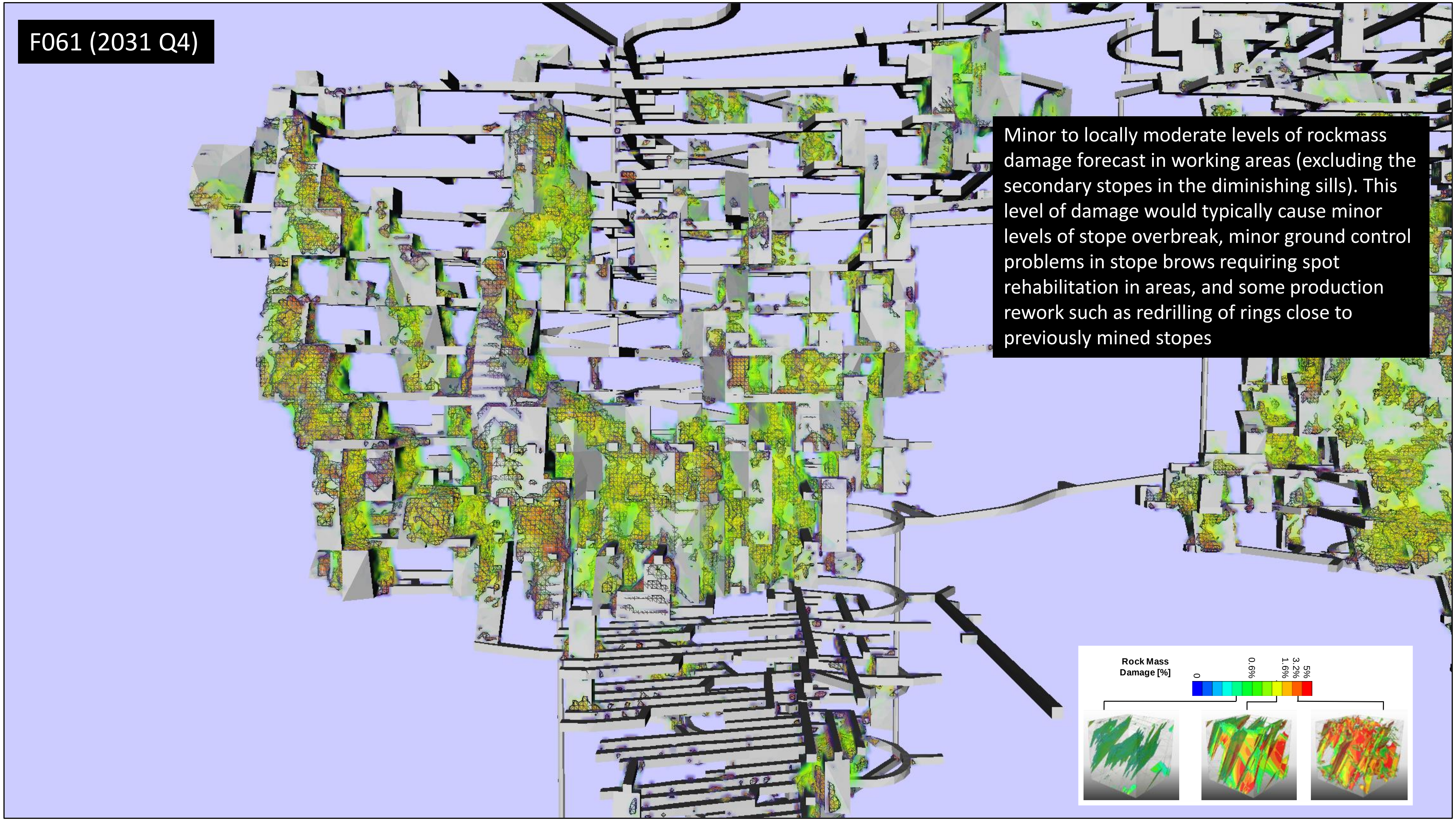


Figure 3-41: Rockmass damage forecasts for the North panel (end 2031)

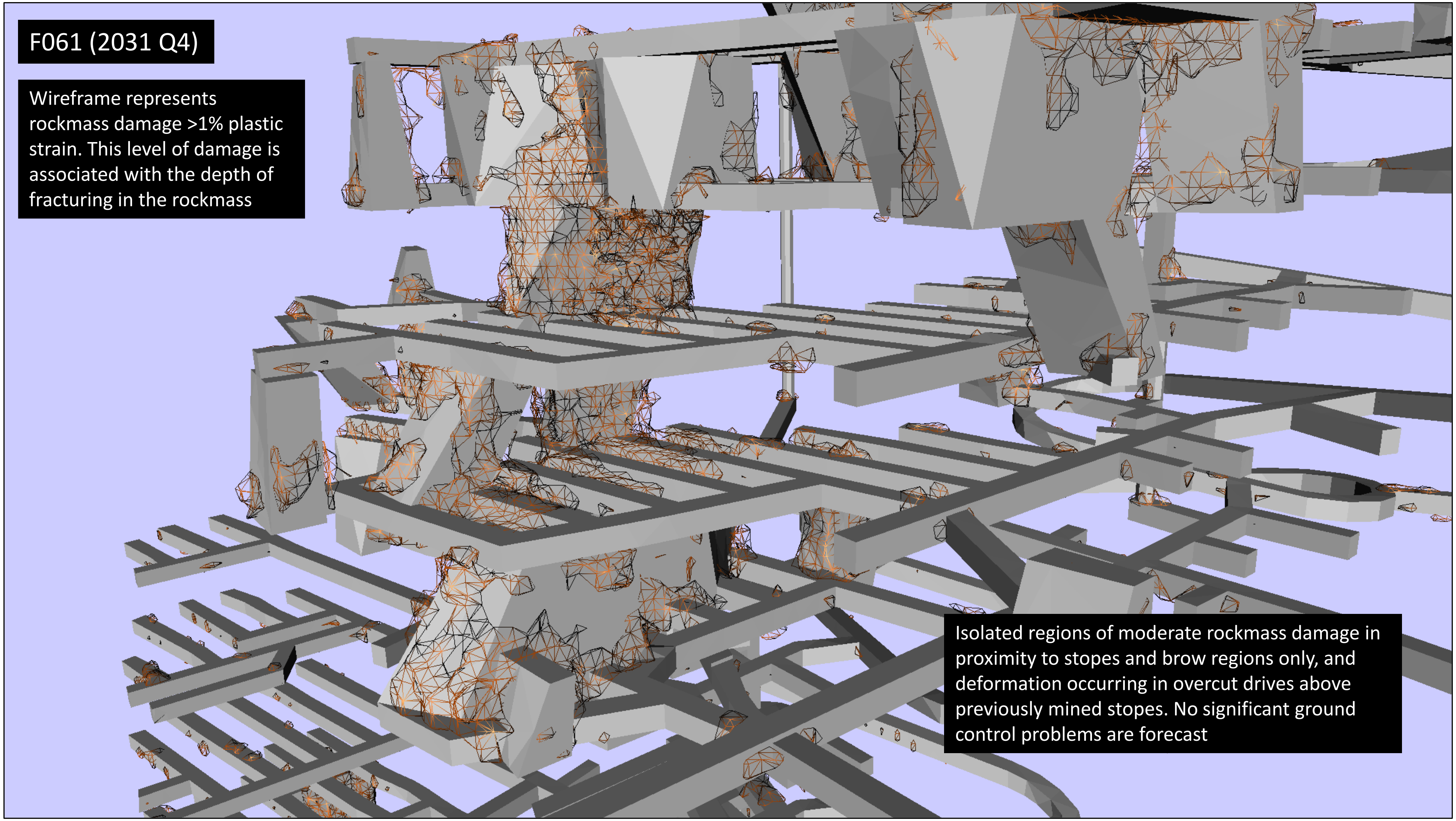


Figure 3-42: Depth of fracturing shown as a wireframe in the longitudinal stoping block in the lower North panel

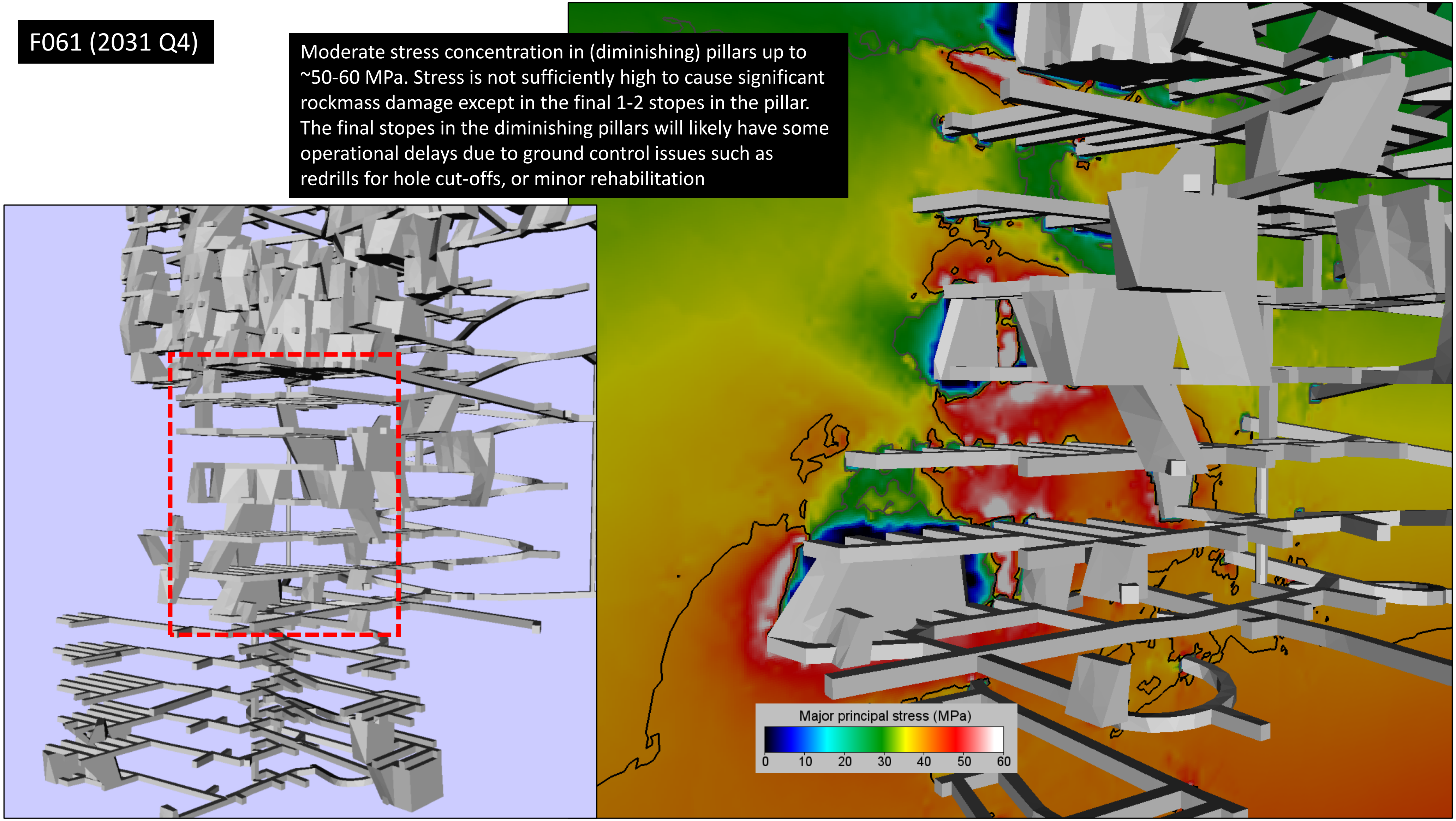


Figure 3-43: Cross section showing major principal stress through the lower North panel and sill pillar between stoping blocks

