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STOCKWORK ZONE PRE-FEASIBILITY STUDY AND UNDERGROUND PRELIMINARY ECONOMIC ASSESSMENT OF THE EL TIGRE SILVER-GOLD PROJECT SONORA, MEXICO

LATITUDE 30°35'15" N, LONGITUDE 109°13'23" W
UTM WGS84 12R 670,380 m E, 3,385,230 m N

FOR
SILVER TIGER METALS INC.

NI-43-101 & 43-101F1
TECHNICAL REPORT

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1.0 SUMMARY

This National Instrument ("NI") 43-101 Technical Report was prepared by P&E Mining Consultants Inc. ("P&E") for Silver Tiger Metals Inc. ("Silver Tiger" or the "Company") to provide an updated Mineral Resource Estimate, updated Pre-Feasibility Study ("PFS") on the Stockwork Zone, and underground Preliminary Economic Assessment ("PEA") for the El Tigre Property (the "Property"), located in the State of Sonora, Mexico. The Property is 100% owned by Silver Tiger.

Input to this Technical Report was also provided by D.E.N.M. Engineering Ltd. ("D.E.N.M.") for mineral processing and metallurgy, Consultores Interdisciplinario en Medio Ambiente S.C. ("CIMA") for environmental aspects, and WSP Canada Inc. ("WSP") for geotechncial and hydrogeological engineering. This Technical Report considers the gold and silver mineralization at El Tigre that is amenable to both surface and underground mining and has an effective date of June 20, 2025.

1.1 PROPERTY, LOCATION, ACCESS

The El Tigre Property extends across the Sierra El Tigre area in northeastern Sonora State, Mexico, situated approximately 90 km south-southeast of the Town of Agua Prieta. The Property's coordinates are approximately 30°35' north latitude and 109°13' west longitude, marked on the Colonia Oaxaca 1:50,000 topographic map.

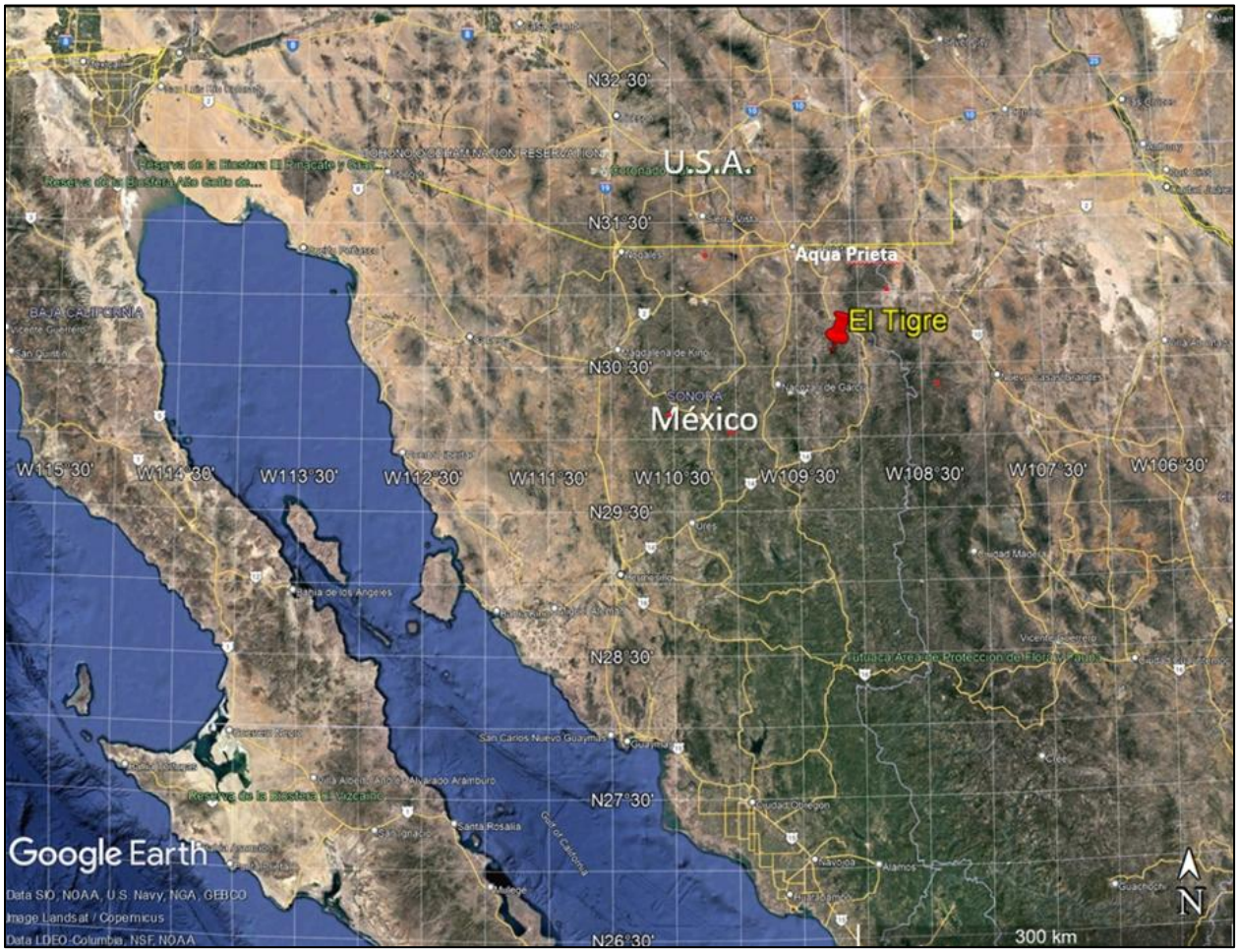
The El Tigre Property consists of 59 Mexican Federal mining concessions totalling 21,775 ha. Four of the concessions are owned by Compañía Minera Talaman S.A. de C.V. ("Talaman") and 55 are owned by Pacemaker Silver Mining S.A. de C.V. ("Pacemaker"). Pacemaker, a subsidiary of Silver Tiger, indirectly holds 100% interest in the four remaining concessions through its 100% ownership of Talaman, such that all 59 concessions are currently controlled by Silver Tiger.

The Property was acquired in November 2015 by Oceanus Resources Corporation ("Oceanus", a precursor Company to Silver Tiger), through the acquisition of all the issued and outstanding common shares of El Tigre Silver Corporation ("El Tigre Silver"), whereby each outstanding El Tigre Silver share was exchanged for 0.2839 of one common share of Oceanus. Following the acquisition of El Tigre Silver, Pacemaker became a 100% indirectly owned Mexican subsidiary of Oceanus. On May 14, 2020, Oceanus announced a name change to Silver Tiger Metals Inc.

Until 2022, the El Tigre Property consisted of nine concessions (El Aguila, Jorge, La Fundadora, Tigre Suertudo, Nik Frac. 2, San Juan, La Carabina Frac 1, La Carabina, Frac 2, and Nik 1 F1). Concession Nik 1 F1 (21,156 ha) expired in 2022 and was subdivided into 51 new valid concessions (Nik 1 F1 D1 to Nik 1 F1 D51) that cover the previous surface of Nik 1 F1 and to which Pacemaker holds legal title. The 59 concessions of the El Tigre Property are all registered with the Registro Público de Mineria as exploitation concessions. EC Rubio, Silver Tiger's Mexican legal counsel, has confirmed that as of June 20, 2025, the concessions are in good standing.

A location map of the El Tigre Property is presented in Figure 1.1.

FIGURE 1.1 EL TIGRE PROPERTY LOCATION MAP



Source: *Micon (2023)*

From Agua Prieta, the El Tigre Property can be reached by driving 75 km south along Mexican Highway 17 to the Town of Equeda, and then 46 km east from there on a dirt road to the El Tigre camp. Large stretches of the road from Esqueda are intermittently maintained by local ranchers on either side of Lake Angostura. Alternate access routes include a crossing at the Lake Angostura dam to the south or at Colonia Morelos or Fresno Ranch to the north. These alternate routes are only viable when the Rio Batiste is low or dry. Access during the monsoon season is hindered by flash floods, which periodically wash out sections of road and generally cause rough road conditions.

In 2023, Silver Tiger completed rehabilitation of a 7 m wide access road and some pluvial works from Colonia Morelos to the exploration camp over 46 km in order that it will be accessible all year round.

The climate of the El Tigre area is typical of the Madrean Archipelago/Sky Island Region, which is semi-arid with bi-seasonal precipitation. Winter precipitation is associated with frontal storms from the Pacific Ocean. Winter conditions generally last from October through May, with the most

intense storms occurring between mid-November and mid-April. Late spring and early summer are typically dry and summer monsoon moisture begins to enter the region in late-June to early-July. Storms are the result of tropical air flowing over heated mountain terrain, with frequent torrential rains occurring during the afternoon and thunderstorms in the evenings.

Temperatures are elevation dependent. In the lowlands, near La Angostura Reservoir, summer temperatures can reach 50°C and winter temperatures can be as low 0°C. At the El Tigre camp site, summer temperatures rarely exceed 40°C and winter temperatures can reach as low as -15°C on the coldest nights.

The El Tigre Property is remote. Food, fuel and lodging are available in Esqueda, a two-to-three-hour drive from the camp. Personnel are currently lodged at the camp, which consists of a 25–person residence, office, shower, washroom, and kitchen facilities. A drill core logging and storage area is also present. Cellular reception is sporadic around the main camp. Satellite internet equipment is present; however, a tower would be required to improve reception. Supplies can be acquired from Esqueda or other communities with proper planning. Heavy equipment or construction materials may require transport from larger cities, such as Hermosillo.

1.2 HISTORY

The Property hosts the historical Lucky Tiger Mine that produced silver and gold intermittently between 1903 and 1938. Mineral exploration was completed by several groups during that time and through the 1970s. Modern exploration was initiated in 1981 by Anaconda Minerals Company through its wholly-owned subsidiary Cobre de Hercules. Anaconda's exploration efforts lasted 29 months and ceased around the time Anaconda terminated all mining and exploration activities in the mid-1980s.

In June 1995, consulting firm Minera de Cordilleras completed a four-hole reverse circulation ("RC") drilling program for a total of 890 m on behalf of a third party. These drill holes were planned to test the concept that the deeper part of the vein system was faulted, such that the mineralized veins were displaced closer to the surface. Assays are available for the drill holes, however, the collar locations are unknown.

1.3 GEOLOGY AND MINERALIZATION

The Sierra El Tigre is one of the large mountain ranges that are part of the Basin and Range Province which is found from northern Nevada to Zacatecas and Jalisco in Mexico. The Sierra El Tigre is part of the massif of the Sierra Madre Occidental and was formed during Cenozoic extensional faulting, which consists of northerly-trending horsts and grabens. Pre-Cenozoic granite and limestone are the oldest rocks exposed in the range and are overlain by remnants of the vast Tertiary rhyolite ignimbrite field of the Sierra Madre Occidental.

Silver, gold, lead, zinc, and copper mineralization occurs in the El Tigre District (the "District") mostly in fissure veins within a narrow, north-trending belt 5.3 km long. The District contains nine known veins. These include the Sooy, El Tigre, Seitz-Kelly and Combination Veins in the southern area and the Aquila, Caleigh, Fundadora, Protectora and Escondida Veins in the northern area of

El Tigre. Silver and gold mineralization in the El Tigre area occurs in both the fissure veins and in a low-grade stockwork halo around the veins.

The veins formed along structurally prepared fissures that generally dip steeply to the west. Vein mineralization consists of quartz and varying proportions of zinc, iron, lead, copper, and silver sulphides with silicified or argillized fragments of host rock. Gold is associated with copper-silver sulphides. The mineralization occurs in discontinuous lenses of elongated high-grade sulphides along the veins and as low-grade impregnations in the vein gangue material. In order of abundance, mineralization consists of pyrite, sphalerite, galena, argentiferous galena, chalcopyrite, tetrahedrite, covellite and gold. Tetrahedrite and galena are the main silver-bearing minerals. Gold occurs in the native state as µm-sized specks and as inclusions in galena and chalcopyrite. The vein host rocks exhibit adularia replacement, with minor silicification, argillization, and propylitization. The El Tigre Veins closely resemble those in quartz-adularia, low sulphidation epithermal deposits.

1.4 EXPLORATION AND DRILLING

Exploration by El Tigre Silver Corporation and Oceanus has included channel sampling of surface mineralization and underground workings, sampling of historical tailings, IP geophysics and diamond drilling. Since summer 2020, exploration activities by Silver Tiger included a channel sampling program of historical underground exploration drifts and surface sampling located on the 3 km of vein extensions that outcrop at surface north of the historical El Tigre Mine. The areas of focus were the Caleigh, Canon Combination, Protectora, and Fundadora Veins. The 2020 channel sampling program was planned to generate additional drill targets and followed-up on the success of the underground channel sampling completed in the same vein extensions in 2019. The 2020 sampling program also returned multiple high-grade values.

Between 1982 and 2013, Anaconda, Minera de Cordilleras and El Tigre Silver Corporation completed a total of 18,114 m of drilling. In 2016 to 2017, Oceanus completed 69 diamond drill holes totalling 12,760 m. Since 2020, Silver Tiger has completed five drilling programs consisting of 478 drill holes totalling 120,006 m on the El Tigre Property. The purpose of the drilling programs was to support updated Mineral Resource Estimates for the El Tigre Property and provide fresh material for metallurgical testing and geotechnical data and information for advanced project development studies. During late 2024, three drill holes totalling 1,031 m were completed by Silver Tiger from an underground level at the El Tigre Property.

High-grade mineralization at the El Tigre Project is open to expansion by drilling down-dip and along strike. In addition to the exploration work completed to date, and based on widely spaced drill holes and average width, length and depth geometry, the Authors have established that the El Tigre mineral deposits contain an additional underground Exploration Target as follows:

- 1 to 4 million tonnes at 200 to 700 g/t AgEq for 23 to 26 Moz AgEq.

The potential quantity and grade of the Exploration Target is conceptual in nature. There has been insufficient work done by a Qualified Person to define this estimate as Mineral Resources. The Company is not treating this estimate as Mineral Resources, and readers should not place undue reliance on this estimate. Even with additional work, there is no certainty that the

estimate will be classified as Mineral Resources. In addition, there is no certainty that this estimate will ever prove to be economically recoverable.

1.5 SAMPLING, ANALYSES AND DATA VERIFICATION

It is the Author's opinion that sample preparation, security and analytical procedures for the El Tigre Project were adequate, and that the data are of satisfactory quality and suitable for use in the current Mineral Resource Estimate. Verification of the El Tigre Project data, used for the current Mineral Resource Estimate, was undertaken by the Authors, and included site visits, due diligence sampling, verification of drilling assay data, and assessment of the available QA/QC data from the historical and recent drilling programs. The Authors consider that there is an adequate correlation between assay values in El Tigre's database and the independent verification samples collected and analyzed, and that the supplied data are of satisfactory quality and suitable for use in this current Mineral Resource Estimate for the El Tigre Project.

1.6 MINERAL PROCESSING AND METALLURGY

Initial preliminary metallurgical testwork of the El Tigre Deposit was completed in 2012 and summarized in a technical report titled "Preliminary Feasibility Study for the El Tigre Silver Project", dated August 15, 2013. The selected process included direct cyanidation followed by Merrill Crowe recovery of Au and Ag at an initial throughput of 200 tpd with future expansion to 400 tpd. The limited amount of cyanidation testwork was undertaken on three composite tailings samples representing visually distinguishable characteristics, however, testwork details are not available.

In August 2022, an initial scoping-level metallurgical testwork program was undertaken at SGS Lakefield ("SGS") located in Ontario, Canada. The objectives of testwork were to develop metallurgical data to evaluate and optimize various processes for the recovery of gold and silver, including whole feed cyanide leaching, Merrill Crowe precipitation, flotation, and heap leach amenability. Mineralogical, environmental, and solid/liquid separation and rheology examinations of both fresh mineralization and leach tailing samples were completed to support the testing program. The subsequent results of this program were presented in the Technical Report titled "NI-43-101 Technical Report and Preliminary Economic Assessment ("PEA") for the El Tigre Project, Pilares – El Tigre Mining District, Sonora, Mexico" Effective Date: October 27, 2023, Report Date: December 14, 2023.

Metallurgical testwork outlined in the PEA focused on the surface mine heap leach mineralized zone to process material in two development phases. Phase I would process 7,500 tpd during the initial three years of production and Phase II would expand production in year four to process 15,000 tpd. Details of the 2022-2023 SGS program are summarized in the report titled, "An Investigation into the El Tigre Deposit," dated October 13, 2023.

As part of this Technical Report, additional metallurgical testing was completed at McClelland Laboratories, Inc. ("MLI") in Sparks, Nevada on mineralization from the proposed starter pit at the El Tigre Project site. Testwork from six locations within the starter pit was undertaken in 2024 to determine recovery variability by heap leach cyanidation processing and to further analyze work completed under the PEA. The testwork results are the foundation of this Technical Report and

support Project process design criteria and process equipment as well as capital and operating costs and financial modelling. Variability in testwork results indicate starter pit mineralization is comprised of multiple geological zones including oxide, transition, and sulphide. This resulted in determination of variable recoveries versus a global recovery assumed in the PEA.

MLI presented the results of the program in a report titled "Report on Metallurgical Testing Program - El Tigre Drill Core Composites" dated August 28, 2024.

Further testwork to coincide with the MLI testing was completed at Laboratorio Tecnológico de Metalurgia ("LTM"), Hermosillo, Mexico. Testwork investigated the zone variability of the proposed mined material for processing. Sample locations were determined by the Silver Tiger geological group.

Table 1.1 summarizes current metallurgical testwork and Process Design Criteria for this Technical Report.

TABLE 1.1 PROCESS DESIGN CRITERIA		
Criteria	**Units**	**Value**
Ore Characteristics		
Specific Gravity	g/cm³	2.6
Plant Availability/Utilization		
Overall Plant Feed-Nominal – Years 1-3	tpy	2,737,500
Overall Plant Feed- Nominal – Years 1-3	tpd	7,500
Overall Plant Feed-Nominal – Years 4 +	tpy	5,475,000
Overall Plant Feed- Nominal – Years 4 +	tpd	15,000
Crushing Plant Availability	%	75
Heap and Processing Plant Availability	%	95
Crushing Product (to pad)	P_{80} inch (mm)	3/8 (9.0)
Process Plant Production		
Plant Feed Characteristics (Life of Mine Average)		
Gold Head Grade	g/t	0.40
Silver Head Grade	g/t	14.9
Metal Recoveries and Process Details		
Anticipated Overall Gold Oxide Recovery- design[1]	%	83
Anticipated Overall Silver Oxide Recovery- design[1]	%	45
Anticipated Overall Gold Sulphide Recovery – design[1]	%	56
Anticipated Overall Silver Sulphide Recovery – design[1]	%	40
Cyanide Addition Rates – design[2]	kg/t	0.25
Lime Addition Rates – design[2]	kg/t	1.5
Merrill Crowe Phase I – design	m^3/h	590
Merrill Crowe Phase II – design	m^3/h	1,180
Merrill Crowe Recovery – Au and Ag	%	99

Source: *D.E.N.M. (2024)*

Notes: *[1] Column testing indicated both higher oxide and sulphide gold from the PEA recovery at a 3/8-inch crush size. In the process design and financial model, these have been discounted by 3% for leaching in the field versus optimum conditions in the laboratory. As noted, recent column testing has indicated sulphide zones are present (15% of the total) that will affect the overall recovery. This variation should be noted in the mine schedule and financial modelling.*

[2] Cyanide Consumption is also discounted for the process design, operating costs, and financial model. Lime is not discounted from the recent column test data.

