

**The risks and potential influence of undermined
areas in the Okiep Copper District with a specific
focus on the Bruinhoek populated area of
Concordia**

by
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DECLARATION

I herewith declare that the entirety of the work contained in this thesis is my own original work and that I am the sole author thereof and that I have not previously in its entirety or in part submitted this work at any University for a degree.

Signature

HJ Luttig

Date:

ABSTRACT

The collapse and/or subsidence of overburden located above an undermined area is inevitably an impending consequence of underground mining activities across the world. One of the legacies left behind of approximately 150 years of mining in the Okiep Copper District is the presence of huge collapsed structures locally known as glory holes. Although some are remote, several holes are located close to populated areas and pose a hazard to the local inhabitants. The Bruinhoek populated area which forms part of the town of Concordia is undermined by the Wheal Julia East Mine. The potential formation of a glory hole within the undermined area poses a risk to local inhabitants of Bruinhoek.

The aim of the study is to determine the likelihood of glory hole formation at Bruinhoek, as well as the inherent risk if this should happen. This aim was achieved by means of a data collection process that involved three reference mines which has glory holes, as well as the Wheal Julia East Mine. In order to determine the likelihood of the formation of a glory hole at the Bruinhoek populated area the main reasons causing the formation of the glory holes at the reference mines were determined first. This was done by the reviewing of previous investigations, studying survey and geological maps gathered from the O'okiep Copper Company (OCC), and field investigations that involved drone aerial surveys, site walkovers and the inspection of the sidewalls of the glory holes. The Wheal Julia East Mine was similarly studied and investigated. After the main reasons for glory hole formation at the reference mines were determined they were used as a proxy for the Wheal Julia East Mine.

It was found that for a glory hole to form an underground cavity must have been created first. The main factor that contributed to the initiation of the failure of overburden at the reference mines was the presence of persistent discontinuities. Investigations proved that the Wheal Julia East Mine has created an underground cavity as well as that persistent discontinuities could occur following the investigations at the reference mines. Therefore it was determined that a glory hole at the Bruimhoek populated area could be expected in the future.

The stope dimensions in plan view for the Wheal Julia East Mine where determined through the creation of overlay maps where a surface image of the Bruinhoek populated area overlies the geological maps of the Wheal Julia East Mine. It was found that the stope lies directly below two plots of the Bruinhoek populated area. The field investigations of the reference mines indicated that instabilities for example self-caving of the sidewalls and rim collapse are common after the formation of a glory hole. This means that in the case of the formation of a glory hole at Bruinhoek it is almost certain that it will self-cave and experience rim collapse, gradually

becoming greater in size. The current situation at Bruinhoek has the potential to develop in a high risk situation endangering people's lives.

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CHAPTER 1: INTRODUCTION

1. 1 Background

The Okiep Copper District is located in the Namaqualand region and lies in the north western corner of the Northern Cape, forming part of the South Africa–Namibia border. The town of Springbok is the main town of the Namaqualand region, with the N7 national road running through the town. Springbok is located 110 km south of the South Africa–Namibia border and 550 km north of Cape Town, as may be seen in Figure 1.1.

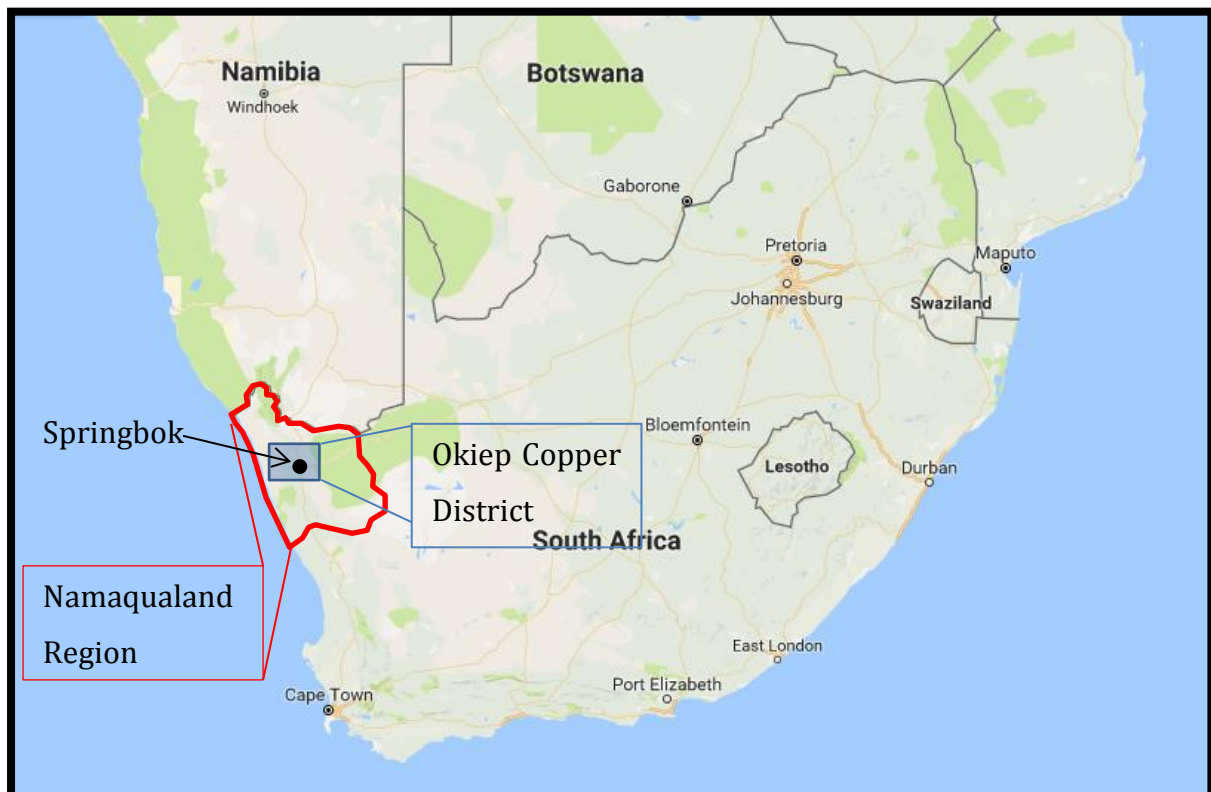


Figure 1. 1: Map of Namaqualand and the Okiep Copper District relative to the rest of South Africa (modified from Google Maps, (2017a)).

In the 17th century the present town of Springbok was a small farm called Springbokfontein owned by the Cloete family. Springbokfontein developed into a small town soon after Philips & King bought a portion of Springbokfontein from the Cloete family, in order to commence with the first commercial mining activities in South Africa (Cairncross, 2004).

Numerous towns exist around the main town of Namaqualand. Most of them were founded as the many rich copper deposits were discovered throughout the region. Okiep, Nababeep, Carolusberg and Concordia are prime examples of such towns. All the copper deposits in the region are located in an area known as the Okiep Copper District. After the mining activities finally ceased in the early 2000s the once flourishing mining towns are now declining. Today Namaqualand is better known as a region

focused on livestock farming and tourism, as beautiful flowers give a new face to the semi-desert region during spring, after plentiful winter rains.

NOTE: For the sake of clarity the reader must note that throughout this research 'O'okiep' will be spelled with an apostrophe as the 'O'okiep' in O'okiep Copper Company and the 'O'okiep' in the titles of the mine maps are spelled with an apostrophe. The only exception is when there will be referred to the 'Okiep Copper District' that will be spelled without an apostrophe as it is often done by many researchers for example Kisters (1993).

1.2 Motivation for research

Copper was mined in the Okiep Copper District for about 150 years (Davenport, 2014) and during the initial stages of mining small areas were focused on for the extraction of copper. As the extent of mining operations became larger and the mining technology improved, numerous shafts were sunk, large stopes were mined and large cavities were created underground. At some of the mines in the Okiep Copper District enormous collapse structures formed as a result of the cavity underneath the surface. These collapse structures are referred to locally, and in this report, as 'glory holes'. The glory holes typically have dimensions of $\pm 2\ 800\ \text{m}^2$ to $\pm 8\ 700\ \text{m}^2$ and can be hundreds of metres deep.

Some of the glory holes are located several kilometres from rural areas; for example those at Rietberg Mine, Hoits Mine and Spektakel Mine. Most of the old mine buildings, offices and structures of these mines have been demolished. The old buildings, structures and/or offices that are still standing today are abandoned, but are located relatively close to the glory holes. An example is the Hoits Mine, located 10 km east of Springbok, next to the N14 national road.

However, some of the glory holes are located in close proximity to populated areas. An example of such a glory hole is that located in the town of Nababeep, 12 km north-west of Springbok. The glory hole has a length of 250 m and is 100 m wide. Should a hole of such dimensions form inside a populated area, it could have catastrophic implications.

The Wheal Julia Mine was in operation from 1955–1959. It is located 12 km north east of the town of Springbok. The mine comprised four parts:

- Wheal Julia Central
- Wheal Julia West
- Wheal Julia North
- Wheal Julia East

A portion of the Bruinhoek Erven, a small portion of the town of Concordia, has been undermined by Wheal Julia East. This means that a cavity exists underground and is located right beneath an area with

regular human activity. If overburden between the underground cavity and surface collapses, as in the case of numerous mines in the Okiep Copper District, the result would potentially be catastrophic.

1.3 Research purpose and objectives

While there are many examples of glory holes in the Okiep Copper District, resulting from underground cavities, the primary purpose of this thesis was to determine the likelihood of glory hole formation at the Bruinhoek area and the intensity of the risk to the people and the surroundings should a glory hole form. It should be remembered that research does not aim to predict when a glory hole might form.

Further objectives include the following:

- Identifying factors contributing to glory hole formation by investigating already established glory holes in the Okiep Copper District and, finally, applying this knowledge to Bruinhoek.
- Developing a glory hole hazard-rating system to determine the intensity of the hazard that each glory hole in the Okiep Copper District creates and compare it to Bruinhoek, should a glory hole form in the future.
- Closely examining the Mine Health and Safety Act and Regulations with regard to the Bruinhoek area.
- Recommending safety, protection and awareness procedures for the undermined area of Bruinhoek.

1.4 Limitations of research

The study excluded modelling to predict when collapse might occur, as this objective did not form part of the objectives of this study.

Information about geological structures and features could be gathered only from older unpublished reports, maps and site visits by the author, because of the difficulty in acquiring fresh geological information, as access to the mines was no longer possible. The walls and surroundings of the glory holes gave insight into the structure of the underground geology. Glory holes could be observed only from a distance, as it was unsafe to move too close to a glory hole.

Due to the lack of information available, rock mass rating (RMR) systems were not used as part of the study, but this is, however, included in the literature study.

1.5 Properties of the Okiep Copper District

The Okiep Copper District is located in the Nama Khoi Local Municipality (see Figure 1.2) and according to (Statistics South Africa, 2017) it then had a total population of 47 041.

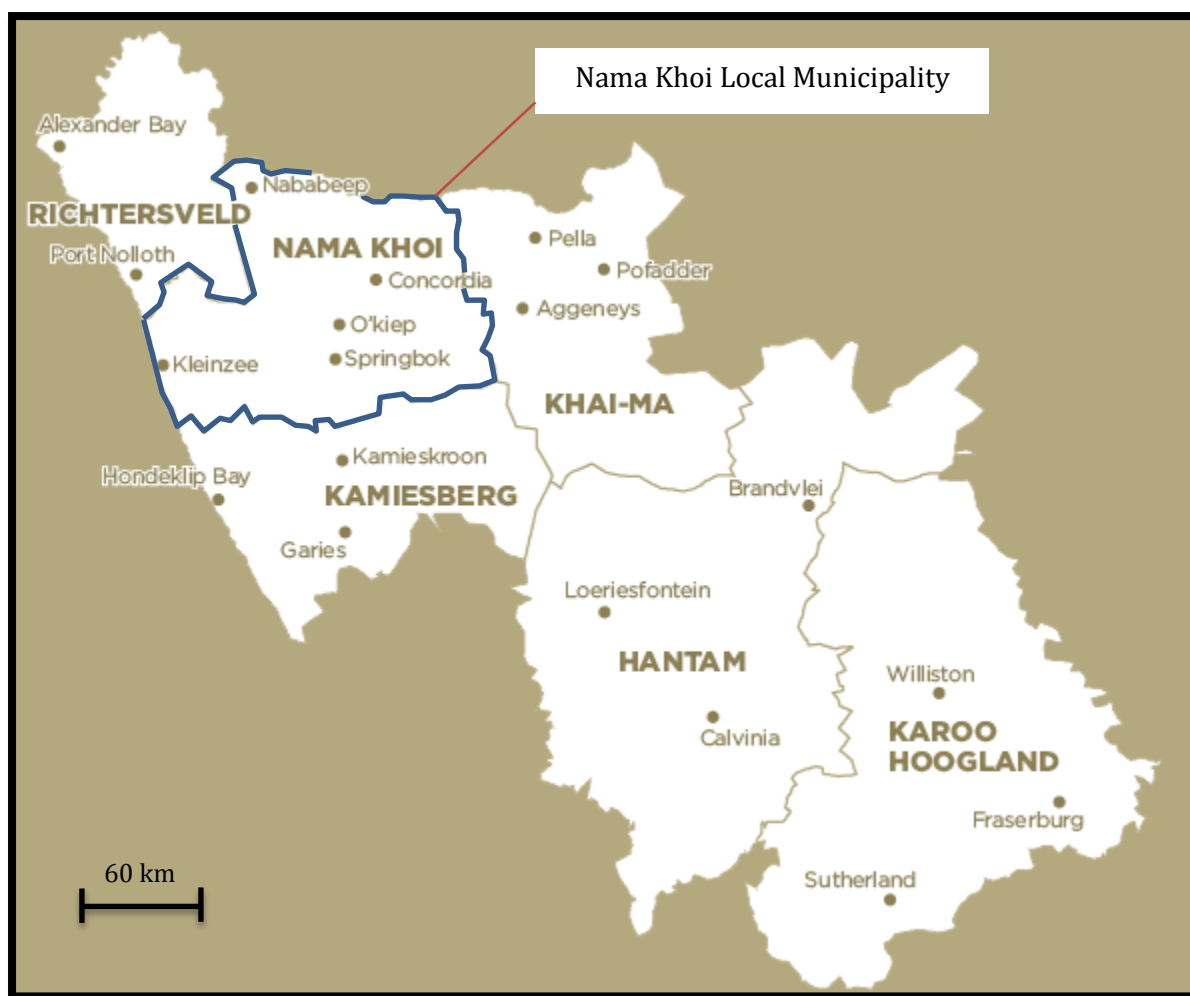


Figure 1. 2: Nama Khoi Local Municipality (Municipalities of South Africa, 2017).

The Okiep Copper District comprises largely granitic and granite gneiss outcrops separated by sand filled valleys. The Okiep Copper District is divided into two regions by the great escarpment that forms a natural boundary. East of the great escarpment the interior plateau can be found, with an average height of approximately 900 m above mean sea level but which can attain heights of 1 300 m above mean sea level. West of the great escarpment the lower lying coastal belt can be found, which is generally 200 m above mean sea level and is situated along the Buffels River. The Okiep Copper District is drained by a number of sandy streams and rivulets, all non-perennial, flowing southwards and westwards into the Modderfontein and Schaap Rivers, cutting eventually through the escarpment and feeding into the major Buffels River, flowing into the Atlantic Ocean at Kleinsee.

Namaqualand forms part of the larger Succulent Karoo biome and can be divided into seven biophysical regions (Desmet, 2007). The Succulent Karoo biome is part of the Cape Floral Kingdom and has recently been added to this unique area of plant diversity. Namaqualand constitutes one quarter of the Succulent Karoo biome and contains 3 500 species of plants, of which 25% are endemic to this region. The Namaqualand region is semi-arid, with a predominantly winter rainfall. In his thesis Smuts (2015) mentions that the average rainfall of the town of Springbok, recorded between the years 1990 and 2004, was 228.2 mm per annum with the highest annual precipitation occurring between

May and August, as seen in Figure 1.3. The average monthly minimum and maximum temperature data recorded over the same time period as the precipitation data is displayed in Figure 1.4. According to Figure 1.4 the warmest month of the year is February and the coldest month is August, with temperatures of 30 °C and 7.8 °C respectively.

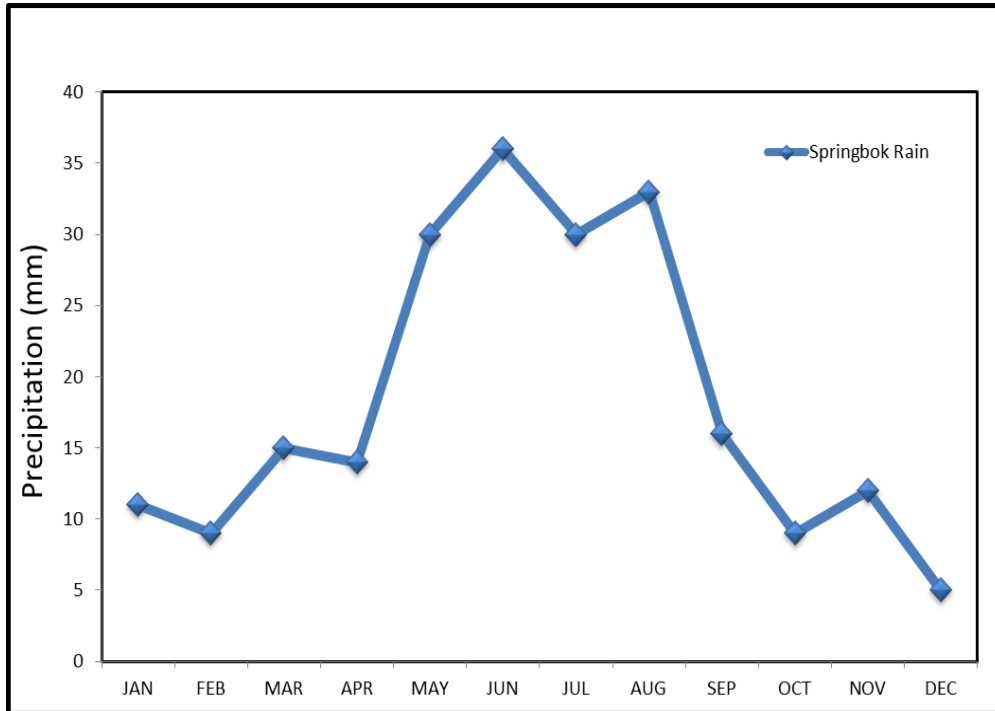


Figure 1. 4: Average monthly precipitation for Springbok (Smuts, 2015).

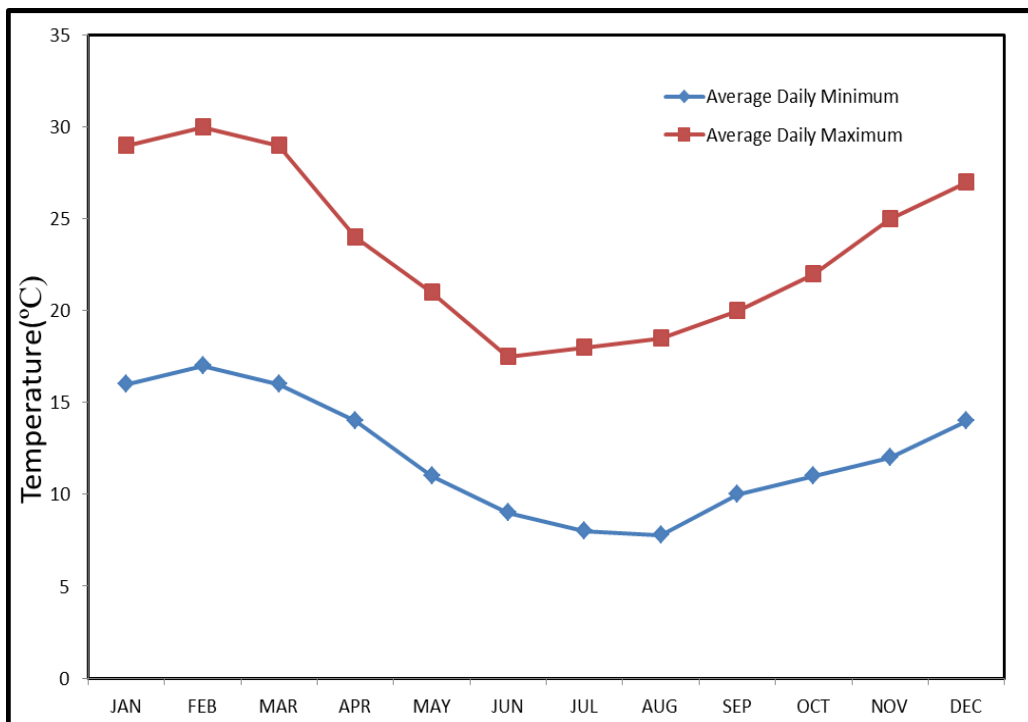


Figure 1. 3: Average daily minimum and maximum temperatures in Springbok (Smuts, 2015).

The Okiep Copper District is located mainly in the Kamiesberg bioregion, which is defined by metamorphosed granite gneiss, forming rolling dome-shaped hills with flattish in-filled valleys in between. A great diversity of soil types can be found in Namaqualand, while the area covered by the Okiep Copper District has shallow, undifferentiated and free-draining red and yellow, variably grained, sandy to loamy soils, which are derived from in situ weathering of the underlying parent material

1.6 Report layout

The thesis comprises five chapters, which include the following:

Chapter 1 provides background information to the research area and explains the motivation for the research. The research purpose and objectives are stated, as well as the limitations. Certain properties of the study area, for example climate and geomorphology, are also discussed.

Chapter 2 begins with a summary of the history of copper mining in the Okiep Copper District followed by a description of the geology of the region, providing the reader with information on, for example, lithostratigraphy and geological structure. Both soft rock mining methods and hard rock mining methods are described, to give the reader an overview of the methods used in both environments followed by a description of the major mining method used in the Okiep Copper District. An overview of land subsidence and collapse in different environments including abandoned mine openings, will be provided. The literature review also includes a discussion on classification systems for rock mass and how it is implemented and used. Chapter 2 concludes with two sections which are Hazard Identification and Risk Assessment (HIRA) and a discussion on the importance and applicability of the Mine Health and Safety Act, 1996 (Act No. 29 of 1996).

Chapter 3 entails the Methodology. The locations of the reference mines and main study area, the Bruinhoek populated area, are identified. Most importantly the methods used to obtain data and results are explained as well as how the results and field data were analysed and used to contribute to certain discussions enabling the author to draw conclusions.

The results obtained in this research can be seen and are discussed in Chapter 4. The main source of results is the survey- and geological maps found at the old mine offices of the O'okiep Copper Company (OCC). Other results include reports on the areas of research, stereonets composed of geological maps data, historical aerial images and glory hole risk ratings.

Chapter 5 is the analysis of captured data and the discussion thereof. Data from drone aerial surveys and field investigations are discussed and analysed along with the results from Chapter 4. New maps including overlay maps and a safety map are also compiled by the author that contributes to discussions focusing on reference mines and the prevailing conditions at the Bruinhoek populated area.

The final chapter, Chapter 6, consists of the conclusions arrived at and recommendations made from the analysis and discussions of the results and field data, as well as risks related to the prevailing conditions at the Bruinhoek populated area.

CHAPTER 2: LITERATURE STUDY

2.1 History of mining in Namaqualand

In the year of 1652 Jan van Riebeeck arrived in South Africa and founded his settlement in the Cape of Good Hope to act as a stopover on the way to the eastern countries where spices were obtained by the VOC for trade in Europe. It was hoped that the settlement would provide water, fresh vegetables and fruit, as well as meat (Davenport, 2014). With regard to valuable mineral deposits, the VOC was interested in the possibility of finding gold deposits in the region of the Cape, with their attention focused mainly on the areas of Devil's Peak, Lion's Head and Table Mountain (Davenport, 2014). The search for gold in the Cape did not, however, deliver results. In 1660 van Riebeeck dispatched a mineral prospecting expedition into the interior of the new land. This expedition took place under the command of Pieter van Meerhof. During this expedition contact was made with the Namaqua people, who wore copper ornaments, resulting in copper being the first metal that drew the attention of the Dutch settlers (Wilson, 1998).

In 1679 Simon van der Stel was appointed as the new Commander at the Cape and he soon found out about van Meerhof's encounter with the Namaqua people. According to Wilson (1998), in 1685 van der Stel led an expedition from the Cape Town to Namaqualand to locate the source of the Namaqua copper. Two months after the expedition started, van der Stel reached the copper mountains and set up camp at the foot of a ridge that is today known as Koperberg (Davenport, 2014). Van der Stel's journey was not followed by the exploitation of copper. Only in 1761 did Dr Karel Rykvoet set off to Namaqualand to examine the geology (Davenport, 2014). He underestimated the mineral wealth of the area and concluded that copper mining would not be profitable. The physical conditions proved to be too harsh and unforgiving, while transport was also a major issue. Only after the expedition of discovery led by Captain Sir James Alexander in the early 1800's a few companies were created to start mining activities in Namaqualand, but these achieved very little success.

The first commercial mining in South Africa took place in 1852, when the mining company Philips & King exported 11 tonnes of what kind of ore (Wilson, 1998). This historical event took place after an increased demand for copper in Europe. Following the demand, Philip and King bought a copper-rich portion of the farm Springbokfontein from the Cloete family in 1850 (Cairncross, 2004). From the beginning of 1854 until mid-1855, a time known as the copper boom, 40 mining companies were formed, heading to Namaqualand with the goal of exploiting copper. The boom lasted for 18 months, with only Philips and King and the Namaqua Mining Company remaining after mid-1855 (Davenport, 2014). In 1862 Philips and King was taken over by the Cape Copper Company. This company was famous for completing a railway line in 1876 that stretched between Okiep and Port Nolloth, located on the west coast (Beale, 1985).

The O'okiep Copper Company (OCC) was formed in 1937 and in 1939 it acquired all the stock and property of both the Namaqua Copper Company and Cape Copper Company. The OCC erected a smelter at Okiep and a treatment plant at Nababeep. The company closed the Okiep-Port Nolloth railway line in 1942, as ore was being transported by road to the railhead at Bitterfontein, from where it was transported by railway to the Cape Town harbour (Beale, 1985). The OCC experienced a period of modest, but continued, growth from 1938–1975, after which there was a decline in the price of copper. The main shareholder in the OCC, at that stage Newmont Mining, sold its stake in the OCC to Gold Fields (Davenport, 2014). As a result of a decline in mining activities only the Nigramoep and Carolusberg mines operated during the early 1990s (Cairncross, 2004). Gold Fields sold its stake in the OCC to Metorex in 1998, which is the same year that the Carolusberg mine closed down. Nigramoep mine produced copper until 2004. This marked the end of the copper mining era in the Okiep Copper District that had lasted for 150 years.

2.2 The Geology of the Okiep Copper District

In the Okiep Copper District, copper is located in one specific geological unit known as the Koperberg Suite. According to Clifford and Barton (2012), the entire Okiep Copper District contains 1 700 small ore bodies that cover up to 0.7% of the entire outcrop area. The economic copper found in the Okiep Copper District is one of only two examples in the world where economic copper mineralization is found in rocks of the Anorthosite–Charnockite family.

2.2.1 Regional setting

According to Johnson *et al.*, (2006), 1 000 Ma–1 200 Ma (Mega annum) ago the Namaqua Orogeny occurred, that formed and metamorphosed the rocks of the Namaqua-Natal Province. Outcrops of these rocks can be found in KwaZulu-Natal and the Northern Cape. Respectively, they can be referred to as the Natal and Namaqua Sectors of the Namaqua-Natal Province. In the Namaqua-Natal Province there are a number of areas that have a common lithostratigraphy and structural fabrics, which is bounded by shear zones (Johnson *et al.*, 2006). These terranes formed during the Namaqua Orogeny. There are three main lithostratigraphic components:

- Modified 2000 Ma Kheisian rocks, which originally were supracrustal rocks. After modification, metasediments, and quartz-feldspar-biotite granulite, and gneiss with intercalations of various metasedimentary rocks, were formed.
- Juvenile plutonic and supracrustal rocks that formed as a result of subduction, ocean spreading and rifting phases 1 200 Ma–1 600 Ma ago. These rocks assembled during intense deformation and metamorphism during collision events.
- Large granitoids formed 1 000 Ma–1 200 Ma ago. Along with the formation of the granitoids small, scattered mafic and ultramafic bodies intruded.

The Namaqua orogenic belt is located in the north western corner of South Africa and in the south western corner of Namibia covering an area of 70 000 km² (Clifford and Barton, 2012). The Okiep Copper District is located in the high grade metamorphic (granulite facies) granite-gneiss domain. It includes mostly quartz-feldspar-biotite, augen gneiss and quartz-microcline granite intrusions containing left over rafts of metasediments (Gadd-Claxton, 1981). The Namaqua orogenic belt in north western South Africa is made up of a series of terranes that are grouped in the following sub provinces seen in Figure 2.1 (Duchesne *et al.*, 2007):

- Gordonia sub province
- Richtersveld sub province
- Bushmanland sub province

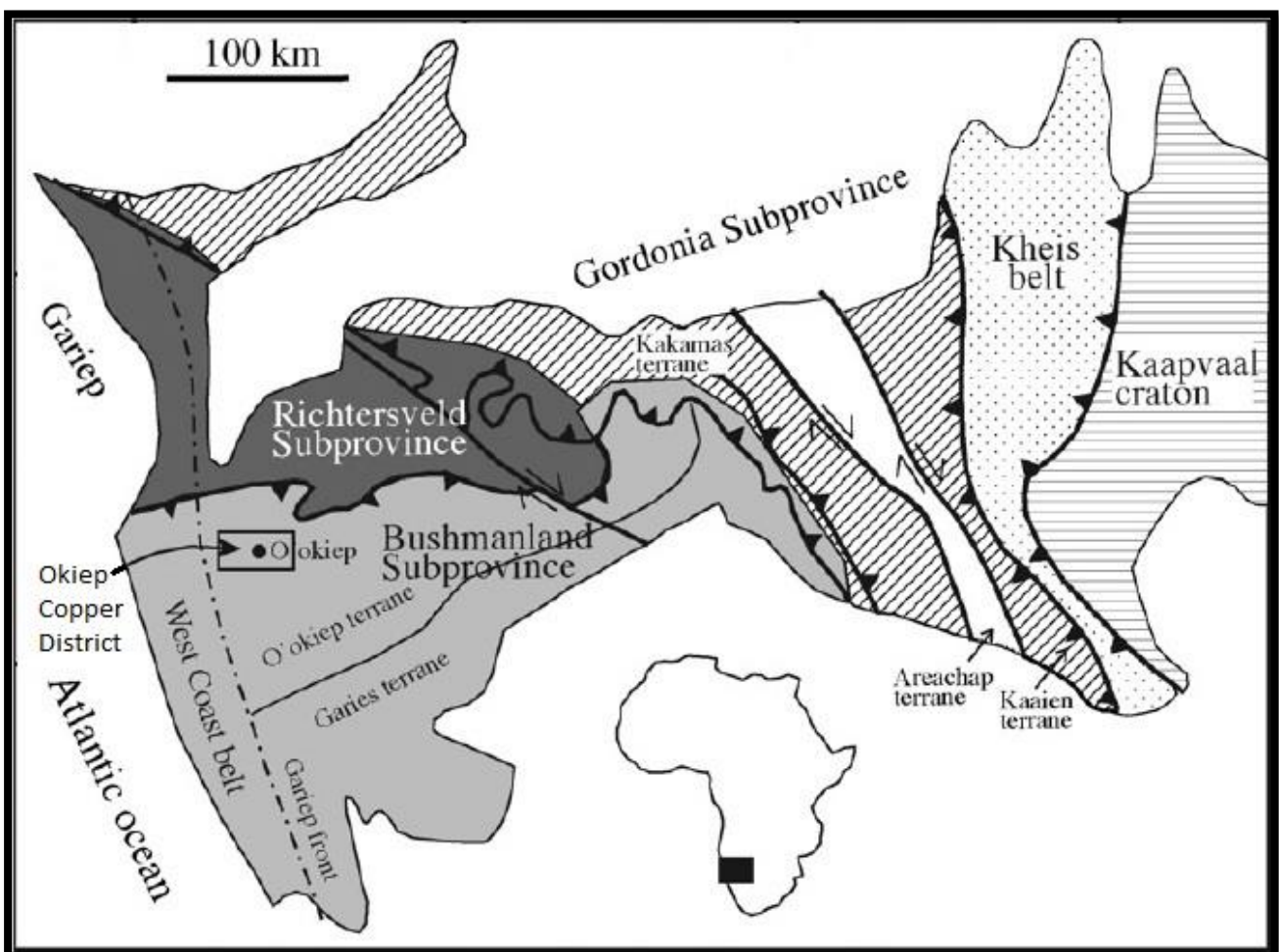


Figure 2. 1: The three sub-provinces of the Namaqua orogenic belt located in south western South Africa (adapted from Duchesne *et al.*, 2007).

The Okiep Copper District is located in the Bushmanland sub-province, made up of 2000 Ma granitic gneisses (basement complex known as the Gladkop Suite), supracrustal rocks of granulite grade with an age of 1600–1200 Ma and granitoids with ages of 1200–1000 Ma that generally have a granite to

charnockitic composition (Johnson *et al.*, 2006). The Bushmaland sub-province is made out of two structural terranes (Duchesne *et al.*, 2007):

- Okiep Terrane
- Garies Terrane

The Okiep Terrane can be distinguished from the Garies Terrane by the fact that the Koperberg Suite only occurs in the Okiep Terrane.

2.2.2 Local geological setting

The rocks of the Okiep Copper District is made up of horizontal regional fragments of metavolcanic and metasedimentary lithologies consisting of quartzite, finely-foliated, biotite-bearing quartzite-feldspar gneiss, and quartz-biotite-garnet-sillimanite schist. Adding to the metasedimentary rocks is a thick Precambrian succession of intrusive biotite-bearing quartzo-feldspathic gneisses, leptites and gneissic granites (Gadd-Claxton, 1981). The region was exposed to granulite facies (high temperature and moderate pressure) metamorphism and experienced several periods of deformations. The north east trending open synforms and doubly plunging antiforms controls the distribution of the rocks in the Okiep Copper District. The metamorphosed sedimentary rocks and intrusive granite-gneiss rocks are intruded by basic bodies known as the Koperberg Suite. These basic bodies are small, irregular dyke-like, plug-like and sill-like cupriferous igneous rocks of intermediate to basic composition (Kisters *et al.*, 1994). The economic copper mineralization is restricted to the Koperberg Suite and varies in composition from anorthosite, to diorite to norite and hypersthene. The metamorphosed granitic basement and rocks of the basic suite are unconformably overlain by sediments that represent the late-Precambrian Nama Group in the western parts of the Okiep Copper District (Kisters *et al.*, 1994). In Table 2.1 the lithostatigraphy of the Okiep Copper District can be seen.

The oldest rocks that will be covered in this study are the meta-volcanosedimentary rocks of the Okiep Group. The Okiep Group is subdivided into the Khurisberg and Lammerhoek Subgroups. According to Gadd-Claxton (1981), the Khurisberg Subgroup is named after the Khurisberg that displays a thick succession of quartz-biotite-garnet-sillimanite schist and quartzite. The Khurisberg Subgroup includes the prominent Springbok Quartzite that is well exposed around the Springbok dome (Lombaard, 1986). The type locality of the Lammerhoek Subgroup is a farm five kilometres south of the town of Springbok. The rocks characterising the subgroup are finely foliated quartz-feldspar-biotite gneisses, grey of colour (Gadd-Claxton, 1981). The Lammerhoek Subgroup is believed to represent a volcano-sedimentary assemblage.

The meta-volcano-sedimentary rocks of the Okiep Group were intruded by granite occurring in the north of the Okiep Copper District. The first major phase of intrusion consists of two successive pulses of granitic emplacement. This phase is known as the Gladkop Suite and consists of the

Brandewynsbank Granite gneiss and the Noenoemaasberg Granite gneiss. The Gladkop Suite occurs mainly in the north western and northern part of the Okiep Copper District and is referred to as grey and pink gneiss. The first pulse of granitic emplacement is the Brandewynsbank Granite gneiss. The rock type is the oldest intrusive rock in the Okiep Copper District and is medium grained and grey of colour (Lombaard, 1986). The second pulse of granitic emplacement is the Noenoemaasberg Granite gneiss confined to the northern part of the district showing reddish tinges when weathered.

The second major phase of intrusion consists of two granitic intrusions grouped as the Little Namaqualand Suite occurring extensively throughout the Okiep Copper District. The two granites forming part of this suite are Nababeep Granite-gneiss and Modderfontein Granite-gneiss. Kisters (1993) describe the Nababeep Granite-gneiss as well-foliated quartz-microcline-biotite gneiss with variable plagioclase, garnet, magnetite and hypersthene. According to Lombaard (1986), the Nababeep Granite-gneiss has a thickness of 600 m in the area of the town of Nababeep and generally consists of sheet like bodies (Johnson *et al.*, 2006). The type locality for the Modderfontein Granite-gneiss is on the farm called Modderfontein. The Modderfontein Granite-gneiss is extensively developed in the Springbok dome and is also well exposed in the east, south and western parts of the Okiep Copper District. The Modderfontein Granite-gneiss mainly has the same composition as the Nababeep Granite-gneiss (Gadd-Claxton, 1981).

The third major phase of intrusion consists of three granitic intrusions grouped as the Spektakel Suite and consists of the Concordia Granite, Rietberg Granite and Kweekfontein Granite. The Concordia and Rietberg Granites are regarded as the two important subgroups of this suite. The type locality for the Concordia Granite is the small town of Concordia. Concordia is located about 8 km northeast of the town of Okiep. The Concordia Granite weathers to form rounded boulders described as woosack weathering (Lombaard, 1986). According to Clifford and Barton (2012), the Concordia Granite is a 1500 m thick sheet like intrusion developed as a quartz-microcline granite with variable amounts of plagioclase, garnet and usually less than three present biotite. The Rietberg Granite gets its name from the Rietberg Mountain. The Rietberg Granite intruded with a sheet like fashion into the rocks of the Okiep Group, Gladkop Suite and Klein Namaqualand Suite as it contains inclusions of each one of them. The Rietberg Granite is has a porphyritic texture and consist out of alkali phenocrysts set in a fine groundmass of quartz, feldspar and minor biotite (Kisters, 1993). The final Subgroup of the Spektakel Suite is the fine grained poorly foliated quartz-microcline bearing Kweekfontein Granite, occurring as irregular dykes in the southern parts of the Okiep Copper District, crosscutting all of the previously mentioned rock types (Johnson *et al.*, 2006).

The most important Suite with regards to copper ore in the Okiep Copper District is the Koperberg Suite. It is made up of intrusive bodies of basic rock that forms the host rock for the copper deposits of the Okiep Copper District. According to Clifford *et al.*, (2004), the Koperberg Suite is part of the

anorthosite-charnockite family and has a restricted compositional range of jotunite, andesine anorthosite, biotite diorite, and leuconorite-norite-hypersthene.

Finally, the sediments of the Nama Group unconformably overlie the Precambrian rocks in the western part of the Okiep Copper District (Lombaard, 1986). The Nama Group represents weakly deformed grits, quartzites, conglomerates, shales and limestones.

Table 2. 1: The lithostatigraphy of the Okiep Copper District (after Lombaard, 1986).

	Rock type	Group/Suite	Description	
NAMAQUALAND METAMORPHIC COMPLEX		Nama Group	Conglomerate, grit, quartzite, shale, limestone	
	unconformity			
	Intrusive rocks		Koperberg Suite	Anorthosite, diorite, norite and hypersthene
			Spektakel Suite	Kweekfontein Granite
				Rietberg Granite
				Concordia Granite
			Klein Namaqualand Suite	Modderfontein Granite-gneiss
				Nababeep Granite-gneiss
		Gladkop Suite	Noenoemaasberg Granite	
			Brandewynsbank Granite-gneiss	
	Supracrustal rocks		Okiep Group	Lammerhoek Sub-group
Khurisberg Sub-group				

2.2.3 Structure

The clearest regional structure in the Okiep Copper District are the open folds defined by the Ratelpoort Synforms and the Springbok Antiform/Dome (Clifford *et al.*, 2004). These folds are generally east-west trending. Gadd-Claxton (1981) describes these folds as having undulating dips of the major rock units and the dominant foliation in the greater part of the Okiep Copper District being less than 30°. However, the earliest deformational event (D_1 -deformation) can be recognized by sharp-hinged intrafolial folds (F_1) (Clifford *et al.*, 2004). Intrafolial folds in older orthogneisses and metasediments north of the Okiep Copper District are recorded in rafts and xenoliths of the Kurisberg Subgroup and Gladkop Suite (Kisters *et al.*, 1994). The second deformational event (D_2 -deformation), also known as the Namaqua-event, reflects large-to-small scale isoclinal folding (F_2) and is specifically well developed in the Springbok Quartzite. Refolding of the F_2 folds and deformation of the subhorizontal lithologies and fabrics of the gneisses of the Okiep Copper District resulted in the formation of the east-west trending open folds (F_3) mentioned previously (Duchesne *et al.*, 2007). On a smaller scale a younger structural feature (F_4) known as steep structures, resulted from a third deformational event (D_3) (Kisters *et al.*, 1994). The (F_4) structural feature can be described as mega kink folds or steep structures where subhorizontal gneissosity has been rotated subvertical altitudes and occurs in parts of the granitic gneisses and metasediments. Ductile deformation of country rock gneisses was also a result of the (D_3) deformational event. Primal regional metamorphism (M_2 in this case granulite facies) accompanied (D_2) deformation and was also coeval with the intrusion of the Koperberg Suite and ductile deformation of the country-rock gneiss. The entire Okiep Copper District is also cross cut by shear faults and intense breccia faults (Lombaard, 1986). Most of the shear faults (D_4) comply to a conjugate set with north western and north eastern strikes They are subvertical mylonitic shear zones recording the last form of ductile deformation in the Okiep Copper District and have maximum widths of 25m (Kisters, 1993). The shear faults are older than the Koperberg material, but they are younger than the sediments of the Nama Group as they are not present in the Nama sediments. The breccia faults (D_5) are older than the shear faults and affect the entire stratigraphic succession. They are subvertical and have a north-south trending strike. They can range from mm-scale, and are then often expressed as cataclasites or pseudotachylite veins, to several tens of meters and are then developed on a regional scale (Kisters, 1993). Normal displacements of the Nama Group along breccia faults indicate late tectonic activity of breccia faults at shallow crustal levels (Kisters, 1993). According to Gadd-Claxton (1981) these breccia faults are important aquifers. According to historic geological maps for example the 70 m level of the Hoits Mine (O'okiep Copper Company, 1975-1993d) moderate to intensely jointed zones often coexist with breccia faults. Note should be taken that access to the mines are currently not possible and therefore very limited information is available to describe faults and joints with regards to discontinuity features for example surface smoothness of joints and faults, joint separation, alteration and weathering of fault and joint planes, joint filling, etc. Table 2.2 displays the sequence of deformational events relative to intrusive events.

Table 2. 2: Historical deformational and intrusive events of the Okie Copper District (after Lombaard, 1986).

Deformational Events	Metamorphic Events	Intrusive Events
D(5) N-S breccia faults		Koperberg Suite Spektakel Suite Klein Namaqualand Suite Gladkop Suite
D(4) NW-SE and NE-SW shear zones		
D(3) F(3) open E-W folding and steep structures	M(2) primal regional metamorphism (granulite facies)	
D(2) F(2) isoclinal folds		
D(1) F(1) intrafolial folding	M(1) overprinted by M(2)	

2.2.4 Field relationships of the copper bearing Koperberg Suite

According to Duchesne *et al.* (2007) bodies of the Koperberg Suite occur in field as elongate bodies or dykes trending in an east-north-easterly direction that can have a length of up to 1 km and width up to 100 m. Individual bodies range from a few square meters to 0.25 km² (Conradie and Schoch, 1986). The suite is revealed as a swarm of dyke-like bodies that have intruded the metasedimentary and gneissic granites (Duchesne *et al.*, 2007). Conradie and Schoch (1986) explain that the mafic intrusions often display pinch and swell structures along the strike of the intrusions and that the contact between the mafic Koperberg Suite and the country rocks is sharp. There is a correlation between spatial associations of steep structures and mafic bodies (Conradie and Schoch, 1986). Steep structures are a geological phenomenon wherein subhorizontal gneissosity has been rotated to subvertical dips (Kisters *et al.*, 1994). In the Okiep Copper District steep structures occur in narrow, easterly trending antiformal zones. According to Gadd-Claxton (1981), mafic material generally occurs in the core of the steep structure, or close to the northern flank.

2.2.5 Weathering and soil formation

Weathering plays a major part in the formation of the soil cover and secondary minerals found in a specific climatic region. The Okiep Copper District is composed mainly of granites and gneisses and, for the purpose of this research, the weathering and soil formation processes are explained below.

According to Brink (1979), the climatic regime of the present and of the relatively recent past plays a fundamental role in the weathering of rocks and the development of the soil profile at any particular point below the surface of the earth. According to Mitchell and Soga (2005), parent material and climate affect the rate at which weathering can proceed, while topography primarily determines the rate of erosion and the depth of soil accumulation. The climate determines the quantity of water present, temperature and vegetation cover. The broad influences of climate are the following:

- At stable temperatures, the rate of weathering is much higher in a wet climate than a dry climate. This can be assumed, provided sufficient drainage is available.
- For a certain quantity of rainfall, chemical weathering advances more rapidly in warm climates than cooler climates.
- The depth of the water table influences weathering, as it determines the depth to which air is available, either as a gas or in solution.
- The type of rainfall is important: short intense rains erode and run off, while rain of light intensity falling over a long duration will soak in and aid in leaching.

Weinert (1964) developed a climatic N-value, calculated from evaporation and precipitation data. The climatic N-values for southern Africa are displayed in Figure 2.2. In areas where the value is less than five, a chemical decomposition mode of weathering predominates, while in areas represented by an N-value greater than five, mechanical disintegration (physical) modes of weathering predominate. According to Brink (1967), two broad but useful generalisations can be made regarding the N-values of soil profiles:

- In the more arid western part of the subcontinent, where N is greater than five, residual soils are generally shallow, and the depth of layers of transported soils can vary greatly in thickness.
- In the more humid eastern and northern parts of the subcontinent, where N is less than five, residual soils are generally deep and transported soils are shallow.

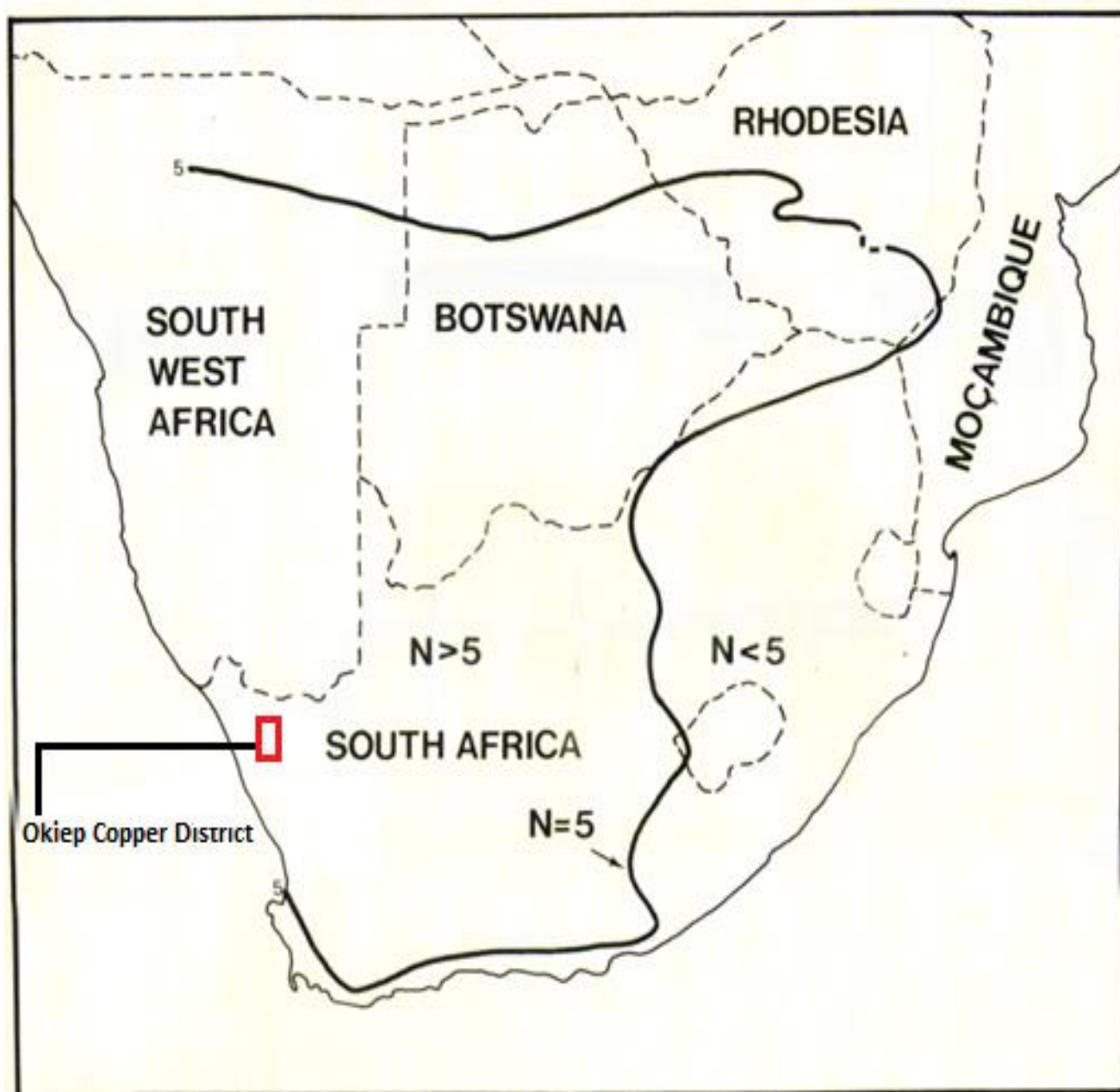


Figure 2. 2: Climatic N-valus plotted accross southern Africa (after Weinert, 1967).

The *in situ* weathering of rock results in the formation of residual soils (Zeevaert, 1983). As seen above, there are two types of weathering, which are largely determined by the N-value of a specific region. According to Figure 2.2 the Okiep Copper District falls into a climatic region with an N-value greater than five. Therefore the Okiep Copper District is dominated by physical weathering and only very little, if any, chemical weathering is present. The process of physical weathering is now explained, followed by a brief description and some common examples of chemical weathering (Mitchell and Soga, 2005):

(a) Physical weathering

Physical weathering is the process of *in situ* breakdown of rock without chemical change.

- Unloading

Cracks and joints may form to depths of hundreds of meters below the ground surface when the effective confining pressure is reduced. The reduction of confining pressure can result from erosion, uplift, or changes in fluid pressure. Exfoliation is the peeling, or spalling off, of surface layers of rocks. Exfoliation may occur during rock excavation and tunnelling. The term 'popping rock' is used to describe the sudden spalling of rock slabs as a result of stress release.

- Thermal expansion and contraction

The effects of thermal expansion and contraction range from the creation of planes of weakness resulting from strains already present in a rock, to complete fracture. Repeated frost and insolation may be an important cause of this weathering in some desert areas. Fires can also cause rapid temperature increase and rock weathering.

- Crystal growth, including frost action

The crystallization pressures of salts, and the pressure associated with the freezing of water in saturated rocks, may cause significant disintegration. Many talus deposits have been formed by frost action.

- Colloid plucking

The shrinkage of colloidal materials on drying can exert a tensile stress on surfaces with which they are in contact.

- Organic activity

The growth of plant roots in existing fractures in rocks is an important weathering process.

Generally, physical weathering processes are the forerunners of chemical weathering. Their main contributions are to loosen the rock mass, reduce particle sizes, and increase the surface area available to chemical attack.

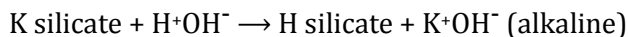
(b) Chemical weathering (Mitchell and Soga, 2005)

This weathering process transforms one mineral to another, or completely dissolves the mineral. Practically all chemical weathering processes depend on the presence of water. Hydration, that is the surface adsorption of water, is the forerunner of all the more complex chemical reactions, many which proceed simultaneously. There are five different forms of chemical weathering which include hydrolysis, chelation, cation exchange, oxidation and carbonation. Out of these five processes hydrolysis and oxidation arguably the most common.

- Hydrolysis

Hydrolysis involves the reaction between a specific mineral and the $[H^+]$ and $[OH^-]$ ions of water. The small size of the ion enables it to enter the lattice of minerals and replace existing cations, for example:

Orthoclase feldspar



or

Anorthosite

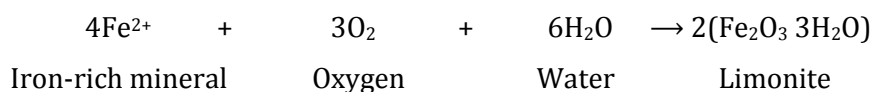


As water is absorbed into feldspar, kaolinite is often produced. In a similar way, other clay minerals and zeolites may form by the weathering of silicate minerals as the associated ions, such as silica, [potassium, calcium, and magnesium are lost into solution.

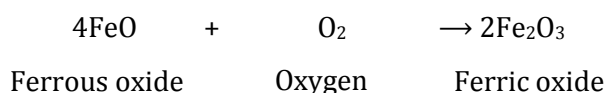
Note should be taken that hydrolysis will not continue in the presence of static water. Continued driving of the reaction requires the removal of soluble materials by leaching, complexing, adsorption, and precipitation, as well as the continued introduction of H^+ ions.

- Oxidation (Monroe and Wicander, 1994)

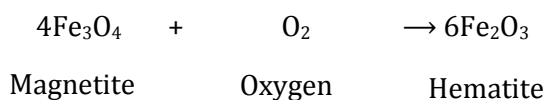
This is the process of electron loss by cations. Most important oxidation products depend on dissolved oxygen in the water. Oxidation produces iron oxide minerals (hematite and limonite) in well aerated regions, usually in the presence of water. Pyroxene, amphibole, magnetite, pyrite, and olivine are most susceptible to oxidation because of their high iron content. Iron contained in minerals combines with oxygen and water to form hydrated iron oxides as follow:



Iron contained minerals can also combine with oxygen in the absence of water for example:



And/or



With high rates of physical weathering and extremely low rates of chemical weathering the Okiep Copper District consists of shallow, undifferentiated and free draining, variably grained, sandy to loamy soils (Desmet, 2007). These soils are derived from the *in situ* weathering of the underlying parent material which is mainly gneisses and granites. The rates of fluvial activity in this region are low but can be sporadically high while wind action causes medium to high rates of erosion. This explains why the Okiep Copper District has shallow residual soils with variations in the thickness of transported soils. Weathering not only happens at ground or near ground level but also deeper down in the parent material. As discussed previously the Okiep Copper District is cross cut by shear faults and intense breccia faults with intense jointing occurring mainly in the regions of these faults. By means of faults and joints access is gained to deeper rock mass material. According to O'okiep Copper Company (1975-1993d), many of these faulted and jointed areas have experienced kaolinisation and cholitisation resulting from chemical weathering in the form of hydrothermal alteration. Hydrolysis might have caused very little of the kaolinisation as the climatic N-value for the Okiep Copper District is not favourable for chemical weathering (Weinert, 1964).

2.3 Hard rock and soft rock mining methods

Before a mining method can be selected the orebody must be probed and outlined. Sufficient information must be gathered in the process so that the most appropriate methods of mining can be elected (Hamrin, 2001). The specific mining method used depends on a number of factors including the geometry of the ore, the quality of rock mass, variability of the ore and whether a proposed method would be economically viable. It is important that a mining method be selected that assures a profit and sufficient safety for workers

The focus will be on underground mining activities. Underground mining activities commence in either soft rock material or hard rock material. Soft rock mining takes place to extract material from sedimentary rock that is usually layered, while hard rock mining takes place in igneous and metamorphic rocks.

For the purpose of this thesis, longwall mining and room-and-pillar mining in soft rock material will be explained, along with the hard rock mining methods used in the Okiep Copper District, to assist the reader in understanding the most important differences between hard rock and soft rock mining methods.

2.3.1 Soft rock mining methods

Two examples of mining methods used to soft rock underground mining are longwall mining and room-and-pillar mining. Although these methods are commonly used in soft rock mining its can be used to extract ore hosted in hard rock material as well. In this section only the room and pillar mining method will be briefly discussed, as sinkholes are prone to develop when this mining method is employed at shallow levels, for example at the Witbank coal fields in South Africa.

2.3.1.1 Room-and-pillar mining

The room-and-pillar mining method can be used for both hard rock and soft rock mining. This mining method is ideal for horizontal flat-lying deposits. According to Lee and Abel (1983), room-and-pillar mining is the most frequently used mining method in coal mines. This method of mining involves a series of parallel drifts that are driven into the ore body. Connections are made between these drifts at a 90° angle and at regular intervals, creating a checkerboard pattern (Pickering *et al.*, 2010). The result is a number of square shaped rooms separated by square shaped pillars. The pillars and rooms are more or less the same in size. Initially only 50% of the material is mined. According to Okubo and Yamatomi (2009), it is not uncommon in non-coal mines for pillars to have an irregular shape. A general layout of a room-and-pillar mine can be seen in Figure 2.3.

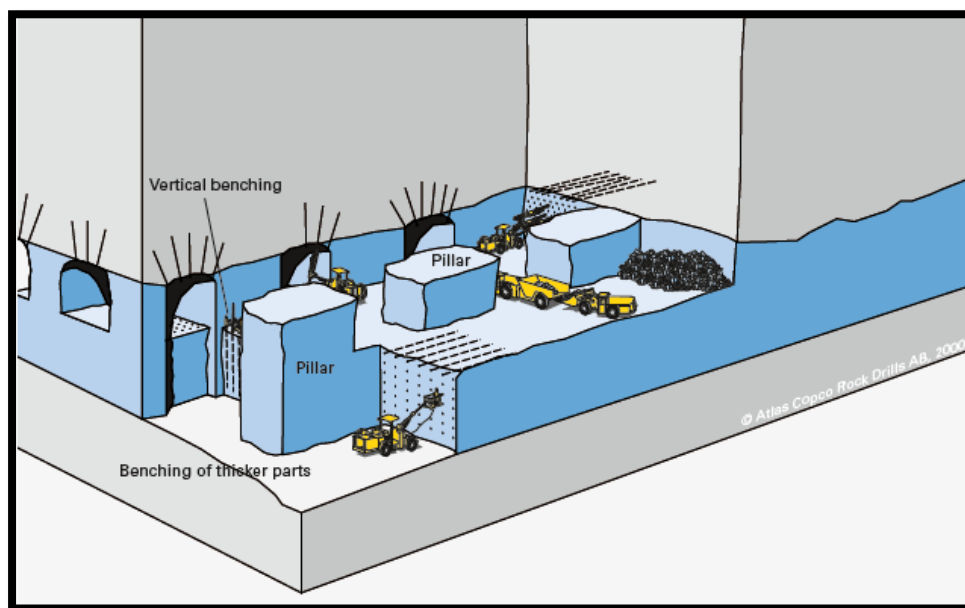


Figure 2. 3: A basic layout of a room-and-pillar mine (Eklind *et al.*, 2007).

A very important aspect of the room-and-pillar method is selecting the optimum pillar size. Pillars too small in size may result in mine collapse, while pillars too large in size can result in valuable ore being left behind, resulting in a loss or at least a reduced profit (Great Mining, 2017). After ore has been transported to the surface, following the development of the rooms and pillars, some of the pillars adjacent to the rooms containing valuable material can be removed. This process is known as pillar robbing. In some cases pillars are only partially robbed so that active roof support remains. Pillar robbing can result in the collapse of the rock above and the gradual settlement of the overburden. Settlement of the overburden can result in surface subsidence. For this reason the percentage of extraction by room-and-pillar mining must be carefully considered (Lee and Abel, 1983).

2.3.2 Hard rock mining methods

Copper is hosted in intrusive bodies of the basic rock of the Koperberg Suite. Examples of common hard rock mining methods include cut and fill, block caving and sublevel caving.

The O'okiep Copper Company (OCC) owned all the mines in the Okiep Copper District when it started mining in 1939, and until the mining activities finally ceased in 2004 (Davenport, 2014). Copper was largely extracted via underground mining methods except, at one mine, Jubilee Mine, which was an open cast mine. Copper was extracted at shallow depths, for example at O'okiep Mine, where extraction took place from 15 m below ground level. However, at other mines ore was extracted at very deep levels, for example, the Carolusberg Mine (Beukes, 2016). In his thesis Gadd-Claxton (1981) explains that the bulk of the ore was extracted with the sublevel stoping method. According to Beukes (2016) cut-and-fill and the Vertical Crater Retreat (VCR) method were also used to extract the steeply dipping ore bodies.

The discussions of hard rock mining methods that follow will focus only on the methods implemented to extract copper ore in the Okiep Copper District.

2.3.2.1 Cut-and-fill mining

- The development of a haulage drive along the footwall of the orebody at main level.
- Development of a stope undercut, as well as drains for excess water.
- A spiral ramp in the footwall, with access to the undercut.
- A raise connecting to levels above for ventilation and providing filling material.

There are two techniques used in cut-and-fill mining, which are known as overhand and underhand cut-and-fill mining. In the underhand method the ore overlies the working area while in the overhand method the ore lies beneath the working area. Ore is removed in horizontal slices, starting from a bottom undercut and progressing upward. The slices are drilled and charged with explosives. After blasting the ore is finally removed by Load Haul Dump (LHD) vehicles and dumped into an inclined tunnel or ore pass. This tunnel transports the ore to a lower elevation in the mine. LHD machines pick up the ore at the other end of the ore pass and transport the ore out of the mine via a ramp or inclined tunnel. After a stope is mined out, the void created is backfilled with deslimed sand tailings or waste rock carried by LHDs from development drives. In some cases cement is added to the backfill material to form a paste like material creating an ideal hard and smooth working surface for LHDs when it has dried out. The backfill material also serves the purpose of support for stope walls and a working platform for machines and equipment when the next slice is mined (Dunbar, 2017). There are a few key developing stages of the cut-and-fill method which includes the development of a footwall haulage drive along the orebody at main level, development of an undercut of the stope area with drains for

water, the development of a spiral ramp in the footwall with access drive to the undercut, and the development of a raise that connects the level above for sufficient ventilation and filling material (Hamrin, 2001).

This mining method is ideal for steeply dipping ore bodies in strata with good to moderate stability. Cut-and-fill is a selective mining method, as the drill pattern can be modified in such a way that ore boundaries can be followed in irregular and discontinuous ore bodies. High grade ore can be extracted, leaving low grade ore in place, or using it as backfill (Didier *et al.*, 2008). Figure 2.4 displays a basic layout of the cut-and-fill method.

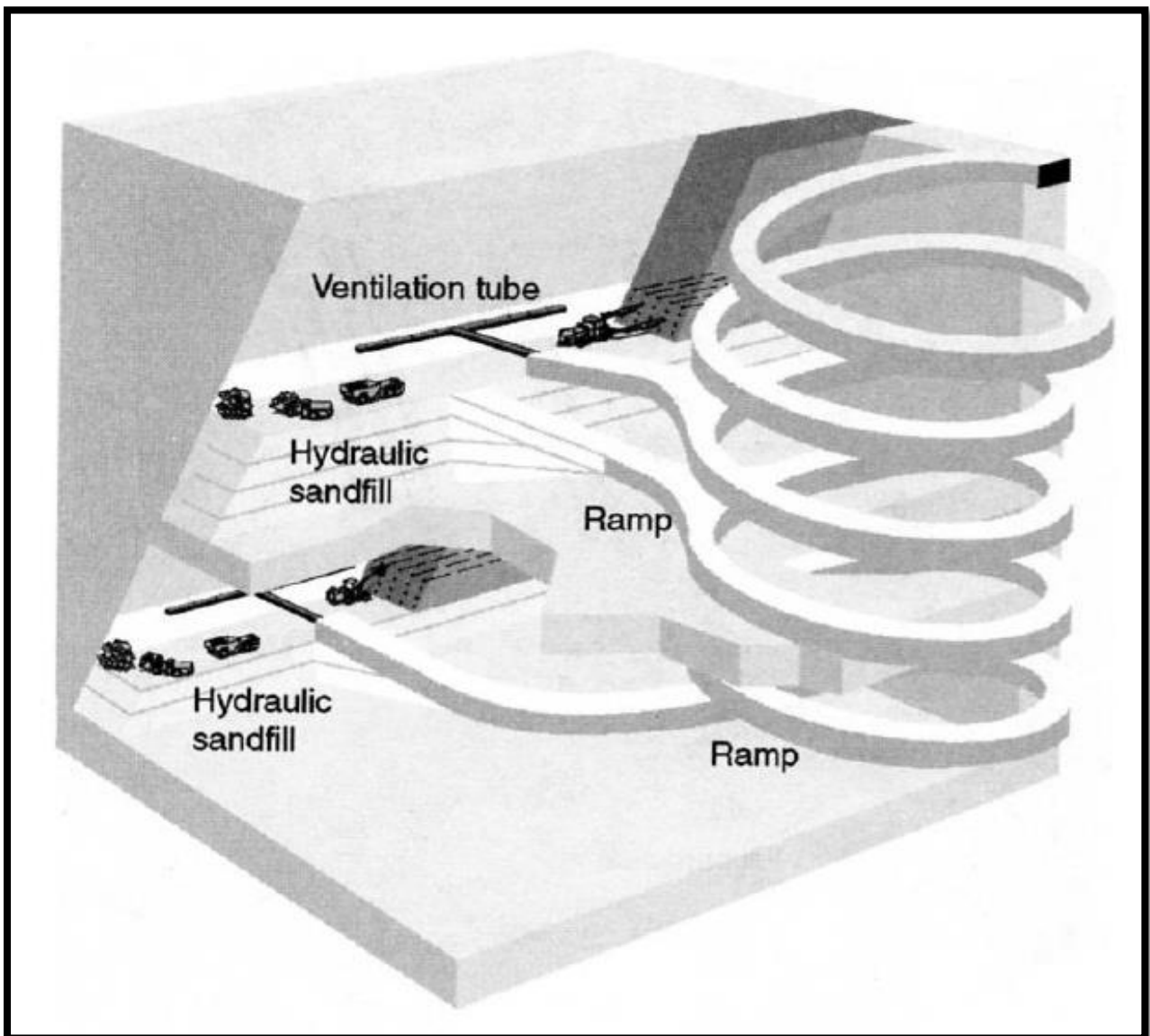


Figure 2. 4: A basic layout of the cut-and-fill mining method (Hamrin, 2001).