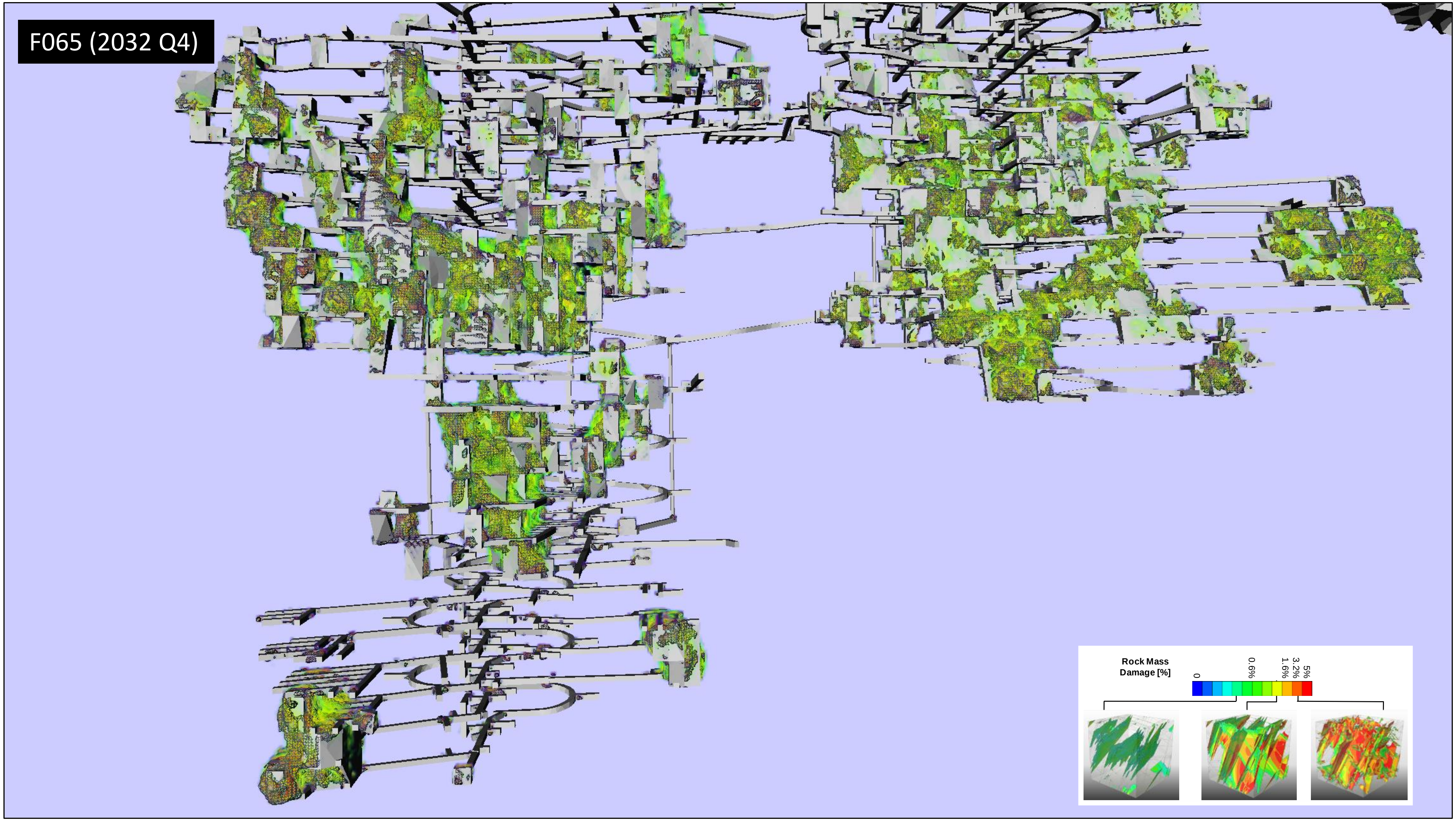


Figure 3-44: Forecast rockmass damage at the end of 2032

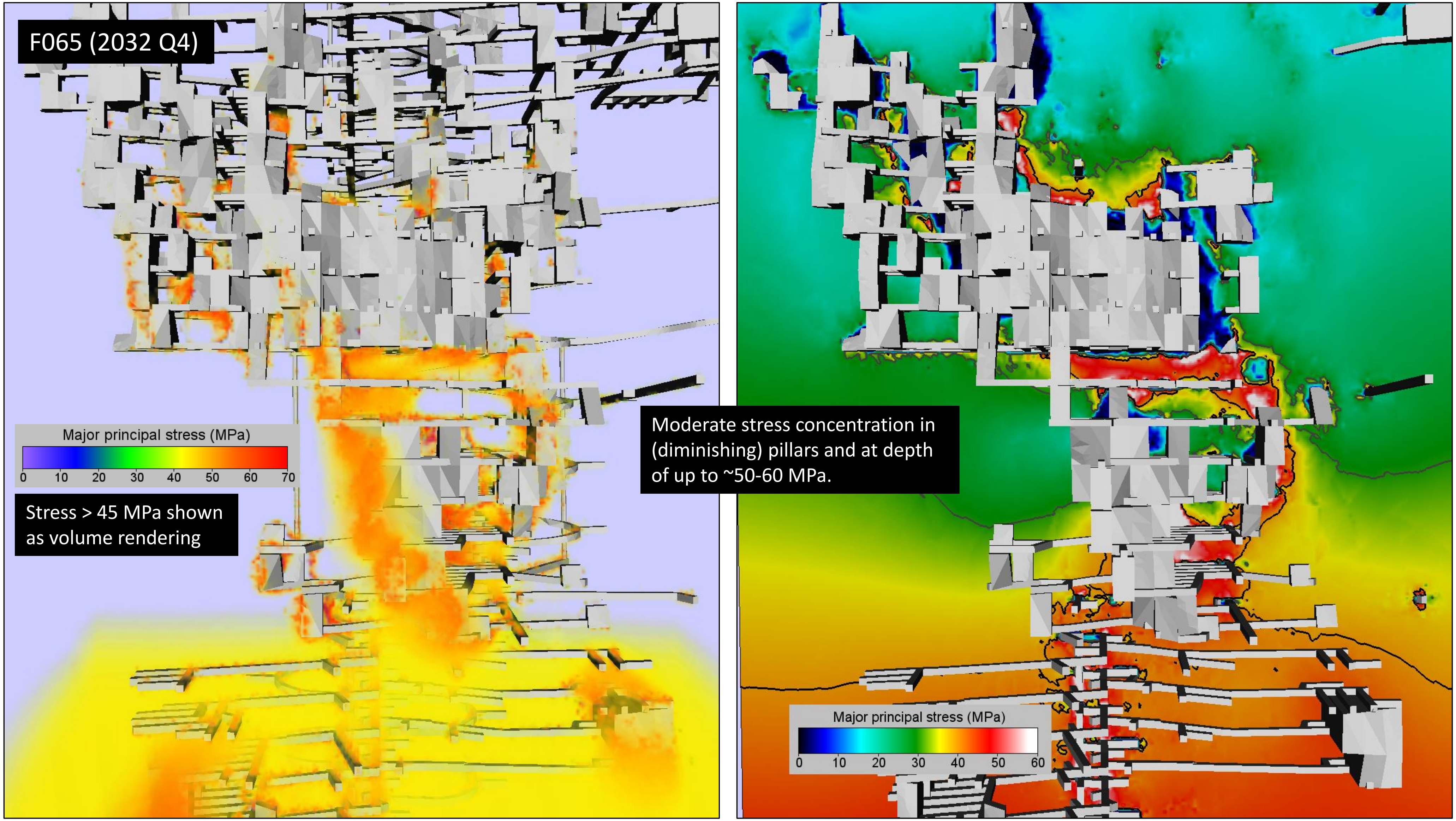


Figure 3-45: Major principal stress forecasts for the North panel in 2032

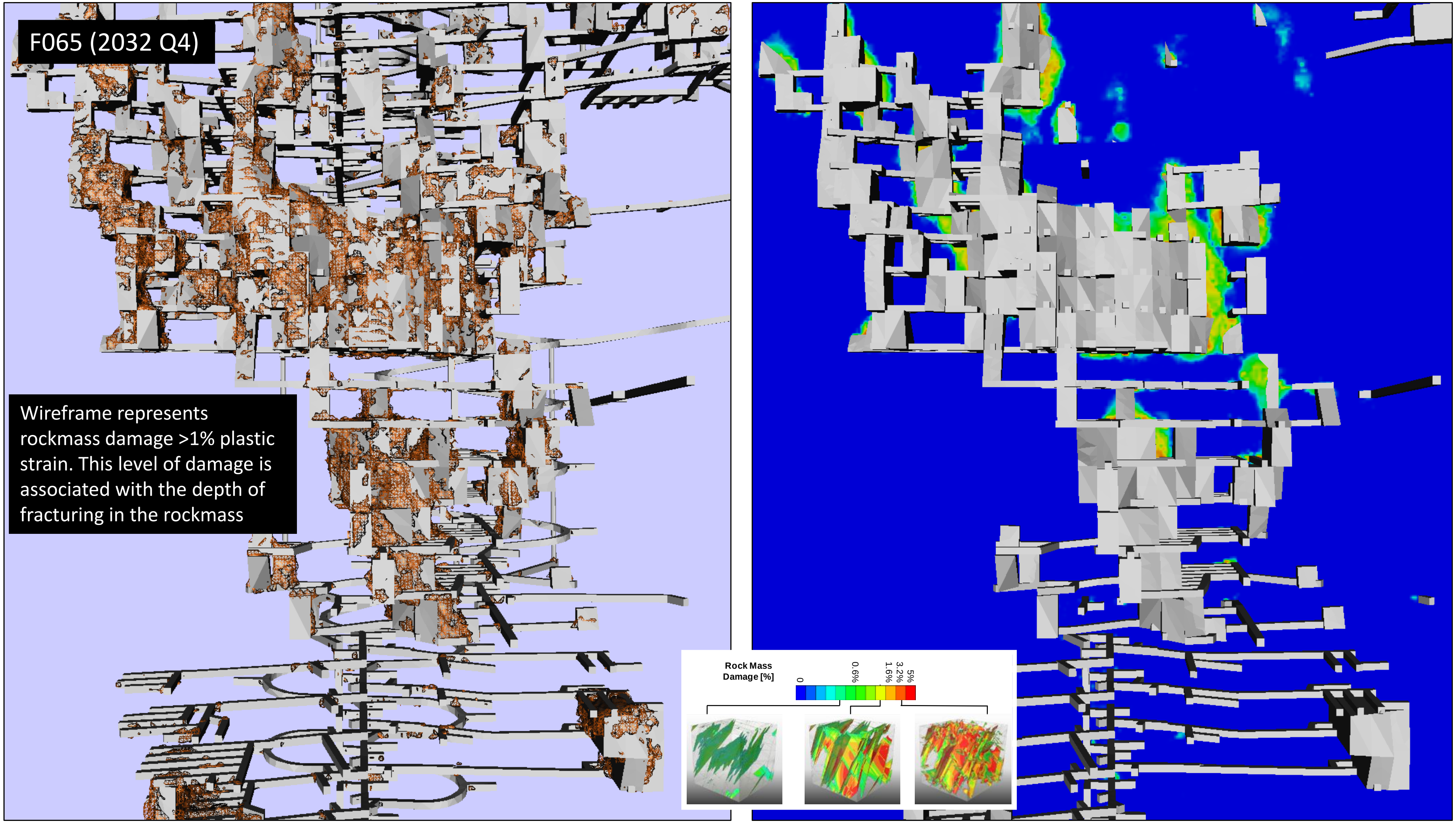


Figure 3-46: Rockmass damage forecasts for the north panel in 2023 (facing east)

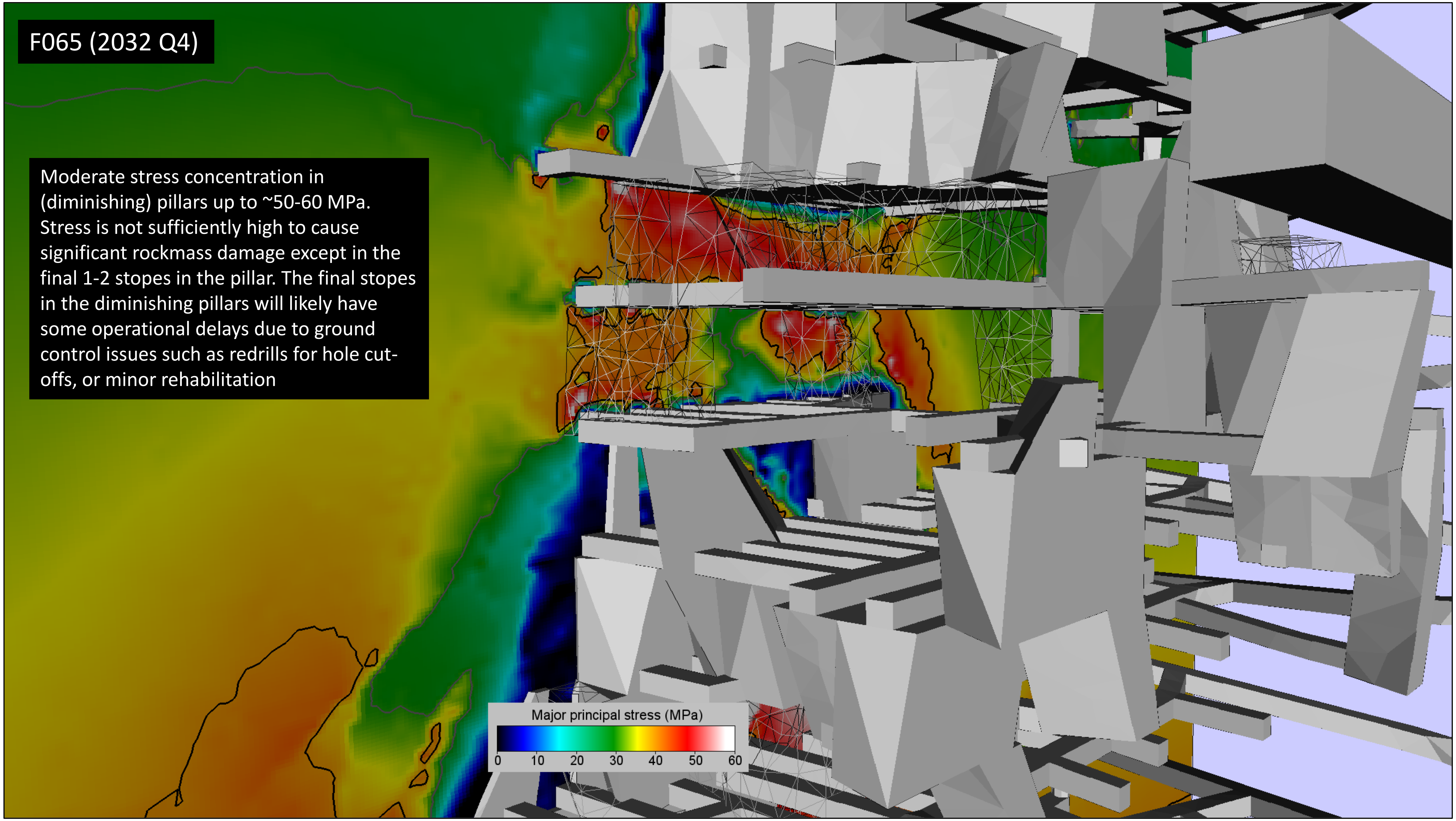


Figure 3-47: Cross section showing major principal stress in the diminishing sill pillar between overhand panels in the middle of the North panel

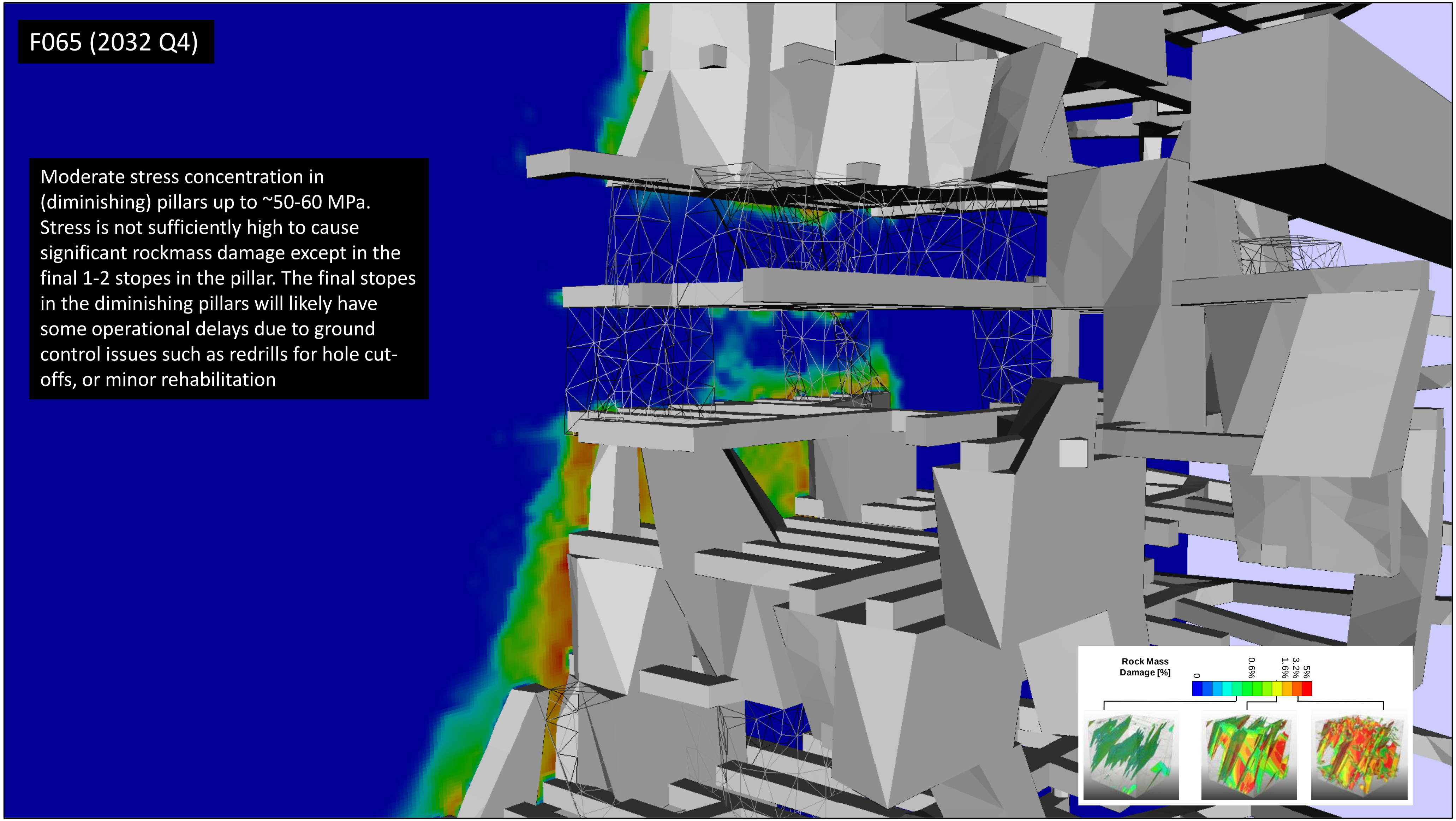


Figure 3-48: Cross section showing rockmass damage forecasts in the diminishing sill pillar between overhand panels in the middle of the North panel

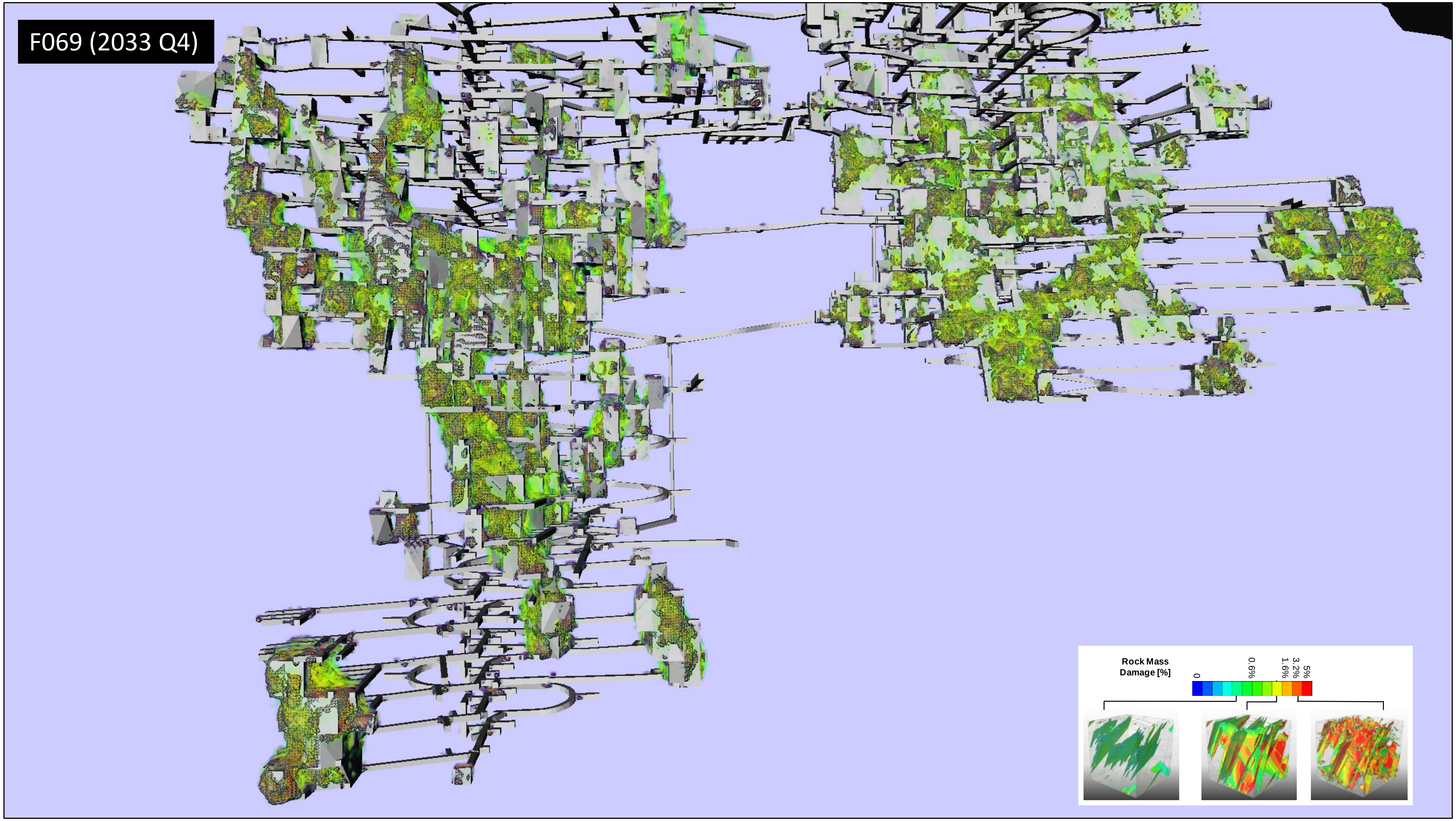


Figure 3-49: Rockmass damage forecasts at the end of 2033

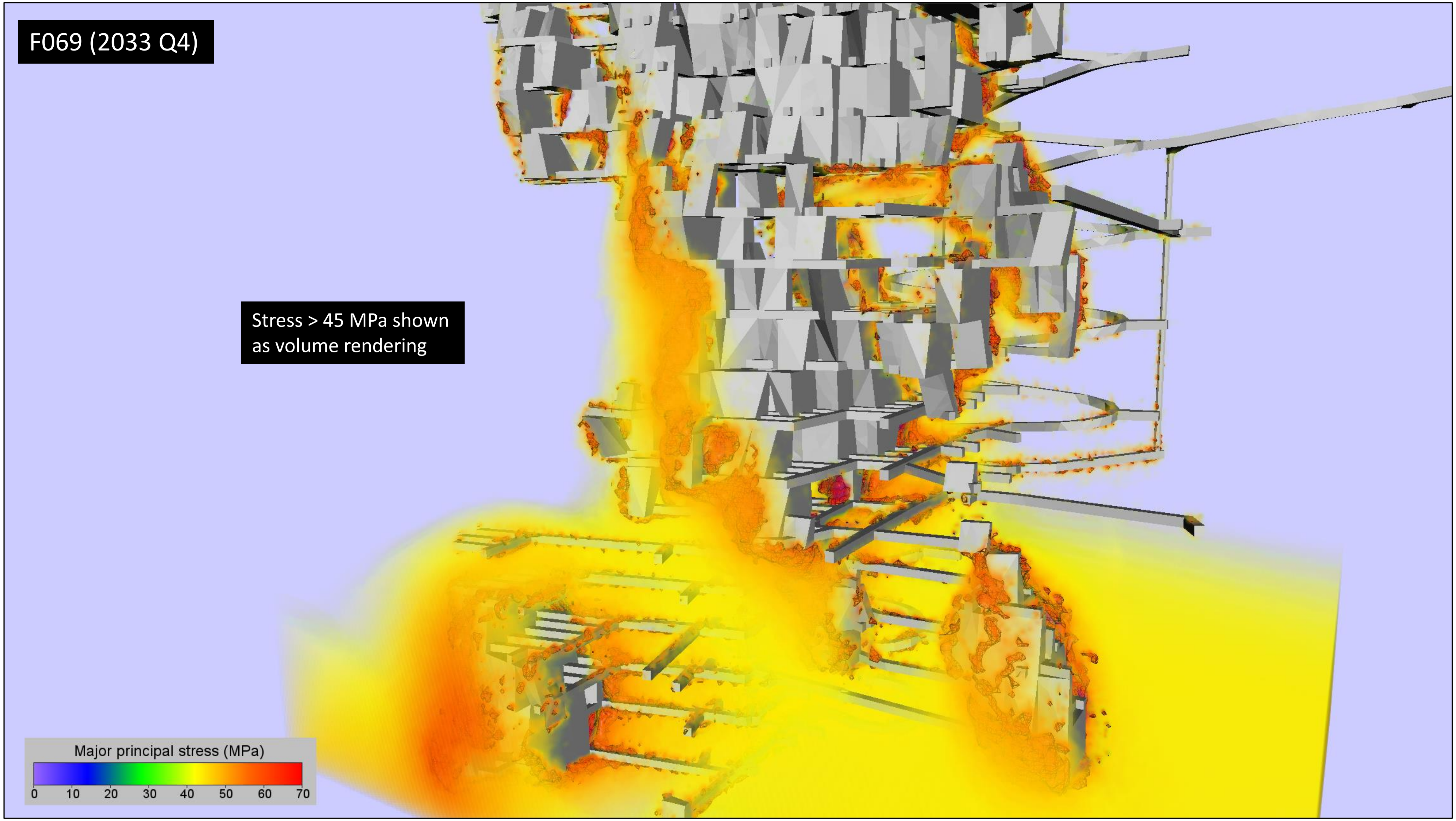


Figure 3-50: Forecast major principal stress (as volume rendering for stress >45 MPa)