1.7 MINERAL RESOURCE ESTIMATE

The basis for the PFS is the updated Mineral Resource Estimate completed by the Authors for the El Tigre Project, which has an effective date of June 20, 2025. The updated Mineral Resource Estimate includes pit-constrained, out-of-pit, tailings and stockpile Mineral Resources. A summary of the updated Mineral Resource Estimate and AuEq cut-off grades is provided in Table 1.2.

The El Tigre Project includes the El Tigre Veins, El Tigre Tailings and the El Tigre Low-Grade Stockpile. The databases used for this Mineral Resource update contain a total of 5,875 collar records that contribute directly to the Mineral Resource Estimate, and includes collar, survey, assay, lithology and bulk density data. Assay data includes Au g/t, Ag g/t, Cu%, Pb% and Zn% grades. The drilling extends ~5 km along strike.

The Authors collaborated with Silver Tiger personnel to develop the mineralization models, grade estimates, and reporting criteria for the Mineral Resources at El Tigre. Mineralized domains initially developed by Silver Tiger were reviewed and modified by the Authors. A total of 28 individual mineralized domains have been identified through drilling and surface sampling. Interpreted mineralized wireframes were developed by Silver Tiger geologists for the El Tigre Veins based on logged drill hole lithology, assay grades and historical records. Silver Tiger identified continuous zones of mineralization from assay grades ≥0.30 g/t AuEq with observed continuity along strike and down-dip, using a calculated Ag:Au equivalent ratio of 75:1. The selected intervals include lower-grade material where necessary to maintain wireframe continuity between drill holes. The Authors developed mineralized domains for the El Tigre Low-Grade Stockpile and the El Tigre Tailings based on lithological logging and LiDAR surface topography.

Assay samples were composited to either 1.00 or 1.50 m for the vein domains. No compositing was used for the Low-Grade Stockpiles and Tailings models. Composites were capped prior to grade estimation based on the analysis of individual composite log-probability distributions.

A total of 5,542 bulk density values were taken by Silver Tiger from drill hole core. Mineralized bulk density values were assigned for each of the El Tigre Main Veins based on the median vein measurement. For the El Tigre North Veins, a bulk density of 2.65 t/m^3 was assigned for the veins and a value of 2.42 t/m^3 was assigned for the Protectora Halo. For the Low-Grade Stockpile a value of 1.60 t/m^3 was assigned, and for the Tailings a value of 1.39 t/m^3 was used based on 37 bulk density measurements.

Vein block grades for gold and silver were estimated by Inverse Distance Cubed ("ID^3") interpolation of capped composites using a minimum of four and a maximum of 12 composites. Vein block grades for copper, lead and zinc were estimated by Inverse Distance Squared ("ID^2") interpolation of capped composites using a minimum of four and a maximum of 12 composites. Nearest-Neighbour ("NN") grade interpolation was used for the Low-Grade Stockpiles. For the Tailings, block grades were estimated by ID^2 estimation of capped assays using a minimum of four and a maximum of 12 samples.

For the El Tigre Veins, blocks within 30 m of three or more drill holes or channels were classified as Measured Mineral Resources, and blocks within 60 m of three or more drill holes or channels were classified as Indicated Mineral Resources. All additional estimated blocks within 180 m of a drill hole were classified as Inferred Mineral Resources.

For the Low-Grade Stockpiles, blocks within 15 m of two or more drill holes were classified as Indicated Mineral Resources. All additional estimated blocks were classified as Inferred Mineral Resources. For the Tailings, blocks within 30 m of three or more auger or core drill holes were classified as Measured Mineral Resources. Blocks within 60 m of two or more auger/drill holes/pits or trenches were classified as Indicated Mineral Resources. All additional estimated blocks were classified as Inferred Mineral Resources.

The Authors consider that the block model Mineral Resource Estimates and Mineral Resource classification represent a reasonable estimation of the global Mineral Resources for the El Tigre Project with regard to compliance with generally accepted industry standards and guidelines, the methodology used for grade estimation, the classification criteria used and the actual implementation of the methodology in terms of Mineral Resource estimation and reporting. The Mineral Resources have been estimated in conformity with the requirements of the CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines as required by the Canadian Securities Administrators' National Instrument 43-101. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

TABLE 1.2
MINERAL RESOURCE ESTIMATE (1-14)

Classification	Cut-off AuEq (g/t)	Tonnes (k)	Ag (g/t)	Ag (koz)	Au (g/t)	Au (koz)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)	AgEq (koz)	AuEq (g/t)	AuEq (koz)
Pit-Constrained Oxide													
Measured	0.08	27,732	39	35,034	0.54	609	NA	NA	NA	136	120,923	0.78	693
Indicated	0.08	19,804	10	6,593	0.36	75	NA	NA	NA	74	46,979	0.42	266
Meas + Ind	**0.08**	**47,536**	**27**	**41,626**	**0.46**	**709**	**NA**	**NA**	**NA**	**110**	**167,902**	**0.63**	**959**
Inferred	0.08	4,475	10	1,453	0.38	18	NA	NA	NA	77	11,045	0.44	63
Pit-Constrained Sulphide													
Measured	0.12	6,031	31	6,042	0.54	105	NA	NA	NA	96	18,667	0.79	154
Indicated	0.12	1,723	14	756	0.46	25	NA	NA	NA	69	3,817	0.57	32
Meas + Ind	**0.12**	**7,755**	**27**	**6,798**	**0.52**	**131**	**NA**	**NA**	**NA**	**90**	**22,484**	**0.74**	**185**
Inferred	0.12	463	74	1,096	0.52	8	NA	NA	NA	136	2,034	1.11	17
Pit-Constrained Total													
Measured	0.08/0.12	33,763	38	41,076	0.54	588	NA	NA	NA	129	139,590	0.78	846
Indicated	0.08/0.12	21,528	11	7,348	0.36	252	NA	NA	NA	73	50,796	0.43	298
Meas + Ind	**0.08/0.12**	**55,291**	**27**	**48,425**	**0.47**	**840**	**NA**	**NA**	**NA**	**107**	**190,386**	**0.64**	**1,144**
Inferred	0.08/0.12	4,939	16	2,550	0.39	62	NA	NA	NA	82	13,078	0.50	79
Out-Of-Pit													
Measured	1.28	1,010	453	14,698	1.05	34	0.13	0.54	0.92	589	19,118	7.09	230
Indicated	1.28	1,374	398	17,588	1.05	47	0.08	0.24	0.41	509	22,482	6.13	271
Meas + Ind	**1.28**	**2,384**	**421**	**32,286**	**1.05**	**81**	**0.10**	**0.37**	**0.63**	**543**	**41,600**	**6.54**	**501**
Inferred	1.28	4,100	294	38,822	1.58	208	0.32	0.75	0.89	497	65,556	6.01	792
Tailings													
Measured	0.45	146	79	371	0.29	1	NA	NA	NA	103	487	1.22	6
Indicated	0.45	706	77	1,752	0.26	6	NA	NA	NA	99	2,252	1.17	27

TABLE 1.2 MINERAL RESOURCE ESTIMATE (1-14)													
Classification	**Cut-off AuEq (g/t)**	**Tonnes (k)**	**Ag (g/t)**	**Ag (koz)**	**Au (g/t)**	**Au (koz)**	**Cu (%)**	**Pb (%)**	**Zn (%)**	**AgEq (g/t)**	**AgEq (koz)**	**AuEq (g/t)**	**AuEq (koz)**
Meas + Ind	**0.45**	**852**	**77**	**2,118**	**0.27**	**7**	**NA**	**NA**	**NA**	**202**	**2,739**	**1.18**	**32**
Inferred	0.45	52	81	134	0.27	1	NA	NA	NA	104	173	1.23	2
Stockpile													
Measured	0.46	0	0	0	0	0	0	0	0	0	0	0	0
Indicated	0.46	101	181	586	0.92	3	0.02	0.23	0.10	256	826	3.17	10
Meas + Ind	**0.46**	**101**	**181**	**586**	**0.92**	**3**	**0.02**	**0.23**	**0.10**	**256**	**826**	**3.17**	**10**
Inferred	0.46	18	146	83	0.46	0.3	0.02	0.17	0.09	183	104	2.27	1
Total													
Measured	NA	34,920	50	56,145	0.55	623	0.13	0.54	0.92	142	159,193	0.96	1,082
Indicated	NA	23,708	36	27,275	0.40	308	0.07	0.24	0.39	100	76,350	0.79	605
Meas + Ind	**NA**	**58,628**	**44**	**83,415**	**0.49**	**931**	**0.09**	**0.36**	**0.61**	**125**	**235,543**	**0.90**	**1,688**
Inferred	NA	9,108	142	41,589	0.92	271	0.32	0.75	0.89	269	78,911	2.99	874

Notes:

1. *Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*
2. *The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.*
3. *The Mineral Resources were estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.*
4. *Historically mined areas were depleted from the Mineral Resource model.*
5. *Prices used are US$2,400/oz Au, US$28/oz Ag, US$4.25/lb Cu, US$0.97/lb Pb and US$1.30/lb Zn.*
6. *The pit-constrained respective oxide and sulphide AuEq cut-off grades of 0.08 and 0.12 g/t were derived from 45% Ag and 83% Au oxide process recovery, 40% Ag and 56% Au sulphide process recovery, US$5.25/t process and G&A cost. The constraining pit optimization parameters were $2.00/t mining cost and 45° pit slopes.*
7. *The out-of-pit AuEq cut-off grade of 1.28 g/t was derived with 93% Ag and 89% Au process recovery, US$28/t process and G&A cost, and a $60/t mining cost. The out-of-pit Mineral Resource grade blocks were quantified above the 1.28 g/t*

AuEq cut-off, below the constraining pit shell and within the constraining mineralized wireframes. Out–of-Pit Mineral Resources are restricted to the El Tigre Main Veins, which exhibit historical continuity and reasonable potential for extraction by cut and fill and long hole mining methods.

8. *The Stockpile AuEq cut-off grade of 0.46 g/t was derived from 85% Ag and 85% Au process recovery, US$28/t process and G&A cost, and a $2/t mining cost.*
9. *The Tailings AuEq cut-off grade of 0.45 g/t was derived from 83% Ag and 82% Au process recovery, US$28.72/t process and G&A cost.*
10. *AgEq and AuEq were calculated at an Ag/Au ratio of 178:1 (oxide) and 120:1 (sulphide) for pit-constrained Mineral Resources.*
11. *AgEq and AuEq were calculated at an Ag/Au ratio of 82:1 for out-of-pit Mineral Resources.*
12. *AgEq and AuEq were calculated at an Ag/Au ratio of 86:1 for Low-Grade Stockpile Mineral Resources.*
13. *AgEq and AuEq were calculated at an Ag/Au ratio of 85:1 for Tailings Mineral Resources.*
14. *Totals may not sum due to rounding.*

1.8 MINERAL RESERVE ESTIMATE

The El Tigre Project will consist of single large surface mining operation. Underground mining is another option and is included as a PEA in Section 24 of this Technical Report, therefore the Mineral Reserves stated herein are based solely on a surface scenario. The Mineral Reserves are summarized in Table 1.3.

The Mineral Reserves are the portion of the Mineral Resource that has been deemed economically mineable by the surface mine production plan and within the designed surface mine. The Mineral Reserves are derived only from Mineral Resources classified as Measured or Indicated. Inferred Mineral Resources are considered as waste material in this Technical Report.

TABLE 1.3
MINERAL RESERVE ESTIMATE (1-7)

Classification	Ore Tonnes (k)	Au (g/t)	Ag (g/t)	AuEq (g/t)	AgEq (g/t)	Contained Au (koz)	Contained Ag (koz)
Proven							
Oxide	19,564	0.40	16.4	0.50	83.4	252	10,316
Sulphide	4,655	0.52	7.6	0.58	65.2	78	1,137
Total Proven	**24,220**	**0.43**	**14.7**	**0.52**	**79.9**	**329**	**11,453**
Probable							
Oxide	14,800	0.37	13.7	0.45	74.3	176	6,519
Sulphide	1,272	0.43	33.8	0.73	82.1	18	1,382
Total Probable	**16,073**	**0.37**	**15.3**	**0.47**	**74.9**	**194**	**7,901**
Proven and Probable							
Oxide	34,365	0.39	15.2	0.48	79.5	428	16,834
Sulphide	5,928	0.50	13.2	0.61	68.8	95	2,520
Total Proven & Probable	**40,292**	**0.40**	**14.9**	**0.50**	**77.9**	**523**	**19,354**

Notes:

1. *The Mineral Reserve Estimate was prepared in accordance with 2014 CIM Definition Standards by Andrew Bradfield, P.Eng.*
2. *The effective date of the Mineral Reserve is June 20, 2025.*
3. *Au price of US$1,900/oz and Ag price of US$23/oz are assumed.*
4. *The Mineral Reserves are based on the surface mine production schedule.*
5. *Mineral Reserves include a combined ore loss and dilution of 2.9%.*
6. *Oxide and Sulphide Mineral Reserves are reported using a cut-off grade of 0.11 g/t AuEq and 0.16 g/t AuEq, respectively. Ore and waste rock mining cost of $2.60/t was assumed, with a process cost of $4.33/t (assumes 15,000 tpd production), G&A cost of $1.00/t, a royalty of 0.5% of revenue, and pit slopes ranging from 46 to 49 degrees. Heap leach recoveries were Oxide 83% Au and 45% Ag, Sulphide 56% Au and 40% Ag.*
7. *Rounding of some figures may lead to minor discrepancies in total.*

To address dilution, the 2.5 m x 5 m x 5 m (X-Y-Z) Mineral Resource block model was regularized from a Geovia GEMS™ volume percent model to a whole block selective mining unit ("SMU")

model. This results in waste and ore portions of a block being combined into a single block volume weighted value. The SMU model is considered a diluted model and was used for pit optimization, production scheduling, and is the basis for the Mineral Reserve Estimate. Net dilution incorporates both the total dilution and ore loss, and is estimated at approximately 2.9%.

1.9 MINING METHODS

The El Tigre Project will consist of a single surface mining operation using conventional equipment. Mining will be initiated in Year -1 (pre-stripping period) and will be completed in Year 9.

Based on kinematic analysis and limit equilibrium approach, an updated design of the PFS pit was assessed considering slightly to moderately weathered rock with quadruple 5 m benches (20 m) for all sectors. For the West Walls a bench face angle ("BFA") of 70° and inter-ramp angle ("IRA") of 49° was recommended while for the East Walls a BFA of 65° and IRA of 46° were suggested. Slopes of appreciable vertical extent were not anticipated on the North and South walls and followed the East Wall mineralized geometry.

The initial ore heap leaching rate (Years 1 to 3) is 7,500 tpd (2.74 Mt per annum ("Mtpa")). In Year 4 the crushing rate will increase to 15,000 tpd (5.5 Mtpa) and this rate is maintained to the end of the mine life.

Figure 1.2 presents the surface mine design and is planned to be mined in three pushbacks or phases. Slope design recommendations were developed for the El Tigre Project site based on data collected as part of a specific geotechnical and hydrogeological investigation. This data was reviewed and interpreted to develop relevant parameters for use in design assessments.

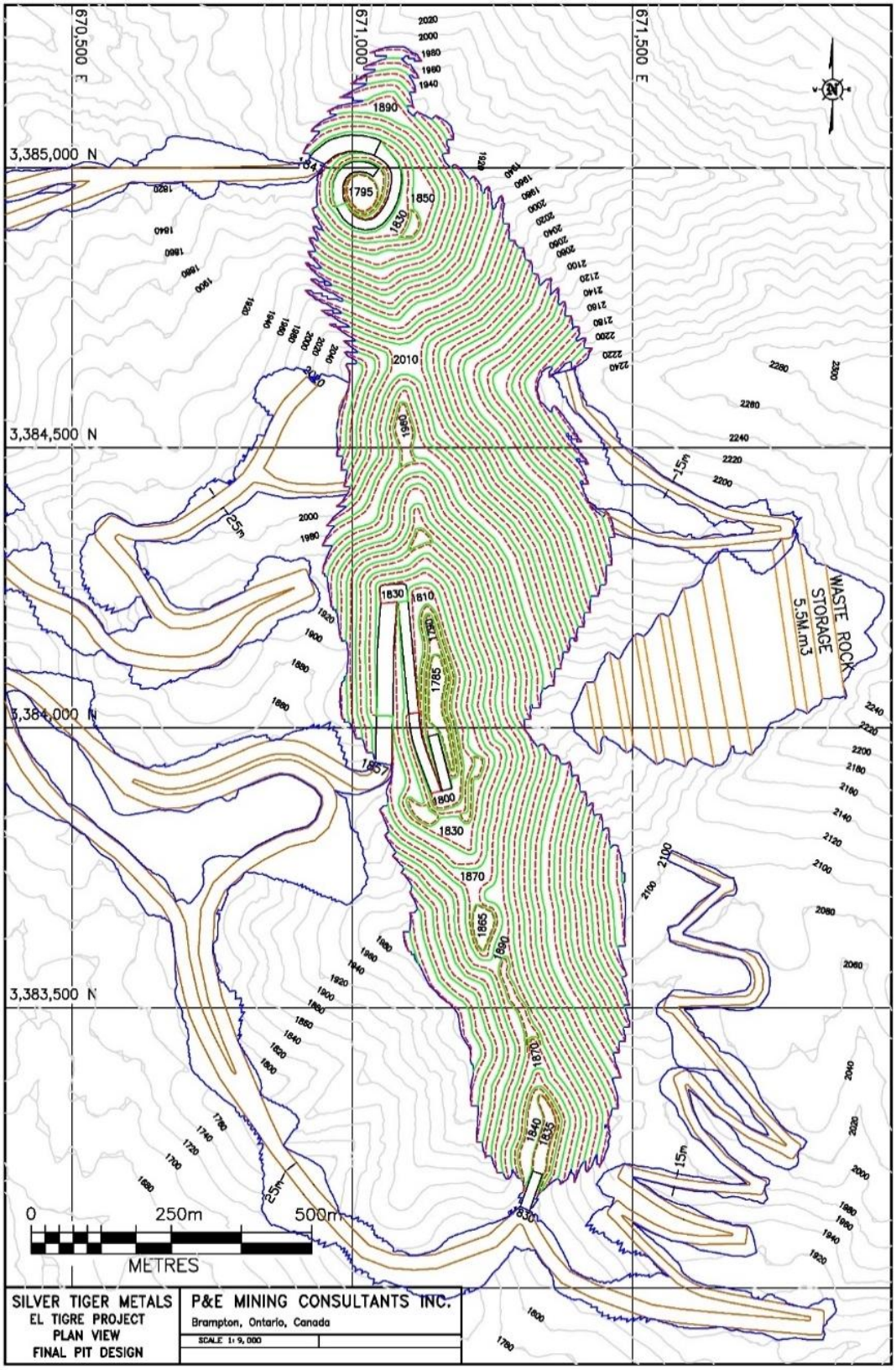
The El Tigre Mine is envisioned to be a contract mining operation. Contractor budgetary quotations for mining the Stockwork Zone were obtained.