2.3.2.2 Vertical crater retreat mining (VCR)

In his thesis Gadd-Claxton (1981), states that the bulk of the ore in the Okiep Copper District was extracted with sublevel open stoping. However, according to Beukes (2016) the Vertical Crater Retreat (VCR) mining method has also been used in some of the mines in the Okiep Copper District. The Canadian mining company, INCO, developed VCR mining, and the method is based on CW Livingston's crater blasting theories, where powerful explosives are placed in holes with a large diameter and then fired, attempting to break the orebody in such a manner that it is able to collect in a draw point (Trotter, 1991). In Figure 2.5 the layout for the VCR mining method is shown.

VCR mining is ideal for steeply dipping layers/veins of ore with competent rock in both ore and host rock. The orebody is divided into stope blocks. Each block is mined individually. Part of the blasted ore remains in the stope over the production cycle, serving as temporary support. The developmental sequence of VCR stopes involves firstly the development and excavation of a haulage drift at drawpoint level along the orebody followed by the second step when a drawpoint loading arrangement is created beneath the stope. The third stope is when the stope is undercut to allow blasted ore to move freely to the drawpoints. The final step of the developmental sequence is the excavation of an overcut access that allows for drilling and charging of the stope (Trotter, 1991) and (Berglund and Günther, 2014).

Vertical drill holes are drilled from the overcut towards the undercut in the stope, where nearly spherical explosive charges are placed. The holes are charged from the overcut and blasting is performed from the base of the vertical holes and progresses in an upward direction. Ore is blasted off in horizontal slices and breaks into the undercut from where it is mucked away with LHDs via the drawpoints. Stopes may or may not be backfilled. When it is planned to backfill a stope it is cleaned out beforehand and filled with cemented hydraulic fill (Eklind *et al.*, 2007).

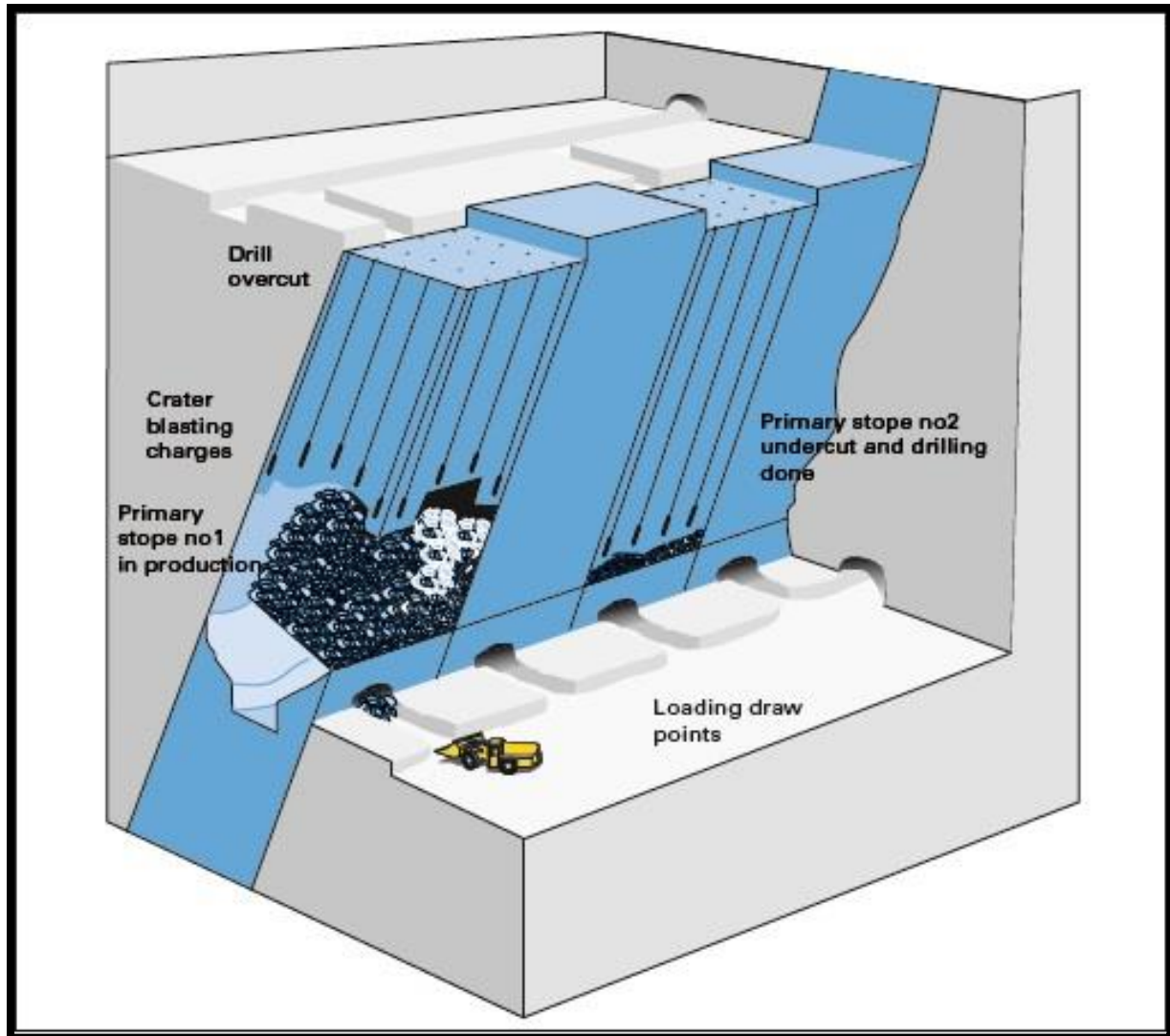


Figure 2. 5: The layout and main activities of a VCR mine (Eklind *et al.*, 2007).

2.3.2.3 Sublevel open stoping

This method of mining was used to extract the bulk of the copper ore in the Okiep Copper District and is therefore discussed in more detail.

This method is also called longhole or blasthole stoping. Sublevel open stoping is ideal for deposits with the following characteristics (Hamrin, 2001):

- Steeply dipping orebodies (the inclination of the footwall must exceed the angle of repose).
- The ore must have regular boundaries.
- The orebody must be competent, with stable rock in the footwall as well as in the hanging wall.
- Rock must be stable in both the footwall and the hanging walls.

According to Didier *et al.*, (2008), stopes can be very large, with the largest dimension being in the vertical direction. Normally, to prevent collapse of the walls, large bodies are divided into two or more

separate stopes. Rock and ore should be competent enough that little or no further support is needed (Gogolewska, 2011).

Sublevel open stoping begins with the splitting up of the ore body between main levels. It is divided into smaller blocks by driving sublevels (Gadd-Claxton, 1981). The selection of sizes for the stopes is important. According to (Berglund and Günther, 2014), the mining efficiency is influenced by the dimensions of the stope and therefore miners always aim for the largest possible stope size. The size of stopes needs to be selected with caution, as the stability of the rock mass is a limiting factor as regards the pillar and stope size. In the case of the mines in the Okiep Copper District, the stopes have a vertical interval of 15.2 m, which in some instances can be increased, up to intervals of 30.6 m (Gadd-Claxton, 1981). Normally, between stopes, ore sections are set aside for pillars to support the hanging wall. Pillars take on the shape of vertical beams. Another form of pillar, known as crown pillars, is also used to support mine workings above a producing stope and essentially they are horizontal sections of ore (Hamrin, 2001). Two advantages of sublevel open stoping is that it allows workers to work more safely under the low backs in the sub-levels, and that the access from the drilling drives to the working place is easy. In Figure 2.6 the layout of a typical mine using the sublevel open stoping mining method can be seen.

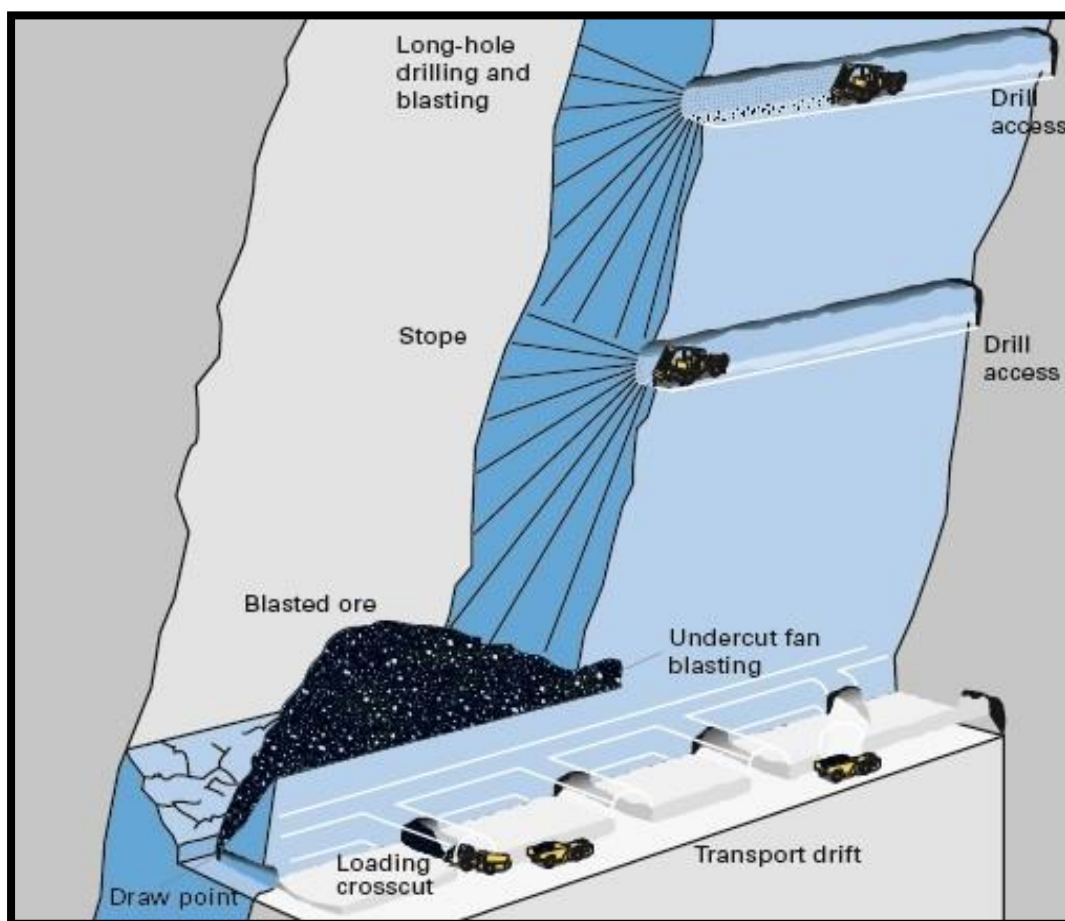


Figure 2. 6: The layout of a typical sublevel open stoping mine (Eklind *et al.*, 2007).

Inside each stope, sublevel drifts are prepared in which drilling must take place. According to Gadd-Claxton (1981) mines in the Okiep Copper District made use of ring drilling. The areas where the drilling is to commence are located in a strategic manner because of a specific blast pattern that needs to be followed. The drill pattern specifies where the blast holes will be located. It is very important that the drill pattern is very accurate with regard to the depth and angle of each hole in order to achieve a successful blast (Hamrin, 2001). Below each stope draw points are developed for successful removal of ore by LHD's. Normally these draw points are spaced at regular intervals and located on the various production levels. According to Gadd-Claxton (1981), two methods of ring drilling were used in the Okiep Copper District, which are the fan drilling pattern and the slot drilling pattern shown in Figure 2.7. When deciding on which pattern is to be used, the width, height and shape of the stope to be mined must be taken into account.

(a) Fan drilling pattern

This method of drilling is used when a large block of uniform ore must be removed in a single-face retreat operation. For this drilling pattern to take place, drilling drives need to be developed along the footwall or hanging wall contact of the orebody at each sublevel. This requires that at least one of the contacts of the orebody needs to be regular in outline. To prevent ore from being diluted as a result of wall rocks mixing with blasted ore or wedges of unbroken ore, the fan pattern must be carefully designed (Gadd-Claxton, 1981).

(b) Slot drilling pattern

When a constant width of a stope needs to be maintained, a more sequential control of the blasting of the rings is required. This is when the slot drilling method is used. The outlines of the orebody could also be irregular. A slot raise is developed along the footwall contact of the orebody. Cross cuts are also developed from the hanging wall to the footwall contacts on each sub-level. Unlike the fan drill pattern, the slot drill pattern makes use of vertical up or down drill holes from a specific level (Gadd-Claxton 1981).

Backfilling of open stopes is not always part of the normal mining procedure. However, when it is required, a delayed fill sequence is implemented using waste rock, or tailings, which may or may not be deslimed.

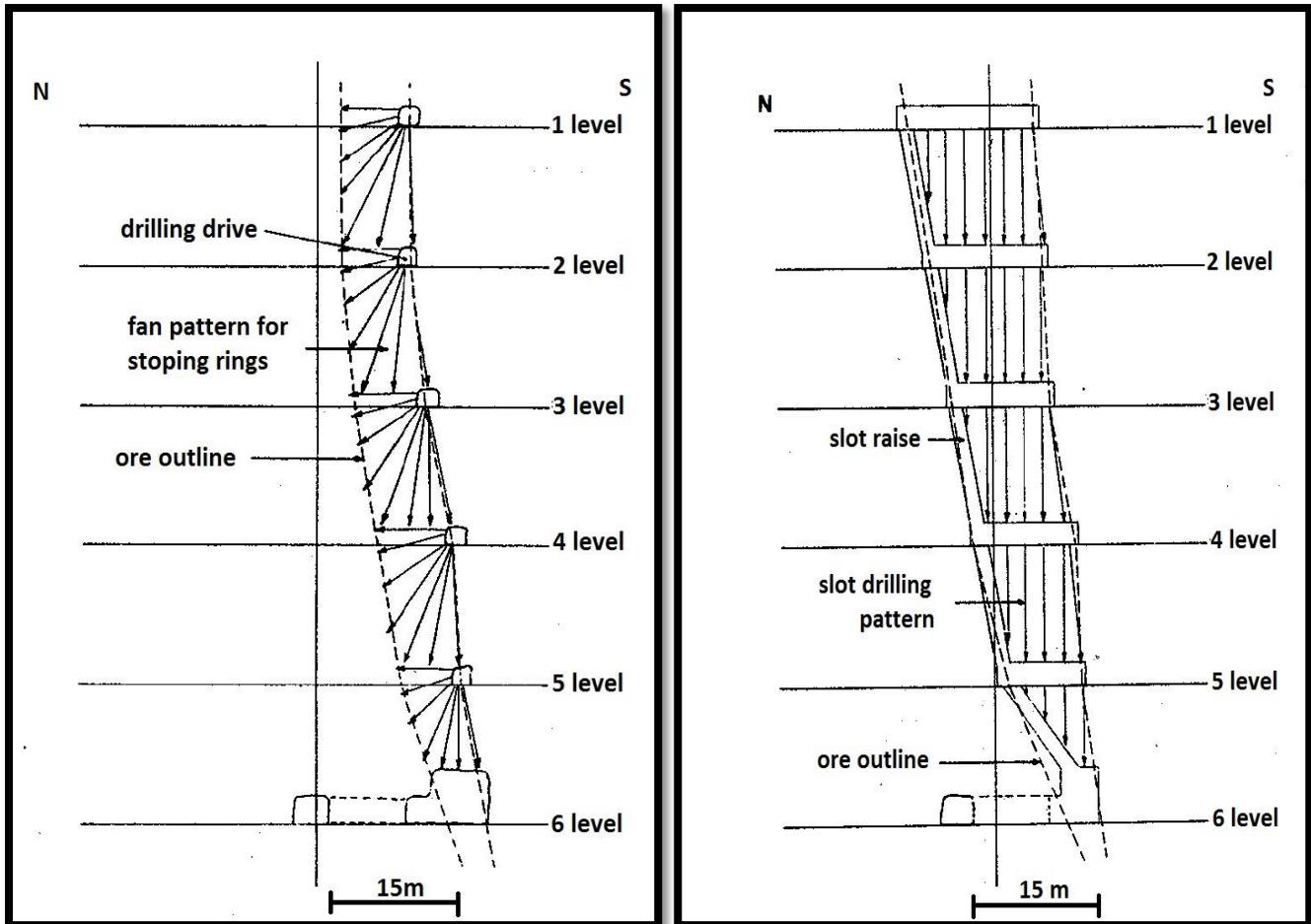


Figure 2. 7: A representation of the two different types of ring drilling that was implemented in the mines of the Okiep Copper District. (a) Fan Drilling Pattern and (b) Slot Drilling Pattern (Gadd-Claxton, 1981).

2.3.2.4 Trackless mining in the Okiep Copper District

Mining engineers in the Okiep Copper District came up with different ways of trackless mining to facilitate the viable extraction of ore from the small, low-grade Okiep type copper deposits. In the Okiep Copper District three methods of trackless mining were used. These are the conveyor decline method, direct trucking method and ore spiral method. Gadd-Claxton (1981) explains the three methods of trackless mining as follow:

(a) Conveyor Decline method

This trackless method involves the development of a conveyor and access decline shaft from an existing crusher level, coupled with the installation of an underground crusher located under a low grade orebody, situated at a greater depth than the originally worked orebody. From the access decline shaft the development of extraction haulage, drawpoints and a crusher chamber take place. After ore is mined and has been crushed below the orebody, it is transported up the conveyor decline to the ore bins. This method of trackless mining has been successfully used at a number of orebodies in the Okiep Copper District. Two examples are the Jan Coetzee Southwest mine and the Homeep East mine. Initially it was used to mine the number 1 Lower Orebody at the East O'okiep Mine.

(b) Direct Trucking method

This method involves the direct trucking of ore from the stope drawpoints to a central crushing plant. The Koperberg Mine was located 3.4 km from the Carolusberg Mine complex, which had its own crushers. In the case of the Koperberg Mine it was much more profitable to transport ore with trucks directly to the Carolusberg Mine crusher, rather than to install additional crushers at Koperberg Mine.

(c) The Ore Spiral method

The reason the Ore Spiral mining method was implemented at the Rietberg mine was to attempt to mechanise development. At the same time, it was attempted to avoid the excessive cost of a waste ramp development in areas of small tonnages of ore reserve. Usually when a sublevel stope is wider than approximately 15 m, where sublevel drives on both the footwall and hangingwall are required for efficient long-hole drilling, the Ore Spiral method is applied. The Ore Spiral method was implemented at the Rietberg Mine. The shape and position of the orebody at the Rietberg Mine permits its exploitation by means of declined and horizontal adits, and complete trackless mechanisation of development and production. The average strike length of the mineable blocks was about 130 m long and the width of an average stope 35 m.

2.4 Subsidence, collapse and mine openings

Subsidence and collapse is a likely consequence of underground mining activities. Subsidence is a phenomenon that can extend over large areas, or it can be localized and small. It can be sudden or time dependant and happen over the course of a couple of days, months or years. According to Singh (1992), since the beginning of mining, the problem of subsidence has been recognised and in literature it was mentioned as far back as Agricola's *De Re Metallica*, by Georg Bauer, published in 1556. Not only mine subsidence itself, but also mine openings left abandoned by mining companies, can be problematic with regard to subsidence and hazards.

Subsidence and collapse will be described in a broad sense in this subchapter including different types of subsidence and collapse followed by how it develops and components and factors influencing these phenomena.

2.4.1 Definitions

Mine subsidence can be defined in many ways and is used by some authors to refer to the collapse of rock mass at ground level. Blodgett and Kuipers (2002) define subsidence as a man-made phenomenon which is associated with a variety of processes, including groundwater dewatering and the mining of coal and metallic ores. In order to give a broad description of subsidence, it may be defined as the collapse of the ground surface over areas where underground material has been removed (Meier and Gibson, 2002). According to Altun *et al.*, (2010), subsidence or collapse is the

vertical downward movement of ground, which can have serious effects on communities, services and buildings. Subsidence can be a process happening gradually over time, but can also be catastrophic and happen very suddenly without any warning. At surface level subsidence is displayed as either troughs or sinkholes and, in some cases, as potholes or glory holes. Mine openings are different from sinkholes and glory holes, and are described as an entrance that gives access to an underground mine, for example shafts, adits and decline tunnels.

2.4.2 Examples of subsidence occurrence and abandoned mine openings

Subsidence and abandoned mine openings are a worldwide phenomenon. According to Meier and Gibson (2002), history has demonstrated that nearly any undermined area, regardless of its depth, where significant amounts of ore has been extracted, is vulnerable to subsidence. Examples of worldwide occurrences of subsidence include:

- In Queensland, Australia, urban development of the town of Ipswich is located above old mine workings, and subsidence of old underground cavities has resulted in the formation of a water filled opening in the back yard of local residents, right next to the house (Independent Expert Scientific Committee on Coal Seam Gas and Large Coal Mining Development, 2014). Also in Australia, in the city of Newcastle, urban infrastructure has periodically been damaged as a result of localised subsidence.
- The Henderson Molybdenum Mine is located in the state of Colorado of the USA. The molybdenum deposit is located beneath the Red Mountain. The mine started with panel caving operations of the ore body in 1976, and late in the year of 1980 the cave zone appeared on the surface as a steep-walled cavity positioned directly above the caved area underground. Tension cracks, proximal to the present day glory hole, were only noticed after the surface initially caved in 1980. It is thought that the large mined-out areas of the Henderson Mine disturbed the in situ stresses of the rock mass, resulting in the development of tension fractures. Fracturing weakened the overburden, initiating caving and eventually resulting in breaching the surface. Although the mine is located 80 km west of Denver, far from urbanised areas, the glory hole created dangerous, unstable conditions on Red Mountain, posing hazards to wildlife and humans (Blodgett and Kuipers, 2002).
- In India life threatening potholes have formed in the past as a result of sudden collapse of overburden into an underground cavity created by the extraction of coal. The potholing phenomenon generally occurs following the failure of a mine roof which has migrated through overlying strata until the failure zone intercepts unconsolidated overburden. Potholes, also known as sinkholes, can also develop from the inflow of weak and weathered strata with water through a fault plane, resulting in the formation of a cavity. When such cavities cave in, a pothole appears at the surface. These holes can have a diameter of 10 m and a depth of 15 m and create a serious hazard to life and property, as they form without any warning. In the town of Jamuna and the city of Kotma potholes with a depth of 63 m have been reported (Singh, 2000).

Subsidence, collapse and mine openings are a problem not only in foreign countries, but are a concern in South Africa. The importance of this problem can be seen by the following discussions and examples:

- On the 5th of February 2016 major land subsidence occurred at the Lily Vintage gold mine in the Barberton goldfields in South Africa (Pijoo and Nyoka, 2017). According to the African News Agency (2016), the reason for the disaster was a crown pillar collapse, in which 79 workers were initially trapped underground, of whom only 76 could be rescued. More recently, a disaster related to mine openings occurred on the 25th of February 2017 in Jerusalem, a small informal settlement outside Boksburg, where a 5 year old child fell into an abandoned shaft (Sebaka-Ginindza, 2016). Figure 2.8 shows the size and extreme hazard the mine opening poses to local residents



Figure 2. 8: The mine opening a five year old child fell into while playing outside of his house in Boksburg (Pijoo and Nyoka, 2017).

- In the Witbank coalfields, coal mining commenced at the turn of the twentieth century. In the early days room and pillar was the extraction technique of choice. A key to successful room and pillar mining is the selection of the optimum pillar size. Pillars too small in size will result in mine collapse and pillars too large in size can reduce the profitability of a mine, since valuable material is left behind. Initially, little or no environmental degradation was associated with mining in the Witbank coalfields; however, in the late 1930s a pillar-robbing programme was undertaken, which had a drastic effect on the environment. Increased stress on the remainder of the pillars led to pillar collapse. Primary effects of the pillar robbing programme included surface subsidence, the appearance of tension cracks, and crownhole development. Secondary effects included spontaneous combustion of the remaining coal, as well as negative impacts on

groundwater resources. Burning of coal accelerated the weakening of the pillars and the collapse of inter-pillar tensional areas resulted in an upward void migration through overlying strata until the weathered zone was reached. In general, the weathered material has subsided by 2-3 m, but in some instances the material has collapsed totally into old workings, leaving cavities 15-20 m deep (Council for Geoscience, 2011).

According to Heath (2009), the closure of mine openings was not a major priority for mine operators in the past, and guidelines as to how openings should be closed off were not available in many countries. Historically in the United Kingdom two common techniques used to close an old shaft were (a) to put a large tree against the shaft sides and then backfilling and (b) building a platform made from wood 3-15 m below the surface and then backfilling (Gallagher *et al.*, 1978). In many instances the wooden platform failed as the wood rotted away. A local example of insufficient closure of a mine opening is in Boksburg, where builders' rubble and waste rock have been dumped into abandoned openings in a random and uncontrolled fashion. According to Meier and Gibson (2002), abandoned mine openings have the potential to expose residents and business owners to dangerous mine-related hazards when urban zones develop within easy commute of these openings.

Abandoned shafts can be filled completely or partially with material, and in some cases left open. It is unlikely that deep shafts are filled completely, while they can be filled partially when a staging is placed at the rock head or just below ground surface. Part of the shaft lining can also be removed and the staging placed on top of the rim of the remaining shaft casing. Usually shafts that are not filled are capped off in some way (Culshaw and Waltham, 1987). According to Heitfeld *et al.*, (2006), abandoned shafts completely filled with unstable filling, filled partially, or containing no fill can lead to sudden collapse features at banking level, and abandoned shafts not filled properly can experience backfill run-out that happens during a flooding period, when underground galleries are connected to the shaft, which can ultimately result in massive surface depression at ground level. In very old abandoned mine openings the remobilisation of backfill material can lead to shaft lining failure and, as a result, surrounding non-competent rock may collapse, leading to crater/glory hole formation with diameters of tens of metres. Surface collapse can also happen from the failure of a stage located on top of the shaft, for example a wooden cap, surface slab or an incorrectly sized plug. If surface collapse happens while the shaft lining remains stable, the depression may be confined to the diameter of the shaft. If lining fails, a larger crater may develop at ground level around the pre-existing shaft location (Didier *et al.*, 2008). The depression created at ground level by shaft failure is displayed in Figure 2.9.



Figure 2. 9: A collapse feature created at ground level as a result of shaft failure (Didier *et al.*, 2008).

It should be noted that both abandoned openings and abandoned mines in South Africa cause problems to the environment and humans:

- According to (Coetzee, 2013) acid mine drainage (AMD) occur on a regular basis at coal mines in the Mpumalanga Highveld, affecting underground water resources, wetlands, and rivers. In the pre-mining environment the water table was higher than during mining activities and flowed naturally through soil and weathered rock. However, during coal extraction, mining opened up new pathways for water ingress, resulting in a lowered water table. Water comes into contact with pyritic coal and the sulphide minerals oxidise to form AMD in an aqueous system in the mining environment. Seepage of AMD causes it to reach rivers, underground water resources and potentially the surface.
- The Edendale Lead Mine (EML) operated from the 1890's through to 1928 when it was abandoned. The mine was abandoned before environmental legislation in South Africa. According to (Glass, 2006), who did a study about the environmental impact of ELM, it was found that soil had such high concentrations of lead that it exceeded the European Union target and intervention standards. The high concentrations of lead in soil were determined to be from the slag heap and waste rock dump that contained an abundance of anglesite (PbSO_4) and leadhillite ($\text{Pb}_4(\text{SO}_4)(\text{CO}_3)_2(\text{OH})_2$). The high concentrations of lead in the soil are of considerable concern, because of its toxicity and the number of people at risk, namely, all

students and staff at the Edendale Primary and High Schools, as well as users of the Edendalespruit and local groundwater sources.

- A study conducted by Heath (2009) with regard to unsafe mine openings found that one of the legacies of abandoned mines in the Central Witwatersrand Mining Basin is the presence of many unsafe openings such shafts and subsidences. Many of these abandoned openings are located dangerously close to settlements and create a substantial risk for persons living in these areas. Heath (2009) mentions a woman seriously injuring herself when she fell 20 m down a narrow (0.5 m) opening as she was inspecting a pipeline in Makause Informal Settlement, Germiston.
- In the Okiep Copper District of Namaqualand abandoned mines have numerous impacts on people and the environment. In the mining town of Nababeep an abandoned copper processing pond is located in shallow, unlined colluvial soils. According to Smuts (2015) acids, used in ore processing, together with acid mine drainage generated from tailings exposure are a contamination risk to water resources and to the biodiversity of the Nababeep area. In a study conducted by Smuts (2015) it was found that the soils from this area do not regulate acid mine drainage from the pond and thus pose a risk for nearby water resources.

For example, copper concentrations in the West O'okiep glory hole, located in the middle of the town of Okiep, are too high for the water to be locally used and have the potential to contaminate local underground water resources (Fourie, 2017).

Glory holes are also an unfortunate legacy left behind by abandoned mines in the Okiep Copper District. The holes create a substantial risk for persons living in close proximity to these holes as well as animals active in the area. Fencing and gabion walls have been erected around some of these holes, aiming to keep people and animals out of these danger zones. Before fencing and walls were installed people used water filled holes for swimming and, in some extreme cases, suicide.

Groundwater solution of the bedrock is the most common form of natural cavity creation. Natural cavities occur significantly in limestone, dolomite, gypsum and salt. The most important cavern forming rocks are limestone and dolomite. Major cavities form in strong carbonates like compact limestone and dolomite that normally display macro-scale cracks, while chalk and travertine, which are a more porous limestone, develop micro-cavities. The cavities are created by groundwater solutions when secondary enlargement of the primary fractures takes place (Culshaw and Waltham, 1987). In the 1960s, the Malmani dolomite landscape of the Transvaal Sequence on the Far West Rand experienced ground subsidence in the form of sudden catastrophic sinkhole formations and extensive development of dolines by gradual subsidence. The management of the gold mines had decided to de-water some of the dolomitic compartments to allow their workers to extract the gold-bearing ore of the underlying Witwatersrand Supergroup economically and safely. On 12 December 1962 the first of

the many sinkholes formed as a result of dewatering, and with a diameter of 55 m and a depth of 30 m, it claimed 29 lives (Brink 1979). Figure 2.10 displays the formation of a sinkhole caused by a dewatering scenario. Figure 2.10 (a) is before de-watering starts and shows that cavities exist within bedrock or overburden, which is in a state of equilibrium because of the stable water table. Figure 2.10 (b) shows the effect of the start of de-watering. The subsurface is made increasingly vulnerable to active subsurface erosion. Erosion will result in transportation of material down the nearest slot. In Figure 2.10 (c) shows the state after the dewatering process, and indicates the formation of the sinkhole as a gradual caving of the overburden, finally reached the surface.

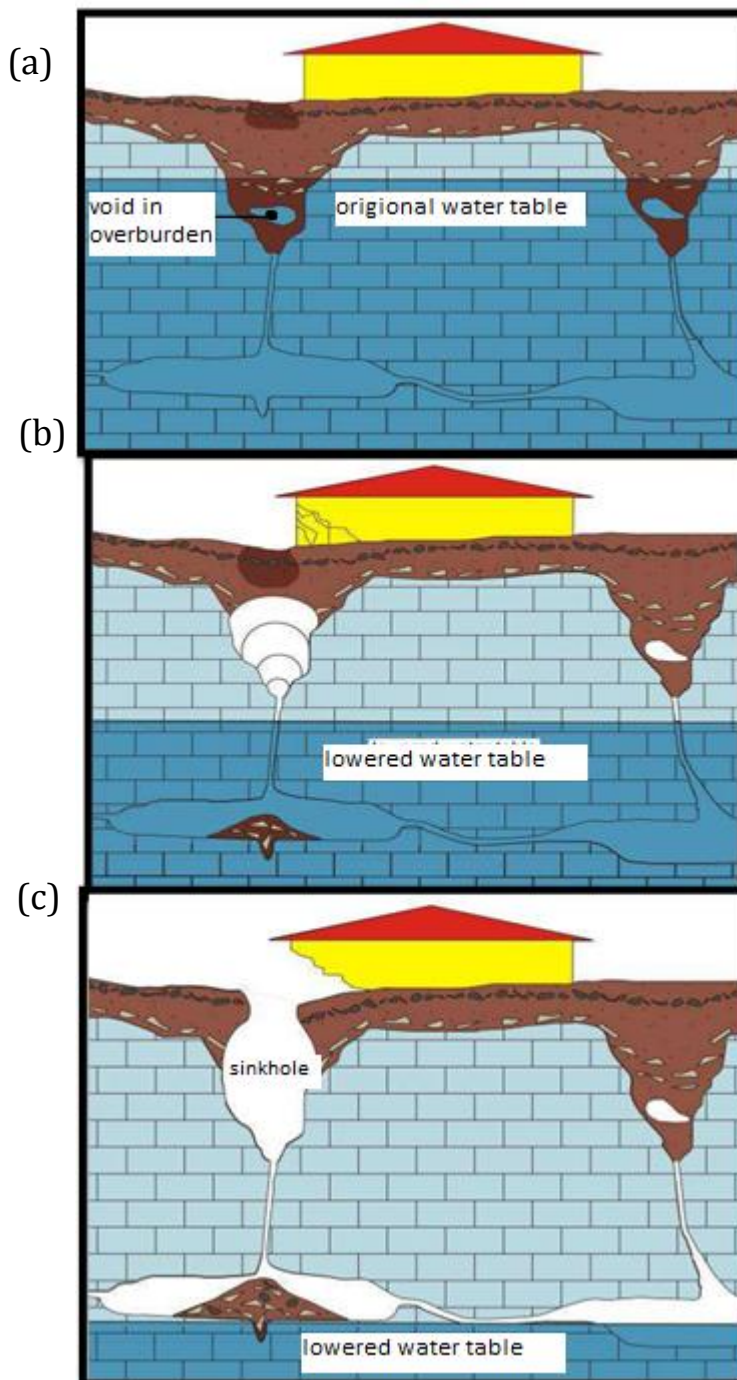


Figure 2. 10: Sinkhole formation by dewatering scenario (Council for Geoscience, 2011).

2.4.3 Subsidence development

2.4.3.1 *Manifestation*

The creation of underground cavities by mining results in the disruption of the stress field of the surrounding strata, and the displacement of strata or material results from the changes in stress. Roof failure and collapse will be initiated by large bending moments in the roof strata of the cavity and eventually the cavity will fill with overburden materials. The downward movement of overlying rock induces lateral movement of rock into the cavity (Lee and Abel, 1983). Eventually deformations will reach the surface, displaying themselves as depressions in the forms of troughs, sinkholes, open fractures and glory holes. Mining at any depth can result in subsidence and the surface area of the subsidence is generally greater than that of the extraction. Surface subsidence expresses itself in a few different ways (Singh, 1992):

- Fissures, cracks or step fractures.
- Troughs, sags and dolines.
- Sinkholes, pits, and potholes or glory holes.

When a small surface area collapses into an underground void it is termed a pit and when the collapsed surface area is large the term glory hole is used. Different countries have different terms to describe the collapse of the surface; for example in Britain these are called 'crownholes' or 'chimneys' (Singh, 1992). Countries extracting underground seams of coal from stratified deposits experience different types of subsidence. In Australia elastic subsidence is experienced under mines a couple of hundred metres below surface. Subsidence expresses itself as sags on the surface, which are termed troughs (Keilich, 2009). In some areas of India, this type of subsidence is termed 'potholes', where coal seams are mined as shallow as 40 meters below the surface. These surface depressions are in the form of a round holes a couple of meters deep (Singh, 2000). In South Africa subsidence is termed, variously, sinkholes, subsidence basins and glory holes. Sinkholes develop mainly at areas underlain by dolomite rock, and that will be discussed later on. Subsidence basins develop at shallow undermined regions, where pillar robbing programmes have been implemented in room and pillar mines, for example at the Witbank coalfield. Shallow undermining by room and pillar mining can also result in sinkholes (Council for Geoscience, 2011). In the copper mines of the Okiep Copper District in Namaqualand, collapse structures resulting from underground mining activities are called glory holes.

2.4.3.2 *Different types of subsidence*

Different types of subsidence create different types of ground movements on the surface. There are two forms of ground movements above underground mines:

- Continuous movements

- Discontinuous movements.

Continuous movements are characterized by smooth, slow and flexible readjustment of the surface and are a well-known phenomenon that leads to topographical depressions without major failure. Subsidence generally results from the collapse of underground cavities that have formed as a result of extraction or the dissolution of deposits. Slopes and deformations are directly proportional to the maximum subsidence in the centres of the subsidence trough, but inversely proportional to the depth of mining activities. The deeper a certain thickness of ore is extracted, the smaller the size of the effect (Didier *et al.*, 2008).

Discontinuous movements include sinkholes, glory holes and discontinuous subsidence. Discontinuous subsidence indicates that the subsidence occurs in shear steps and the subsidence trough or profile cannot be represented by a continuous function. Glory holes and sinkholes can be described as the sudden display at the surface of a collapse crater. The diameter of sinkholes can vary from a few metres to several tens of metres. Glory holes are larger in size and can have diameters of ± 100 m, while the depth of the crater depends on the depth of the underground mine workings or the cavity below. Discontinuous movements should not be confused with discontinuous subsidence; however, discontinuous subsidence forms part of discontinuous movements. Discontinuous subsidence influences the stability over several square metres and is caused by dynamic failures of part or all of the mining works. The height of the maximum subsidence, at the central part of the trough, can reach several metres to tens of meters. The dissolution of salt layers is also a cause of discontinuous subsidence. In contrast with the gradual slope of continuous subsidence, discontinuous subsidence has an edged trough, with open, sub-vertical cracks. These troughs have marked out steps and are hazardous to people, property and wildlife (Didier *et al.*, 2008).

2.4.4 Subsidence in different environments

2.4.4.1 Room-and-pillar mining in soft rock environments

Continuous subsidence generally happens during the underground mining of stratified or vein deposits. One of the most common minerals to be mined as a stratified deposit is coal. Coal is often mined via the longwall or room-and-pillar mining method

Historically, room and pillar mining was an economic method for mining coal beds less than 300 m deep with a strong immediate roof rock and variations in the quality of coal (Bétournay, 2004). Today this mining technique is applied to flat deposits of moderate thickness and inclined deposits with thicker beds. Room-and-Pillar mining can also be used for mining flat-bedded hard rock materials, for example copper. In a general sense the impacts associated with active longwall mining activities are experienced more frequently than impacts from room-and-pillar mining, because of the high production rates of longwall mining, larger zones of full coal extraction and the fact that with planned mine subsidence the overburden collapses immediately after coal extraction (Altun *et al.*, 2010).

As mentioned the surface settlement experienced from room-and-pillar mining is not as uniform as with longwall mining as room-and-pillar mining has several stages of coal removal and the deterioration of pillars is slow. Generally, settlement is rather an erratic, intermittent and long delayed process (Lee and Abel, 1983). Because room-and-pillar mining happens at a relatively shallow depth subsidence features can easily reach the surface.

Active subsidence in room-and-pillar mining without the recovery of pillars is generally small and ground level may experience a variable frequency of subsidence in this period. The reason for this is that surrounding rock and pillars are relatively intact during this period causing only minor deflections of the roof to be carried to the surface. Complete collapse of abandoned pillars and the adjacent strata may occur as a result of human activities or natural causes some time after underground extraction. Complete collapse of the underground void may also happen when pillars are too extensively or completely robbed and this can result in a more uniform settlement at surface level. Collapse and subsidence are likely to take place until the voids created by room-and-pillar mining have been filled with caved material. This consequently results in residual subsidence being the major subsidence measured at ground level in room-and-pillar mining (Singh, 1992). According to Singh (1992), studies have shown that no matter how well designed a room and pillar layout is, the extra weight carried by pillars due to excavations will result in measurable deformations of the pillars. These movements will finally be carried to the surface and, depending on the amount of pillar loading, the characteristics of the pillars and the overhanging material, the surface deflections may vary from nearly undetectable to negligible to considerable.

The area of material extracted plays a very important role with regard to the profile of the depression created by subsidence. There are three classifications of extraction area influencing the profile of the depression. These three classifications are expressed in terms of the panel width (W) and the depth of cover (H) known as the W/H ratio (Keilich, 2009). The classifications are:

- sub-critical extraction
- critical extraction
- super-critical extraction

Different countries or areas have different ratio values for each classification, according to local factors, for example overburden geology. For the sake of an example, the classification values used in the Southern Coal Fields of New South Wales will be made use of here. Non effective width to depth ratio (NEW) implies that extraction is so little that the W/H ratio is too low for any subsidence to take place. NEW values are in the region of 0.2-0.8. Sub-critical extraction is when insufficient extraction takes place to produce maximum subsidence. W/H values for sub-critical extraction values lie between 0.8-1.4. A critical extraction is one just large enough to produce maximum subsidence in the centre of a panel with W/H ratios 1.4-2.0. Super-critical extractions are those having W/H values of 2.0 and

higher. Super-critical extraction allows development of the full potential subsidence. The difference between super-critical and critical extraction is that in super-critical extraction maximum subsidence will occur over a greater area at ground level while critical extraction allows maximum subsidence at one point at ground level (Singh, 1992). Figure 2.11 (a) shows a comparison of sub-critical, critical and super-critical depression shapes. Figure 2.11 (b) shows a comparison between the three types of extraction, including a non-effective width to depth ratio, and the amount of subsidence taking place above each type of extraction.

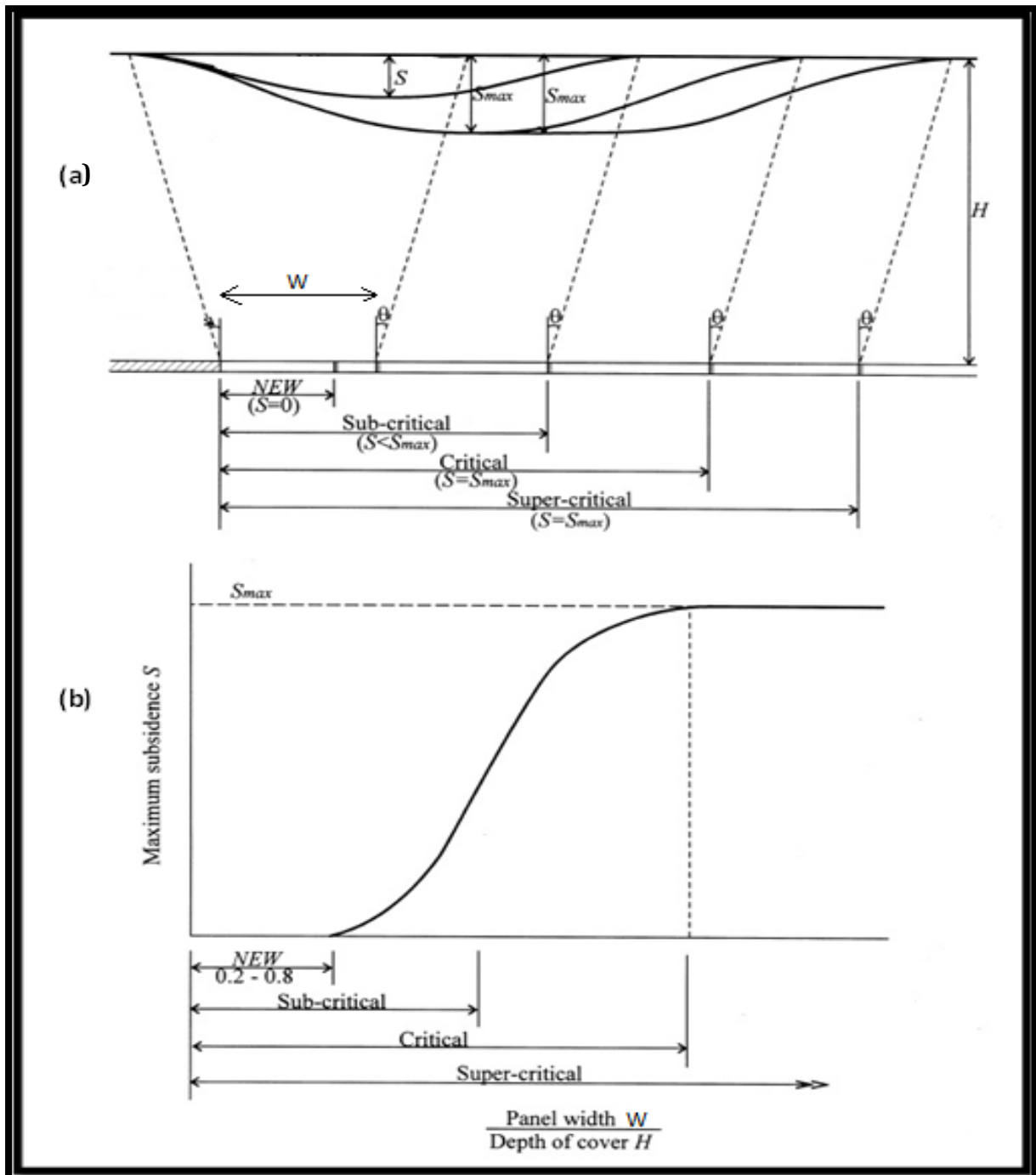


Figure 2. 11: The three classifications of extraction in (a) and the amount of subsidence taking place for each classification in (b) (after Keilich, 2009).

Mechanisms responsible for residual subsidence over room-and-pillar mining include (Singh, 1992):

- **Pillar Failure**

Pillar failure can occur during the time of extraction or after considerable delay and is caused by environmental changes or increased loading. With regard to pillar geometry, pillar failure does not commonly occur at a shallow depth. When pillars are too small to carry the overburden roof, the result is significant loading of adjoining pillars by arching, causing extended failure. Pillar failures normally correspond to some phase of mining, which includes robbing of pillars on retreat, abandoning of a certain mining area, and the mining of seams in close proximity to one another

- **Squeeze and crush**

Squeeze and crush is a mechanism that takes place when abandoned pillars are punched into the immediate floor or roof that might have been altered or weakened by weathering processes or the action of water. When the bearing capacity of the floor or roof is exceeded, squeezing will occur and is favoured by the following factors resulting in the settlement of the ground surface in the form of troughs or basins (Gray *et al.*, 1977):

- High pillar stress
- Underclay mine floor
- Flooded mine conditions

- **Roof collapse**

Roof collapse is the most common mechanism of failure over remnant pillars in an abandoned room-and-pillar mine. When failure is initiated and the caving process starts, it can reach the ground surface, or the process may be arrested at some point in the overburden, depending on the geotechnical and geometric factors present. Surface expressions of failure are commonly in the form of localised depressions or pits. The collapse process reaches a certain height, which is a function of:

- The volume of the room or mine opening
- The bulking factor of the strata
- The thickness and location of possibly competent overlying strata

Troughs and sinkhole subsidence are the two main forms of surface instability caused by room-and-pillar mining. A variation in ore quality and ore thickness results in irregular pillar distribution,

causing sinkhole subsidence and trough type failure as a result of roof spans which are too great and/or pillar failure. Figure 2.12 displays the formation of sinkhole subsidence. The sinkhole is created from an upward progressing chimney shaped failure, starting from a mine junction or room as indicated on Figure 2.12. The chimney shaped failure progresses through overlying strata and eventually reaches the surface, forming a sharply delineated sinkhole.

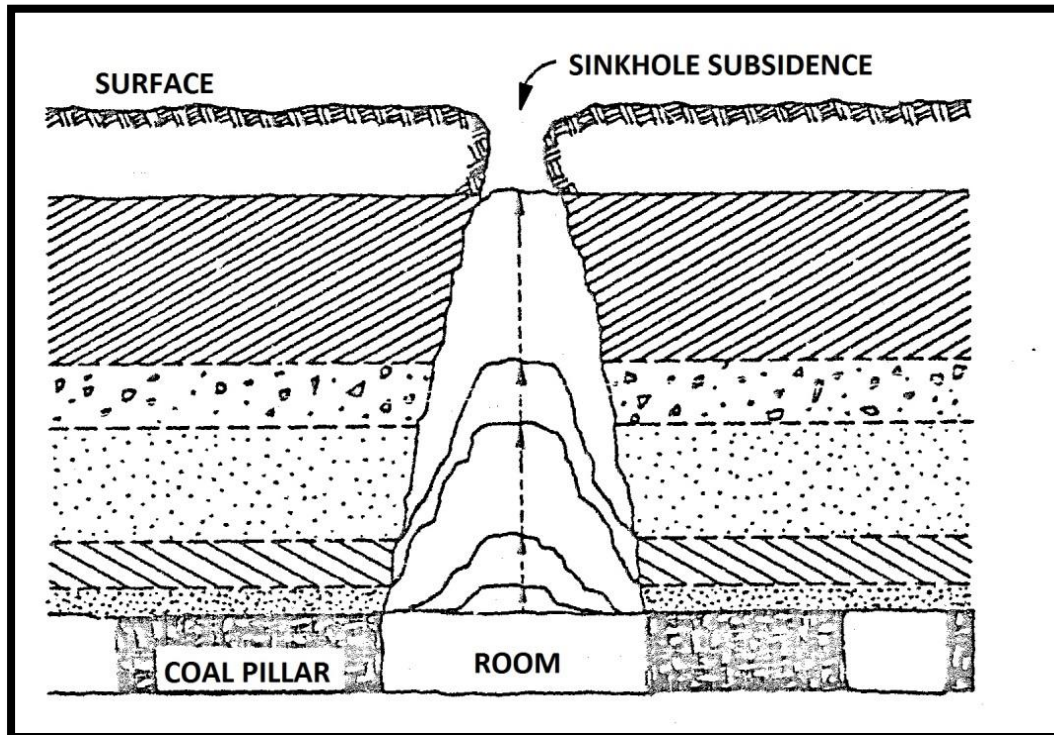


Figure 2. 12: A visual display of how sinkhole subsidence occurs from room-and-pillar mining activities (after Bétournay, 2004).

The second form of room-and-pillar induced subsidence resembles saucer shaped troughs and results from the downward sag of overburden caused by multiple pillar failures. The deterioration of clay bearing lithologies results from flooding of mines, ultimately leading to roof and floor deterioration, which initiates rock mass displacements and causes trough subsidence. The troughs are usually not much deeper than one meter at the centre but can cover an area up to 300 m² (Bétournay, 2004).

2.4.4.2 Hard rock Mines

Metal-based mineral deposits can originate in many ways and by various modes of formation, which include depositional, magmatic, intrusive, hydrothermal, epithermal, and enrichment. These modes of formation are associated with the basic geological origins such as sedimentary, igneous and metamorphic. Geological processes such as metamorphism, faulting, folding, and alteration are overprinted on these origins and can provide numerous geomechanical rock mass environments in which mining can take place, as seen in Figure 2.13:

- (a) Orebody in poorly jointed rock mass.
- (b) Orebody in blocky and jointed rock mass.

- (c) A weak schistose orebody located between competent walls.
- (d) Massive orebody located between weak schistose walls.
- (e) Ore located in generally foliated or slaty rock mass.
- (f) Ore located in well develop stratified rock mass.
- (g) Orebody in fault weakened and altered rock mass.

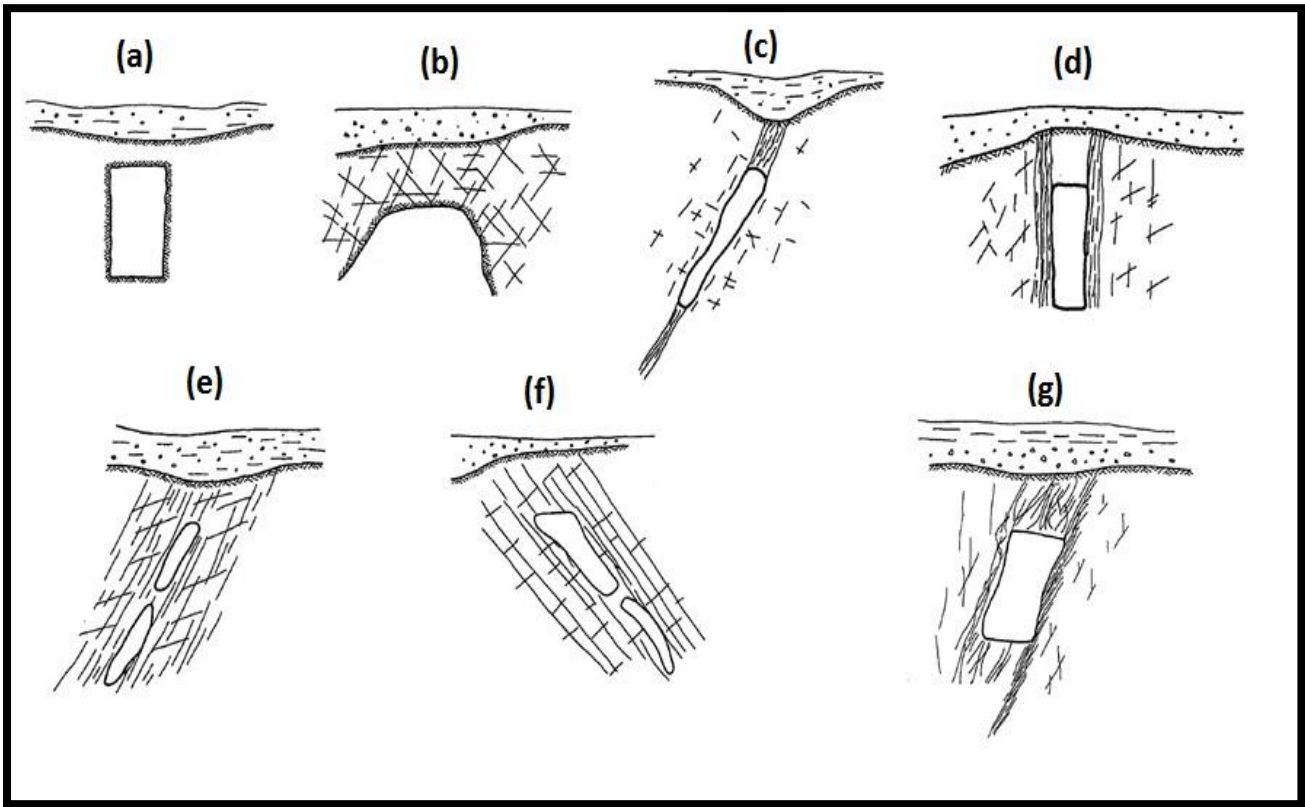


Figure 2. 13: Geomechanical rock mass environments in which mining can take place (after Bétournay, 1995).

Because metal orebodies often extend to the upper limit of bedrock, shallow underground stopes are in many instances created close to ground level. In other instances, mainly related to metamorphic formations, ore horizons do not extend as close to ground level, but contain extensive rock mass weakness that could generate failures to the surface. Generally, geological terrains are poor and have variable rock mass quality in the upper reaches of bedrock. The first 5-15 m of bedrock can be moderately to seriously altered, displaying higher jointing density and apertures. Weak rock units formed by geological processes such as faulting, shearing and metamorphism can exist at any level from the surface down to great depths and therefore represent the initial area and path of certain types of failure (Bétournay, 2004).

Five common types of failure mechanism in shallow stopes of hard rock mines include (Bétournay, 2004) and (Didier *et al.*, 2008):

(a) Rock fracturing

When high horizontal and/or vertical stresses are present in a rock mass, this can result in the fracturing of the rock mass. Fractures develop perpendicular to the direction of the least amount of principle stress. In a mining environment, when a rock mass is too intensely fractured, the rock mass can collapse into a stope as a result of rupture of the surface crown pillar.

(b) Plug failure

According to Didier *et al.*, (2008), plug failure is the sudden fall of the surface crown pillar, as an integral block, by gravity, delineated by well-defined boundary plane discontinuities, into a shallow stope. The planes are well defined with near to vertically dipping, uninterrupted discontinuities with low shear strength. The thickness of pillars can vary. Failure of pillars of up to 600 m has occurred (Allen, 1937). The potential for failure reduces significantly when bounding planes are not continuous and the dip of planes decreases from vertical to horizontal. The potential for failure depends a lot on the confinement available, from redistributed stresses around the stope, to resist movement. Areas of numerous regional shallow stopes are prime candidates for plug failure. Visual precursor movements rarely occur, however, small scale movements can cause discontinuities to open up, allowing the inflow of water. Plug failure occurs suddenly and completely, as a block.

(c) Ravelling

When left unsupported, gradual failure of the periphery resulting from unfavourably oriented rock blocks and the subsequent enlargement of an opening, is commonplace when the span exceeds self-support capabilities. Blocks of the rock mass fail, without the remainder of the rock mass mobilising on large scale, unless a stable self-supported arch cavity is formed that transfers the rock load from the unsupported span to the sides of the underground cavity. Ravelling can reach the top of the bedrock in unfavourable conditions, and can cause destabilisation of the crown pillar.

(d) Chimneying disintegration

Overburden failure is referred to as chimneying disintegration when the failure is not caused mainly by joint orientation (Dyne, 1998). Disintegration progresses upwards within a weak rock mass, forming a cavity with a lateral extent of ± 5 m with near vertical sides, as seen in Figure 2.14. The cavity develops from an underground opening towards the surface. It involves horizontal to semi-horizontal strata failure and the progressive failure of weak rock masses. According to (Bétournay, 1995), when weak, steeply-dipping material is bounded by a competent hanging wall, the disintegration path follows the contact. Prime material for

chimney disintegration is weak homogenous material with low cohesion and a shearing path of no resistance. Figure 2.14 represents an image displaying chimneying disintegration.

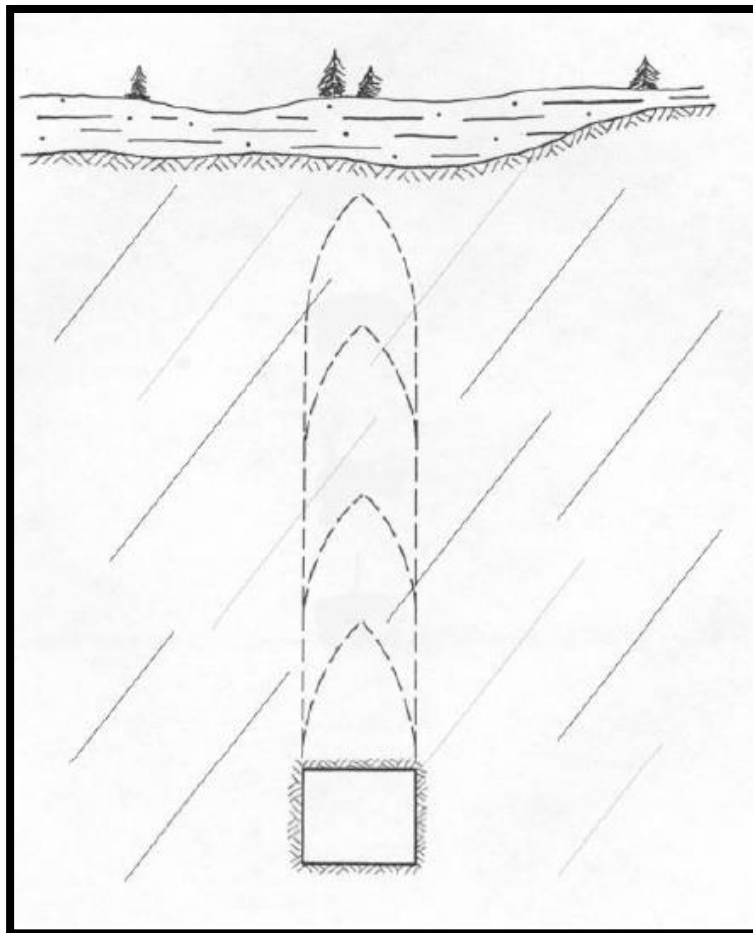


Figure 2. 14: A visual display of chimneying disintegration (Bétournay, 1995).

(e) Rock mass caving

This mechanism involves the break-up and the mobilisation of blocks into an opening by means of gravity. It results in a progressive failure front that moves towards the surface. The initiating factor for rock mass caving can be block ravelling; however, the mechanical action involved in breaking and mobilising the rock mass is difficult to quantify and predict.

The following rock mass conditions may lead to the initiation and continuation of rock mass caving in areas of low confining stresses:

- Persistent discontinuities.
- Shallow angle discontinuities.
- Blocks of similar shape.
- Low block surface friction.

Stopes with a high underground opening span, located in areas of low confining stresses, may also experience rock mass caving.

The surface effects and the dimensions of the phenomenon have been defined by Janelid and Kvapil (1966) by means of an ellipsoid drawing pattern seen in Figure 2.15. The volume of rock mass that caved into a stope is defined by the draw ellipsoid. Broken material that has moved and expanded under gravity to fill this volume is defined as the limit ellipsoid. When the draw ellipsoid intersects the surface the complete failure of the rock mass in the surface crown pillar can be expected.

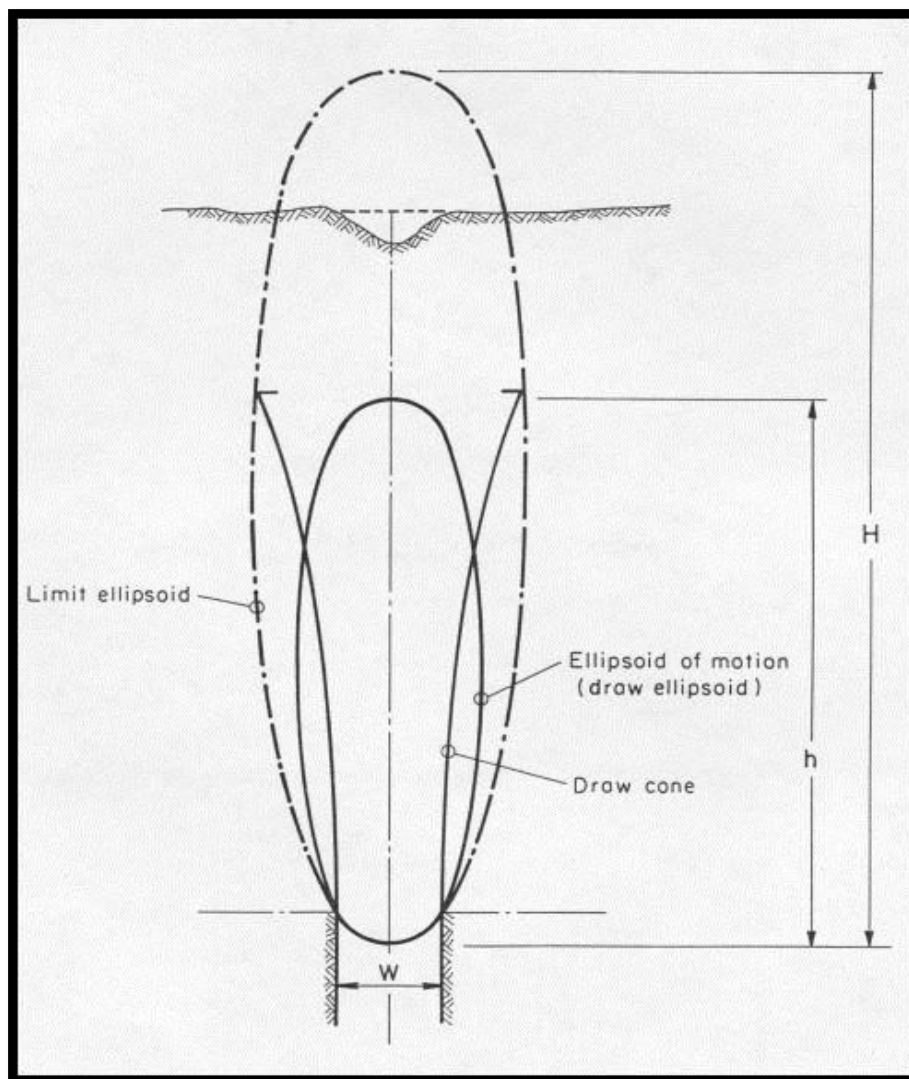


Figure 2. 15: The effect and dimensions of rock mass caving displayed by means of an ellipsoid drawing pattern by Janelid and Kvapil (1966).