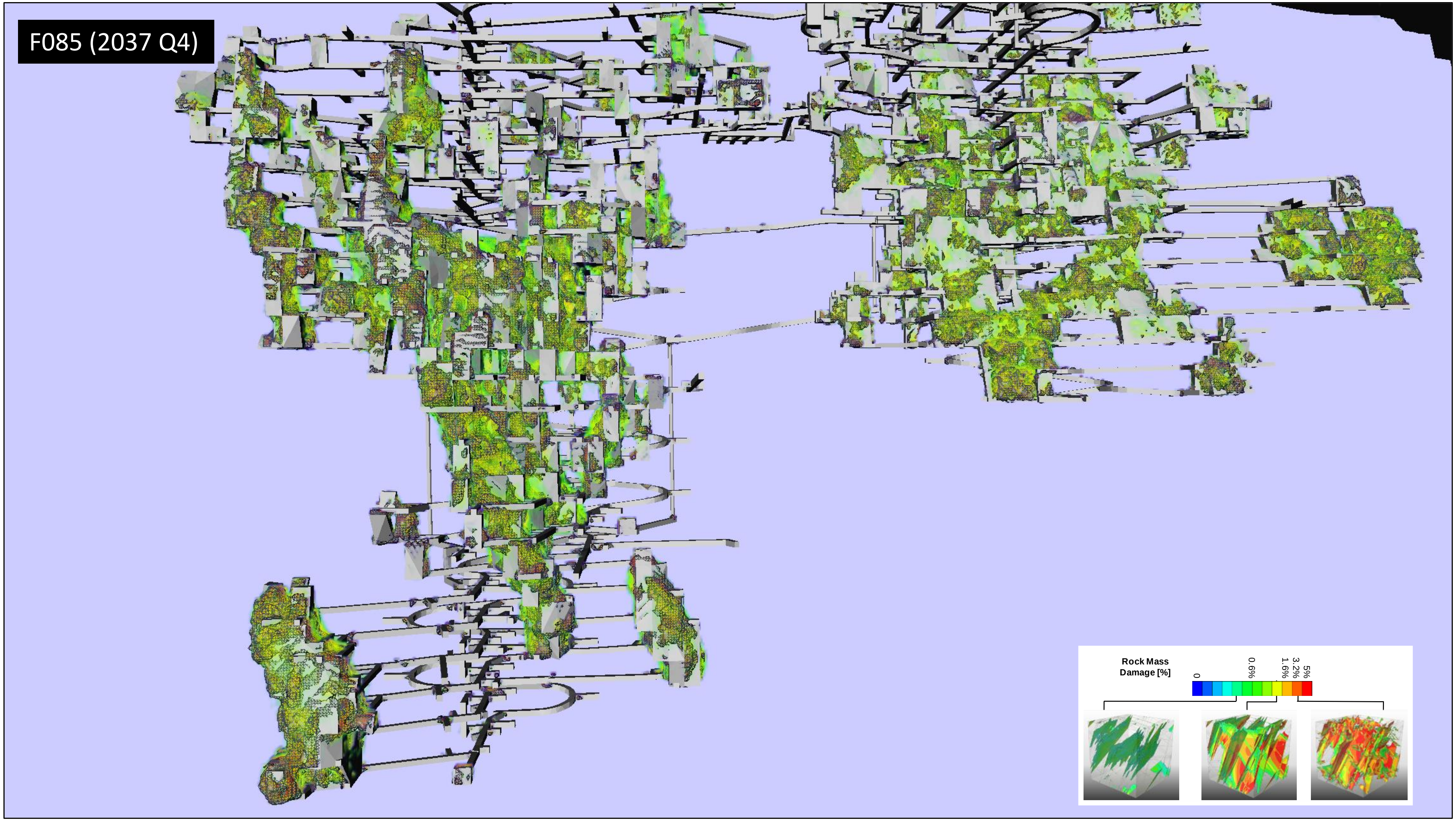


Figure 3-51: Rockmass damage forecasts at the end of mining in 2037

4 CONCLUSIONS, RECOMMENDATIONS & LIMITATIONS

Main findings

Infrastructure and decline layout:

1. The stoping sequence, level layout, decline and infrastructure position vary significantly in each mine design option assessed in this project. This means direct trade-offs and comparisons are difficult as there are multiple variables that change when comparison options. Each design aspect of each option has been assessed individually for the three mine design options provided.
2. Each design option and mining sequence has advantages and disadvantages over its counterparts. The recommended adjustments to the mine plan draws on components from each of the three mine plans which are summarised here and described in detail in the body of the report.
    - The main difference for Design option 4 from the other designs assessed (including the relatively similar design in FW S01) is the additional footwall drive connections in the longitudinal panels. This has potential productivity benefits, however the additional accesses and planned central retreat stoping sequence forms diminishing pillars. This mostly occurs in the South panel. Although this is generally considered unfavourable from a geotechnical perspective, the rockmass conditions are favourable and depth is relatively shallow (i.e. low stress conditions). This results in limited stability and ground control problems are forecasts in these areas.
    - Central declines (and central diminishing pillars) are generally not recommended below a depth of ~500m due to stress concentration causing ground control problems, particularly in the final stopes and near the level accesses. Both the north and south production panels have large barren areas which form waste pillars. This enables a central decline to be positioned with a large waste pillar near the level access without significant ground control problems developing due to the large waste pillars and low extraction ratio. i.e. a diminishing pillar is only formed in isolated parts of the panel due to the barren regions. However, infill drilling may (or may not) identify additional mineralisation and the orebody, and planned stoping may become more continuous than currently known and planned. This would not adversely impact the mine plan above a depth of 400-500m due to the favourable rockmass conditions. However, below a depth of ~500-600m, diminishing central pillars would inevitably be subject to increasing ground control problems such as deformation in drives, elevated stope over break, production delays for rehabilitation and redrilling and potential seismicity.
3. Tight spiral declines and stacked level accesses (i.e. FW S01 design) are generally not recommended due to the high local excavation ratio. However, no adverse stress concentration is forecast in the model for the tight spiral declines or stacked level accesses in the FW S01 option due to the favourable rockmass conditions and relative shallow mining depth. However, a figure 8 type decline such as the layout in Design option 4 is considered to be more (geotechnically) favourable.

Transverse vs longitudinal mining:

4. Each of the four mine designs evaluated in this project include both longitudinal and transverse stope development and mining sequences.
    - Design option #4 and the FW S01 option are predominately longitudinal access drives, with transverse stope development for thicker sections of the orebody in both the north and south panels.
    - Design option 4 has additional longitudinal mining fronts that are formed by more centrally located access drives. These form a central retreat sequence in places of the South panel, including diminishing pillars.
    - Design option 4 can be summarised by:
        - Decline and infrastructure layout is very similar to FW option S01
        - Longitudinal stoping with transverse access in thicker sections of the orebody

- Semi-continuous overhand stoping in the south panels
- Overhand stoping in the thicker sections north panel is primary-secondary overhand stoping, and is less continuous compared to the south panel
- Thicker areas of mineralisation (sometimes multiple lenses) are mined with a mix of longitudinal and transverse drives in the mid section of the north panel
- Diminishing sill pillars within each panel, either the sill pillars between stoping blocks, and where longitudinal stoping retreats to a central access (in some places)
- Six stopes in the current design have crown pillars <10-20m with the top of fresh rock boundary.

5. Multiple Galway splays are present in the FW of the south panel. This causes a number of adverse fault interactions, including:
    - Longitudinal ore drives for the FW stopes are aligned with faults on some levels. These faults are, in areas, subparallel and run along the longitudinal ore drives.
    - Footwall drives run subparallel/parallel to Galway faults in places
    - Galway splay faults bound the FW of the orebody. This causes the final drawpoint brow position to be located at the same location of stopes with transverse access
    - Fault interaction is variable on each level and through the decline in Design option 4. Galway splay faults intersect level accesses, FW infrastructure and intersect part of the decline, particularly for the south mining panel
6. Faults in the north panel are more widely distributed. Generally, interaction with faults, namely the Galway splays, is more adverse in the south panel compared to the north panel.
7. Transverse stoping in thicker sections of the north and south panels provides more active mining areas and is likely to be more favourable for productivity. Long lead-lags between primary and secondary stopes should be avoided (target 2 stope maximum where possible) and a semi-continuous chevron/pyramid mining front should be targeted to minimise extensive diminishing pillars with previously mined panels above as the overhand stoping panels converge with previously mined stoping blocks.

Design option #4

8. The mine layout and sequence is somewhat similar to the FW S01 mine plan
9. Generally isolated areas of minor to moderate rockmass damage (only) for early and mid stages of mining in both the north and south panels
10. Moderate stress concentration occurs in the diminishing central pillar (~30-40 MPa) in the South panel. Stress concentration is limited due to the orientation of the major principal stress and depth of mining. The increase in stress in the diminishing pillar is not sufficiently high to cause significant damage in production areas during extraction of the stopes in the diminishing pillar.
11. Stress concentration of ~50-60 MPa is forecast in the central diminishing pillar in the mid section of the North panel. The stress is not sufficiently high to cause significant or extensive rockmass damage throughout the central pillar. However, this level of stress is sufficiently high (>0.4 x UCS) to cause low levels of seismicity in diminishing pillars.
12. Moderate damage occurs in the secondary stopes in the thicker mid section of the North panel (i.e. the pillars between the primary stopes). These stopes have elevated potential for crown instability, which would impact the ability to re-access the overcut drives and cause production delays during rehabilitation.
13. Primary-secondary stoping with long (vertical) lead-lags not recommended in the lower North panel due to rockmass damage caused in the secondaries. Altering the sequence in the thicker (overhand) stoping blocks to be a more chevron shaped front will help to mitigate ground control problems associated with the close out (sill) pillars. Most problems are forecast to occur in the secondary stopes in the late stages of each stoping block.
14. Transverse layout and sequence has advantages in the thicker section of the North panel. However, the current sequence in design #4 is unfavourable and forms diminishing sill pillars with a flat mining front (including

secondary stopes in the diminishing sill). The upper transverse mining panel is currently split into 2 x two level panels, rather than a single 4 level panel.

15. Transverse stoping and accesses may not be warranted in the narrow sections where single stopes across the orebody width are planned. Galway splay faults adversely impact final stope brows (along the footwall), the footwall drives and some sections of the longitudinal ore drives. We note that characterisation and definition of the major faults in the underground mining precinct is ongoing.
16. A section of the lower North panel is mined using multiple north south ore drives (longitudinal layout) for the multiple lenses. The mineralisation and stope layout is separated for each lens which differs from the more continuous mineralisation and stoping in the (transverse) panel above. Some stopes are close to the main access cause minor to moderate damage locally to the access drive. Recommend to delay these stopes and mine them as production retreats back to the access and FW drive.
17. There are stopes mined out of sequence in both north and south panels. These out of sequence stopes mine out areas along the level access prior to other stopes being mined. A more continuous / progressive retreat sequence towards the level access is recommended. It is likely these out of sequence stopes are minor scheduling errors only, and would be corrected in future design iterations.
18. The barren pillars in both the south and north panel prevents true diminishing pillars from being formed by planned stoping. This of course is not applicable if infill drilling determines the mineralisation is more continuous than currently known.

Recommendations

South panel

1. Six stopes in the current design have crown pillars <10-20m with the top of fresh rock boundary. Recommend to flag these stopes for potential redesign, pending ongoing updates to the geological interpretation of the cover sequence boundaries. These stopes are mined late in the mine life (as per previous recommendations) and this assessment can be done in years to come.

North panel

2. Transverse layout and sequence has advantages in the thicker section of the North panel. However the current sequence in design #4 is unfavourable and forms diminishing sill pillars with a flat mining front (including secondary stopes in the diminishing sill). The upper transverse mining panel is currently split into 2 x two level panels, rather than a single 4 level panel. These should be joined to a single four level panel.
3. The stoping sequence in the North panel should be adjusted to be more continuous in the thicker transverse stoping areas. Primary-secondary stoping in the diminishing sill pillars should be avoided as far as practical.
4. For the lower North panel is mined using multiple north south ore drives (longitudinal layout). We recommend to delay mining of stopes close to the main level accesses prior to mining of the main stoping areas on each level. This will help to delay potential damage to the accesses, as well as delays for bogging, filling and curing of stopes near the access, rather than mining the larger groups of stopes on the levels.

General

5. Confirmation of the pre-mining stress state. Some stress measurements completed are impacted by mining induced effects of the pit and (potentially) the Glenfiddich fault and Cowal shear. The current interpretation of the insitu stress regime provided for this assessment is not aligned with the regional stress field or measured stress at nearby mines. Additional analysis, field measurements and field observation to further confirm the pre-mining stress is required.
6. Compilation and review of rock strength and characterisation data for the open pit and underground datasets. There is no single dataset available at present.

7. Characterisation of known faults, including strength classification, thickness, infill material and general geological description.
8. Fault continuity should be confirmed by Evolution from ongoing drilling and underground mapping.
9. Review faults with updated mapping and ID of faults during mine development. Plan to adjust the mine design for the location of drives, intersections, stope brows etc as required.
10. Adjust the location of intersections to avoid faults during development, where practical. Additional cablebolting will be required where major faults interact with intersections.
11. Do not mine 4-way intersections, as far as practical. Use two 3-way intersections instead.
12. Review mining sequence on each level to ensure level access is not lost due to stopes mined out of sequence.

Limitations

In addition to the normal resolution limits associated with the current finite element model, the main limitations of this assessment are:

1. The most significant limitation of this assessment include:
    a. Confidence in the insitu stress regime. A sensitivity model run was completed in addition to the project scope to assess the effects of a pre-mining stress state more closely aligned with the regional stress field.
    b. Fault properties used in the model are aligned with recent underground modelling projects. It is noted these properties are stronger than previous assessments from 2014 to 2019, and are considered to be high given the geological description of the faults
    c. Fault continuously is unknown and must be confirmed by Evolution from ongoing drilling and underground mapping
2. There is no intermediate scale fault model available and a DFN has not been included in the model
3. Variability in the rock strength properties provided for each lithology domain, including differences in the open pit and underground datasets

Enquiries

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5 REFERENCES

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6 APPENDIX A - LRX FRAMEWORK

Contents

A. CONSTITUTIVE MODEL AND PHYSICAL COMPOSITION

A.1. The LRX constitutive framework

The Levkovitch-Reusch (LRx) constitutive framework is a package of tools that describe the stress-strain behaviour of rock masses and structures. The framework is continuously evolving and developed and a variety of different versions exist. The current version is LR4. The main features of LRx are:

- The continuum regions of the rockmass are modelled as strain-softening dilatant materials. This means that as strain increases the material softens, weakens and dilates. All parameters can vary at different rates with respect to strain changes, and this allows approximation of complex stress-strain behaviour of real rock masses. A generalisation of the Hoek-Brown yield criterion (Hoek et al. 2002) was used for the continuous regions of the rockmass, as described below.
- The behaviour of explicit discontinuities is approximated using cohesive elements. These elements are used because they can capture the mechanical response of thin structures at large deformations, which normal tetrahedral finite elements cannot achieve effectively. Cohesive elements allow simulation of the discrete behaviour associated with structures and can be used to construct a rockmass model compromising continuum regions separated by discontinuities. The structures are free to dislocate, dilate and degrade.
- Small scale structures can be represented in detailed models explicitly as cohesive elements, or ubiquitously by smearing the effects of the joints within the continuum parts of the rockmass.
- Tetrahedral higher-order elements are used for the discretization of the model geometry. These are considered essential for FE models where large gradients of displacements and damage are expected.
- The LRX framework includes provision for hydromechanical coupling when necessary which means that the material constitutive equations (governing mechanical behaviour) are solved at the same time as the equations governing fluid flow in porous media (Darcy's equation), or solved in sequential or staggered incremental schemes, depending on the problem. This means that the modelling framework can capture the effects of pore water pressure on the strength of the rock (as may caused by groundwater percolation through the rockmass itself).
- Seismic potential can be assessed by considering the modelled rate of energy release (RER), which is the maximum instantaneous rate of energy release within a unit volume during a model frame. RER can be correlated with seismic potential and has been successfully applied to forecast seismic potential in several projects. This requires calibration using seismic data for quantitative evaluations of seismic potential.

Model outputs include displacement, stress, strain, and pore water pressure fields, where the presence of pore-water pressure is implemented. Plastic strain, reported as the plastic strain tensor or as scalar equivalent plastic strain measure, represents the amount of plastic rockmass deformation after yield. The plastic strain can be interpreted as rockmass damage and usually correlates well with most engineers' visual interpretation and intuitive understanding of rockmass damage. BE's damage scale is based on plastic strain (see further below how modelled rockmass damage can be interpreted).