The Mining Contractor will undertake all drilling, loading, hauling, and mine site maintenance activities. A separate Blasting Contractor will provide blasthole loading and charging services using an ammonium nitrate fuel oil mixture ("ANFO"). The Owner will provide overall mine management and technical services, such as mine planning, grade control, geotechnical, and surveying services.

It is expected that mid-sized diesel-hydraulic excavators and diesel-powered front-end loaders will be used. The anticipated truck size is 91 tonnes, similar to the CAT 777 truck, although alternate truck sizes may actually be used.

Several waste rock storage facilities will be utilized depending on where the mining activity is occurring. The Main Waste Rock Storage Facility will be located to the west of the surface mine. Two smaller waste rock facilities will be required when stripping the upper benches since haul distances to the Main Facility will be lengthy. 68.3 Mt of waste rock will be mined, with 40.3 Mt ore, for a total of 108.6 Mt at a strip ratio of 1.7:1.

FIGURE 1.2 SURFACE MINE DESIGN



Source: *P&E (2024)*

1.10 RECOVERY METHODS

The testwork results presented in Section 13 of this Technical Report form the basis for flowsheet development and design criteria. The process plant design is based on a nominal 7,500 tpd (Years 1 to 3) and 15,000 tpd (Years 4 to 9) of ore with average grades of 0.40 g/t Au and 14.9 g/t Ag. The recovery methods are consistent with the crushing, leach pad loading, leaching, Merrill Crowe precipitation, refinery, and support processing methods presented in the 2023 PEA.

Revised recovery methods included in this Technical Report are as follows:

- Process plant facility located adjacent to the heap pond infrastructure to improve process operability and eliminate distant water pumping and power demand. This is illustrated in Section 18 of this Technical Report.

- Water will be supplied from two surface water wells. Infrastructure also includes high head pumps and distribution system to deliver fresh water to the site.

- Selected equipment for crushing and recovery during Phase I has been "upsized" to lower Phase II direct capital requirements and related construction time. This includes the primary crusher system and material handling, discharge conveyors to the leach pad loading, and Merrill Crowe deaeration, vacuum and process pumps. The refinery has also been designed to treat Phase II metal production.

The process plant flowsheet design comprises of three stage conventional crushing, material handling of crushed product and loading onto lined heap leach pads. Solution ponds and pumping system allows irrigation of loaded ore and subsequent collection of the pregnant solution. The pregnant solution is pumped to the Merrill Crowe recovery facility for precipitation of gold and silver in solution and filtered accordingly. The resultant precipitate will be smelted at an on-site refinery to produce doré bars for market. The barren solution from the recovery process is recirculated to the heap leach pond (barren) for cyanide addition and pumping to the heaps for leaching.

Unit operations and support facilities includes the following:

- ROM material and primary crushing with internal stockpile;

- Secondary cone(s) crushing with screens;

- Tertiary cone(s) crushing and screens (Final crush size of P_{80} – 3/8 in.);

- Material handling and closed circuit crushing and heap leach pad loading;

- 80 mm lined (LLDPE) heap pads capable of the supporting the entire Mineral Reserve (two stages to allow placement of life-of-mine 40.3 Mt). Phase I – 168,000 m^2 and Phase 2 – 248,000 m^2;

- 60 mm lined (LLDPE) solution ponds – barren, pregnant, and emergency ponds complete with internal pumping, piping, and flow distribution system to the pads;

- Conventional Merrill Crowe (zinc precipitation) process plant;
- Doré refinery with dust collection system;
- Regent preparation facilities (main plant and pad);
- Assay and metallurgical laboratory; and
- Utilities (water, air, standby power).

Make-up water for reagent mixing along with water evaporation and general process requirements will be pumped from two wells as noted above.

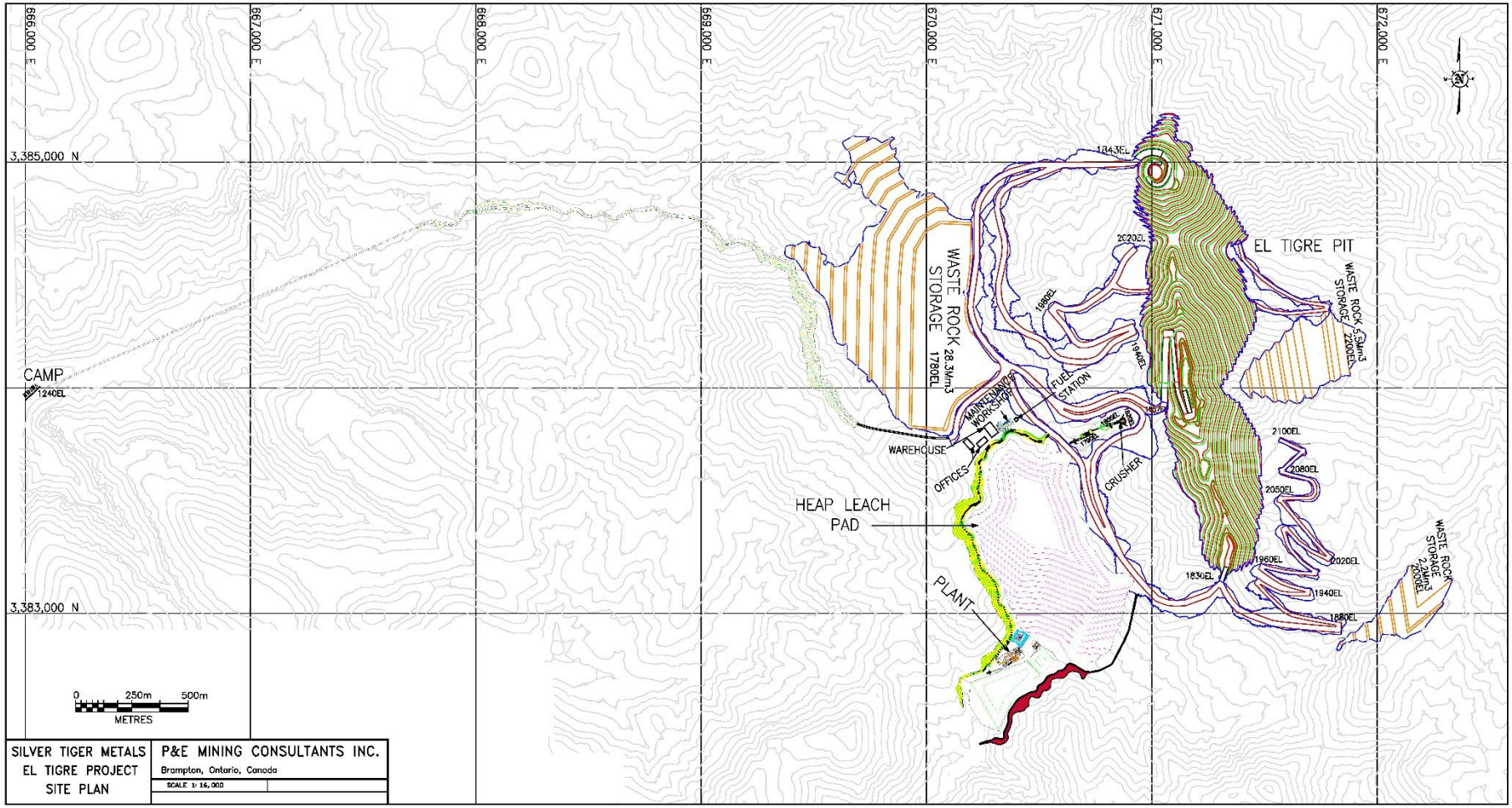
1.11 PROJECT INFRASTRUCTURE

Current infrastructure at the El Tigre Project consists of an exploration camp that has accommodation for drillers and site administration personal. Power for the camp is supplied by diesel generators, and water for drilling is supplied by existing underground workings (Level 7). There are existing impounded tailings on site and low-grade stockpile material.

Permanent road access to the site is from Colonia Morelos, located in the Town of Agua Prieta. Total distance to the site is 46 km and it traverses the mountain ranges of the Sierra Pilares de Teras, Sierra las Delicias, and the Sierra de Enmedio. The road was upgraded in 2023 so it is accessible all year round. After final upgrading, the road will be adequate for personnel movement, access for process plant construction equipment and material, mining and drilling equipment, consumables and safety and emergency requirements.

Figure 1.3 shows the major surface infrastructure proposed at the El Tigre Project.

FIGURE 1.3 PLANNED SURFACE INFRASTRUCTURE AT THE EL TIGRE PROJECT



Source: *P&E (2024)*

Major infrastructure will consist of: surface mine and waste rock storage facilities; three-stage crushing circuit; heap leach pad with pregnant, barren, and overflow ponds; process plant with Merrill Crowe circuit and refinery for doré production, and assay laboratory; buildings such as a maintenance workshop, admin offices, warehouse; diesel fuel storage and distribution station; explosives storage; camp facility; and electrical power supply from the national grid, and distribution.

Water needed for infrastructure and the process plant will be supplied via a designed pump system from two surface wells.

Power to the El Tigre Property will be supplied by the national grid via a 72 km 34.5 kV overhead power line. The routing of the line will be from Esqueda. Overhead power lines will connect 13.8 kV, three phase and 60 Hz via a sub-station located near the process plant area.

The Authors are of the opinion that there appears to be no obvious impediments to building a surface mine, processing or heap leach facility within the area of the El Tigre Project Concessions.

1.12 MARKET STUDIES AND CONTRACTS

Metal prices are based on an approximate average of December 31, 2025, three-year monthly trailing averages, and Consensus Economics Inc. long term price forecasts, and are presented in Table 1.4. The Mexican Peso:US Dollar exchange rate is based on the approximate past three-year average. The metal prices and exchange rate are subject to spot market conditions. There are no metals streaming or hedging agreements in place.

TABLE 1.4
METAL PRICES AND EXCHANGE RATE

Item	Price
Gold (US$/oz)	3,200
Silver (US$/oz)	38.00
Exchange Rate ($MXN:US$)	20

Note: *$MXN = Mexican peso, US$ = United States dollars.*

Currently there are no contracts in place that are material to the El Tigre Project.

1.13 ENVIRONMENTAL STUDIES, PERMITS, AND SOCIAL OR COMMUNITY IMPACTS

Exploration activities have been completed under annual permits from the Mexican government Secretary of the Environment and Natural Resources ("SEMARNAT"). An extensive list of Federal and State permits will be required before mining can commence, along with environmental impact studies. Silver Tiger has engaged the Mexican firm CIMA to conduct an environmental baseline study. CIMA has completed a socio-economic baseline study that considers economic, cultural, social, demographic, and geographical aspects of the local communities. On July 27, 2023, an Environmental Impact Statement on El Tigre was submitted and is currently being

evaluated by the regulatory authorities. The mining Project is expected to represent minimal risk to the processes and structure of the local ecosystem, and to the roads, livestock and agricultural activities. The baseline study concluded that there are no archaeological zones within the Project environmental system.

1.14 CAPITAL AND OPERATING COSTS

All costs are presented in Q3 2024 US dollars. No provision has been included in the cost estimates to offset future escalation. A contingency of 12% is added to all capital costs ("CAPEX"). No contingency is added to operating costs ("OPEX").

The total initial CAPEX of the El Tigre Project is estimated at $86.8M. Expansion and sustaining capital costs incurred during the nine production years are estimated to total $20.1M and $6.2M, respectively. Total capital costs over the life-of-mine ("LOM") are estimated at $113.1M and are presented in Table 1.5.

TABLE 1.5
CAPITAL COST SUMMARY

Item	Initial ($k)	Expansion ($k)	Sustaining ($k)	Total ($k)
Process Plant Direct Costs	42,851	13,584	1,600	58,034
Process Plant Indirect Costs	8,121	-	-	8,121
Mining Direct Costs	2,660	4,362	3,956	10,978
Mine Pre-stripping	3,362	-	-	3,362
Infrastructure	20,489	-	-	20,489
Subtotal	**77,483**	**17,946**	**5,556**	**100,985**
Contingencies @ 12%	9,298	2,199	622	12,118
Total	**86,780**	**20,145**	**6,178**	**113,103**

The initial capital cost estimate addresses the engineering, procurement, construction and start-up of the El Tigre Project, with an 18-month construction period to develop a surface mine, build a process plant capable of treating 7,500 tpd, prepare a heap leach pad, and install associated ancillary surface facilities.

Expansion capital costs estimated at $20.1M are incurred in production years three and four to prepare haul roads to the next surface mine development phase and increase the process plant and heap leach pad capacity to 15,000 tpd.

Sustaining capital represents capital expenses for additional costs that are not included in the normal operating costs, equipment purchases that will be necessary during the operating life of the Project, heap leach pad capacity increases, and all capital costs associated with surface mining. LOM sustaining capital is estimated at $6.2M.

The operating costs estimate includes the cost of mining, processing, and General and Administration ("G&A") services. Total OPEX over the LOM is estimated at $491.0M, averaging $12.19/t processed, as presented in Table 1.6.

TABLE 1.6
OPERATING COST SUMMARY

Area	Unit Cost	LOM Cost ($k)
Surface Mining ($/t mined)	2.24	
Heap Leach and Processing Phase I ($/t processed)	5.79	
Heap Leach and Processing Phase II ($/t processed)	4.74	
OPEX ($/t processed)		
Heap Leach and Processing	4.95	199,604
Surface Mining	5.96	240,120
G&A	1.28	51,300
Total	**12.19**	**491,024**

An electrical power rate supplied by Commission Federal de Electricity ("CFE") has been quoted at $0.113/kWh. The delivered diesel price has been quoted by a reputable fuel distribution firm at $1.11/litre.

The Project is subject to a 1.0% net smelter return ("NSR") mining duty tax payable to the Mexican government. Total costs associated with this NSR royalty tax over the LOM are estimated at $16.2M.

Costs to be incurred after the LOM plan were estimated by CIMA at $6.2M to close and rehabilitate the Project site.

Cash costs over the LOM, including Mexican mining taxes, are estimated to average $11.70/oz AgEq or $997/oz AuEq. All-In Sustaining Costs ("AISC") over the LOM are estimated to average $14.53/oz AgEq or $1,238/oz AuEq and include closure costs.

1.15 ECONOMIC ANALYSIS

Cautionary Statement - The results of the economic analyses discussed in this Technical Report represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented herein.

The El Tigre Project economic evaluation conclusions are summarized in Table 1.7. At base case metal prices of US$3,200/oz Au and US$38/oz Ag the Project has an estimated US$455.6M after-tax net present value ("NPV") at a 5% discount rate ("NPV5%"), and an after-tax internal rate of

return ("IRR") of 66%. The payback period is estimated to be 1.4 years from the start of production.

TABLE 1.7
ECONOMIC EVALUATION SUMMARY

Item	Pre-Tax	After-Tax
NPV0% ($M)	1,000.8	625.3
NPV5% ($M)	737.2	**455.6**
NPV7% ($M)	656.5	403.6
IRR (%)	87.7	**65.7**
Payback period (years)	1.2	**1.4**

Table 1.8 provides further details on the Project financial model.

TABLE 1.8
PROJECT FINANCIAL SUMMARY

Assumption / Result	Unit	Value	Assumption / Result	Unit	Value
Ore Mined	kt	40,292	Net Revenue	US$M	1,623.9
Waste Rock Mined	kt	68,321	Initial Capital	US$M	86.8
Strip Ratio	w:o	1.7:1	Expansion Capital	US$M	20.1
Silver Grade	g/t	14.9	Sustaining Capital	US$M	6.2
Gold Grade	g/t	0.40	Mining Costs	$/t Mined	2.24
Silver Recovery (Oxide/Sulphide)	%	45 / 40	Processing Costs (Phase I and Phase II)	$/t Ore	5.79/4.74
Gold Recovery (Oxide/Sulphide)	%	83 / 56	G&A Costs	$/t Ore	1.27
Silver Price	US$/oz	38.00	Operating Cash Cost	US$/oz AgEq	11.7
Gold Price	US$/oz	3,200	All-in Sustaining Cost	US$/oz AgEq	14.5
Payable Silver Metal	Moz	8.57	After-Tax NPV (5% discount)	US$M	455.6
Payable Gold Metal	koz	408	Pre-Tax NPV (5% discount)	US$M	737.2
Payable AgEq	Moz	42.9	After-Tax IRR	%	65.7
Production and Reclamation	years	10	Pre-Tax IRR	%	87.7
Average Material Mined	tpd	30,000	After-Tax Payback Period	years	1.4

Table 1.9 presents a metal price sensitivity analysis and Table 1.10 presents a cost sensitivity analysis.

TABLE 1.9
SILVER AND GOLD PRICE SENSITIVITY NPV, IRR AND PAYBACK

Sensitivity	-20%	-10%	Base Case	+10%	+20%
Silver Price (US$/oz)	30	34	**38**	42	46
Gold Price (US$/oz)	2,560	2,880	**3,200**	3,520	3,840
After-Tax NPV (5%) (US$M)	303.9	379.8	**455.6**	531.5	607.4
After-Tax IRR (%)	49.9	58.1	**65.7**	73.0	79.9
After-Tax Payback (years)	1.7	1.5	**1.4**	1.3	1.2

TABLE 1.10
CAPITAL AND OPERATING COST SENSITIVITY OF NPV AND IRR

Sensitivity	-20%	-10%	Base Case	+10%	+20%
Operating Costs – NPV_5 (US$M)	502.0	478.8	**455.6**	432.5	409.3
Operating Costs – IRR (%)	70.7	68.2	**65.7**	63.2	60.6
Capital Costs – NPV_5 (US$M)	470.2	462.9	**455.6**	448.3	441.0
Capital Costs – IRR (%)	77.7	71.2	**65.7**	61.1	57.0

The after-tax base case NPV's and IRR's are most sensitive to metal prices followed by operating costs and capital costs.

1.16 RISKS AND OPPORTUNITIES

A preliminary risk analysis was completed on the Project by the Authors using a low-medium-high ranking system. The highest risk items were identified to be mining contractor cost assumptions and variable domain metallurgical recoveries (including oxide, transition and sulphide). The main opportunity is that the Deposit remains open along strike and down dip, and there is potential to increase the current Mineral Resource Estimate.

1.17 CONCLUSIONS

The base case surface mine LOM is scheduled over a production period of nine years. The Proven and Probable Mineral Reserve Estimate for the El Tigre Project totals 40.3 Mt at average grades of 0.40 g/t Au and 14.9 g/t Ag. Approximately 60% of the Mineral Reserve is classified as Proven.

Based on the work undertaken to date, as summarized in this Technical Report, and the individual Qualified Persons conclusions listed in Section 25, the base case operating plan has identified a viable and attractive development opportunity. Overall Project risks are perceived as low.

Current base case economic analysis indicates that the El Tigre Project is forecast to be profitable. It is recommended that Silver Tiger continue the Project development plans, and implement the recommendations set out in Section 26 of this Technical Report.

1.18 RECOMMENDATIONS

The El Tigre Project contains a significant precious metal Mineral Reserve and the Authors recommend that Silver Tiger proceed to develop the Project. Silver Tiger has indicated that it plans to proceed straight to detailed engineering and project construction as soon as the surface mine and processing permits are in place. Detailed engineering (FEED) is estimated to take nine months to complete at a cost of $1.0 M.