Surface impacts from movement of a hard rock mass s are significantly different from broad regional subsidence relating to longwall mining or troughs that exist over failed areas of pillars. In those cases the surface of the bedrock and overlying soil is lowered and a smooth dipping surface results, with no sudden changes in elevation. However, cavities in hard rock mines can result in the sudden collapse of overburden, establishing a glory hole without any warning. Generally the surface does not significantly

subside until the last portion of the upward developing failure reaches, or is about to reach, the top of the bedrock (Bétournay, 2004).

2.4.5 Factors affecting subsidence

Subsidence is affected by many interdependent factors that control the surface expression, magnitude, and rate of development and which include:

1. **Mining method** (Blodgett and Kuipers, 2002)

The method used to extract ore has a substantial influence on surface deformations, for example longwall mining, room-and-pillar and *in situ* mining methods will affect the rock mass in different ways. Factors depending on the mining method, such a cavity shape, pillar layout and the volume of material removed, depend on the method of extraction and influence the timing and configuration of surface expressions. For example, room-and-pillar mining can delay the deformation of roofs and walls, whereas longwall mining produces contemporaneous subsidence.

2. **Depth of extraction** (Blodgett and Kuipers, 2002)

The effects of subsidence can be more prolonged the deeper the extraction takes place below ground level. However, subsidence is independent of depth, which essentially means that even if a greater thickness of overburden is left above a cavity, subsidence will still happen. The greater the ratio of overburden thickness to the depth of mining, the more likely it is that subsidence will be prolonged.

3. **Thickness of the ore body** (Blodgett and Kuipers, 2002) and (Lee and Abel, 1983)

There is a correlation between the thickness of the extracted material and the amount of surface subsidence resulting from the cavity. The amount of surface subsidence will be greater when the thickness of the extracted ore body is larger. For example, in the United Kingdom, the maximum depth of a subsidence basin resulting from coal mining of 0.9 times the thickness of the mined out seam, is reached when the span of the void reaches or exceeds 1.4 times the depth of the deposit.

4. **Lithology, structure and geological discontinuities** (Singh, 1992)

The strength and deformational properties of rock masses are largely controlled by rock type and structural features such as joints, faults and foliation planes. When mining takes place it disturbs the forces that are in equilibrium within the strata and can trigger movement along a fault plane. Overlying strata having high rock mass quality can be weakened by discontinuities that can ultimately trigger or initiate subsidence. A cavity roof formed by the blocks bounded by joints may fail by shearing along the planes of weakness when vertical stresses exceed the

shear resistance along the joints. Geological discontinuities are responsible for significant variations in the development of subsidence, and especially the extent and timing of subsidence. Fissures and joints in strata can affect subsidence in a similar way that joints do, but on a smaller scale.

5. Degree of extraction (Altun *et al.*, 2010) and (Singh, 1992)

Delayed subsidence can result from lower extraction ratios. For example, in room-and-pillar mines, lower extraction ratios mean a greater thickness of pillars, resulting in delayed subsidence. Subsidence tends to occur more rapidly and extensively when the amount of pillar support is decreased. In hard rock mines, especially at shallow stopes, when a great amount of ore extraction takes place, especially in a horizontal direction, the rock mass can become de-stressed and insufficient to prevent gravity failures.

6. Mined area (Singh, 1992) and (Lee and Abel, 1983)

The mined-out area is an important factor governing subsidence, for example for maximum subsidence to happen in longwall coal mines, the critical width needs to be exceeded. In hard rock mines the larger the extraction area, the more disruption of rock mass in equilibrium will happen, and that can contribute to subsidence. Also the larger the mined area, the larger will the effect of subsidence be at ground level.

7. Nature of overburden (Altun *et al.*, 2010) and (Blodgett and Kuipers, 2002)

Subsidence can be delayed when strong and massive beds, or other competent geological material, are present above the mined out area. The strength of the overlying material is a factor of the extent and timing of subsidence. Overburden composed of soft sedimentary rocks is associated with a gradual lowering of ground level, while delayed, sudden collapse is more typical of overburden composed of hard rocks weakened by joints, faults and rock fabrics such as foliation.

8. Time (Bétournay, 2004) and (Singh, 1992) and (Blodgett and Kuipers, 2002)

The duration of subsidence is composed of an active phase and a residual phase. According to Blodgett and Kuipers (2002), active subsidence and underground extraction happens simultaneously, for example in soft rock environments where longwall mining is used for coal extraction. Residual subsidence takes place after mining activities, for example when support pillars in room-and-pillar mines deteriorate or when caving of the overburden of a shallow stope in a hard rock mine reaches the surface forming a glory hole.

Subsidence is a function of time and interdependent on the method used for extraction. Surface effects are almost immediately noticeable where block caving mining methods are used. In

room-and-pillar mining no surface effects can be visible for a number of years after mining has been completed. Surface subsidence features will only start to emerge as soon as pillars deteriorate or punch into the mine floor. In pre-existing rock mass conditions that will not rapidly lead to failures, for example the existence of two joint sets only, where three is needed for block development which can be thought of as higher quality rock masses, failure will only occur until a much later time frame. However, rock masses with poor rock mass qualities will be expected to fail much sooner. The duration of surface movements are often delayed for longer periods the deeper the mining activity takes place. Subsidence can occur much later in deep hard rock mines, as a result of the caving process that may take much longer to reach the surface. In most cases, in hard rock environments the surface does not significantly subside until the last portions of the upward developing failure reach or are about to reach the top of bedrock.

9. Groundwater (Blodgett and Kuipers, 2002) and (Shapiro, 2015)

When strata around a mine are disturbed, drainage gradients can be altered and the rock mass may then become weakened when saturation and erosion patterns change. In areas of high rainfall, subsidence can be induced where surface water moves through fissures and fractures into the strata/rock mass at or near the mining area, serving as a lubricant in material close to cavities or mined areas, ultimately inducing subsidence. This happens because the shear resistance between fractures, joint planes, shear zones or blocks is lowered as a result of the lubricating effect of the water.

Human activity can also result in subsidence:

- The first example is what happens when groundwater is located in an area where ore needs to be extracted and the ore zone needs to be made more accessible by the withdrawal of groundwater, causing the formation of an underground cavity, once filled with water. This cavity, like cavities directly created by mining, may result in subsidence.
- The second example is known as induced seismicity, which can also result in the failure of overburden. Induced seismicity can also result in subsidence. It happens (a) when water is injected into a permeable aquifer and the pore fluid pressure increases. The rise in pore fluid pressure can cause ancient faults to be reactivated, by decreasing the frictional strength of the fault. Seismicity can also be induced (b) by changing the shear stresses acting on a fault. This happens when a volume and/or mass gets pulled out of the ground, changing the loading conditions on a fault. Both examples mentioned can cause mini earthquakes that can ultimately result in subsidence of the overburden.

10. Surface topography (Lee and Abel, 1983)

Where a horizontal ground level is present, the stresses produced by the overburden on subsurface rock are uniform; however, regions of irregular topographic relief produce irregular stress distributions in rock masses, where the height of a column of rock above a particular point varies. Hillsides, for example, tend to emphasise the surface manifestation of subsidence.

11. *In situ* stress (Lee and Abel, 1983)

Horizontal stresses restrain the development of surface subsidence by maintaining a strong ground arch in the immediate mine roof. The stability and height of the arch are sensitive to the ratio of vertical to horizontal *in situ* stresses. An arch that is highly stressed has the potential to fail suddenly as a result of gradual thinning. An arch can also fail when overburden material is de-stressed as a result of extensive horizontal mining or intense regional shallow stope mining. Gravitational forces can ultimately cause de-stressed arches to fail.

2.4.6 The Effects of Subsidence, Collapse and Mine Openings on the Environment

According to Lee and Abel (1983), damage resulting from subsidence over underground mines has been a geological hazard in urban areas for a number of years. In the United States of America there are several cases where cities and small towns have developed within a relatively close distance of undermined areas. These areas are running the risks of potential injuries and house damages as a result of subsidence. Subsidence is not limited to urban areas but can be a major problem in rural areas too. Residents of such areas are exposed to dangerous mine-related problems, including mine subsidence and/or collapse, abandoned shaft openings and poisonous gases. Subsidence causes the ground surface to deform, affecting structures including homes, bridges, communication lines and water lines. Subsidence resulting in deep surface depressions can pose a threat to human life (Meier and Gibson, 2002). A glory hole located in the middle of the town of Okiep in the Okiep Copper District is an example of a deep surface depression that poses a threat to humans living nearby.

Heath (2009) researched the location of unsafe mine openings in the Witwatersrand, South Africa and evaluated these openings based on location, hazard assessment and sealing of the openings. According to Heath (2009) informal settlements have developed over areas of abandoned mines and are located dangerously close to mine openings, making the situation life threatening. As discussed earlier, abandoned openings such as shafts can fail if not sealed properly and can ultimately result in a collapse structure at ground level creating a hazardous zone that could lead to the possibility of people or objects falling into such an opening.

Subsidence and collapse over undermined areas is able to create tension fractures, as mentioned previously, and these fractures can create pathways for gas such as methane to escape (Pennsylvania Department of Environmental Protection, 2016). Methane gas can kill woody plants and causes agricultural livestock not to graze in certain areas (Pennsylvania Department of Environmental

Protection, 2016). Tension fractures can also result in the loss of underground water sources (Lee and Abel, 1983). Troughs created by subsidence and collapse can fill with water, creating new habitats and/or swamps, and can even trap animals. Cracks and fissures can also cause the drain of water from the topsoil, which is required for the growth of plants and crops (California University of Pennsylvania, 2005). Gradient changes are also a consequence of subsidence and collapse and can cause an increase in the velocity of surface water runoff, ultimately increasing the effects of erosion. A change in gradient can also result in soil being saturated with water (Altun *et al.*, 2010).

2.5 Rock mass classification systems

For the sake of completeness of the literature study it is important to know that various rock mass classification systems exist to describe the quality of a rock mass of a certain region. However, rock mass classification systems are not further used in this study, since insufficient data are available to successfully classify the rock masses of the areas being studied.

It is important to distinguish between the properties of a rock and the properties of a rock mass. In a mining environment, it is important to know the quality of the rock mass, which can be determined by means of rock mass rating systems. The rock properties play an important role in defining a rock mass. Rock mass properties are greatly affected by lithological changes, but structural features such as faulting, folding and jointing affect the properties of a rock mass (Budavari, 1983). Singh and Goel (2011) agrees with Budavari (1983) by also explaining that the properties of a rock mass are governed by the discontinuities in the mass, and the properties of the intact rock materials. A rock mass can be defined as a matrix that consists of rock material and discontinuities. According to Singh and Goel (2011), the majority of the discontinuities occurring in a rock mass are joints. Features like faults, dykes and bedding planes are dealt with individually, as they are more localised to a specific area. The rock mass properties therefore depend on rock material properties, rock joints and boundary conditions. In the field a rock mass is exposed to certain conditions, such as *in situ* stress, and groundwater, which govern the *behaviour* of the rock. Rock mass classifications, on the other hand, are used to quantify the quality of a rock. According to Singh and Goel (2011), the behaviour and properties of a rock mass depend very strongly on the existence of rock joints and other discontinuities, as seen in Table 2.3.

Table 2. 3: Important parameters governing rock mass properties (after Singh and Goel (2011)).

Joint Parameters	Material Parameters	Boundary Conditions
Number of joint sets	Compressive strength	Groundwater pressure and flow
Orientation	Modulus of elasticity	<i>In situ</i> stress
Spacing		
Aperture		
Surface roughness		
Weathering and alteration		

Throughout the years different systems of classification for rock masses have been developed, and more than 100 years ago, in 1879, the development of an empirical approach to tunnel design was attempted (Hoek *et al.*, 2000). Some of the earliest rock mass classification systems are no longer used. Others have been developed in such a way over time that a modified version of the old systems is still in use today. According to Hoek *et al.*, (2000), the majority of the multi-parameter classification systems were developed from case histories relating to civil engineering, case histories in which the components of the engineering geological characteristics of a specific rock mass were included. In preliminary design and engineering, rock mass classifications are widely used. If these classifications are properly used and applied they serve as a powerful tool for estimating rock mass stability, predicting behavioural characteristics of rock and the selection of support systems for underground workings (Rahmannejad and Mohammadi, 2007). Milne *et al.*, (1998) explain that rock mass classifications are traditionally used to group areas that have comparable geomechanical characteristics. The classification system provides guidelines on stability performance and suggests appropriate support systems to be used. There are a number of existing rock mass classification systems, and each one places different emphases on different parameters; therefore it is recommended that at least two methods should be used during the early stages of a project (Hoek *et al.*, 2000). An effective classification system should include the following (Budavari, 1983):

- Rock deformation factors.
- Rock strength factors.
- Formation homogeneity, continuity and isotropy factors.

2.5.1 Terzaghi (Rock Load Factor)

The earliest classification system used to classify a rock mass was developed by Terzaghi in 1946 (Jauch, 2000). He classified rock masses by means of a Rock Load Factor (RLF), using a descriptive classification. His classification system focuses on the characteristics that dominate the behaviour of a rock mass, especially in settings where gravity is the dominant driving force (Hoek *et al.*, 2000). According to Jauch (2000), this classification system has been used in the past in the United States of America to select steel supports for rock tunnels. When a rock mass was evaluated, it was classified into nine categories/classes ranging from hard and intact to blocky, and to swelling rock. After the classification of a rock mass, the support for the steel arch was determined on the base of the tunnel rock load concept, developed by Terzaghi (1946). Nowadays Terzaghi's classification system is regarded as being too general to provide a sufficient evaluation of rock quality, as it does not provide quantitative information on the properties of rock masses (Jauch, 2000). As a result, the system is not applied in practise anymore.

2.5.2 Rock Quality Designation (RQD)

According to Budavari (1983) in the early 1980s, the Rock Quality Designation (RQD) classification system was the most popular system in use. For more than twenty years the RQD has been used as an individual index for the quality of rock (Jauch, 2000). The RQD index was developed by D. U. Deere in 1964 (Deere and Deere, 1989) and the original purpose of the system was to quantitatively estimate the quality of a specific rock mass from drill core logs. Jauch (2000) also explains that the RQD simply gives a description of the quality of the drill core that has been recovered by measuring the percentage of good rock within a borehole. According to Singh and Goel (2011) the RQD is a reflection of the degree of fracturing of a rock mass. The RQD does not take into account the geometrical and mechanical properties of joints or the strength of the rock, therefore the RQD only partially reflects the quality of a rock mass.

The RQD is essentially the recovery percentage of core. It is calculated by adding all the pieces of intact recovered ore longer than 100 mm. That length will then be divided by the total length of core run. That value will then be expressed as a percentage that represents the RQD (Budavari, 1983). The RQD is a very easy system to use and no special instruments are required. The calculation can be done in the field. The procedure for calculating the RQD is illustrated in Figure 2.16.

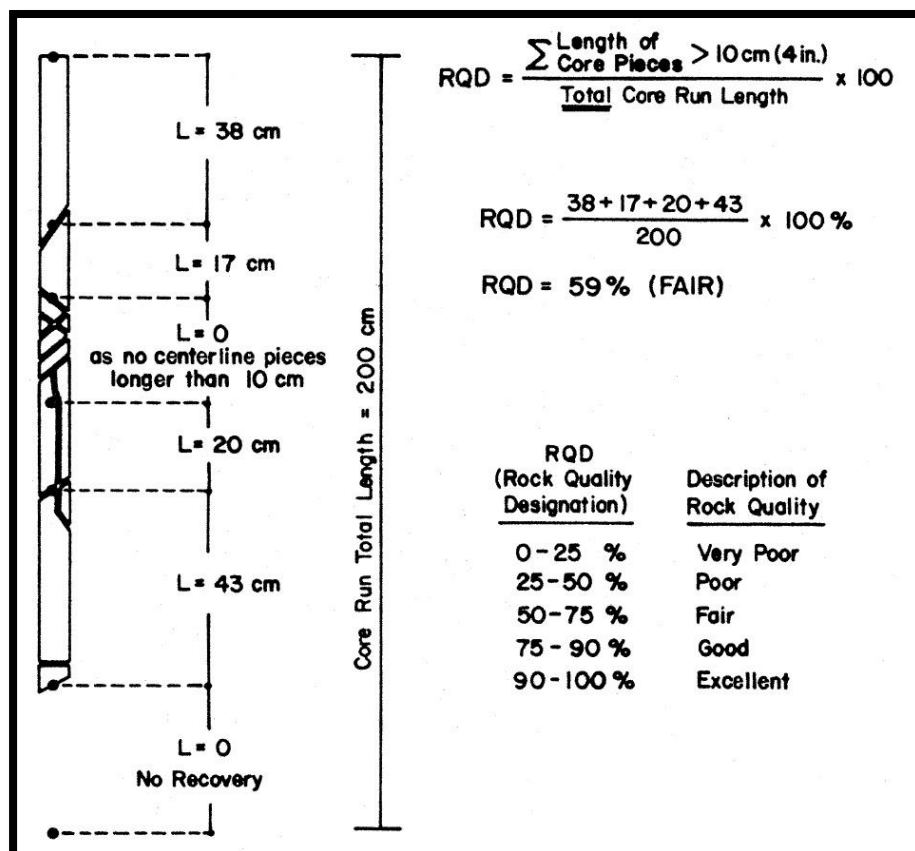


Figure 2. 16: The procedure for determining the RQD (Deere and Deere,, 1989).

Hoek *et al.*, (2000) explain that the RQD is a representation of the *in situ* quality of a rock mass and that when the RQD is calculated the core being evaluated should be handled with care. Not working cautiously with the drill core might result in fractures caused by human interference. If fractures in a drill core result from human interference they should be marked and ignored when calculating the RQD, otherwise the value will be miscalculated. As mentioned, it is a very simple system to use and it requires no special equipment. According to Jauch (2000), rock mass quality depends on several factors, for example the degree of jointing and alteration. Keeping this in mind it should be noted the RQD is also a stand-alone system and does not include many important features that have an influence on the quality of rock masses, for example compressive strength, lithology, orientation of joints, water conditions, etc.

RQD is not used on its own anymore, but rather as a basic element along with rock mass classification systems such as the RMR and Q-systems Singh and Goel (2011). It is considered as being insufficient to provide a classification of a rock mass on its own.

2.5.3 Q-System

This system was developed by Barton *et al.*, (1974) from the NGI (Norwegian Geotechnical Institute) between 1971 and 1974. The Q-system aims to determine the characteristics of a rock mass and serves as a guideline for support requirements for tunnels (Hartman and Handley, 2002). According to Norwegian Geotechnical Institute (2015), this system is used for the classification of rock masses around underground openings and it is also used in field mapping. However, the Q-system was primarily developed for classifying rock masses in the process of tunnel designing. It is applicable to core logging and borehole investigations, but it is important to keep in mind that in such cases it might be difficult to estimate some of the parameters. Q-values determined from boreholes and field mapping may be more uncertain than those from underground openings, and should be handled with care, as human interference with samples can lead to false results during calculation of the Q-value (Norwegian Geotechnical Institute, 2015). The most correct Q-values are obtained from underground geological mapping and is estimated based on six rock mass parameters determined according to numerical values in look-up tables. Table A-1.2 - A-1.7 in Addendum A displays the numerical values assigned to a geologically described situation. These six rock mass parameters are:

- RQD - Rock Quality Designation
- J_n - joint set number
- J_r - joint roughness number
- J_a - joint alteration number
- J_w - joint water reduction factor
- SRF - stress reduction factor

The six rock mass parameters are paired to express three main factors which describe the stability of the rock mass of an underground opening where:

1. $\frac{RQD}{J_n}$ = discontinuous structure of rock mass and crude measurement of block size..
2. $\frac{J_r}{J_a}$ = roughness and frictional characteristics of discontinuity surfaces and filling materials.
3. $\frac{J_w}{SRF}$ = empirical stress factor describing active stress in the rock mass.

The Q-value is determined using the following equation:

$$Q = (RQD/J_n) * (J_r/J_a) * (J_w/SRF) \quad 2.1$$

The Q-value finally describes the quality of the rock mass of an underground opening in jointed rock masses. The value of Q varies on a logarithmic scale from 0.001 to a maximum of 1 000. Q-values represent specific rock mass quality descriptions and classes ranging between A and G, as seen in Table A-1.1 in Addendum A.

With regard to the support design of underground openings, two factors along with the Q-value are important. These are the safety requirements and the dimensions of the underground opening, which are used to obtain a value representing the Equivalent Dimension (D_e).

$$D_e = \frac{\text{span or wall height}}{ESR} \quad 2.2$$

The Excavation Support Ratio (ESR) is obtained from Table A-1.8 in Addendum A. According to Norwegian Geotechnical Institute (2015), it is recommended that ESR = 1.0 should be used when the Q-value is smaller than or equal to 0.1 for B, C and D type excavations explained in Table A-1.8, in Addendum A.

After the Q-value and Equivalent Dimension (D_e) have been successfully obtained, the permanent support design for an underground tunnel/opening is determined using the rock support chart in Table A-1.9 and A-1.10 in Addendum A.

One disadvantage of the Q-system is that it is a difficult system for inexperienced users to use and therefore is recommended that the Q-system should be used only by experts or highly experienced users.

2.5.4 Rock Mass Rating (RMR)

The RMR was developed in 1973 by Bieniawski, but only in 1976 were the details for the Geomechanics Classification, also known as the Rock Mass Rating (RMR) published for the first time. The system was developed by the examination of different case study records. As more of these case study records were examined, the RMR system was improved. Over the course of the development of

the system the rating assigned to the different parameters of the system changed. The latest improved version of the RMR system is the 1989 version of the RMR system by Bieniawski (1989). According to Aksoy (2008), every area of study is unique and has its own specific features and it is therefore important that when the RMR system is used it must be used by an expert in the field of rock mass classifications.

When the RMR system is used the first step is to divide the rock mass into a number of structural regions in such a way that certain features are more or less uniform within each region (Hoek *et al.*, 2000). The boundaries of the structural regions will most likely be separated by a major geological feature such as a shear zone, faults or dykes (Bieniawski, 1989). The specific structural regions are classified according to six parameters that are measureable in the field. These parameters are measured and a value is assigned to each of them. Each parameter's value contributes to the final RMR value of the structural region, and the higher the value of each parameter, the better the quality of the assessed region. How the system assigns a value to each measured parameter can be seen in Table A-2.1 in Addendum A. The six parameters used in the RMR classification system are:

- Strength of intact rock
- Drill core quality/Rock Quality Designation (RQD)
- Spacing of discontinuities
- Condition of discontinuities
- Groundwater
- Orientation of discontinuities

The values of the ratings that can be obtained for the conditions of discontinuities (parameter number 4 of Table A-2.1) lie between 0 and 30. In section A (4) of Table A-2.1 in Addendum A, descriptions are given to assist with the allocation of a specific rating, but it is recommended that section E of Table A-2.1 should be used as guideline, as it provides more precise numerical evaluation for the details of the discontinuities (Jauch, 2000).

For the five parameters in section A of Table A-2.1 in Addendum A, an individual rating is obtained from the property of each parameter (Singh and Goel, 2011). Ratings obtained for each parameter in section A are summed together to obtain a value out of a 100; for example 80 out of 100. At this point only five of the six parameter values are summed, with the remaining value being the orientation of discontinuities parameter. The value for the orientation of discontinuity parameter is obtained by using section B and section F of Table A-2.1 in Addendum A. The value for the sixth parameter (orientation of discontinuities) is then summed to the value initially obtained out of a 100 using section A of Table A-2.1 in Addendum A. This new value is the final RMR value and is then classed into a specific rock mass class number, using in Section C of Table A-2.1 in Addendum A. The class number

is then finally used in section D of Table A-2.1 in Addendum A, to determine the meaning of the class number assigned to the rock mass (Jauch, 2000).

Ideally, the groundwater conditions should be calculated on the basis of the estimated water inflow for every 10 m of tunnel length (Jauch, 2000). Sometimes this is not possible and the easiest way to determine the conditions is to declare a specific rock mass as 'flowing', 'dripping', 'wet', 'damp', or 'completely dry'. It can be assumed that 'flowing' corresponds to a water inflow greater than 125 l/min per 10 m length of tunnel (Jauch, 2000)

Problems might occur with regard to measuring the conditions of the discontinuities in the field by means of bore holes. When relying on bore holes to assist with the calculation of the RMR rating, the discontinuity length and discontinuity aperture cannot be measured. As a result, sections A and E of Table A-2.1 in Addendum A cannot be used to determine a rating for the parameter. This problem can be overcome by making use of joint parameters, which are usually described on borehole cores. Arbitrary values are assigned to these parameters, which have to be converted to the ratings of section A of Table A-2.1 in Addendum A of the RMR system (Jauch, 2000). By introducing the following assumptions an attempt to allocate RMR values to core logs can be made (Jauch, 2000):

- A value of 2 will be assigned when a discontinuity is a fracture, value of 1 when it is a fault and a value of 0 when it is a stress release joint.
- A value of 2 will be assigned to irregular discontinuities, a value of 1 to undulating discontinuities and a value of 0 to planar discontinuities.
- For rough surfaces a value of 2 will be assigned, while for a smooth surface a value of 0 will be allocated.
- With regards to the degree of alteration of the discontinuity surface, a value of 2 will be assigned if it is unaltered, a value of 1 if the surface is slightly altered and a value of 0 if it is greatly altered.
- When an infill material is absent in the discontinuities, a value of 2 is assigned and 0 is allocated to when infill material is present.

The highest single value for the discontinuity conditions will be allocated to the most favourable conditions (unaltered and irregular surface joints with no infill material) and the lowest single value will be given to the least favourable or worst discontinuity conditions (stress release joints with a smooth planar surface and infill material) and finally, the values for each discontinuity condition will be summed.. The value obtained will be between 0 and 10, which is then compared to the values of section A of Table A-2.1 in Addendum A in the RMR system, and a correspondence can be obtained.

A very useful output from the RMR system, relating to mines, chambers and tunnels, is the prediction of stand-up time and the maximum stable rock span for a given RMR, for example a underground stope in mining conditions. Stand-up time is the amount of time an underground roof span (in metres) can be left unsupported in a rock mass with a specific RMR rating before it collapses. This proves to be useful with regard to tunnel collapse as well. This graph illustrating the relationship between stand-up time and roof span for a certain RMR value can be seen in Figure A-2.3 in Addendum A.

According to Hoek *et al.*, (2000), the two most widely used rock mass classifications are Barton *et al.*'s (1974) Q-system and Bieniawski's (1989) Rock Mass Rating. The main difference between these two systems is that the RMR system lacks a stress parameter. Therefore, as mentioned previously, it is advised that when a rock mass need to be evaluated, at least two classification systems should be used. The most common method used in South Africa is Bieniawski's (1989) Rock Mass Rating (Croukamp, 2016).

2.6 Hazard identification and risk assessment (HIRA)

Risk assessments are important as they form an integral part of an occupational health and safety management plan. It can create awareness of hazards and risks with regards to the public and/or local inhabitants and can ultimately prevent unnecessary injuries when utilized properly for example in the situation of the Bruinhoek populated area.

For this subsection it is important to define certain concepts (Western Sydney University, 2017) and (NRVRC, 2017):

- **Hazard:** Anything, for example a condition, situation, practice or behaviour that has the potential to cause harm, including injury, disease, death, or environmental, property and equipment damage. A hazard can be a thing or a situation.
- **Risk:** The likelihood or possibility that harm, for example injury, illness, death or damage may occur from exposure to a hazard.
- **Hazard Identification:** This is the process of examining an area with the goal of identifying hazards inherent to the specific location. The main objective of hazard identification is finding what could cause harm in the area.
- **Risk assessment:** Is defined as the process of assessing the risks associated with each of the hazards identified, so that the nature of the risk can be understood. This includes the nature of the harm that may result from the hazard, the severity of that harm and the likelihood of this occurring.
- **Risk analysis:** A systematic use of the available information to determine how often specified events might occur and the magnitude of their likely consequences.

Risk assessments is an opportunity to have a thorough look at situations or processes that may cause harm, particularly to humans. Following the identifications of hazards it must be evaluated to see how likely and severe the risks related to the hazards are. When this is determined measures should be put in place to effectively eliminate or control the harm from happening (Canadian Centre for Occupational Health and Safety, 2018)

According to Paithankar (2011) there are three types of hazard identification and risk analysis (HIRA).

(a) Baseline hazard identification and risk assessment

This type of risk assessment would be conducted if no previous risk assessment has been conducted. The purpose of this HIRA is to establish risk profile or setoff risk profiles. It is used to prioritise action programmes for issue-based risk assessments. This risk assessment report should reflect information on:

- All activities and tasks performed
- All workplaces
- All machinery or tools used

The frequency at which these risk assessment reports should be reviewed is not prescribed, however, annual reviews are commonly required.

(b) Issue based hazard identification and risk assessment

The purpose of conducting an issue-based HIRA is to conduct a detailed assessment study that will result in the development of action plans for the treatment of significant risk. This type of risk assessment is typically conducted for the following situations:

- After a new machine was purchased. The purpose of this assessment is to determine if the machine is safe and in compliance with legal and other requirements.
- After an incident. During the incident investigation process, weaknesses of internal policies or procedures could have been identified. The incident investigation may also show that additional precautionary measures are required as to prevent a re-occurrence.
- When new legislation has been promulgated. New legislation may prescribe requirements which are currently not being complied with. Assessing the contents of the legislation will identify any non-conformance and will allow employers to ensure compliance.

(c) Continues hazard identification and risk assessment

Regular inspections conducted on equipment or machinery, such as a daily inspection of mobile plant, could be regarded as continuous risk assessment. The purpose of conducting continuous HIRA is to:

- Identify operational health and safety hazards, with the purpose of immediately treating significant risks.
- Gather information to feed back to issue-based HIRA.
- Gather information to feed back to baseline HIRA.

The risk assessment procedure has five different steps and is displayed in Figure 2.17 below.

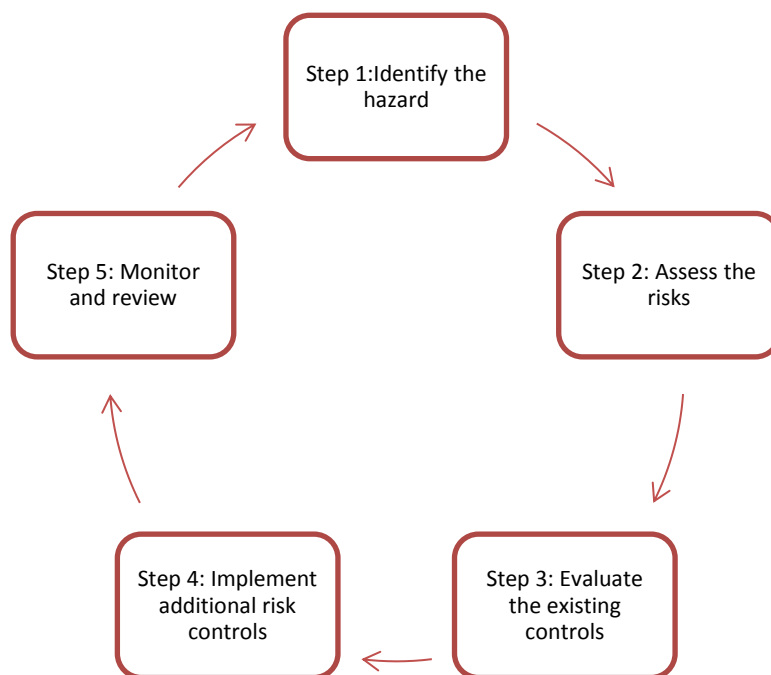


Figure 2. 17: Important steps in risk assessment (Paithankar, 2011).

- **Step 1: Hazard identification**

The purpose is to develop a list of hazards that are likely to expose people to injury, illness or even death when not effectively controlled.

- **Step 2: Risk assessment**

Risk assessment, as explained above, is the process used to determine the likelihood of people being exposed to injury, illness or disease in the workplace, arising from any situation identified during the hazard identification process.

Risk occurs when a person is exposed to a hazard and it is a measure of the probability and potential severity of harm or loss.

According to South Africa, Department of Environmental Affairs and Tourism, (2005), risk equals hazard multiplied by probability or likelihood. In order to develop a risk rating matrix, hazard/s need to be identified, as well as the likelihood of being exposed to the hazard. Table 2.4 shows a risk evaluation matrix table, scoring 2 functions which are consequence/severity and likelihood/frequency/probability, in order to calculate risk and to display it in table format.

- Step 3: Risk control

Risk control is the process used to identify, develop, implement and continually review all practicable measures for eliminating or reducing the likelihood of an injury, illness or diseases.

- Step 4: Implementation of risk controls

All hazards that have been assessed should be dealt with in order of priority in one or more of the following hierarchy of controls.

The most effective methods of control are:

- I. Elimination of hazards
 - II. Substitution of something safer
 - III. Use engineering/design controls
 - IV. Use administrative controls such as safe work procedures
 - V. Protect the workers
- Step 5: Monitor and review
- Hazard identification, risk assessments and control are on-going processes and therefore regularly reviewing the effectiveness of the hazard assessment and control measures is important.

An example of a baseline risk assessment can be seen in research conducted by Heath (2009) in the Central Witwatersrand Mining Basin where mining related openings were assessed to determine which of them should be considered for sealing. Forming part of his baseline hazard identification and risk assessment Heath (2009) developed a simple 'hole risk rating system' that assisted in identifying openings that requires sealing. He identified two criteria that were considered as being relevant in determining the risk posed by each hole:

- Depth of the hole i.e. the deeper the hole, the greater the likelihood of injury. Therefore depth represented likelihood.
- Proximity of the hole to human settlements i.e. the closer the opening to a local settlement or thoroughfare, the greater the probability of someone falling into an opening.

Heath (2009) allocated scores out of five to both criteria with scores increasing with depth and the closer a hole is located to settlements. When a hole was assessed the two scores out of five were added together that calculated an overall risk rating out of ten. Low scores indicated a low risk and high scores a high risk. Table 2.4 demonstrates the system developed and used by Heath (2009) in his research. Mine openings that had a high overall risk rating were the one's considered for sealing.

Table 2. 4: An example of a risk evaluation matrix with two scoring functions to calculate risk rating developed by Heath (2009).

Hazard rating	Description	Remarks
1	Sealed Adequately	These openings had been sealed prior to this project and currently pose no danger.
2	Slightly hazardous	Small settlement or shallow surface opening or trench. Depth < 1,5m. Minor injuries possible.
3	Hazardous	Surface opening 1,5-3m deep. Minor injuries probable.
4	Very hazardous	Surface opening is 3-10m deep. Serious injuries probable though probably non fatal.
5	Extremely hazardous	Surface opening > 10m deep, serious injuries likely and probably fatal. Inclined holes: fatal injuries due to fall of ground, gas, drowning, getting lost.
Proximity Rating	Description	
1	Opening is in a vacant field (> 1000m from any settlement or thoroughfare), area frequented by few people.	
3	Opening is <1000m from a human settlement or thoroughfare. Area occasionally frequented by passers by.	
5	Opening within or close to an urban settlement or where the holing threatens a road or walkway.	
Overall Risk Rating		
Low	1-3	Hole shallow or far away from settlements i.e insignificant threat
	4-5	Hole either a) deep (>10m) and far away (>1000m) or b) close but shallow (<3m) or c) moderately deep (3-10m) and reasonably far away (>1000m)
Moderate	6-7	Hole either a) hole deep (>10m) but far away (>1000m) or b) moderately deep (3-10m) and reasonably close to settlements (<1000m)
High	8-10	Hole deep (>10m) and close to settlements (<1000m)

2.7 Mine Health and Safety Act and Regulations

According to Department of Mineral Resources (2017), the Mine Health and Safety Inspectorate was established in terms of the Mine Health and Safety Act No. 29 of 1996, as amended, for the purpose of executing the statutory mandate of the Department of Mineral Resources to safeguard the health and safety of mine employees and communities affected by mining operations. It has a mission statement committing it to strive towards a safe and healthy mining industry and this is to be achieved by reducing mining-related deaths, injuries and ill health through the formulation of national policy and legislation, the provision of advice, and the application of systems that monitor and enforce compliance with the law in the mining sector.

The main functions of the Mine Health and Safety Inspectorate are the provision of policy inputs for the establishment, and the application of mine safety standards at mining operations, and the promotion of application thereof. In addition, the inspectorate policy will develop inputs towards the establishment and application of safety standards for mine equipment at mining operations, and promote their application; the establishment and application of mine health standards at mining operations and the promotion of these applications; and ensuring an effective support and inspection service

The Department of Mineral Resources was visited at the Springbok regional office to obtain a copy of the Mine Health and Safety Act No. 29 of 1996 (South Africa, Department of Mineral Resources, 2018) from the senior inspector of mines, Mr Willem Welding. During a discussion with Mr Welding the legislation that is applicable to this thesis was discussed. According to Welding (2017) two legislative chapters are very important for the purpose of this thesis:

- Chapter 17.7 (a)

*The employer must take reasonable measures to ensure that no mining operations are carried out within horizontal distance of 100 (one hundred) meters from **reserve land**, buildings, road, railways, dams, waste dumps, or any other structure whatsoever, including such structures beyond the mining boundaries, or any surface, which it may be necessary to protect in order to prevent any significant risk, unless a lesser distance has been determined safe by **risk assessment** and all restriction and conditions determined in terms of the **risk assessment** are complied with.*

- Chapter 17.8

*No person may erect, establish or construct buildings, roads, railways, dams, waste dumps, **reserve land**, excavations or any other structures whatsoever within a horizontal distance of 100 (one hundred) meters from **workings**, unless a lesser distance has been determined safe -*

- in the case of the employer, by **risk assessment** and if all restrictions and conditions determined in terms of the **risk assessment** are complied with; or
- in the case of any other person, by a **professional geotechnical specialist** and if all restriction and conditions determined by him or her or by the Chief Inspector of Mines are complied with.

The concept of horizontal distance is visually displayed in Figure 2.18, which explains, like the legislative phrases above, that no structures of any kind are allowed to be built above an undermined area except when the structure is erected beyond a distance of 100m away from the undermined region. According to the Mine Health and Safety Act No. 29 of 1996 (South Africa, Department of Mineral Resources, 2018), when a structure is erected within a horizontal distance of 100m from an undermined area it is regarded as unsafe.

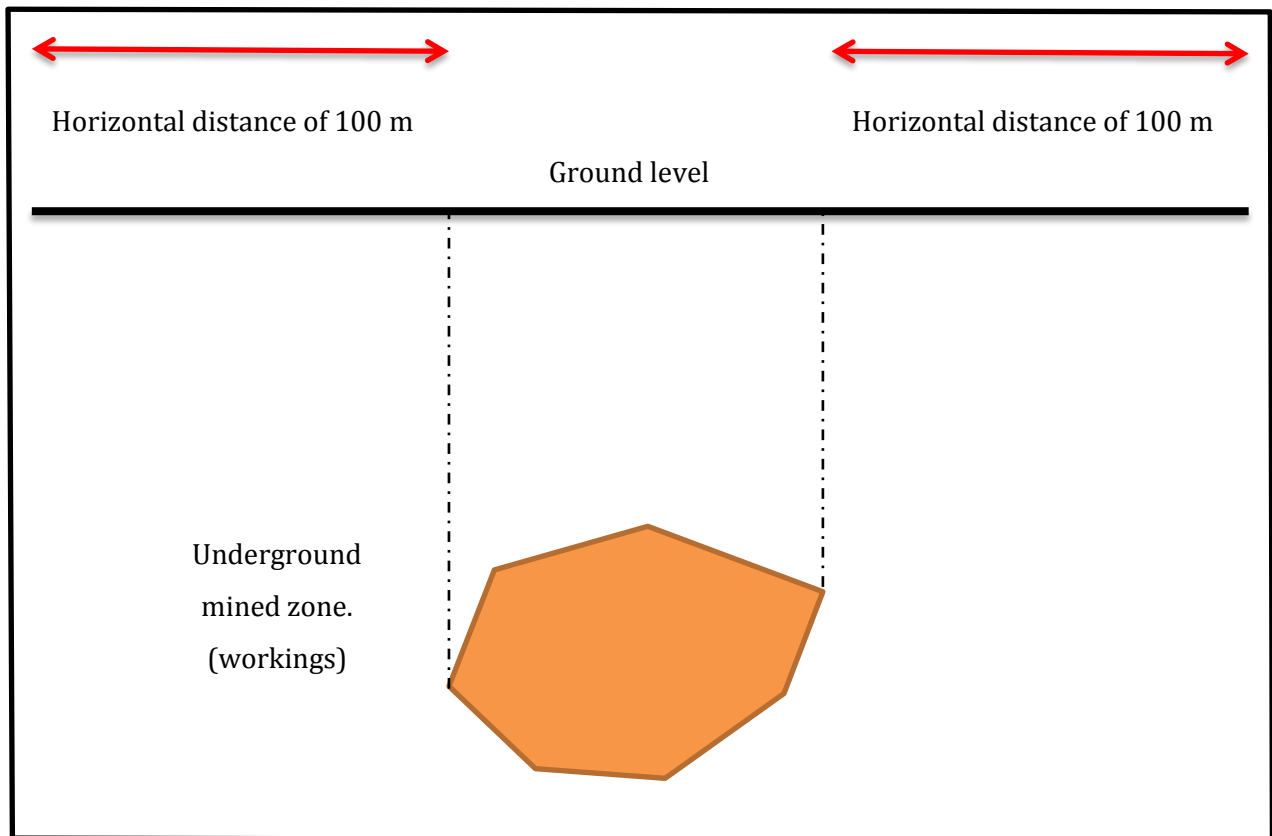


Figure 2. 18: The concept of 100 m horizontal distance explained.

CHAPTER 3: METHODOLOGY

Data collection makes out an important part of research as it is the source of the results which gets evaluated in order to draw conclusions and make recommendations. The research of this thesis is based on previous reports, a desk study and field works while this chapter deals with the methods associated with the collection of data.

3.1 Demarcation of the Research Area

The study area is located in the Okiep Copper District in Namaqualand in the south-western corner of South Africa. The Okiep Copper District has many glory holes that resulted from underground mining activities. In order to assist with the evaluation of the Bruinhoek populated area at Concordia it was decided to identify three reference mines where glory holes formed in the past. Reference mines were also investigated. The following mines were elected as reference mines:

- Rietberg Mine
- Okiep Mine consisting of an eastern and western part
- Hoits Mine

Reference mines and the Wheal Julia East Mine, which undermines the Bruinhoek populated area, are located relatively close (± 10 km) to each other as seen in Figure 3.1. Reference mines have at least one glory hole making them suitable for the collection of data. The localities are easy accessible for the researcher to conduct research and is also one of the reasons why they were chosen as reference mines. Reference mines play an important role in risk analysis and determining the reason for glory hole formation. Figure 3.1 visually displays the location of the reference mines relative to each other and Wheal Julia East (Bruinhoek rural area) which is the main focus of this thesis.

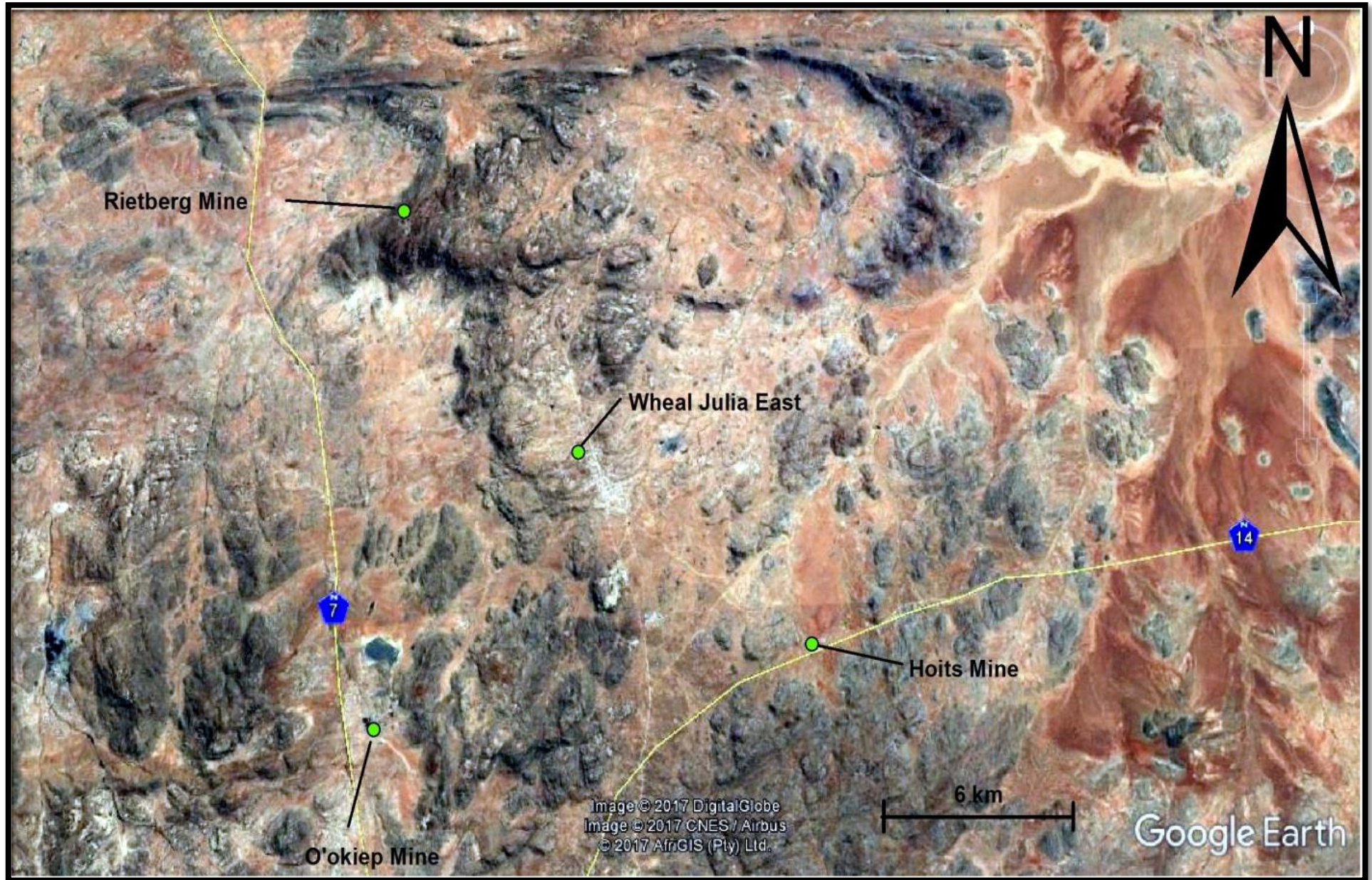


Figure 3. 1: The locations of the reference mines and Wheal Julia East relative to one another (modified from Google Maps, (2017b)).

3.2 Methods

3.2.1 Data collection and desktop study

(a) Reports

The first step in the data collection process was the gathering of reports by the author, which was done on the abandoned mines of the Okiep Copper District by Site Plan Consulting (2005) and Spiessens (2015). Comments in the reports that were specifically aimed at the mining areas concerned with this thesis, were studied to provide the author with background information on the relevant mining areas, including the undermined Bruinhoek area of Concordia. These reports confirmed some of the results obtained during the data collection and field investigation processes. They were also a source of support during the writing of Chapter 5, where the results from Chapter 4 were discussed and interpreted. The reports collected included:

- A general report consisting of remarks and some recommendations for all of the mines in the Okiep Copper District by Site Plan Consulting (2005).
- An inspection report by the principal inspector of mines (Spiessens, 2015) of the Northern Cape region.

Note should be taken that very few reports could be found.

(b) Geological- and survey maps

When mining is conducted, a record of all mining activities is kept and displayed on maps. Survey- and geological maps are arguably the most common maps in mining. The headquarters of the O'okiep Copper Company (OCC) is located in Nababeep and was visited to collect survey- and geological maps of each of the mines forming part of this study. One characteristic shared by both survey- and geological maps is that they were compiled according to the LO 17 coordinate system. Note should be taken that only a few maps were found for some of the reference mines.

- Survey maps in plan view and vertical section were collected, as they portrayed valuable information not previously known to the author. Features such as shaft locations, borehole locations, locations of underground tunnels, the locations of stopes, locations of crown pillars and depth of mining are displayed on these survey maps and this is a source of the results that will be discussed in Chapter 4 and are used for the purpose of data manipulation and map compilation processes. In this study the information from survey maps (plan view and vertical sections) was used to determine possible factors contributing to glory hole formation in Chapter 5.
- Geological maps are a crucial source of valuable underground information as access into mines is not possible. Like the survey maps, geological maps indicate the stope and

tunnel dimensions at each level of the mine. Additionally, geological maps contain crucial structural and lithological characteristics not provided by survey maps. Structural features such as breccia faults, joints and the strike and dip thereof, are indicated on these maps as well as their concentration. The locations of lithologies across the maps at each level are indicated too. The geological maps are the most important source of results for this research and what they show are displayed and discussed in Chapter 4. In Chapter 5 these maps are processed and used along with other maps/images to determine causes for glory hole formation.

It should be remembered that it was not possible for the author to gain access to mines. Therefore underground information with regards to structural and lithological information was only gathered from geological and survey maps and the comments made on the maps by the surveyors and geologists that composed them.

The author did not conduct a survey of joints and faults in the field. Mines had been abandoned and entrances sealed off, preventing access to fresh geological outcrops. Instead, with help from Beukes (2016), who was a geologist in the Okiep Copper District, data from joints and faults gathered from the geological maps was processed with Stereonet version 9.9.4, (2017). The software produces stereonet, diagrams displaying the fault and joint data visually. Stereonets are added onto the original geological maps in Chapter 4 as well as the maps created with ArcMap™ 10.5 in Chapter 5. Stereonets play an important part in Chapter 5 during the data interpretation process and contributes to identifying the reasons for glory hole formation. Examples of stereonet created by Stereonet version 9.9.4, (2017) can be seen in Figure 3.2.

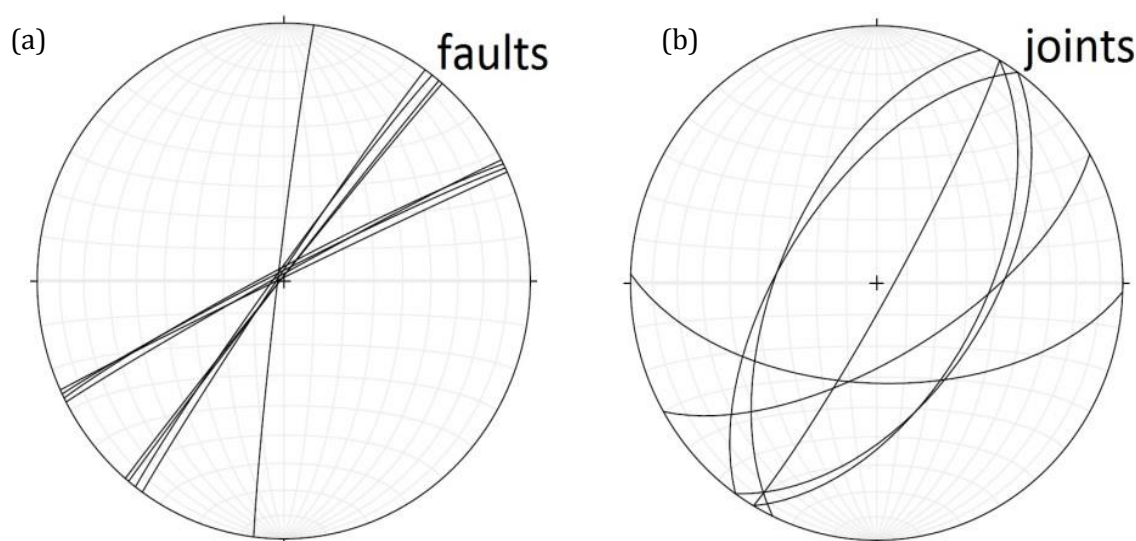


Figure 3. 2: Fault data (a) and joint data (b) visually displayed on two separate stereonet produced by Stereonet version 9.9.4, (2017).

(c) Mine Health and Safety Act No. 29 of 1996 (South Africa, Department of Mineral Resources, 2018)

Mr Willem Welding, Senior Inspector of Mines, at the Department of Mineral Resources (Springbok regional office) was visited to help identify the legislative chapters from the Mine Health and Safety Act and Regulations that are applicable to research. The chapters discussed in Chapter 2.7 of this thesis were used to partially develop the risk matrix that is used to rate the intensity of risks that is created by the glory holes at the reference mines used in this research. Secondly, in Chapter 5 it is also partially used to develop a safety map focusing on the undermined Bruinhoek populated area. The development of the risk matrix and safety map is explained later on in this chapter. Finally, the Mine Health and Safety Act No. 29 of 1996 (South Africa, Department of Mineral Resources, 2018) assists in making recommendations in Chapter 6 which corresponds with the legislation explained in Chapter 2 of this thesis.

3.2.2 Field investigation

(a) Drone aerial survey

The first step in the field investigation is the surveying of reference mines, including Bruinhoek, by means of a drone flying at an altitude of 70 m above ground level. The drone is a DJI Inspire 1 model with a Zenmuse X3 camera attached to it. The areas are photographed in such a way that the glory holes are fully captured. Photos are taken by a camera attached to the bottom of the drone. After the photographs of each area are taken the photos are uploaded to DroneDeploy™ cloud and processing of the photographs is done by DroneDeploy™ that develops the following products available for each area:

- 2D orthomosaic photo
- Elevation data in the form of:
 - i. Digital Elevation Model (DEM)
 - ii. Contour map
 - iii. 2D elevation map with different colours indicating different heights
- 3D topographic model

This research makes use of two products from DroneDeploy™ which is the 2D orthomosaic photos and the 2D elevation maps for each area. The 2D orthomosaic photos are used in ArcMap™ 10.5 for the further production of maps and along with historical Google Earth images to determine how much self-caving of a certain glory hole have taken place over a period in time. The 2D elevation maps are used in Chapter 5, to indicate any depressions existing around glory holes and at the undermined region of the Bruinhoek populated area. These two products created by DroneDeploy™ helps to show the unstable nature of glory holes and supports discussions in Chapter 5. It should be noted that an aerial survey for the Okiep Mine could not be done.

(b) Site survey

Site surveys consist of site walkovers and inspections of the glory holes and its surroundings. The soil thickness, saprolite, fractures and weathering of the more solid rock material are described. While the mines are not accessed by the author as mentioned previously it should be noted that glory holes cannot be accessed as it is not possible because of safety reasons and therefore during the site surveys all of the joint conditions are estimated. Secondly, instabilities are identified in the form of depressions, cracks around the rim of the glory hole, overhanging rims, persistent faulting and jointing, and unstable rock masses spalling from the side walls of the hole. Thirdly, the method of protection and its effectiveness is described.

3.2.3 Risk matrix development

In this study the first two steps of the baseline risk assessment are done which include hazard identification and risk assessment. Recommendations with regards to risk control are made in Chapter 6 while the implementation thereof falls outside the scope of this study.

The glory holes in this study are identified as the hazards because it has the potential to cause harm and in extreme cases, death. According to Paithankar (2011), the risk rating can be calculated as likelihood/probability multiplied by the severity of a hazard. The risk created by these glory holes is assessed by means of a risk matrix that is qualitatively developed. The two functions identified to calculate the intensity of the risks exhibited by each glory hole are hazard severity and probability.

(a) Hazard severity

For the risk matrix to function the severity of the hazard was represented by a numerical range of 1-5, (1) indicating lowest severity of hazard and (5) the greatest severity:

- (1) = A shallow glory hole with stable and shallow slopes and an easy exit.
- (2) = A shallow glory hole with unstable and shallow slopes and a potential for exit.
- (3) = A deep glory hole with some unstable and shallow/steep slopes and a difficult exit.
- (4) = A deep glory hole with steep and unstable slopes and no chance of exit.
- (5) = A deep glory hole with vertical and unstable slopes and no chance of exit.

(b) Hazard probability

The second part of the risk rating equation defined is the probability of someone gaining access to a glory hole. Two factors are combined to define the probability, which were distance of settlements or human activity from a glory hole and effectiveness of protection around a glory hole. A distance of a 100m was generally taken as the threshold distance based on the

regulations from the Mine Health and Safety Act No. 29 of 1996 (South Africa, Department of Mineral Resources, 2018). The hoist cable fences and the concrete palisades are regarded as insufficient forms of protection as cables in fencing are too far apart and palisades are easily damaged. However, gabion walls were regarded as an effective form of protection. For the risk matrix to function the probability was represented by (1) unlikely access and (5) certain access:

- (5) = Located 100 m or closer to human activity with no form of protection.
- (4) = Located 100 m or closer to human activity with ineffective protection (Hoist cable fence or concrete palisade).
- (3) = Located 100 m or closer to human activity with moderate protection (Gabion wall).
- (2) = Located further than 100 m from human activity with ineffective protection (Hoist cable fence or concrete palisade).
- (1) = Located further than 100 m from human activity with moderate protection (Gabion wall).

It should be noted that the risk matrix is developed to calculate the risk at mines where glory holes have formed. In Chapter 4 the risk matrix system is used to calculate the risk created by glory holes at each of the reference mines. In Chapter 5 the risk ratings are compared with the prevailing situation at the Bruinhoek populated area and gives an indication of the risk intensity that could be expected if a glory hole forms in the future at Bruinhoek.. The risk matrix system can be seen in Figure 3.3 indicating high risk as red, moderate risk as yellow and low risk as green. Should a glory hole form at Bruinhoek in the future the same risk matrix can be used to calculate risk intensity.

Table 3. 1: The risk matrix that rates the intensity of risks created glory holes from the reference mines.

		Probability					Risk rating	
		1 rare	2 unlikely	3 possible	4 likely	5 almost certain		
Hazard severity	1 insignificant	1	2	3	4	5	high risk	
	2 minor	2	4	6	8	10	medium risk	
	3 moderate	3	6	9	12	15	low risk	
	4 major	4	8	12	16	20		
	5 catastrophic	5	10	15	20	25		

3.2.4 GIS map production and data manipulation

During the map production process ArcMap™ 10.5 was used.

- The first step in the map production process entailed the importing of 2D orthomosaic photos, using ArcMap™ 10.5, of the reference mines and Bruinhoek. The 2D orthomosaic maps were edited to indicate certain features, including the glory holes, cracks around glory holes etc. The second step was to import plan view survey- and geological maps into ArcMap™ 10.5. Survey maps were also edited, and features such as shaft locations, borehole locations, crown pillars and undermined regions, were indicated. Geological maps were edited too. The different lithologies, mined areas, zones of alteration, zones of kaolinisation, mine tunnels, and zones of structural importance for example jointed and brecciated zones were highlighted as indicated on the maps by the geologists who compiled these maps during the operational years of the mine. The stereonet developed by Stereonet version 9.9.4, (2017) from these historical geological maps, as explained above, were placed on the maps at a specific region of structural importance.
- The 2D orthomosaic maps, the edited survey maps and the edited geological maps were projected according to the GCS Hartbeeshoek 1994, LO 17 projection and this allowed the author to develop edited geological- and survey maps overlying the 2D orthomosaic map of

study areas. These maps served as an important source of information in Chapter 5 as they displayed the relationship between underground features and surface features for example the relationship between the location of glory holes and zones of structural importance (jointed- and brecciated zones). Overlay maps also assisted in determining the outlines of the undermined area at Bruinhoek In Chapter 5 these maps contributed to the discussion as to why glory holes in the Okiep Copper District formed, and if it is likely that a glory hole can form at the undermined Bruinhoek populated area.

- The 2D orthomosaic maps produced by DroneDeploy™ were used along with historical imagery from Google Earth. The rims of the glory holes on the 2D orthomosaic photo were compared with the rims of the same holes from historical Google Earth images. In this way the amount of rim collapse and self-caving of the glory holes at each study area was observed and compared. This information partially contributed to the development of the safety map in Chapter 5 and indicates what can be expected following the formation of a glory hole.
- In Chapter 5 a safety map was produced for the undermined Bruinhoek populated area in Concordia. The safety map was developed so that it had a red zone representing a danger area, a yellow zone representing an unsafe area and a green zone representing a safe area. The first step in developing the map was to outline the undermined area with help from the geological maps of the Wheal Julia East Mine. ArcMap™ 10.5 was used to outline the undermined area on the 2D orthomosaic map produced by DroneDeploy™. The second step was to outline the red zone (danger area) around the undermined area. The regulations from the Mine Health and Safety Act No. 29 of 1996, as discussed in Chapter 2.7, were applied to create the red zone. The third step was to outline the yellow zone. It is based on a conservative amount of rim collapse and self-caving i.e. more than observed at the reference mines seen from historical imagery from Google Earth and the 2D orthomosaic from DroneDeploy™. In the fourth step the green zone was indicated and stretches beyond the yellow zone, and represents the safest area from the undermined zone.

CHAPTER 4: RESULTS

Chapter 4 will consist of four sections that will represent the three reference mines and the Bruinhoek populated area undermined by the Wheal Julia Mine. Each section will display results in the form of the desktop study, which consists of previous reports, and historical survey and geological maps from the O'okiep Copper Company offices. The second part of the results for each section comes from historical imagery from Google Earth and the third part includes the calculation of the risk ratings of each glory hole, using the risk matrix displayed in Table 3.3. It should be noted that not many previous reports were found and it is the same with maps for some of the reference mines. The reports include a general report and an inspection report. Site Plan Consulting (2005) visited all the abandoned mines in the Okiep Copper District to compile a general composite report containing information about each mine. The inspection report was conducted by Spiessens (2015) the senior inspector of mines and focuses specifically on the Bruinhoek populated area. For the Rietberg Mine and O'okiep Mine only survey maps were found, while survey and geological maps were found for Hoits Mine and Wheal Julia Mine. The survey maps indicate the dimensions of the mines and certain underground features relative to surface features, while the geological maps indicates important structural features of the geology below the surface.

4.1 Rietberg Mine

According to Beukes (2016), the Rieberg Mine was in production from 1976-1981 and has therefore been abandoned for at least 38 years. Beukes (2016) also mentions that ore was initially extracted via conventional mining methods at the beginning of the lifetime of the mine; however, sub-level open stoping was implemented shortly after. This mining method extracts large areas of material which can be seen in Figure 4.1 as this vertical section indicates that the area that has been mined has a height of approximately 180m and a width of approximately 200m. According to these dimensions, the Rietberg mine has a vertical mined area of approximately 36 000m². It should be noted that all of the red angled lines, regardless of their slope direction, represent the mined area. Figure 4.1 also indicates that mining activities took place at depths as shallow as 15m below ground level. According to a report by Site Plan Consulting (2005) the Rietberg Mine has two glory holes both being deeper than 200m and filled with water to unknown depths. The report states that the rims of the glory holes are dangerously overhanging the openings, and that the sidewalls are vertical and unstable. According to the report, the rehabilitation involved filling of ventilation shafts and adits with waste rock covered with sand, constructing and a 1.4m high gabion wall that now surrounds the glory hole, aiming to keep people and animals away. The gabion wall can be seen in Figure 4.2. Further caving of these glory holes is highly likely to happen in the future (Site Plan Consulting, 2005). Figure 4.3 shows a historical Google Earth image indicating the outlines of the glory holes in May 2002 as well as outlines of the holes in January 2017 illustrating the growth of the holes over a space of 15 years. Utilising the risk matrix from Table 3.3, the risk intensity of the Rietberg Mine glory holes can be calculated, which is done in

Table B 1.1 in Addendum B. Because these glory holes are remote, the possibility of people coming across them is unlikely however; these glory holes are extremely hazardous as they are very deep, with vertical and unstable sidewalls as mentioned. The holes, therefore, have a low risk rating of 5.