A.2. Constitutive model for the continuum parts

The relation between stress, strain, strength and degradation is described by the constitutive model. Generally, constitutive models consist of 3 main parts:

(i) a stress dependent yield criterion,
(ii) a plastic strain potential, which describes how the material will deform because of changes in stress due to damage and
(iii) a description of how stress and strain are related.

In the LRX framework, a generic yield criterion is used that can approximate almost any common rock mechanics yield criterion. In BE models, Hoek-Brown is applied as the base case for most problems.

The starting point for the generic criterion that can approximate Hoek Brown, Mohr coulomb or other criteria is the Menetrey/Willam strength criterion (1), described by the following function

$$\left[\frac{q}{\sigma_{ci}}\right]^2 + m\left[\frac{1}{3}\frac{q}{\sigma_{ci}}R(\theta,e) - \frac{p}{\sigma_{ci}}\right] - s = 0 \qquad \text{A-1}$$

The material constants s and m are the measures of the cohesive and frictional strength, and σ_{ci} represents the uniaxial compressive strength of intact rock. Further,

$p = -\frac{1}{3}\,\boldsymbol{I}\cdot\boldsymbol{\sigma}$ is the hydrostatic pressure,

$q = \sqrt{\frac{3}{2}\,\boldsymbol{S}\cdot\boldsymbol{S}}$ the Mises equivalent stress and

$r = \left[\frac{9}{2}\,\boldsymbol{S}\cdot(\boldsymbol{S}\,\boldsymbol{S})\right]^{1/3}$ the third stress invariant

with $\boldsymbol{S}$ being the deviatoric part of the Cauchy stress $\boldsymbol{\sigma}$. The dependence on the third invariant is introduced via the convex elliptic function in the deviatoric stress plane

$$R(\theta,e) = \frac{4(1-e^2)\cos^2\theta+(2e-1)^2}{2(1-e^2)\cos\theta+(2e-1)\sqrt{4(1-e^2)\cos^2\theta+5e^2-4e}}. \qquad \text{A-2}$$

Here, the variable θ, defined via $\cos 3\theta = (r/q)^3$, is the deviatoric polar angle (also known as Lode angle) and the material constant e is the deviatoric eccentricity that describes the "out-of-roundedness" of the deviatoric trace of the function $R(\theta,e)$ in terms of the ratio between the Mises stress along the extension meridian ($\theta = 0$) and the compression meridian ($\theta = \pi/3$). For $\theta = 0$ and $\theta = \pi/3$ the function becomes $1/e$ and 1 respectevely. The convexity of $R(\theta,e)$ requires that $0.5 \le e \le 1$.

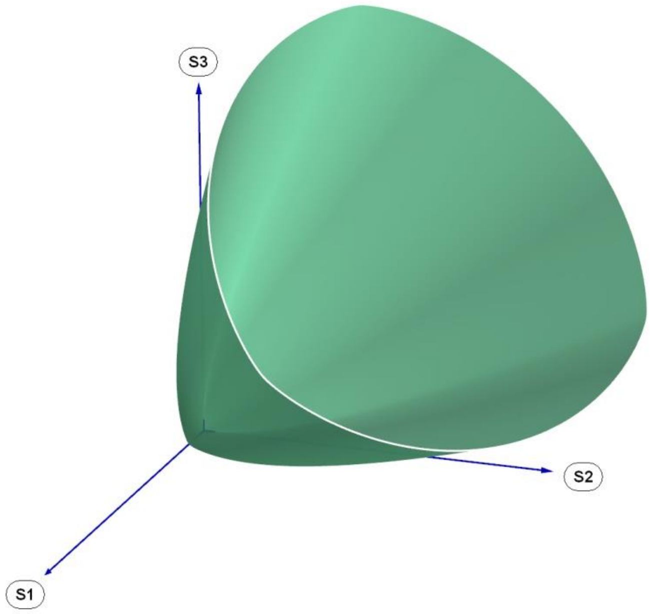

Figure A.1 Three-dimensional representation of the Menetrey/Willam failure surface in the principal stress space

In the case of $e = 0.5$ the Menetrey/Willam failure function represents a circumscribed approximation of the Hoek-Brown (2) strength criterion

$$\left(\frac{\sigma_1 - \sigma_3}{\sigma_{ci}}\right)^2 + m\frac{\sigma_3}{\sigma_{ci}} - s = 0, \qquad \text{A-3}$$

where σ_1 and σ_3 are the major and minor principal stresses at failure. In order to recognize the similarity between both criteria we rewrite the principal stresses representation using the relation between the stress invariants and the principal stresses

$$\sigma_1 = -p + \frac{2}{3}q\cos\theta \text{ and } \sigma_3 = -p + \frac{2}{3}q\cos\left(\theta + \frac{2}{3}\pi\right).$$

Inserting the upper expressions for the principal stresses into [3] one obtains the Hoek/Brown strength criterion in terms of the stress invariants

$$\left[\frac{2}{\sqrt{3}}\frac{q}{\sigma_{ci}}\sin\left(\theta + \frac{\pi}{3}\right)\right]^2 + m\left[\frac{2}{3}\frac{q}{\sigma_{ci}}\cos\theta - \frac{p}{\sigma_{ci}}\right] - s = 0. \qquad \text{A-4}$$

Setting $e = 0.5$ results in an exact match between both criteria at the extension and compression meridians. For $\theta = 0$ and $\theta = \pi/3$ both expressions are reduced respectively to

$$\left[\frac{q}{\sigma_{ci}}\right]^2 + m\left[\frac{2}{3}\frac{q}{\sigma_{ci}} - \frac{p}{\sigma_{ci}}\right] - s = 0 \qquad \text{A-5}$$

$$\left[\frac{q}{\sigma_{ci}}\right]^2 + m\left[\frac{1}{3}\frac{q}{\sigma_{ci}} - \frac{p}{\sigma_{ci}}\right] - s = 0. \qquad \text{A-6}$$

Thus, for $e = 0.5$ the Menetrey/Willam criterion can be considered as a circumscribed approximation of the Hoek/Brown function (Fig.A.2-2).

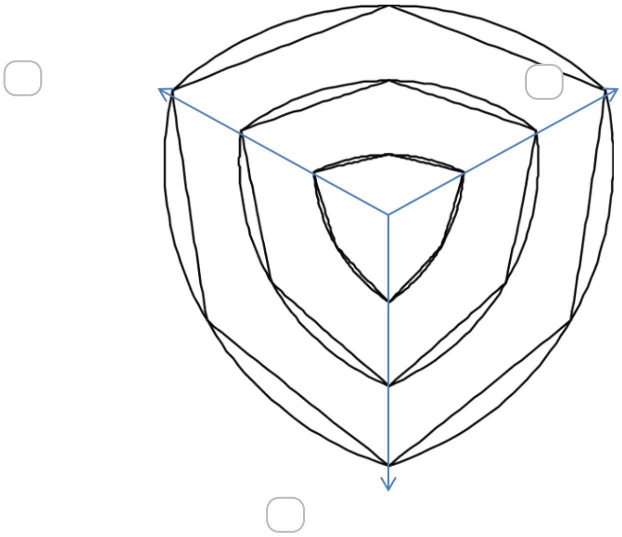

Figure A-2: Comparison between the Deviatoric traces of the Menetrey/Willam failure model (smooth curves) and the 1980 Hoek-Brown criteria at three levels of confinement in the principal stress space

In contrast to the Hoek/Brown model that does not account for the intermediate principal stress, the dependence on σ_2 in the case of the Menetrey/Willam criterion [1] is governed by the eccentricity parameter e. Increasing eccentricity values cause a higher dependence on σ_2 with the deviatoric trace of the Menetrey/Willam model approaching a circle (Fig A.2-3).

Thus, the Menetrey/Willam model possesses a material parameter that can be adjusted to match the true triaxial failure data if this is required.

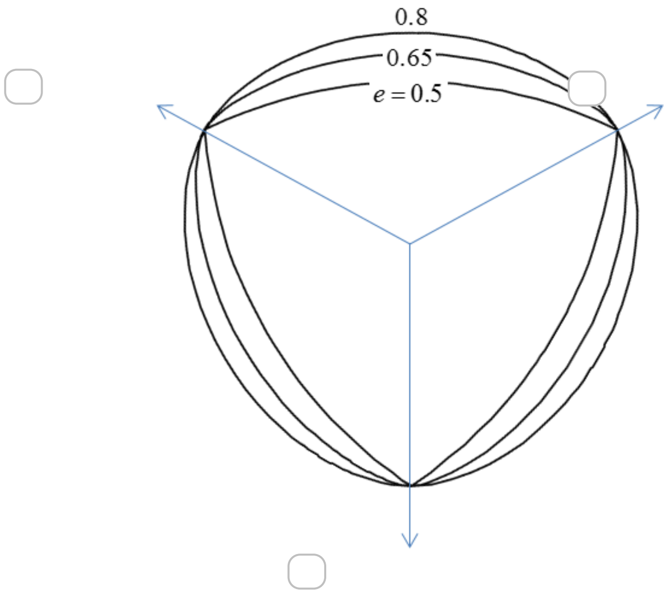

Figure A-3: Deviatoric traces of the Menetrey/Willam failure function for three different eccentricity values.

In 1992 the original Hoek/Brown criterion was extended (3) by an additional parameter a to the following form

$$\left(\frac{\sigma_1-\sigma_3}{\sigma_{ci}}\right)^{\frac{1}{a}} + m\frac{\sigma_3}{\sigma_{ci}} - s = 0, \qquad \text{A-7}$$

that allows to change the curvature of the failure envelope, particularly in the very low normal stress range to account for very low or zero tensile strength in heavily jointed or very poor rock masses. A corresponding extension of the Menetrey/Willam model takes the form

$$\left[\frac{q}{\sigma_{ci}}\right]^{\frac{1}{a}} + m\left[\frac{1}{3}\frac{q}{\sigma_{ci}}R(\theta,e) - \frac{p}{\sigma_{ci}}\right] - s = 0, \qquad \text{A-8}$$

which is the failure criterion in the framework of the LRX model.

Accordingly, the above failure function [7] can be considered as a circumscribed approximation of the 1992 Hoek/Brown (3) criterion.

The plastic strain potential is given by the relation

$$\boldsymbol{D}_p = \dot{\lambda}\frac{\partial G}{\partial \boldsymbol{\sigma}}, \qquad \text{A-9}$$

where $\dot{\lambda}$ is the magnitude of the plastic strain increment and G is the flow potential

$$G = \sigma_{ci}\left[\frac{q}{\sigma_{ci}}\right]^{\frac{1}{a}} + \frac{1}{3}mqR(\theta, e) - d_g p. \qquad \text{A-10}$$

Here, d_g is the dilation parameter in the bulk. If the flow potential differs from the yield function the flow rule is non-associative which is the case for most geotechnical materials.

The model is implemented in such a way that all the strength parameters as well as the dilation and the Elastic modulus can be prescribed as piecewise linear functions of the equivalent plastic strain which is the accumulated deviatoric plastic strain

$$peeq^{dev} = \int_0^t \left(\dot{\lambda}\left\|\left(\frac{\partial G}{\partial \boldsymbol{\sigma}}\right)^{dev}\right\|\right)dt \qquad \text{A-11}$$

to account for the stress-strain behaviour of the rock type, i.e. s, m_b, d_g and the Young's modulus are piecewise linear functions of $peeq^{dev}$. $\|\boldsymbol{A}\|$ is the norm of a tensor $\boldsymbol{A}$ and $(\boldsymbol{A})^{dev}$ the deviatoric part of a tensor $\boldsymbol{A}$.

A.3. Representation of explicit structure

The behaviour of explicit discontinuities is approximated using cohesive elements (formulation COH3D6 in ABAQUS). These elements are used because they can capture the mechanical response of thin structures at large strains, which normal tetrahedral finite elements cannot achieve effectively. Cohesive elements allow simulation of the discrete behaviour associated with structures and can be used to construct a rockmass model compromising continuum regions separated by discontinuities. The structures are free to dislocate, dilate and degrade. The constitutive behaviour of the cohesive elements can be defined using the LRX continuum-based constitutive model, or a constitutive model specified directly in terms of traction versus separation with Coulomb yield criterion with cohesion.

The first approach is typically used to model layers of finite thickness, while the second approach is useful in applications for discontinuities of zero thickness such as fractures. Both models have the LRX feature of elastic-plastic material behaviour in such a way that all the strength parameters as well as the dilation and the Elastic modulus can be prescribed as piecewise linear functions of accumulated plastic strain or the accumulated fault slip.

Discontinuities modelled with continuum LRX material behaviour have the same set of material properties as LRX bulk materials (s. chapter A.2 Constitutive model for the continuum parts).

The main feature of the traction-separation fault behavior is the onset of the fault slip is described by the following cohesive-frictional criterion

$$\tau - p_n \tan\beta - c = 0 \qquad \text{A-12}$$

with c and β being the fault cohesion and friction angle, respectively. Further, τ is the magnitude of the shear stress resolved onto the fault plane and p_n the normal stress acting across the fault. The kinematic of the fault slip deformation is described by the plastic strain rate

$$\boldsymbol{D}_p = \dot{\gamma}[\mathrm{sym}(\boldsymbol{s}\otimes\boldsymbol{n}) + \tan\psi\ \boldsymbol{n}\otimes\boldsymbol{n}] \qquad \text{A-13}$$

with $\dot{\gamma}$ being the fault slip rate and ψ the fault dilation angle. Further, $\boldsymbol{n}$ is the unit normal vector of the fault plane (i.e. the orientation of the finite element) and $\boldsymbol{s}$ the unit vector into the direction of the resolved shear stress. The constitutive fault parameters c, β and ψ are prescribed as piecewise linear functions of the accumulated fault slip γ. The required parameter to define the mechanical behaviour of a traction-separation cohesive section are:

D	Constitutive thickness	
ρ [kg/m³]	Density	
E [GPa]	Elastic modulus	These parameters are a function of the accumulated fault slip.
v	Poisson's ratio	
d	Dilation	
s	Fault cohesion	
a	Fault friction angle	

Table A-1 Material properties for traction-separation cohesive sections

A.4. Extension for the case of transversal isotropy

The isotropic LRX framework is extended for the case of transversal isotropy using the theory of liner stress transformation. The main assumption in this theory is that the anisotropic yield function of the actual stress $\boldsymbol{\sigma}$ is equivalent to an isotropic yield function of the linear transformed stress $\boldsymbol{\sigma}^*$

$$f_{aniso}(\boldsymbol{\sigma}) = f_{iso}(\boldsymbol{\sigma}^*) \qquad \text{A-14}$$

With this approach the usage of an arbitrary isotropic yield function is possible.

The linear stress transformation:

$$\boldsymbol{\sigma}^* = \boldsymbol{L\sigma} \qquad \text{A-15}$$

is performed via a fully symmetric 4th order tensor $\boldsymbol{L}$ that has to satisfy the material symmetry conditions (similar to the elastic stiffness tensor). It is also called the stress weighting tensor. Depending on the material anisotropy type it has different number of independent material constants.

Rock with a population of parallel weakness planes or cracks can be considered as transverse isotropic. With x_3 axis being the symmetry axis and written in the material symmetry frame (Fig A.1-4),

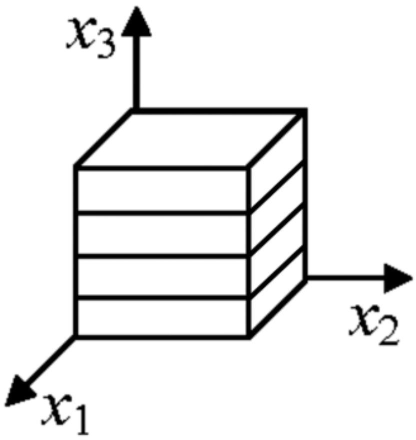

Figure A-4: Material symmetry frame of a transverse isotropic material.

$\boldsymbol{L}$ has the following form:

$$\mathrm{L} = \begin{pmatrix} \mathrm{n} & 0 & 0 & 0 & 0 & 0 \\ 0 & \mathrm{n} & 0 & 0 & 0 & 0 \\ 0 & 0 & 1 & 0 & 0 & 0 \\ 0 & 0 & 0 & \mathrm{n} & 0 & 0 \\ 0 & 0 & 0 & 0 & \mathrm{s} & 0 \\ 0 & 0 & 0 & 0 & 0 & \mathrm{s} \end{pmatrix} \qquad \text{A-16}$$

with only two independent material constants n and s.

To extend the LRX framework for the case of transverse isotropy, the actual stress in the equation [8] is replaced by the stress transformed via [16]

$$\boldsymbol{\sigma}^* = \boldsymbol{L\sigma} = \begin{pmatrix} \sigma_{11}n \\ \sigma_{22}n \\ \sigma_{33} \\ \sigma_{12}n \\ \sigma_{23}s \\ \sigma_{13}s \end{pmatrix} \qquad \text{A-17}$$

The meaning of the anisotropy constants s and n becomes clear if the yield function is analysed for the case of pure shear loading parallel to the cracks and of uniaxial compressive loading parallel to the cracks, respectively.

In the case of pure shear loading parallel to the cracks the yield condition reads:

$$f_{iso}(\boldsymbol{L\sigma}) = f_{iso}(\sigma_{13}s) = 0$$

and $\sigma_{13}s = CS_{iso}$ follows. Accordingly, parameter s represents the reduction factor of the cohesive strength with respect to the isotropic case if shear loading is applied parallel to the cracks.

For the case of uniaxial compressive loading parallel to the cracks (loading direction x_1 or x_2) the yield criterion reads

$$f_{iso}(\boldsymbol{L\sigma}) = f_{iso}(\sigma_{11}n) = 0$$

and $\sigma_{11}n = UCS_{iso}$ follows. Accordingly, parameter n represents the reduction factor of the uniaxial compressive strength with respect to the isotropic case if the uniaxial compressive load is applied parallel to the cracks. If compressive load is applied in x_3 direction $\sigma_{33} = UCS_{iso}$ follows which means that the uniaxial compressive strength perpendicular to the cracks is not influenced by them.

For an arbitrary direction of the uniaxial compressive load with respect to the material symmetry frame the stress weighting tensor $\boldsymbol{L}$ has to be transformed into the loading coordinate system. As a result, the simple diagonal shape is lost and the components of the transformed stress tensor $\boldsymbol{\sigma}^* = \boldsymbol{L\sigma}$ attains shear components that depends also on constant s. Accordingly, the uniaxial compressive strength for such a transverse isotropic material depends on both anisotropy constants.