It is also recommended that Silver Tiger complete further metallurgical testwork to confirm previous metallurgical investigations of the El Tigre Project Deposits; and undertake additional geotechnical and hydrogeological studies to confirm aspects of the completed analysis. A minimal drilling program designed to potentially upgrade Inferred Mineral Resources contained within the surface mine design is also recommended.

1.19 PRELIMINARY ECONOMIC ASSESSMENT

The 2025 PEA is an alternative development option done at the conceptual level based on Mineral Resources that are considered amenable to underground mining methods.

The 2025 PEA mine plan is partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be classified as Mineral Reserves, and there is no certainty that the 2025 PEA based on these Mineral Resources will be realized.

The information presented in Sections 2 to 14 of this Technical Report, which includes the Mineral Resource Estimate, also pertains to the 2025 PEA, as do Section 23 and Sections 25 to 27. Information relating solely to the 2025 PEA is summarized in the following sub-sections.

1.19.1 Mineral Processing and Metallurgy

Metallurgical testwork on the El Tigre underground veins has been completed by SGS Lakefield, Metso, and LTM Hermosillo between 2022 and 2025 on Black Shale, Deep Sulphide, and Vein mineralization. SGS conducted composite preparation, mineralogical analyses, comminution testing, and extensive flotation programs, including sequential and locked-cycle evaluations. Composite weights ranged from approximately 45 to 56 kg, with representative charges generated through riffle splitting. Rougher flotation produced high silver recoveries, with Black Shale achieving up to 94% Ag and Deep Sulphide up to 99% Ag, while concentrate grades reached 1,560 and 2,509 g/t Ag, respectively. Cleaner and locked-cycle testing confirmed that the mineralization can produce saleable Cu, Pb, Zn, and bulk concentrates. Metso thickening and filtration testwork defined achievable underflow densities of 50 to 55% solids and cake moistures of 17 to 23%.

LTM bottle roll testing on 16 vein samples demonstrated gold and silver extractions of up to 98.38% and 96.07%, respectively. Limitations were related to liberation requirements and variable sample representativeness. Results indicate the mineralization is amenable to conventional flotation and cyanidation, although further optimization is required to refine recovery assumptions.

1.19.2 Underground Mining Methods

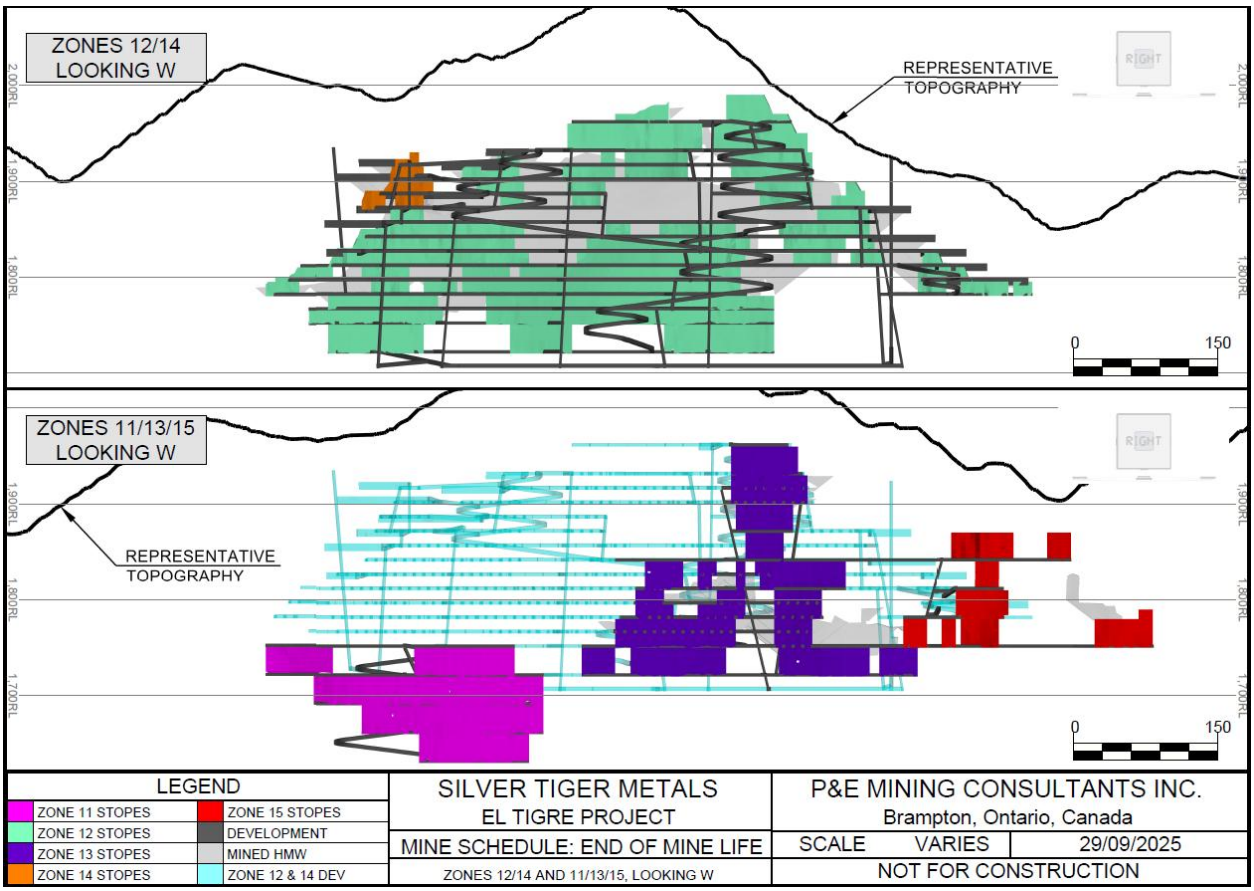
The El Tigre Underground Project is planned as a contractor operated longhole mining operation with cemented paste backfill supplemented by minor mechanized cut and fill mining and recovery of existing low grade and tailings stockpiles. Underground mining contributes approximately 4,000 kt of mineralized material, with 119 kt from a historical low-grade stockpile and 904 kt from a historical tailings stockpile, for a total of 5,022.8 kt grading over the LOM on average 331.7 g/t AgEq, 251.3 g/t Ag, 0.91 g/t Au, 0.033% Cu, 0.099% Pb and 0.166% Zn; base metals provide no economic benefit under the cyanidation process. Underground production is scheduled at 290 kt per annum ("ktpa"), giving an underground mine life of approximately 14 years and an overall Project life of approximately 17 years, including process plant commissioning with low-grade stockpile material in Year -1 and processing of the tailings stockpile at 365 ktpa after process plant feed from underground mining is depleted.

Longitudinal and transverse longhole stopes, which account for 84% of underground tonnes, are arranged on 30 m level intervals with 15 m sublevels. Typical strike lengths are approximately 25 m for areas utilizing longitudinal retreat mining and 10 m for areas utilizing transverse mining. Stopes are supported by cemented paste backfill with strengths of 0.5, 1.0 or 1.4 MPa depending on span. Crown pillars follow a 2:1 thickness-to-span ratio with a minimum thickness of 15 m. Average internal dilution is estimated at 50.4%, external dilution at 4.8% and overall mining recovery at 93.8%, comprising 99.0% for development and 93.0% for production.

The underground mining fleet comprises, at steady state, 7-t and 10-t load-haul-dump units, 30-t haul trucks, longhole drills, development drill jumbos, rock bolters, shotcrete machines and ancillary support equipment sized for the planned production rate. Materials handling and storage consists of dry stack tailings for cemented paste backfill, waste development storage stockpiles and dedicated stockpiles for low-grade development and historical backfill material. Approximately 437.1 kt of historical backfill and 2,438.9 kt of development waste, totaling 2,876.0 kt, are mined over the LOM. The mine plan includes Inferred Mineral Resources that are too speculative geologically to be classified as Mineral Reserves, and there is no certainty that these Inferred Mineral Resources will be converted to Mineral Reserves or that the mine plan based on them will be realized.

A longitudinal projection of the planned underground workings for Zones 11 to 15 is presented in Figure 1.4.

FIGURE 1.4 LONGITUDINAL PROJECTION AT END OF UNDERGROUND LOM, ZONES 11 TO 15



Source: *P&E (2025)*

1.19.3 Recovery Methods

Processing of the El Tigre mineralized material will begin with historical tailings and stockpile sources that average 78 g/t Ag and 0.27 g/t Au for tailings and 176 g/t Ag and 0.85 g/t Au for stockpiles, followed by underground feed that averages 293 g/t Ag and 1.05 g/t Au. Pre-production operations will treat 1,000 tpd while production from the underground veins will be processed at 800 tpd within a 350 day/year operating schedule. The flowsheet includes two stage crushing, ball milling, pre-leach thickening, six stage leaching, counter current decantation, and Merrill Crowe precipitation to produce gold and silver doré. Based on metallurgical testing, expected metal recoveries are 82% Au and 83% Ag for tailings, 92% Au and 71% Ag for stockpiles, and 89.7% Au and 84.4% Ag for underground material. Water will be supplied from two surface wells. Reagents include lime and sodium cyanide. Tailings will be dewatered for dry stack placement and will support paste backfill. Infrastructure includes a metallurgical laboratory, reagent preparation systems, and grid power supplied by a 34.5 kV transmission line.

1.19.4 Project Infrastructure

Major surface infrastructure proposed at the El Tigre Project, will consist of:

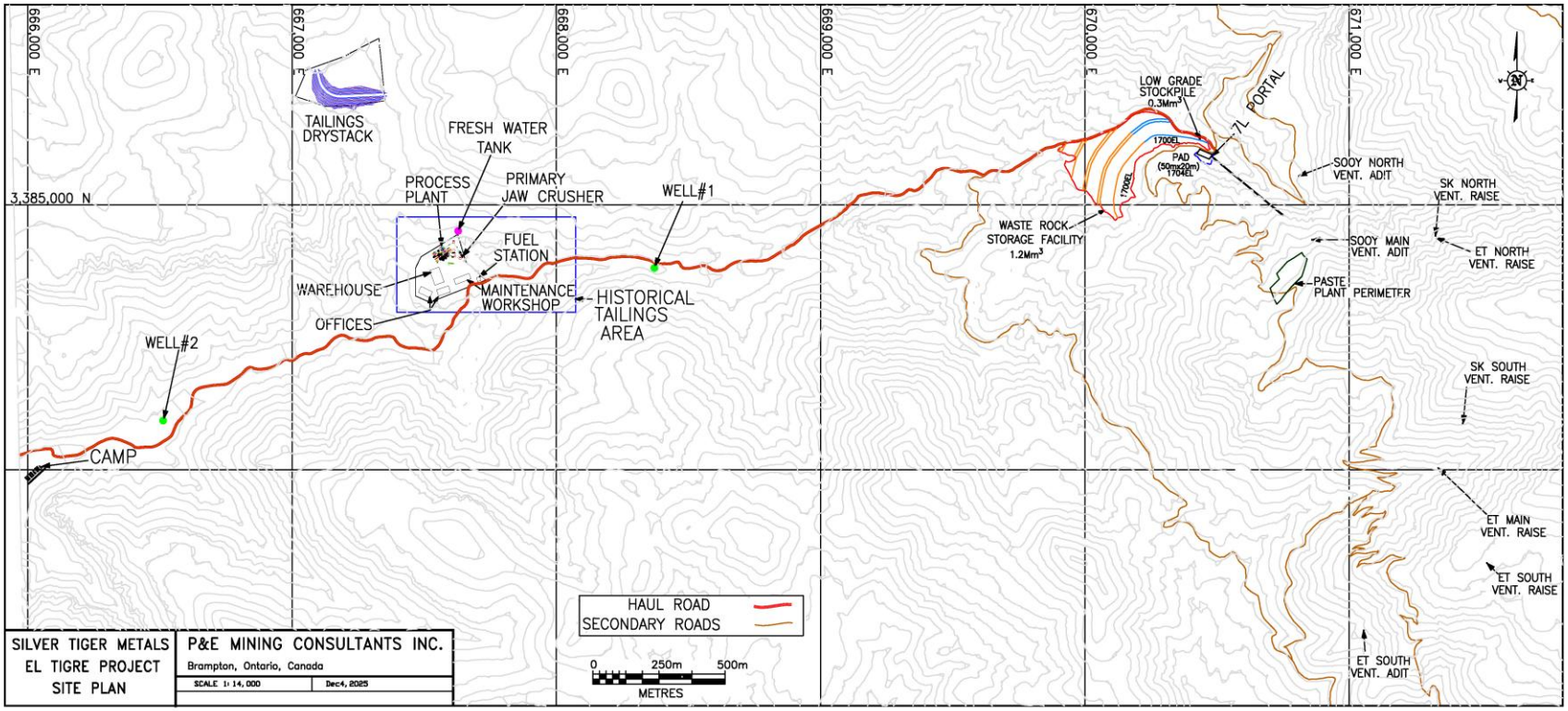
- Portal for Level 7 access to the underground mine workings;
- Low-grade mineralization stockpile and waste rock storage facilities near the portal;
- Site access roads;
- Locations of ventilation raises and paste backfill plant;
- Process plant and primary jaw crusher, with water supplied by two wells;
- Dry stack tailings storage facility; and
- Historical tailings area.

Additional infrastructure to be installed includes:

- Gatehouse and security on the main access road;
- Main office for administration, purchasing, safety and technical personnel;
- Warehouse for all mechanical and process plant parts;
- Fuel storage and distribution facility to be installed by fuel supplier;
- Maintenance workshop to be installed by mining contractor;
- Communications - telephone, cellular, and internet;
- Water wells, storage, potable and sewage treatment plants;
- Secure explosives storage facility; and
- 300-person camp to house all site personnel.

A plan view of the surface infrastructure is presented in Figure 1.5.

FIGURE 1.5 PROPOSED SURFACE INFRASTRUCTURE AT THE EL TIGRE PROJECT



Source: *P&E (2025)*

1.19.5 Market Studies and Contracts

A gold price of $3,200/oz and silver price of $38/oz are based on an approximate average of December 31, 2025, three-year monthly trailing averages, and Consensus Economics Inc. long term price forecasts. The Mexican Peso:US Dollar exchange rate of 20:1 is based on the approximate past three-year average. The metal prices and exchange rate are subject to spot market conditions. There are no metals streaming or hedging agreements in place. Currently there are no contracts in place that are material to the El Tigre Project.

1.19.6 Capital and Operating Costs

All costs are presented in Q4 2025 US dollars. No provision has been included in the cost estimates to offset future escalation. A contingency of 15% is added to all capital costs ("CAPEX"). No contingency is added to operating costs ("OPEX").

The total initial CAPEX of the El Tigre 2025 PEA is estimated at $83.5M including $10.9M in contingency costs. Pre-production revenue of $29.2M is included as a credit against initial capital costs. Sustaining capital costs incurred during the ~17 production years are estimated at $213.6M. Total capital costs over the LOM are estimated at $297.1M and are presented in Table 1.11.

TABLE 1.11
2025 PEA CAPITAL COST SUMMARY

Item	Initial ($k)	Sustaining ($k)	Total ($k)
Process Plant Directs	22,298	-	22,298
Process Plant Indirects	8,811	-	8,811
Underground Mining	49,518	173,508	223,026
Infrastructure	6,449	-	6,449
Dry Stack Tailings	3,978	12,230	16,208
Owner's Costs	10,743	-	10,743
Pre-production revenues[2]	(29,175)	-	(29,175)
Subtotal	**72,622**	**185,738**	**258,360**
Contingencies @ 15%	10,893	27,861	38,754
Total[1]	**83,515**	**213,599**	**297,114**

Notes: *[1] Totals may not sum due to rounding.*
[2] Revenue is generated during the pre-production period from processing historical stockpiled mineralized material.

The initial capital cost estimate addresses the engineering, procurement, construction and start-up of the El Tigre Project, with a two-year construction period to develop an underground mine, build a process plant capable of treating 800 tpd, construct a paste backfill plant, and install associated ancillary surface facilities including preparation of a dry stack tailings facility.

Sustaining capital represents capital expenses for additional costs that are not included in the normal operating costs, equipment purchases that will be necessary during the operating life of the

Project, dry stack tailings capacity increases, and all capital costs associated with underground mining.

Total OPEX over the life-of-mine ("LOM") is estimated at $595.6M, averaging $118.57/t processed as presented in Table 1.12. The operating costs estimate includes the cost of mining, processing, and General and Administration ("G&A") services.

TABLE 1.12
2025 PEA OPERATING COST SUMMARY

Item	Unit Cost ($/t processed)	LOM Cost ($k)
Processing UG Feed	41.55	166,193
Processing Tailings/Stockpile Feed	24.25	21,489
Subtotal Processing[1]	**37.37**	**187,681**
Underground Mining	76.80	307,200
Haulage UG Portal to Process Plant, Return with Tailings	1.33	6,672
Subtotal Mining and Hauling[2]	**62.49**	**313,872**
General and Administration[3]	18.71	94,000
Total	**118.57**	**595,553**

Note: *1+2+3=total.*

An electrical power rate supplied by Commission Federal de Electricity ("CFE") has been quoted at $0.113/kWh. The delivered diesel price has been quoted by a reputable fuel distribution firm at $1.11/litre.

The Project is subject to a 1.0% NSR Mexican government mining duty. Total costs associated with this NSR duty over the LOM are estimated at $16.7M.

Costs to be incurred after the LOM plan to close and rehabilitate the Project site were estimated at $4.0M.

Cash costs over the LOM, including Mexican mining duties, are estimated to average $16.05/oz AgEq or $1,351/oz AuEq. All-In Sustaining Costs ("AISC") over the LOM are estimated to average $23.98/oz AgEq or $2,019/oz AuEq and include closure costs.

1.19.7 Economic Analysis

Cautionary Statement - The reader is advised that the 2025 PEA summarized in this Technical Report is intended to provide only an initial, high-level review of the Project potential and design options. The 2025 PEA mine plan and economic model include numerous assumptions and the use of Inferred Mineral Resources. Inferred Mineral Resources are considered to be too speculative to be used in an economic analysis except as allowed by NI 43-101 in PEA studies. There is no guarantee the Project economics described herein will be achieved.

The El Tigre Project economic evaluation conclusions are summarized in Table 1.13. At base case metal prices of US$3,200/oz Au and US$38/oz Ag the Project has an estimated US$304.0M after-tax net present value ("NPV") at a 5% discount rate ("NPV5%"), and an after-tax internal rate of return ("IRR") of 43%. The payback period is estimated to be 2.6 years from the start of production.

TABLE 1.13
2025 PEA ECONOMIC EVALUATION SUMMARY

Item	Pre-Tax	After-Tax
NPV0% ($M)	753.5	495.6
NPV5% ($M)	487.3	**304.0**
NPV7% ($M)	414.5	252.6
IRR (%)	65.2	**42.8**
Payback period (years)	1.8	**2.6**

The estimated annual production and LOM cash flows for the El Tigre 2025 PEA are summarized in Table 1.14.