Vertical section of Rietberg Mine

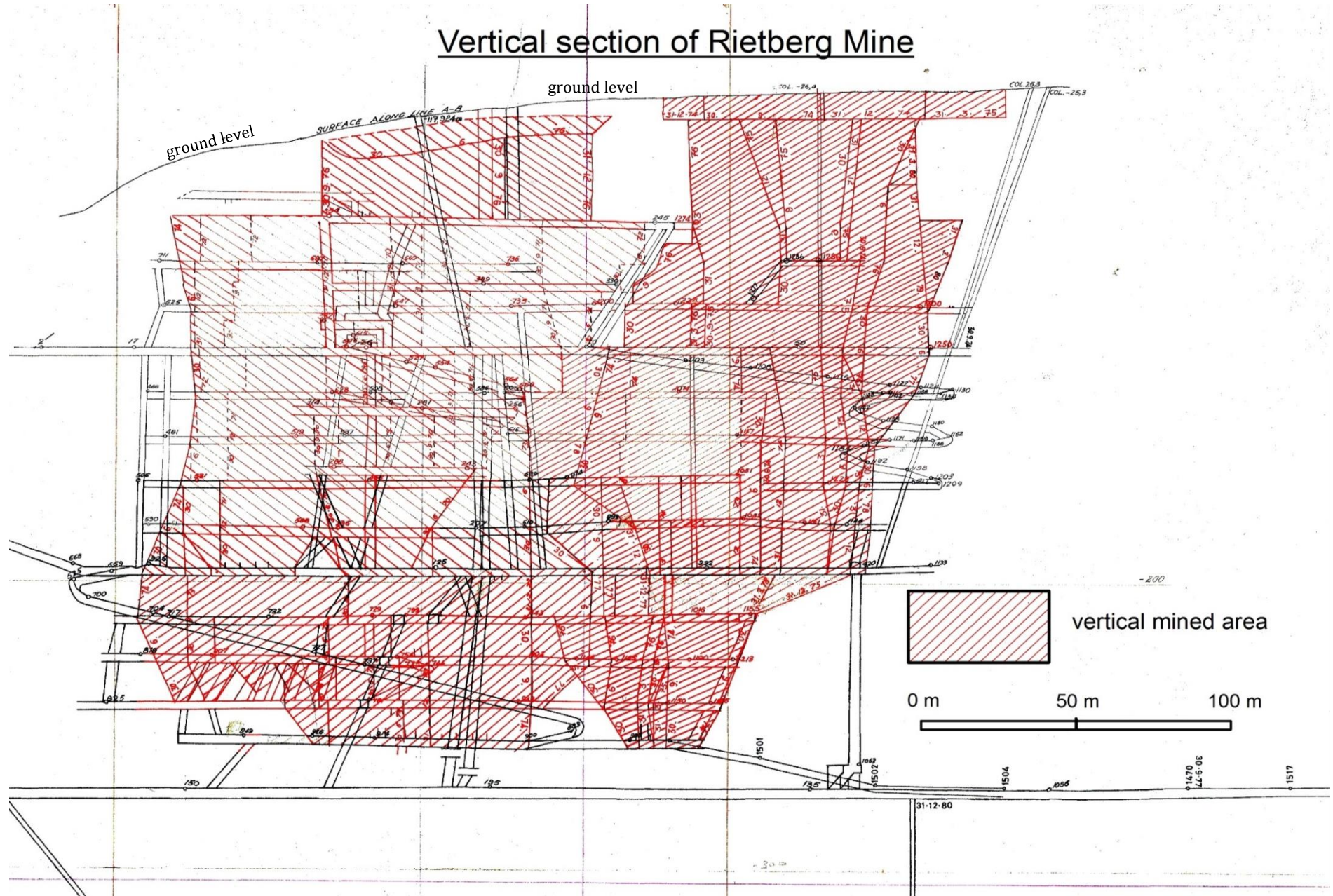


Figure 4. 1: Vertical section of the Rietberg Mine showing the total vertical mined underground area (O'okiep Copper Company, 1976-1981).



Figure 4. 2: The gabian wall surround the Rietberg Mine glory holes.



Figure 4. 3: An aerial view of the Rietberg Mine glory holes showing the extent of their growth from 2002-2017 (modified from Google Earth Pro., (2018b)).

4.2 O'okiep Mine

The town of O'okiep is a historical mining town in the Okiep Copper District. The O'okiep mine is located in the town of O'okiep. For many years copper was extracted from the mine consisting of two parts:

(a) West O'okiep

According to Beukes (2016), this part of the O'okiep Mine is the older of the two. Copper was first discovered and mined in 1855 (Smallberger, 1975). Not many maps were found for West O'okiep. A survey map displayed in Figure 4.4 shows an underground layout with some tunnels, while the most important features of this map are the two shafts located above the underground mined areas. The shafts are indicated as old shaft and no 3 shaft on the map. According to the vertical section in Figure 4.5 mining activities were located approximately 80m below no 3 shaft, while workings took place at slightly shallower depths of ± 40 m below old shaft because of the ore boundary extending upwards in this area. Figure 4.5 also shows that the underground mined area is large, having a vertical length of approximately 250 m and an average height of at least 100 m. According to Site Plan Consulting, (2005) the glory hole located at West O'okiep is approximately 150 m deep and filled with copper-contaminated water; the main reason for the origin of the hole being the collapse of the old shaft. A 2.5m concrete palisade is located around the entire glory hole serving as a protection mechanism to prevent human and animal activity near the hole Figure 4.6. Comparing historical Google Earth images from May 2002 and March 2017 caving of the rims of the glory hole has taken place although very little is seen in Figure 4.7. It is notable that in 2002 the hole was only partially filled with water, whereas in 2017 it is filled completely. This hole obtains a risk rating of 12 (high risk) as it a deep, water-filled hole with some steep and unstable rims and it is located closer than a 100m from human activity with an ineffective concrete palisade as protection. The calculation for the risk rating can be seen in Table B-1.1 in Addendum B.

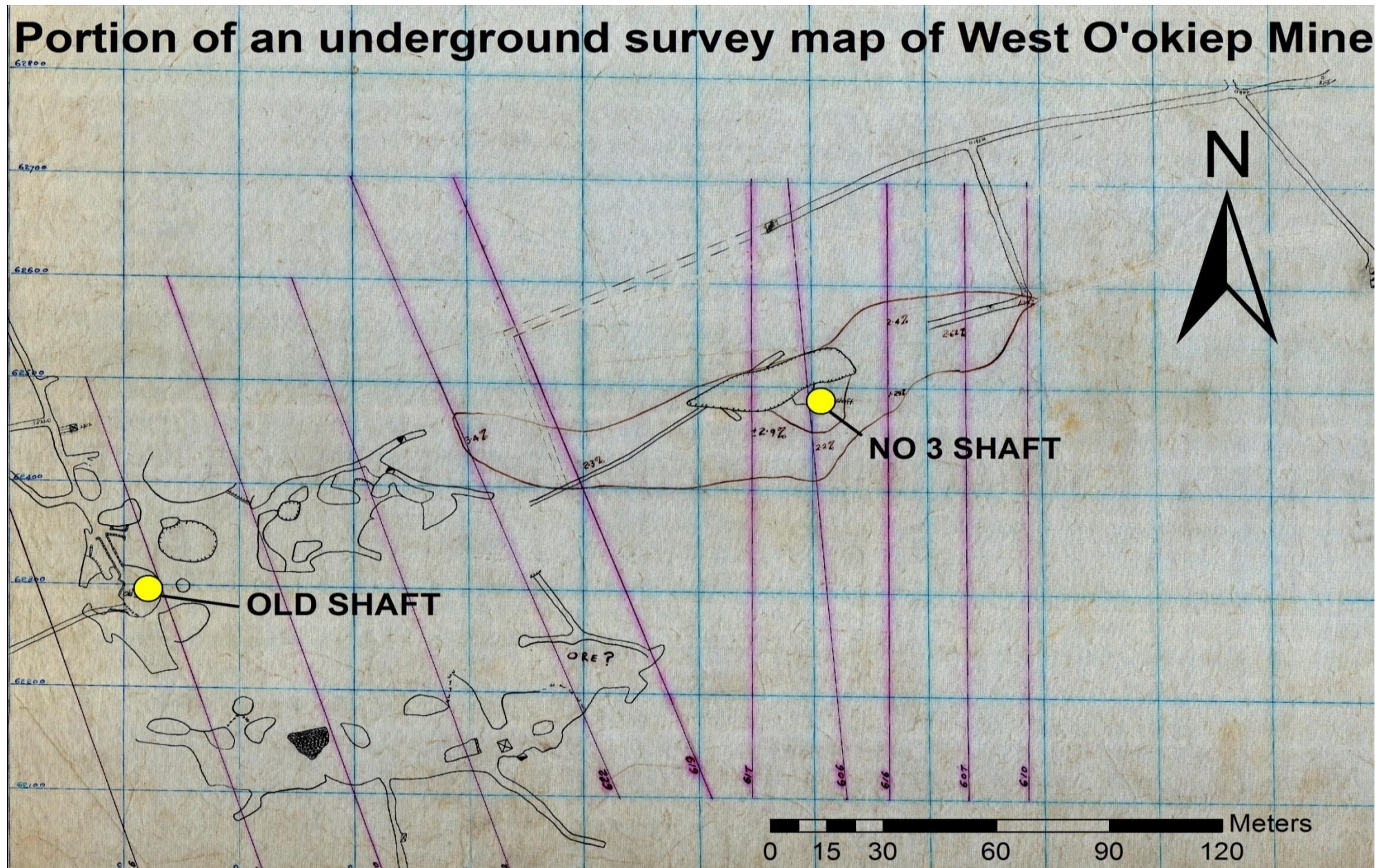


Figure 4. 4: Underground survey map of a portion of the West O'okiep Mine (O'okiep Copper Company, 1942-1975a).

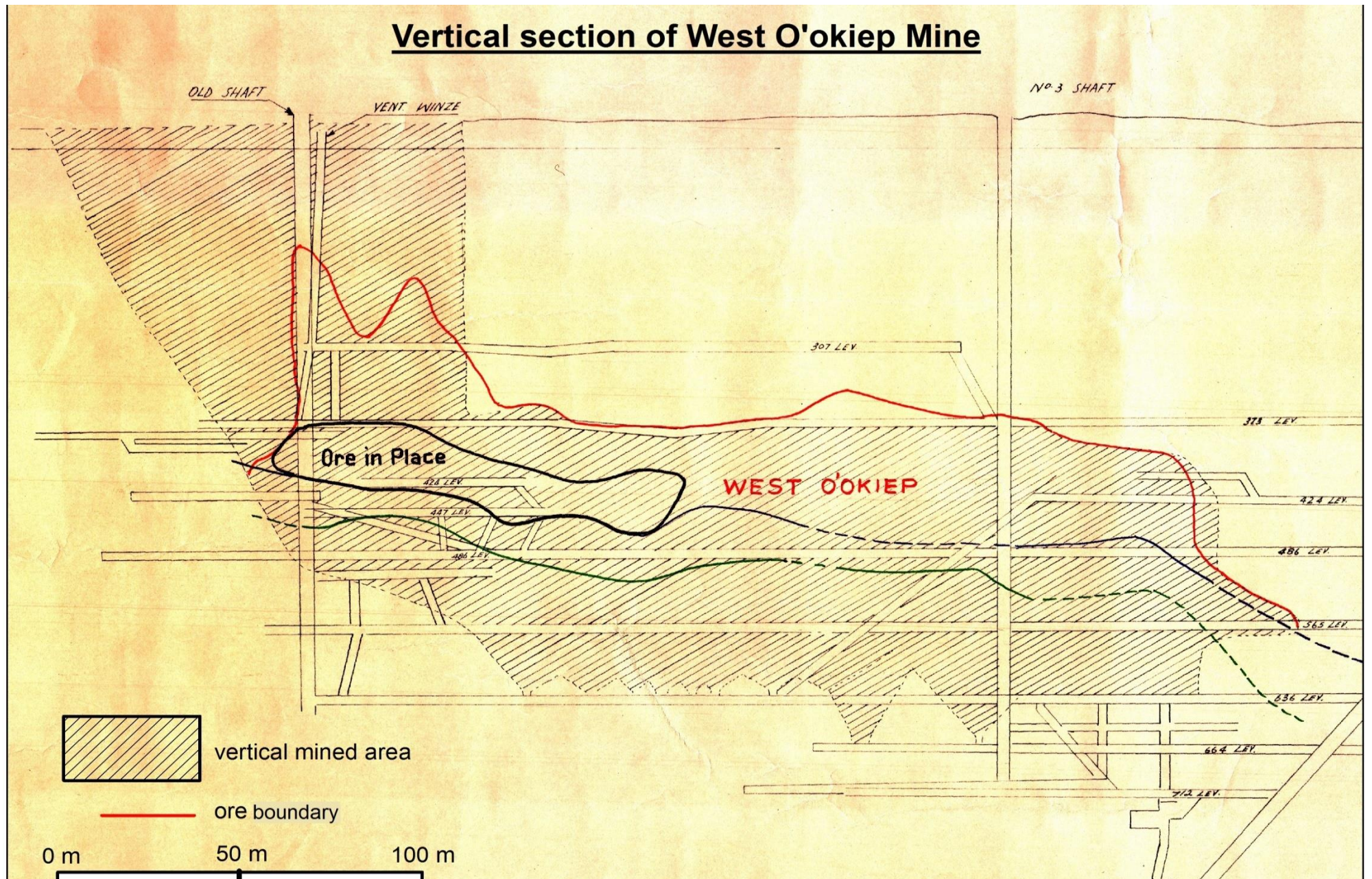


Figure 4. 5: Vertical longitudinal section of the West O'okiep Mine (O'okiep Copper Company, 1942-1975b).



Figure 4. 6: The 2.5m high concrete palisade surrounding the West O'Okiep Mine glory hole.



Figure 4. 7: An aerial view of the West O'okiep Mine glory hole and the extent of its growth from 2002-2017 (modified from Google Earth Pro. (2018c)).

(a) East O'okiep

This part of the mine is the younger part of the O'okiep Mine. Figure 4.8 shows a portion of the East O'okiep Mine in plan view, with the striped and orange-shaded area highlighting a portion of the total underground mined area. The map indicates several surface features such as structures, roads, boreholes and a caved zone. The caved zone is located in a portion of 9 Stope and in the same area where a road pillar has been left below a section of the private road. The function of the road pillar was to provide stability at ground level. A Google Earth image from May 2002 clearly shows the opening of the caved zone while an image from March 2017 indicates that the area has been filled up, as it is not visible anymore. Two more roads located in the western part of the map are also located in the orange area which means they have also been undermined; however, no indication of caving is indicated at these roads. Other surface structures such as the post office and hotel are not located in the orange shaded area indicating that the extraction of copper ore did not take place directly below them; however, they are undermined by mine tunnels. According to Site Plan Consulting (2005), undermining of structures, such as dwellings and the hotel, may cause cracking and/or the collapse of these structures in the future. In Figure 4.9 a vertical section of approximately the same portion of the mine as seen in Figure 4.8 can be seen, showing that mining activities took place at shallow depths that ranged between 15m and 40m below ground level. It is indicated that the underground cavity, at least 700m long (O'okiep Copper Company, 1942-1975c), with heights ranging between 35m and 80m, have been backfilled with sand distributed via the boreholes indicated on Figure 4.9. From Figure 4.8 it can be seen that extensive horizontal mining took place to exploit the ore body and, although the underground cavity was backfilled and pillars were left to provide stability at surface level, a glory still nevertheless formed. Although the glory hole has a shallower depth of approximately 4 m-5 m with unstable and shallow slopes it is located, like the West O'okiep glory hole, less than a 100m from of frequent human activity (Site Plan Consulting, 2005) and it is also not protected in any way. Therefore the East O'okiep glory hole has a medium risk rating of 10 as seen in Table B-1.1 in Addendum B.

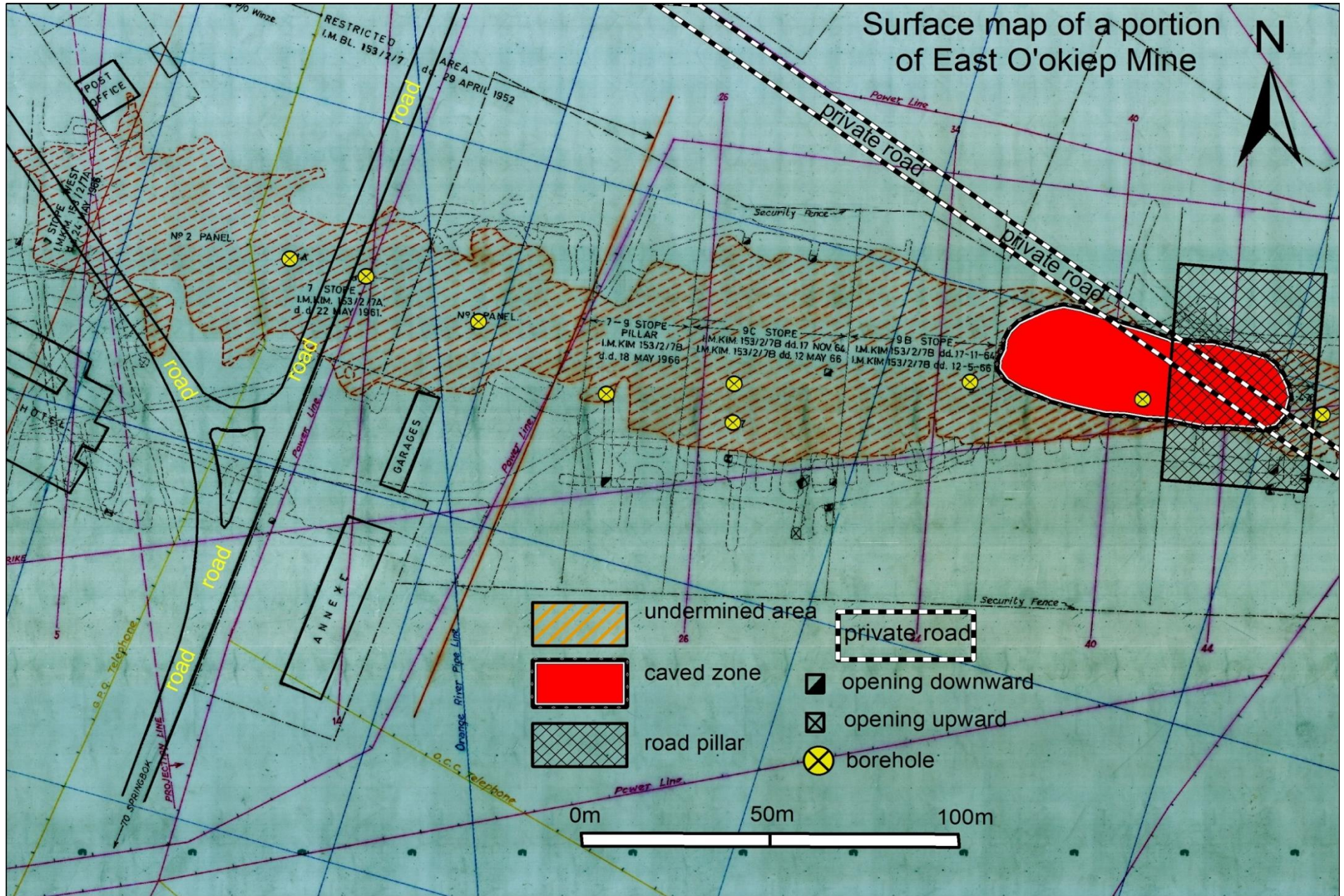


Figure 4. 8: A plan view map displaying a portion of the East O'okiep Mine (O'okiep Copper Company, 1942-1975c).

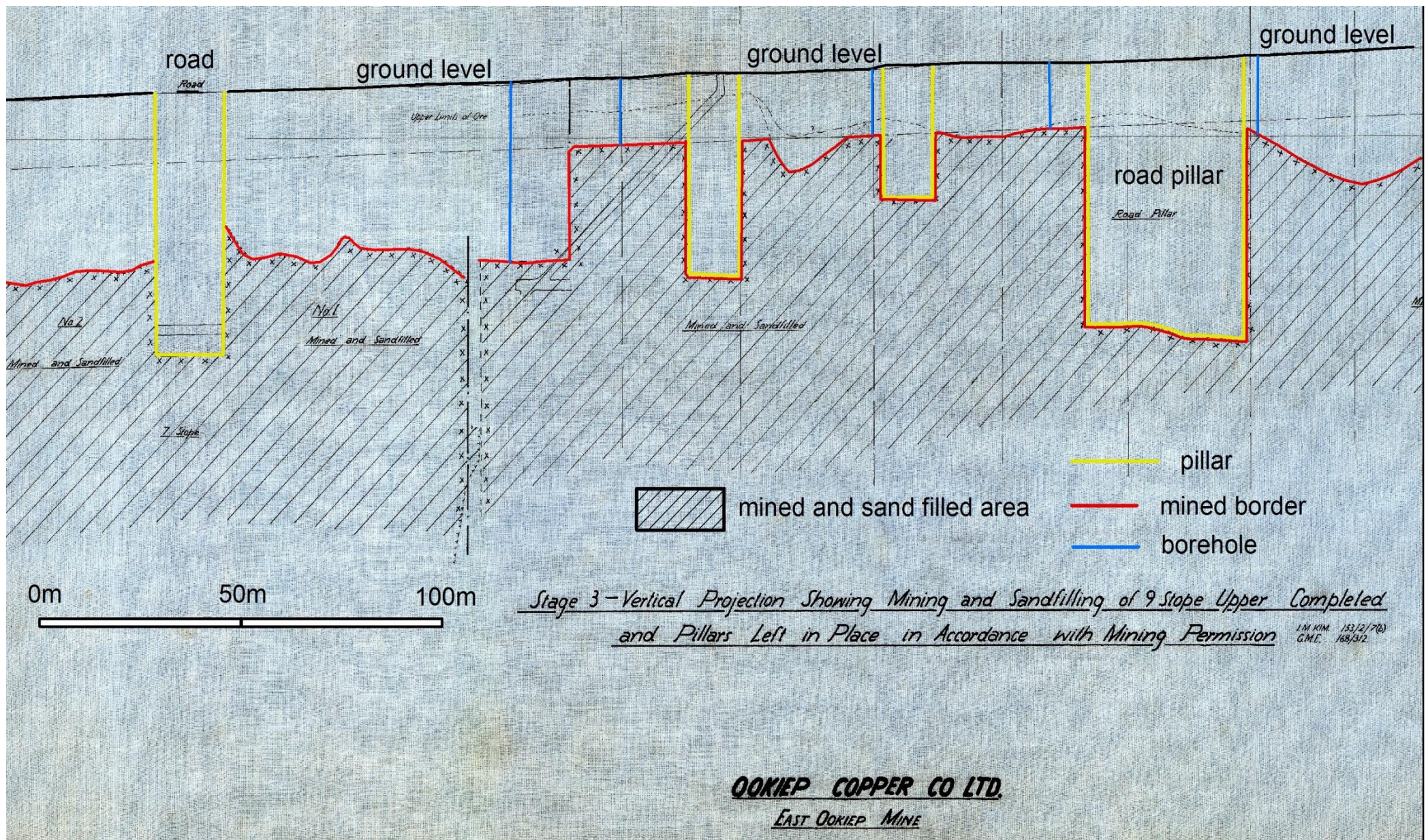


Figure 4. 9: Vertical section of a portion of the East O'okiep Mine (O'okiep Copper Company, 1942-1975d).

4.3 Hoits Mine

This mine operated from 1975-1993 and has been abandoned for at least 25 years (Beukes, 2016). A plan view map of the layout of surface structures and features such as old mine buildings, the decline shaft, the vertical shaft, ventilation shafts and a projection of the orebody can all be seen in Figure 4.10. It should be noted that some of the old mine structures have now been renovated as a farm house. According to Site Plan Consulting (2005), the vertical shaft and ventilation shafts have been capped, while the decline shaft was filled with waste rock to an unknown depth and topped with soiled. A vertical section of the mine is displayed in Figure 4.11, and indicates that the mine had a total of 9 stopes covering a large vertical area with 1 Stope-3 Stope covering approximately 12800m². It is indicated that only 4 Stope-6 Stope are backfilled via boreholes (fill holes), while 3 Stope is partially filled with material from an ex waste-dump. Mining activities at 1 Stope to 4 Stope took place at 55m below ground level with the respective stopes reaching depths of about 270m below ground level. The remaining stopes, 5 Stope-9 Stope, were mined at greater depths. According to Site Plan Consulting (2005), a 100m deep glory hole is located less than a 100m north from the renovated mine buildings. This glory hole is indicated on Figure 4.11, where it is situated above 1 Stope and 2 Stope. It should be noted that the glory hole might be deeper than 100 m, as 1 Stope reached depths of 160m and 2 Stope 190m, below ground level. The overburden between the underground cavity and ground level collapsed to form the glory hole at the surface. According to Spiessens (2015) the sidewalls and rims of the glory hole are unstable, as rim collapse has been observed in the past. Historical images from May 2003 and January 2015 visually indicate the unstable nature of the side walls and rims, through the expansion of the glory hole seen in Figure 4.12. In 2003 it measured 45m from east to west and in 2015 it measured 60m. The glory hole has a safety fence surrounding the rims of the hole made up of steel pipes and hoist cables seen in Figure 4.13 (Spiessens, 2015). A high risk rating of 20 has been obtained for this glory hole, as it has vertical and unstable sidewalls located 100m and closer to human activity, and with ineffective protection. The risk rating calculation can be seen in Table B-1.1 in Addendum B. According to Site Plan Consulting (2005), more glory holes are likely to occur at this mine in the future.

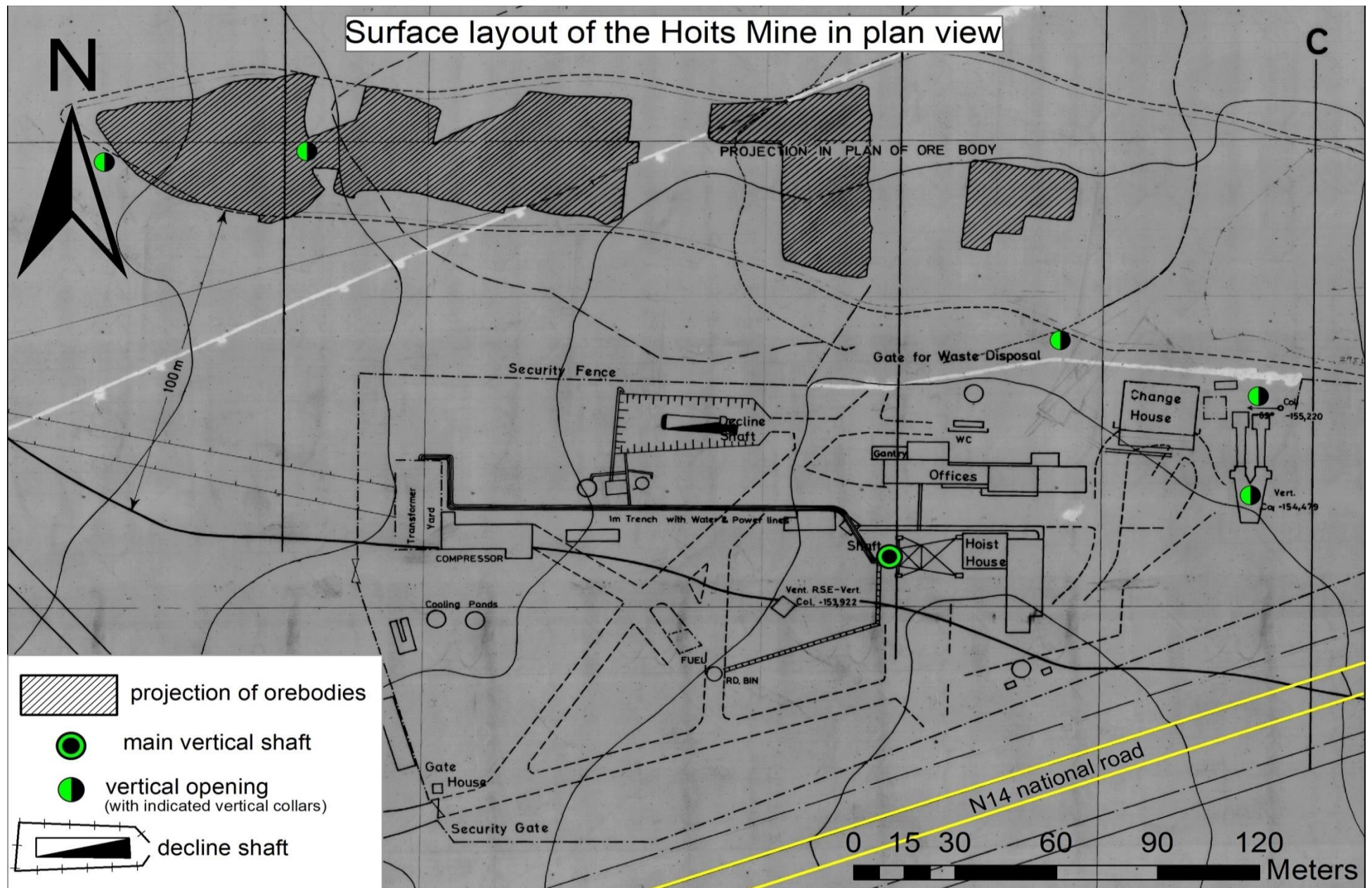


Figure 4. 10: A plan view of the Hoits Mine displaying its different surface features (O'okiep Copper Company, 1975-1993a).

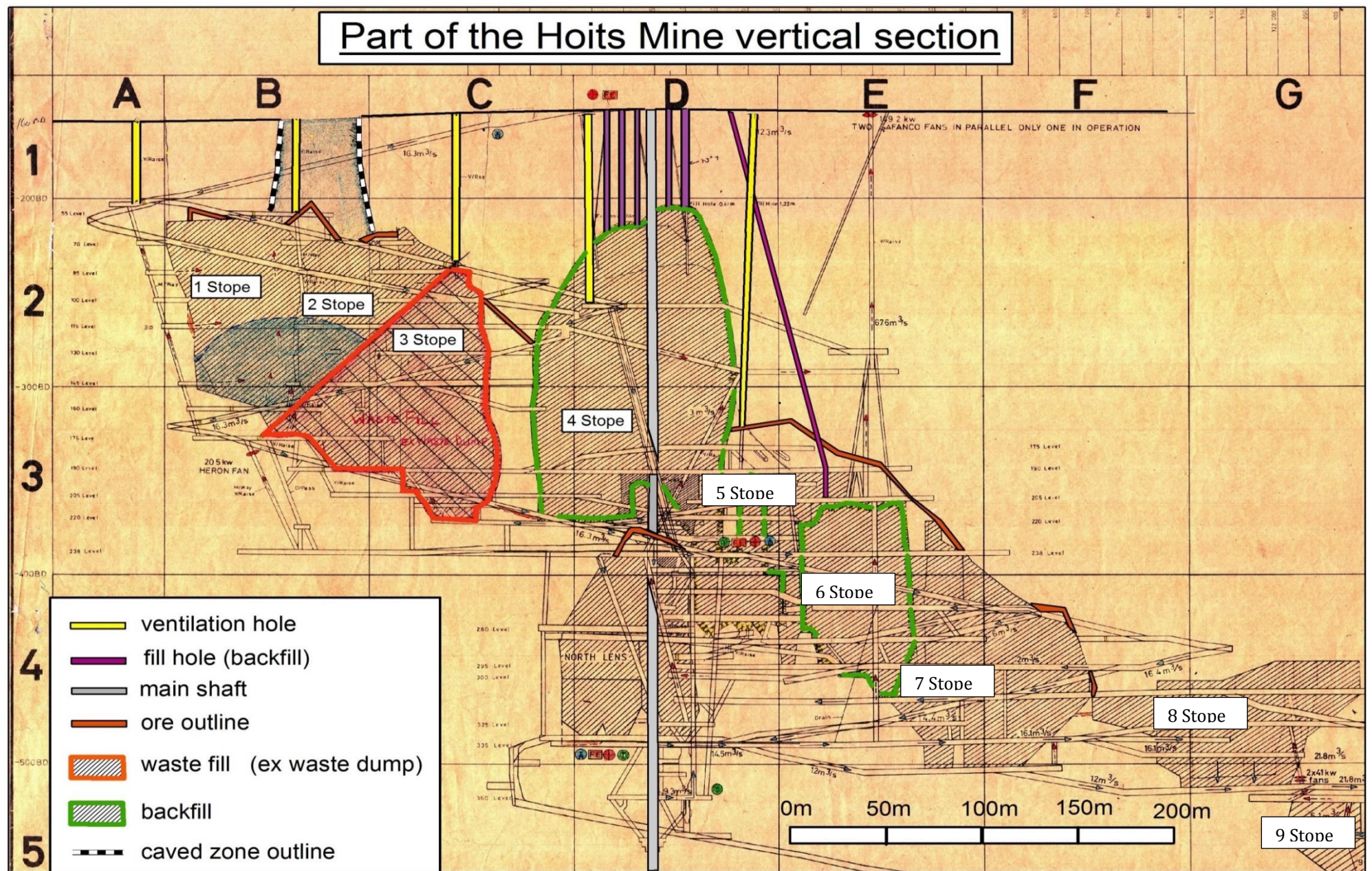


Figure 4. 11: A vertical section of part of the Hoits Mine (O'okiep Copper Company, 1975-1993b).



Figure 4. 12: An aerial view of the Hoits Mine glory hole and the extent of its growth from 2002-2017 (Google Earth, 2018).



Figure 4. 13: The safety fence surrounding the Hoits Mine glory hole.

A total of nine geological maps were found and for the purpose of this thesis five of these maps, those showing the 55 m level to the 115 m level, will be reviewed and discussed. Zones of structural importance will also be identified. It should be noted that the geologist that compiled these maps made comments on the maps with regard to fault and joint fillings, alterations and the degree of jointing and faulting and that these comments are highlighted on the maps by the author. It is also important to note that no quantification of the degree of jointing and faulting is available, but only the comments made on the maps by the geologist. Displacement and the direction along shear faults are also not indicated as these movements are minor (Kisters *et al.*, 1996). Stereonets will also be compiled from the joint and fault data gathered from the maps to indicate structural relationships between these features. The remaining geological maps, 130m level-190m level, are not discussed in depth and the maps are included in Addendum C.

55 m level

Figure 4.14 represents the geological map of the 55 m level (uppermost level) of the Hoits Mine. This level is composed mainly of the Nababeep Granite-gneiss, diorite and minor patches of norite. Although not large amounts of ore were extracted from this level, various areas comprising important structural zones occur throughout the level.

- Zone 1: Highly jointed Nababeep Granite-gneiss with kaolinisation at joint planes. The strikes of the joints displayed on the stereonet indicates that two joint sets are present forming a steeply dipping conjugate set. Among the joints is a steeply dipping strike slip fault mimicking the strike (NE) of one of the joint sets.
- Zone 2: Similar to zone 1, this zone is a highly jointed area where kaolinisation occurs at joint planes, and is located mainly in the Nababeep Granite-gneiss. A stereonet indicates one major steeply dipping joint set. Among the joints are two steeply dipping strike slip faults intersecting each other as well as the joint set.
- Zone 3: Located in a diorite outcrop this zone consists of an area of a moderately concentrated steeply dipping conjugate set of joints, as well as a steep dipping breccia fault. It is notable that the strike of the breccia fault correlates with the faults in zone 1 and zone 2 as indicated by the stereonets. Chloritisation is present between the joints and the breccia fault of the diorite outcrop.

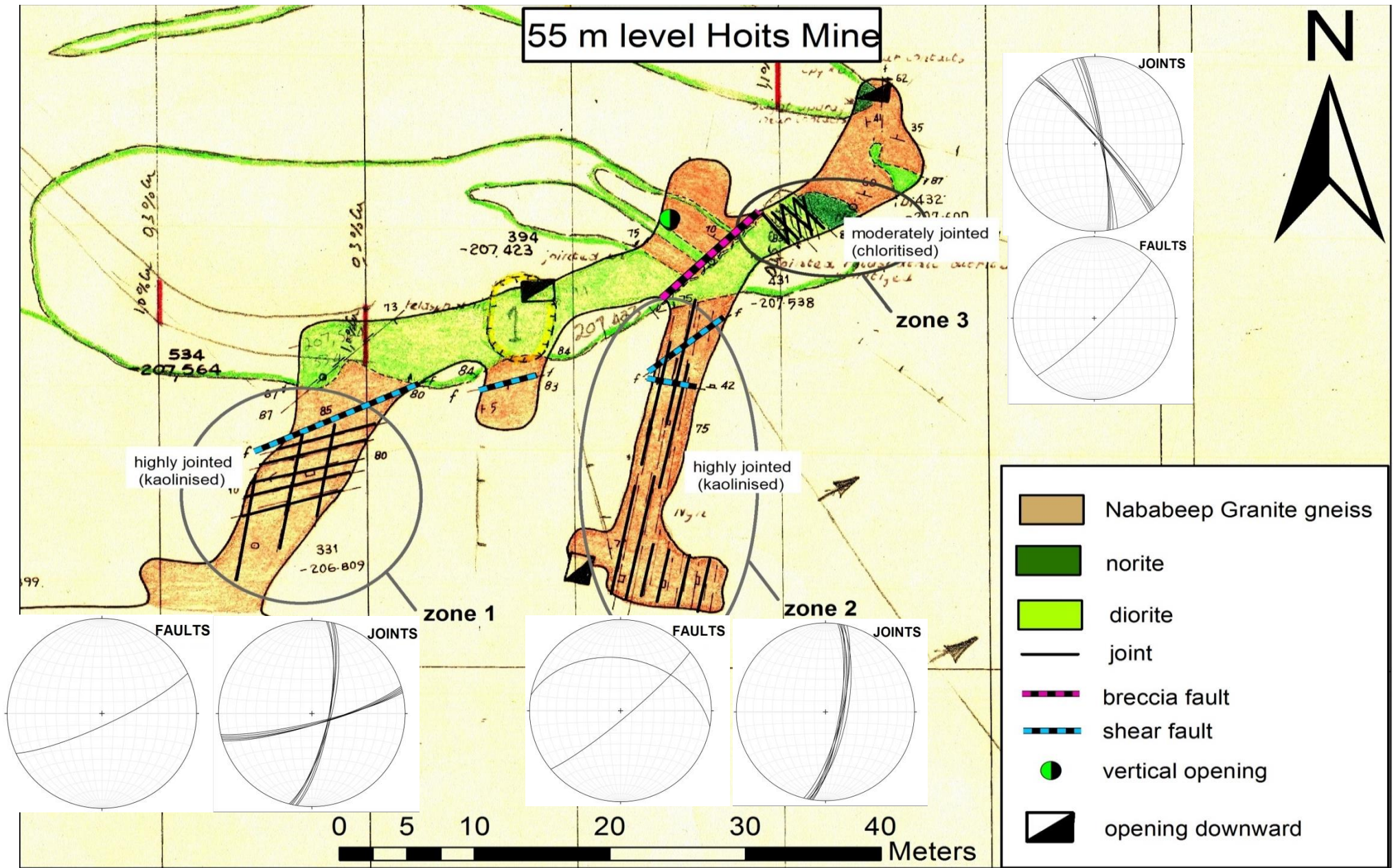


Figure 4. 14: 55m level map of the Hoits Mine (O'okiep Copper Company, 1975-1993c).

70 m level

This level is composed of the NababEEP Granite-gneiss and noritic outcrops seen in Figure 4.15. Joints and breccia faults are randomly located throughout the level while there are identified zones of weakness and structural importance.

- Zone 1: This is a highly jointed zone with northerly orientated strikes having mostly steep dips in an easterly direction as seen on the zone 1 stereonet. Steeply dipping breccia faults are also present in this zone and mimic the strikes of the joints. This zone represents the tunnel that breaks away from the decline shaft.
- Zone 2: This zone is of structural importance as it represents an area of breccia faulting and intense jointing. Brecciation occurs in both the norite and NababEEP Granite-gneiss, while the degree of brecciation and jointing in the NababEEP Granite-gneiss is more intense than that in the noritic rocks. The zone 2 stereonets indicate that two joint sets, the N and NE orientations, are present and that the breccia faults have a preferred NE strike, slightly crosscutting some of the joints. According to the stereonets, both jointing and breccia faulting have steep dips, with one joint set slightly mimicking the faults. While jointing is more localised in the NababEEP Granite-gneiss, as mentioned the breccia faults occur randomly throughout the NababEEP granite-gneiss and norites. Chloritisation of norites and kaolinisation of the NababEEP Granite-gneiss occur in the jointed and brecciated zones. Two major fault structures are also indicated on the map cutting through the entire zone, and these correlate with the orientations of the breccia faults. It seems as if the two faults meet up with each other in the middle of the zone and join to form one major fault structure. This area can be described as a fault zone, as it consists of numerous closely spaced fault surfaces separating masses of broken rock (Davis *et al.*, 2011). According to Davis *et al.*, (2011), a single fault zone can also thicken or become thinner along a strike, as individual fault strands merge or split away from the zone, as can be seen from Figure 4.15.
- Zone 3: A jointed and brecciated area with steep dips located in the NababEEP Granite-gneiss and noritic rocks. Although the distribution of faults and joints is less intense in this zone, two joint sets can be noticed, with the NE preferred orientation of the faults correlating with one of the joint sets and crosscutting the other.

85 m level

The main lithologies of this level are the Nababeep Granite-gneiss and noritic rocks. The distribution of joints and breccia faults at this level is random throughout, with the exception of Zone 1.

Zone 1: A highly brecciated and jointed zone, as seen in Figure 4.16. Joints and breccia faults occur in both the Nababeep Granite-gneiss and the norites. The breccia faults and joints in both are steeply dipping and have preferred NE strikes, as may be seen in the stereonet, while there is an indication of a single breccia fault having a NW orientation. A major fault structure is also indicated and, similar to what occurs in the 70 m level, it cuts through the entire zone, mimicking the strikes of the joints and breccia faults. This area is also a fault zone, as it consists of closely spaced fault surfaces.

The two areas of 'timber' on Figure 4.16 indicate zones where rock mass was reinstated, with wooden support structures.

100 m level

This level can be seen on Figure 4.17 and is made up of the Nababeep Granite-gneiss and noritic outcrop. Like the 85 m level, it exhibits randomly distributed, steeply dipping and NE orientated joints with a moderate concentration throughout. Breccia faults are present too but are more concentrated at Zone 1.

Zone 1: This area is highly brecciated and represents the Nababeep Granite-gneiss and noritic rocks. The breccia faults have steep dips and are orientated in a NE direction, while the steeply dipping joints mimic the strikes of the breccia faults. This can be seen from the stereonet data. A shear fault is also located just west of this zone, with a strike in an easterly direction. Two major fault structures are indicated, and these have similar orientations to those of the breccia faults and joints. The fault structure starts off as a single fault and splits up into two as it goes through the area. Like the major fault structures from previous levels, these fault structures seem to cut through the entire zone. Similar to the 70 m level, this area is also a fault zone, as it is made up of numerous closely spaced fault surfaces.

Another area of a section of tunnel reinforced by timber can be seen just east of Zone 1 as indicated in Figure 4.17.

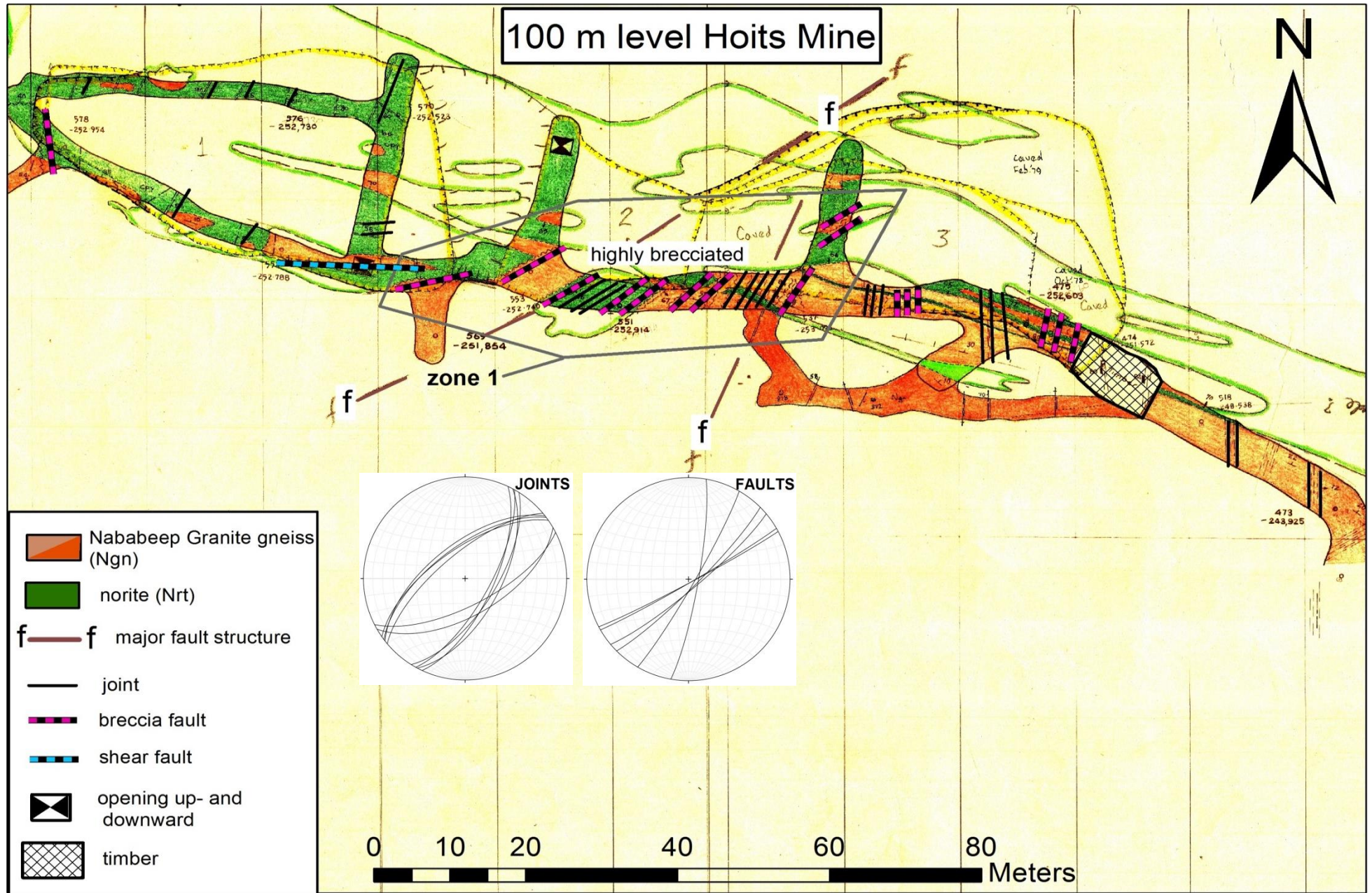
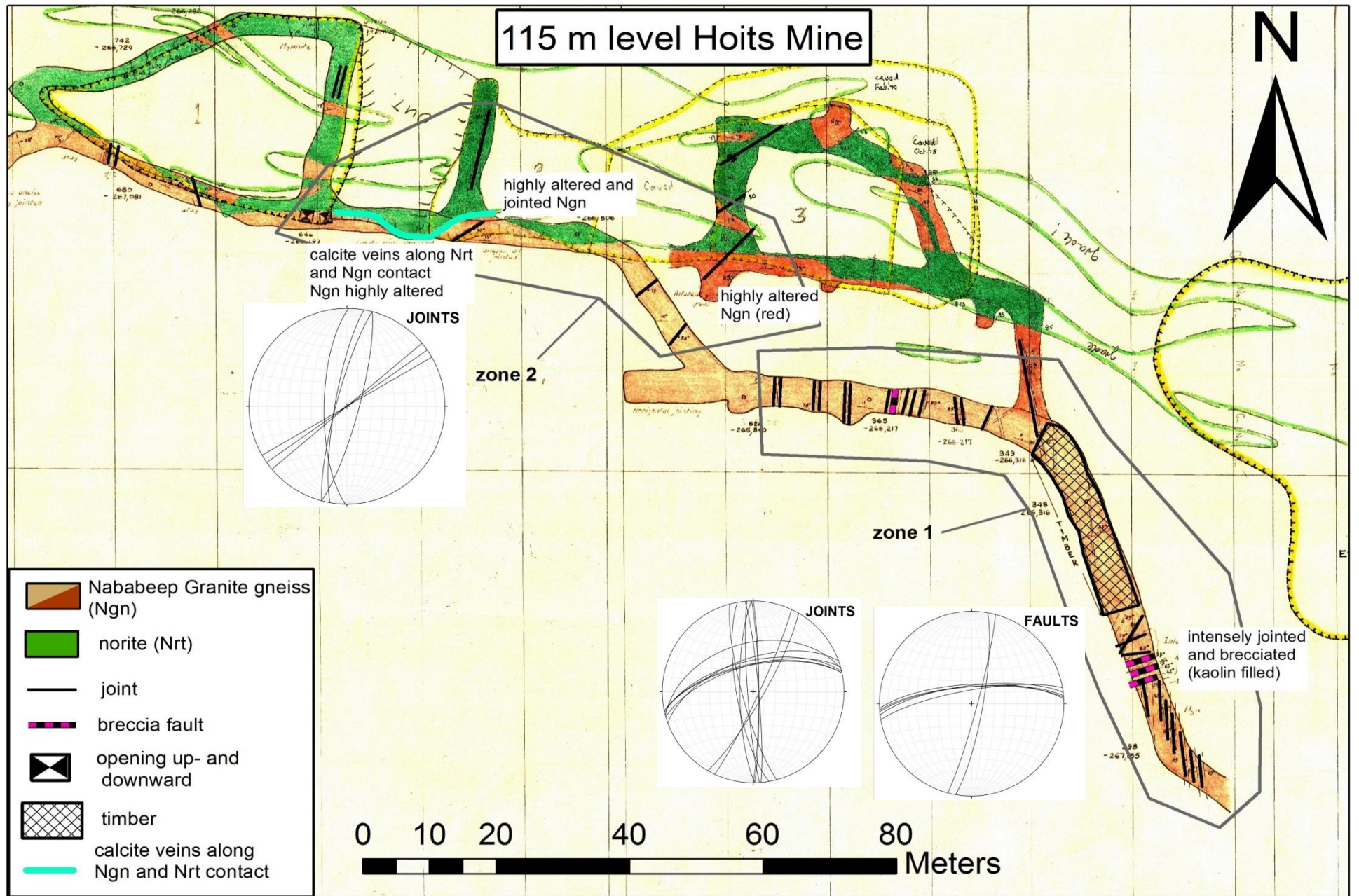


Figure 4. 17: 100m level map of the Hoits Mine (O'okiep Copper Company, 1975-1993f).

115 m level

Like previous levels, NababEEP Granite-gneiss and noritic rocks comprise the entire level. From Figure 4.18 it can be seen that the area of mining has increased in area comparing it to those from shallower levels. A moderate concentration of steeply dipping joints is present throughout the 115 m level, with strikes varying between N and NE. According to the geological map displayed in Figure 4.18, not many breccia faults are indicated in this area. There are two zones of structural importance at this level.

- Zone 1: Representing a zone composed of NababEEP Granite-gneiss that is intensely jointed with areas of kaolinised brecciation. A large tunnel area has also been reinstated, with timber for support purposes. According to the stereonet for Zone 1, there is one main steeply dipping joint set with an N orientation. Brecciation crosscuts some of the joints with an E strike.
- Zone 2: Composed of both NababEEP Granite-gneiss and norite, this zone has calcite veins along some of the contacts of the two lithologies, as indicated on Figure 4.18. The gneiss, specifically, has a reddish colour because of alteration. It is also highly jointed and steeply dipping, with a preferred NE strike, as indicated by stereonets.



In general, each area where a high concentration of breccia faults and joints occur was identified. More often than not, these discontinuities are accompanied by chloritisation of noritic rocks and kaolinisation of the Nababeep Granite-gneiss. The overall indication of stereonet is that both breccia faults and joints are sub vertical displaying steep dips. Breccia faults and joints have a preferred strike of NE, but can occasionally have a slight N orientation as well. Discontinuities with N orientations occur randomly throughout the geological maps and can crosscut NE orientated discontinuities when they occur in the same area as seen, for example, on the Zone 2 stereonet from 70m level. According to strike data from the stereonet, in areas of highly brecciated zones the orientations of joints mimic those of the faults. It is often indicated by the geologist that these zones form part of a major fault structure, for example, at 70m, 85m and 100m level. According to Davis *et al.*, (2011), an area consisting of a zone of closely spaced fault surfaces can be identified as a fault zone. It is noticeable that, while very few shear faults are indicated from the 55 m level to the 115 m level the levels below the 115 m level have more shear faults, according to the geological maps. Like the breccia faults, the shear faults are also steeply dipping with a preferred NE strike. In the geological maps the direction of displacement and the amount of displacement is not indicated, while Kisters (1993) states in his thesis that the NE trending shear zones of the Okiep Copper District display sinistral displacement. Some of these shear zones also form part of the fault zones mentioned previously. Finally, the contact zones between the noritic rocks and the Nababeep Granite-gneiss occur randomly throughout each level, with the contact angle between the two lithologies generally being steep. At some levels, for example 115 m level and 130 m level, the presence of calcite veins, located at the contact zones between the noritic rocks and Nababeep Granite-gneiss, has been noted by the geologist. These veins resulted from mineral precipitation during hydrothermal activity between contact zones and small openings in the rock mass. The presence of hydrothermal activity also explains why alteration features such as kaolinisation and chloritisation are found in many areas of fault weakened and jointed rock mass. The movement of hydrothermal fluids through these weakened rock masses result in the alteration of fault and joint planes causing the formation of secondary minerals. Further discussions on kaolinisation, chloritisation and secondary minerals will be done in Chapter 5.

4.4 Wheal Julia Mine

According to Spiessens (2015), stoping operations were stopped in October 1959, indicating that this mine has been abandoned for nearly 60 years. It comprises three parts, which are Wheal Julia West, Central and East. The main shaft was located at Wheal Julia Central, while Wheal Julia West and East were accessed via underground tunnels from Wheal Julia Central. Wheal Julia East also had a smaller shaft, which is locally known as the 'Curry Shaft'. Spiessens (2015) states that the stopes of this mine were likely to have been mined by the Vertical Crater Retreat (VCR) mining method, which can relate in some way to sublevel open stoping. The VCR method blasts away large sections of the orebody and stopes are not always backfilled (Eklind *et al.*, 2007) which is similar to sublevel open stoping as seen in Chapter 2. According to Spiessens (2015), houses, dwellings and plots of the Bruinhoek populated

area are located above zones which are undermined by Wheal Julia East. This essentially means that an underground cavity that is not backfilled is located below this populated zone. In Figure 4.19 a vertical section through Wheal Julia East can be seen, indicating the different levels of, the mine and, more importantly, the vertical mined area. According to Spiessens (2015), the surface dimensions of the undermined area in plan view are 80m in length and 60m in width and according to Figure 4.19, the depth of the mine is approximately 150m, while the overburden separating the mined zone and ground level is approximately 70m thick. According to these dimensions, the Wheal Julia East Mine has an approximate vertical mined area in the region of 12 000m². It should be noted that the shallowest level of the Wheal Julia East Mine, 'Old Cousin Jack level', is located 60m below the surface, but seems to be only a tunnel where no copper extraction took place, according the Wheal Julia East geological maps. Spiessens (2015) states that rehabilitation of the Wheal Julia Mine was done only in 1999, although it is not clear what the rehabilitation entailed, and whether it was done properly. It is only known that the Main Shaft, 'Curry Shaft' located at the Bruinhoek, and the ventilation openings were capped with concrete blocks. Spiessens (2015) states in his report that if the rehabilitation of stopes were not done effectively there is a chance that caving of the overburden might take place that could eventually cause the formation of a glory hole at surface level. Spiessens (2015) and Site Plan Consulting (2005) recommend that undermined houses and dwellings in close proximity to the undermined zone should be evacuated immediately, as they might be cracked or damaged by the process of subsidence. In extreme cases, houses and dwellings might even disappear when a glory hole is formed and lives may be lost.

Vertical section of the Wheal Julia East Mine

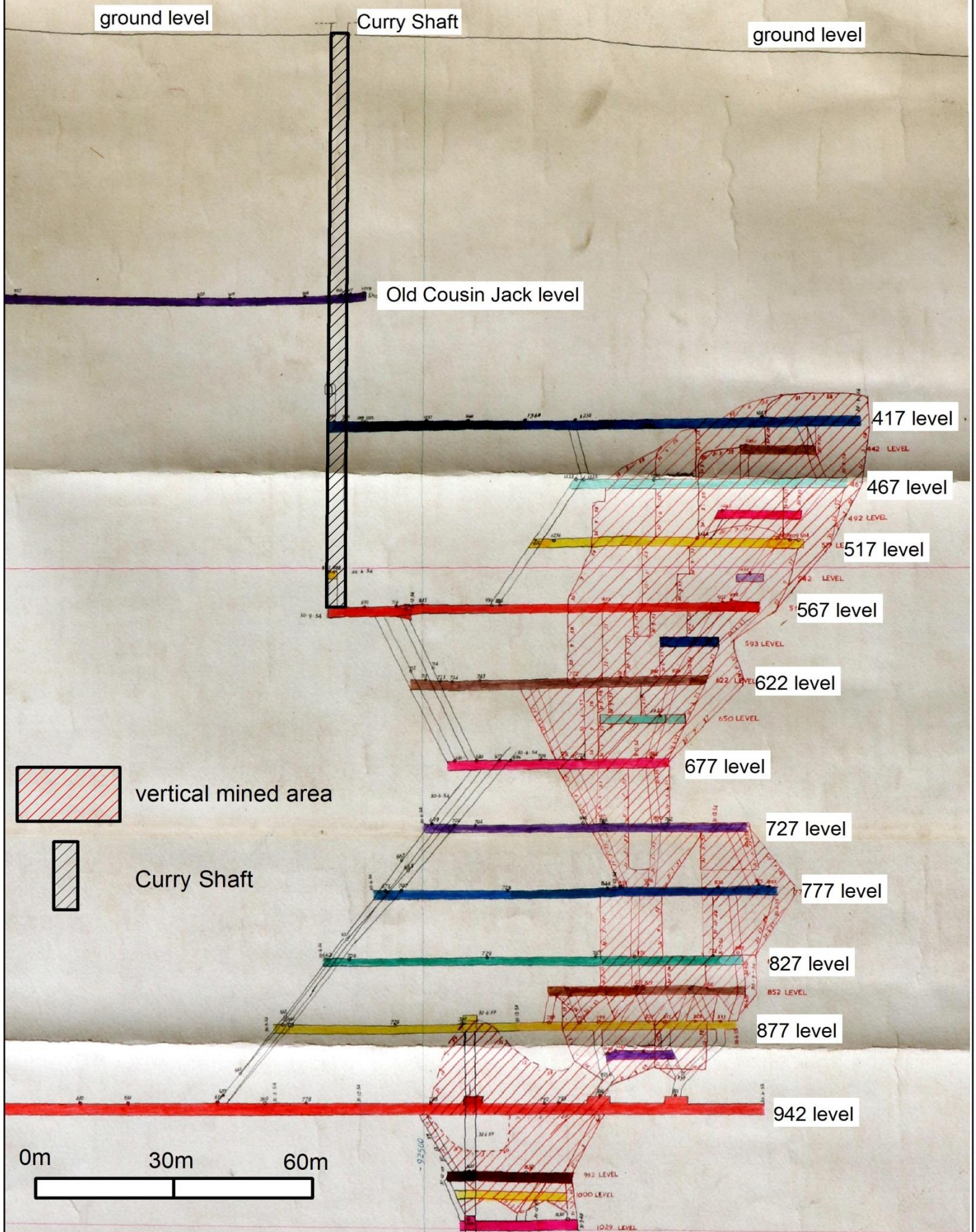


Figure 4. 19: Vertical section of Wheal Julia East Mine (O'okiep Copper Company, 1955-1959m).

A total of 12 geological maps for Wheal Julia East were found. It must be noted that these maps are at least 60 years old and differ considerably from the geological maps found for Hoits Mine. Colours, symbols and notes made by the geologists who compiled these maps have become dull over the years, making it difficult to see what the maps display. The maps indicate different rock types, tunnels, and the area where ore was extracted at each level; however, very few structural data are available on these maps. The geologists who compiled these maps focused on mineralogical characteristics and not so much on the structural features of the rock mass. Therefore, the author was not able to recover the same type and amount of information as was possible with the geological maps from Hoits Mine. In Figure 4.20 an example of a geological map from Wheal Julia East, the 467 level, can be seen, while the rest of these maps are included in Addendum C. This map is one of the few maps where a structural feature could be found, in the dioritic rocks that most likely represent a form of jointing with NE strikes. Another structural feature is the NE orientated 'calcite slip'. This feature is not explained on the geological maps and it can therefore be assumed that the 'calcite slip' represents a fault that has been filled by calcite by means of hydrothermal activity. Other structural information can be seen on the 877 level, where contorted NababEEP Granite -gneiss is indicated and on 942 level where a 'minor' breccia fault is noted. On two separate levels chloritisation was also indicated. Similar to Hoits Mine, the host rock for the copper mineralisation at Wheal Julia East is located in, and surrounded by, the NababEEP Granite-gneiss. According to the geological maps the host rock for copper mineralisation is diorite with only a few patches of norite present at some levels. Other information indicated on this map as well as the other geological maps that is irrelevant to this research includes the concentration of copper in some rocks and mineralogical descriptions of the rock mass for example areas of disseminated chalcopyrite.

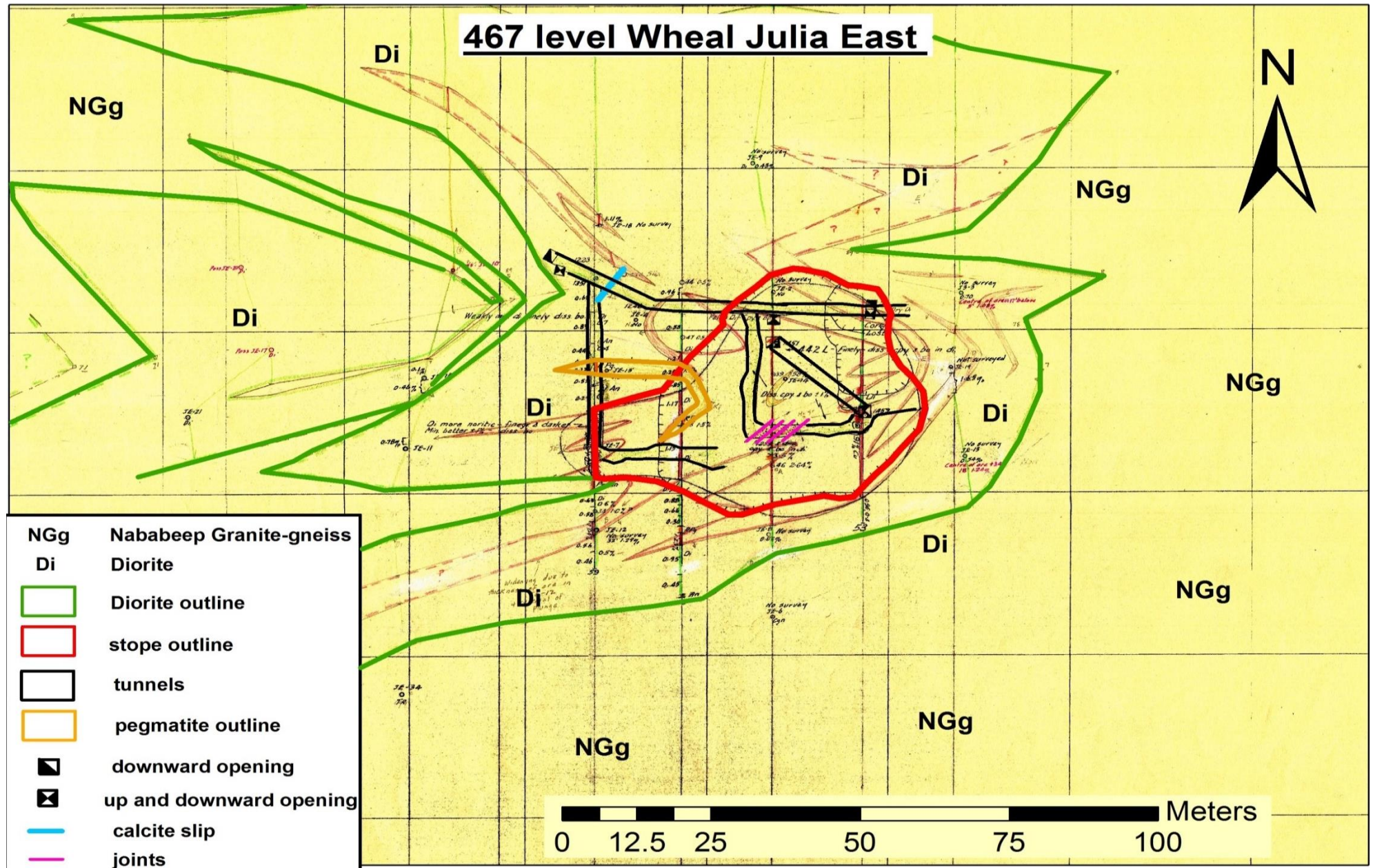


Figure 4. 20: An example of an underground geological map from Wheal Julia East Mine (O'okiep Copper Company, 1955-1959c).

Chapter 5: DATA ANALYSIS AND DISCUSSION

This chapter entails the analysis and discussion of data collected from the research reported in Chapter 4. Adding to these results, the author conducted field investigations by means of a drone aerial survey of each mine, as well as site walkovers where surface and rock mass features are described and incorporated into the analysis of data and the overall discussion of each mine. From the reported results and the field investigations the causes of the development of glory holes at each mine will be determined and discussed. The analysed data and discussions of the reference mines will finally be used as a proxy to add to the discussion and analysis of the Bruinhoek populated area, which is undermined by the Wheal Julia Mine. To conclude the chapter three different safety zones will be identified for the Bruinhoek populated area using the Mine Health and Safety Act No. 29 of 1996 (South Africa, Department of Mineral Resources, 2018), the risk rating matrix from Figure 3.3 and discussions from Chapter 5.