The pictures below show the dependence of UCS from the rotation angle of the load axis relative to x_3 axis for load direction varying from 0^0 (perpendicular to the cracks) to 90^0 (parallel to the cracks) for different combinations of s and n values.

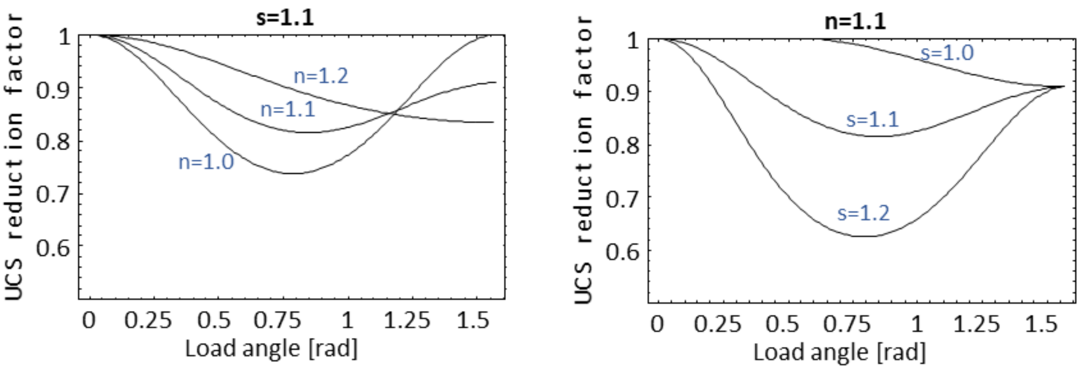

Figure A-5: Influence of the loading direction on UCS for different combinations of n and s values

A.5. Model parameter to determine rock strength

The application of the constitutive model for a particular rock type or the mechanical behaviour of a discontinuity requires the determination of a set of model parameters. One common approach is to determine the model parameter with help of the GSI (geological strength index) system (see (3) and (4) for the application) and optionally the value m_i (frictional strength of the intact rock mass): This allows an initial determination of elastic properties E and ν, the frictional strength of the broken rock m_b and the cohesive strength s as well as the dilation.

UCS [MPa]	Uniaxial Compressive Strength	
GSI	Geological Strength Index	
m_i	Frictional strength of intact rock	
D	Damage parameter (Hoek-Brown)	

ρ [kg/m³]	Plastic strain	
m_b	HB parameter for frictional strength of broken rock	These parameters are a piecewise linear function of the accumulated plastic strain.
E [GPa], v	Elastic modulus, Poisson's ration	
d	Dilation	
s	cohesive strength parameter	
a	strength parameter	

Table A-2 Material properties for continuum LRX material

An example for the documentation of material properties is provided in the next figure:

Material properties M01

Name	ρ [kg/m³]	UCSi [MPa]	GSI	n_{aniso}	s_{aniso}	Level	ε_{plast} [%]	E [GPa]	v	s	m_b	a	e	dilation
ALBITOFIRO	2850.00	230.00	77.50	1.00	1.00	PEAK	0.00	49.28	0.25	7.32E-3	4.30	0.50	0.60	0.72
						TRANS	0.91	41.26	0.25	8.93E-4	2.29	0.50	0.60	0.38
						RES	5.26	37.41	0.25	1.00E-5	1.01	0.50	0.60	0.00
ALBITOFIRO_C	2850.00	230.00	77.50	1.00	1.00	PEAK	0.00	41.26	0.25	8.93E-4	2.29	0.50	0.60	0.38
BHT	2850.00	230.00	77.50	1.00	1.00	PEAK	0.00	49.28	0.25	7.32E-3	4.30	0.50	0.60	0.72
						TRANS	0.91	41.26	0.25	8.93E-4	2.29	0.50	0.60	0.38
						RES	5.26	37.41	0.25	1.00E-5	1.01	0.50	0.60	0.00
BHT_C	2850.00	230.00	77.50	1.00	1.00	PEAK	0.00	41.26	0.25	8.93E-4	2.29	0.50	0.60	0.38
KA	2450.00	100.00	65.00	1.00	1.00	PEAK	0.00	19.55	0.25	2.10E-3	1.73	0.50	0.60	0.29
						TRANS	1.38	16.96	0.25	4.62E-4	1.10	0.50	0.60	0.18
						RES	11.38	15.00	0.25	1.00E-5	0.59	0.50	0.60	0.00
KA_C	2450.00	100.00	65.00	1.00	1.00	PEAK	0.00	16.96	0.25	4.62E-4	1.10	0.50	0.60	0.18
KPCLS	2850.00	230.00	77.50	1.00	1.00	PEAK	0.00	49.28	0.25	7.32E-3	4.30	0.50	0.60	0.72
						TRANS	0.91	41.26	0.25	8.93E-4	2.29	0.50	0.60	0.38
						RES	5.26	37.41	0.25	1.00E-5	1.01	0.50	0.60	0.00
KPCLS_C	2850.00	230.00	77.50	1.00	1.00	PEAK	0.00	41.26	0.25	8.93E-4	2.29	0.50	0.60	0.38
KPCMIX	2800.00	122.00	60.00	1.00	1.00	PEAK	0.00	22.94	0.25	1.27E-3	1.67	0.51	0.60	0.28
						TRANS	0.62	21.04	0.25	5.69E-4	1.31	0.51	0.60	0.22
						RES	8.81	17.72	0.25	1.00E-5	0.66	0.51	0.60	0.00
KPCMIX_C	2800.00	122.00	60.00	1.00	1.00	PEAK	0.00	21.04	0.25	5.69E-4	1.31	0.51	0.60	0.22
KPCSB	2850.00	170.00	54.50	1.00	1.00	PEAK	0.00	30.59	0.25	7.34E-4	1.74	0.51	0.60	0.29
						RES	5.84	23.83	0.25	1.00E-5	0.82	0.51	0.60	0.00
KPCSB_C	2850.00	170.00	54.50	1.00	1.00	PEAK	0.00	30.59	0.25	7.34E-4	1.74	0.51	0.60	0.29
LAVAS	2850.00	230.00	77.50	1.00	1.00	PEAK	0.00	49.28	0.25	7.32E-3	4.30	0.50	0.60	0.72
						TRANS	0.91	41.26	0.25	8.93E-4	2.29	0.50	0.60	0.38
						RES	5.26	37.41	0.25	1.00E-5	1.01	0.50	0.60	0.00
LAVAS_C	2850.00	230.00	77.50	1.00	1.00	PEAK	0.00	41.26	0.25	8.93E-4	2.29	0.50	0.60	0.38
FAULTS_LR	2700.00	60.00	40.00	1.00	1.00	PEAK	0.00	9.54	0.25	1.72E-4	0.64	0.52	0.60	0.11
						RES	12.74	7.73	0.25	1.00E-5	0.46	0.52	0.60	0.00

Figure A-6: Example for documentation of material properties of the LRX framework.

A.6. Modelling softening behaviour

A set of these parameters describes the onset of yielding for a rock type. To describe the post-yield behaviour of stress-strain relation of the rock, the implementation of the constitutive model allows an arbitrary number of characteristic points to describe the stress-strain curve of the material.

The image below shows frequently used idealizations for the softening behaviour of the rock materials. (P) denotes the peak strength material, (T) indicates the onset of softening and (R) examples for the residual strength level.

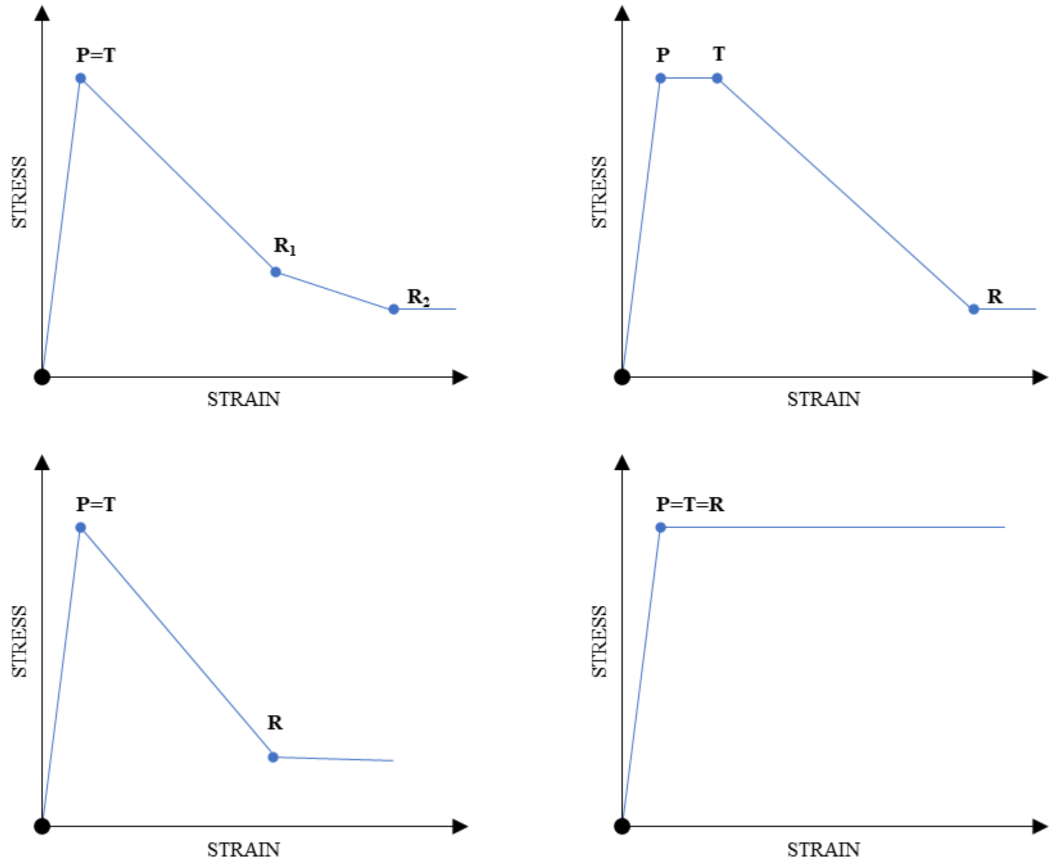

Figure A-7: Idealizations for the softening behaviour of the rock materials. (P) denotes the peak strength material, (T) indicates the onset of softening and (R) examples for the residual strength level.

In the LRX framework the softening behaviour is introduced in such a way that all the strength parameters as well as the dilation and the Elastic modulus can be prescribed as piecewise linear functions of accumulated plastic strain to account for the stress-strain behaviour of the rock type, i.e. d_g, s and $m_b$and the Young's modulus can evolve independently according to the available laboratory data or available description of the deformation and damage behaviour rock mass.

A.7. The common damage scale

As a purely phenomenological model the constitutive equations do not incorporate a damage variable that allows the direct quantification of the damage state of the rock.

For non-linear elastic-plastic models as used in the LRX framework the rock mass damage is related to the amount of accumulated equivalent plastic strain, which is the amount of permanent (irreversible) rockmass deformation after yield. The table below shows a possible correlation of plastic strain values with the damage state of the rock. The specific correlation of plastic strain levels with damage states is often referred to as the "common damage scale (CSD)", which can vary depending of the softening behaviour of the investigated rock.

Plastic strain	Damage state	Observed behaviour
>5%	Very significant	Gross distortion and comminution.
~3%	Significant	Extensive fracturing of intact rock.
~1.5%	Moderate	Constant load leads to increasing deformation.
~0.7%	Minor	No significant decrease in strength or stiffness.
<0.35%	None to very minor	Undisturbed in situ conditions.

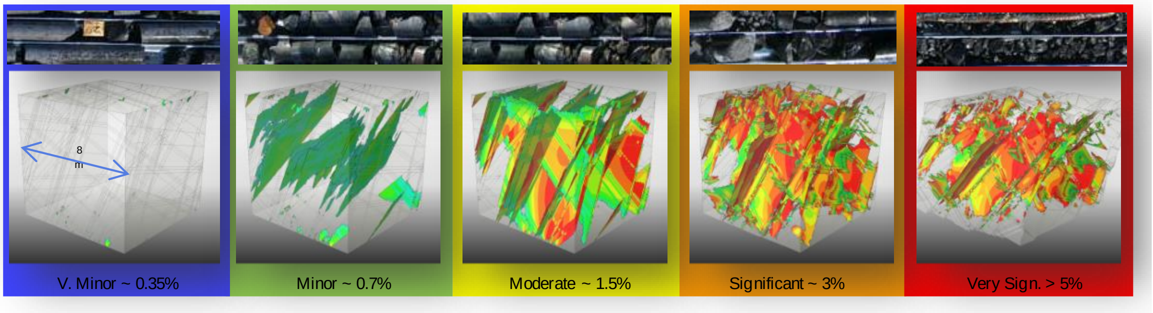

Table A-8 Correlation of plastic strain values with the damage state of the rock

A.8. Assessing seismic potential with RER

The mining of excavations in rock re-distributes stress and causes damage to the rock mass and discontinuities. The resulting reduction in strength and degradation in stiffness of the damaged rock and structures leads to further deformation and release of stored elastic strain energy.

One portion of this released energy is consumed by the damage process - frictional sliding and the creation of new surfaces. This energy cannot be retrieved, so is counted as 'dissipated'. If the value of the released elastic energy is higher than the energy dissipated by the irreversible damage, the surplus is emitted into the surrounding rock. These release events are seismic events.

The magnitude (and/or the rate) of the released energy during these events can be measured in a mine using a seismic monitoring system or calculated using a model. The instantaneous, peak (i.e. maximum) rate of energy release from a volume of rock (i.e. the energy that is not dissipated) is the Rate of Energy Release (RER).

The calculated rate of energy release (RER) is used to represent seismic potential in the model. Levkovitch et al. (2013) describe RER in some detail. RER is calculated as follows:

Each model frame comprises many numerical time steps as part of the explicit FE solution procedure. For each time step, the instantaneous rate of energy release is calculated for each finite element. This is the change in elastic strain energy less the dissipated plastic energy, and represents the energy radiated from the element out into the surrounding environment. The dissipated plastic energy represents irreversible work done on the rockmass through processes such as friction on joint surfaces and creation of new fractures, and is calculated from the plastic strain condition of the element.

The RER is the maximum value of the instantaneous rate of energy release calculated all the time steps during a model frame.

RER is recorded for every tetrahedral element and every cohesive element in the FE simulation at every frame. This allows RER to be calculated for the homogenised rockmass (represented with tetrahedral elements), and for the explicit structures (represented with cohesive elements). Both are important: The largest events are expected on structures, but many lower magnitude events are expected in the homogenised rockmass.

A.9. Mechanical response in the presence of pore-water pressure

In the LRX framework the governing rock or soil is regarded as a deformable porous medium, consisting of a solid skeleton and a pore space. A fluid (e.g., water) may partially or fully saturate this pore space and is allowed to flow through connected pores, i.e, to permeate through the rockmass. Within the conceptual modelling approach both the skeleton and the voids are considered to be homogeneously smeared within the Representative Volume Element (RVE), where the proportion of pore volume space to the bulk volume is denoted as porosity.

At any material point in the model, the fluid is subjected to a fluid pressure. The spatial distribution of the fluid pressure does vary and results from the respective hydro-geological setting. This pressure is obtained as a result of a separate hydrological analysis.

The fluid interacts with the solid rock skeleton. In case of a single-phase water flow the respective fluid pressure acting on the solid skeleton is referred to as pore-water-pressure p_w, or, in case of a multi-phase flow, as wetting phase pressure.

The stresses of the entire RVE, denoted as total stresses, can be decomposed in two parts. One part is represented by the effective stresses of the solid skeleton, and the other part by the fluid pressure acting onto the solid skeleton. This is referred to as effective stress concept of Terzaghi (1936):

$$\boldsymbol{\sigma}_{tot} = \boldsymbol{\sigma}_{eff} + \alpha_B p_w \mathbf{1}\,. \tag{A-18}$$

The sign convention is such that p_w being positive in compression, and of $\boldsymbol{\sigma}$ negative in compression, i.e., $p = -1/3\,\mathrm{tr}(\boldsymbol{\sigma})$. Further, α_B denotes the Biot coefficient which is a material parameter depending on the rock type that is generally bound between $0 < \alpha_B \leq 1$. Typical values for the Biot coefficient are summarized in the literature for a range of materials. Total stresses are always used to fulfil the linear momentum (equilibrium). The constitutive response of the porous material, however, is always updated using the effective stresses. Hence, the presence of pore-water pressure reduces the skeleton stresses such that the effective confinement pressure is reduced, and the material may be subject to earlier yielding. As a special case, a pore-water pressure exceeding the total confining pressure, i.e., $p_w > -1/3\,\mathrm{tr}(\boldsymbol{\sigma}_{tot})$, results in a plastic apex-mode deformation, also referred to as tensile cracking. This situation may arise in cases where a large p_w is present in a de-stressed material region, such as near a free surface.