TABLE 1.14
2025 PEA PROJECT CASH FLOW SUMMARY

Assumption / Result	Unit	Value	Assumption / Result	Unit	Value
Plant Feed Processed	kt	5,023	Net Revenue	US$M	1,665.4
Waste Rock Mined	kt	2,439	Initial Capital	US$M	83.5
Silver Grade	g/t	251.3	Sustaining Capital	US$M	213.6
Gold Grade	g/t	0.91	Mining and Haulage Costs	$/t processed	62.49
Silver Recovery	%	84	Processing Costs (UG and Stockpile)	$/t processed	41.55 / 24.25
Gold Recovery	%	90	G&A Costs	$/t processed	18.71
Silver Price	US$/oz	38.00	Operating Cash Cost	US$/oz AgEq	16.0
Gold Price	US$/oz	3,200	All-in Sustaining Cost	US$/oz AgEq	24.0
Payable Silver Metal	Moz	33.6	After-Tax NPV (5% discount)	US$M	304.0
Payable Gold Metal	koz	130	Pre-Tax NPV (5% discount)	US$M	487.3
Payable AgEq	Moz	38.5	After-Tax IRR	%	42.8
Production and Reclamation	years	18	Pre-Tax IRR	%	65.2

TABLE 1.14 2025 PEA PROJECT CASH FLOW SUMMARY					
Assumption / Result	**Unit**	**Value**	**Assumption / Result**	**Unit**	**Value**
Average Feed Processed	tpd	800	After-Tax Payback Period	years	2.6

The results of the sensitivity analyses on the after-tax NPV with a 5% discount rate are shown in Table 1.15 and Table 1.16.

TABLE 1.15 2025 PEA METAL PRICE SENSITIVITY					
Sensitivity	**-20%**	**-10%**	**Base Case**	**+10%**	**+20%**
Silver Price (US$/oz)	30	34	**38**	42	46
Gold Price (US$/oz)	2,560	2,880	**3,200**	3,520	3,840
After-Tax NPV (5%) (US$M)	153.9	229.1	**304.0**	378.9	453.9
After-Tax IRR (%)	24.8	33.8	**42.8**	51.8	61.0
After-Tax Payback (years)	3.5	3.0	**2.6**	2.2	1.9

TABLE 1.16 2025 PEA CAPITAL AND OPERATING COST SENSITIVITY					
Sensitivity	**-20%**	**-10%**	**Base Case**	**+10%**	**+20%**
Operating Costs – NPV_5 (US$M)	354.9	329.5	**304.0**	278.6	253.2
Operating Costs – IRR (%)	48.2	45.5	**42.8**	40.1	37.4
Capital Costs – NPV_5 (US$M)	337.1	320.6	**304.0**	287.5	271.0
Capital Costs – IRR (%)	55.1	48.4	**42.8**	38.1	34.0

The after-tax base case NPV's and IRR's are most sensitive to metal prices followed by operating costs and capital costs.

If the 2025 PEA underground mine plan was to be implemented after surface mining commenced as detailed in the PFS sections of this Technical Report there would be a savings in initial capital costs of the 2025 PEA estimate by approximately $17.0M. The savings would consist of infrastructure such as the electrical powerline for grid power, and camp and office facilities, plus much of the owner's costs.

1.19.8 El Tigre North Analysis

Included in the MRE stated in Section 14 of this Technical Report are additional Mineral Resources located approximately 700 m north of the planned mining areas discussed in the 2025 PEA. An arbitrary division of the Property into "El Tigre South" ("ETS") and "El Tigre North" ("ETN") utilized the Combination Canyon Fault as the cut-off boundary. Significant Mineral Resources exist on the north side of the fault at ETN that are potentially amenable to underground mining. A mine design was completed for ETN, however, detailed costing/scheduling activities have not been performed. The mine plan indicates that a total of 2.95 Mt of mineralized material at an average grade of 343 g/t AgEq could potentially be recovered from ETN utilizing longitudinal longhole retreat mining with PF, showing the potential for a mine life of slightly in excess of 10 years at 290 ktpa (nominally 800 tpd). Table 1.17 presents the mine plan portion of the Mineral Resource in the ETN mining area.

TABLE 1.17
ETN MINE PLAN PORTION OF THE MINERAL RESOURCE

Item	Tonnes (k)	AgEq (g/t)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
Measured	118.7	333.9	0.79	264.0	0.053	0.263	0.363
Indicated	188.0	336.2	1.06	242.3	0.050	0.076	0.088
Total M&I[1]	**306.8**	**335.3**	**0.96**	**250.7**	**0.051**	**0.148**	**0.194**
Inferred	2,641.3	343.9	1.59	203.1	0.332	0.735	0.822

Note: *[1.] Totals may not sum due to rounding*
M&I = Measured and Indicated Mineral Resources.

1.19.9 Risks and Opportunities

A preliminary risk analysis was completed on the 2025 PEA by the Authors using a low-medium-high ranking system. The highest risk items were identified to be lack of comprehensive geotechnical analysis, mining contractor cost assumptions, and that further testwork is required on metallurgical recoveries. The main opportunities are that the Deposit remains open along strike and down dip, which has the potential to increase the current Mineral Resource Estimate, El Tigre North could extend the mine life, and that the Project economics improve substantially if current spot metal prices are input to the cash flow model.

1.19.10 Conclusions and Recommendations

The 2025 PEA underground mine and mineral processing plan is scheduled over a production period of ~17 years. The mine plan portion of the Mineral Resource Estimate for the El Tigre 2025 PEA totals 5.0 Mt at average grades of 331.7 g/t AgEq, 251.3 g/t Ag, 0.91 g/t Au, 0.033% Cu, 0.099% Pb and 0.166% Zn (base metals provide no economic benefit under the proposed cyanidation process). Approximately 85% of the mine plan portion of the Mineral Resource, including historical tailings and low-grade stockpile mineralization, are classified as Measured and Indicated.

Based on the work undertaken to date, as summarized in this Technical Report, and the individual Qualified Persons conclusions listed in Section 25, the base case operating plan has identified a viable and attractive development opportunity. Overall Project risks are perceived as low.

Current base case economic analysis indicates that the El Tigre 2025 PEA Project is forecast to be profitable. It is recommended that Silver Tiger continue the 2025 PEA Project development plans, and implement the recommendations set out in Section 26 of this Technical Report.

The El Tigre Property contains a significant precious metal Mineral Resource, and the Authors recommend that Silver Tiger proceed to develop the 2025 PEA Project and proceed with a PFS on underground mining. It is recommended that the Company complete further metallurgical testwork and undertake geotechnical and hydrogeological studies. An infill drill program designed to potentially upgrade Inferred Mineral Resources contained within the underground mine design is also recommended. The estimated cost of the recommended work program is $7.8M.

2.0 INTRODUCTION AND TERMS OF REFERENCE

2.1 TERMS OF REFERENCE

Silver Tiger Metals Inc. ("Silver Tiger") retained P&E Mining Consultants Inc. ("P&E") to complete an independent NI 43-101 Technical Report consisting of an updated Mineral Resource Estimate, an updated Stockwork Zone Pre-Feasibility Study ("PFS") and an underground Preliminary Economic Assessment ("PEA") on the El Tigre Property, located in the State of Sonora, Mexico. Input to this Technical Report was also provided by D.E.N.M. Engineering Ltd. ("D.E.N.M."), Consultores Interdisciplinario en Medio Ambiente S.C. ("CIMA") and WSP Canada Inc. ("WSP"). This Technical Report considers gold and silver mineralization at the El Tigre Property (the "Property" or the "Project") that is amenable to both surface and underground mining.

The Updated Mineral Resource Estimate presented in this Technical Report has been prepared in full conformance and compliance with the "CIM Standards on Mineral Resources and Reserves – Definitions (2014) and Best Practices Guidelines (2019)" as referred to in NI 43-101 and Form 43-101F, Standards of Disclosure for Mineral Projects and in force as of the effective date of this Technical Report.

This Technical Report was prepared by the Authors, at the request of Mr. Glenn Jessome, President and CEO of Silver Tiger, a company incorporated pursuant to the Canada Business Corporations Act, trading under the symbol of "SLVR" on the TSX-V Exchange, and with its corporate office at:

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1969 Upper Water St. Halifax, Nova Scotia
Canada
B3J 3N2

The authors ("Authors") of this Technical Report understand that the Report will support the public disclosure requirements of Silver Tiger and will be filed on SEDAR+ as required under NI 43-101 disclosure regulations.

This Technical Report is considered effective as of June 20, 2025.

2.2 INDEPENDENT SITE VISITS

Mr. David Burga, P.Geo., of P&E, an independent Qualified Person under the terms of NI 43-101, completed a site visit to the El Tigre Property from January 20 to 21, 2016. Mr. Fred Brown, P.Geo., of P&E, an independent Qualified Person under the terms of NI 43-101, completed a site visit to the Property from June 19 to 20, 2016 and on May 22, 2017. Mr. Yungang Wu, P.Geo., of P&E, an independent Qualified Person under the terms of NI 43-101, completed a site visit to the Property from July 13 to 14, 2017. Data verification sampling programs were completed as part of the on-site review.

More recently, Mr. David Burga completed site visits to the Property on August 5 and 6, 2023, and from May 20 to 21, 2024. Data verification sampling programs were completed as part of the on-site review and the results are described in Section 12 of this Technical Report.

Mr. Greg Robinson, P.Eng., of P&E, an independent Qualified Person under the terms of NI 43-101, completed a site visit to the Property from May 5 to 9, 2025. The purpose of the visit was to assess engineering aspects of the Project, particularly for infrastructure and mining studies, and undertake due diligence sampling. Drill core verification samples were collected from nine diamond drill holes that were completed on the Property in 2024.

Mr. David Salari, P.Eng., of D.E.N.M., an independent Qualified Person under the terms of NI 43-101, visited the Property on February 1 to 2, 2024 for the purpose to inspect the proposed location of the surface mine, heap leach pad(s), ponds, and process recovery facility. Inspection and location of the mineral testing holes was also reviewed with associated drill core. Site access, water source and other associated infrastructure was also addressed and inspected.

Mr. César Fernadez Villalobos, Eng., of CIMA, completed a site visit to the El Tigre Project on May 26 to 27, 2022. A baseline environmental data program was reviewed when on site.

A Qualified Person site visit for surface mine geotechnical and hydrogeological services was not completed for the PFS. WSP geotechnical staff (other than the Qualified Person) have visited the site on multiple occasions prior to and during the geotechnical drilling campaign, for the PFS, to review data collection procedures, training, site layout, and other relevant aspects. WSP's Qualified Person provided oversight for these visits.

2.3 SOURCES OF INFORMATION

This Technical Report is based, in part, on internal Company technical reports, and maps, company letters, memoranda, public disclosure and public information as listed in the References under Section 27 of this Report. Sections from reports authored by other consultants have been directly quoted or summarized in this Technical Report and are indicated where appropriate.

P&E prepared a Technical Report and Stockwork Zone Pre-Feasibility Study of the Project with an effective date of October 22, 2024. Micon International Limited prepared a previous Technical Report and Preliminary Economic Assessment on the El Tigre Project with an effective date of October 27, 2023. The PEA was based on the Updated Mineral Resource Estimate on the Project prepared by P&E with an effective date of September 12, 2023. P&E also prepared a previous Updated Mineral Resource Technical Report of the Project with an effective date of September 7, 2017. The reports are referenced in the Reference section (Section 27) of this Technical Report.

The Authors and co-Authors of each section of this Technical Report are presented in Table 2.1. In acting as independent Qualified Persons as defined by NI 43-101, they take responsibility for those sections of this Report as outlined in their "Certificate of Author" included in Section 28 of this Report.

TABLE 2.1 QUALIFIED PERSONS RESPONSIBLE FOR THIS TECHNICAL REPORT		
Qualified Person	**Contracted By**	**Sections of Technical Report**
Andrew Bradfield, P.Eng.	P&E Mining Consultants Inc.	2, 3, 16, 19, 22, 24.1, 24.2, 24.3.1, 24.3.6, 24.3.7, 24.3.9, 24.3.12 and Co-author 1, 15, 18, 21, 24.3.8, 25, 26, 27
Jarita Barry, P.Geo.	P&E Mining Consultants Inc.	11 and Co-author 1, 12, 25, 26, 27
Fred H. Brown, P.Geo.	P&E Mining Consultants Inc.	Co-author 1, 14, 25, 26, 27
David Burga, P.Geo.	P&E Mining Consultants Inc.	Co-author 1, 10, 12, 25, 26, 27
D. Grant Feasby, P.Eng.	P&E Mining Consultants Inc.	20 and Co-author 1, 25, 26, 27
Eugene Puritch, P.Eng., FEC, CET	P&E Mining Consultants Inc.	Co-author 1, 14, 25, 26, 27
Greg Robinson, P.Eng.	P&E Mining Consultants Inc.	24.3.3, 24.3.4, 24.3.10, 24.3.11 and Co-author 1, 12, 24.3.8, 25, 26, 27
William Stone, Ph.D., P.Geo.	P&E Mining Consultants Inc.	4, 5, 6, 7, 8, 9, 23 and Co-author 1, 10, 25, 26, 27
David Salari, P.Eng.	D.E.N.M. Engineering Ltd.	13, 17, 24.3.2, 24.3.5 and Co-author 1, 18, 21, 24.3.8, 25, 26, 27
James Smith, P.Eng.	WSP Canada Inc.	Co-author 1, 2, 15, 25, 26, 27

2.4 UNITS AND CURRENCY

Unless otherwise stated all units used in this Technical Report are metric. Gold ("Au") and silver ("Ag") assay values are reported in grams of metal per tonne ("g/t"). The US$ is used throughout this Report unless otherwise specified.

The coordinate system used by Silver Tiger for locating and reporting drill hole information is the UTM system. The Property is in UTM Zone 12 and the WGS84 datum is used. Maps in this Technical Report use the UTM coordinate system, unless otherwise indicated.

Table 2.2 shows the meaning of the terminology and abbreviations for technical terms used throughout the text of this Technical Report. Table 2.3 lists unit abbreviations.

TABLE 2.2 TERMINOLOGY AND ABBREVIATIONS	
Abbreviation	**Meaning**
°	degree(s)
°C	degrees Celsius
Ø	diameter
>	greater than
<	less than

TABLE 2.2 TERMINOLOGY AND ABBREVIATIONS	
Abbreviation	**Meaning**
%	percent
µm	micrometre, micron
σ_{ci}	unconfined compressive strength
3-D	three-dimensional
AA	atomic absorption
AAS	atomic absorption spectrometry
ABA	acid-base accounting
Actlabs	Activation Laboratories Ltd.
Ag	silver
AgEq	silver equivalency
AGP	acid generating potential
Ai	abrasion index
AISC	all-in sustaining costs
ALS	ALS Minerals, ALS Global, ALS Worldwide Labs, ALS Geochemistry
AN	ammonium nitrate
Anaconda	Anaconda Minerals Company
ANFO	ammonium nitrate and fuel oil
ANP	acid neutralization potential
AP	acid potential (potential acid generation)
AP/NP	acid potential/neutralization potential
As	arsenic
ATV	Acoustic Televiewer (survey)
Au	gold
AuEq	gold equivalency
Authors	the authors, Qualified Persons, of this technical report
BFA	bench face angle
BLS	barren leach solution
BS	Black Shale
Bureau Veritas	Bureau Veritas S.A.
BWI	Bond Work Index
C&F	cut-and-fill
$CaCO_3$	calcium carbonate
CaO	calcium oxide, lime
CAPEX	capital expenditure
CCD	counter current decantation (circuit)
Cd	cadmium
CDN	CDN Resource Laboratories Ltd.

TABLE 2.2 TERMINOLOGY AND ABBREVIATIONS	
Abbreviation	**Meaning**
CFE	Comision Federal de Electricidad (Mexican Federal Electrical Commission)
CFP	chamber filter
CIM	Canadian Institute of Mining, Metallurgy, and Petroleum
CIMA	Consultores Interdisciplinario en Medio Ambiente S.C.
cm	centimetre(s)
CN	cyanide
CN/FA	cyanide soluble/fire assay
CNA	Consejo Nacional Agropecuario
COBR	coarse bottle roll
Cobre	Cobre de Hercules
COG(s)	cut-off grade(s)
Company, the	Silver Tiger Metals Inc.
CONAGUA	Comision Nacional del Agua
CONANP	National Commission of Natural Protected Areas
Copper Queen	Copper Queen Mining Co.
CoV	coefficient of variation
CRM(s)	certified reference material(s)
C_T	total carbon
C_{TOC}	total organic carbon
Cu	copper
CuO	copper oxide
$CuSO_4$	copper sulphate
CWi	crusher work index
$M	dollars, millions
$MXN	Mexican peso
D&F	drift-and-fill
DCOG	development cut-off grade(s)
D.E.N.M.	D.E.N.M. Engineering Ltd.
Deposit, the	El Tigre silver and gold deposit
District, the	El Tigre District
DSO	Deswik Stope Optimizer
DTM	digital terrain model
DTU	Unified Technical Document of the Procedure for Change of Use of Forest Land
E	east
ECOG	economic cut-off grade
EDS	energy-dispersive x-ray spectroscopy

TABLE 2.2 TERMINOLOGY AND ABBREVIATIONS	
Abbreviation	**Meaning**
EGM	Engineering Geology Model
EL or El	elevation
El Tigre Silver	El Tigre Silver Corporation
ES	environmental system
EPCM	engineering, procurement, construction management
Eq. m	equivalent metres
ETN	El Tigre North
ETS	El Tigre South
ETS	El Tigre Silver Corporation
F	fluorine
FA	fire assay
FA-AAS or FA/ASS	fire assay- atomic absorption spectrometry
FA-GRAV or FA/GRAV	fire assay-gravimetric
FAR(s)	fresh air raise(s)
Fe	iron
FEED	Front End Engineering Design
FEL	front-end loader
FoS	Factor of Safety
ft	foot, feet
FW	footwall
FWD(s)	footwall drift(s)
FX	foreign exchange
g	gram
g/L or g/l	grams per litre
g/t	grams per tonne
G&A	general and administration
gal/min	gallons per minute
GPS	Global Positioning System
GSI	geological strength index
H	height
H_2SO_4	sulphuric acid
H: V	horizontal versus vertical (ratio)
ha	hectare(s)
HCl	hydrochloric acid
HDPE	high-density polyethylene
Herdron	Herdron Capital Corp.
Hg	mercury
HG	high grade

TABLE 2.2 TERMINOLOGY AND ABBREVIATIONS	
Abbreviation	**Meaning**
HMW	Historical Mine Workings
HP	horsepower
HRC	Hard Rock Consulting
HW	hanging wall
H:V	horizontal to vertical ratio
ICP	inductively coupled plasma
ICP-AES	inductively coupled plasma-atomic emission spectroscopy
ICP-ES/MS	inductively coupled plasma-emission spectroscopy/mass spectrometry
ICP-MS or ICP/MS	inductively coupled plasma-mass spectrometry
ICP-OES	inductively coupled plasma - optical emission spectroscopy
ID	identification
ID^2	inverse distance squared
ID^3	inverse distance cubed
IMSS	Mexican Institute of Social Security
in or in.	inch(es)
INAH	Instituto Nacional de Antropología e Historia
IP	induced polarization
IRA	inter-ramp angle
IRR	internal rate of return
ISO	International Organization for Standardization
ISO/IEC	International Organization for Standardization/International Electrotechnical Commission
ISO-ILAC-IAF	International Organization for Standardization-International Laboratory Accreditation Cooperation-International Accreditation Forum
ITH	in-the-hole
ITS	indirect tensile strength
Jaleros del Tigre	Cia. Jaleros del Tigre, S.A. de C.V.
k	thousand(s)
$k\text{-}m^3$	thousand(s) of cubic metres
kg	kilograms(s)
kg/t	kilograms(s) per tonne
km	kilometre(s)
koz	thousands of Troy ounces
kt	kilotonnes or thousands of tonnes
ktpa	thousands of tonnes per annum or year
kV	kilovolt
kVA	kilovolt-ampere