5.1 Rietberg Mine

From the drone aerial survey, a high resolution 2D elevation map was produced, which is displayed in Figure 5.1. The lighter colours on the map represent higher elevation, whereas the darker zones indicate areas of lower elevation. The figure shows that the ground level around the two Rietberg glory holes is not obviously lower lying than its surroundings. It indicates that intense surface subsidence is not a precursor to the development of these glory holes and that glory hole formation has happened suddenly, without any warning. However, there is a zone of slight subsidence located at a portion of the rim of the C-shaped glory hole. Other elevation differences that can be seen on the map are the glory holes themselves, relative to ground level; the natural slope of the terrain from east to west and the progressive rim collapse of the O-shaped glory hole. Another high resolution 2D map can be seen Figure 5.2, which is an aerial view of the Rietberg glory holes. Zooming in on the C-shaped hole, prominent surface cracks can be seen outlining some areas of the rim. It must be noted that some of these cracks are located in exactly the same area as the previously mentioned surface subsidence of the rim seen on Figure 5.1.

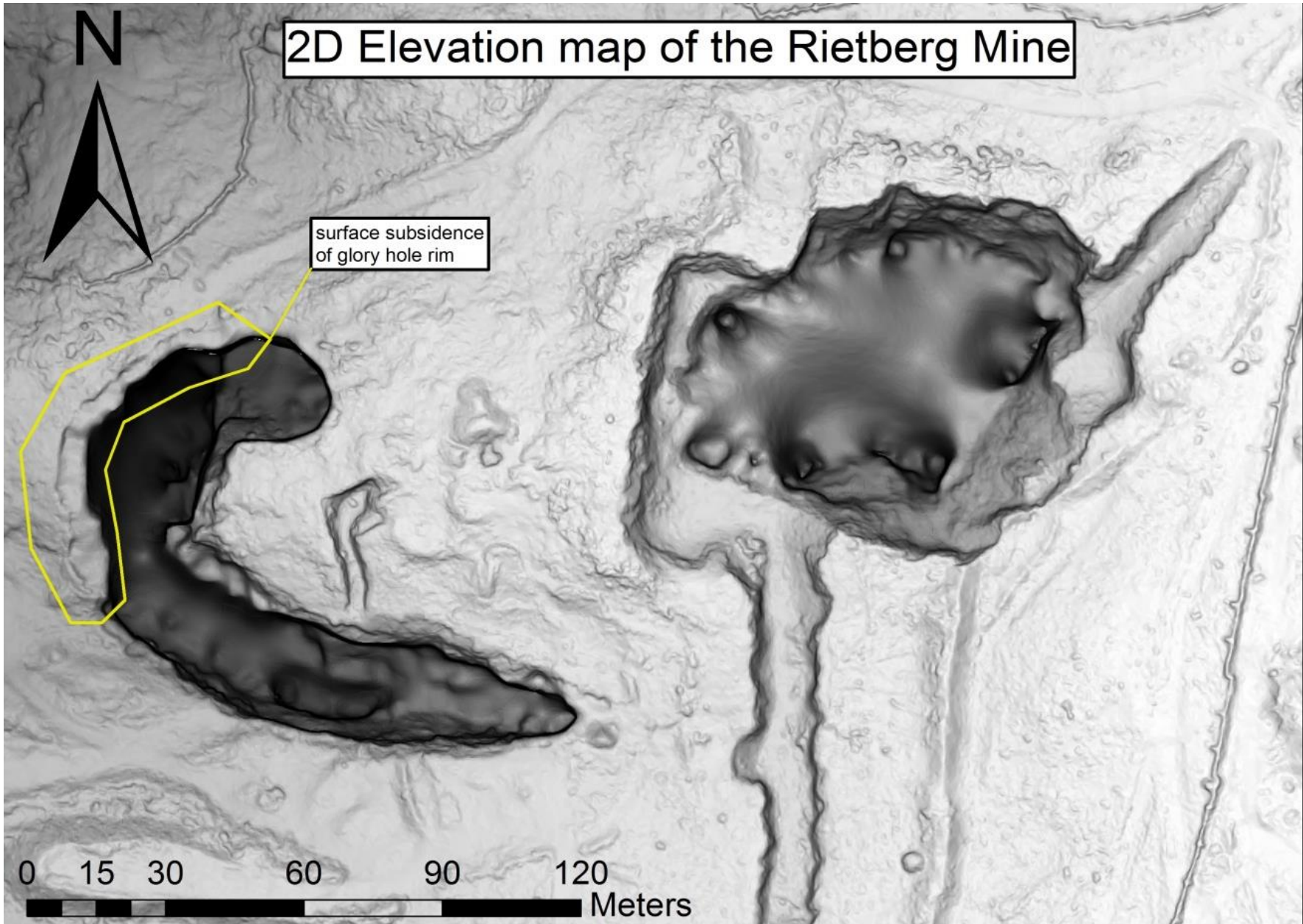


Figure 5. 1: 2D Elevations map of the displaying the elevations at and around the Rietberg glory holes.

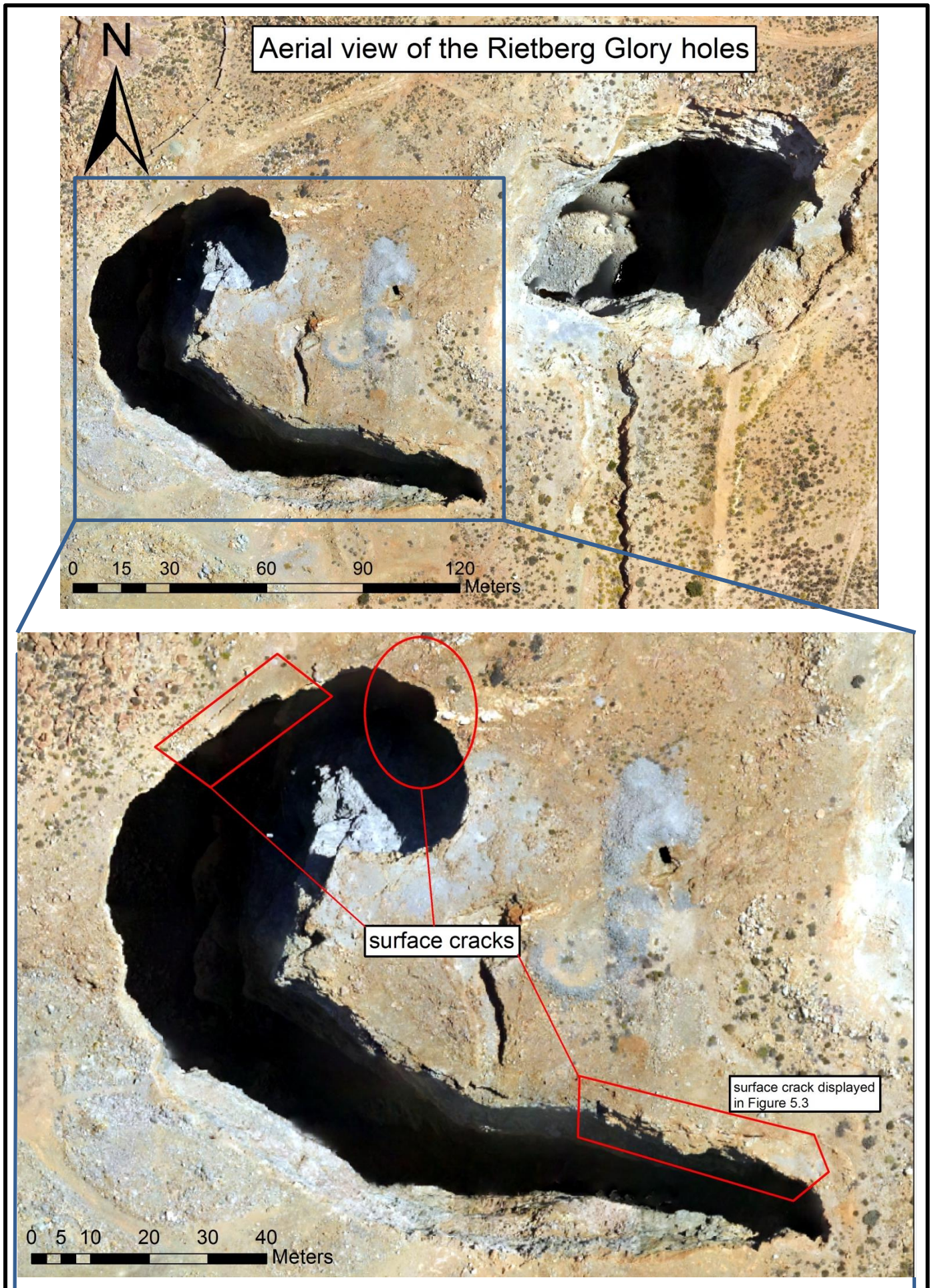


Figure 5. 2: 2D aerial view of the Rietberg glory hole with an enlarged image of the C-shaped glory hole.

In Figure 5.3 one of these prominent surface cracks can be seen up close. It extends for approximately 10m-20m, has a width ranging between 5cm-10cm and is partially filled with material from the shallow soil layer.

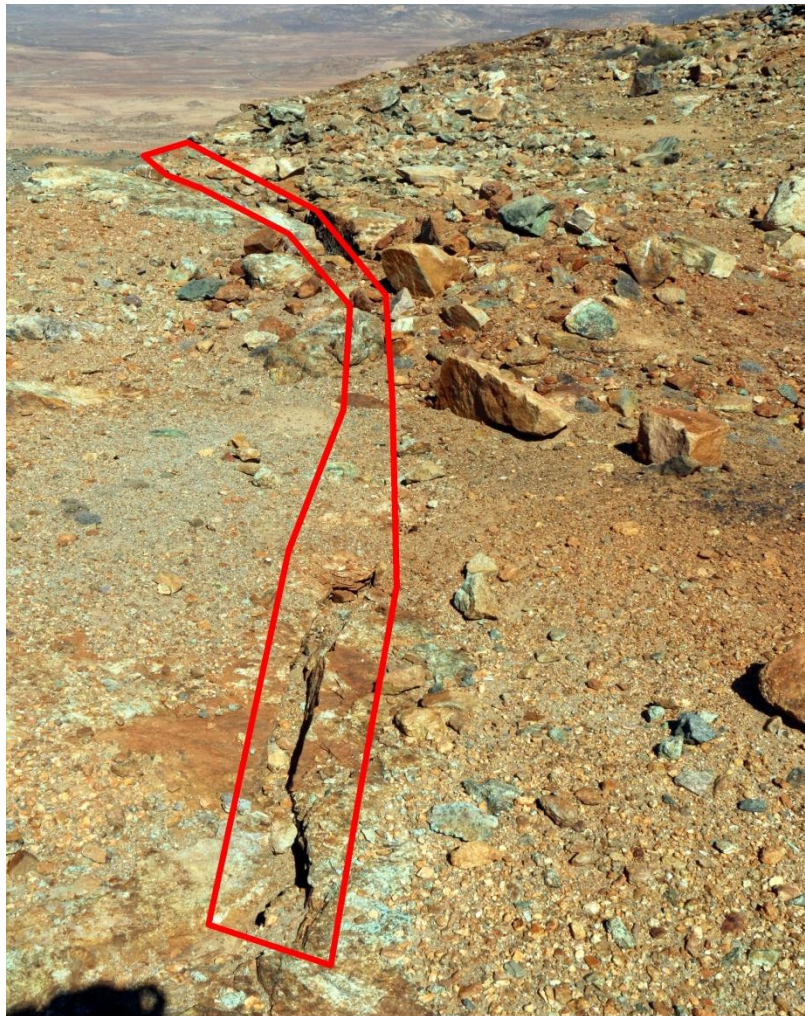


Figure 5. 3: A surface crack located at the C-shaped glory hole outlined in red.

During the field investigation it was also found that the soil cover at and around the glory holes is extremely thin, having a maximum thickness of 20cm. It is followed by a 2 m - 5 m zone of weak bedrock material composed of Rietberg granite (Kisters, 1993), as seen in Figure 5.4. The material contains many joints and has reddish-orange and greenish colour stains. The aperture of these joints ranges between 1mm and 5mm, having rough to slightly rough surfaces. Most of these joints have no infilling, while a few have a hard filling for example sand like material. In general, for this 2 m - 5 m zone of weak bedrock, joints are moderately weathered. This zone cannot quite be defined as saprolite, as it is not deeply weathered material. According to Bétournay (2004), commonly the first 5m to 15m of bedrock can be moderately to seriously altered with higher jointing density and apertures.

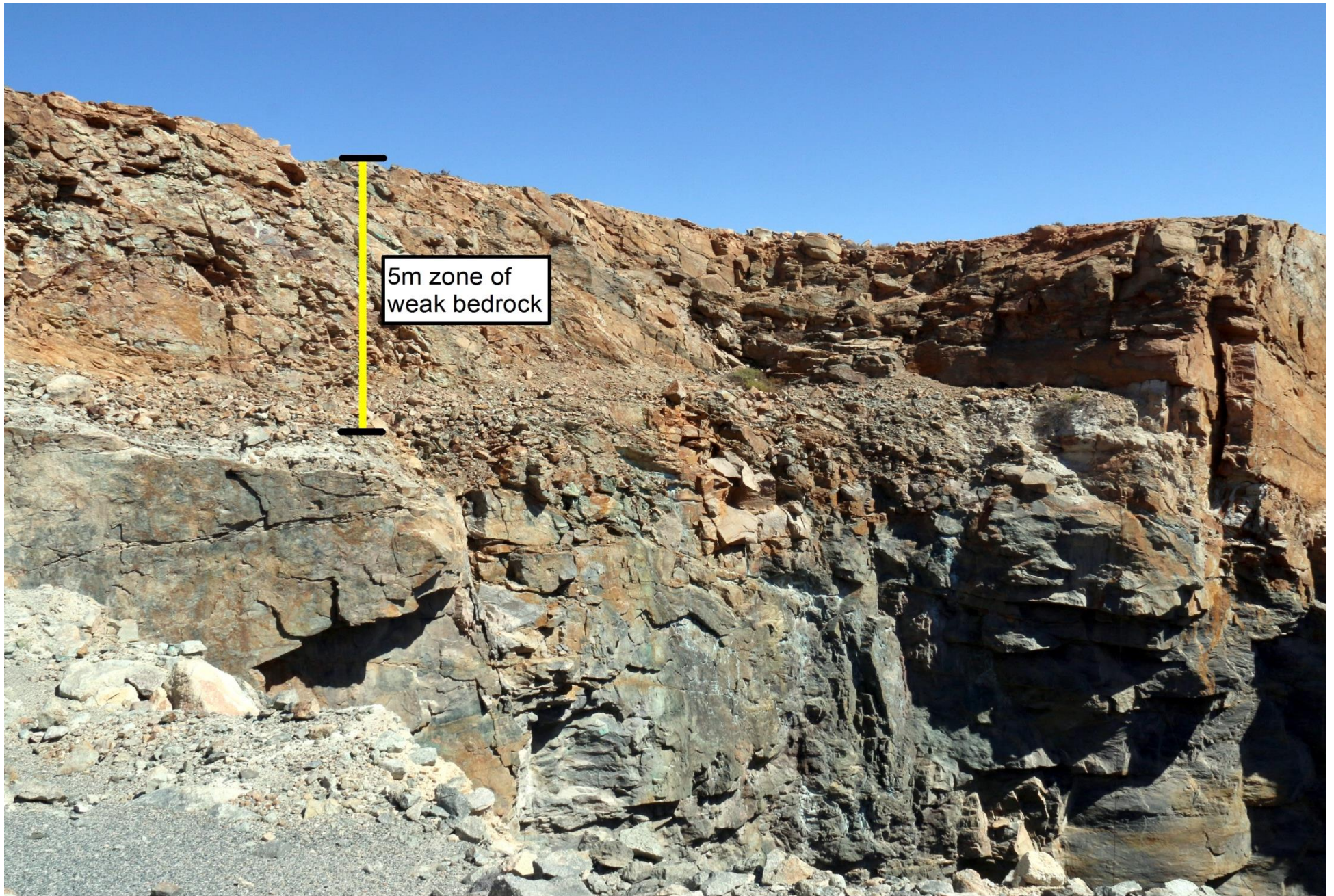


Figure 5. 4: The 5m zone of weak bedrock located below the thin soil layer.

Below this zone of weak bedrock a more stable rock mass is encountered, which stretches down into the glory hole (see Figure 5.5). Although the rock mass appears to be more stable, jointing is nevertheless present, forming blocks not always equal in size and shape. From Figure 5.5 it can be seen that the majority of joints displayed have steep dips. The author was unable to determine the exact length of the joints; however, it can be estimated to range between less than 1m to 3m. For the majority of the joints the apertures are less than 1mm, with joint surfaces varying between slightly rough and smooth. No joint filling could be seen and overall joints are weathered on their surfaces. It should be noted that all of the joint conditions could only be estimated from a distance of about 10 m, as direct access to the glory hole walls was not possible.

According to a report from Site Plan Consulting (2005), cited in Chapter 4, it was mentioned that the sidewalls of the Rietberg glory holes are vertical and unstable. The vertical and unstable nature of the sidewalls can be seen in Figures 5.5 and 5.6. Figure 5.5 displays two zones where block-like rock material had come loose from the sidewall. This happens because of joint intersections in the rock mass, creating blocky rock material of different sizes where at least one joint intersect has a dip that allows the block structure to exceed the friction angle of the joint surface, ultimately allowing material to detach itself from the sidewall. Figure 5.6 is an example of a large individual block in the process of spalling from the sidewall. The spalling process of large blocks happens because of vertical discontinuities in the rock mass that can cause a block or slab to come loose and, according to Terzaghi (1946), the damaging of the rock mass during underground excavations and extraction of material contribute to the spalling phenomenon. It is not only the side walls of the glory holes that are unstable, but also the dangerously overhanging rims of both glory holes (Site Plan Consulting, 2005). The unstable nature of the rims was seen in Figure 4.2 of Chapter 4, where a piece of overburden that had formerly spanned over the C-shaped hole has now collapsed. Figure 5.7 shows an image of a piece of overburden that forms the rim of the C-shaped hole, and at the same time is overhanging a portion of the glory hole. The majority of the overhang is composed of the previously described weaker bedrock material and, as mentioned, this material is moderately weathered and has a high concentration of joints. Because the collapse of a weak overburden has happened in the past, the collapse of the dangerously overhanging rims of both glory holes can be expected to happen in the future.

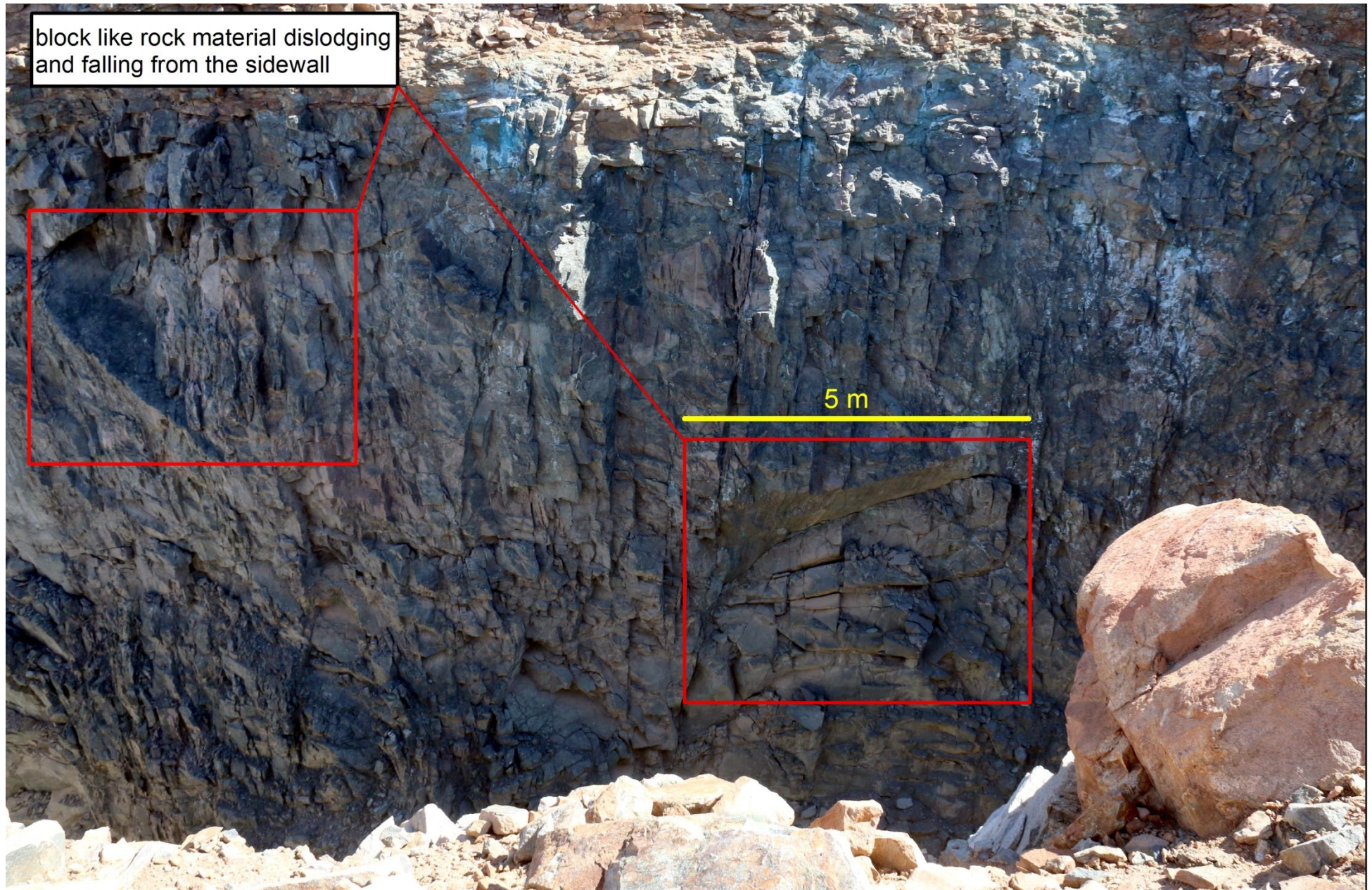


Figure 5. 5: Block like rock material coming loose and falling from the sidewall.

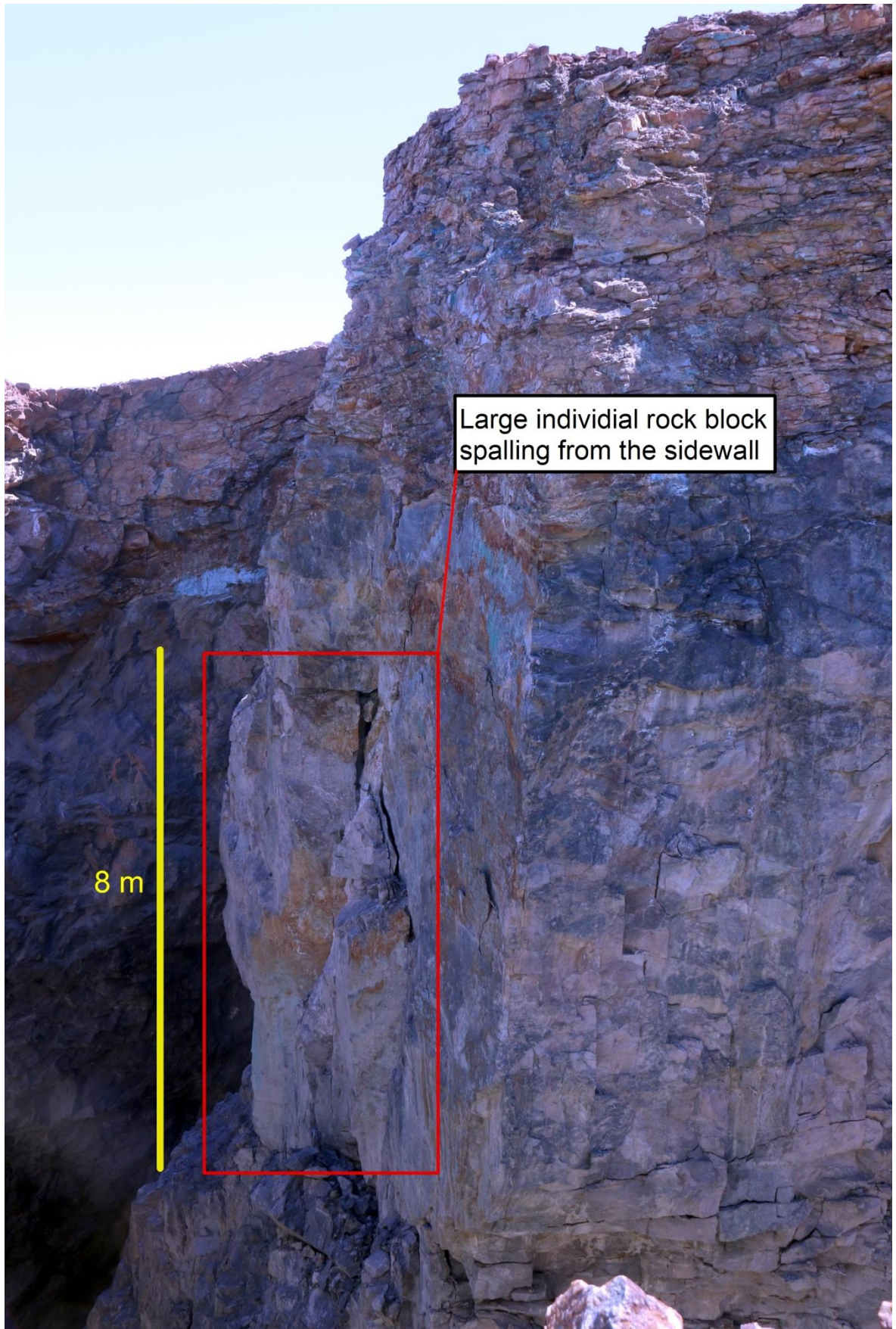
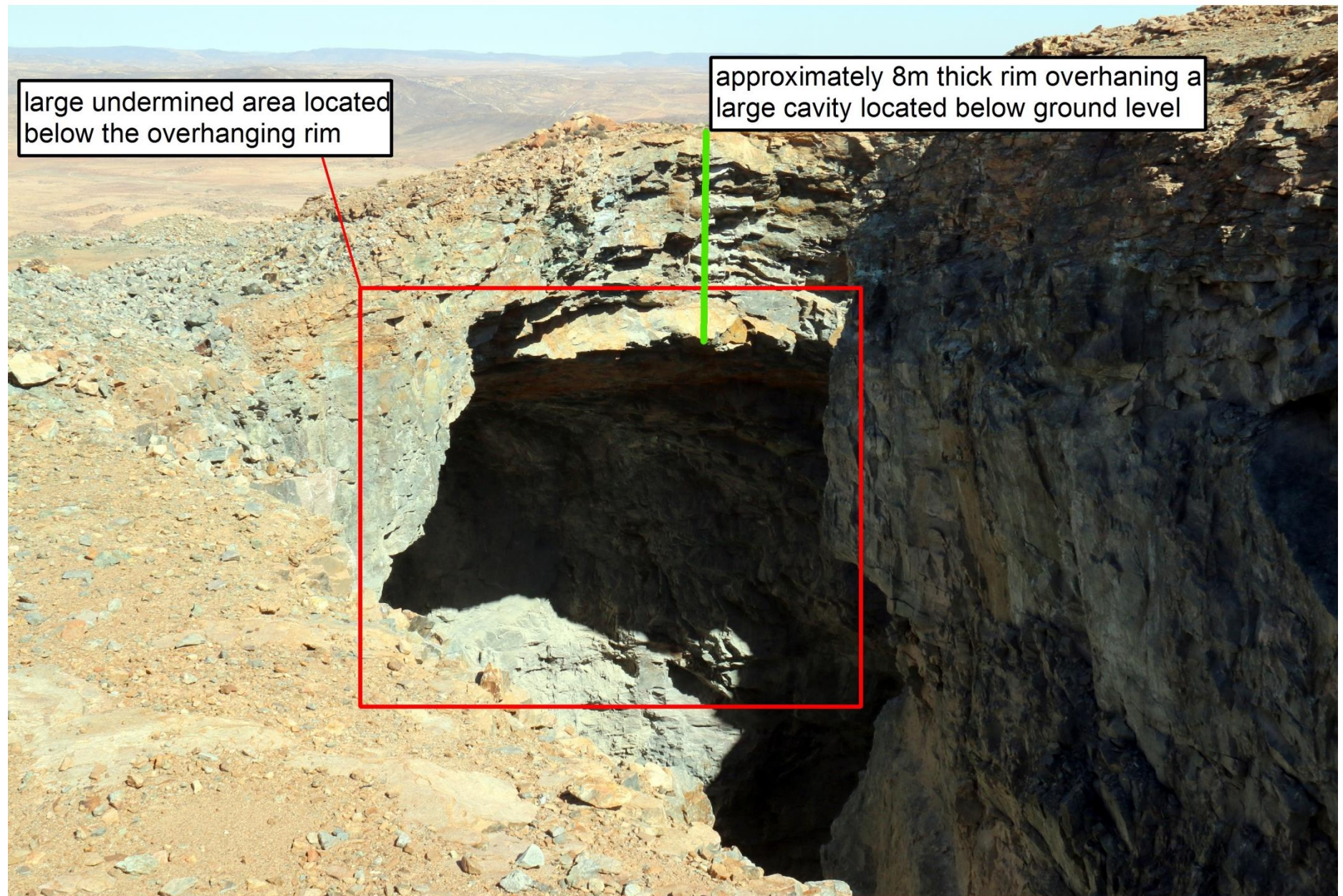


Figure 5. 6: A large unstable rock block spalling from the sidewall.



large undermined area located below the overhanging rim

approximately 8m thick rim overhanging a large cavity located below ground level

Figure 5. 7: A 8m thick rim overhanging a large undermined area.

The main reasons for the formation of the Rietberg glory holes are the large underground cavities created by mining, and the state of the uppermost rock mass. As mentioned in Chapter 4, the Rietberg Mine covers a vertically mined area of approximately 36 000m², where mining activities have taken place as close as 15m from the surface. According to Bétournay (1995), when shallow mining of large underground areas take place the rock mass becomes de-stressed as confining pressures reduce. This essentially causes a reduction in the horizontal stress of the weak rock mass that help formed and support the arch over the underground cavities. The horizontal stress is essential to help keep the weak and jointed rock mass of the arch intact. However, the reduction in horizontal stress which is caused in this situation also reduces the shear resistance on joint planes. This allows the blocks formed by these joints to slide out of their original position, initiating raveling of the arch, as explained in Chapter 2 by Bétournay (2004). Eventually, the arch completely loses its ability to support itself and it collapses. It is also possible, as mentioned previously, that the dangerously overhanging rims of the glory holes may collapse in future and this is likely to happen as a result of the same mode of failure as the arch.

As mentioned in Chapter 4, the Rietberg Mine glory holes are surrounded by a 1.4 m high gabion wall. It should be noted that this is only a moderate form of protection as animals and most people would not be able to climb over it.

5.2 O'okiep Mine

It was previously mentioned that the town of O'okiep is a historic mining town, while the O'okiep Mine itself is of historic importance to the history of mining in South Africa (Smallberger, 1975). According to Chapter 4 of this thesis, very few maps could be found for the O'okiep Mine and the reader should be aware of this also throughout the discussion in Chapter 5. The survey maps allow for the understanding of some of the mining activities, the dimensions of underground activities and the location of underground mining relative to the surface. With no geological maps found, little is known of the geological structure of the rock mass, specifically at the O'okiep Mine. However, as stated by Lombaard (1986) the geology of the Okiep Copper District has experienced the same deformational events, indicating that rock masses from different areas can have similar structural properties.

In Figure 4.4 a portion of a survey map of West O'okiep Mine can be seen, which indicates two shaft positions that are located in the same region above underground workings, as was found in Chapter 4. In Figure 5.8 this same map can be seen where it is overlain by a surface level image of the exact same area, indicating that the old shaft is located in the middle of the West O'okiep glory hole and no 3 shaft is located at the Cornish pump house. According to Figure 4.5 the two shafts are separated from the underground cavity by a 40 m-50 m thick overburden. The vertical undermined area located below the shafts is approximately 25000m², as mentioned in Chapter 4.

The drone aerial survey could not be conducted for this mine. The terrain around the glory hole is flat, except for 3 smallish mine dumps located to the west of it. The concrete palisade mentioned by Site Plan Consulting (2005) in Chapter 4 surrounds the entire glory hole and is approximately 2.5 m high. The palisade has proved to be an ineffective solution, as it has been vandalized in some areas, failing to prevent access to the hole and thus not ensuring the safety of the people of the town. For safety reasons the palisade was not crossed by the author in his attempt to investigate the areas close to the rim. There is no evidence of surface instabilities outside the borders of the palisade, as no surface cracks or depressions (sagging) were observed. As mentioned by Site Plan Consulting (2005) in Chapter 4, the glory hole is filled with water and only 2 m of the rock mass is visible above the water level. Similar to the Rietberg Mine, the soil cover is very thin, being only 20 cm thick, followed by weak bedrock material with mostly rough surfaced sidewalls and a few smooth surfaces. These smooth surfaces are exposed discontinuity planes that were created when blocks of rock mass dislodged during the caving process. Although the overall density of discontinuities in the bedrock appears to be lower than those in the Rietberg Mine, zones of weakness can nevertheless be identified. Steep and shallow dipping discontinuities, with variations in strike, are present throughout the rock mass exposed above water level. The density of discontinuities is inconsistent, as the rock mass displays blocky textures as well as more uniform material with smooth sides. The spacing of discontinuities in the blocky rock mass is approximately 15 cm-30 cm with apertures of approximately 1 mm-5 mm. In other zones of more uniform bedrock the spacing between discontinuities is 0.5 m-2 m with apertures ranging between 1 cm and 5 cm. These discontinuities can partially be seen in Figure 5.9. The weathering of discontinuities, their filling and its roughness could not be observed. It should be noted that these observations were made from a small portion of exposed rock mass and that these properties may differ for the rock mass located deeper into the glory hole. Finally, the abundance of water indicates the presence of breccia faults as they are important aquifers in the Okiep Copper District and cause intense brecciation of the country rocks (Gadd-Claxton, 1981) and (Kisters *et al.*, 1996).

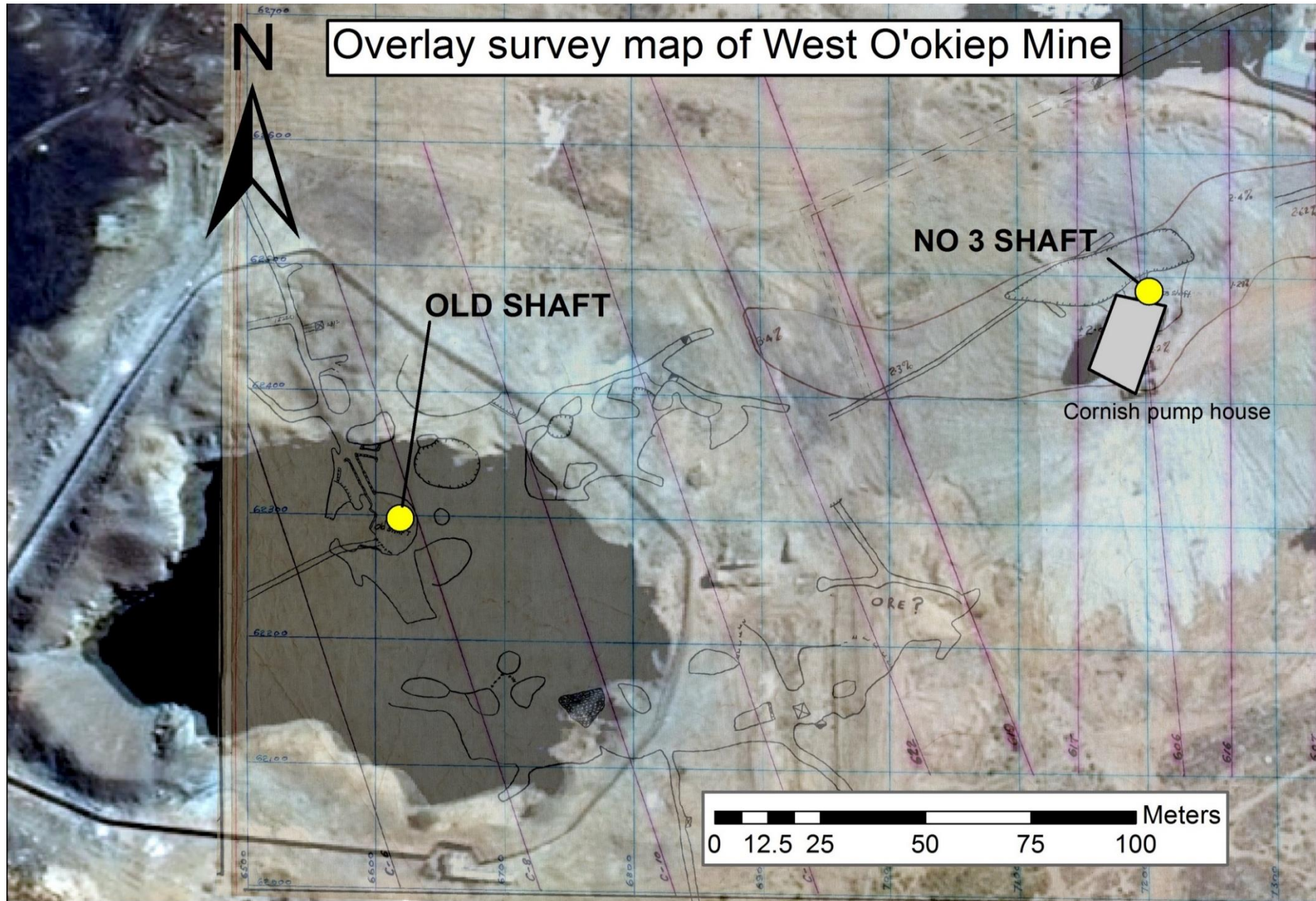


Figure 5. 8: A portion of a survey map overlying the West O'okiep Mine (O'okiep Copper Company, 1942-1975a).

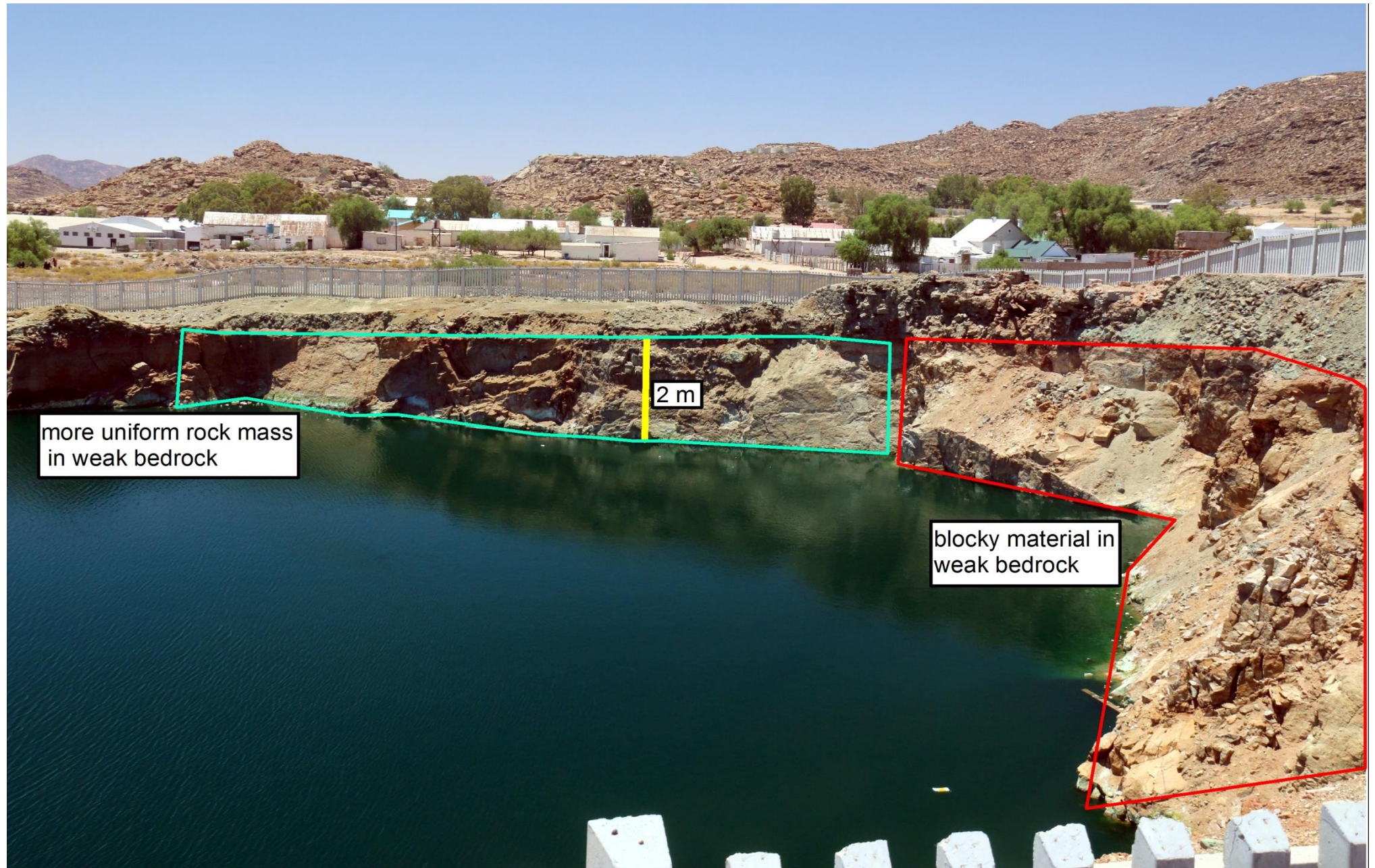


Figure 5. 9: Zones of blocky and more stable rock masses in the zone of weak bedrock.

Although there were limitations with regard to the field investigation of West O'okiep and not many maps could be found, as reported in Chapter 4, certain factors that could affect and cause the formation of the glory holes were identified. The initiating factor involves the main shaft (old shaft) of the mine. The West O'okiep Mine was operational more than a century ago during the late 1800's and early 1900's (Beukes, 2016). No information with regard to closure methods for mine openings at West O'okiep could be found. In his thesis, Heath (2009), states that the closure of mine openings was not a priority for mine operators in the past, as guidelines to how they should be closed and sealed were not available in many countries. Although it is not known how the main shaft (old shaft) of West O'okiep was rehabilitated, or if it was ever rehabilitated, the chances of it being done effectively are unlikely as this is a very old mine mentioned in Chapter 4. According to Heitfeld *et al.*, (2006), abandoned shafts completely filled with unstable filling, filled only partially, or containing no fill, can lead to failure of the shaft lining. The outcome of this may be the caving and collapse of surrounding incompetent rock mass, creating a crater at surface level with greater dimensions than the shaft diameter itself. The initial stages of the caving process at West O'okiep started off with the collapse of the main shaft (old shaft). Following the collapse of the shaft, three main interdependent factors contributed to further caving of the of the rock mass to create the West O'okiep glory hole. The first factor is the nature of the discontinuities in the overburden. The bedrock encountered below the soil layer is weak. The discontinuities in this bedrock create zones of unstable blocky material with wide apertures. The unstable nature of the blocky mass can be seen in Figure 4.7, where the expansion of the glory hole rim is displayed. In the more uniform zones of the weak bedrock fewer discontinuities were seen with seemingly wider spacing between them, while at the same time they have very wide apertures and smooth discontinuity surfaces, seen in Figure 5.9. Although the properties of the rock mass below the water surface could not be seen, discontinuities can be expected to occur in the lower levels because, as noted earlier, the same deformational events are experienced by the rock succession of the Okiep Copper District (Lombaard, 1986) and, because of this, the West O'okiep Mine can in this sense relate to the Rietberg Mine, where discontinuities were present in the lower levels of the rock mass. Finally, breccia faults responsible for intense country rock deformation can also be expected in the lower rock mass, as explained earlier. These discontinuities create planes of weakness along which material may shear when vertical stress exceeds shear resistance. This initiates raveling and/or rock mass caving of the overburden that can result in the formation of a glory hole. The second factor that contributed to the formation of the glory hole is the degree of extraction. According to Altun *et al.*, (2010) and Singh (1992), when a great amount of extraction takes place in shallow stopes, especially in a horizontal direction, rock mass becomes de-stressed and can have insufficient internal stress to prevent gravity failures. A third factor contributing to glory hole formation is groundwater. The Cornish pump house started pumping water from the mine in the late 1800's (Smallberger, 1975) indicating that groundwater has been a problem since the early stages of mining. According to Blodgett and Kuipers (2002), the disturbance of strata in a mined area can alter the drainage

gradients, explaining why the West O'okiep Mine has experienced ground water problems since early mining days in a region where breccia faults are encountered. Ground water can now move through discontinuities in the rock mass, increasing pore fluid pressures and decreasing frictional strength between discontinuity planes. Groundwater essentially serves as a lubricant in rock mass material and can ultimately induce caving of the overburden.

The second part of the O'okiep Mine, East O'okiep Mine, is located approximately 300m east of the West O'okiep Mine glory hole and therefore similar geological conditions to those of West O'okiep Mine are likely to prevail here. Chapter 4 indicated that, although the underground cavity was backfilled and crown pillars were left in place to provide surface stability of the rock mass, a glory hole nevertheless did occur. Similar to West O'okiep, the glory hole at East O'okiep formed mainly because of interdependent factors. In the case of East O'okiep the nature of the overburden, depth of extraction and the *in situ* stress of the rock mass are the main contributors to the formation of the glory hole. Having very similar rock mass properties as the West O'okiep Mine, it can be expected that the overburden separating the underground cavity and ground level is made up of fairly weak rock mass. This, along with extensive horizontal mining at very shallow depths, caused the weak and thin overburden to de-stress, which allowed gravitational forces to induce movements along the planes of weakness in the overburden itself. These prevailing conditions have the potential to result in the failure of overburden. To counter this process, in an effort to prevent it from taking place, the mined area was backfilled with sand and crown pillars, spanning across the width of the cavity, were left behind.

According to Bétournay (2004), the shallow section of the Hollinger Mine in Canada ranged between depths of 150m deep to near surface level. The stopes located in this shallow section of the mine were backfilled with gravel through openings located at surface level. Bétournay (1995) explains that the fill material would thus tend to form cones at the material's angle of repose, thereby leaving considerable void space, which finally resulted in the collapse of the unfilled cavity. As indicated in Chapter 4, the East O'okiep Mine underground cavity was backfilled via surface boreholes, which is a similar technique to that used at the Hollinger Mine in Canada. As backfill material was filled into the underground cavity of the East O'okiep Mine, the sandy backfill material was not evenly distributed, which ultimately resulted in the creation of cones, leaving behind large void spaces. Another problem with this backfill technique is that the sand fill is not compacted properly and may subside or run out over time, creating even more underground void spaces. The glory hole at the East O'okiep mine formed as a result of this insufficient backfill technique. It allowed the road pillar and overburden composed of fairly weak and de-stressed rock mass material to collapse into a void space created by the insufficient backfill technique.

In Chapter 4 it was found that the East O'okiep glory hole was filled with a layer of sand and gravel. The filled area can be seen in Figure 5.10. Although a drone aerial survey could not be done for the

O'okiep Mine, the field investigation proved that although the glory hole has been filled, instabilities still prevail in this area. Figure 5.11 displays two images that show 15 cm-20 cm wide surface cracks that evolved around the edges of the filled glory hole, emphasising the unstable nature of this area. These surface cracks are created by gradual subsidence of the fill material as it self-compacts, as well as by the displacement of underground fill material, overburden and pillar material into void spaces that remain underground. Another factor contributing to the creation of void spaces is the run out of backfill, resulting from loose backfill material and surface water drainage after heavy rains. These factors have the potential to result in enough subsidence to create another large surface depression at the filled glory hole.

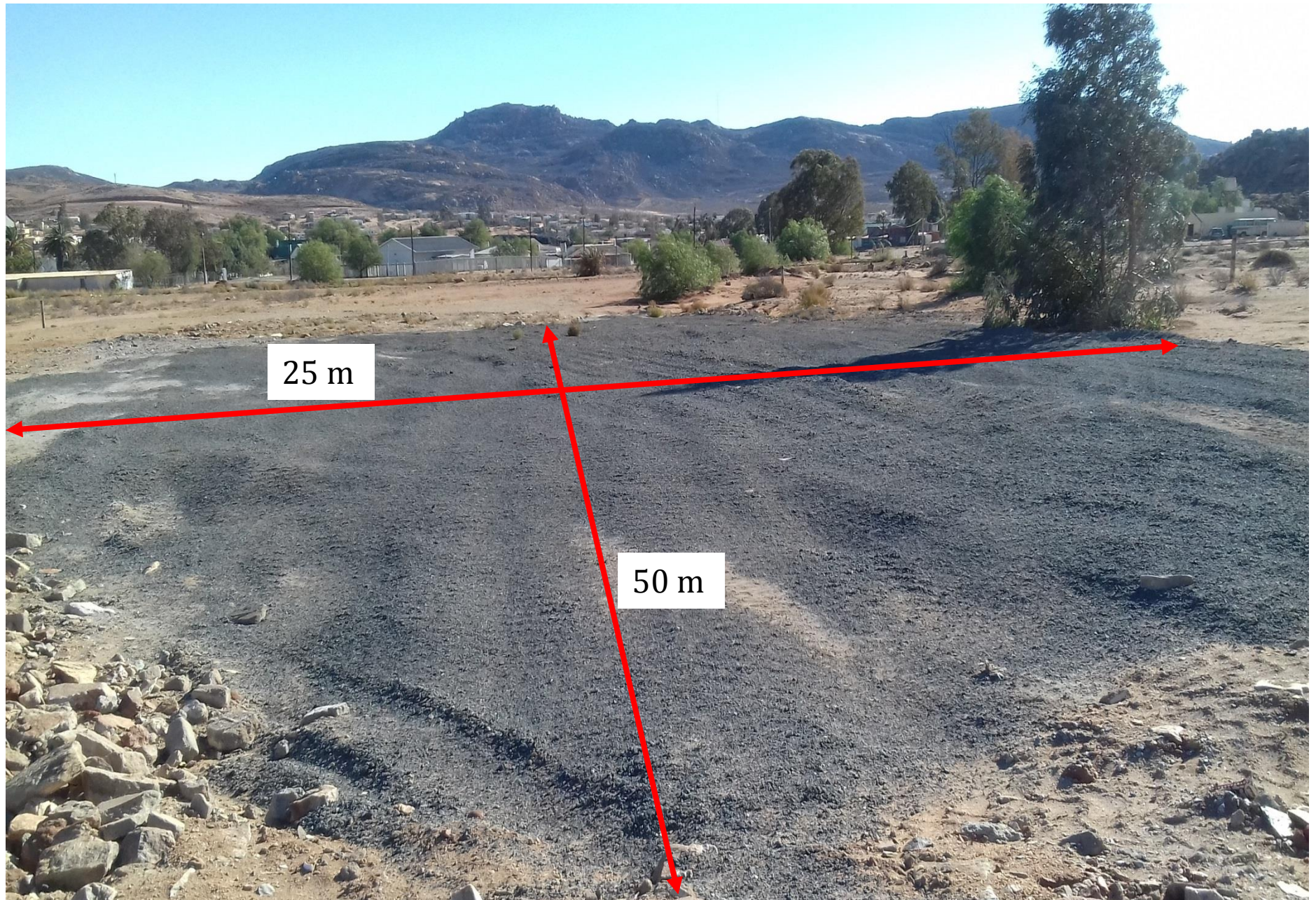


Figure 5. 10: The sand and gravel filled East O'okiep glory hole.



Figure 5. 11: The 15 cm-20 cm wide surface cracks outlining the gravel and sand filled East O'okiep glory hole.

5.3 Hoits Mine

In contrast to the Rietberg and O'okiep Mines, survey maps, as well as geological maps of the Hoits Mine were found. As seen in Chapter 4 the geological maps are an important source of information as regards the structural characteristics of the rock mass. Zones of structural importance were indicated for each level and, as explained previously, these zones represent an area where the concentration of joints, faults or both are high, as indicated on each map by the geologist who compiled the maps. These identified zones raise concerns regarding the stability of the rock mass. To identify the locations of these zones of weakness in the rock mass relative to ground level, overlay images were created which show the geological maps from Chapter 4 overlaid by a 2D orthomosaic surface map of the Hoits Mine. An example of an overlay map can be seen in Figure 5.12, while the remaining overlay maps are included in Addendum D. Examining Figure 5.12 and the overlay maps in Addendum D, it was found that each overlay map the glory hole overlies an area of rock mass weakened by closely spaced, steeply dipping fault surfaces, accompanied by steeply dipping joints. Therefore, at each level, these weakened zones form part of a fault zone (Davis *et al.*, 2011) that cuts through the location of the glory hole, brecciating both the noritic rocks and NababEEP Granite-gneiss and creating a zone of weakness within the rock mass. According to Davis *et al.*, (2011), fault zones made up of closely spaced breccia faults typically represent masses of broken rock that can be incohesive and are often accompanied by numerous joints. Although descriptions of the fault weakened rock mass were not provided on the geological maps, notes by the geologist indicate that in brecciated and jointed areas rock alterations in the form of chloritisation of the noritic rocks and kaolinisation of the NababEEP Granite-gneiss are common. The secondary minerals are not a product of chemical decomposition, as the Okiep Copper District has a Weinert N-value higher than five, indicating that physical weathering dominates in the region (Weinert, 1967), and therefore, secondary minerals are a product of hydrothermal activities. Secondary minerals can further affect the stability of the fault zone, as they have very low frictional properties (Palmström, 2014), essentially lowering the shear resistance along discontinuity planes. Based on the geological maps, the Hoits Mine orebody, or at least part of the orebody, is located in a fault-weakened and jointed rock mass which is, according to Bétournay (1995), a rock mass environment in which underground mining sometimes take place in order to extract ore. The weak and unstable rock mass in the overburden make this area a likely site for two failure mechanisms, as explained by (Bétournay, 2004) and (Didier *et al.*, 2008) and reported in Chapter 2. The first mechanism is raveling, which occurs, as explained previously, when the jointed span of an opening is left unsupported and it then exceeds its self-support capabilities. This results in the gradual failure of the periphery, because of unfavourably oriented rock blocks. When raveling reaches the top of the bedrock in unfavourable rock mass conditions, for example highly jointed and/or fault weakened material it can cause destabilisation of the crown pillar or overburden. The second mechanism is rock mass caving that involves the break-up and mobilisation of blocks into an opening by means of gravity, when a progressive failure front moves towards the surface. According to Bétournay(1995), several

rock mass conditions can lead to the initiation of rock mass caving. Two of these conditions are persistent discontinuities and low block surface friction, which are both present at the Hoits Mine.

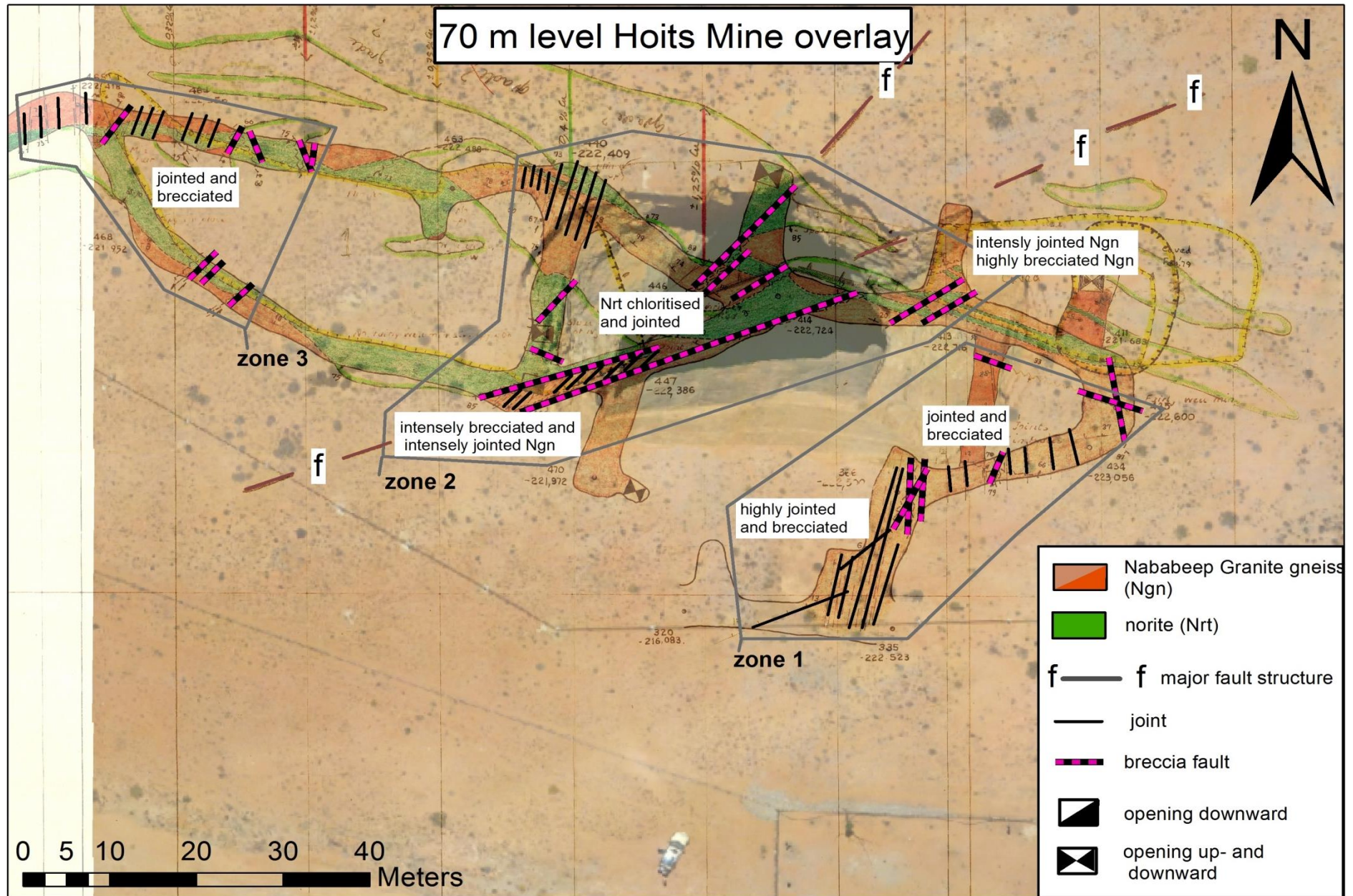


Figure 5. 12: 70m level map overlay of the Hoits Mine (O'okiep Copper Company, 1975-1993d).

In Figure 5.13 a high resolution 2D elevation map can be seen where darker zones indicate depressions and the lighter zones the more elevated areas. The most prominent feature on the map is the dark zone represented by the Hoits Mine glory hole. In Figure 5.13 there are no clear dark areas surrounding the rim of the glory hole, indicating that gradual subsidence of the surface is not taking place now and did not take place prior to the formation of the hole. It indicates that the formation of the glory hole at surface level was a sudden event and not a gradual one. The darkish area located south of the hole is the result of the natural slope of the terrain towards a dry riverbed.

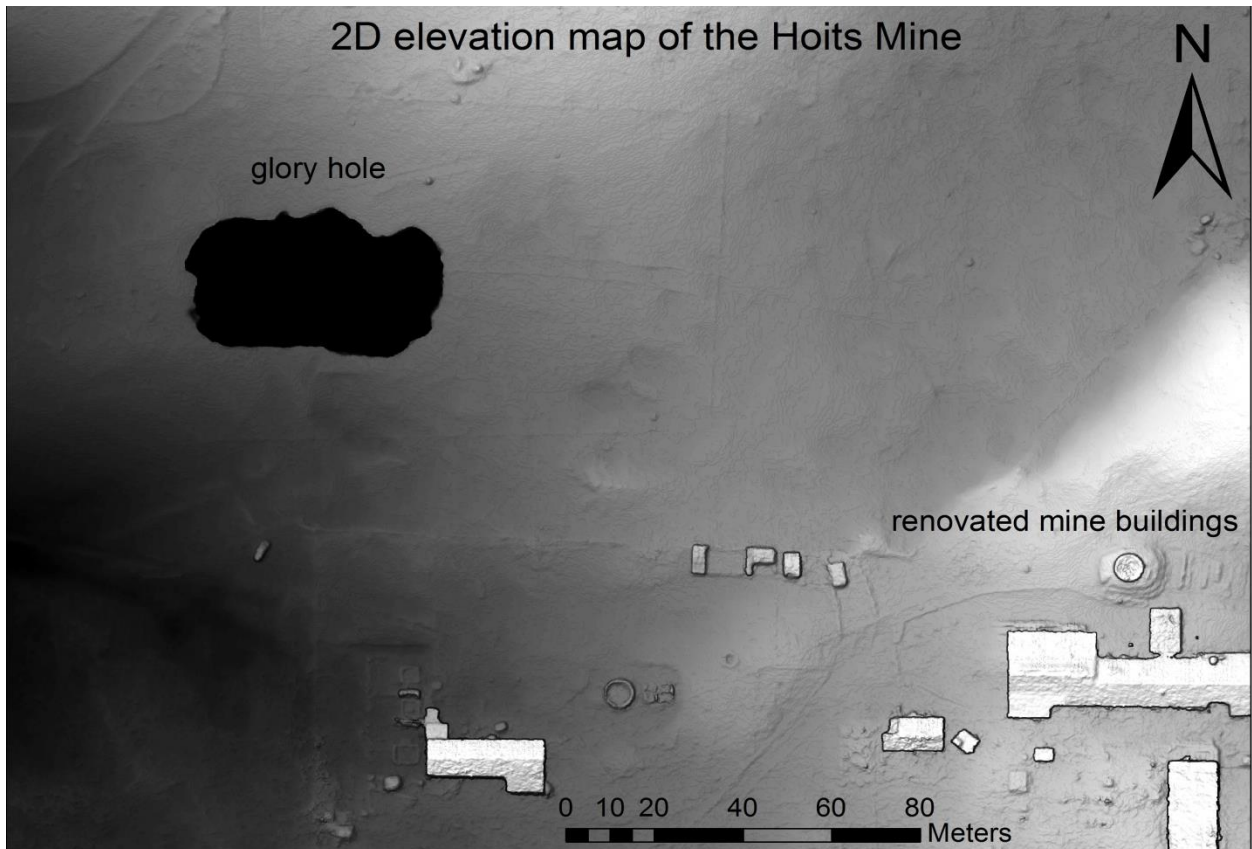


Figure 5. 13: 2D elevation map of Hoits Mine.

Figure 5.14 displays the same surface area of the Hoits Mine in the form of a 2D orthomosaic map. Here it can be seen that the glory hole is located less than 100m from areas of regular human activity, which confirms the same statement made by Site Plan Consulting (2005), as reported in Chapter 4. On the western, eastern and northern rims of the glory hole cracks can also be seen, indicating that the hole is gradually self-caving and growing in size. Parts of these cracks seen in Figure 5.15 are located on the eastern rim of the glory hole. The full length of these cracks reaches 20 m, with a spacing of 1m between adjacent cracks. These cracks have a minimum width of 5 cm and are only partially filled with material from the shallow, 20 cm soil layer surrounding the area of the glory hole. These prevailing conditions support the statement by Spiessens (2015) in Chapter 4 that the rim of the glory hole is unstable and will cave in the future.