A.10.References

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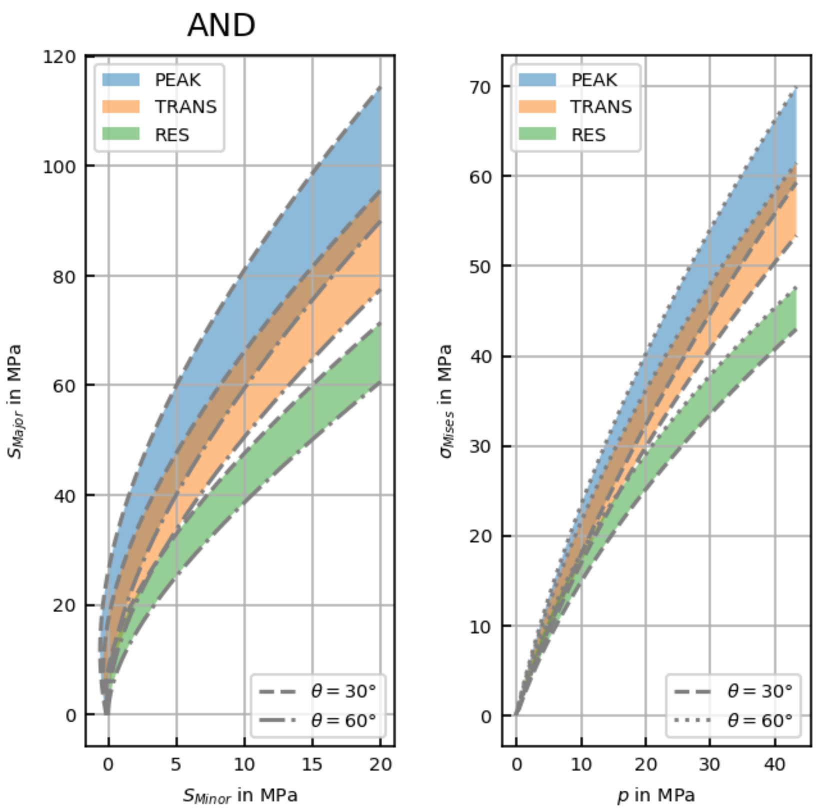
7 APPENDIX B – MATERIAL PROPERTIES

General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2770.00
UCS_i [MPa]	125.00
GSI	65.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	TRANS	RES
ε_{plast} [%]	0.00	0.96	7.32
E [GPa]	26.27	22.53	19.38
ν	0.25	0.25	0.25
s	2.10e-03	5.82e-04	1.00e-05
m_b	1.97	1.34	0.67
a	0.50	0.50	0.50
e	0.60	0.60	0.60
d	0.33	0.22	0.00

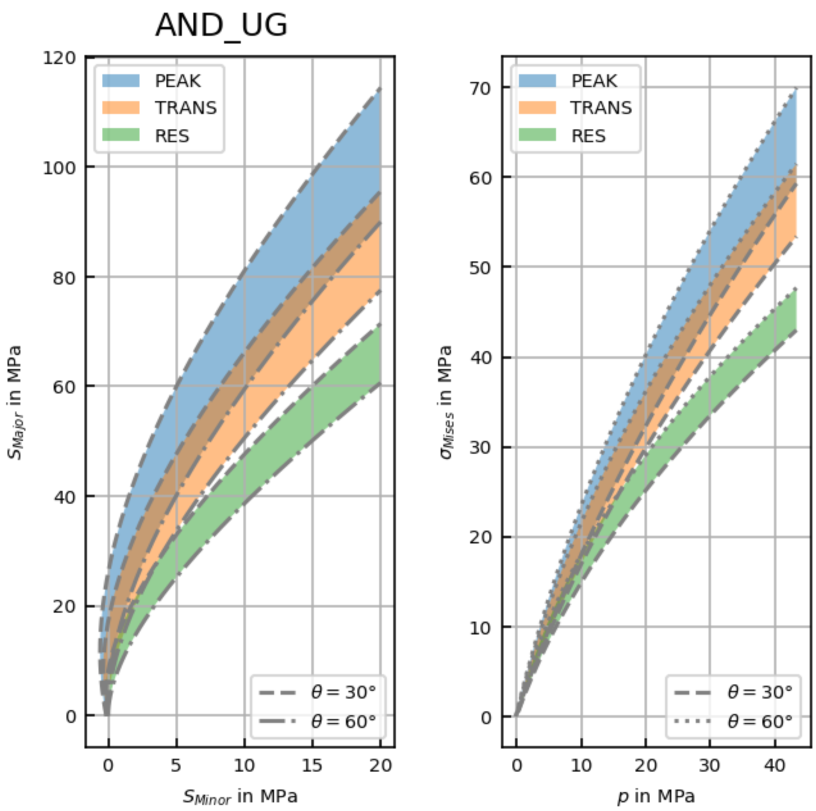


General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2770.00
UCS_i [MPa]	125.00
GSI	65.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	TRANS	RES
ε_{plast} [%]	0.00	0.96	7.32
E [GPa]	26.27	22.53	19.38
ν	0.25	0.25	0.25
s	2.10e-03	5.82e-04	1.00e-05
m_b	1.97	1.34	0.67
a	0.50	0.50	0.50
e	0.60	0.60	0.60
d	0.33	0.22	0.00

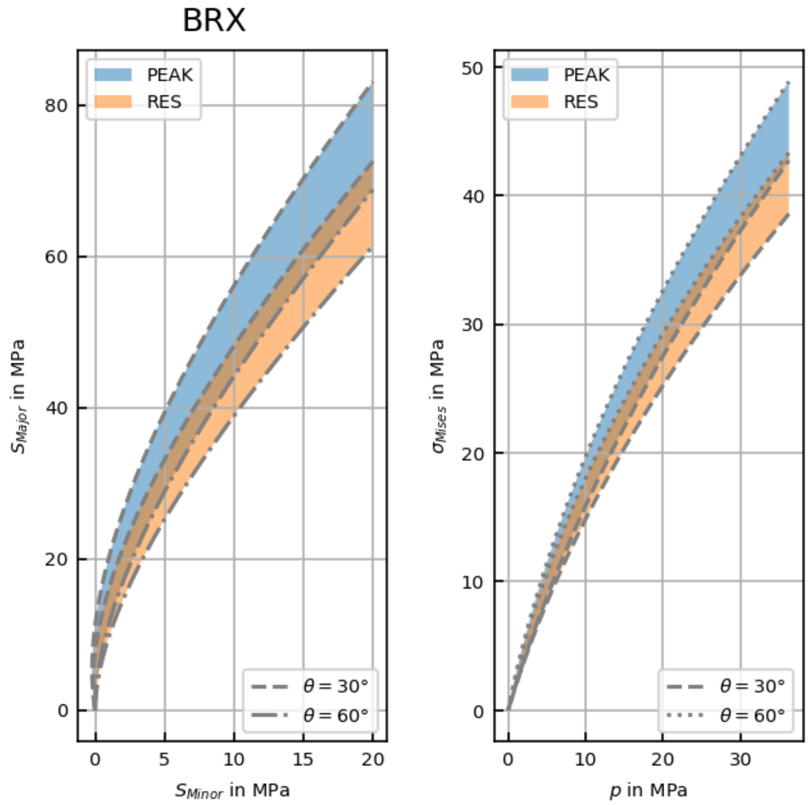


General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2800.00
UCS_i [MPa]	132.00
GSI	40.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	RES
ε_{plast} [%]	0.00	2.77
E [GPa]	20.83	19.80
ν	0.25	0.25
s	1.72e-04	1.00e-05
m_b	0.96	0.70
a	0.52	0.52
e	0.60	0.60
d	0.16	0.00

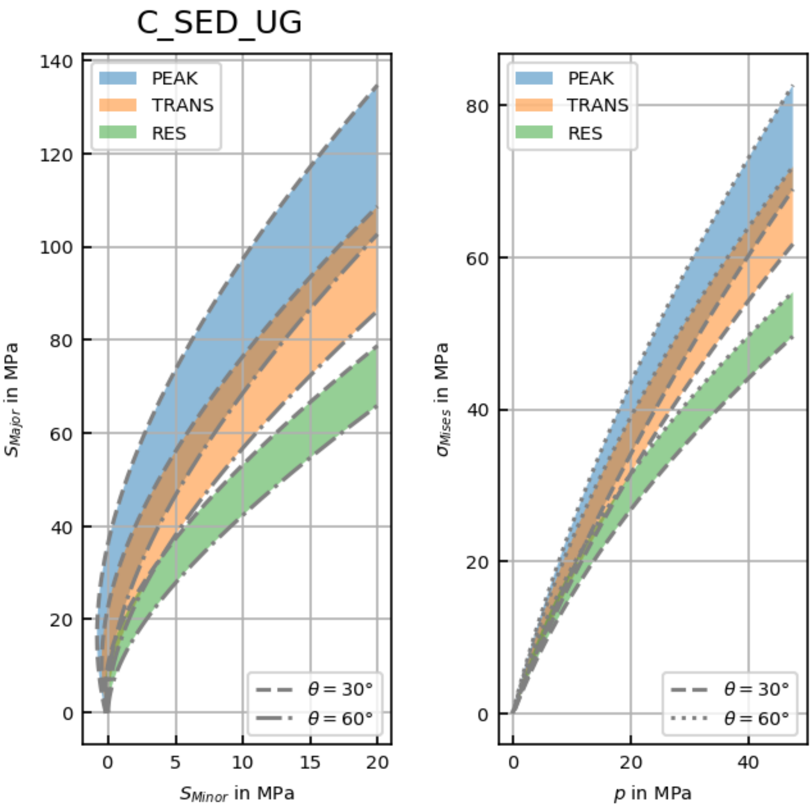


General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2700.00
UCS_i [MPa]	145.00
GSI	68.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	TRANS	RES
ε_{plast} [%]	0.00	0.96	6.46
E [GPa]	28.90	24.15	20.49
ν	0.25	0.25	0.25
s	2.83e-03	6.64e-04	1.00e-05
m_b	2.36	1.53	0.74
a	0.50	0.50	0.50
e	0.60	0.60	0.60
d	0.39	0.25	0.00

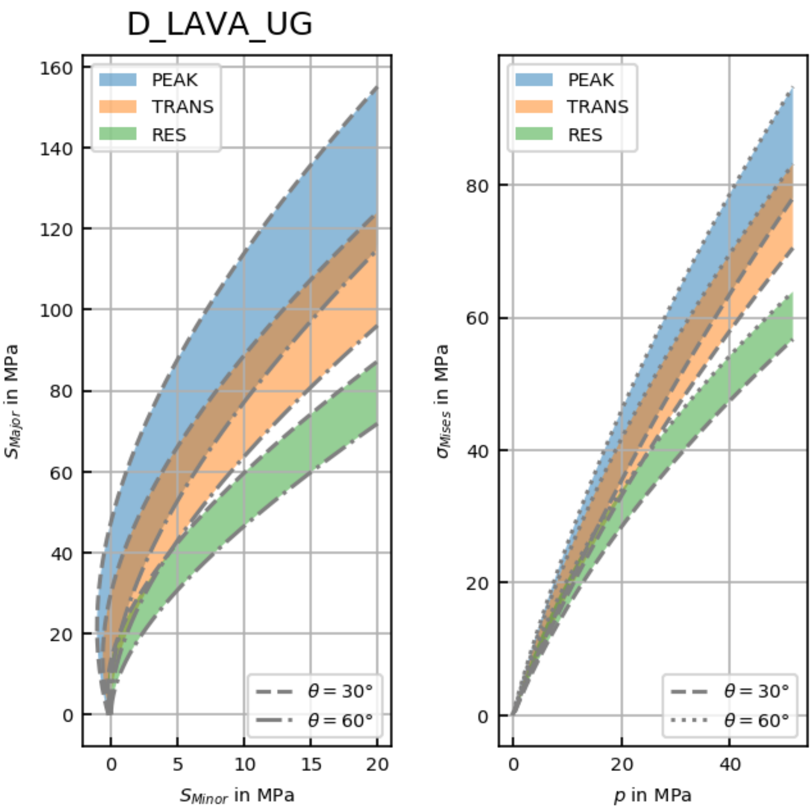


General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2700.00
UCS_i [MPa]	168.00
GSI	69.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	TRANS	RES
ε_{plast} [%]	0.00	0.83	5.59
E [GPa]	30.67	25.61	21.46
ν	0.25	0.25	0.25
s	3.13e-03	7.42e-04	1.00e-05
m_b	2.67	1.74	0.81
a	0.50	0.50	0.50
e	0.60	0.60	0.60
d	0.45	0.29	0.00

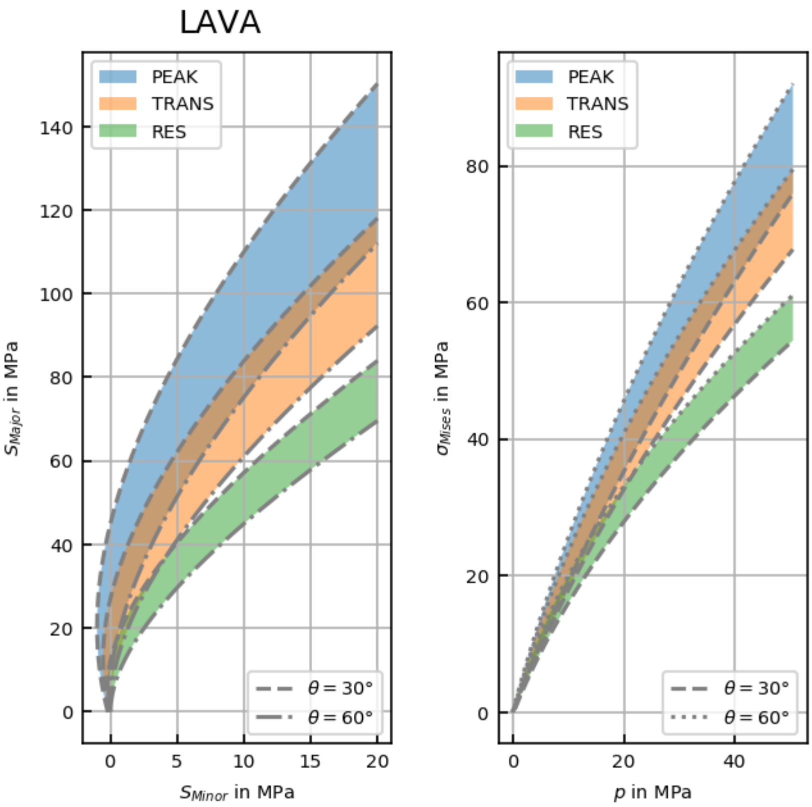


General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2760.00
UCS_i [MPa]	159.00
GSI	70.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	TRANS	RES
ε_{plast} [%]	0.00	0.96	5.99
E [GPa]	30.57	25.08	21.11
ν	0.25	0.25	0.25
s	3.46e-03	7.13e-04	1.00e-05
m_b	2.66	1.65	0.78
a	0.50	0.50	0.50
e	0.60	0.60	0.60
d	0.44	0.28	0.00



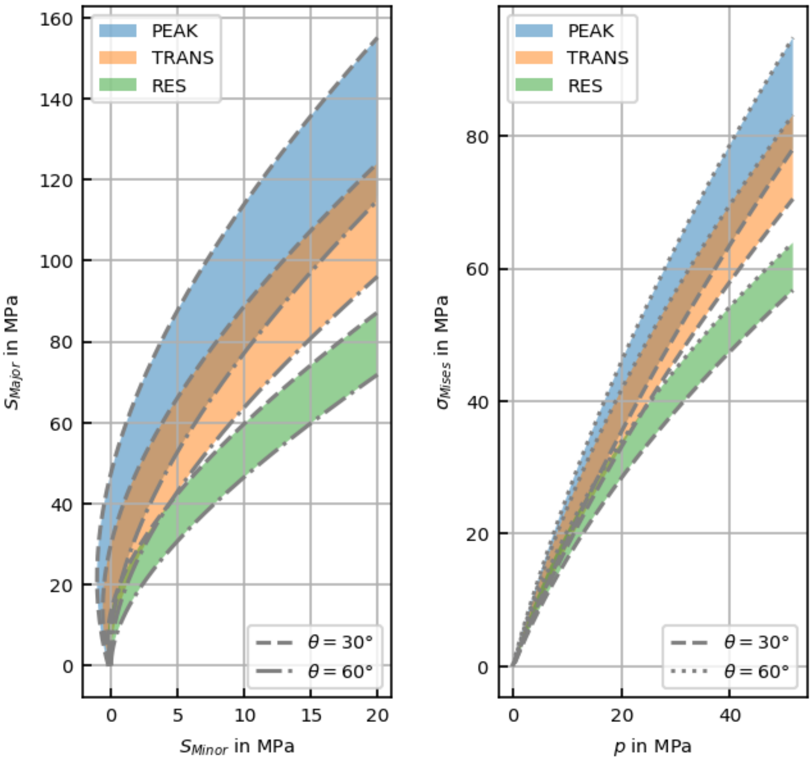
LAVA_UG

General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2700.00
UCS_i [MPa]	168.00
GSI	69.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	TRANS	RES
ε_{plast} [%]	0.00	0.83	5.59
E [GPa]	30.67	25.61	21.46
ν	0.25	0.25	0.25
s	3.13e-03	7.42e-04	1.00e-05
m_b	2.67	1.74	0.81
a	0.50	0.50	0.50
e	0.60	0.60	0.60
d	0.45	0.29	0.00



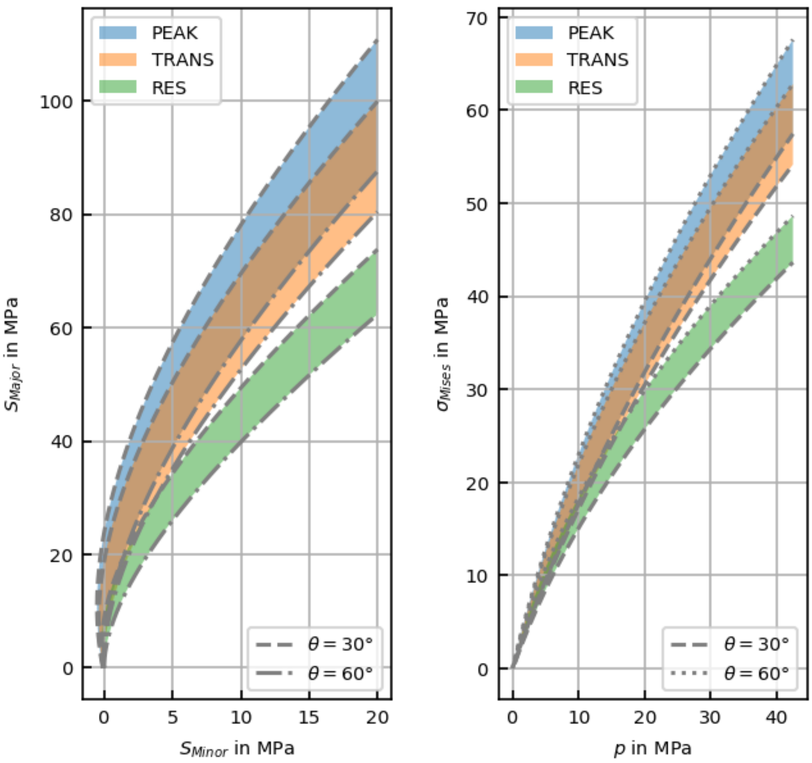
DIOR

General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2800.00
UCS_i [MPa]	132.00
GSI	60.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	TRANS	RES
ε_{plast} [%]	0.00	0.52	6.56
E [GPa]	25.17	23.14	19.80
ν	0.25	0.25	0.25
s	1.27e-03	6.12e-04	1.00e-05
m_b	1.75	1.40	0.70
a	0.51	0.51	0.51
e	0.60	0.60	0.60
d	0.29	0.23	0.00