TABLE 2.2 TERMINOLOGY AND ABBREVIATIONS	
Abbreviation	**Meaning**
L	length
L or l	litre(s)
L/s	litres per second
lb	pound (weight)
LCT	locked cycle test
LDFS	General Law of Sustainable Forest Development
LE	limit equilibrium
level	mine working level referring to the nominal elevation (m RL), e.g., 4285 level (mine workings at 4285 m RL)
LG	low grade
LGEEPA	General Law of Ecological Balance and Environmental Protection
LH	longhole
LHD	load-haul-dump
LiDAR	Light Detection and Ranging
Lin. m	linear metre
LLDPE	linear low-density polyethylene
LOM	life of mine
LTM	Laboratorio Tecnológico de Metalurgia LTM S.A. de C.V.
LVS	Lower Volcanic Series
M	million(s)
m	metre(s)
m^2	square metre(s)
m^3	cubic metre(s)
Ma	millions of years
masl	metres above sea level
MC	Merrill Crowe
MCOG	marginal cut-off grade
Metso	Metso (Canada) Incorporated
mi	Hoek-Brown constant
MIBC	methyl isobutyl carbinol
Minera de Cordilleras	Minera de Cordilleras S.A. de R.L. de C.V.
MLC(s)	mine load centres
MLI	McClelland Laboratories, Inc.
mm	millimetre
Mm^3	millions of metres cubed
Mn	manganese
Mo	molybdenum
Moz	million ounces

TABLE 2.2 TERMINOLOGY AND ABBREVIATIONS	
Abbreviation	**Meaning**
MPa	megapascal
MPs	mineralized passes
MRE	Mineral Resource Estimate
MS	mass spectrometer
Mt	mega tonne or million tonnes
Mtpa	millions of tonnes per annum or year
MW	megawatts
N	north
N	total population size/number
NaCN	sodium cyanide
NAG	net acid generating / net acid production potential
NaOH	sodium hydroxide
Net NP or NNP	net neutralization potential
NI	National Instrument
NN	Nearest Neighbour
NNP or Net NP	net neutralization potential
No.	number
NP	neutralization potential
NPV	net present value
NSR	net smelter return
NV-ABA	Nevada modified acid base accounting
NV-NAG	Nevada modified net acid generating
Oceanus	Oceanus Resources Corporation
OP	open pit
OPEX	operating expenditure
OX	oxide ore
oz	Troy ounce
oz/ton	Troy ounce(s) per ton
P_{80}	80% percent passing
P&E	P&E Mining Consultants Inc.
Pacemaker	Pacemaker Silver Mining S.A. de C.V.
PAX	potassium amyl xanthate
Pb	lead
PbO	lead oxide
PEA	Preliminary Economic Assessment
P.Eng.	Professional Engineer
PEX	potassium ethyl xanthate
PF	paste backfill

TABLE 2.2 TERMINOLOGY AND ABBREVIATIONS	
Abbreviation	**Meaning**
PF	powder factor
PFD	process flowsheet
PFS	Pre-Feasibility Study
P.Geo.	Professional Geoscientist
Photosat	Photosat Information, Ltd
PLT(s)	point load test(s)
PM	planned maintenance
ppm	parts per million
PROFEPA	Procudraduria Federal de Protección al Medio Ambiente
Project, the	the El Tigre Project that is the subject of this Technical Report
Property, the	the El Tigre Property that is the subject of this Technical Report
psi	(pound-force) per square inch
QA	quality assurance
QA/QC	quality assurance/quality control
QC	quality control
QEMSCAN	quantitative evaluation of minerals by scanning electron microscopy
R^2	coefficient of determination
RAR(s)	return air raise(s)
RC	reverse circulation
Report, the	this Technical Report
RF	revenue factor
RMR	rock mass rating
ROM	run of mine
RQD	rock quality designation
S	south
S	sulphur
S= or $S^{=}$ or S^{2-}	sulphide sulphur
Sb	antimony
scfm	standard cubic feet per minute
SCT	Secretary of Communications and Transport
SD	standard deviation
SEDENA	Secretariat of National Defense
SEM	scanning electron microscope
SEMARNAT	Secretariat of Environment and Natural Resources
SGM	Servicio Geologica Mexicano
SGS	SGS Lakefield that is part of SGS S.A.
Silver Tiger	Silver Tiger Metals Inc.
Skyline	Skyline Assayers and Laboratories Inc.

TABLE 2.2 TERMINOLOGY AND ABBREVIATIONS	
Abbreviation	**Meaning**
SMOB	Sierra Madre Occidental Belt
SMU	selective mining unit
Sonrisa	Cia. Minera Sonrisa, S.A. de C.V.
SPs	stockpiles
SPM	SPM Mineria S.A. de C.V.
ST or S_T	total sulphur
Std Dev	standard deviation
SU	sulphide ore
t	metric tonne(s)
t/h	tonnes per hour
t/m^3	tonnes per cubic metre
Talaman	Compania Minera Talaman S.A. de C.V.
TCS	triaxial compressive strength
TD-ICP	total digestion-inductively coupled plasma
Technical Report	this NI 43-101 Technical Report
TM Landsat	Thematic Mapper (sensor) Landsat
ton	short ton(s)
tpd	tonnes per day
UCS	uniaxial compressive strength
UG	underground
US$	United States dollar(s)
UTM	Universal Transverse Mercator grid system
UVS	Upper Volcanic Series
VW	vibrating wire
VWP	vibrating wire piezometers
W	west
W	width
w/w	weight by weight
WGS84 or WGS 84	World Geodetic System 84, used by the GPS
WPs	waste passes
WSP	WSP Canada Inc. or WSP Grupo Mexico S. A. de C. V.
wt%	weight percent
WWTP	Wastewater Treatment Plant
XRD	X-ray diffraction
YR	year
Zn	zinc
$ZnSO_4$	zinc sulphate

TABLE 2.3 UNIT MEASUREMENT ABBREVIATIONS			
Abbreviation	**Meaning**	**Abbreviation**	**Meaning**
µm	microns, micrometre	m^3/d	cubic metre per day
$	dollar	m^3/h	cubic metre per hour
$/t	dollar per metric tonne	m^3/s	cubic metre per second
%	percent sign	m/h	metre per hour
% w/w	percent solid by weight	m/s	metre per second
¢/kWh	cent per kilowatt hour	mg	milligram
°	degree	mg/L	milligram per litre
°C	degree Celsius	MHz	megahertz
cm	centimetre	Mt	million tonnes
d	day	Mtpa or Mtpy	million tonnes per annum or year
ft	feet	min	minute
GWh	gigawatt hours	min/h	minute per hour
g/mL, g/ml, g.ml	grams per millilitre	mL	millilitre
g/t	grams per tonne	mm	millimetre
h	hour	Mm^3	millions of metres cubed
ha	hectare	MPa	megapascal
hp	horsepower	Mt	million tonnes or megatonnes
Hz	hertz	MV	medium voltage
k	kilo, thousands	MVA	mega volt-ampere
kg	kilogram	MW	megawatts
kg/t	kilogram per metric tonne	oz	ounce (troy)
kHz	kilohertz	Pa	pascal
km	kilometre	pH	measure of acidity
kPa	kilopascal	ppb	part per billion
kt	thousands of tonnes or kilotonnes	ppm	part per million
kV	kilovolt	psi	(pound-force) per square inch
kVA	kilovolt-ampere	s	second
kW	kilowatt	t or tonne	metric tonne
kWh	kilowatt-hour	tpd	metric tonne per day
kWh/t	kilowatt-hour per metric tonne	t/h	metric tonne per hour
L	litre	t/h/m	metric tonne per hour per metre
L/s	litres per second	$t/h/m^2$	metric tonne per hour per square metre
L/min, l/min	liters per minute	t/m	metric tonne per month

TABLE 2.3 UNIT MEASUREMENT ABBREVIATIONS			
Abbreviation	**Meaning**	**Abbreviation**	**Meaning**
L/h/m^2, l/h/m^2, L/hr/m^2, l/hr/m^2	liters per hour per square metre	t/m^2	metric tonne per square metre
lb	pound(s)	t/m^3	metric tonne per cubic metre
M	million	T	short ton
m	metre	tpy or tpa	metric tonnes per year or per annum
m^2	square metre	V	volt
m^3	cubic metre	W	watt
m^3/y	cubic metre per year	wt% or wt %	weight percent
mØ	metre diameter	yr	year

3.0 RELIANCE ON OTHER EXPERTS

Although the Authors carefully reviewed all available information presented, they cannot guarantee its accuracy and completeness. The Authors reserve the right, and will not be obligated to revise the Technical Report and conclusions, if additional information becomes known subsequent to the effective date of this Technical Report.

Copies of the tenure documents, operating licenses, permits, and work contracts were not reviewed. Information on tenure was obtained from Silver Tiger and included a legal due diligence title opinion dated June 20, 2025 supplied by Silver Tiger's Mexican legal counsel, Mr. Pablo Méndez Alvidrez, titled "Opinion regarding Compañia Minera Talaman, S.A. de C.V. and Pacemaker Silver Mining, S.A. de C.V., and confirmation of title regarding the "El Tigre" mining concessions". Mr. Pablo Méndez Alvidrez is with the firm EC Rubio located at Punto Alto E4, Penthouse Centro Ejecutivo N0 5500, 31125 Chihuahua, Chihuahua. The Authors have relied on tenure information from Silver Tiger and have not undertaken an independent detailed legal verification of title and ownership of the El Tigre Project. The Authors have not verified the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s) between third parties, and have relied on and considers that they have a reasonable basis to rely on Silver Tiger to have completed the proper legal due diligence.

The Authors have relied on Mr. Keith Abriel, CPA, CA, CFA, Chief Financial Officer of Silver Tiger, for assistance with the taxation calculations in the financial model, as presented in section 22 of this Technical Report.

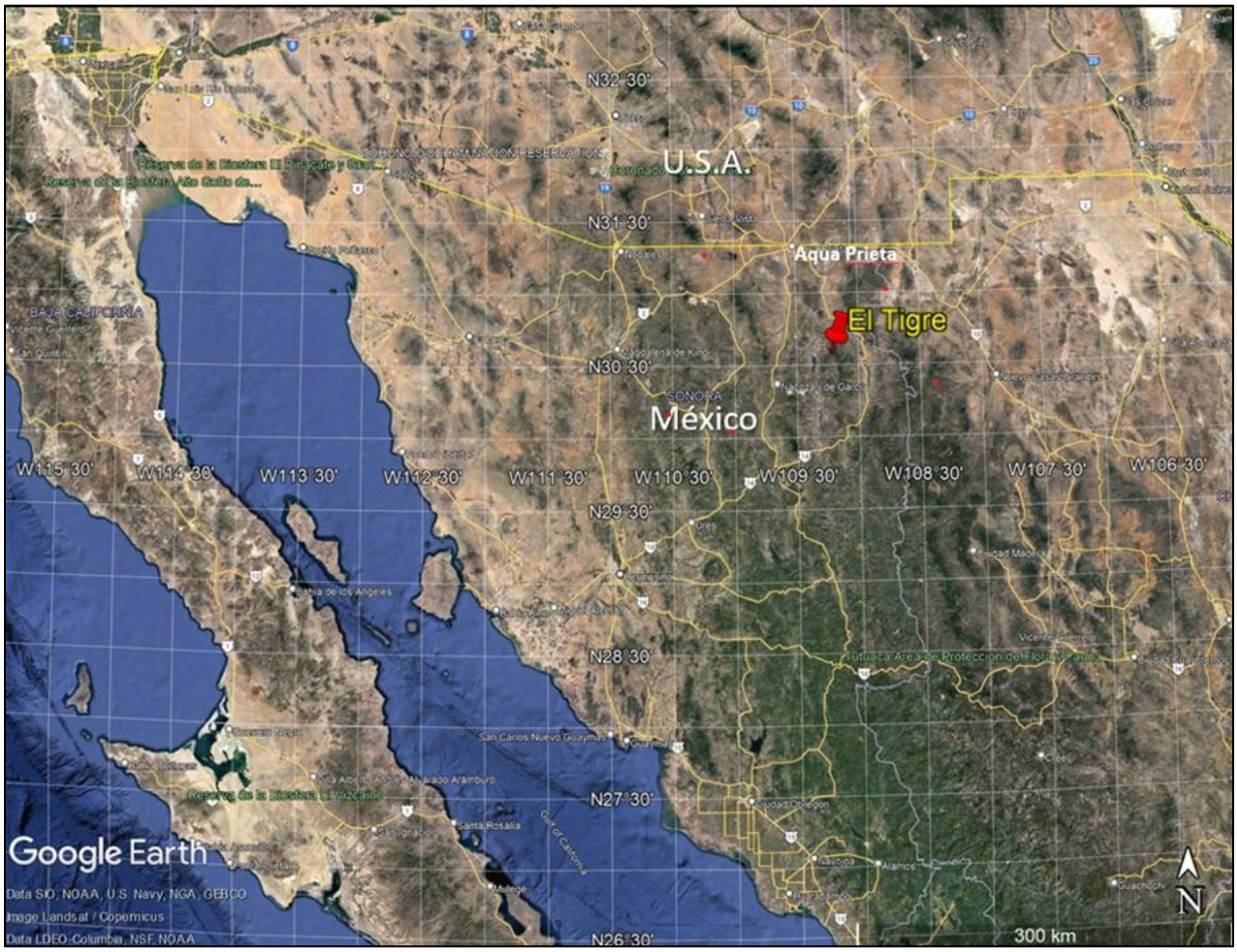
A draft copy of this Technical Report has been reviewed for factual errors by Silver Tiger and the Authors have relied on Silver Tiger's historical and current knowledge of the Property in this regard. Any statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the effective date of this Technical Report.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The El Tigre Property is located in the Sierra El Tigre area of northeastern Sonora State, Mexico. The Property is ~90 km south-southeast of the border Town of Agua Prieta (Figure 4.1). The Property is centred at approximately 30°35' north latitude and 109°13' west longitude (UTM WGS84 12R 670,380 m E and 3,385,230 m N) on the Colonia Oaxaca 1:50,000 topographic map sheet (H12B66) of the Servicio Geologica Mexicano (SGM).

FIGURE 4.1 EL TIGRE PROPERTY LOCATION MAP

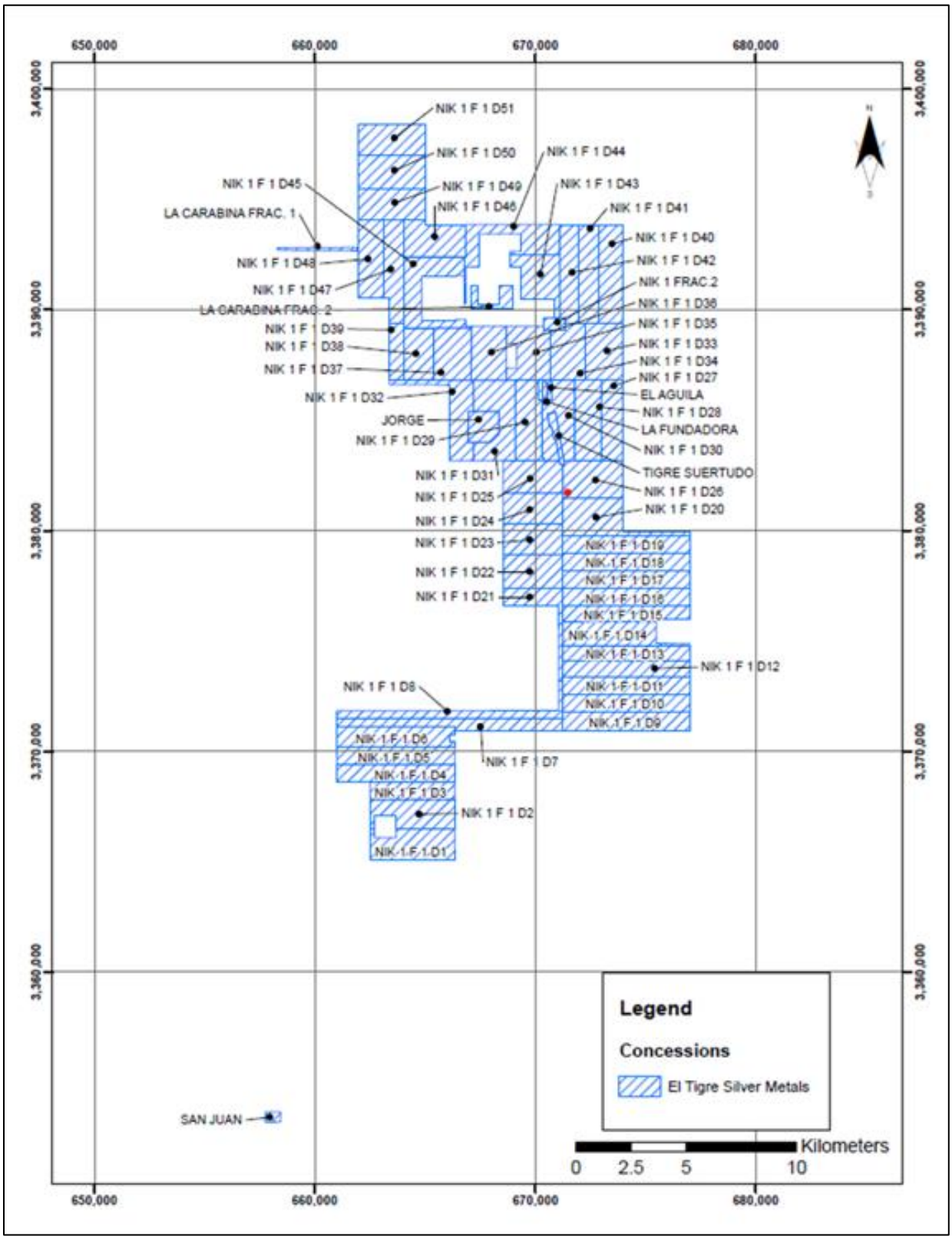


Source: *Micon (2023)*

4.2 PROPERTY DESCRIPTION AND MINERAL TENURE

The El Tigre Property consists of 59 Mexican Federal mining concessions totalling 21,775 ha (Figure 4.2; Table 4.1). Fifty-eight of the concessions are contiguous and one, San Juan, is not: the latter being located ~10 km southwest of the core El Tigre Property (Figure 4.2).