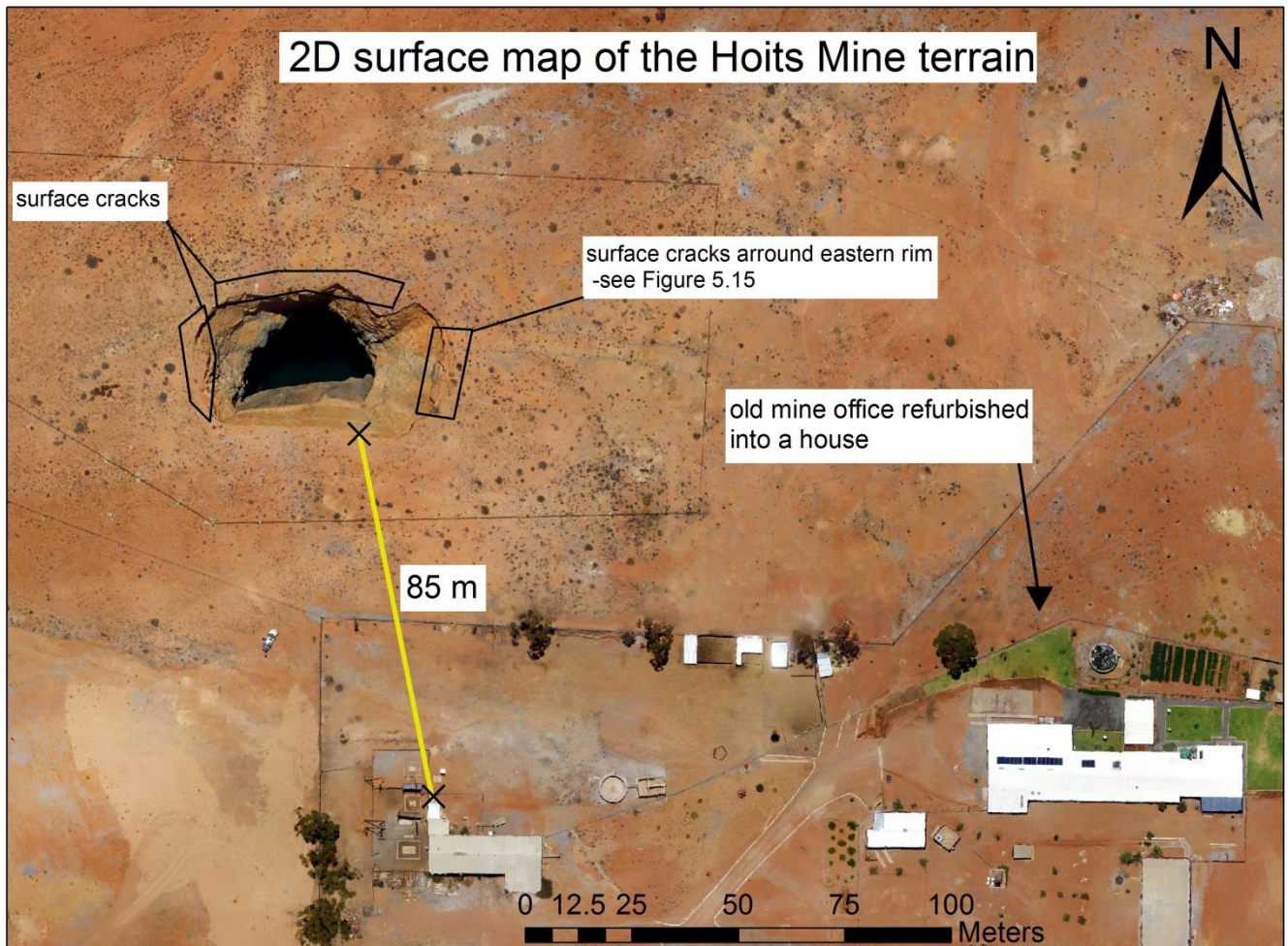


Figure 5. 14: 2D surface map of the Hoits Mine.

Figure 5.16 shows the northern sidewall of the glory hole. The yellow-outlined zones represent the two areas in the rock mass through which the fault zones, which are indicated as major fault structures on geological maps in Chapter 4, cut through. While the largest part of the sidewall is made up of unstable block like rock mass material (orange outlined zones) the two yellow outlined zones are specifically dominated by closely spaced vertical discontinuities. These steeply dipping features have resulted from the development of the fault zones, as well as the strain that this put on the surrounding rock mass. As mentioned previously, these features cut through the area where the glory hole is currently located and essentially created an area of weakened rock mass located right above an undermined zone before the glory hole was formed. Figure 5.17 displays the western and southern sidewalls. The outlined area on the western wall indicates the location just before the two fault zones seen in Figure 5.16 come together to form a single fault zone, as can also be seen on the 70 m level overlay map in Figure 5.12. Like the fault zones on the northern face, the rock mass seen on the western sidewall is also composed of vertical, closely spaced discontinuities, creating an unstable rocks mass of broken rock material. This wall is also located in the same area as the prominent surface cracks around the western rim of the glory hole. This emphasises the unstable nature of the rock mass and the likelihood of further caving in the future. The southern wall is especially noticeable because of

its unusually smooth surface in comparison to other sidewalls (Figure 5.17). On the surface of the smooth wall, thin lines can be seen running across the face of the wall which are almost parallel to each other and to the ground level. These lines are slickenlines. According to Davis *et al.*, (2011), slickenlines are generally straight, fine-scale, skin deep lines that occupy a fault surface, and can record the direction of slip. This sidewall, most likely the result of a strike slip fault, is a significant contributor to the formation of the glory hole, as it creates an almost frictionless failure path along which the fault weakened rock mass can move. The smooth surface was most likely created as a result of the movement of the fault along the foliation of the rock mass. As explained by Kisters *et al.*, (1994), the copper bearing orebodies of the Okiep Copper District are associated with steep structures. This is where the subhorizontal gneissosity of the country rocks (Nababeep Granite-gneiss) has been rotated to subvertical attitudes. The subvertical orientation of the well-foliated country rocks most likely allowed the surface of the southern wall of the glory hole to become slightly polished by the movement of the strike slip fault, creating the smooth sidewall as it is seen today.

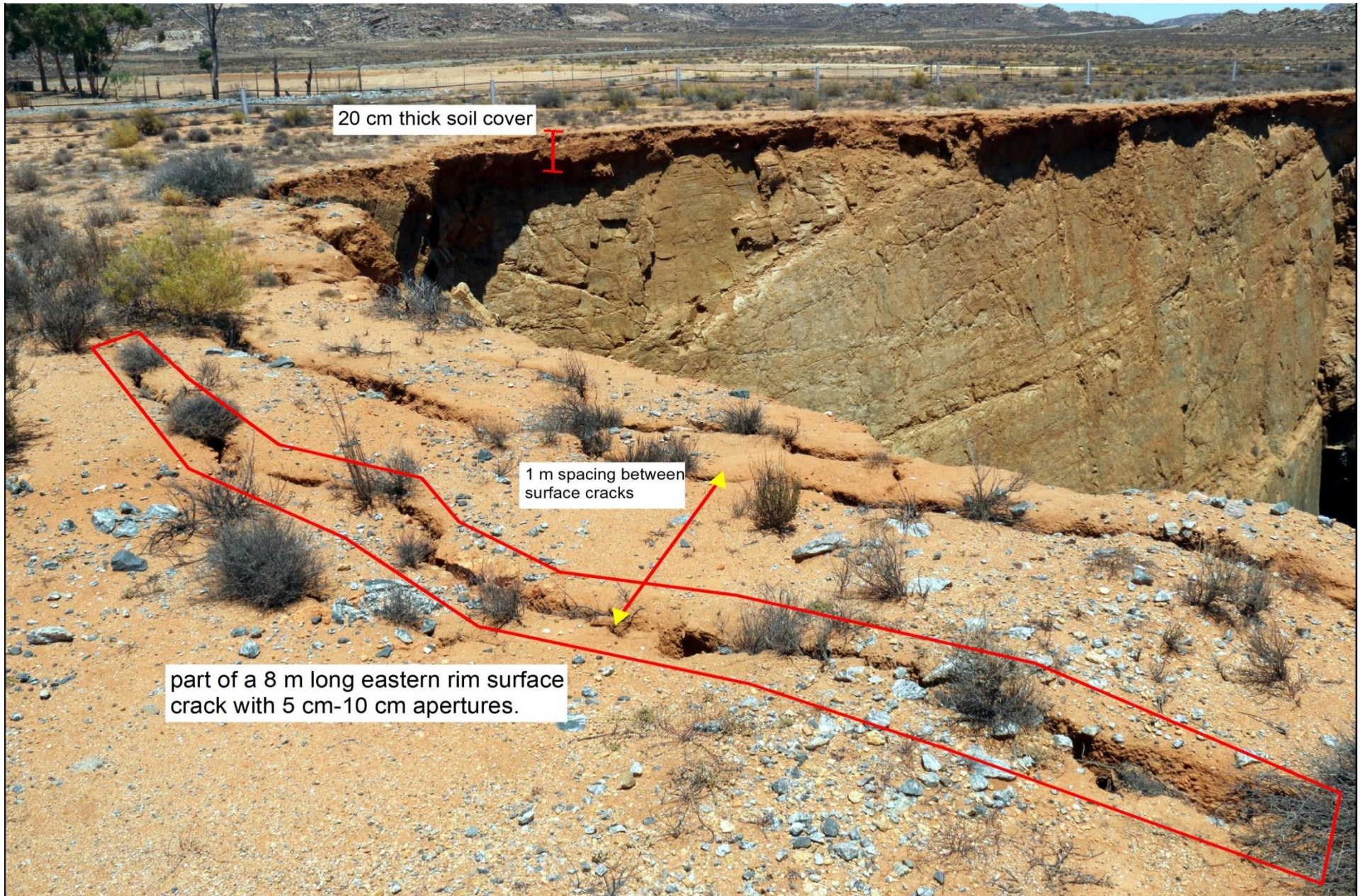


Figure 5. 15: Surface cracks around the eastern rim of the Hoits Mine glory hole.

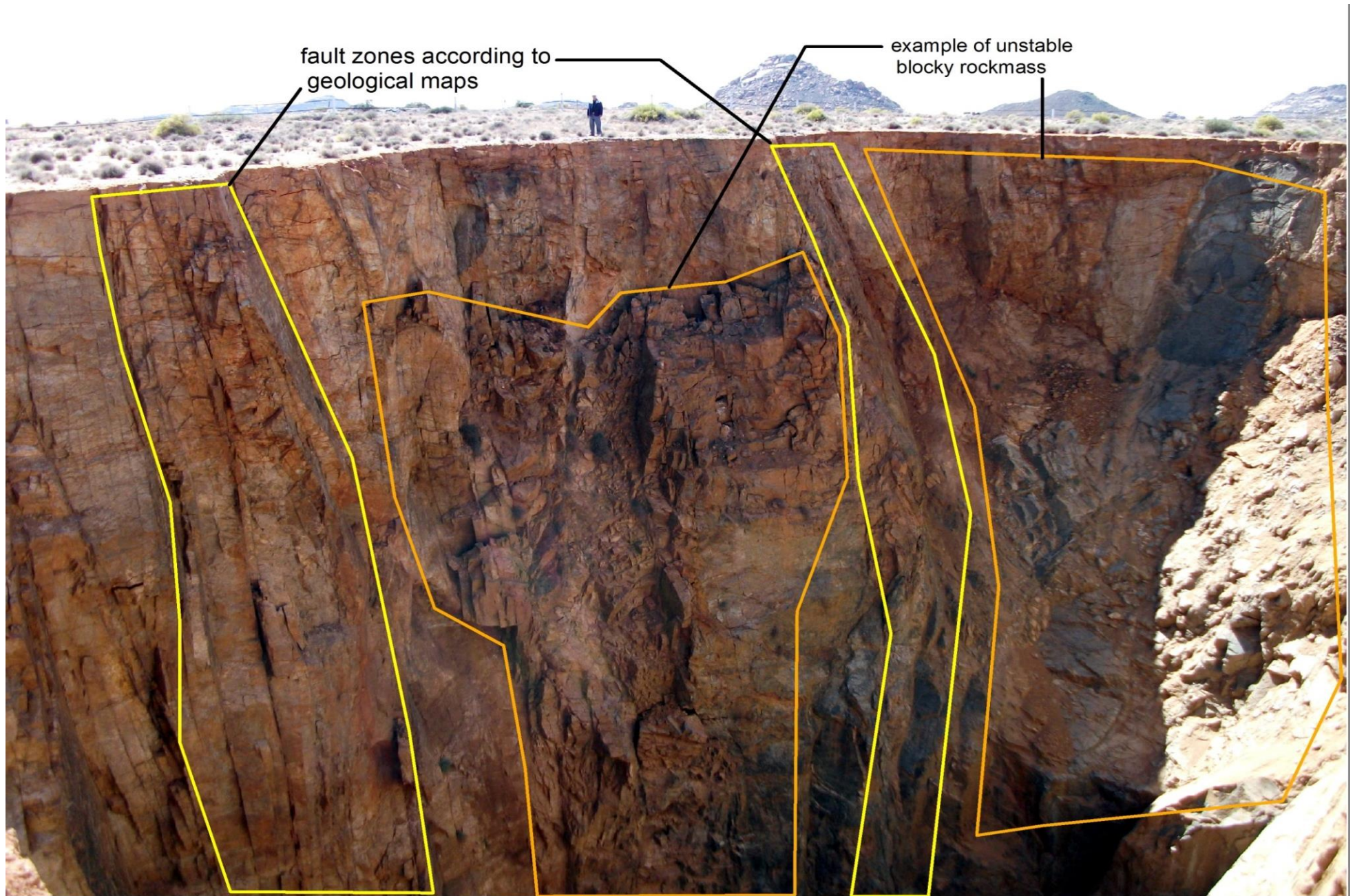


Figure 5. 16: Northern sidewall of the Hoits Mine glory hole

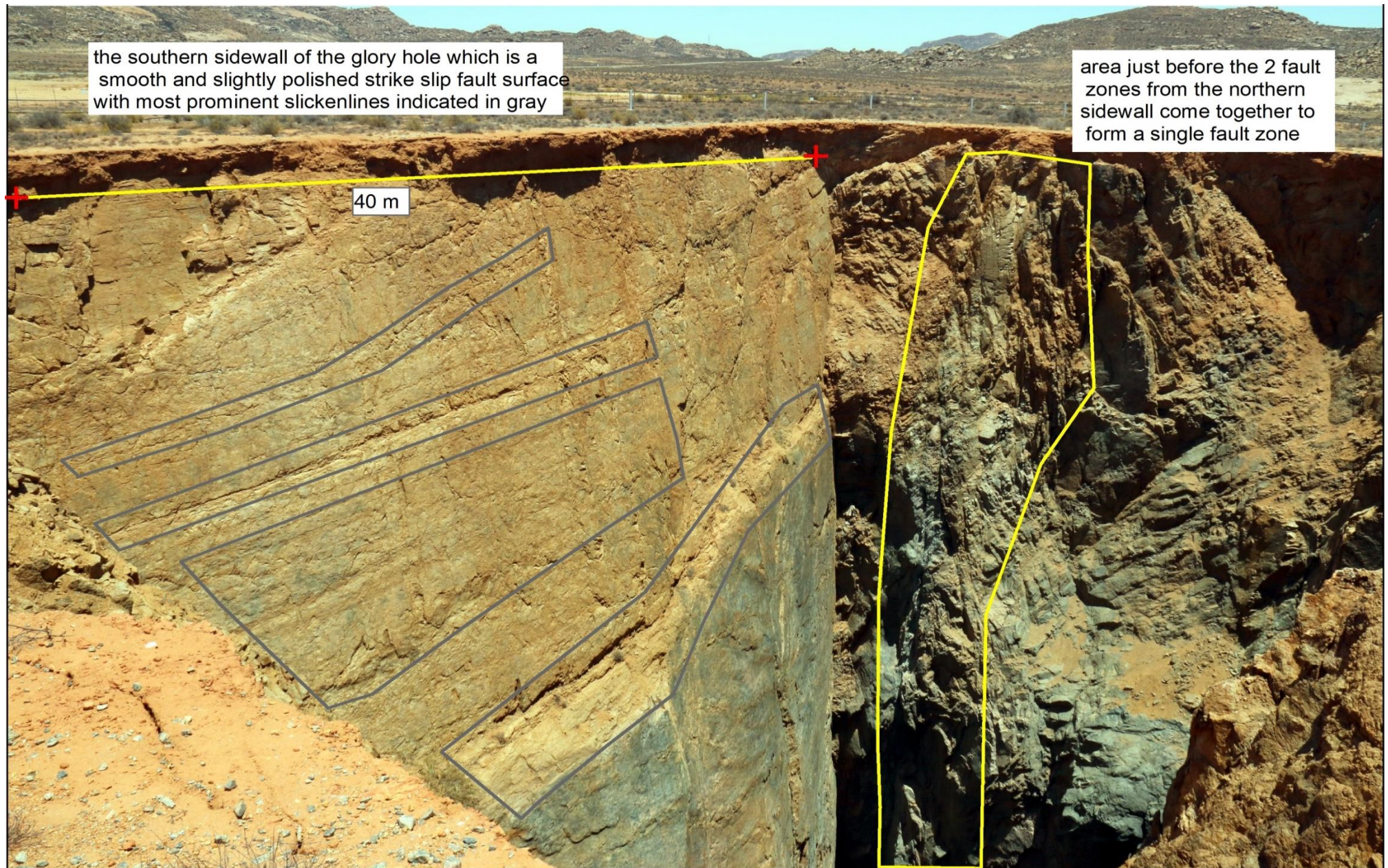


Figure 5. 17: The smooth southern sidewall as well as the western sidewall of the Hoits Mine glory hole.

To summarize, the reasons for the formation of the glory hole are first, the creation of an underground cavity during mining operations. According to Gadd-Claxton (1981) the mining method used to extract ore in the Okiep Copper District was sublevel open stoping, a method that extracts large areas of ore without backfilling the cavities in some cases. As seen in Chapter 4, 1 Stope–2 Stope were not backfilled while, 3 Stope was only partially filled with ‘ex waste dump’ material. The underground cavity below 1 Stope–3 Stope has a vertical mined area of 12 800m², indicating that a large area of rock mass that was initially in equilibrium is now disturbed, as explained by Lee and Abel (1983) in Chapter 2. As seen in Chapter 4, 55 m of overburden separated the ground level and the underground cavity from each other before the glory hole formed. The overburden was fault-weakened by two fault zones composed of closely spaced vertical breccia faults accompanied by steeply dipping joints, creating a broken rock mass likely to be incohesive. According to the geological maps in Chapter 4, chloritisation and kaolinisation is often found along jointed and brecciated zones that can have poor frictional properties. The rock mass is therefore jointed, fault-weakened and altered in certain areas and, as discussed previously, this makes the overburden susceptible to two mechanisms of failure, which are raveling and rock mass caving. Because of the weakened overburden, material was mobilised into the underground cavity by means of gravity. This happened because of the low cohesion of the weakened rock material and the low frictional properties of broken rock. As the caving process progresses the failure front moves towards the surface, until the underground opening span can no longer support its own weight because of low confining pressures. The complete failure of the overburden and the formation of the glory hole happened when the remaining overburden failed and slid down along the smooth, steeply dipping southern sidewall of the glory hole. The shear resistance of the smooth southern sidewall is not sufficient enough to prevent the failure and sliding of the de-stressed overburden, contributing to the formation of the glory hole at ground level. Interdependent factors such as the mining method, depth of extraction, mined area and the structural properties of the rock mass contribute to the formation of the glory hole, while the strength and deformational properties of a rock mass is largely controlled by structural features such as faults, joints and in some cases foliation planes (Singh, 1992). The rock mass around the glory hole continues to be unstable as it progressively caves and gradually expands in size. Progressive caving of the rims can be seen from historical images (Figure 4.12) and the unstable nature of the rims can be seen from the surface cracks surrounding the glory hole in Figure 5.15. Further caving of the rim and sidewalls of the glory hole can therefore be expected. As seen in Chapter 4, Spiessens (2015) also found that the rims and sidewalls of the Hoits Mine glory hole were unstable and, according to him, further caving will happen in the future.

It should be noted, as mentioned in Chapter 4, that the Hoits Mine glory hole is only surrounded by a hoist cable fence (Figure 4.13). This form of glory hole protection is ineffective as anything or anyone can cross this fence with very little effort.

5.4 Wheal Julia East

As seen in Chapter 4, the geological maps portray very little structural information about the rock mass of the Wheal Julia Mine. The main focus of these maps is mineralogical and not structure related. Only a few of maps had the occasional structural feature, for example a single 'calcite slip', but not nearly enough information is available on these maps to get a general indication of the quality of the rock mass. It should be remembered that one of the aims of this research is not to predict when a glory hole at the Bruinhoek populated area might form, but rather whether it is likely that one will form in the future. The reference mines are, therefore used as a proxy to determine the likelihood of a glory hole forming at Bruinhoek. This determination will be done later in this section.

Similar to the Rietberg Mine and Hoits Mine, the drone aerial survey produced a high resolution 2D elevation map, as well as a 2D orthomosaic map. In Chapter 4 it was seen that, although the geological maps portrays very little to no structural information, it outlines the stope dimensions at each level. Overlay images were created, similar to those created for Hoits Mine, with the geological maps and the 2D orthomosaic map from the drone aerial survey. This assisted with the outlining of the underground tunnels and the dimensions of the stope at ground level. An example of such an overlay map can be seen in Figure 5.18 whereas the rest are included in Addendum D. The 2D orthomosaic map can be seen in Figure 5.19, where it indicates the plots and roads of the Bruinhoek populated area and most importantly the area undermined by the Wheal Julia East Mine. The area undermined by the Wheal Julia East stopes is outlined in red, while the tunnels of the Old Cousin Jack level are outlined in green. The colour change is to distinguish between the areas undermined by tunnels and those undermined by stopes. According to Figure 5.19, there are two plots, as well as a section of the frequently used tarred road, that are located directly above the undermined zone. The tarred road leads to old mine houses that are currently occupied by people. This confirms the facts in a report by Spiessens (2015) reporting that plots and houses of the Bruinhoek populated area are located above the undermined zone. The Curry Shaft, now sealed with a concrete block, is located next to the frequently used tarred road, while a section of the Old Cousin Jack mine tunnel also runs below the road. There are at least 20 other plots and houses located 100m or less from the undermined zone. The 2D elevation map is displayed in Figure 5.20 and, on this map the undermined zone has a slightly darker colour than the surroundings north, west and south of it, indicating that the undermined zone is slightly lower lying than its surroundings. However, this is not an indication of subsidence, as the undermined zone is surrounded by hillocks, causing it to be a naturally lower lying area. According to Singh (1992), one of the major ways surface subsidence manifests itself is by fractures and surface cracks. The undermined surface area displays no obvious surface features that manifest subsidence. During the field investigations some of the houses located in close proximity of the undermined zone were inspected and in some instances houses located less than a 100m from the undermined zone showed minor cracks, that can be seen in Figure 5.21. These cracks most likely resulted from minor foundation

subsidence causing the wall to crack and not from surface subsidence of the undermined region. No surface cracks were found around these houses.

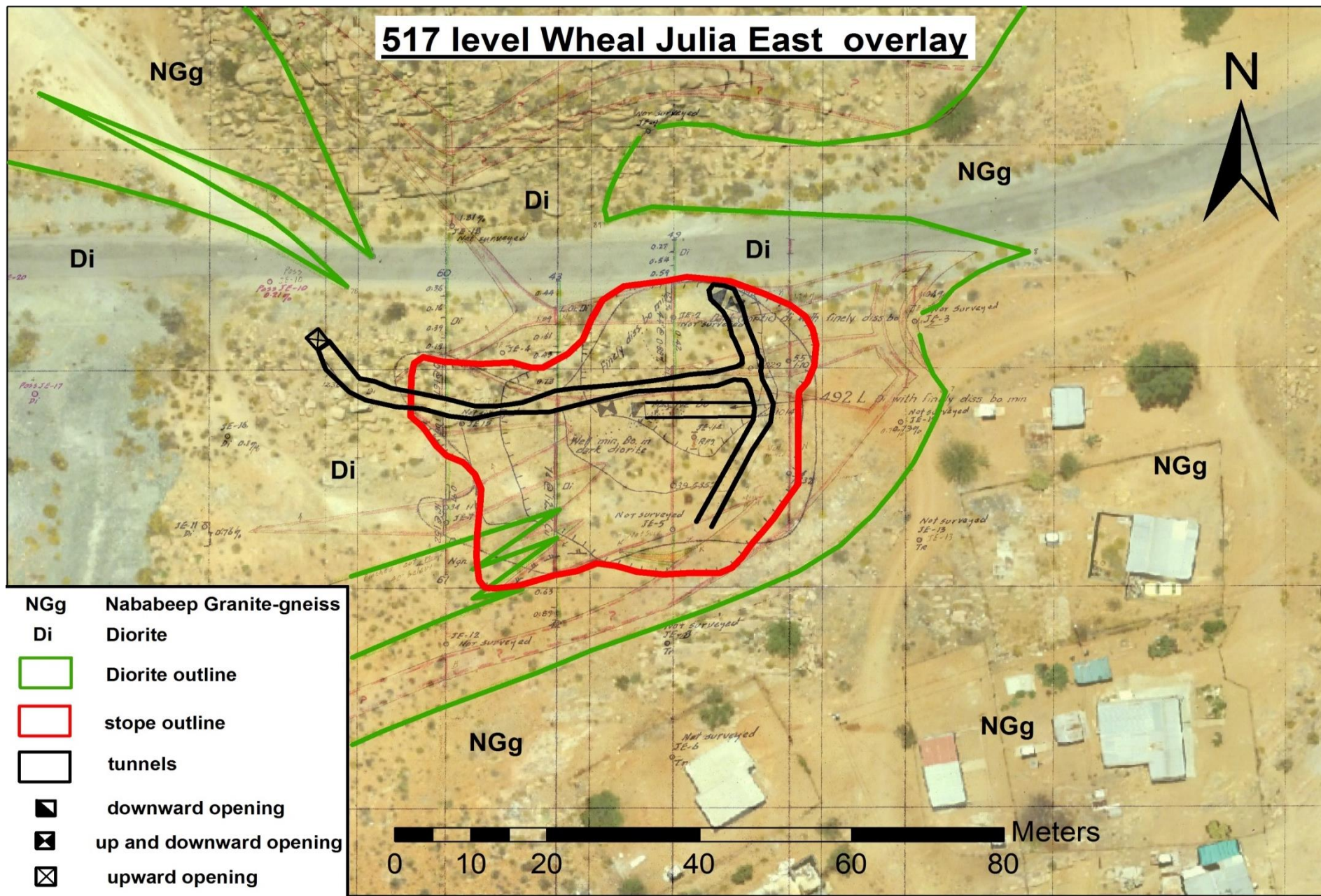


Figure 5. 18: Map of 517 level overlay of Wheal Julia East Mine O'okiep Copper Company, 1955-1959d).

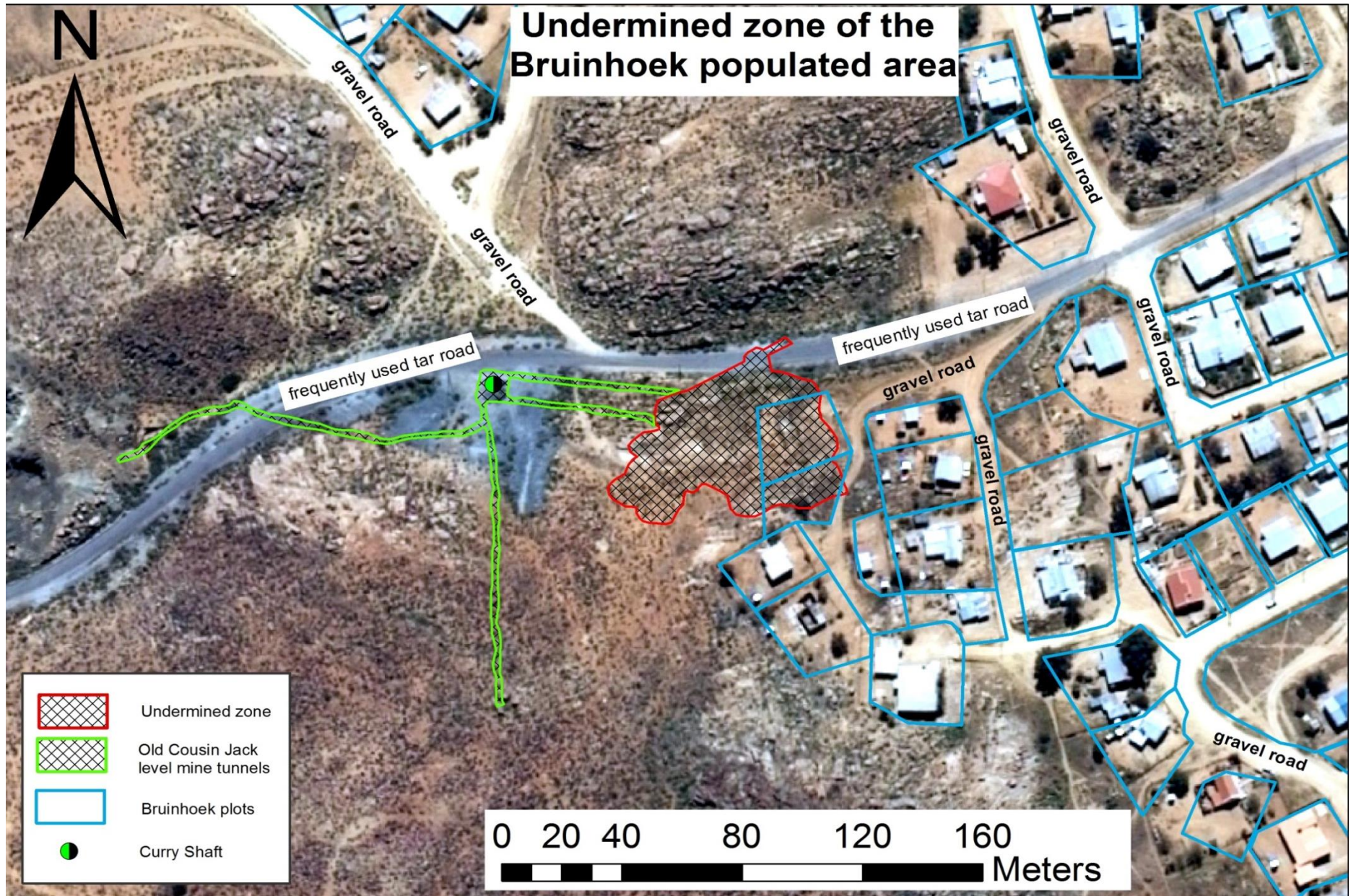


Figure 5. 19: Surface map of a portion of the Bruinhoek populated area and the zone undermined by Wheal Julia East.

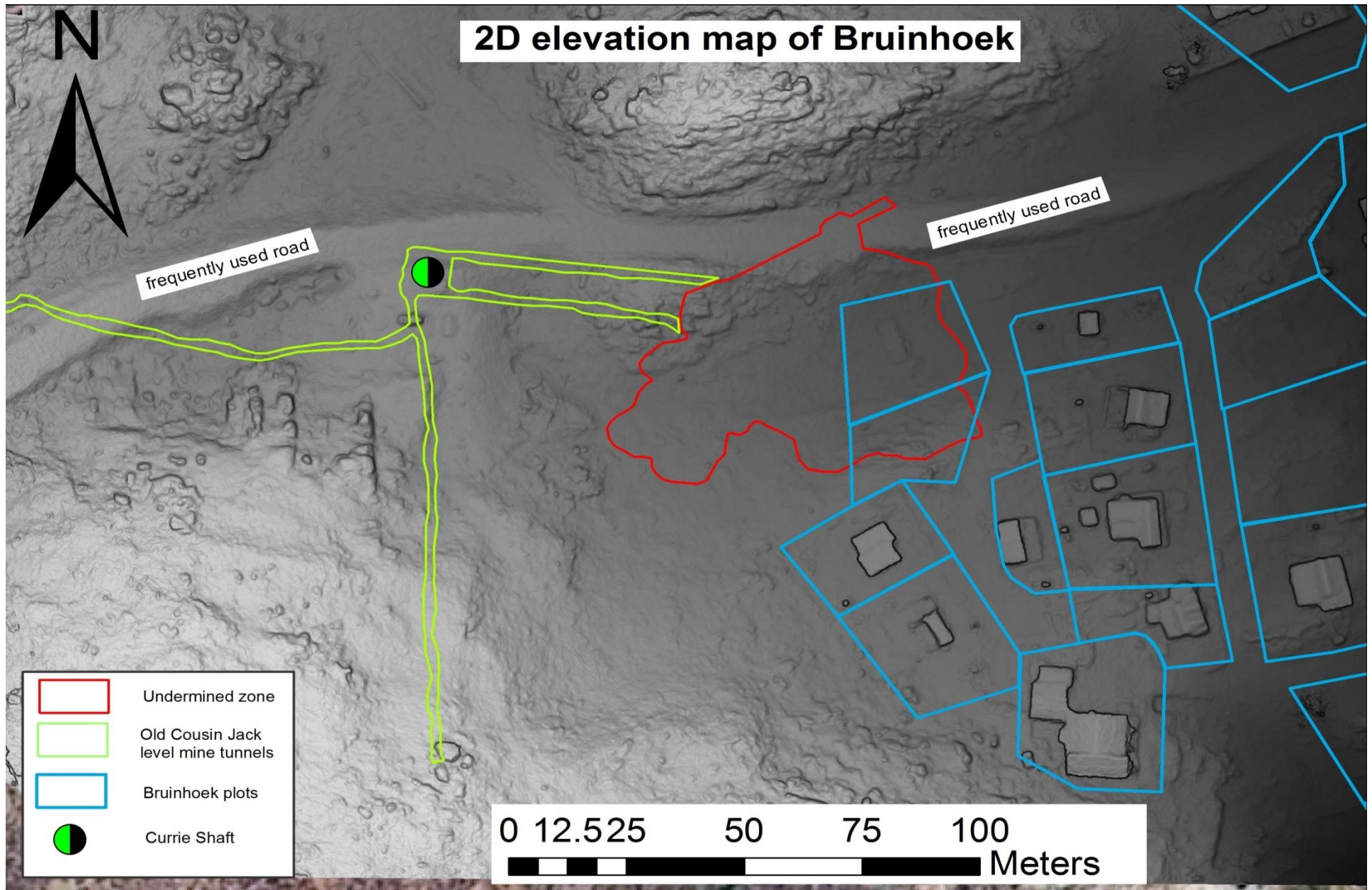


Figure 5. 20: 2D elevation map of the portion of the Bruinhoek populated area undermined by Wheal Julia East.



Figure 5. 21: A wall crack inside one of the Bruinhoek houses.

The biggest difference between Wheal Julia East Mine and the reference mines is the fact that there is no glory hole located at the area undermined by Wheal Julia East. Despite the fact that all the reference mines have glory holes, there is one common factor that the reference mines share and that is the creation of a large underground cavity as a result of the extraction of copper ore. According to Gadd-Claxton (1981), the bulk of the copper ore of the Okiep Copper District was extracted using the sublevel open stoping mining method. As previously explained, this method extracts large bodies of ore and often leaves the cavity behind without backfilling the void. According to the survey maps from Chapter 4, these cavities vary in the size of their vertical mined areas from 12 800m² below 1 Stope-3 Stope at Hoits Mine to as large as 36 000m² at Rietberg Mine. According to these results, as well as field investigations, regardless of the sizes of the underground cavities, glory holes nevertheless form. At the reference mines, copper extraction took place at different depths below ground level. At the Rietberg Mine mining activities occurred as shallow as 15m below ground level, while ore extraction at the Hoits Mine started at 55m below ground level. Regardless of the thickness of the overburden at the reference mines, glory holes still formed. According to geological maps and field investigations there is a second contributing factor to the formation of glory holes that the reference mines share, and that is the presence of persistent geological discontinuities in the overburden rock mass. This is identified as a main contributing factor, as it enables ravelling and rock mass caving of the overburden until the rock mass becomes so de-stressed that it causes the overburden to collapse. At Hoits Mine, for example, it was seen that fault zones caused portions of the rock mass to be incohesive. As seen at West O'okiep Mine and Hoits Mine, ground water and secondary minerals with low frictional properties are also present at reference mines. Both of these have the potential to reduce the shear resistance on discontinuity planes, promoting the failure process of the overburden. Following the formation of the glory holes, instabilities of the sidewalls and rims are present. As discussed previously, this was seen in the form of self-caving and spalling of the sidewalls, rim collapse and surface cracks. As seen in Chapter 4, the presence of prevailing instabilities of and around the glory holes is supported by Spiessens (2015) and Site Plan Consulting (2005). At East O'okiep Mine it was found that the distribution of backfill material into underground cavities was done insufficiently well, indicating that glory holes can be expected at undermined areas that have been backfilled.

To determine whether a glory hole is likely to form at the undermined Bruinhoek populated area, the reference mines are used as a proxy for the Wheal Julia East Mine. It was determined earlier that a characteristic that all the reference mines share is that a glory hole formed at each mine where a large underground cavity had been created. The Wheal Julia East Mine had also had a large underground cavity created during ore extraction, as seen in Figure 4.19 and Figure 5.19. This cavity is located directly below the Bruinhoek populated area. The reference mines also proved, as mentioned previously, that although the thickness of the overburden between ground level and the underground cavity can vary for different reference mines, glory holes form nevertheless. According to Bétournay (2004) the formation of a glory hole can be delayed by the depth the underground cavity is located

below ground level and that this could likely be a reason why a glory hole has not yet formed at the Bruinhoek populated area. Figure 3.1 indicates that the Wheal Julia East Mine is surrounded by, and located in the middle of, the three reference mines. According to Lombaard (1986) the regional geology of the Okiep Copper District is fairly uniform as the major rock types, Granite-gneisses and granites, had all experienced the same deformational events (Table 2.2). Therefore the rock mass characteristics seen at the reference mines could prevail at the Wheal Julia East Mine, which includes persistent steeply dipping joints and breccia faults, fault zones and sub vertical orientated foliation of the country rock. Other characteristics seen at the reference mines, such as secondary minerals and groundwater are also likely to be present at the Wheal Julia East Mine. With a large underground cavity located below the Bruinhoek populated area and based on the results, discussions and analysis of the reference mines, the same mechanisms of failure, ravelling and rock mass caving, of the overburden are likely to occur at the Wheal Julia East Mine. This is also supported by the presence of 'calcite slips', jointing and chloritisation that can be seen on some of the geological maps of the Weal Julia East Mine. These are similar features that were observed at reference mines. As seen in Chapter 4 and during field investigations, instabilities in the form of self-caving sidewalls, rim and overburden collapse, and surface cracks are very common events and features at the reference mines. To add to this, the glory holes all have steep sidewalls and are deeper than a 100m (Site Plan Consulting, 2005) and (Spiessens, 2015), with the only exception being the East O'okiep Mine glory hole that was shallower because of the underground cavity having been backfilled insufficiently. Based on the foregoing it can therefore be accepted that a glory hole at the Bruinhoek populated area would be very deep (possibly as deep as 100 m according to Figure 4.16), will have steep and unstable sidewalls susceptible to self-caving, will be unstable at surface level displaying either overburden and/or ongoing rim collapse with surface cracks outlining the rim of the hole. A glory hole at Bruinhoek would be a massive risk to the people living in the area, as it would obtain a risk rating of at least 20, and possibly 25 using the risk matrix displayed in Table 3.1. A risk rating calculation can be seen in Table B-1.1 in Addendum B if a glory hole should form at Bruinhoek.

Based on the results, discussions and analysis a safety map for the Bruinhoek populated area has been compiled, also utilizing Chapter 17.8 of the Mine Health and Safety Act No. 29 of 1996 (South Africa, Department of Mineral Resources, 2018). As seen in Chapter 2 Chapter 17 of the Act explains that structures erected within a horizontal distance of 100m from an undermined area, including underground tunnels, is regarded as unsafe. The first zone of the safety map is therefore a red area, representing a danger zone, which includes all structures within a horizontal distance of 100 m from the undermined area. The second zone of the safety map represents a yellow area that stretches 50 m beyond the red area, displaying a conservative safety factor based on observations made at the reference mines regarding instabilities prevailing at glory holes in the form of self-caving sidewalls and continuous rim collapse. Examples are the Rietberg Mine, where rock spalling of the sidewalls (Figure 5.6) and overburden collapse (Figure 4.3) were noted, as well as extensive rim collapse and

surface cracks at the Hoits Mine. The yellow area represents an unsafe zone. The third and final zone of the safety map is a green area, which represents a safe zone where locals will not be directly affected by the formation of a glory hole. The green area stretches beyond the yellow unsafe area. The complete safety map can be seen in Figure 5.22.

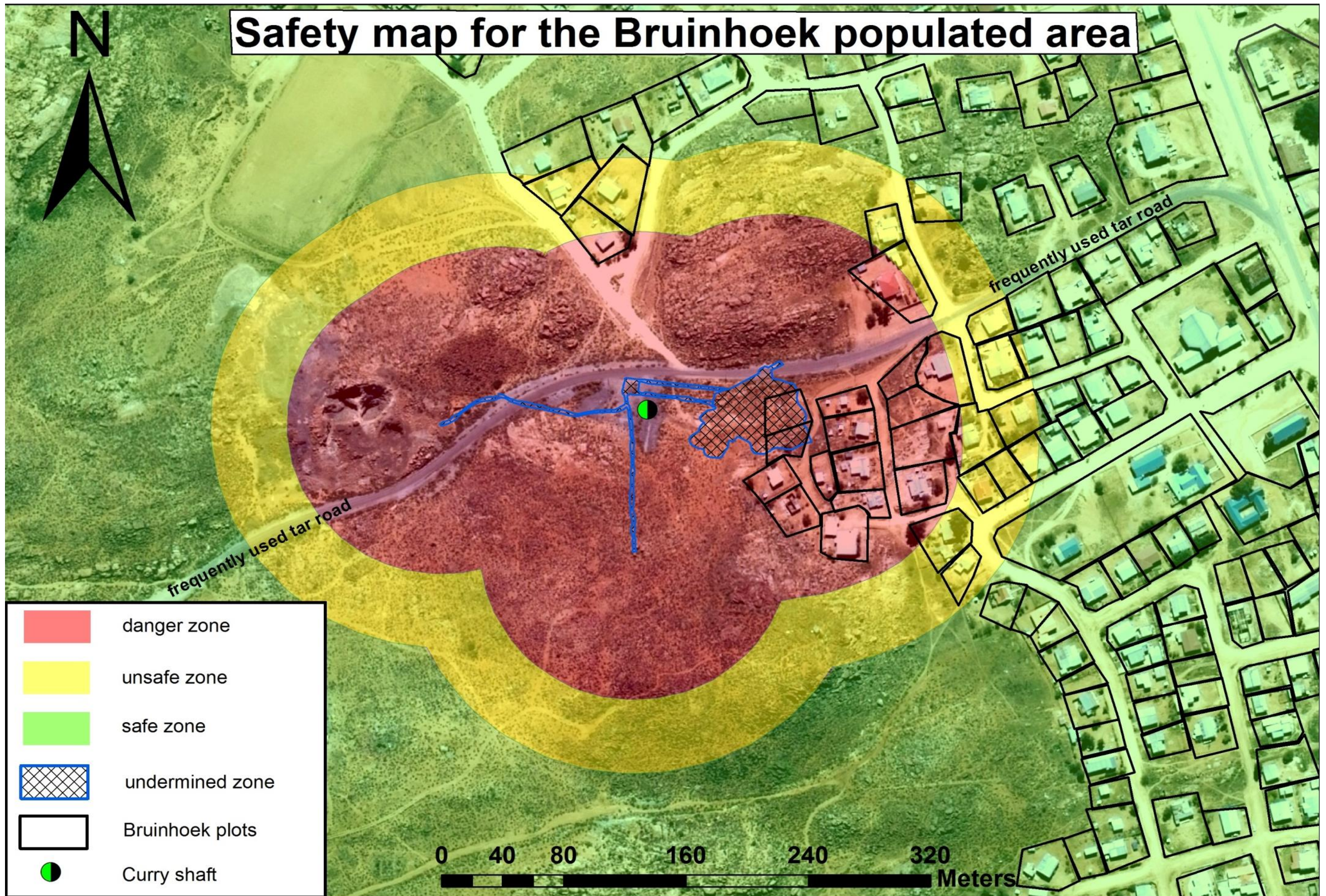


Figure 5. 22: A surface map of the undermined zone in the Bruinhoek populated area divided into different safety zones.

CHAPTER 6: CONCLUSION AND RECOMMENDATIONS

Throughout this research it was seen that one of the legacies of copper mining in the Okiep Copper District is the numerous glory holes that have formed at areas undermined by the stopes of abandoned mines. It was realised early on that deep chemical weathering (chemical decomposition) of the rock mass has had very little, if any, influence on the formation of glory holes as the Okiep Copper District has a Weinert N-value higher than five, indicating that physical weathering dominates (Weinert, 1967). During the investigation process it was discovered that despite the size of the underground cavity or the thickness of the overburden separating the cavity and ground level, glory holes nevertheless formed. The investigation of the reference mines indicated that areas where glory holes are found had one common feature, which was the presence of an underground cavity. It was determined that an underground cavity is not the main contributing factor in the formation of the glory holes, however, for the formation of large (radius of approximately 50m and more) and deep (deeper than a 100m) glory holes to take place, an underground cavity is a requirement. During the discussions and analyses of geological maps, stereonet and field investigations it was found that the main contributing factor to the formation of glory holes in the Okiep Copper District was the presence of persistent steeply dipping discontinuities, which were causing weakening of the overburden in such a way that failure by means of raveling and/or rock mass caving was initiated. Other factors that were found also to contribute to the formation of the holes were secondary minerals with low frictional properties, groundwater, foliation of the rock mass and, sometimes low confining pressures. The failure of the overburden is helped on by the presence of secondary minerals and groundwater, as it lowers the shear resistance on discontinuity planes, and enabling blocky rock mass material to slide off or fall out more easily. As the caving of the overburden takes place, the failure front migrates closer to ground level. As the failure front reaches the upper portion of the overburden, there is a decrease in confining pressures, resulting in the overburden's inability to support its own weight, which leads to the formation of a glory hole at surface level. It was also found that after the formation of the glory holes, rock spalling and caving of the sidewalls, overburden collapse and rim collapse are common. Surface cracks around the rims of the glory holes are also a common sight. These features are examples of the instabilities that prevail after glory hole formation.

The reference mines are, finally, used as a proxy for the Wheal Julia East Mine to determine the probability of glory hole formation in the Bruinhoek populated area. As seen in Figure 4.19 the Wheal Julia East Mine has created a large underground cavity, approximately 12 000m², which is, as mentioned earlier, a requirement for glory holes to form. In Chapter 5 it was emphasised that the Wheal Julia East Mine is surrounded by and located centrally among the reference mines (Figure 3.1), and that the regional geology of the Okiep Copper District is fairly uniform (Lombaard, 1986). In addition to this, when the results, discussions and analyses of the reference mines are taken into account it can, in conclusion, be said that all factors contributing to the formation of glory holes in the

Okiep Copper District such as persistent steeply dipping discontinuities, secondary minerals such as chlorite and kaolinite, ground water and low confining pressures will also be present at the Wheal Julia East Mine. Therefore one or both of the failure mechanisms identified at the reference mines, ravelling and/or rock mass caving, could also occur. Based on this fact, it is concluded that a glory hole could form at the Bruinhoek populated area, and this idea is also supported by Spiessens (2015). It should be remembered that not all the underground cavities of the reference mines were backfilled. However, the stopes in the Okiep Copper District that were backfilled were filled using ineffective methods, as was seen at East O'okiep Mine. Despite some stopes being backfilled, this does not ensure that glory holes will not form. Glory holes could still form, although a cavity has been backfilled. It was also proven by the reference mines that after the formation of glory holes, various forms of instabilities prevail. Instabilities include spalling and self-caving of sidewalls, rim collapse, collapse of overhanging overburden and extensive surface cracks outlining the rims of the glory holes. These instabilities cause the glory holes to grow larger in size as years go by. When a glory hole at Bruinhoek does form, it is almost certain that similar instabilities will prevail, enabling it to enlarge to cover a greater extent of the surface area.

Based on the findings at the reference mines, a glory hole at Bruinhoek will be deep, at the least over 50 m, and will also have steep sidewalls. The implications of such a glory hole would be the destruction of a portion of the frequently used tarred road and the disappearance of the two undermined plots seen in Figure 5.19. As discussed previously, after the formation of glory holes instabilities continue to prevail, emphasising the certainty that a glory hole at Bruinhoek would enlarge as time goes on, covering a gradually larger surface area. This creates an even greater hazard to the locals, as it would cause the disappearance and destruction of houses and can potentially cause injury or the loss of lives.

The following recommendations are made regarding the prevailing conditions at the Bruinhoek populated area:

- The houses of the Bruinhoek populated area located in the red area in Figure 5.22, representing a danger zone, according to the Mine Health and Safety Act, 1996 (Act No. 29 of 1996) must be evacuated, as must the houses that fall into the yellow area (unsafe zone), representing the conservative safety factor, in case of extensive glory hole rim collapse/self-caving. These houses must be evacuated by the NamaKhoi Municipality.
- The frequently used tarred road leading to the mine houses of the Wheal Julia Mine should be realigned in such a way that it does not cut through the yellow or red zones. The realignment of this road must also be done by the NamaKhoi Municipality.

- The undermined zone of the Bruinhoek populated area, which includes zones undermined by both the stopes of the Wheal Julia East Mine as well as its tunnels, must be secured with at least a gabion wall and proper warning notices must be conspicuously displayed to ensure the safety of the local people as far as possible. This must be done by the O'okiep Copper Company as the undermining was done by the company and to date no closing certificate has been issued to the company by the Department of Mineral Resources.
- Regular workshops should be held by the Department of Mineral Resources along with the NamaKhoi Municipality, to inform the public of the prevailing situation and of the procedures that will be followed to make a proper assessment, and to plan the future in co-operation with the affected persons and the community at large.
- A Geotechnical specialist must be appointed by the relevant authorities to do a full scale investigation to determine whether there are any safe areas within the 100m restricted zone for possible relocation (Mine Health and Safety Act No. 29 of 1996 (South Africa, Department of Mineral Resources, 2018)) and should then propose a long term development plan for the area.
- A Risk Management Plan must be drawn up by local authorities in consultation with the Department of Mineral Resources to ensure that the area remains safe for residential purposes.
- The installation of a monitoring system developed and advised by the Geotechnical specialist must be undertaken to ensure the future safety of the undermined zone and the residents.

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ADDENDUM A: ROCK MASS CLASSIFICATION SYSTEMS

A-1: Q-system

Table A-1.1: Certain ranges of Q-values assigned to specific classes and rock mass quality descriptions.

Q-value	Class	Rock mass quality
400 ~ 1000	A	Exceptionally Good
100 ~ 400	A	Extremely Good
40 ~ 100	A	Very Good
10 ~ 40	B	Good
4 ~ 10	C	Fair
1 ~ 4	D	Poor
0.1 ~ 1	E	Very Poor
0.01 ~ 0.1	F	Extremely Poor
0.001 ~ 0.01	G	Exceptionally Poor

Table A-1.2: The RQD table used in the Q-system.

1 RQD (Rock Quality Designation)			RQD
A	Very poor	(> 27 joints per m ³)	0-25
B	Poor	(20-27 joints per m ³)	25-50
C	Fair	(13-19 joints per m ³)	50-75
D	Good	(8-12 joints per m ³)	75-90
E	Excellent	(0-7 joints per m ³)	90-100

Note: i) Where RQD is reported or measured as ≤ 10 (including 0) the value 10 is used to evaluate the Q-value
ii) RQD-intervals of 5, i.e. 100, 95, 90, etc., are sufficiently accurate

Table A-1.3: The joint set number contributing to the final Q-value.

2 Joint set number		J_n
A	Massive, no or few joints	0.5-1.0
B	One joint set	2
C	One joint set plus random joints	3
D	Two joint sets	4
E	Two joint sets plus random joints	6
F	Three joint sets	9
G	Three joint sets plus random joints	12
H	Four or more joint sets, random heavily jointed "sugar cube", etc	15
J	Crushed rock, earth like	20
Note: I) For tunnel intersections, use $3 \times J_n$ II) For portals, use $2 \times J_n$		

Table A-1.4: The joint roughness table used to contribute in the process of calculating the Q-system.

3 Joint Roughness Number		J_r
a) Rock-wall contact, and b) Rock-wall contact before 10 cm of shear movement		
A	Discontinuous joints	4
B	Rough or irregular, undulating	3
C	Smooth, undulating	2
D	Slickensided, undulating	1.5
E	Rough, irregular, planar	1.5
F	Smooth, planar	1
G	Slickensided, planar	0.5
Note: I) Description refers to small scale features and intermediate scale features, in that order		
c) No rock-wall contact when sheared		
H	Zone containing clay minerals thick enough to prevent rock-wall contact when sheared	1
Note: II) Add 1 if the mean spacing of the relevant joint set is greater than 3 m (dependent on the size of the underground opening) III) $J_r = 0.5$ can be used for planar slickensided joints having lineations, provided the lineations are oriented in the estimated sliding direction		

Table A-1.5: The joint alteration number used for determining the Q-value.

4 Joint Alteration Number		Φ_r approx.	J_a
a) Rock-wall contact (no mineral fillings, only coatings)			
A	Tightly healed, hard, non-softening, impermeable filling, i.e., quartz or epidote.		0.75
B	Unaltered joint walls, surface staining only.	25-35°	1
C	Slightly altered joint walls. Non-softening mineral coatings; sandy particles, clay-free disintegrated rock, etc.	25-30°	2
D	Silty or sandy clay coatings, small clay fraction (non-softening).	20-25°	3
E	Softening or low friction clay mineral coatings, i.e., kaolinite or mica. Also chlorite, talc gypsum, graphite, etc., and small quantities of swelling clays.	8-16°	4
b) Rock-wall contact before 10 cm shear (thin mineral fillings)			
F	Sandy particles, clay-free disintegrated rock, etc.	25-30°	4
G	Strongly over-consolidated, non-softening, clay mineral fillings (continuous, but <5 mm thickness).	16-24°	6
H	Medium or low over-consolidation, softening, clay mineral fillings (continuous, but <5 mm thickness).	12-16°	8
J	Swelling-clay fillings, i.e., montmorillonite (continuous, but <5 mm thickness). Value of J_a depends on percent of swelling clay-size particles.	6-12°	8-12
c) No rock-wall contact when sheared (thick mineral fillings)			
K	Zones or bands of disintegrated or crushed rock. Strongly over-consolidated.	16-24°	6
L	Zones or bands of clay, disintegrated or crushed rock. Medium or low over-consolidation or softening fillings.	12-16°	8
M	Zones or bands of clay, disintegrated or crushed rock. Swelling clay. J_a depends on percent of swelling clay-size particles.	6-12°	8-12
N	Thick continuous zones or bands of clay. Strongly over-consolidated.	12-16°	10
O	Thick, continuous zones or bands of clay. Medium to low over-consolidation.	12-16°	13
P	Thick, continuous zones or bands with clay. Swelling clay. J_a depends on percent of swelling clay-size particles.	6-12°	13-20

Table A-1.6: Joint water reduction factor used in the Q-system.

5 Joint Water Reduction Factor		J_w
A	Dry excavations or minor inflow (humid or a few drips)	1.0
B	Medium inflow, occasional outwash of joint fillings (many drips/"rain")	0.66
C	Jet inflow or high pressure in competent rock with unfilled joints	0.5
D	Large inflow or high pressure, considerable outwash of joint fillings	0.33
E	Exceptionally high inflow or water pressure decaying with time. Causes outwash of material and perhaps cave in	0.2-0.1
F	Exceptionally high inflow or water pressure continuing without noticeable decay. Causes outwash of material and perhaps cave in	0.1-0.05
Note: i) Factors C to F are crude estimates. Increase J_w if the rock is drained or grouting is carried out ii) Special problems caused by ice formation are not considered		

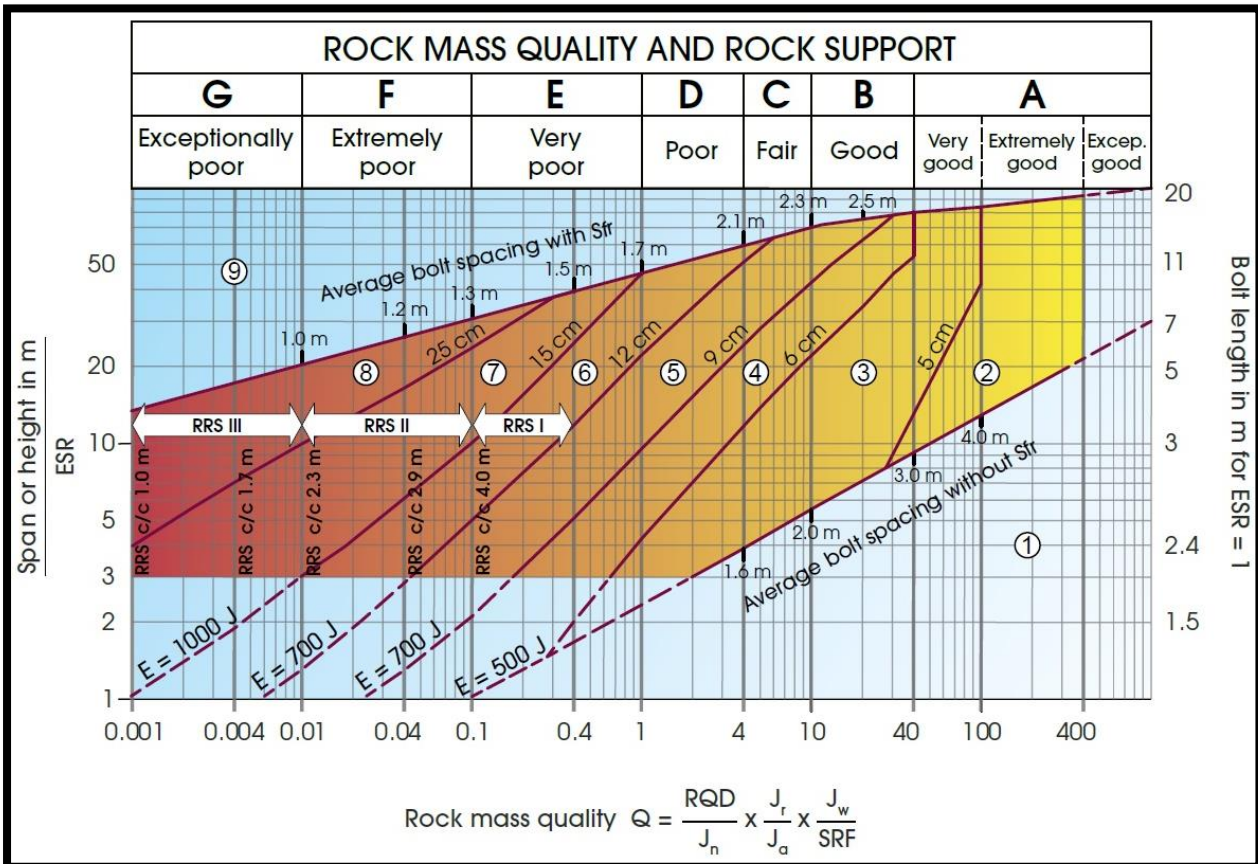
Table A-1.7: The stress reduction factor contributing to the final Q-value.

6 Stress Reduction Factor			SRF
a) Weak zones intersecting the underground opening, which may cause loosening of rock mass			
A	Multiple occurrences of weak zones within a short section containing clay or chemically disintegrated, very loose surrounding rock (any depth), or long sections with incompetent (weak) rock (any depth). For squeezing, see 6L and 6M		10
B	Multiple shear zones within a short section in competent clay-free rock with loose surrounding rock (any depth)		7.5
C	Single weak zones with or without clay or chemical disintegrated rock (depth $\leq 50\text{m}$)		5
D	Loose, open joints, heavily jointed or "sugar cube", etc. (any depth)		5
E	Single weak zones with or without clay or chemical disintegrated rock (depth $> 50\text{m}$)		2.5
Note: i) Reduce these values of SRF by 25-50% if the weak zones only influence but do not intersect the underground opening			
b) Competent, mainly massive rock, stress problems			
		σ_c / σ_1	σ_θ / σ_c
F	Low stress, near surface, open joints	>200	<0.01
G	Medium stress, favourable stress condition	200-10	0.01-0.3
H	High stress, very tight structure. Usually favourable to stability. May also be unfavourable to stability dependent on the orientation of stresses compared to jointing/weakness planes*	10-5	0.3-0.4
J	Moderate spalling and/or slabbing after > 1 hour in massive rock	5-3	0.5-0.65
K	Spalling or rock burst after a few minutes in massive rock	3-2	0.65-1
L	Heavy rock burst and immediate dynamic deformation in massive rock	<2	>1
Note: ii) For strongly anisotropic virgin stress field (if measured): when $5 \leq \sigma_1 / \sigma_3 \leq 10$, reduce σ_c to $0.75 \sigma_c$. When $\sigma_1 / \sigma_3 > 10$, reduce σ_c to $0.5 \sigma_c$, where σ_c = unconfined compression strength, σ_1 and σ_3 are the major and minor principal stresses, and σ_θ = maximum tangential stress (estimated from elastic theory)			
iii) When the depth of the crown below the surface is less than the span; suggest SRF increase from 2.5 to 5 for such cases (see F)			
c) Squeezing rock: plastic deformation in incompetent rock under the influence of high pressure			
		σ_θ / σ_c	SRF
M	Mild squeezing rock pressure	1-5	5-10
N	Heavy squeezing rock pressure	>5	10-20
Note: iv) Determination of squeezing rock conditions must be made according to relevant literature (i.e. Singh et al., 1992 and Bhasin and Grimstad, 1996)			
d) Swelling rock: chemical swelling activity depending on the presence of water			SRF
O	Mild swelling rock pressure		5-10
P	Heavy swelling rock pressure		10-15

Table A-1.8: The type of excavation used to calculate the Q-value.

7 Type of excavation		ESR
A	Temporary mine openings, etc.	ca. 3-5
B	Vertical shafts*: i) circular sections ii) rectangular/square section * Dependant of purpose. May be lower than given values.	ca. 2.5 ca. 2.0
C	Permanent mine openings, water tunnels for hydro power (exclude high pressure penstocks), water supply tunnels, pilot tunnels, drifts and headings for large openings.	1.6
D	Minor road and railway tunnels, surge chambers, access tunnels, sewage tunnels, etc.	1.3
E	Power houses, storage rooms, water treatment plants, major road and railway tunnels, civil defence chambers, portals, intersections, etc.	1.0
F	Underground nuclear power stations, railways stations, sports and public facilities, factories, etc.	0.8
G	Very important caverns and underground openings with a long lifetime, ≈ 100 years, or without access for maintenance.	0.5

Table A-1.9: The rock support chart



Support categories

- ① Unsupported or spot bolting
- ② Spot bolting, **SB**
- ③ Systematic bolting, fibre reinforced sprayed concrete, 5-6 cm, **B+Sfr**
- ④ Fibre reinforced sprayed concrete and bolting, 6-9 cm, **Sfr (E500)+B**
- ⑤ Fibre reinforced sprayed concrete and bolting, 9-12 cm, **Sfr (E700)+B**
- ⑥ Fibre reinforced sprayed concrete and bolting, 12-15 cm + reinforced ribs of sprayed concrete and bolting, **Sfr (E700)+RRS I +B**
- ⑦ Fibre reinforced sprayed concrete >15 cm + reinforced ribs of sprayed concrete and bolting, **Sfr (E1000)+RRS II+B**
- ⑧ Cast concrete lining, **CCA** or **Sfr (E1000)+RRS III+B**
- ⑨ Special evaluation


Bolts spacing is mainly based on $\varnothing 20$ mm


E = Energy absorption in fibre reinforced sprayed concrete


ESR = Excavation Support Ratio

Areas with dashed lines have no empirical data

RRS - spacing related to Q-value

 **Si30/6 $\varnothing 16$ - $\varnothing 20$ (span 10m)**
D40/6+2 $\varnothing 16$ -20 (span 20m)

 Si35/6 $\varnothing 16$ -20 (span 5m)
D45/6+2 $\varnothing 16$ -20 (span 10m)
D55/6+4 $\varnothing 20$ (span 20m)

 D40/6+4 $\varnothing 16$ -20 (span 5 m)
D55/6+4 $\varnothing 20$ (span 10 m)
Special evaluation (span 20 m)

Si30/6 = Single layer of 6 rebars,
30 cm thickness of sprayed concrete

D = Double layer of rebars

$\varnothing 16$ = Rebar diameter is 16 mm

c/c = RSS spacing, centre - centre

Figure A-1.1: Support categories for the rock support chart.