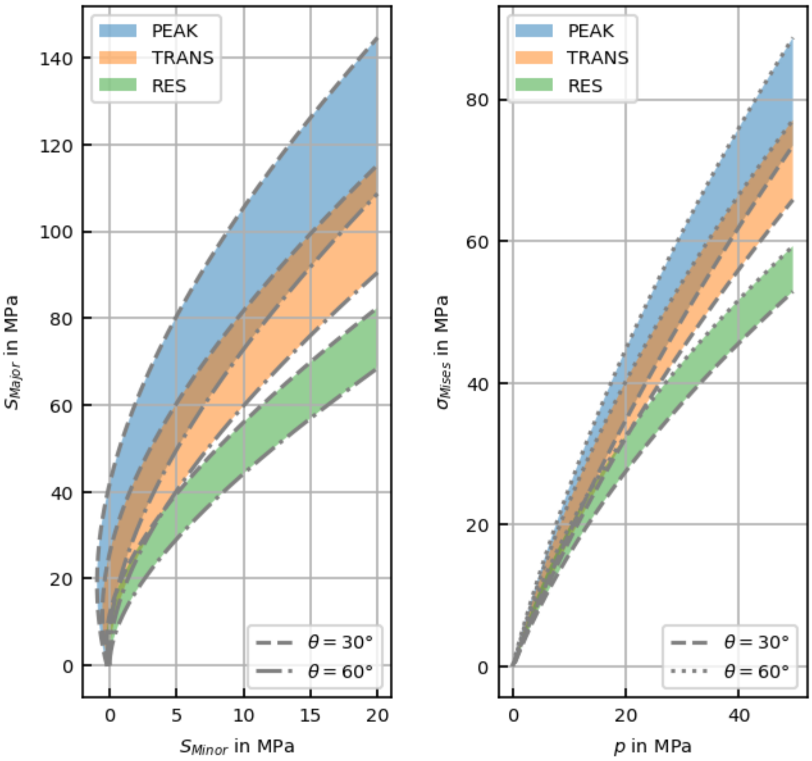
DIOR_UG

General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2900.00
UCS_i [MPa]	155.00
GSI	69.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	TRANS	RES
ε_{plast} [%]	0.00	0.93	6.09
E [GPa]	29.94	24.83	20.95
ν	0.25	0.25	0.25
s	3.13e-03	7.00e-04	1.00e-05
m_b	2.54	1.62	0.77
a	0.50	0.50	0.50
e	0.60	0.60	0.60
d	0.42	0.27	0.00



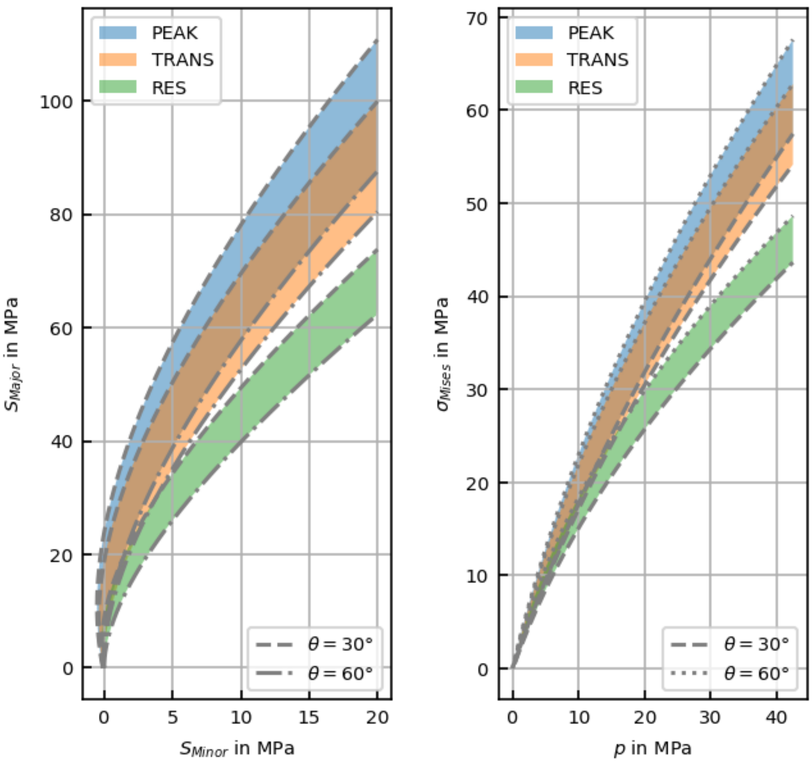
DIPP

General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2770.00
UCS_i [MPa]	132.00
GSI	60.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	TRANS	RES
ε_{plast} [%]	0.00	0.52	6.56
E [GPa]	25.17	23.14	19.80
ν	0.25	0.25	0.25
s	1.27e-03	6.12e-04	1.00e-05
m_b	1.75	1.40	0.70
a	0.51	0.51	0.51
e	0.60	0.60	0.60
d	0.29	0.23	0.00



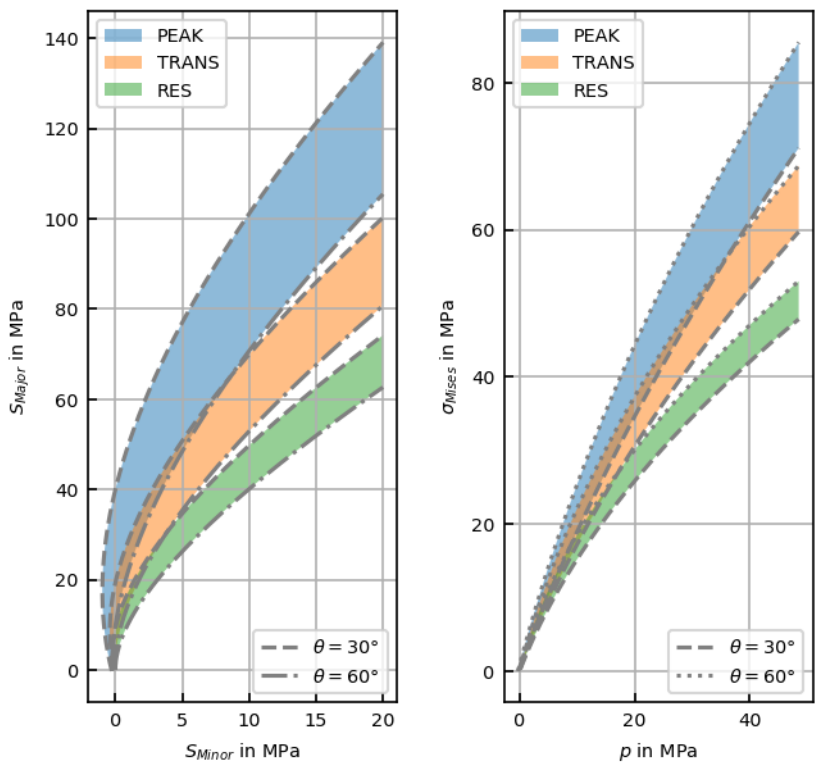
General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2800.00
UCS_i [MPa]	132.00
GSI	75.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	TRANS	RES
ε_{plast} [%]	0.00	1.59	7.63
E [GPa]	30.58	23.14	19.80
ν	0.25	0.25	0.25
s	5.70e-03	6.12e-04	1.00e-05
m_b	2.74	1.40	0.70
a	0.50	0.50	0.50
e	0.60	0.60	0.60
d	0.46	0.23	0.00

DOL_UG



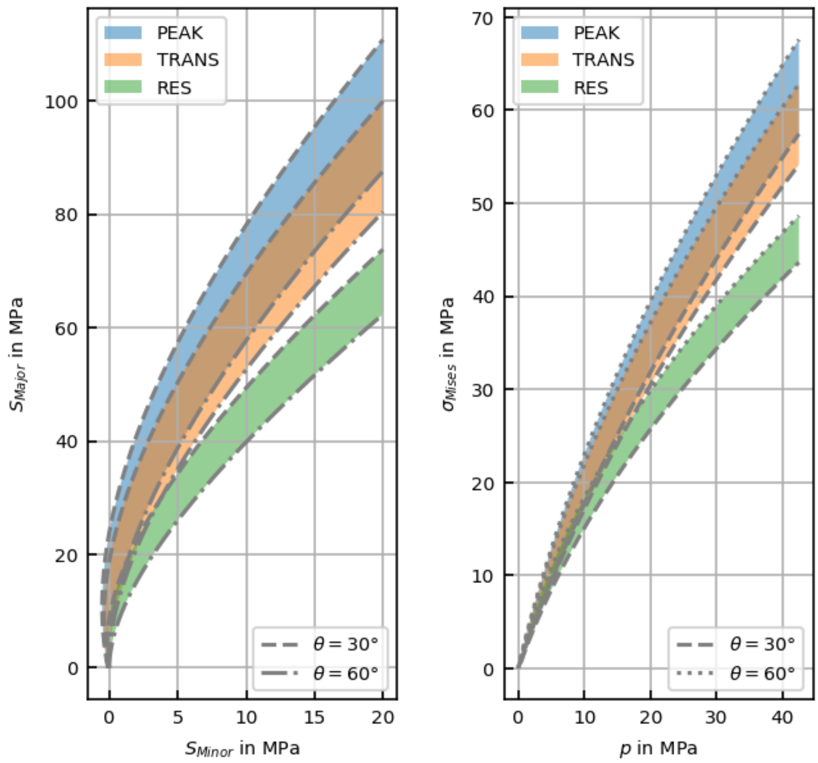
General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2770.00
UCS_i [MPa]	132.00
GSI	60.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	TRANS	RES
ε_{plast} [%]	0.00	0.52	6.56
E [GPa]	25.17	23.14	19.80
ν	0.25	0.25	0.25
s	1.27e-03	6.12e-04	1.00e-05
m_b	1.75	1.40	0.70
a	0.51	0.51	0.51
e	0.60	0.60	0.60
d	0.29	0.23	0.00

DYDI_E42



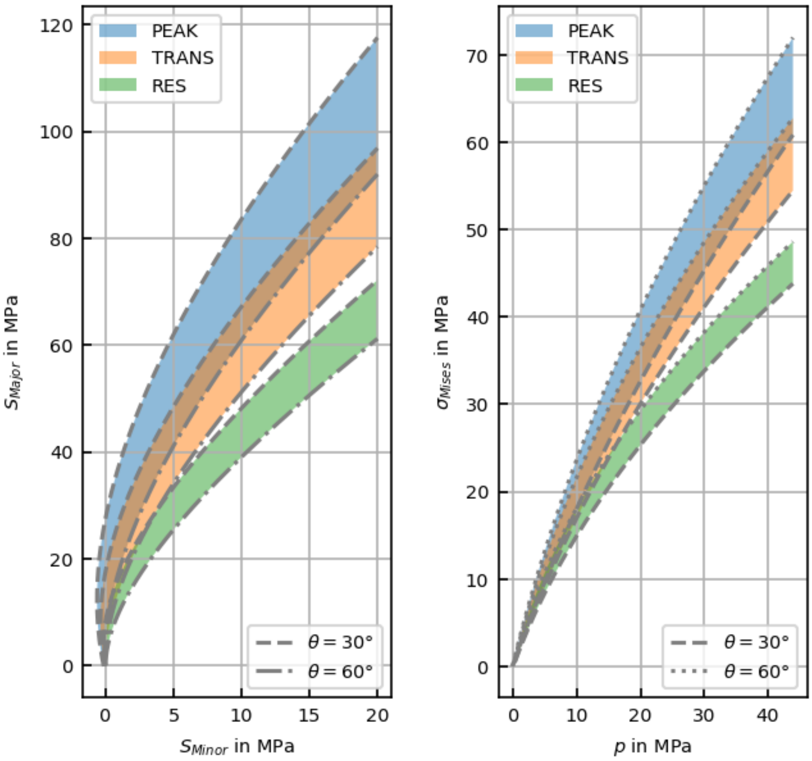
E_VOL

General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2770.00
UCS_i [MPa]	127.00
GSI	66.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	TRANS	RES
ε_{plast} [%]	0.00	1.01	7.27
E [GPa]	26.79	22.71	19.50
ν	0.25	0.25	0.25
s	2.32e-03	5.91e-04	1.00e-05
m_b	2.04	1.36	0.68
a	0.50	0.50	0.50
e	0.60	0.60	0.60
d	0.34	0.23	0.00



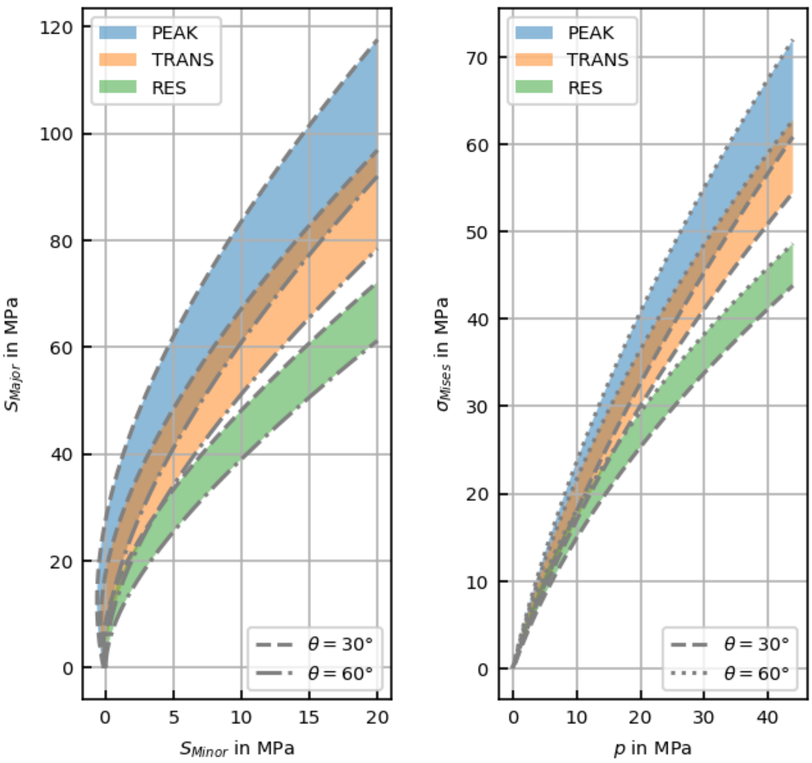
F_SED_UG

General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2800.00
UCS_i [MPa]	127.00
GSI	66.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	TRANS	RES
ε_{plast} [%]	0.00	1.01	7.27
E [GPa]	26.79	22.71	19.50
ν	0.25	0.25	0.25
s	2.32e-03	5.91e-04	1.00e-05
m_b	2.04	1.36	0.68
a	0.50	0.50	0.50
e	0.60	0.60	0.60
d	0.34	0.23	0.00

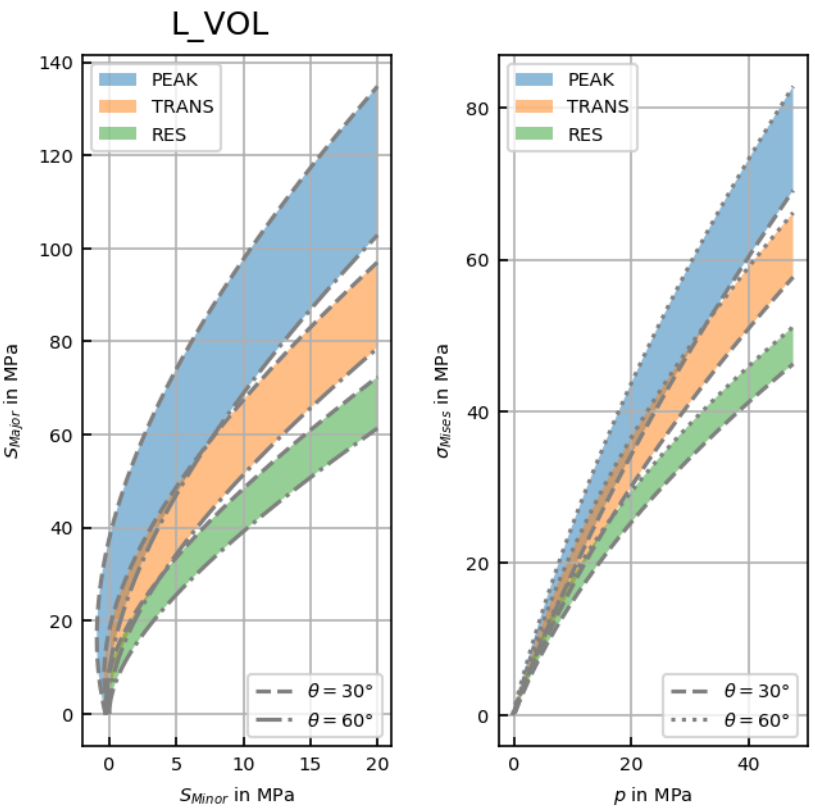


General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2770.00
UCS_i [MPa]	127.00
GSI	75.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	TRANS	RES
ε_{plast} [%]	0.00	1.67	7.94
E [GPa]	30.12	22.71	19.50
ν	0.25	0.25	0.25
s	5.70e-03	5.91e-04	1.00e-05
m_b	2.68	1.36	0.68
a	0.50	0.50	0.50
e	0.60	0.60	0.60
d	0.45	0.23	0.00