FIGURE 4.2 EL TIGRE MINERAL CONCESSION MAP



Source: *P&E (2024)*
Note: *Concessions information effective as of June 20, 2025.*

TABLE 4.1
EL TIGRE PROPERTY MINERAL TITLES

No.	Lot	Title	Area (ha)	Validity Start	Validity End	Current Validity (years)	Location	Title Holder / Owner	Liens or Legal Issues	Status
1	EL AGUILA	172113	38	26/09/83	25/09/33	39	Nacozari de Garcia, Sonora	Compañía Minera Talaman S.A. de C.V.	Free	Active
2	JORGE	194087	188.48	19/12/91	18/12/41	31	Nacozari de Garcia, Sonora	Compañía Minera Talaman S.A. de C.V.	Free	Active
3	LA FUNDADORA	172112	20	26/09/83	25/09/33	39	Nacozari de Garcia, Sonora	Compañía Minera Talaman S.A. de C.V.	Free	Active
4	TIGRE SUERTUDO	168634	66	26/06/81	25/06/31	41	Nacozari de Garcia, Sonora	Compañía Minera Talaman S.A. de C.V.	Free	Active
5	NIK 1 FRAC. 2	230001	50	05/07/07	28/03/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
6	SAN JUAN	228337	32.06	08/11/06	07/11/56	16	Valle Hidalgo, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
7	LA CARABINA FRAC. 1	229274	35.63	29/03/07	28/03/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
8	LA CARABINA FRAC. 2	229275	188.27	29/03/07	28/03/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
9	NIK 1 F1 D1	247064	453.39	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
10	NIK 1 F1 D2	247065	434.61	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
11	NIK 1 F1 D3	247066	304	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
12	NIK 1 F1 D4	247067	409	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
13	NIK 1 F1 D5	247068	424	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
14	NIK 1 F1 D6	247069	471	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
15	NIK 1 F1 D7	247070	457	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
16	NIK 1 F1 D8	247071	434	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
17	NIK 1 F1 D9	247072	456	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
18	NIK 1 F1 D10	247073	456	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
19	NIK 1 F1 D11	247074	456	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
20	NIK 1 F1 D12	247075	399	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
21	NIK 1 F1 D13	247076	477	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
22	NIK 1 F1 D14	247077	384	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
23	NIK 1 F1 D15	247078	456	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active

TABLE 4.1
EL TIGRE PROPERTY MINERAL TITLES

No.	Lot	Title	Area (ha)	Validity Start	Validity End	Current Validity (years)	Location	Title Holder / Owner	Liens or Legal Issues	Status
24	NIK 1 F1 D16	247079	456	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
25	NIK 1 F1 D17	247080	456	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
26	NIK 1 F1 D18	247081	456	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
27	NIK 1 F1 D19	247082	456	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
28	NIK 1 F1 D20	247083	489	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
29	NIK 1 F1 D21	247084	284	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
30	NIK 1 F1 D22	247085	405	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
31	NIK 1 F1 D23	247086	378	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
32	NIK 1 F1 D24	247087	378	08/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
33	NIK 1 F1 D25	247088	378	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
34	NIK 1 F1 D26	247089	420.58	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
35	NIK 1 F1 D28	247090	417.35	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
36	NIK 1 F1 D29	247091	441.07	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
37	NIK 1 F1 D30	247092	429.68	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
38	NIK 1 F1 D31	247093	498.84	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
39	NIK 1 F1 D32	247094	450	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
40	NIK 1 F1 D34	247095	400	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
41	NIK 1 F1 D35	247096	434.68	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
42	NIK 1 F1 D27	247097	324	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
43	NIK 1 F1 D33	247098	400	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
44	NIK 1 F1 D40	247099	450	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
45	NIK 1 F1 D41	247100	405	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
46	NIK 1 F1 D42	247101	405	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
47	NIK 1 F1 D36	247102	399.32	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active

TABLE 4.1 EL TIGRE PROPERTY MINERAL TITLES										
No.	**Lot**	**Title**	**Area (ha)**	**Validity Start**	**Validity End**	**Current Validity (years)**	**Location**	**Title Holder / Owner**	**Liens or Legal Issues**	**Status**
48	NIK 1 F1 D37	247103	400	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
49	NIK 1 F1 D38	247104	400	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
50	NIK 1 F1 D39	247105	125	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
51	NIK 1 F1 D43	247106	400.92	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
52	NIK 1 F1 D44	247107	410.14	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
53	NIK 1 F1 D45	247108	447.72	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
54	NIK 1 F1 D46	247109	438	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
55	NIK 1 F1 D47	247110	399	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
56	NIK 1 F1 D48	247111	420	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
57	NIK 1 F1 D49	247112	420	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
58	NIK 1 F1 D50	247113	450	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active
59	NIK 1 F1 D51	247114	420	15/06/22	04/07/57	15	Nacozari de Garcia, Sonora	Pacemaker Silver Mining S.A. de C.V.	Free	Active

Notes: *Concession information effective as of June 20, 2025, date of Legal Opinion by EC Rubio for Silver Tiger.*

Four of the concessions are owned by Compañía Minera Talaman S.A. de C.V. (Talaman) and 55 (including San Juan) are owned by Pacemaker Silver Mining S.A. de C.V. ("Pacemaker"). Pacemaker is a wholly-owned subsidiary of Silver Tiger and Talaman is a wholly-owned subsidiary of Pacemaker, and therefore these four concessions are controlled by Silver Tiger. Concessions El Aguila (Title: 172113), La Fundadora (Title: 172112), Tigre Suertudo (Title: 168634), NIK 1 F1 D26 (Title: 247089), NIK 1 F1 D30 (Title: 247092), NIK 1 F1 D34 (Title: 247095), and NIK 1 F1 D35 (Title: 247096) cover the area of the current Mineral Resources reported in Section 14 of this Technical Report.

The Property was acquired in November 2015 by Oceanus Resources Corporation ("Oceanus", the precursor company to Silver Tiger), through the acquisition of all the issued and outstanding common shares of El Tigre Silver Corporation ("El Tigre Silver"), whereby each outstanding El Tigre Silver share was exchanged for 0.2839 of one common share of Oceanus. Following the acquisition of El Tigre Silver, Pacemaker became a 100% indirectly owned Mexican subsidiary of Oceanus. On May 14, 2020, Oceanus announced a name change to Silver Tiger Metals Inc.

Until 2022, the El Tigre Property consisted of nine concessions (El Aguila, Jorge, La Fundadora, Tigre Suertudo, Nik Frac. 2, San Juan, La Carabina Frac 1, La Carabina, Frac 2, and Nik 1 F1). Concession Nik 1 F1 (21,156.3 ha) expired in 2022 and was subdivided into 51 new valid concessions (Nik 1 F1 D1 to Nik 1 F1 D51) that cover the previous surface of Nik 1 F1 and to which Pacemaker holds legal title. The 59 concessions of the El Tigre Property are all registered with the Registro Público de Mineria as exploitation concessions. EC Rubio, Silver Tiger's Mexican legal counsel, has confirmed that as of August 12, 2024, the concessions are in good standing.

4.3 MEXICAN MINERAL POLICY

Mining exploration in Mexico is regulated by the Mining Law of 1992, amended in 2005, which establishes that all minerals are owned by the Mexican nation and that private parties may exploit such minerals (except oil, gas and nuclear fuel minerals) through mining licenses, or concessions granted by the federal government.

A mining concession gives the holder both exploration and exploitation rights subject to the payment of relevant taxes. Mining concessions have a term of 50 years from the date the exploration or exploitation concession was registered and are renewable for an additional 50-year term. Concessions may be granted to (or acquired by) Mexican individuals, local communities with collective ownership of the land, known as "ejidos", and companies incorporated in Mexico in accordance with Mexican law.

Mining concessions must be registered with the Registro Público de Mineria as either an exploration, exploitation, or beneficial plant concession. The 2005 amendment changed the term of exploration concessions from six years to 50 years, matching the term granted for exploitation concessions. The amendment also allowed for exploration concessions to be renewed for an additional 50-year term.

Mexican mining law requires a concession applicant to hire a licensed land surveyor (a "Perito Minero") to locate the corners and boundaries of the concession with respect to a substantial

physical concession location monument (a "Punto Partida"). The Punto Partida is constructed at a prominent location within the concession by the applicant and is painted white, and then the name of the claim is painted, engraved or affixed in some other permanent manner to it. The land surveyor locates the Punto Partida in UTM coordinates with a specified datum. The corners of the concession are surveyed in UTM coordinates using the Punto Partida as the principal reference point. The survey data collected becomes the legal description of the concession with the concession is granted. After the concession has been granted, the concession number must be affixed to the Punto Partida. Although some corner markers may become lost or destroyed over time, these locations can be re-established via the Punto Partida, which the owner is obliged to maintain in an identifiable condition.

4.4 SURFACE RIGHTS

Under Mexican mining law, tailings revert to the owner of the surface estate when the concession owner who created the tailings allows the concession to lapse. Currently, ownership and responsibility for the tailings at El Tigre belongs to the owner of the surface estate, who is a private landowner. On May 11, 2017, El Tigre Silver completed full consolidation of the El Tigre Property by signing a Lease/Purchase agreement with Martin Lopez Lauterio's executor, Mrs. Maria Angelica Mares Mungaray, for the surface land and tailings from the historical operation of the Lucky Tiger Combination Gold Mining Company. The tailings are located on the El Tigre mining concessions. Under the terms of the Lease/Purchase Agreement, El Tigre Silver, through its wholly owned Mexican subsidiary, Pacemaker, can process the tailings and extract the contained metal at any time. Under the terms of the agreement, Pacemaker will pay the owner US$1,030,000 in 84 equal monthly payments. Pacemaker is also required to pay the owner a fee of either US$0.50, $1.00, $1.50 or $2.00/t extracted, depending on the commercial price of gold (<US$1,300), from US$1,301 to $1,500, from US$1,501 to $1,800, and >US$1,801). Upon reaching commercial production, Pacemaker is required to pay the owner US$500,000 as a bonus payment, with the payment to be made in 12 equal monthly installments.

Silver Tiger controls 21,775 ha of mineral rights and has agreements in place with local ranchers sufficient to support a mining operation, including areas for mining, leaching, processing, tailings and waste rock disposal.

4.5 ENVIRONMENTAL AND PERMITTING

The El Tigre Mining District is typical of many historical mining districts in Mexico, in that it has numerous open shafts, open stopes, drifts, historical buildings and foundations, tailings, and water draining out of flooded workings. Water drains from the Level 7 portal at an average rate of 38 l/min (10 US gal/min).

There are no known cultural restrictions on exploration activity. However, it is important to respect the historical mining ruins. A small historical church is near the main camp and is maintained and visited by residents of the region. A graveyard is also present near the main camp and appropriate care will need to be taken to prevent disturbance of the site. An Environmental Impact Statement (an "informe preventivo") must be issued and filed with Secretaría de Medio Ambiente y Recursos Naturales ("SEMARNAT") for any expected surface land disturbance, such as road building or

mining. This statement must outline the work to be done, state any surface disturbance planned, and what measures will be taken to mitigate surface and other environmental disturbances.

If SEMARNAT determines that the environmental disturbance will be significant, a reclamation bond may be required before work can resume. If extensive road building is required, a "Cambio uso de Suelo" plan may need to be filed with the Procudraduria Federal de Protección al Medio Ambiente ("PROFEPA"). Extensive road building is not considered to be necessary for exploration at El Tigre.

4.6 COMMENTS ON SECTION 4

The Author is not aware of any significant factors or risks besides those discussed in this Technical Report that may affect access, title or right or ability to perform work on the Property by Silver Tiger, or any other party that may be engaged to undertake work on the Property by Silver Tiger. It is the Authors' understanding that further permitting and environmental studies would be required if the Property were to advance beyond the project development stage.

The El Tigre Project area is large enough to accommodate the necessary infrastructure to support a mining and processing operation, should the economics of the mineral deposits be sufficient to warrant production.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The information in this section has been summarized from Micon (2023) and updated where applicable.

5.1 ACCESS

From Agua Prieta, the El Tigre Property can be reached by driving 75 km south along Mexican Highway 17 to the Town of Esqueda, and then 46 km east from there on dirt road to the El Tigre camp (Figure 5.1). Large stretches of the road from Esqueda are intermittently maintained by local ranchers on either side of Lake Angostura. Alternate access routes include a crossing at the Lake Angostura dam to the south or at Colonia Morelos or Fresno Ranch to the north. These alternate routes are only viable when the Rio Bavispe is low or dry. Access during the monsoon season is hindered by flash floods, which periodically wash out sections of road and generally cause rough road conditions.

Road improvements were made during the final 5 km of road leading to the Property to allow safe access of drilling and support equipment. These improvements included grading, widening the road in places, adding fill in spots and installing some ditches and water restriction bars. Although the area is dry for much of the year, provisions have been made for drainage during the monsoon season, in order to ensure long term use of the road surface.

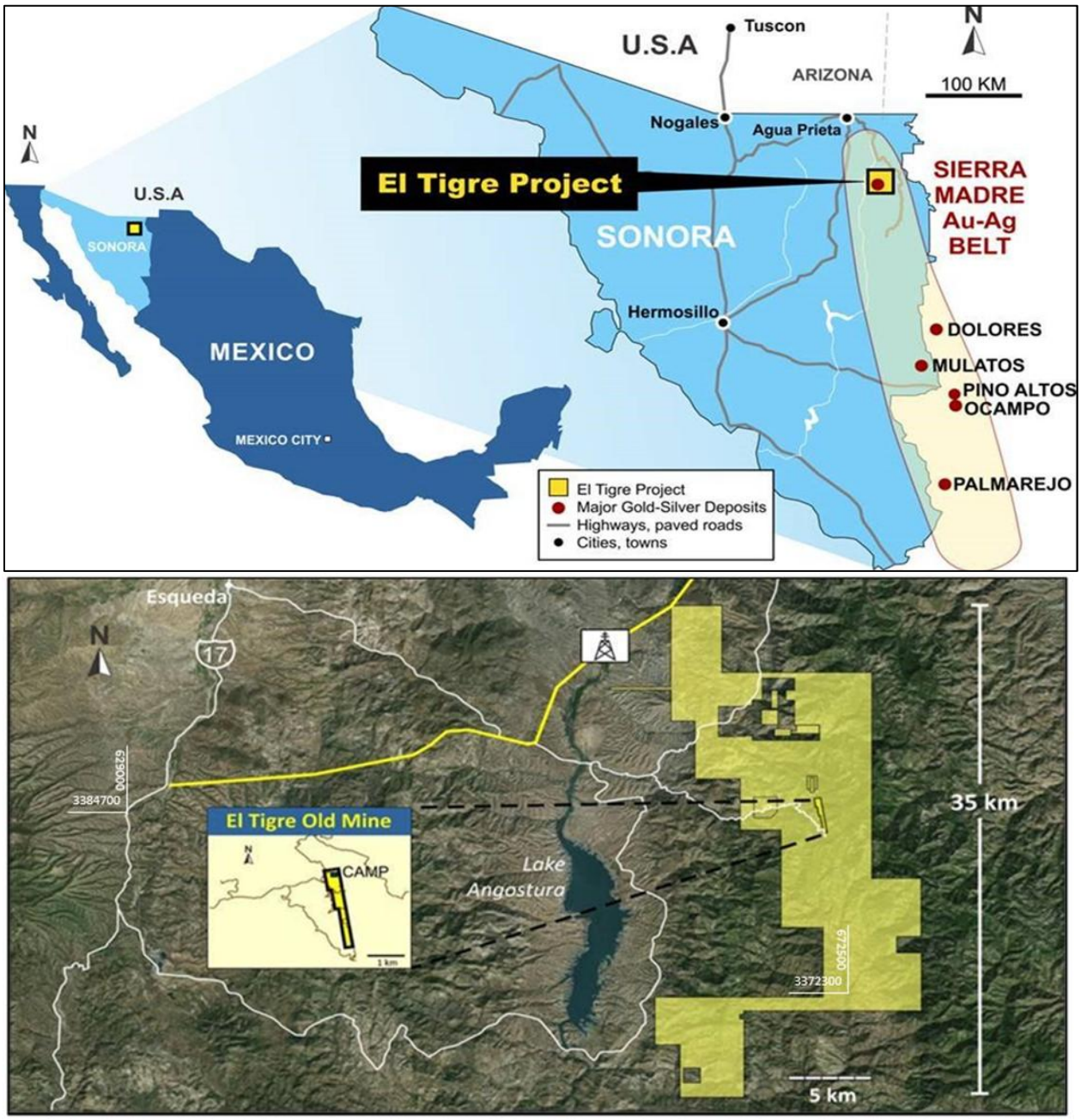
In 2023, Silver Tiger completed rehabilitation of a 7 m wide access road and some pluvial works from Colonia Morelos to the exploration camp over 46 km in order that it will be accessible all year round.

5.2 CLIMATE

The climate of the El Tigre area is typical of the Madrean Archipilego/Sky Island Region, which is semi-arid with bi-seasonal precipitation. Winter precipitation is associated with frontal storms from the Pacific Ocean. Winter conditions generally last from October through May, with the most intense storms occurring between mid-November and mid-April. Late spring and early summer are typically dry and summer monsoon moisture begins to enter the region in late-June to early-July. Storms are the result of tropical air flowing over heated mountain terrain, with frequent torrential rains occurring during the afternoon and thunderstorms in the evenings.

Temperatures are elevation dependent. In the lowlands, near La Angostura Reservoir, summer temperatures can reach 50°C and winter temperatures can be as low 0°C. At the El Tigre camp site, summer temperatures rarely exceed 40°C and winter temperatures can be as low as -15°C on the coldest nights. Winter precipitation generally falls as rain; however, the higher peaks of the Sierra El Tigre can be snow covered.

FIGURE 5.1 EL TIGRE PROPERTY ACCESS



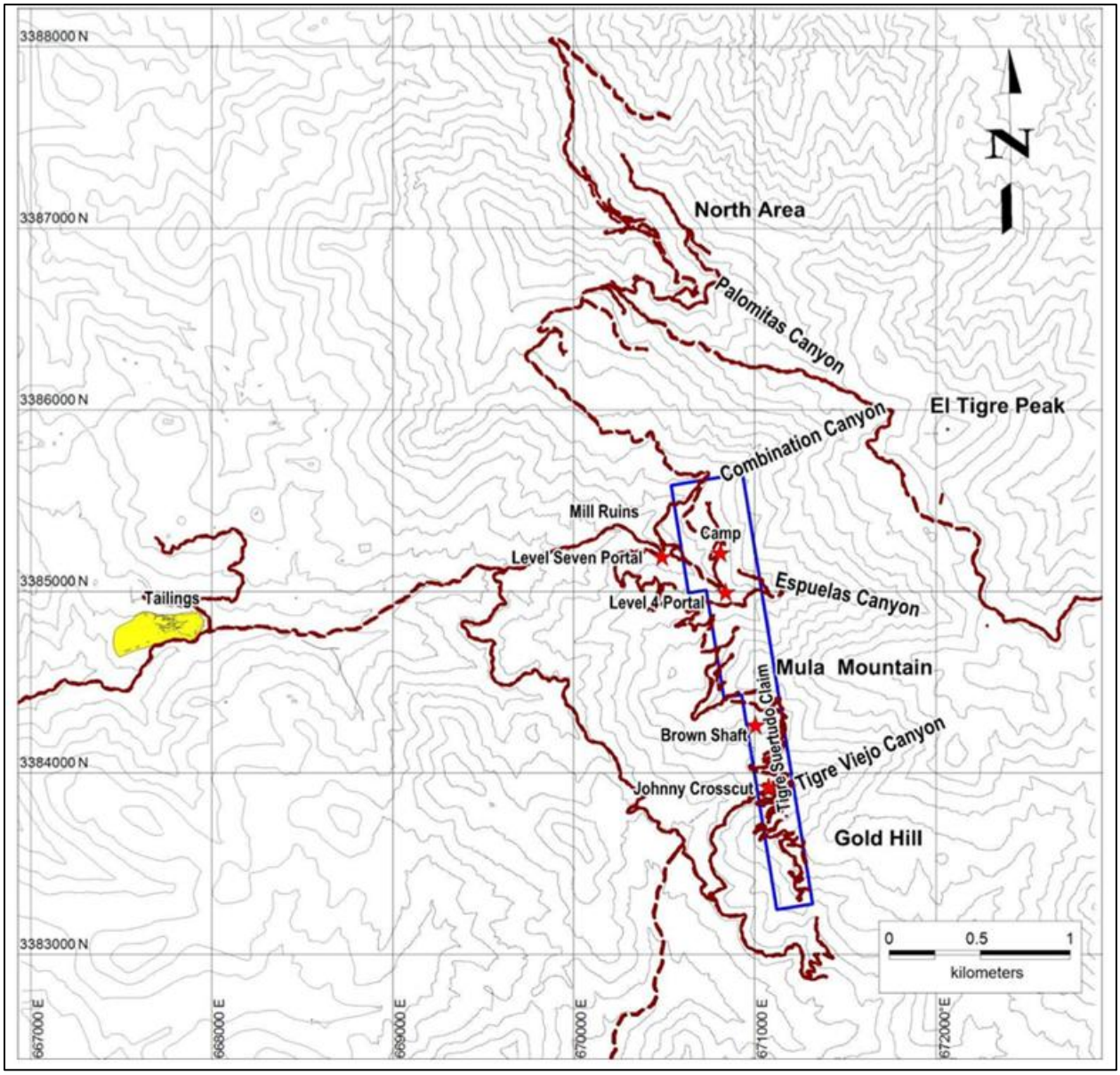
Source: *Modified by P&E (This Report) from Micon (2023)*

5.3 LOCAL RESOURCES AND INFRASTRUCTURE

The El Tigre Property is remote with food, fuel and lodging available in Agua Prieta, a two hour drive from the camp. Personnel are lodged at the camp, which consists of a 45–person residence, office, shower, washroom, and kitchen facilities. A drill core logging and storage area is also present. Cellular reception is sporadic around the main camp. Satellite internet equipment is present at the camp; however, a tower would be required to improve cell reception. Supplies can be acquired from Agua Prieta or other communities with proper planning. Heavy equipment or

construction materials may require transport from larger cities such as Hermosillo. The general site layout is shown in Figure 5.2.