A-2: Rock Mass Rating

Table A-2.1: Rock Mass Rating system

A. CLASSIFICATION PARAMETERS AND THEIR RATINGS									
Parameter			Range of values						
1	Strength of intact rock material	Point-load strength index	>10 MPa	4-10 MPa	2-4 MPa	1-2 MPa	For this low range –uniaxial compressive test is preferred		
		Uniaxial comp. strength	>250 MPa	100-250 MPa	50-100 MPa	25-50 MPa	5-25 MPa	1-5 MPa	<1 MPa
	Rating	15	12	7	4	2	1	0	
2	Drift cone Quality RQD		90% - 100%	75% - 90%	50% - 75%	25% - 50%	<25%		
	Rating		20	17	13	8	3		
3	Spacing of discontinuities		>2m	0.6–2m	200–600mm	60-200mm	<60mm		
	Rating		20	15	10	8	5		
4	Condition of discontinuities (See E)		Very rough surfaces Not continuous No separation Unweathered wall rock	Slightly rough surfaces Separation <1mm Slightly weathered walls	Slightly rough surfaces Separation <1mm Highly weathered walls	Slicksided surfaces or Gouge <5mm thick or Separation 1-5mm, continuous	Soft gouge >5mm thick Or Separation >5mm Continuous		
5	Ground water	Inflow per 10m tunnel length (l/m)	None	<10	10-25	25-125	>125		
		(Joint water press)/ (Major principal??)	0	<0.1	0.1-0.2	0.2-0.5	>0.5		
		General condition	Completely dry	Damp	Wet	Dripping	Flowing		
	Rating		15	10	7	4	0		
B. RATING ADJUSTMENT FOR DISCONTINUITY ORIENTATIONS (See F)									
Strike and dip orientations			Very favourable	Favourable	Fair	Unfavourable	Very Unfavourable		
Ratings	Tunnels & mines		0	-2	-5	-10	-12		
	Foundations		0	-2	-7	-15	-25		
	Slopes		0	-5	-25	-50			
C. ROCK MASS CLASSES DETERMINED FROM TOTAL RATINGS									
Rating			100←81	80←61	60←41	40←21	<21		
Class number			I	II	III	IV	V		
Description			Very good rock	Good rock	Fair rock	Poor rock	Very poor rock		
D. MEANING OF ROCK CLASSES									
Class number			I	II	III	IV	V		
Average stand-up time			20 years for 15m span	1 year for 10m span	1 week for 5m span	10 hours for 2.5m span	30 min for 1m span		
Cohesion of rock mass (kPa)			>400	300-400	200-300	100-200	<100		
Friction angle of rock mass (deg)			>45	35-45	25-35	15-25	<15		
E. GUIDELINES FOR CLASSIFICATION OF DISCONTINUITY conditions									
Discontinuity length (persistence)			<1m	1-3m	3-10m	10-20m	>20m		
Rating			6	4	2	1	0		
Separation (aperture)			None	<0.1mm	0.1-1.0mm	1-5mm	>5mm		
Rating			6	5	4	1	0		
Roughness			Very rough	Rough	Slightly rough	Smooth	Slicksided		
Rating			6	5	3	1	0		
Infilling (gouge)			None	Hard filling <5mm	Hard filling >5mm	Soft filling <5mm	Soft filling >5mm		
Rating			6	4	2	2	0		
Weathering			Unweathered	Slightly weathered	Moderately weathered	Highly weathered	Decomposed		
Rating			6	5	3	1	0		
F. EFFECT OF DISCONTINUITY STRIKE AND DIP ORIENTATION IN TUNNELING									
Strike perpendicular to tunnel axis					Strike parallel to tunnel axis				
Drive with dip-Dip 45-90°			Drive with dip-Dip 20-45°		Dip 45-90°		Dip 20-45°		
Very favourable			Favourable		Very favourable		Fair		
Drive against dip-Dip 45-90°			Drive against dip-Dip 20-45°		Dip 0-20-Irrespective of strike*				
Fair			Unfavourable		Fair				

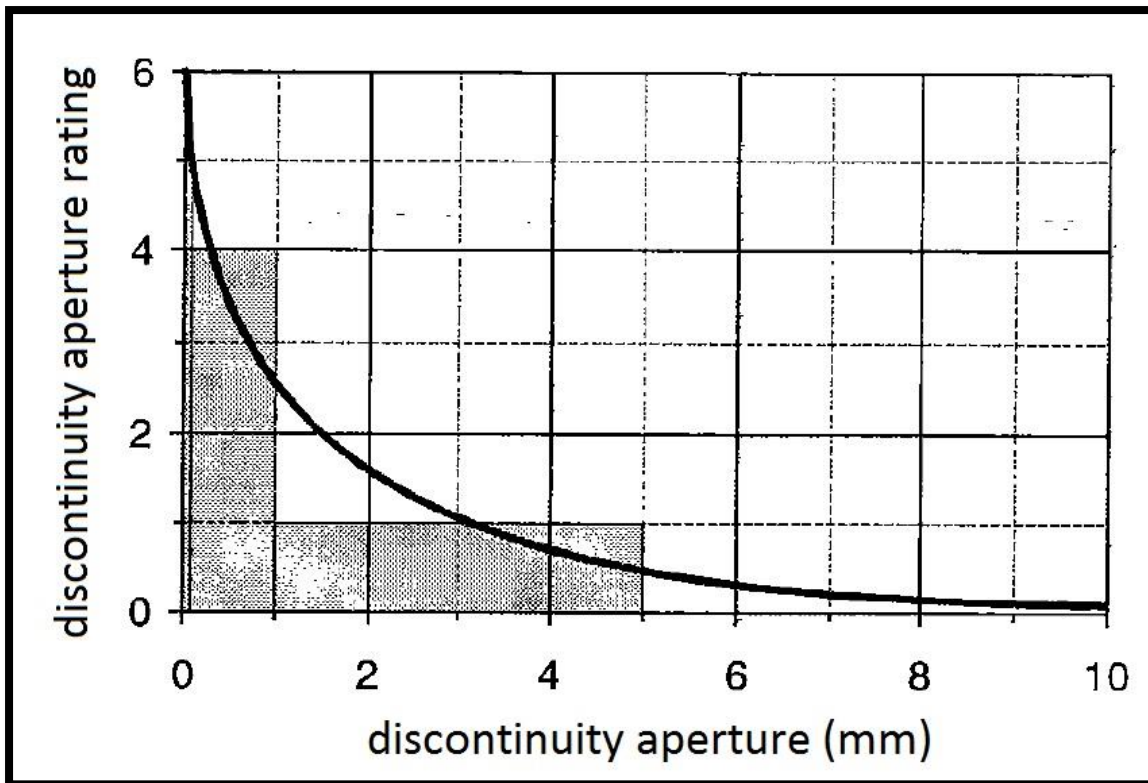


Figure A-2.1: The joint aperture measurement corresponding to discontinuity aperture rating values.

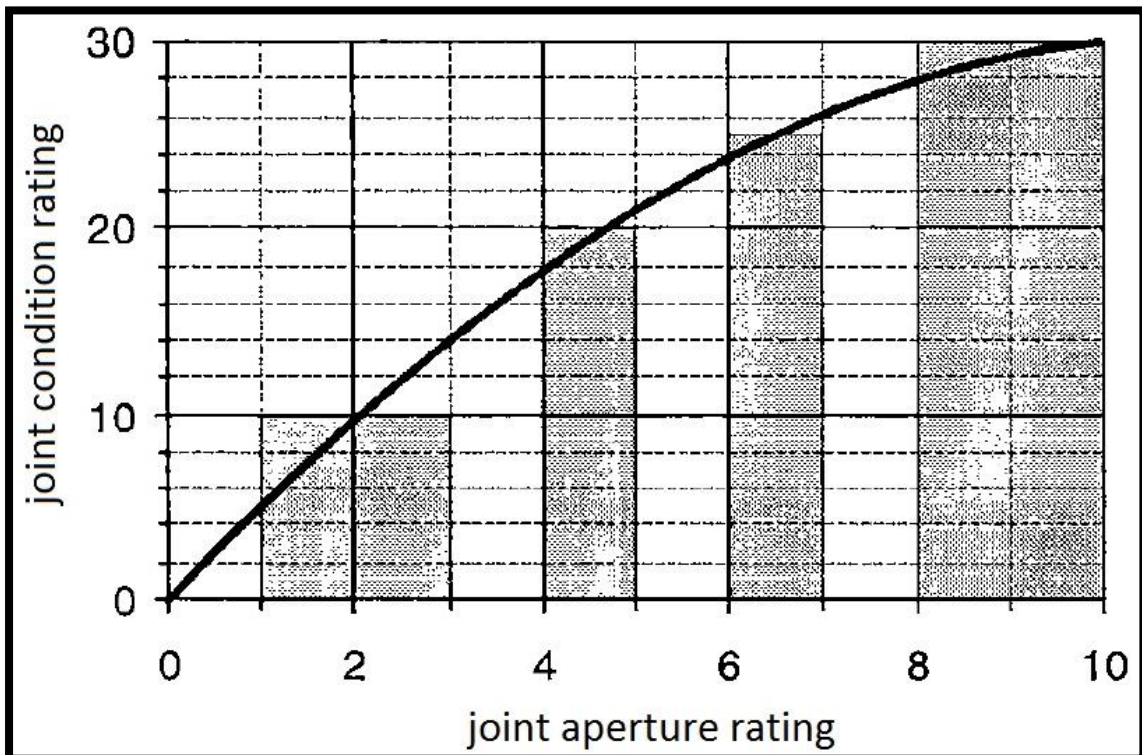


Figure A-2.2: Joint aperture ratings corresponding to the final joint condition ratings.

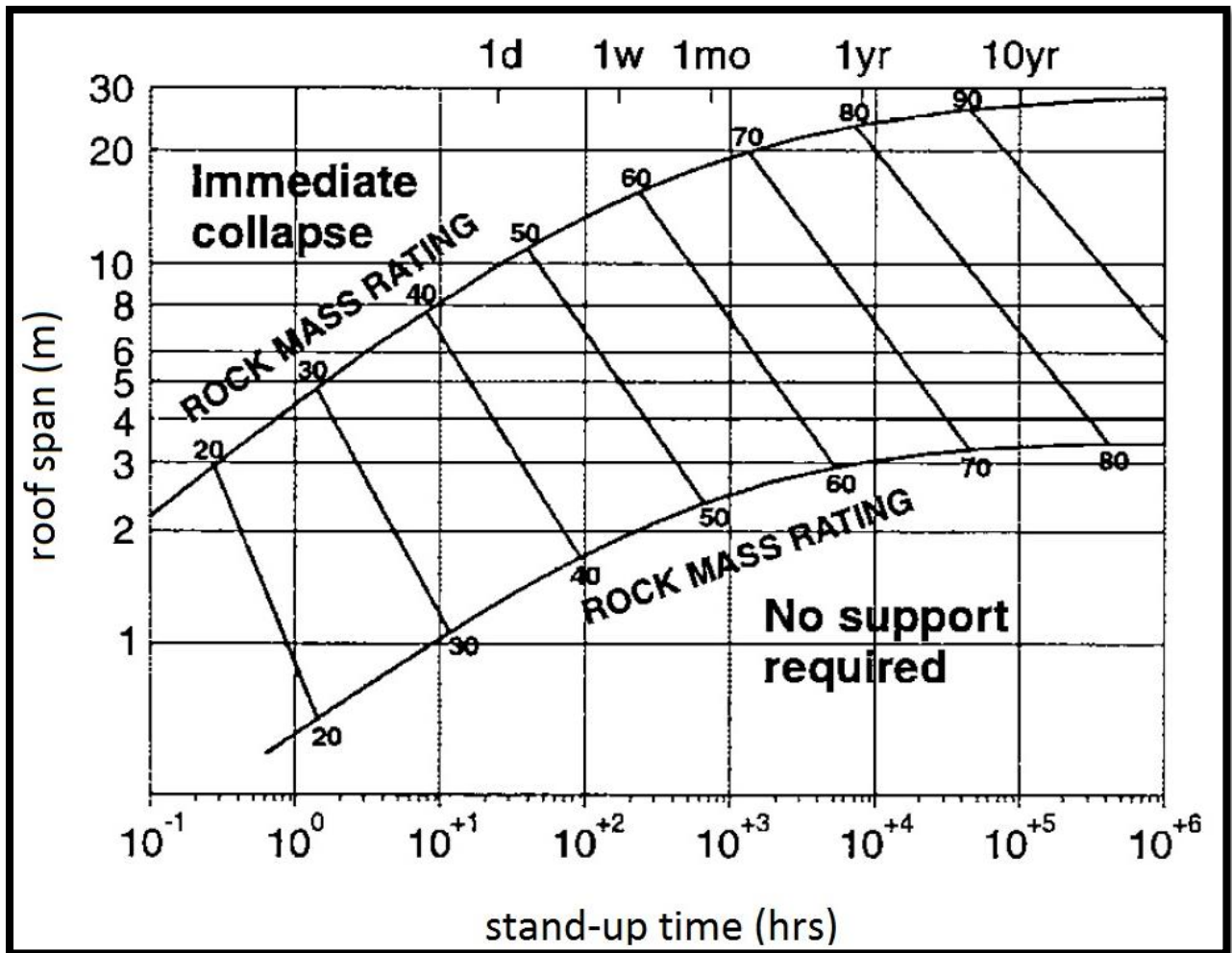


Figure A-2.3: The relationship between stand-up time and span for various RMR values.

ADDENDUM B: GLORY HOLE RISK RATINGS

Table B-1. 1: The risk rating calculations for glory holes at specific mines.

Glory hole (where)	Description	Hazard probability	Hazard intensity (HI)	Risk rating (HP x HI)
Rietberg Mine	Distance: >100 m from human activity	1	5	5
	Protection: moderate (gabion wall)			
	Sidewalls: vertical and unstable			
	Depth: very deep (> 200 m) no chance of exit			
West O'okiep Mine	Distance: < 100 m from human activity	4	3	12
	Protection: ineffective (concrete palisade)			
	Sidewalls: some steep and unstable as well as rims			
	Depth: 150 m, but water filled (difficult exit)			
East O'okiep Mine	Distance: < 100 m from human activity	5	2	10
	Protection: no protection			
	Sidewalls: shallow slopes and unstable			
	Depth: shallow (4 m-5 m) potential for exit			
Hoits Mine	Distance: < 100 m from human activity	4	5	20
	Protection: ineffective (Hoist cable fence)			
	Sidewalls: vertical and unstable			
	Depth: deep (>100 m) no chance of exit			
Wheal Julia East Mine (should a glory hole form in the future)	Distance: < 100 m from human activity	5	4 or 5	20 or 25
	Protection: no protection			
	Sidewalls: either vertical or steep and unstable			
	Depth: most likely around 100 m or deeper			

Hazard probability	Description
Rare (1)	Located further than 100 m from human activity with moderate protection (Gabion wall).
Unlikely (2)	Located further than 100 m from human activity with ineffective protection (Hoist cable fence or concrete palisade).
Possible (3)	Located 100 m or closer to human activity with moderate protection (Gabion wall).
Likely (4)	Located 100 m or closer to human activity with ineffective protection (Hoist cable fence or concrete palisade).
Almost certain (5)	Located 100 m or closer to human activity with no form of protection.

Hazard intensity	Description
Insignificant (1)	A shallow glory hole with stable and shallow slopes and an easy exit
Minor (2)	A shallow glory hole with unstable and shallow slopes and a potential for exit.
Moderate (3)	A deep glory hole with some unstable and shallow/steep slopes and a difficult exit
Major (4)	A deep glory hole with steep and unstable slopes and no chance of exit.
Catastrophic (5)	A deep glory hole with vertical and unstable slopes and no chance of exit.

Risk rating	
high risk	
medium risk	
low risk	

ADDENDUM C: GEOLOGICAL MAPS

C-1: Hoits Mine

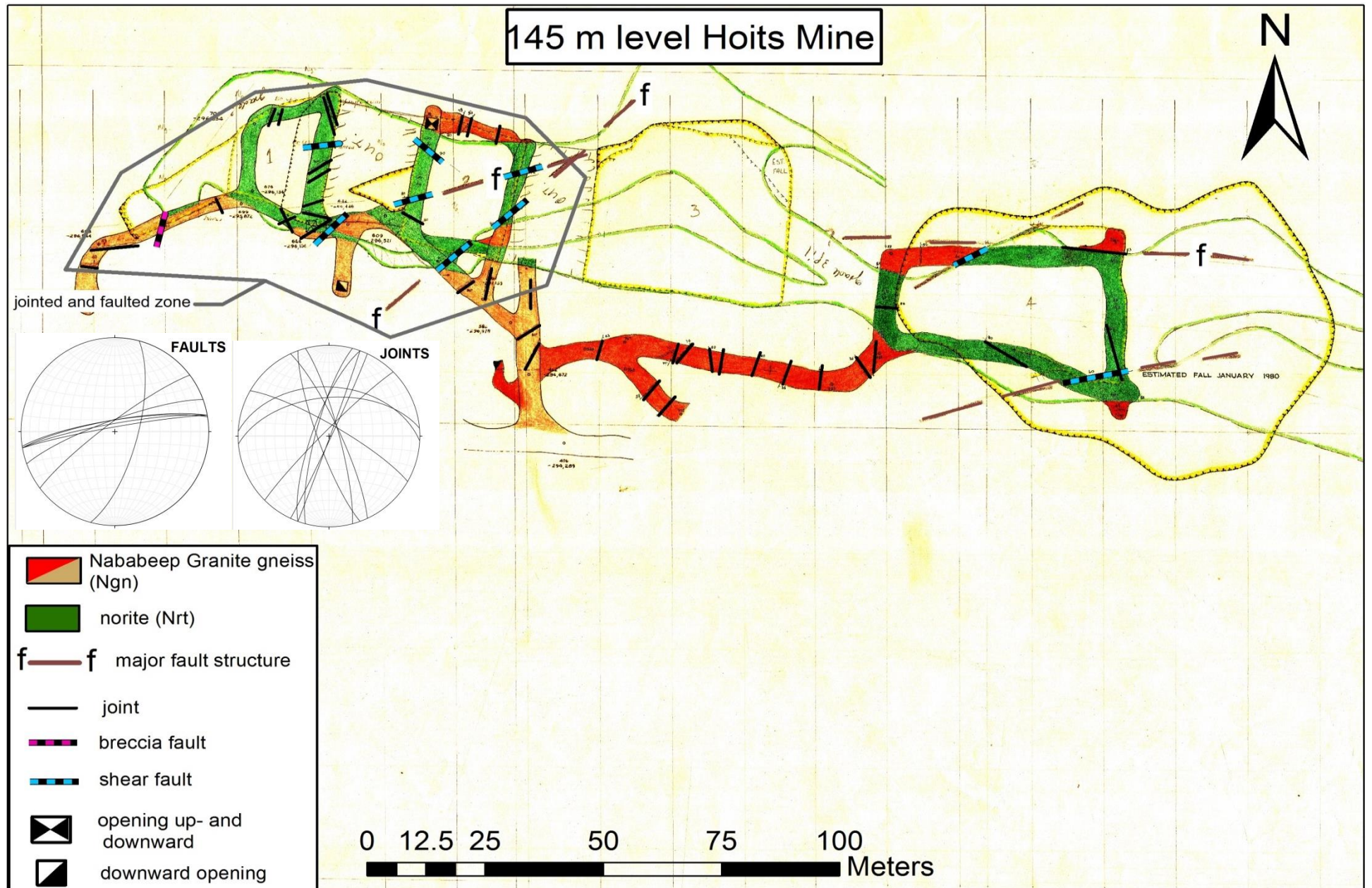


Figure C-1. 2: Map of 145 m level of the Hoits Mine (O'okiep Copper Company, 1975-1993i).

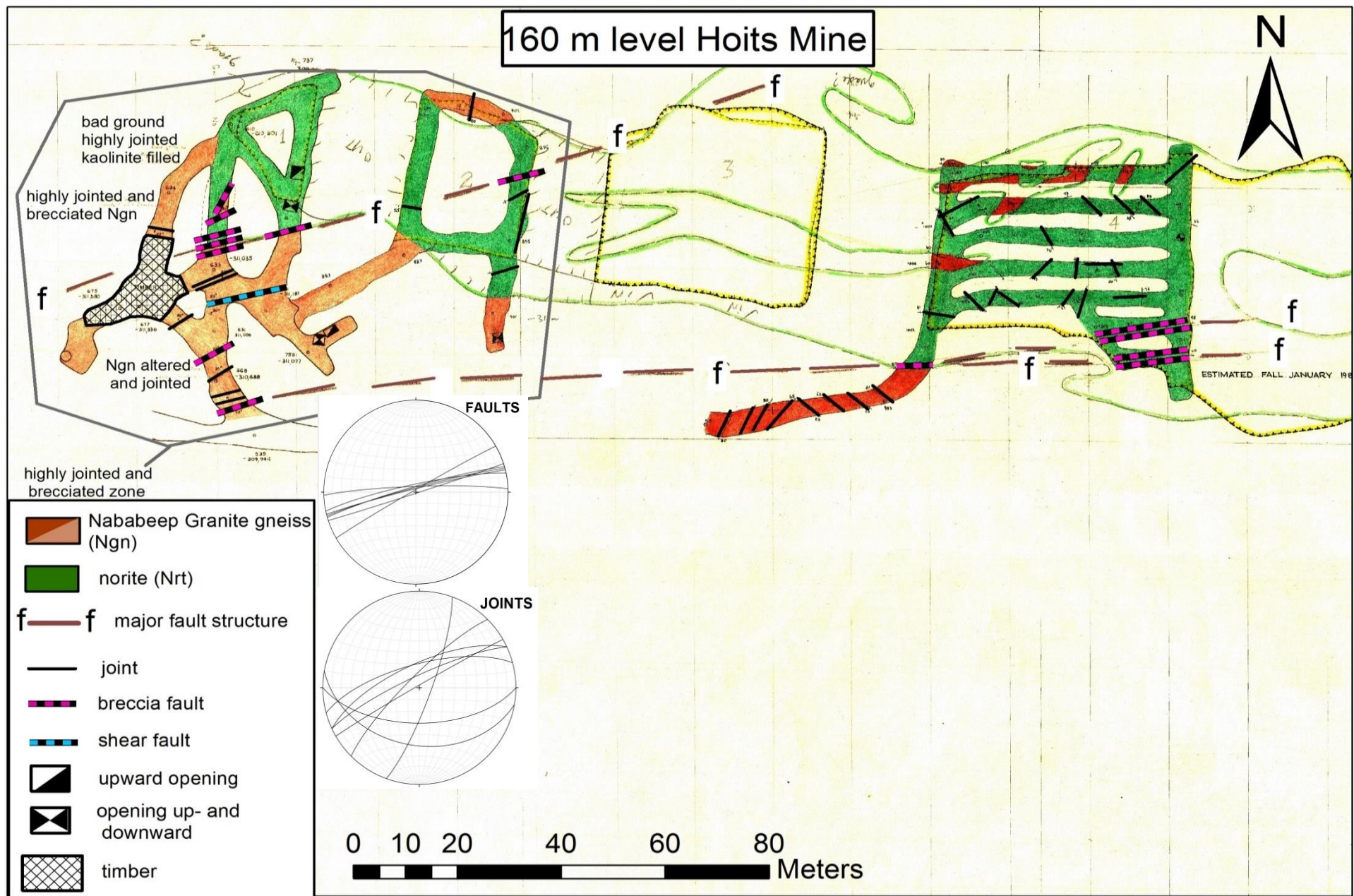


Figure C-1. 3: Map of 160 m level of the Hoits Mine (O'okiep Copper Company, 1975-1993j).

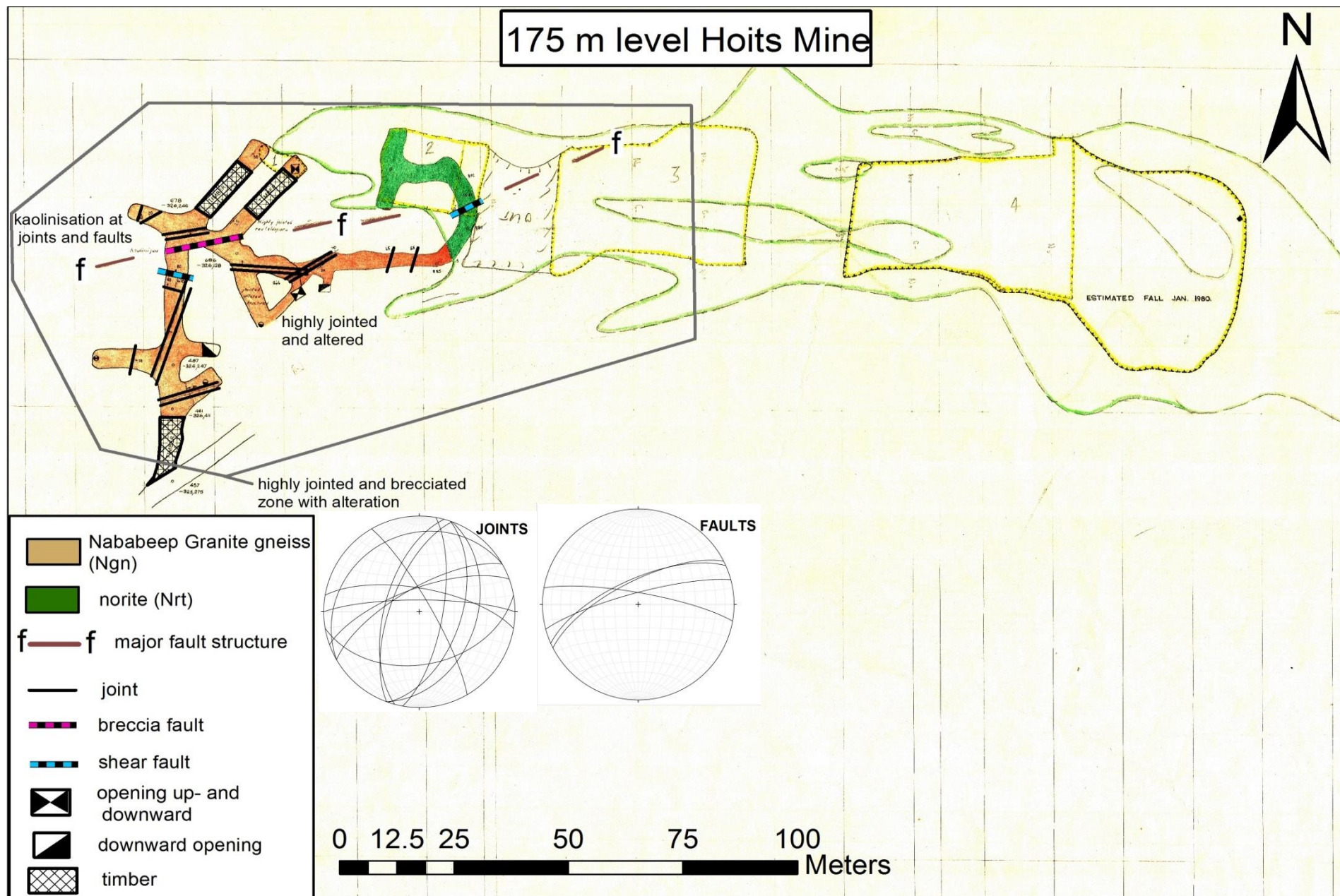
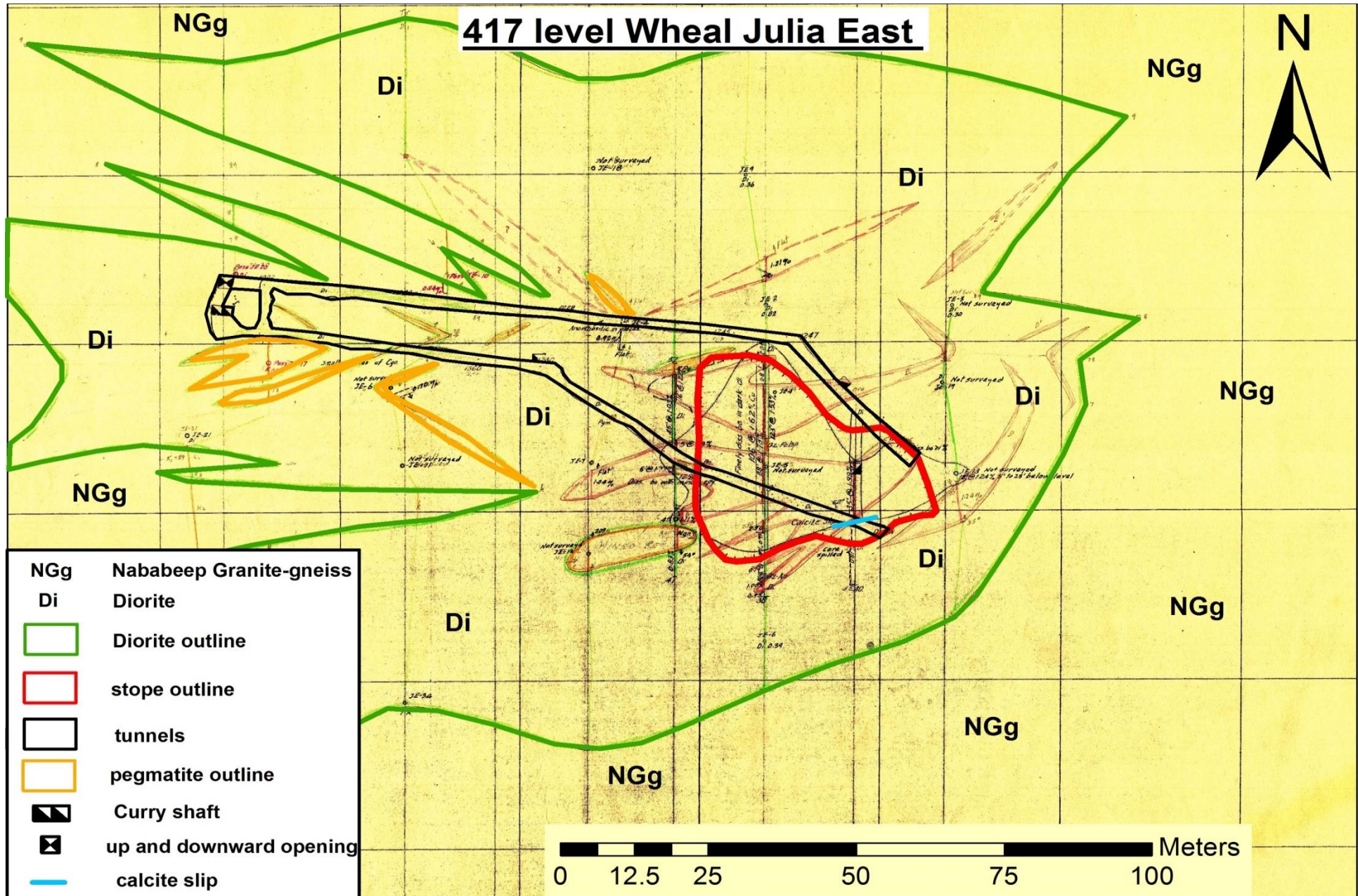
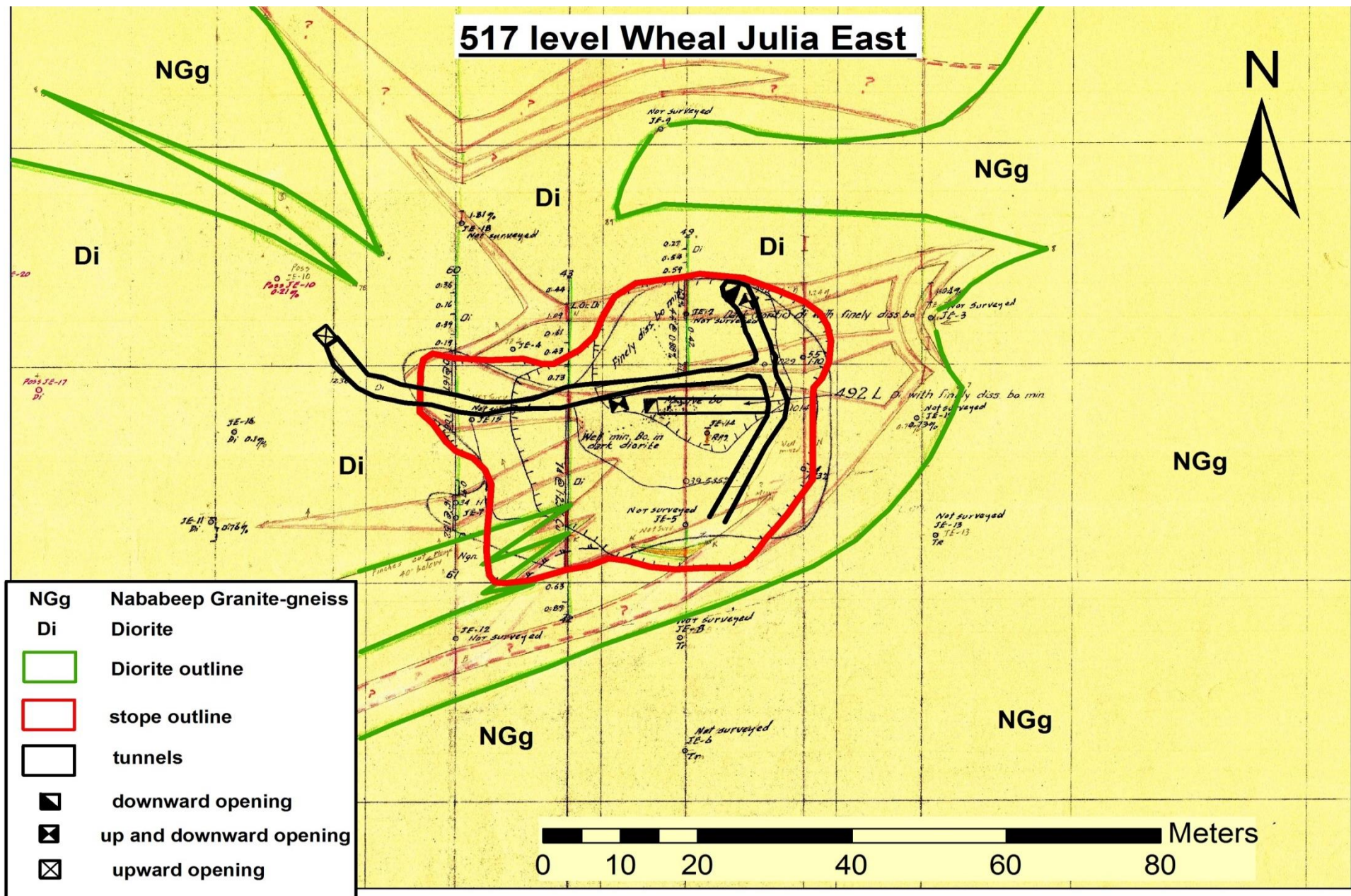


Figure C-1. 4: Map of 175 m level of the Hoits Mine (O'okiep Copper Company, 1975-1993k).

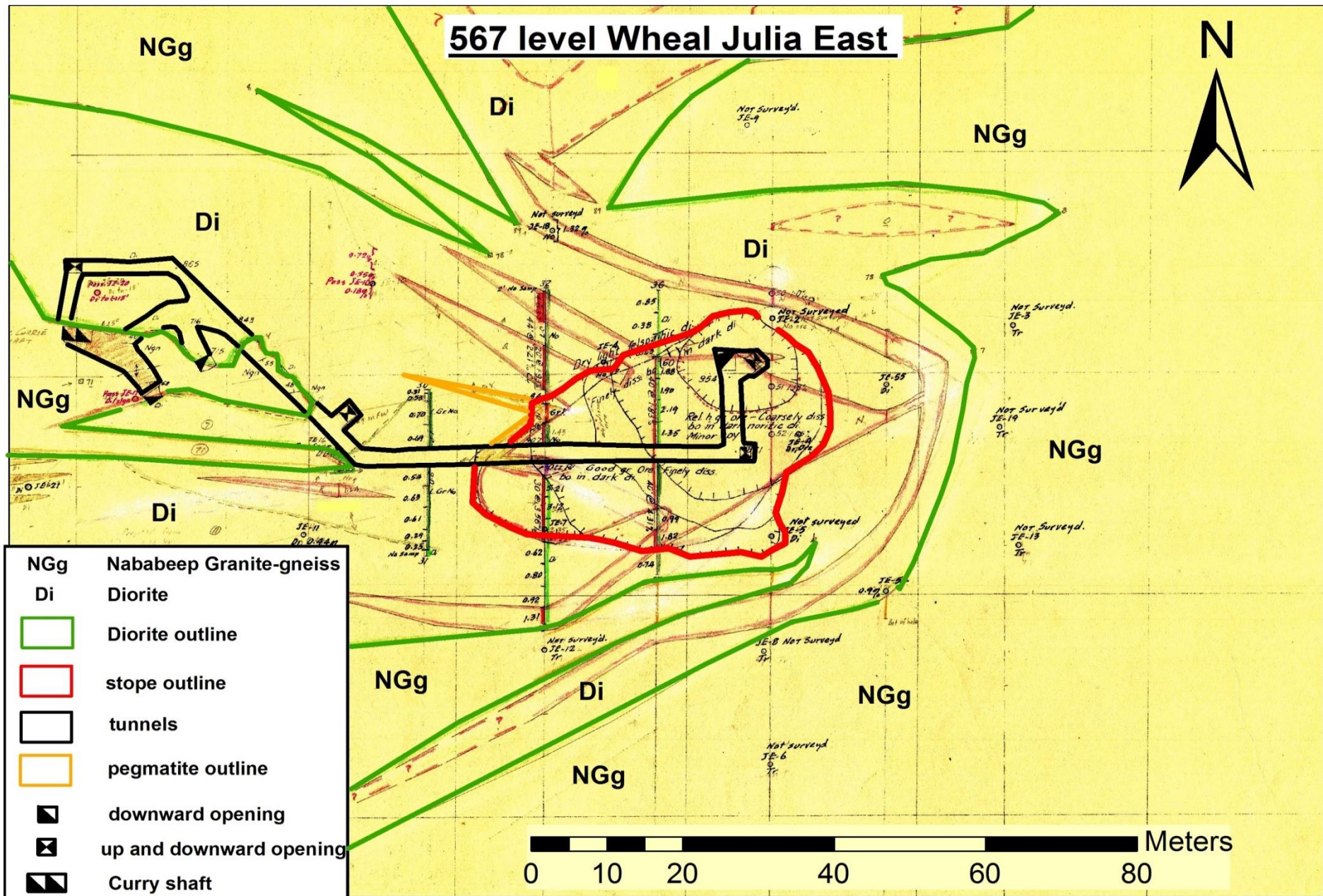
C-2: Wheal Julia East Mine



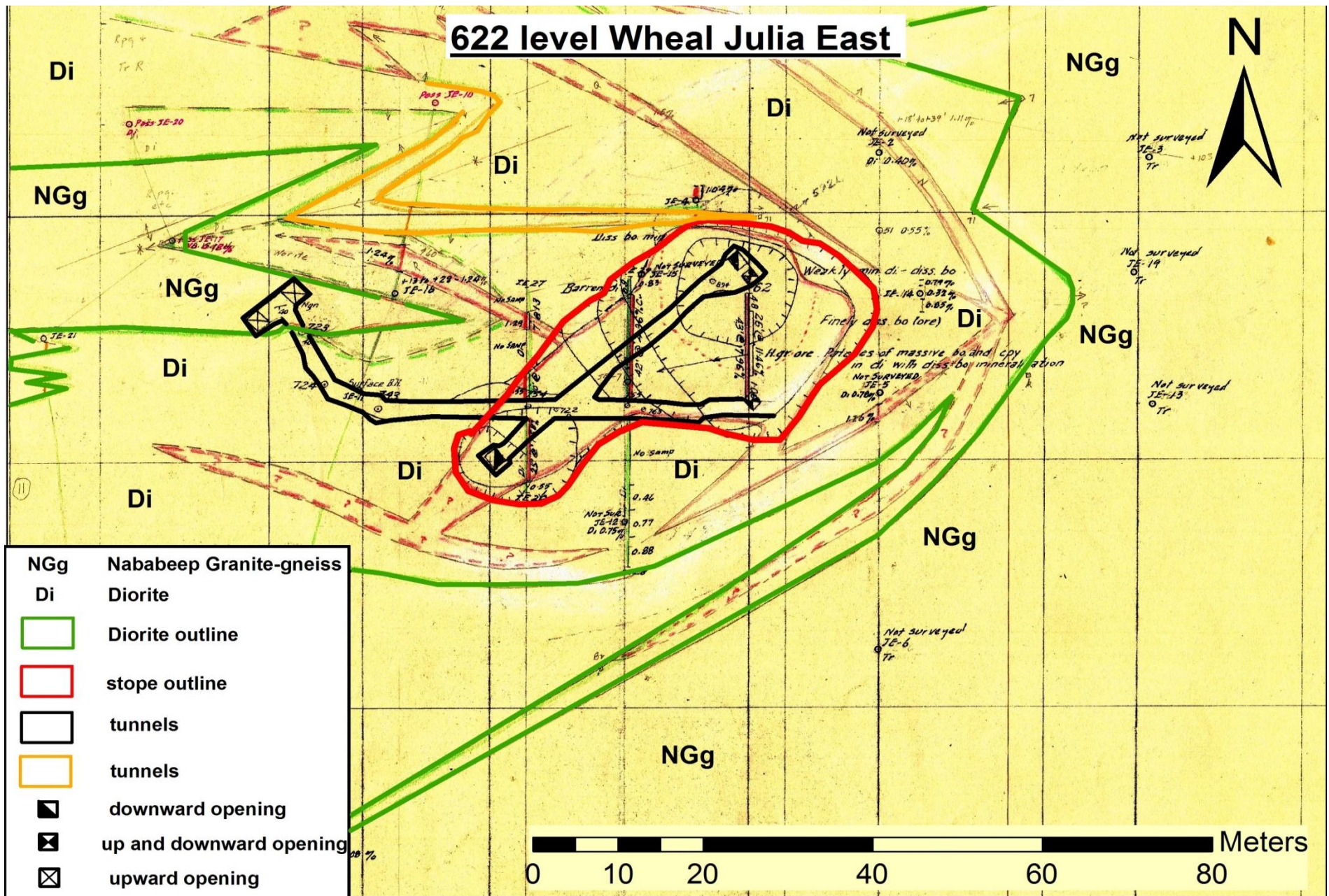
C-2. 2: Map of 417 level of the Wheal Julia East Mine (O'okiep Copper Company, 1955-1959b).



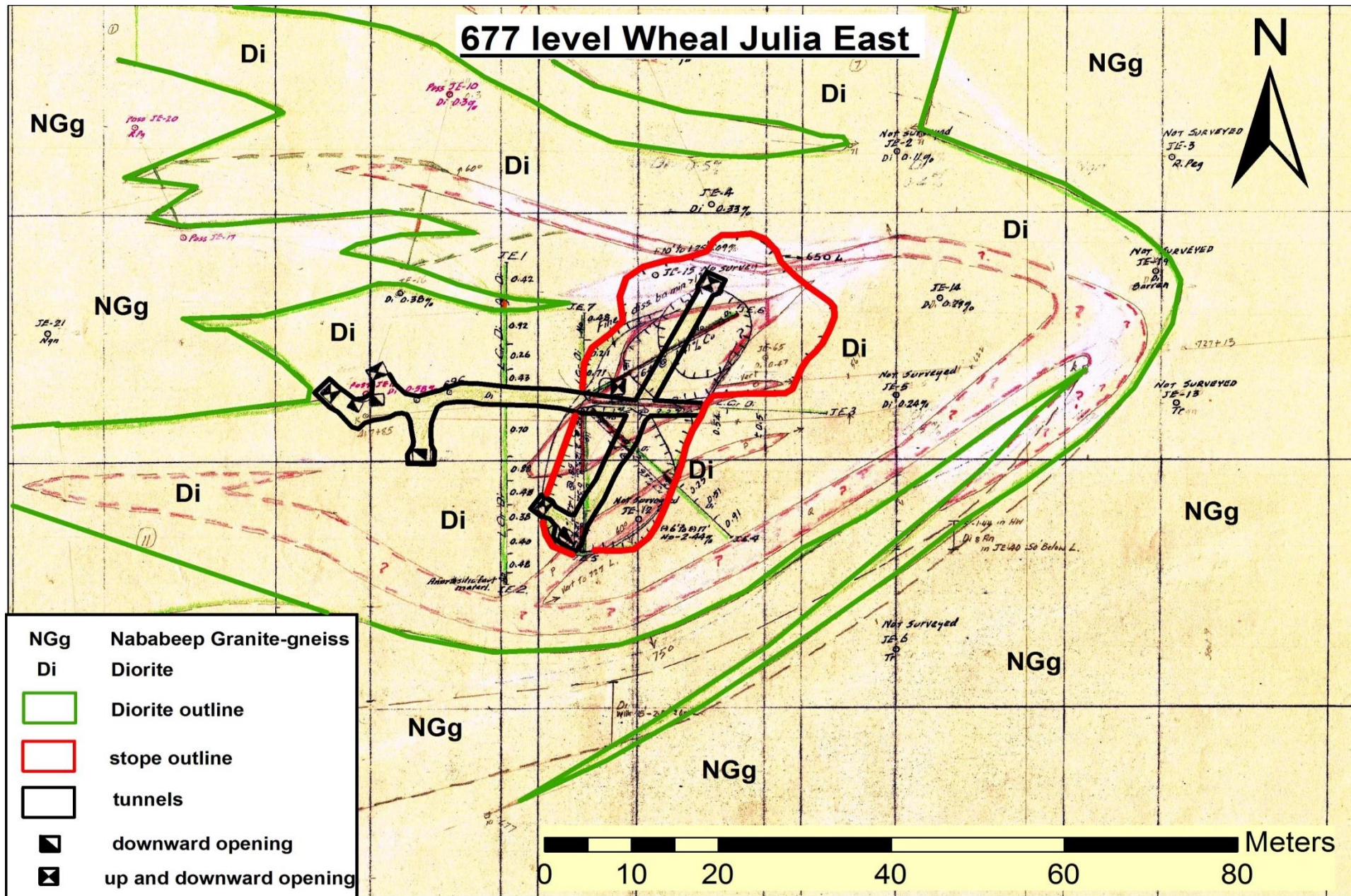
C-2. 3: Map of 517 level of the Wheal Julia East Mine (O'okiep Copper Company, 1955-1959d).



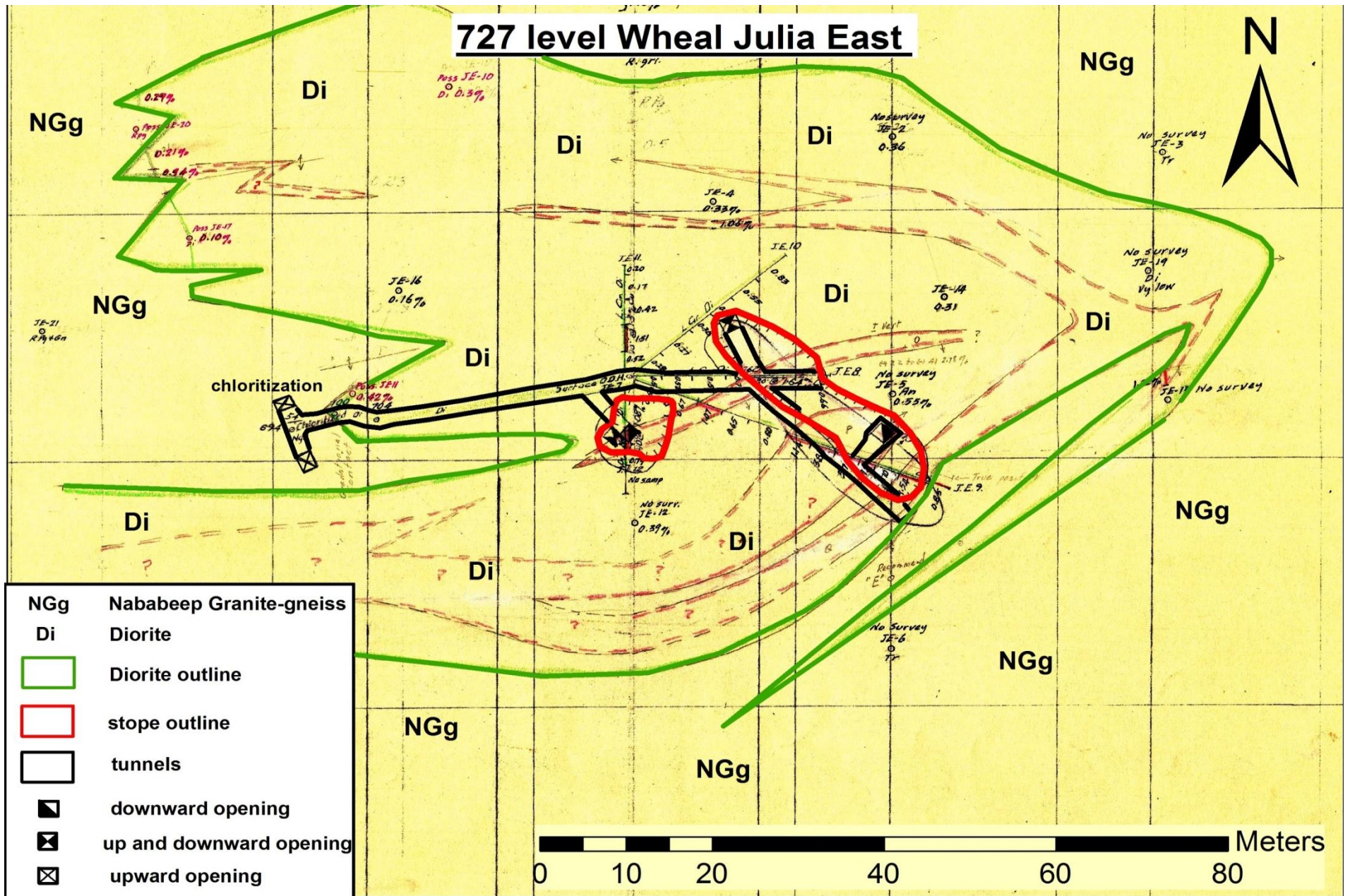
C-2. 4: Map of 567 level of the Wheal Julia East Mine (O'okiep Copper Company, 1955-1959e).



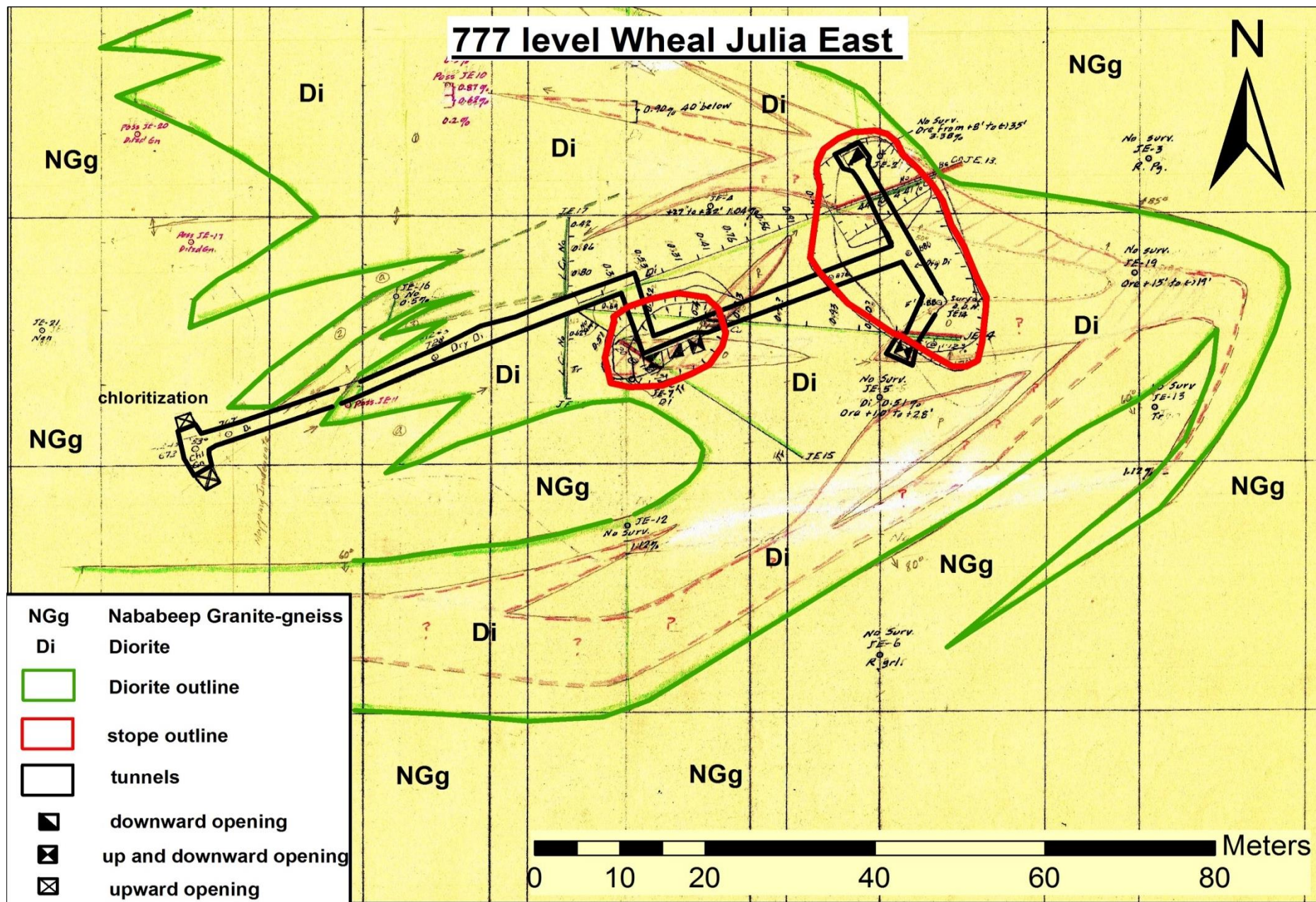
C-2. 5: Map of 622 level of the Wheal Julia East Mine (O'okiep Copper Company, 1955-1959f).



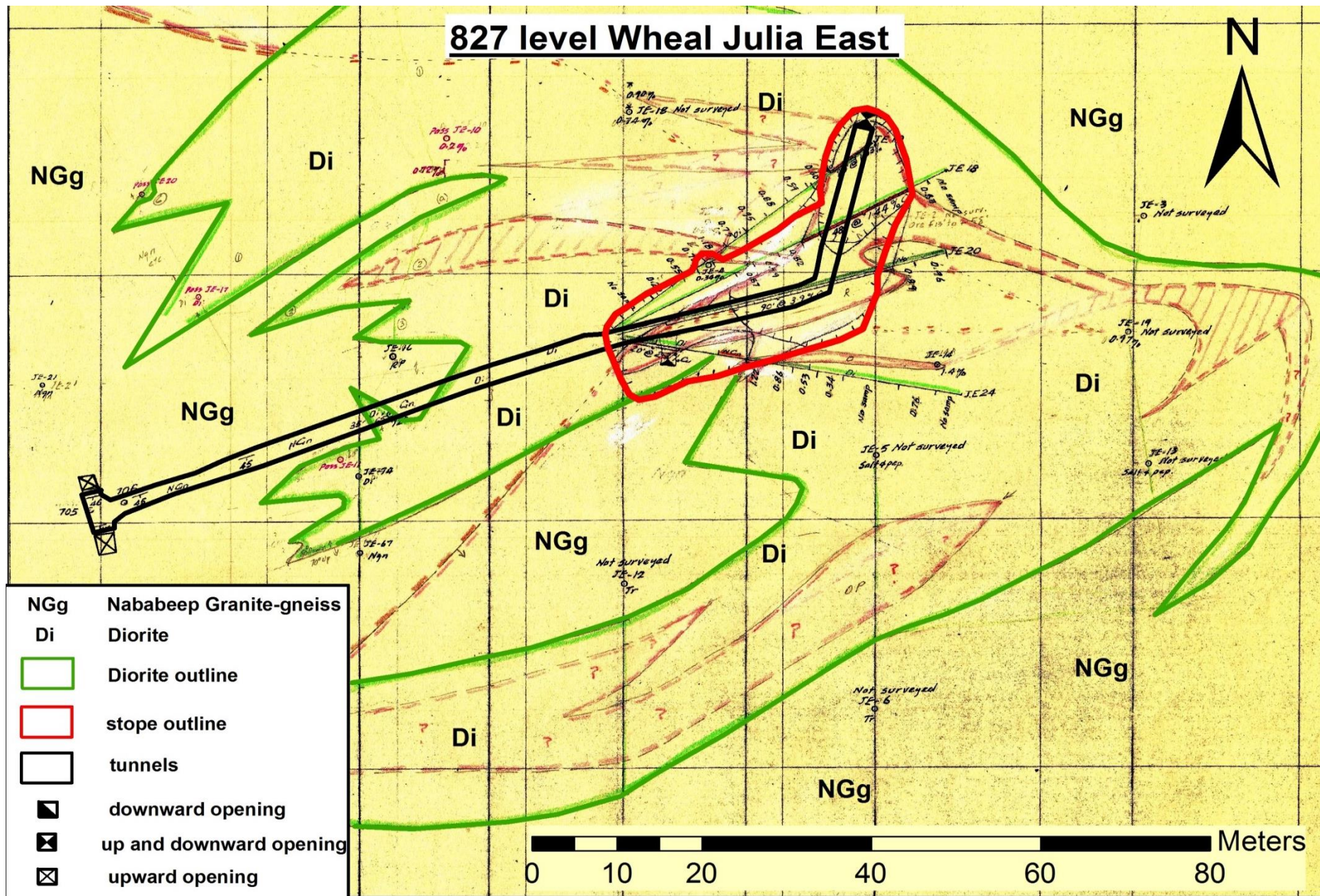
C-2. 6: Map of 677 level of the Wheal Julia East Mine (O'okiep Copper Company, 1955-1959g).



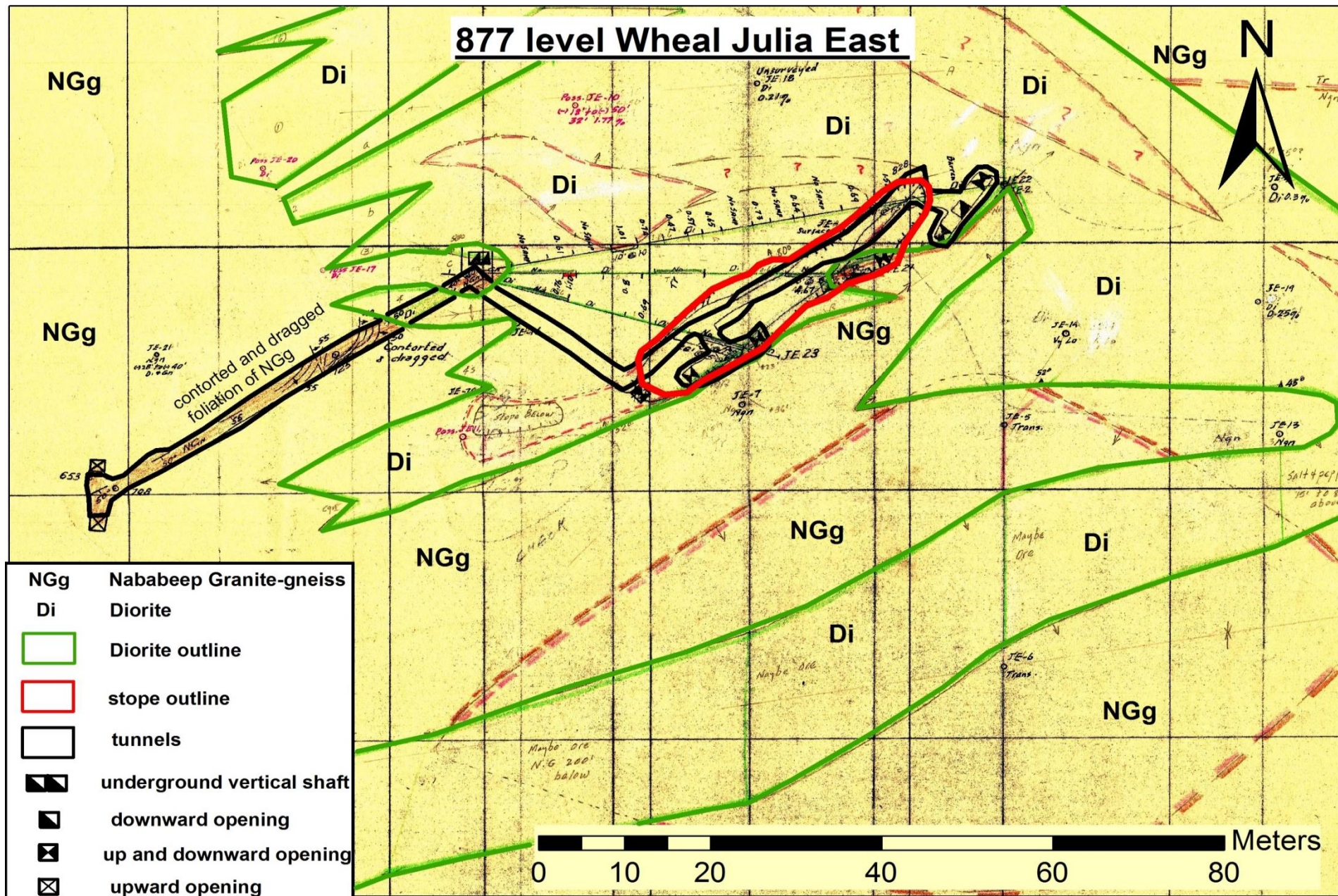
C-2. 7: Map of 727 level of the Wheal Julia East Mine (O'okiep Copper Company, 1955-1959h).



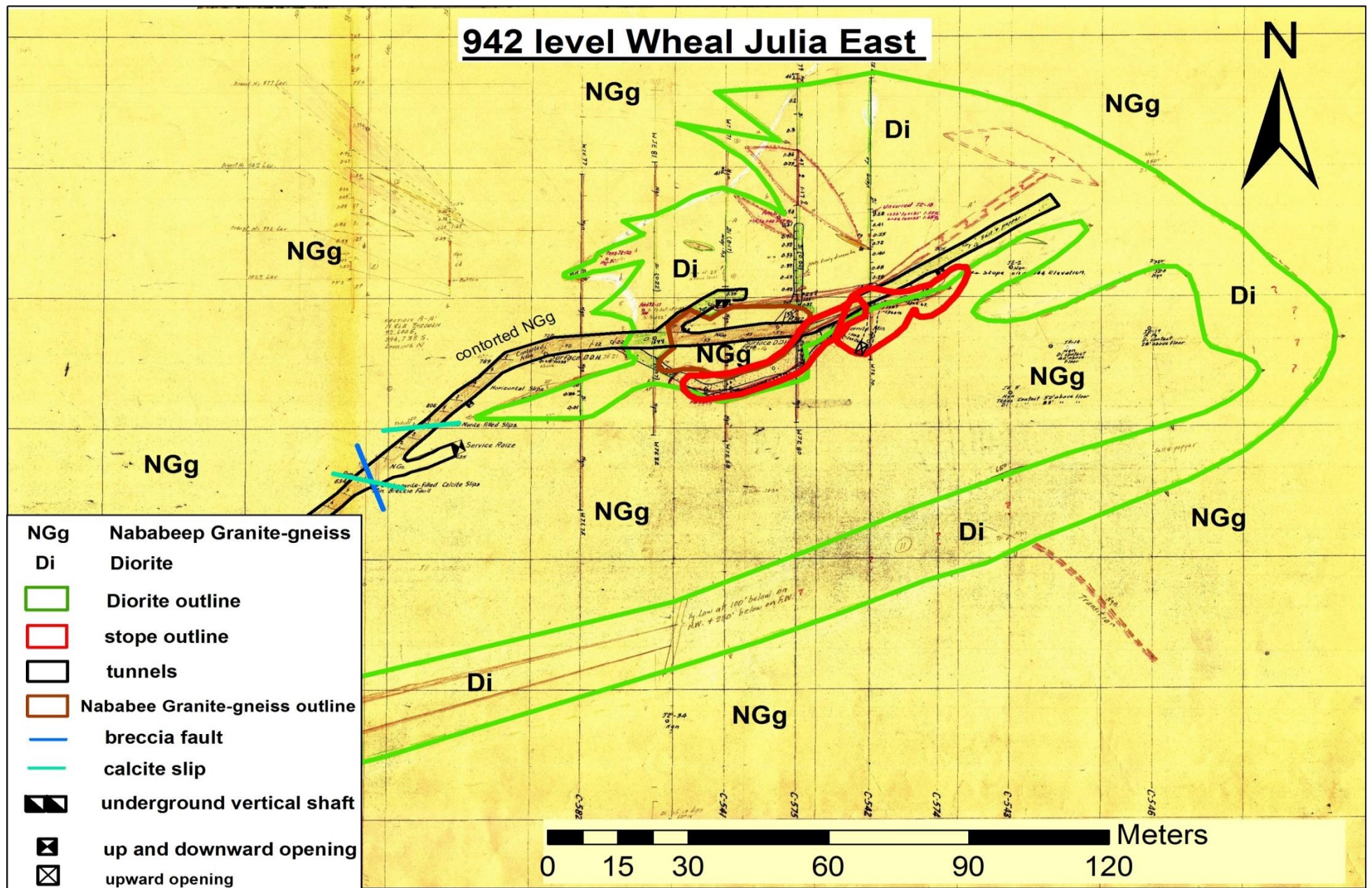
C-2. 8: Map of 777 level of the Wheal Julia East Mine (O'okiep Copper Company, 1955-1959i).



C-2. 9: Map of 827 level of the Wheal Julia East Mine (O'okiep Copper Company, 1955-1959j).



C-2. 10: Map of 877 level of the Wheal Julia East Mine (O'okiep Copper Company, 1955-1959 k).



C-2. 11: Map of 942 level of the Wheal Julia East Mine (O'okiep Copper Company, 1955-1959).