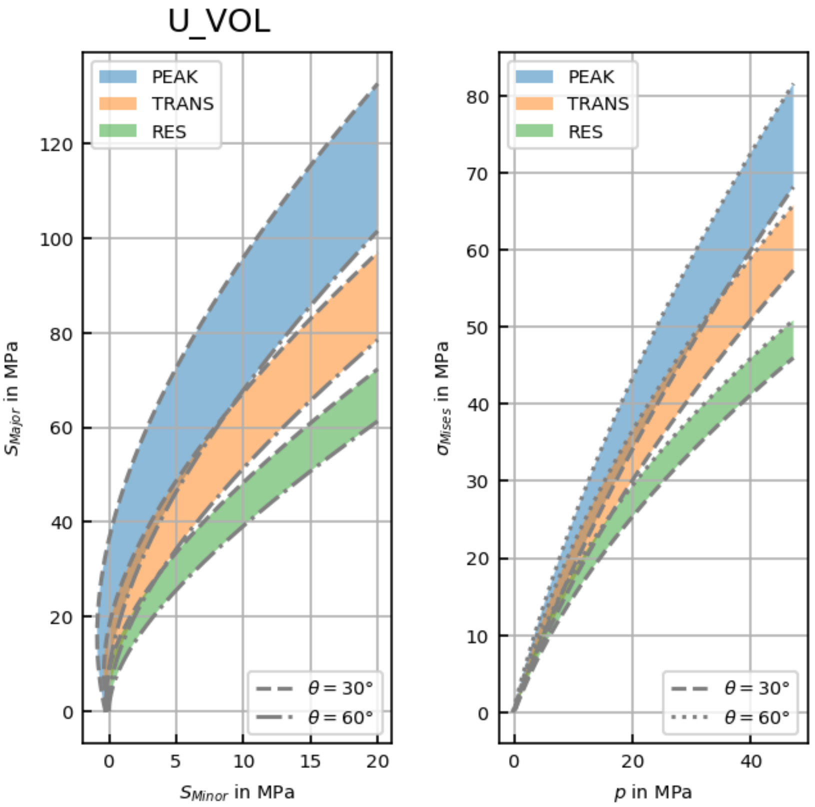


General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2700.00
UCS_i [MPa]	127.00
GSI	74.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	TRANS	RES
ε_{plast} [%]	0.00	1.60	7.86
E [GPa]	29.74	22.71	19.50
ν	0.25	0.25	0.25
s	5.16e-03	5.91e-04	1.00e-05
m_b	2.60	1.36	0.68
a	0.50	0.50	0.50
e	0.60	0.60	0.60
d	0.43	0.23	0.00



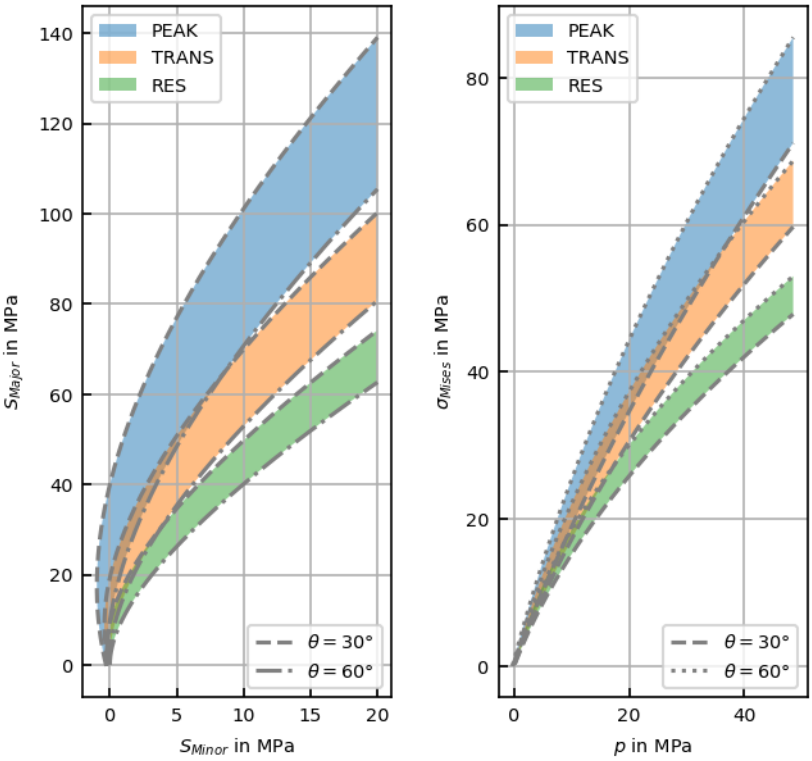
General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2800.00
UCS_i [MPa]	132.00
GSI	75.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	TRANS	RES
ε_{plast} [%]	0.00	1.59	7.63
E [GPa]	30.58	23.14	19.80
ν	0.25	0.25	0.25
s	5.70e-03	6.12e-04	1.00e-05
m_b	2.74	1.40	0.70
a	0.50	0.50	0.50
e	0.60	0.60	0.60
d	0.46	0.23	0.00

MONZ



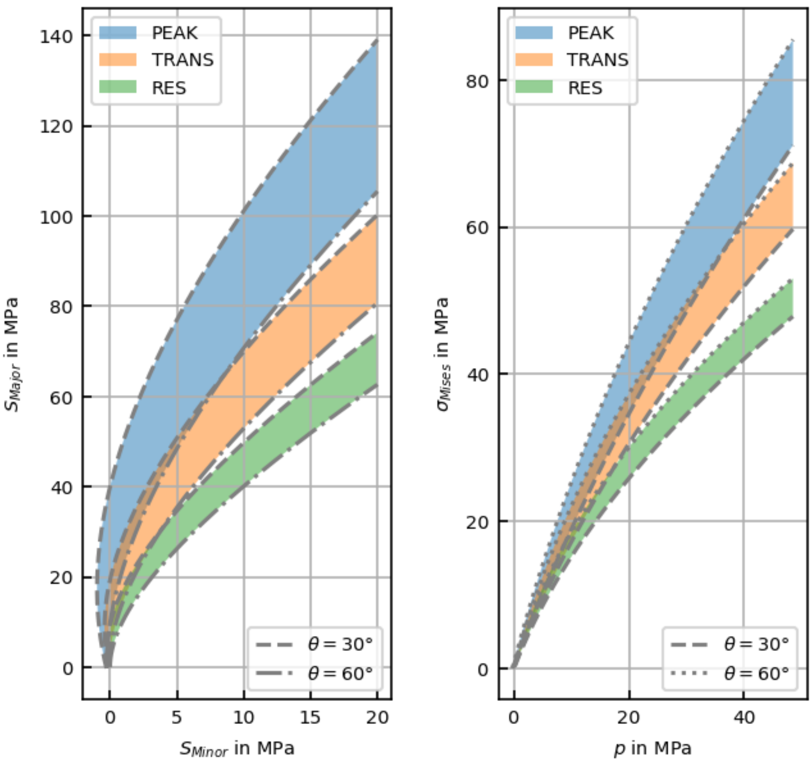
General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2800.00
UCS_i [MPa]	132.00
GSI	75.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	TRANS	RES
ε_{plast} [%]	0.00	1.59	7.63
E [GPa]	30.58	23.14	19.80
ν	0.25	0.25	0.25
s	5.70e-03	6.12e-04	1.00e-05
m_b	2.74	1.40	0.70
a	0.50	0.50	0.50
e	0.60	0.60	0.60
d	0.46	0.23	0.00

MONZ_UG

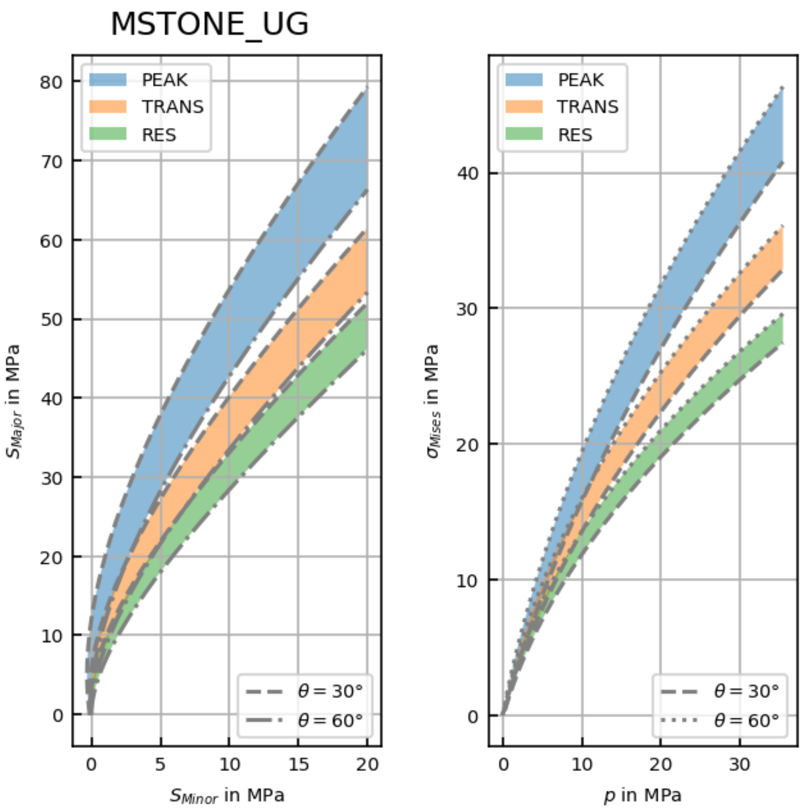


General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2700.00
UCS_i [MPa]	70.00
GSI	67.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	TRANS	RES
ε_{plast} [%]	0.00	2.34	12.41
E [GPa]	19.99	15.70	14.36
ν	0.25	0.25	0.25
s	2.56e-03	2.96e-04	1.00e-05
m_b	1.54	0.80	0.50
a	0.50	0.50	0.50
e	0.60	0.60	0.60
d	0.26	0.13	0.00

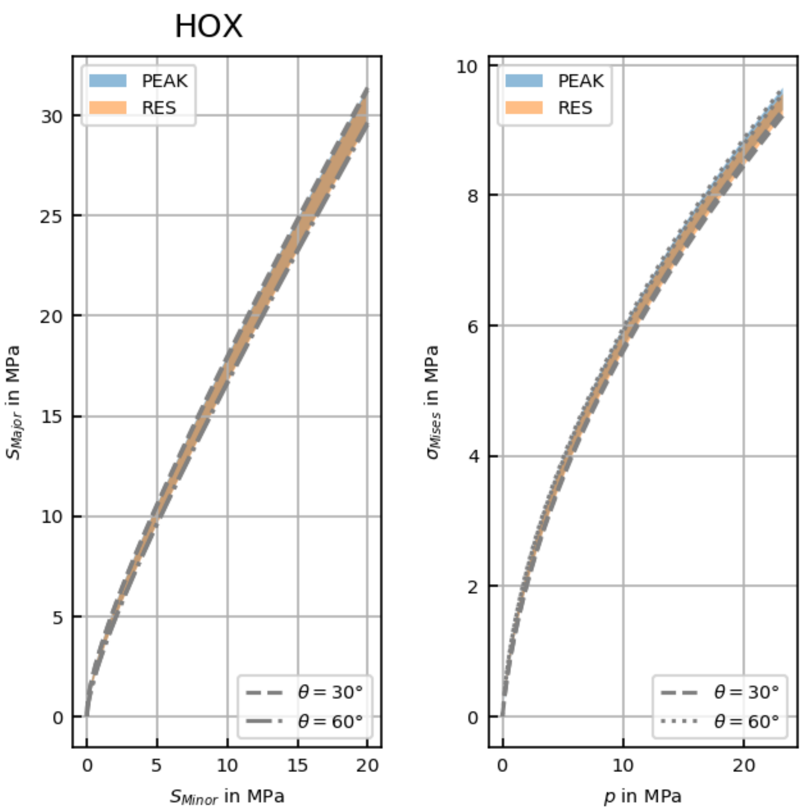


General Properties

Type	Computator V14.3 PLR
ρ [kg/m³]	2000.00
UCS_i [MPa]	15.00
GSI	30.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	RES
ε_{plast} [%]	0.00	30.84
E [GPa]	4.25	4.24
ν	0.25	0.25
s	6.34e-05	1.00e-05
m_b	0.32	0.32
a	0.53	0.53
e	0.60	0.60
d	0.05	0.00

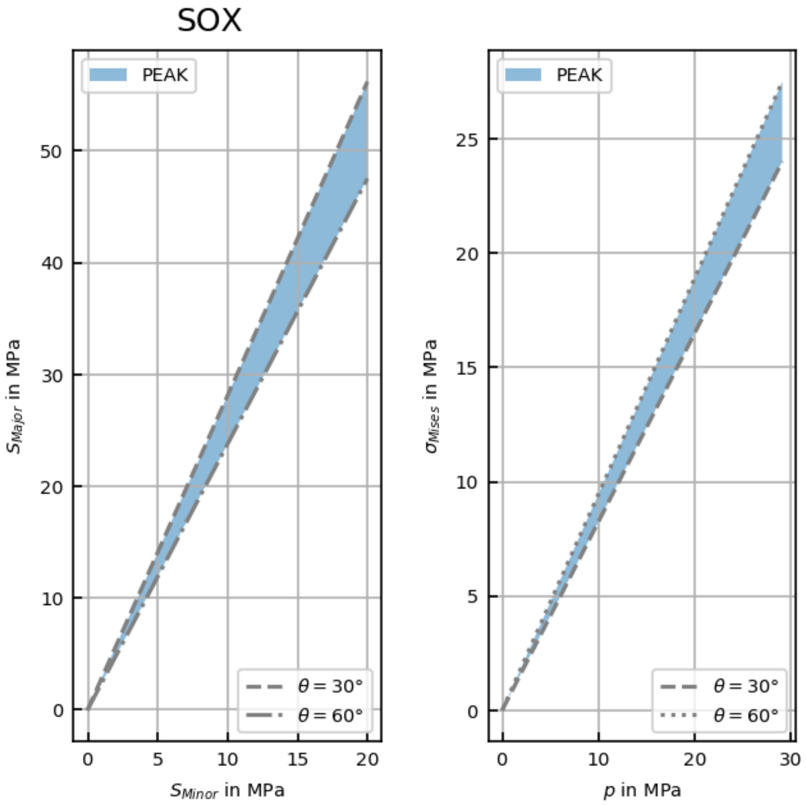


General Properties

Type	MOHR-COULOMB PLR
ρ [kg/m³]	1950.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK
ε_{plast} [%]	0.00
E [GPa]	0.40
ν	0.30
c [kPa]	28.00
ϕ [°]	24.00
d	0.25

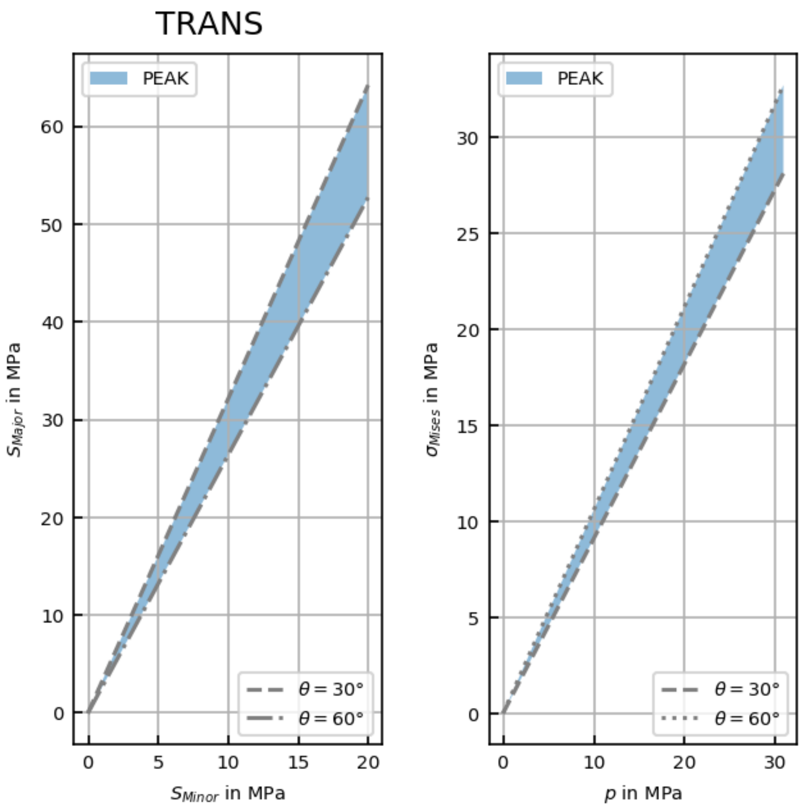


General Properties

Type	MOHR-COULOMB PLR
ρ [kg/m³]	1850.00
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK
ε_{plast} [%]	0.00
E [GPa]	0.40
ν	0.30
c [kPa]	26.00
ϕ [°]	26.70
d	0.25

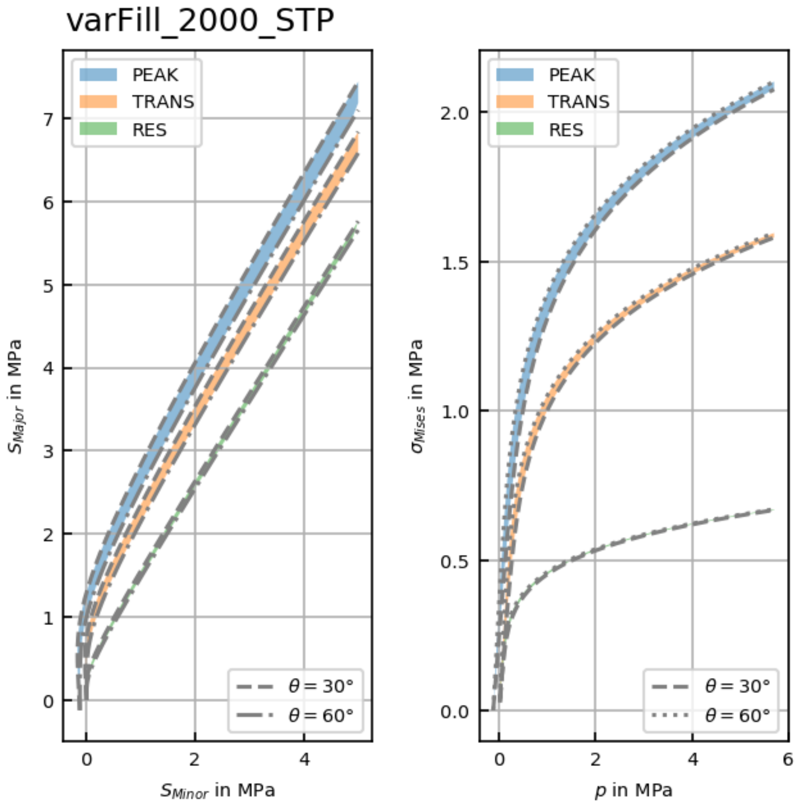


General Properties

Type	arbitrary PLR
ρ [kg/m³]	------
UCS_i [MPa]	1.00
GSI	------
n_{aniso}	1.00
s_{aniso}	1.00

LR2 Parameters

	PEAK	TRANS	RES
ε_{plast} [%]	0.00	1.50	3.50
E [GPa]	0.20	0.20	0.20
ν	0.20	0.20	0.20
s	1.00	1.00e-03	1.00e-05
m_b	8.00	2.00	0.03
a	0.20	0.20	0.20
e	0.60	0.60	0.60
d	0.25	0.25	0.25



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