ADDENDUM D: GEOLOGICAL OVERLAY MAPS

D-1: Hoits Mine

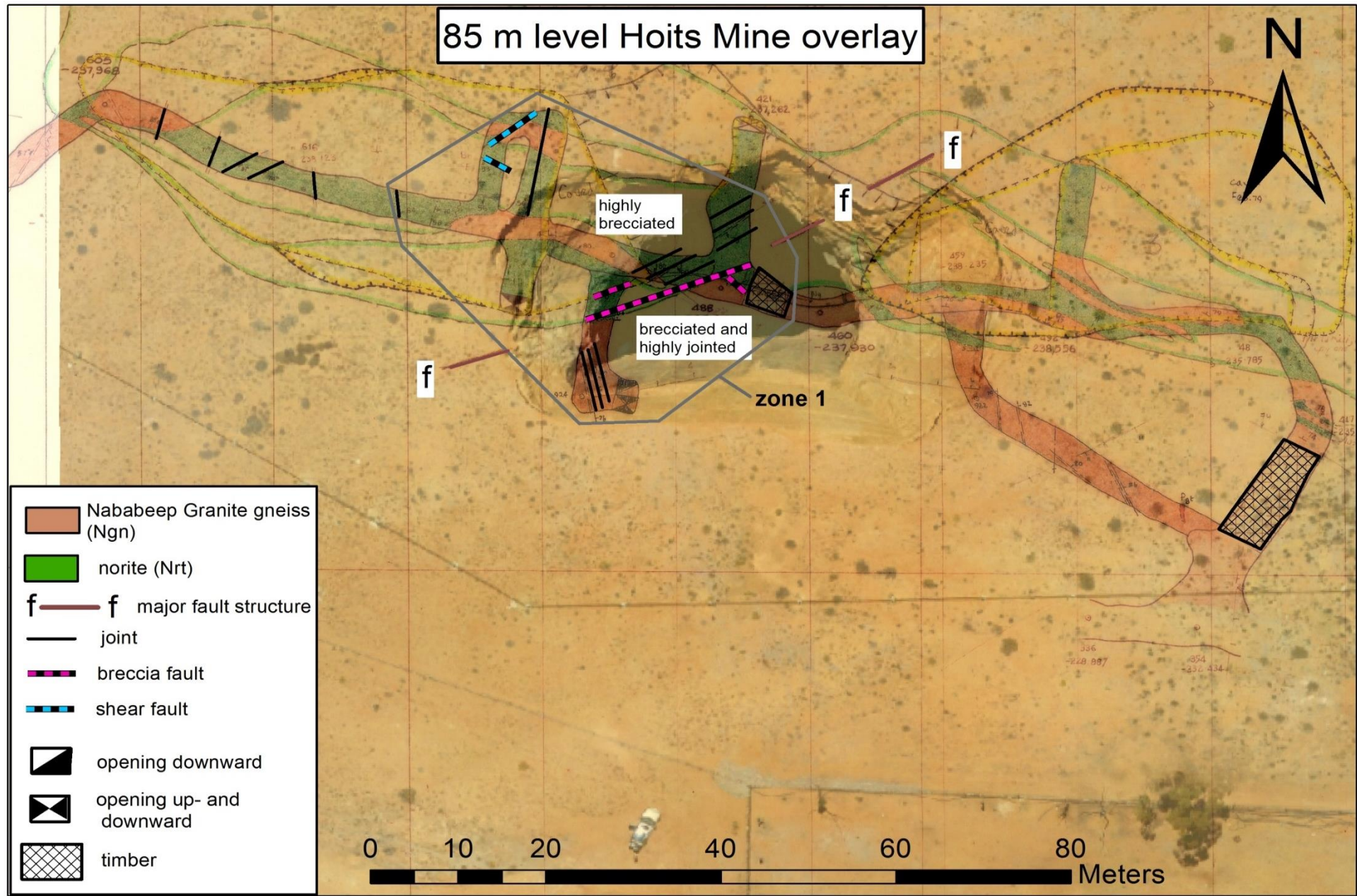


Figure D-1. 2: Map of 80 m level overlay of the Hoits Mine (O'okiep Copper Company, 1975-1993e).

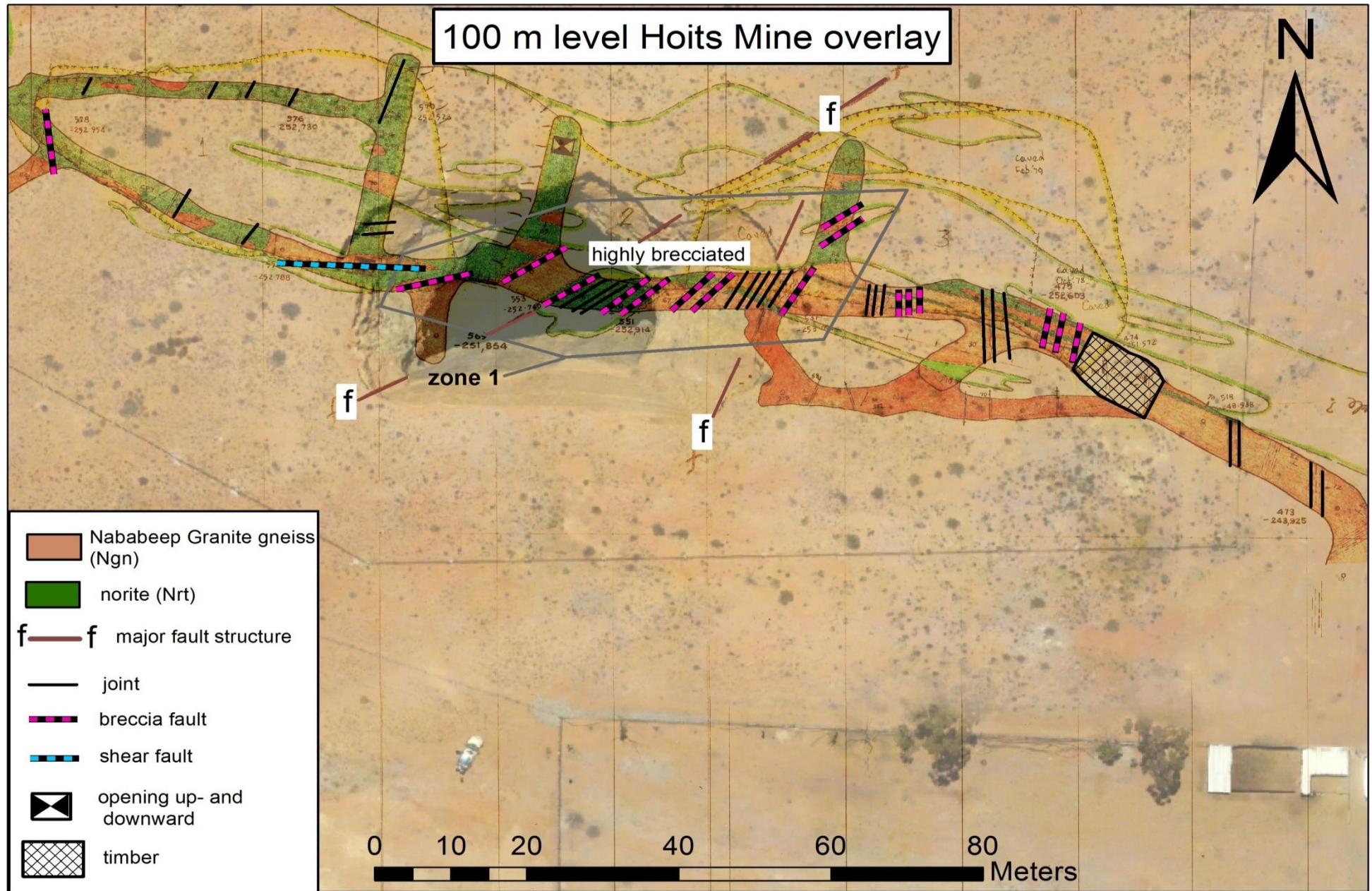


Figure D-1. 3: Map of 100 m level overlay of the Hoits Mine (O'okiep Copper Company, 1975-1993f).

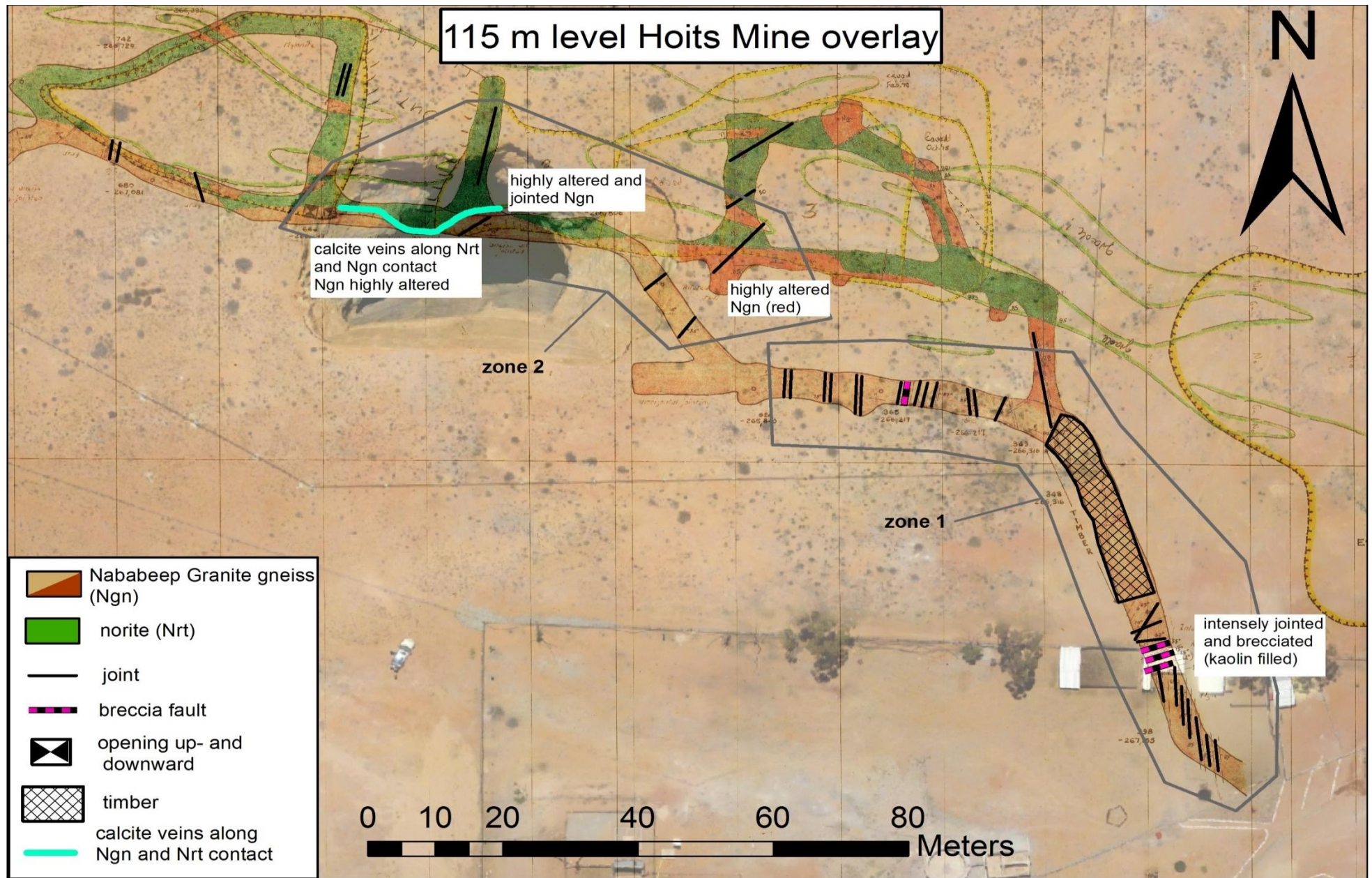


Figure D-1. 4: Map of 115 m level overlay of the Hoits Mine (O'okiep Copper Company, 1975-1993g).

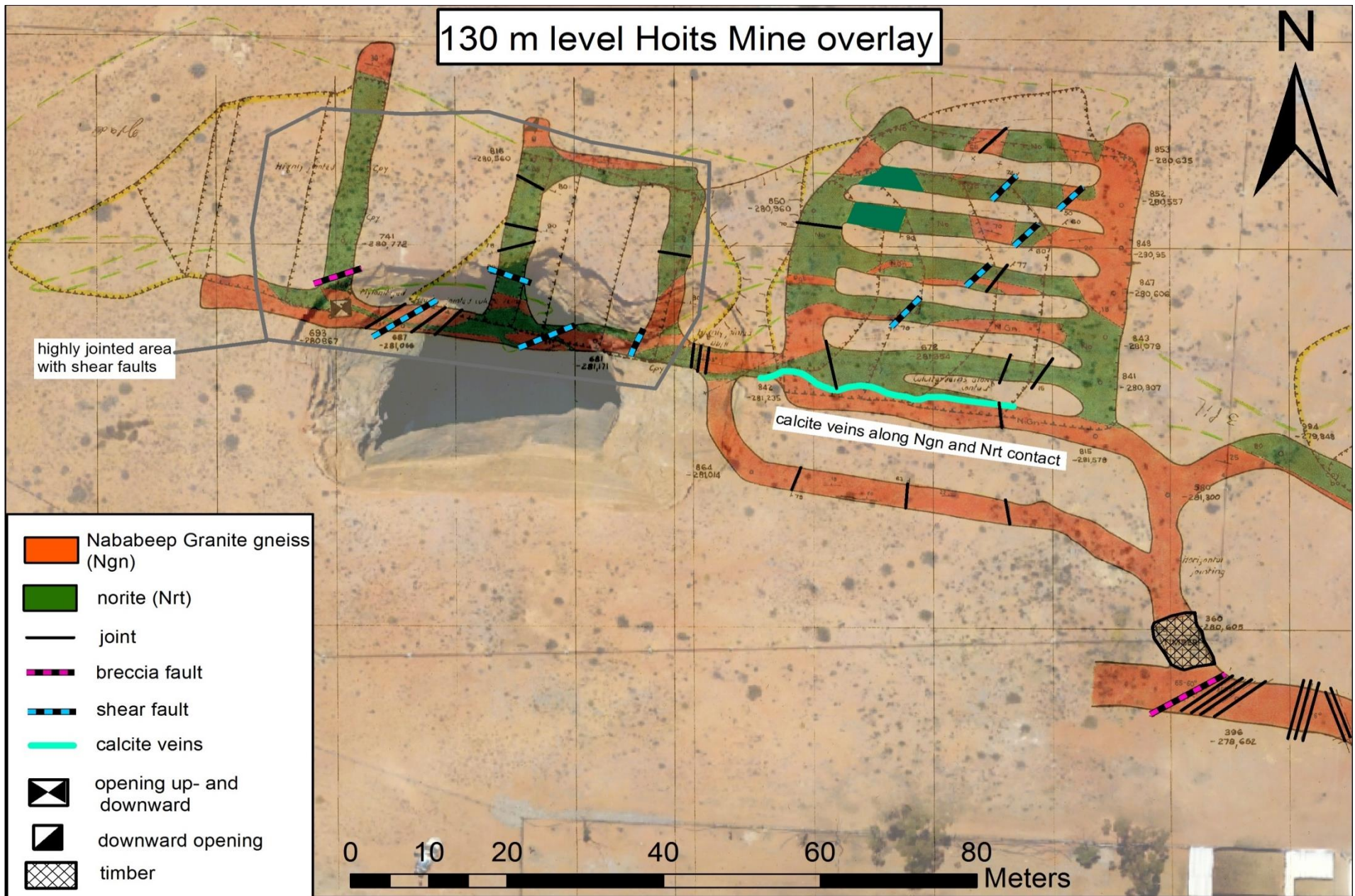


Figure D-1. 5: Map of 130 m level overlay of the Hoits Mine (O'okiep Copper Company, 1975-1993h).

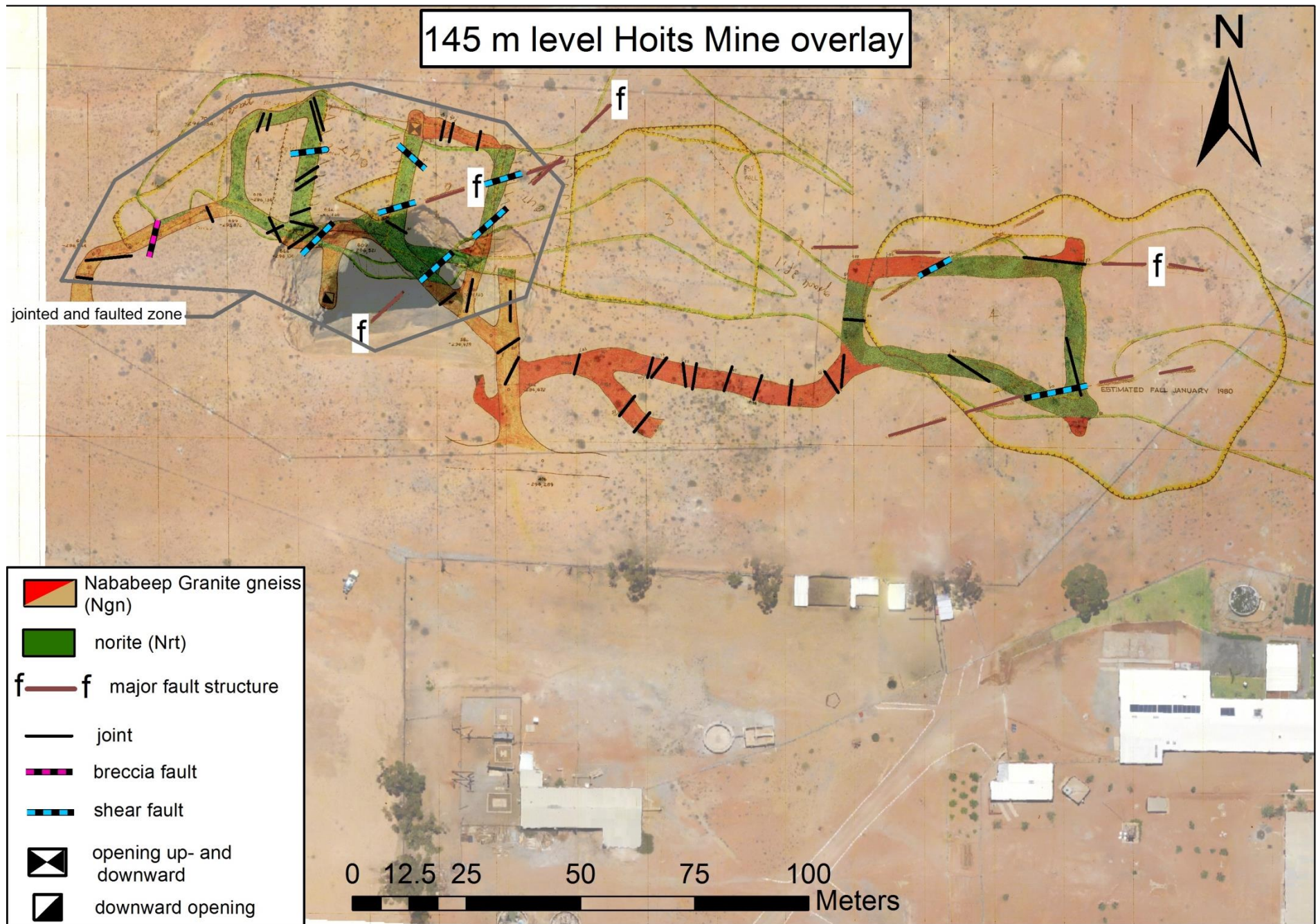


Figure D-1. 6: Map of 145 m level overlay of the Hoits Mine (O'okiep Copper Company, 1975-1993i).

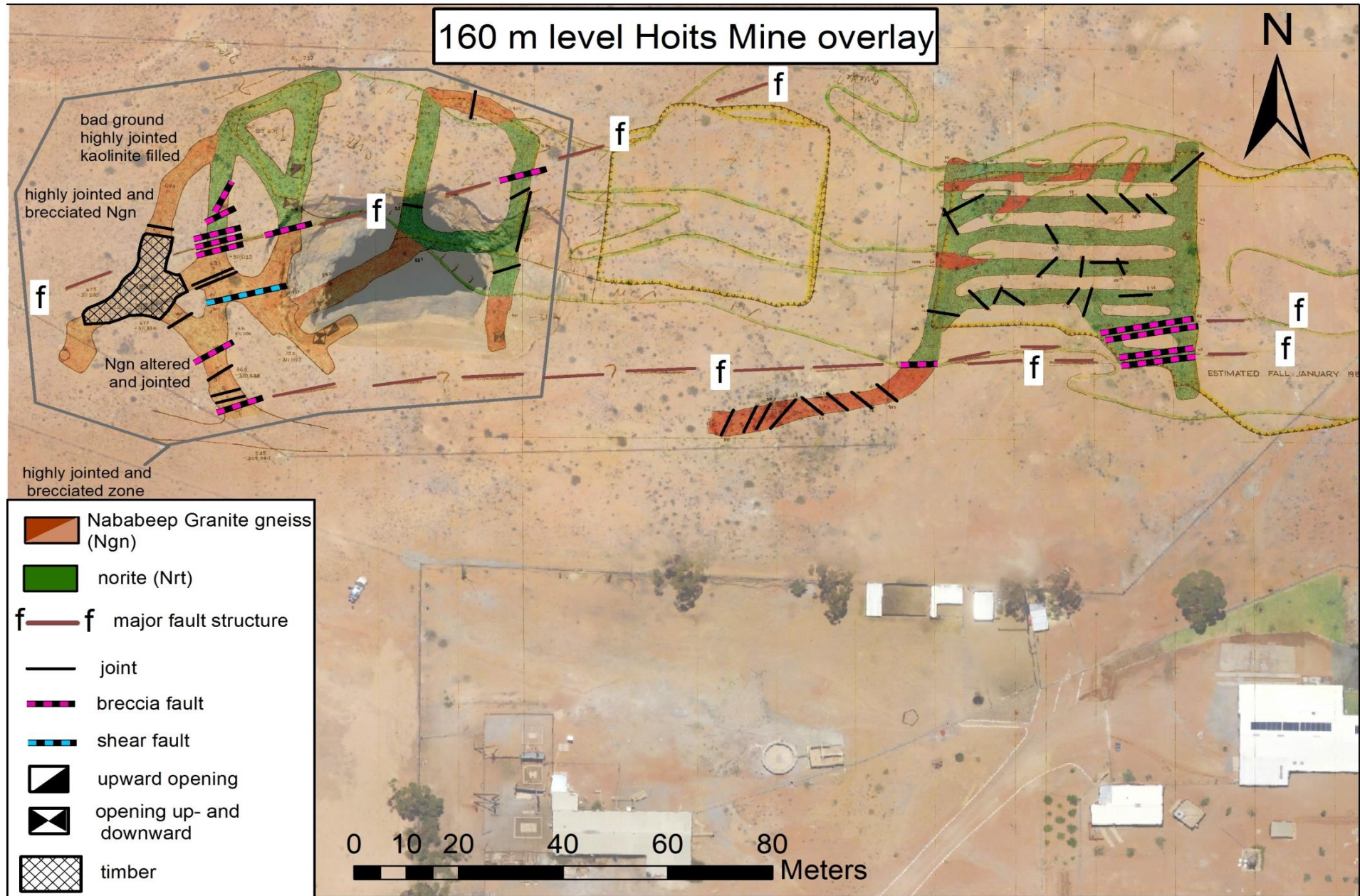


Figure D-1. 7: Map of 160 m level overlay of the Hoits Mine (O'okiep Copper Company, 1975-1993j).

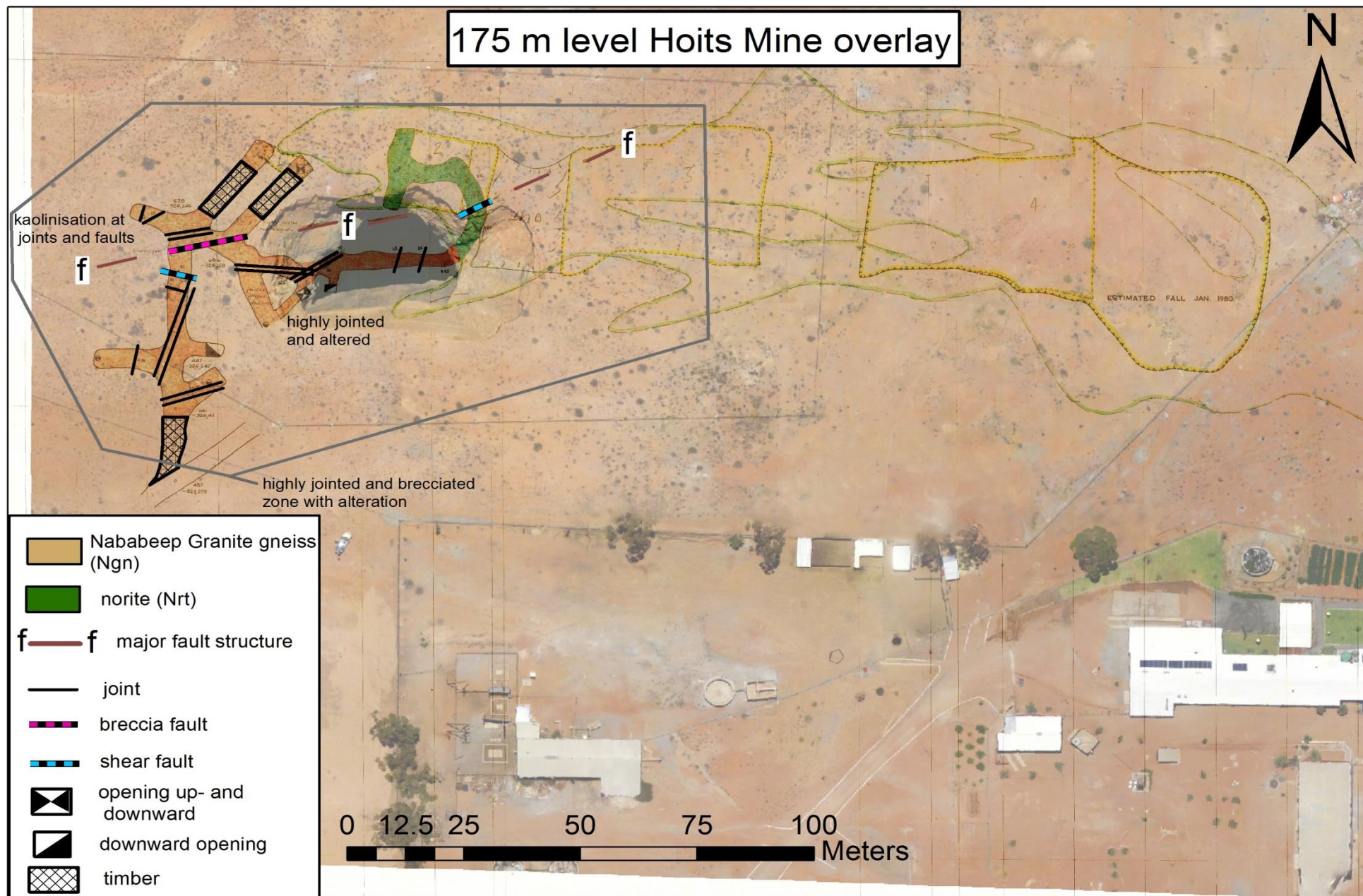


Figure D-1. 8: Map of 175 m level overlay of the Hoits Mine (O'okiep Copper Company, 1975-1993k).

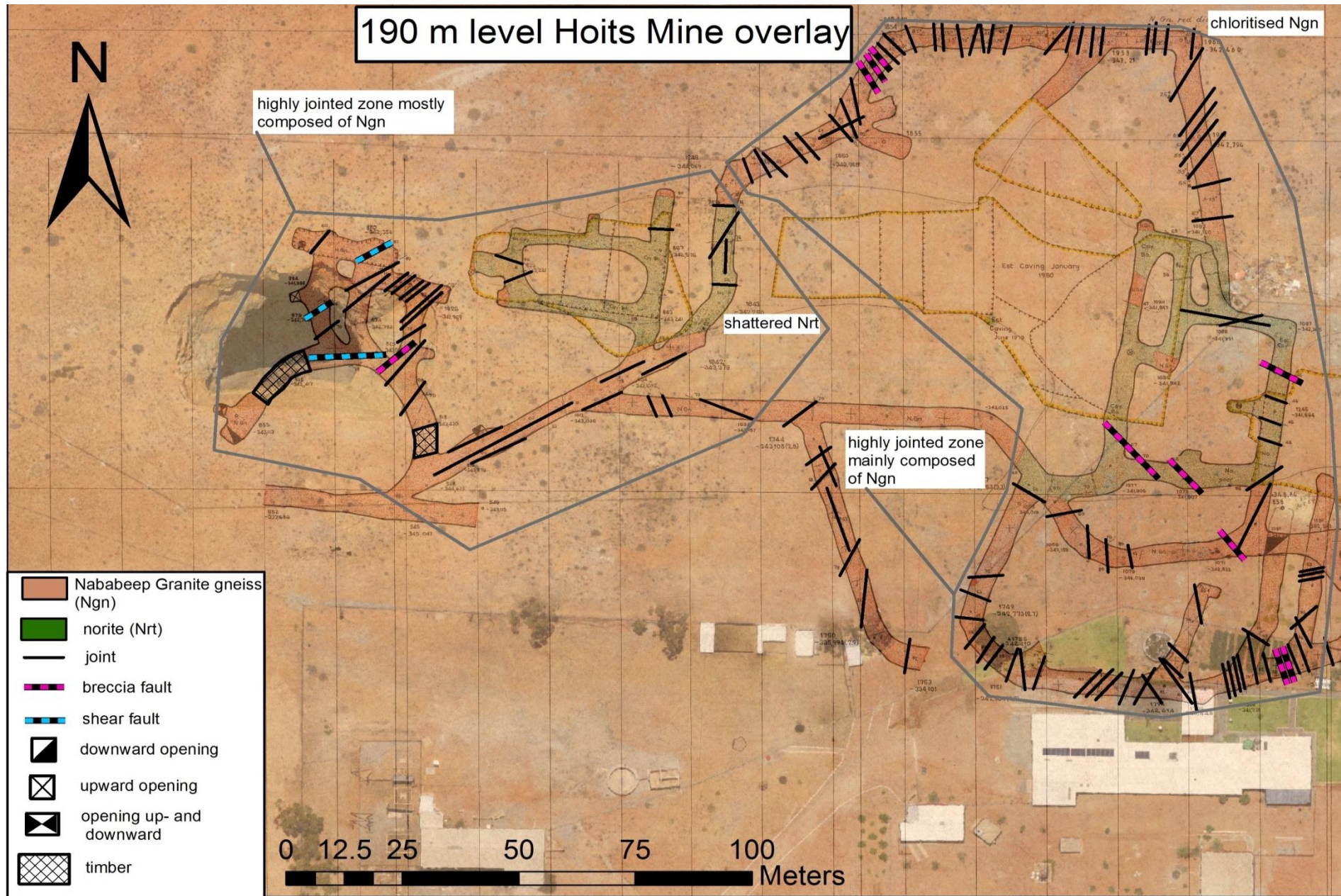


Figure D-1. 9: Map of 190 m level overlay of the Hoits Mine (O'okiep Copper Company, 1975-1993).

D-2: Wheal Julia East Mine

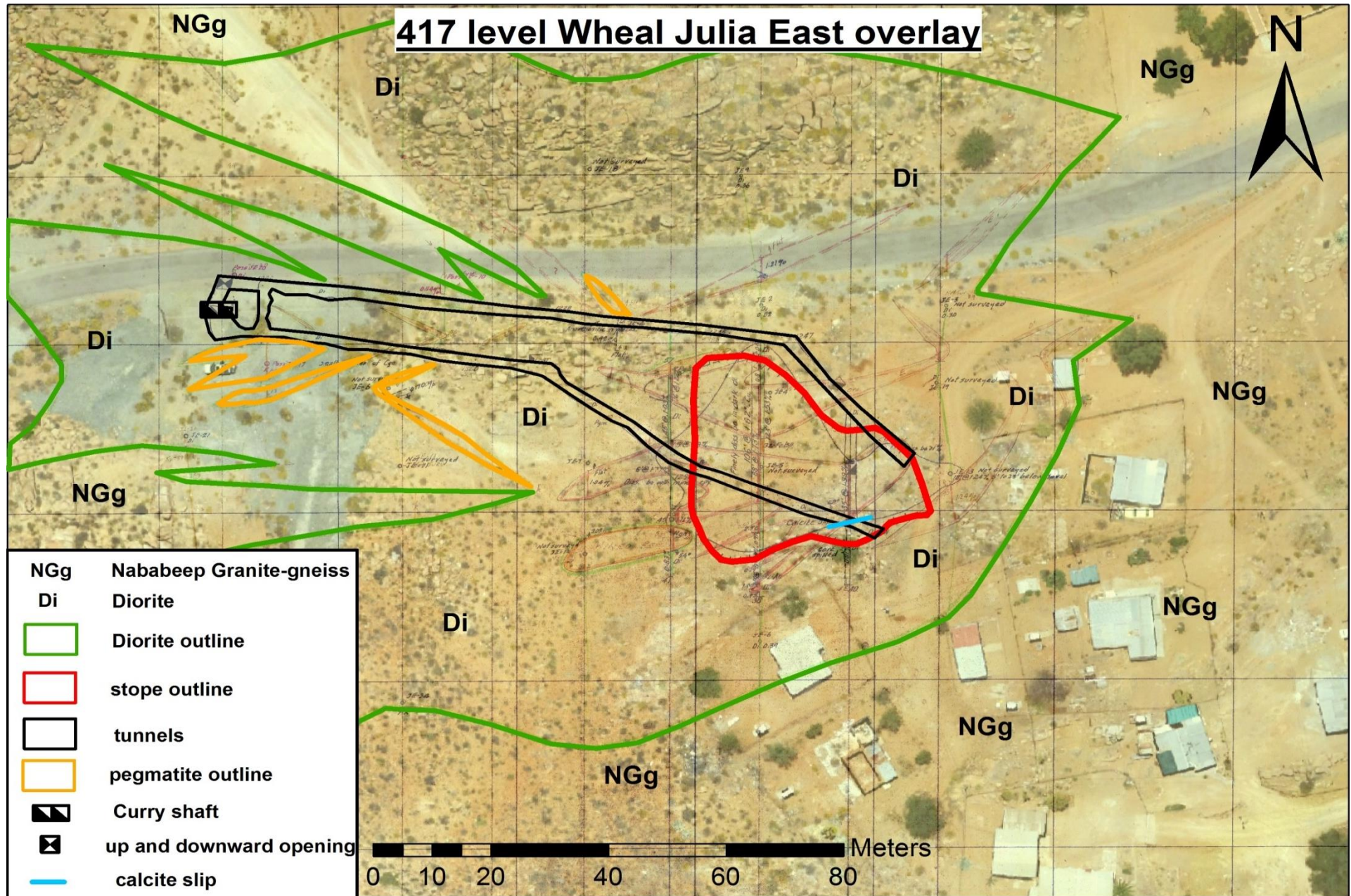


Figure D-2. 2: Map of 417 level overlay of the Wheal Julia East Mine (O'okiep Copper Company, 1955-1959b).

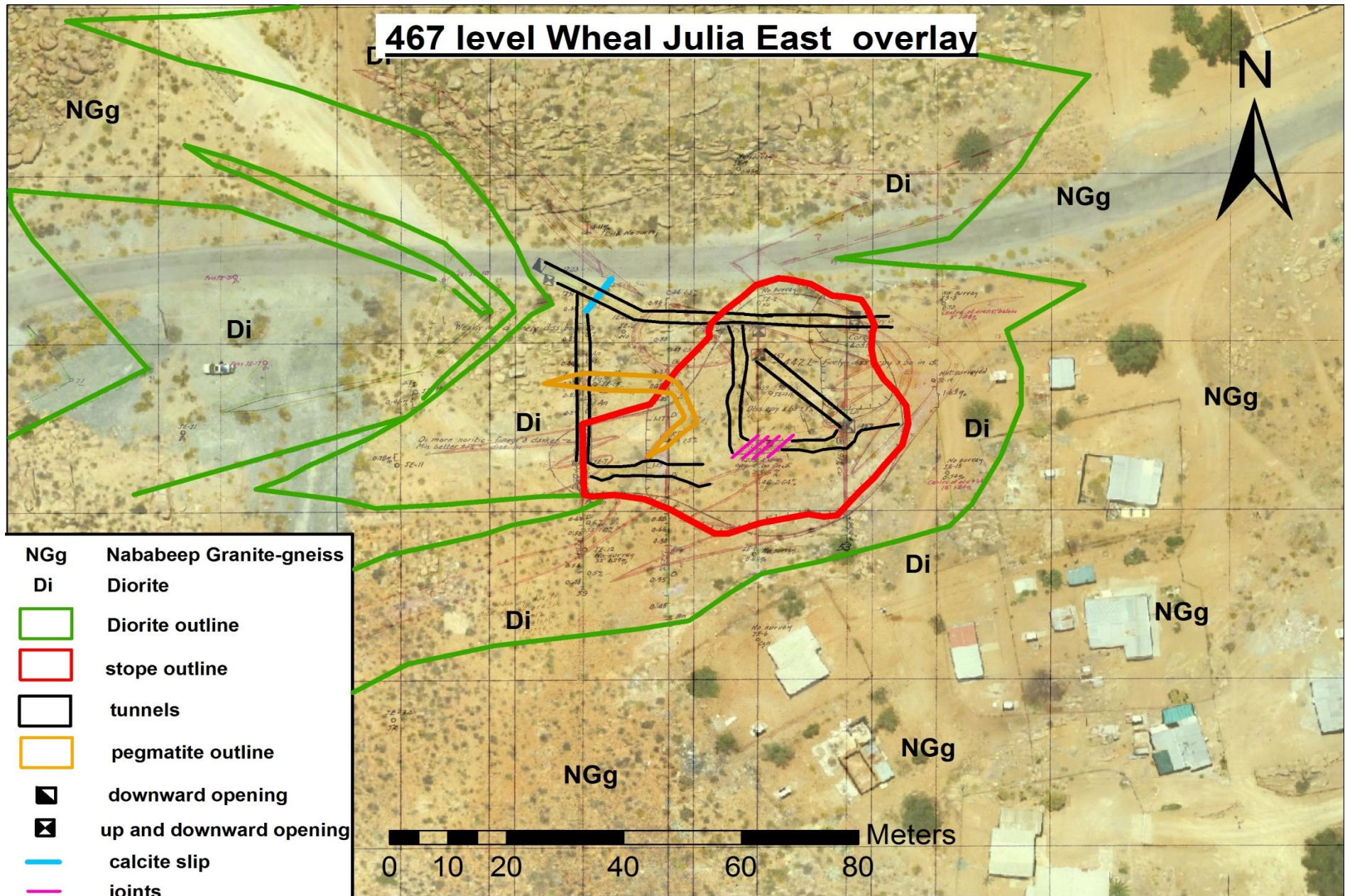


Figure D-2. 3: Map of 467 level overlay of the Wheal Julia East Mine (O'okiep Copper Company, 1955-1959c).

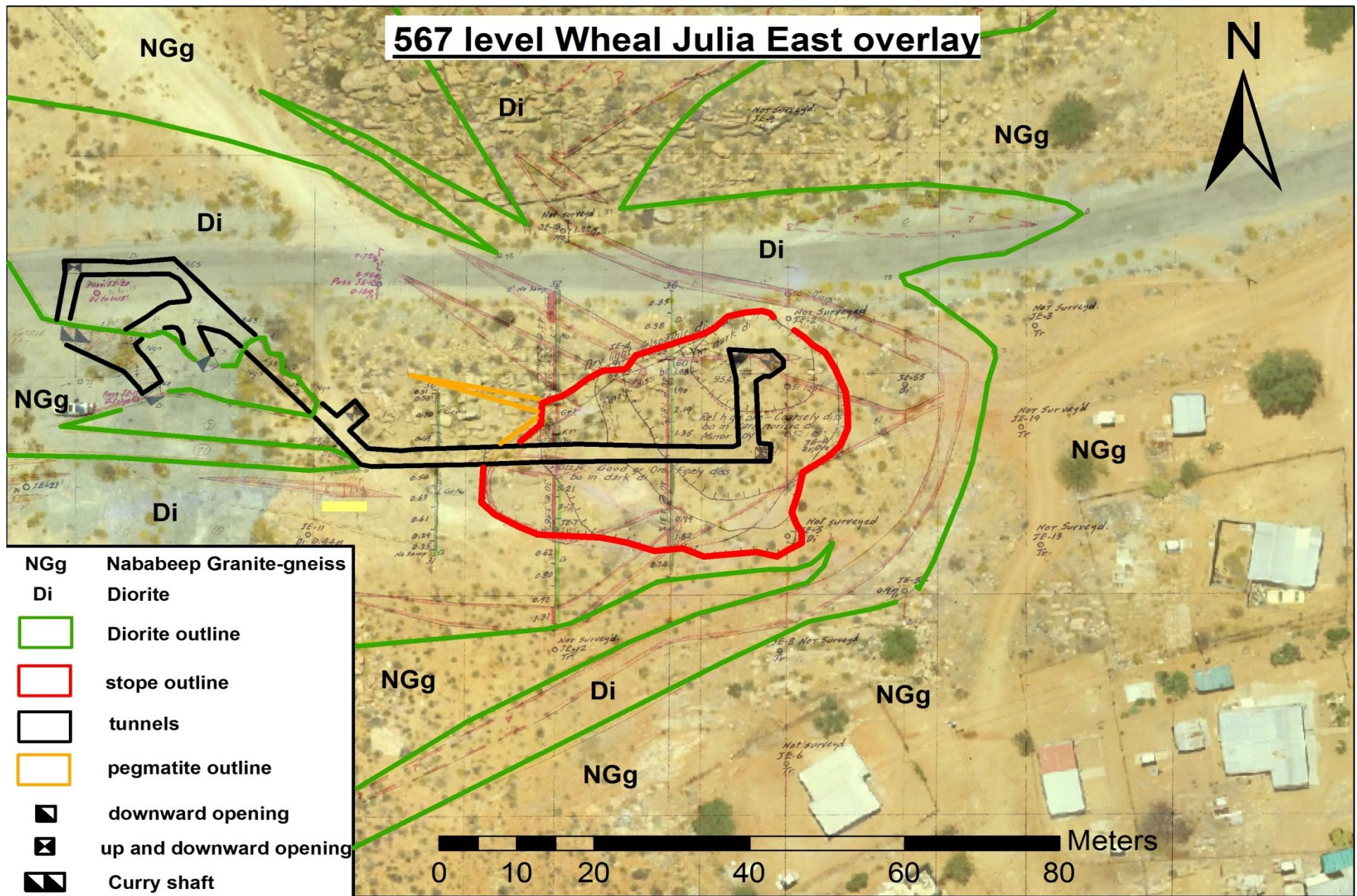


Figure D-2. 4: Map of 567 level overlay of the Wheal Julia East Mine (O'okiep Copper Company, 1955-1959e).

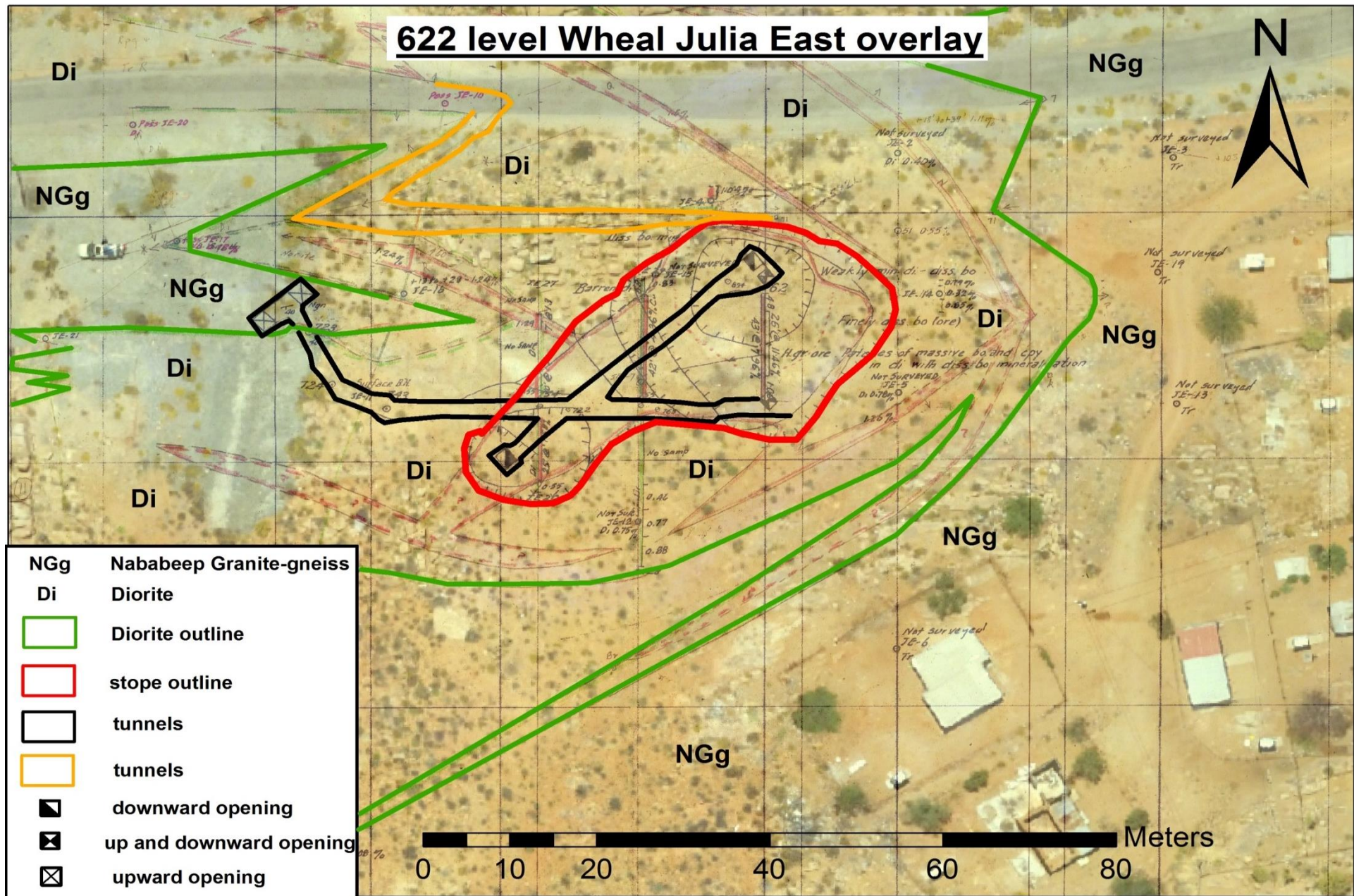


Figure D-2. 5: Map of 622 level overlay of the Wheal Julia East Mine (O'okiep Copper Company, 1955-1959f).

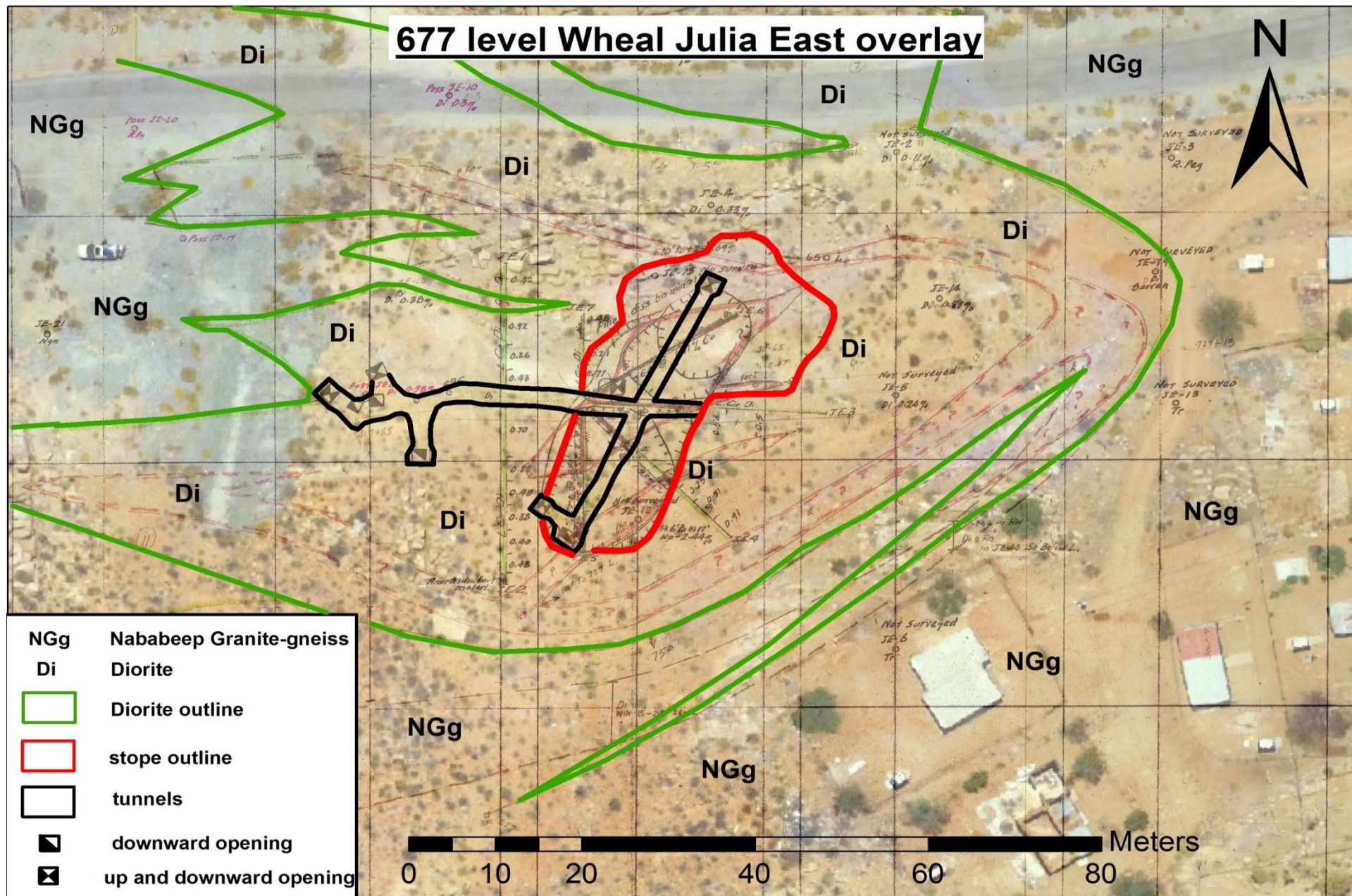


Figure D-2. 6: Map of 677 level overlay of the Wheal Julia East Mine (O'okiep Copper Company, 1955-1959g).

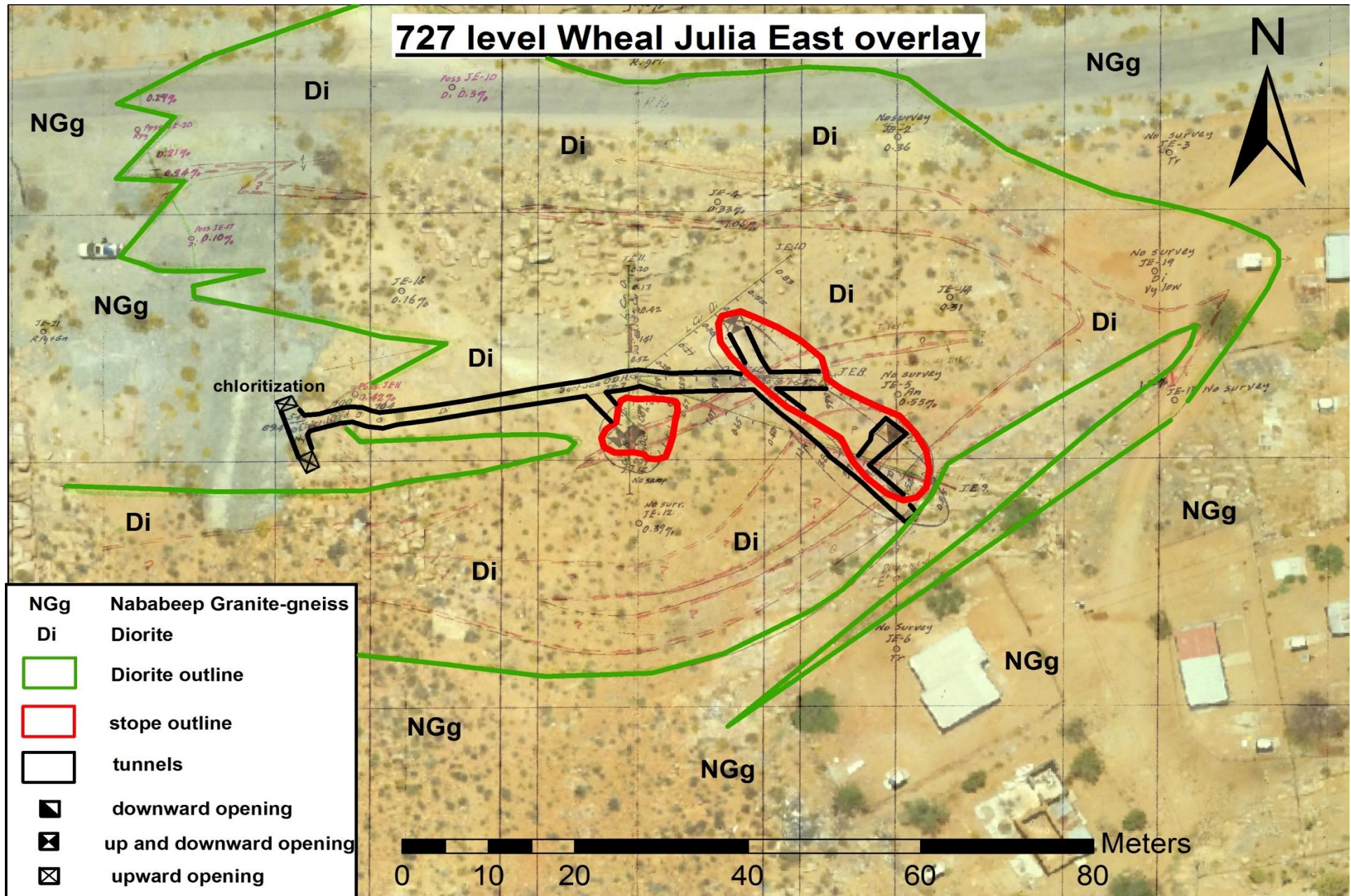


Figure D-2. 7: Map of 727 level overlay of the Wheal Julia East Mine (O'okiep Copper Company, 1955-1959h).

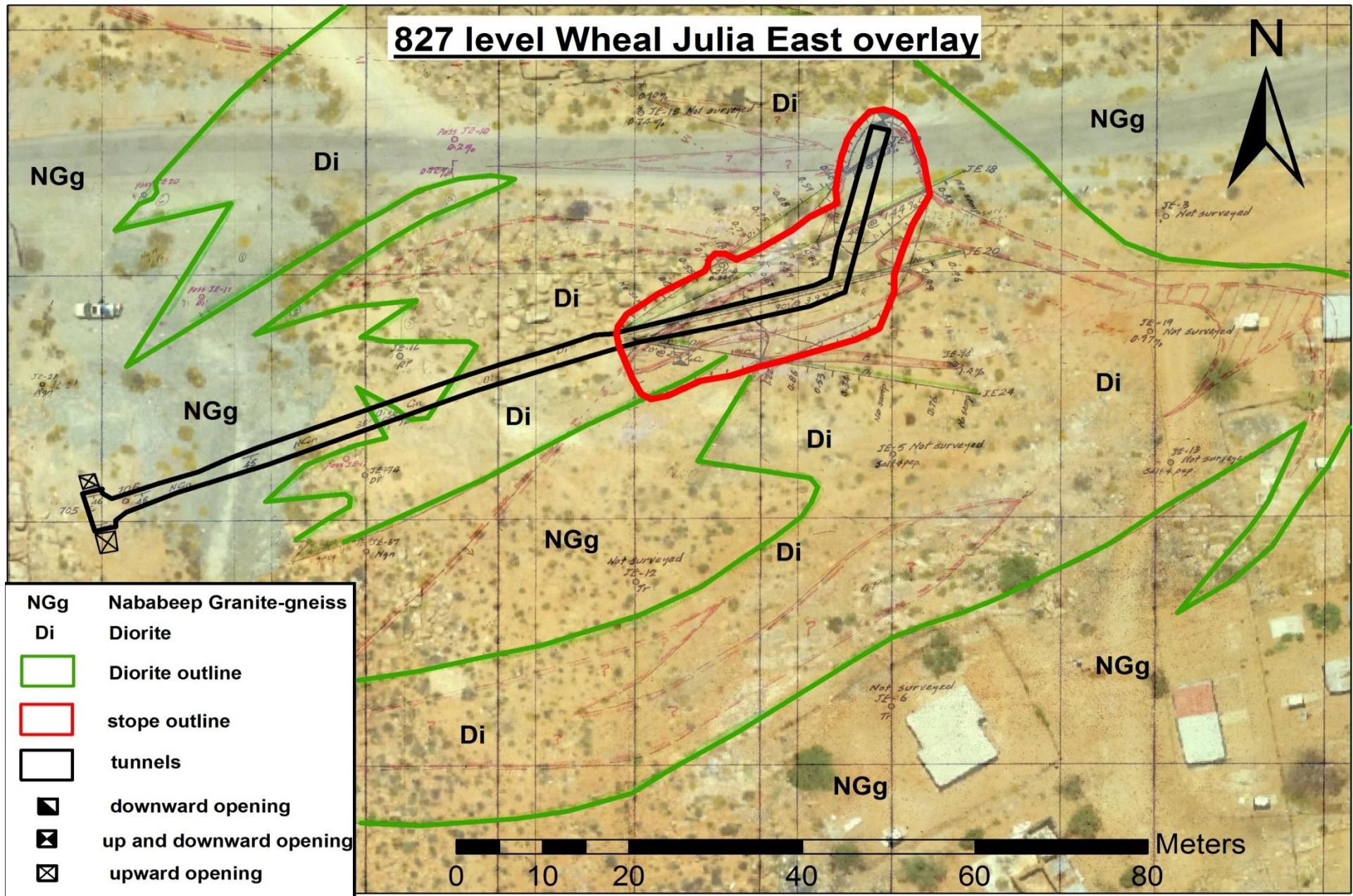


Figure D-2. 9: Map of 827 level overlay of the Wheal Julia East Mine (O’okiep Copper Company, 1955-1959j).

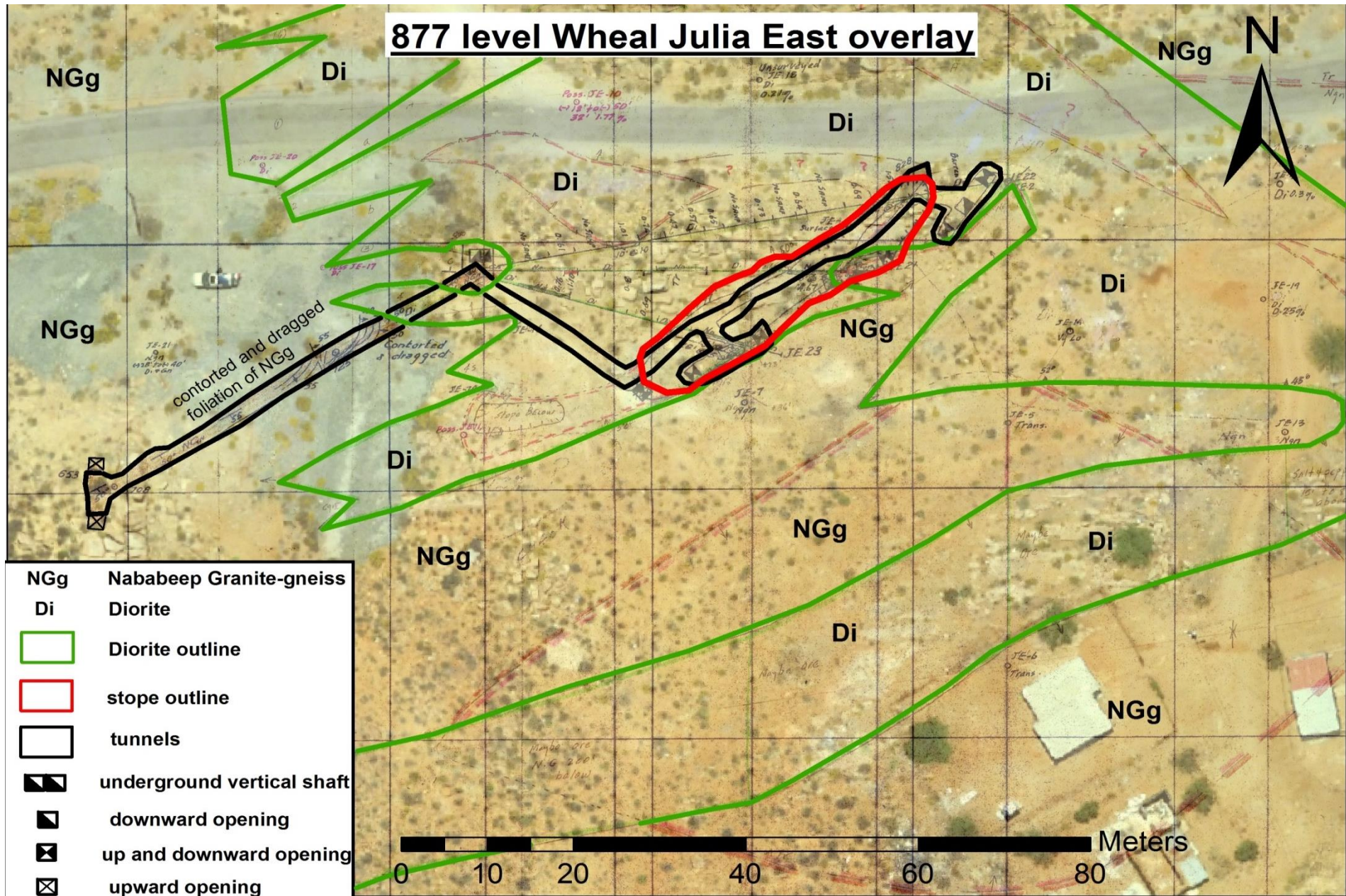


Figure D-2. 10: Map of 877 level overlay of the Wheal Julia East Mine (O'okiep Copper Company, 1955-1959k).

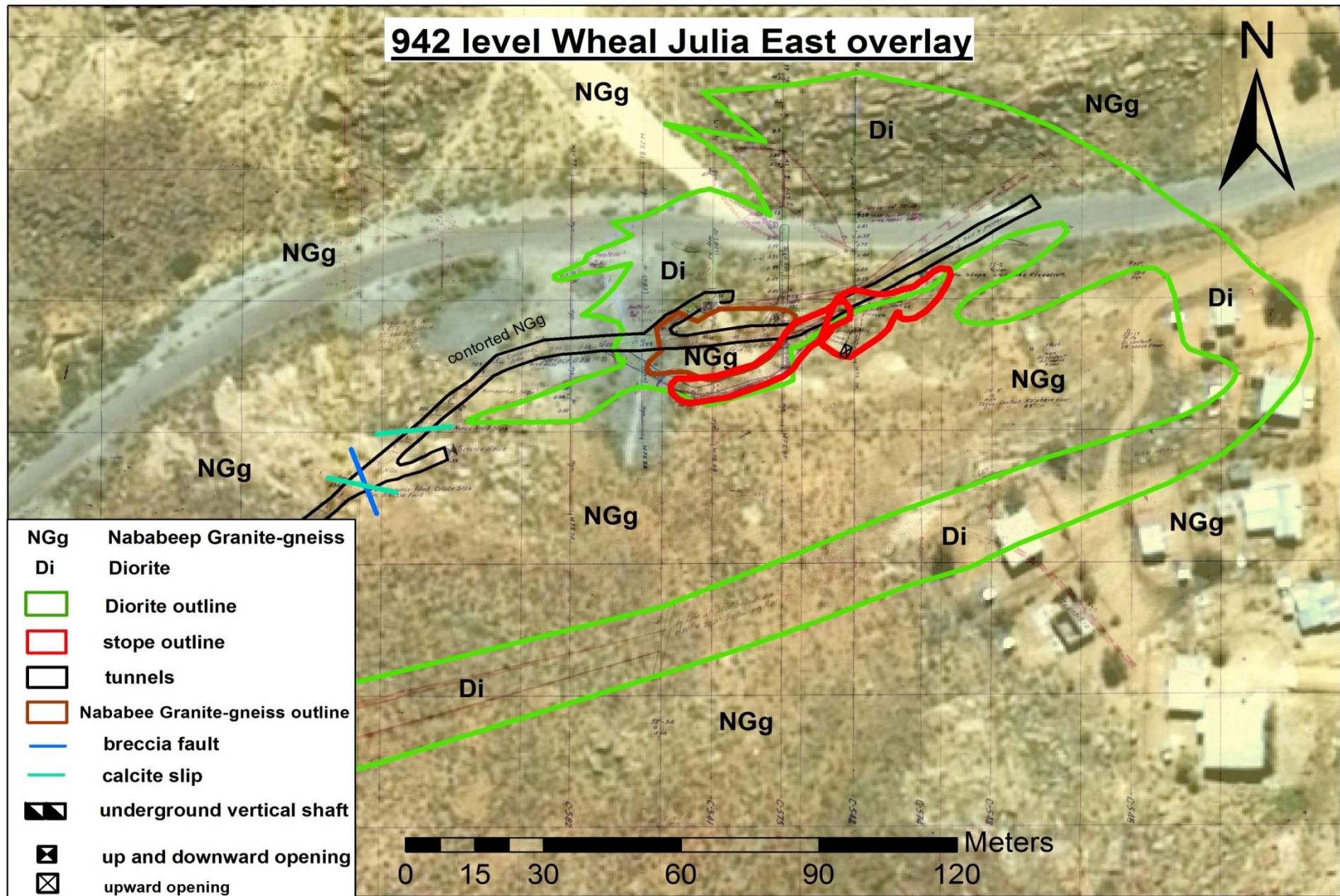


Figure D-2. 11: Map of 942 level overlay of the Wheal Julia East Mine (O'okiep Copper Company, 1955-1959).