FIGURE 5.2 EL TIGRE GENERAL SITE LAYOUT



Source: *Black and Choquette (2013)*

Electricity at the El Tigre camp is provided by a portable generator. Sufficient water for camp, exploration and operating purposes comes from a spring uphill of the camp and from the Level 7 underground workings outflow of ~38 l/m (US 10 gal/min).

Mining personnel can be sourced locally or from Hermosillo. The Town of Esqueda, to the west of the camp, has historically supplied labour for the mining activities at El Tigre. Experienced mining and processing personnel can be sourced from the nearby mining towns of Nacozari and Agua Prieta, where the La Caridad and the Cananea copper mines operate.

5.4 PHYSIOGRAPHY

The El Tigre Property is situated on the western slope of the Sierra El Tigre. Elevation on the Property ranges between 1,500 and 2,000 masl and the terrain is rugged and mountainous (Figure 5.3). Drainage is via tributaries of the Rio Bavispe, many of which are seasonal, through several cliff-forming bedrock formations. Vegetation types are zoned in bands on mountains reflecting increased rainfall and decreasing temperatures and areas at higher elevations. Vegetation requiring more water is found above drier vegetation and species richness generally increases at higher elevations. Oak woodlands and pine-oak forests are found at higher elevations, which have more species of vegetation than lowland areas. Vegetation varies from the upper Sonoran yucca-ocotillo to manzanit-oak-pinyon-chaparral.

FIGURE 5.3 TERRAIN OF THE EL TIGRE PROPERTY

Source: *Wood (2009)*

5.5 COMMENTS ON SECTION 5

The Author considers that exploration activities and any potential mining operations can continue year-round, except during periods of heavy rainfall. The Author also considers that the El Tigre Property is sufficient in size to locate all potential infrastructure to host a mining operation.

6.0 HISTORY

The information for this section has been summarized from P&E (2017, 2023) and further information researched by the Author.

All drill hole survey and assay values are expressed in metric units, with grid coordinates reported in Universal Transverse Mercator (UTM) coordinate space relative to WGS 1984 UTM Zone 12.

6.1 EARLY HISTORY OF THE EL TIGRE PROJECT AREA

Visible gold was discovered in red hematite, iron-clay gangue near Gold Hill, ~1-km south of the present El Tigre Mine. The El Tigre Mining Company in Sonora, which was owned by the Lucky Tiger Combination Gold Mining Company of Kansas City, Missouri began mining in 1903.

The mineralization was rich enough to warrant direct shipment to the Douglas Smelter in Arizona. The original stamp mill in conjunction with concentrating tables and Frue vanners was constructed on-site to increase the recovery of the lower-grade ores. However, the silver in the form of cerargyrite (silver chloride) from the upper workings could not be concentrated by this method.

A 100 tpd concentration plant was constructed in 1907 using staged crushing with rollers to minimize the loss of slime. The new plant proved profitable largely due to the mine producing mineralized material with very little cerargyrite when it began operating and with the gold and silver intimately associated with galena, zinc-blende, chalcopyrite, and iron pyrite. The new plant in conjunction with the older stamp mill was capable of processing 175 tpd. This operation was profitable for several years.

In 1910, the average assay of the 38,610 short tons of mined material processed was 0.256 oz/ton Au, 33.568 oz/ton Ag, 1.06% CuO and 0.568% PbO, with zinc also present. A 250 ton/day cyanide plant was constructed on-site and was in operation by the end June 1911, treating 175 tons of mined mineralized material and 75 ton/day of old tailings from the dumps below the plant. The mineralized material was ground to at least 200 mesh to leach the silver. Copper in the mineralized material significantly increased cyanide consumption, although maximizing copper recovery in the concentration plant, reduced cyanide consumption was achieved, and the operation was profitable. Figure 6.1The general view, general plan and sectional views of the concentrator, and cyanide plant at the El Tigre Mine in 1911 are shown in Figure 6.1 to Figure 6.3 (Forbes, 1912).

Approximately 62 to 64% of the gold and silver in the mined material was recovered in the form of concentrates and hand sorted high grade. Recovery by the cyanidation of the concentrator-tailings was ~82% and of the dump-tailings of ~ 85%. Total recovery by hand sorting, milling and cyanidation was ~93 to 95%, according to Forbes (1912).

A 65 mile (105 km) electrical power line was constructed from the power plant of the Copper Queen Mining Co. ("Copper Queen") in Douglas, Arizona to the El Tigre process plant, to power the new process plant and cyanide treatment plant. The electrical power line resulted in a reduction of approximately two-thirds in the cost of power compared to the previous cost when gas and steam engines were used at the Mine. Interruptions to the transmission from lightning occurred during the summer rainy season, however, they seldom exceeded an hour in duration.

FIGURE 6.1 GENERAL VIEW OF THE MILL AND CYANIDE PLANT AT THE EL TIGRE MINE IN 1911

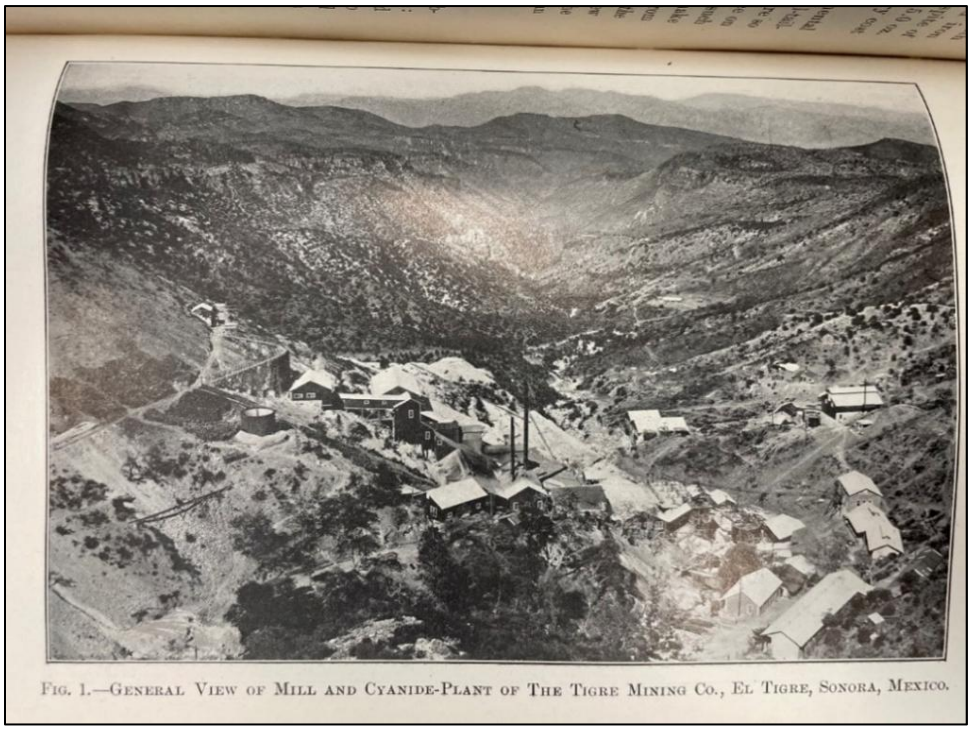


FIG. 1.—GENERAL VIEW OF MILL AND CYANIDE-PLANT OF THE TIGRE MINING CO., EL TIGRE, SONORA, MEXICO.

Source: *D.L.H. Forbes (1912)*

FIGURE 6.2 SECTIONAL VIEWS OF THE MILL AND CYANIDE PLANT

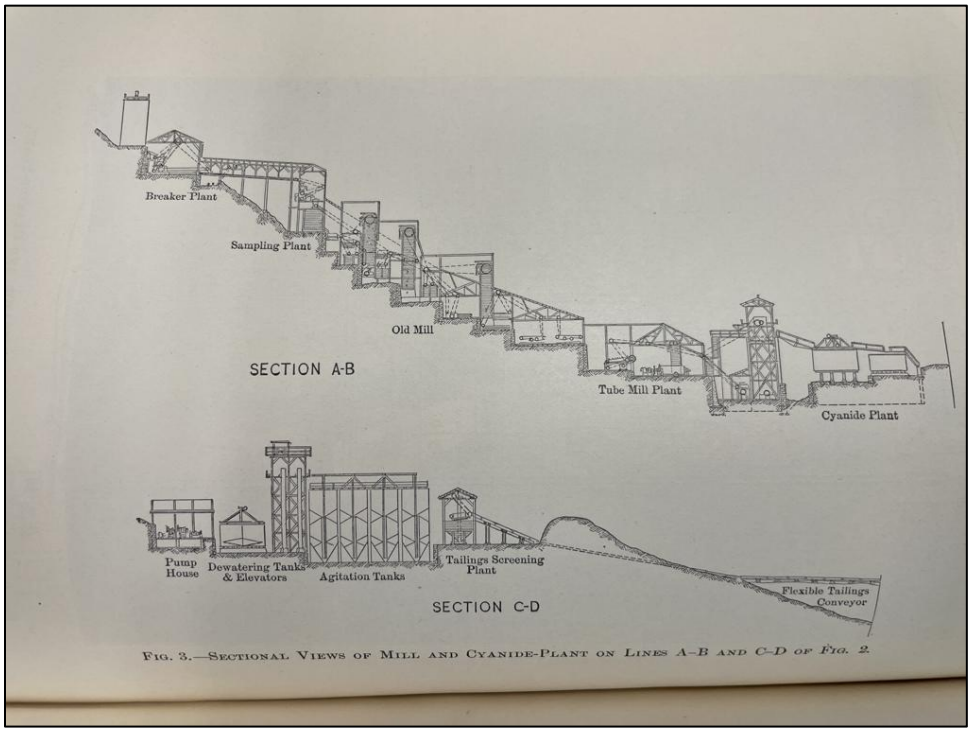


FIG. 3.—SECTIONAL VIEWS OF MILL AND CYANIDE-PLANT ON LINES A–B AND C–D OF FIG. 2.

Source: *D.L.H. Forbes (1912)*

FIGURE 6.3 PLAN VIEW OF THE CONCENTRATOR AND CYANIDE PLANT

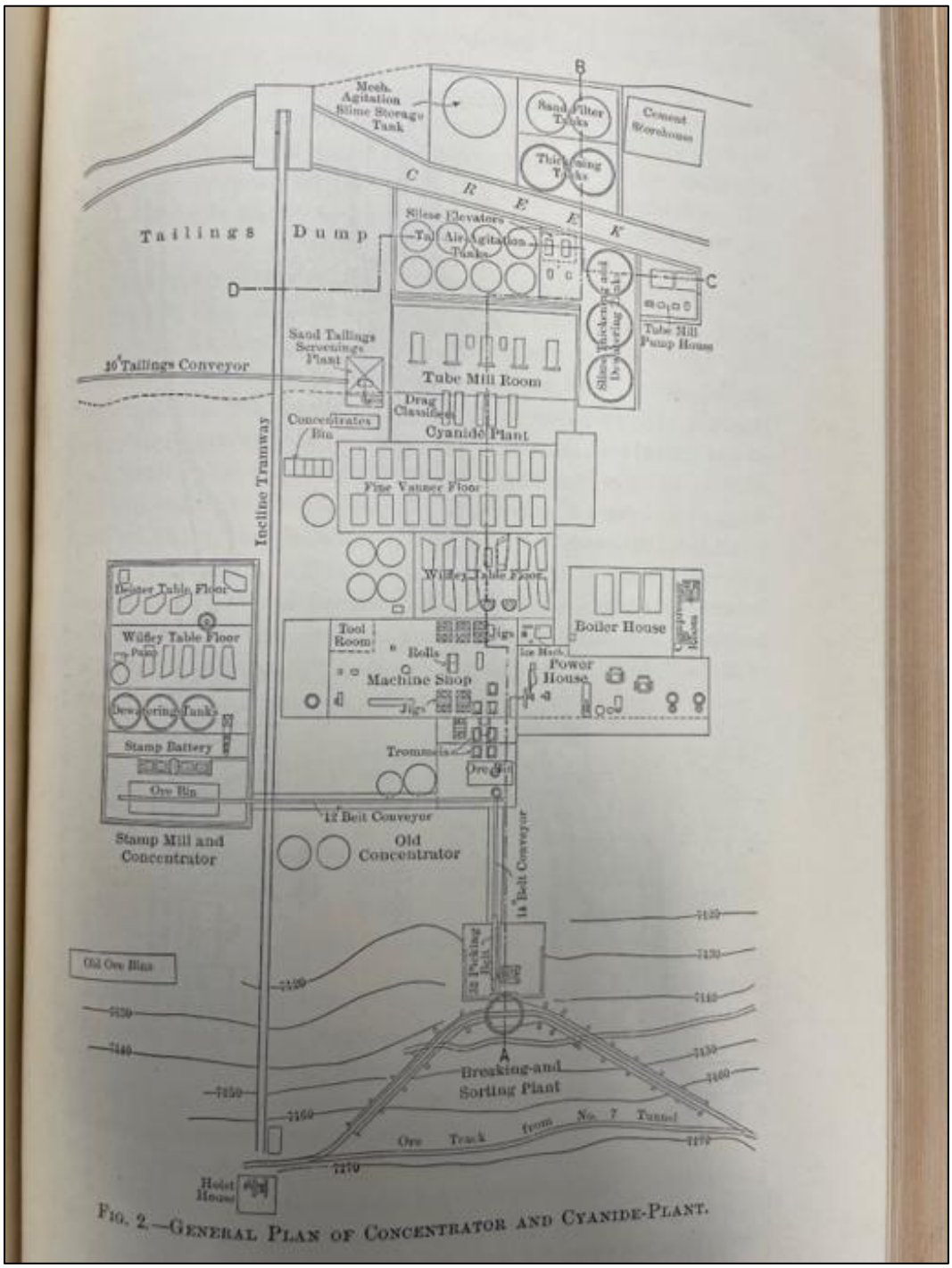


FIG. 2.—GENERAL PLAN OF CONCENTRATOR AND CYANIDE-PLANT.

Source: *D.L.H. Forbes (1912)*

Tailings were discharged directly into Tigre Canyon during the first year of operating the cyanide leach plant and were washed downstream by rains. Lucky Tiger was forced by the Mexican government to develop a tailings impoundment because the downstream cyanide contamination was killing cattle. The tailings impoundment was built in 1912 on a small mesa above the canyon

(16.2 ha), approximately two miles (3.2 km) west of the process plant. Dam berms were constructed from dried tailings.

The impoundment was divided into two cells: one for active storage of wet tailings, and one for drying. Lucky Tiger was able to continue operations after the tailings impoundment area was completed.

In September 1912, the El Tigre Mine was attacked by Orozquistas[1] causing $30,000 in damages. They looted the town and carried off $50,000 in gold and silver bullion. However, the mine manager saved the bullion by casting it into 400-pound ingots that caused the overloaded mules to collapse, and the bullion was recovered from the retreating revolutionaries. In 1913, El Tigre shipped nine million pesos worth of metal.

During the 1910 Mexican Revolution, Pancho Villa attacked Cananea, Nacozari and El Tigre and numerous properties in Sonora and other Mexican states became idle, were severely damaged or overburdened with numerous taxes raised by both sides in the conflict. After 1917, when the main conflicts died down, mining and smelters were slow to re-open. By 1922, a revival of the mining industry was underway. In 1922, El Tigre reported its lowest costs since 1916: wages and marketing costs were down and production in tons-per-man-shift was up 17%.

During the early years of mining, processing and cyanide leaching of the tails, the operation shipped hand-sorted high-grade ore, gravity concentrates and bullion. It was reported that overall silver and gold recovery was 93 to 95%. The feed to the process plant was ~30 oz/t to 40 oz/t Ag and 0.10 oz/t to 0.15 oz/t Au. The high-grade hand-sorted mineralized material and gravity concentrate each were approximately 350 oz/t Ag and 1.5 oz/t Au.

In 1923, the US Department of Commerce reported that up to 1920, 11,217 ft of development work had been undertaken at the mine and that the company, organized in 1903 at Kansas City, MO, had paid dividends up to and including 1920, of US$7,469,572. In the 1920s, mining at greater depths resulted in less oxidized material and cyanide leaching of the tailings became impractical. The process plant was modified to accommodate a new flotation process and only flotation tailings were discharged.

A court case in the US for the recovery of an alleged overpayment of income taxes for the year 1925 noted that, "By the end of 1925 the total amount of silver extracted from the mine since March 1, 1913, was 30,535,243 ounces, or 1,056,756 ounces more than the estimated contents of the mine, and there were large reserves of mineralized material remaining. In 1925, the Mexican Company recovered from the Mine 2,455,156 ounces of silver" (*Lucky Tiger-Combination Gold Mining v. Crooks*, 95 F.2d 885, 886 (8th Cir. 1938)).

Mineral Reserves were mainly depleted by 1930, and Lucky Tiger closed the El Tigre operation in 1931. Operations were resumed with the introduction of unionized labourers. However, in 1938, the mine closed permanently due to low silver prices, increased union demands, and a new 11% production royalty that caused the mine to become uneconomic. Unregulated mining continued by

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Orozquistas (Colorados) Followers of Pascual Orozco, also known as the Colorados (Red Flaggers). They first fought for Madero, 1910-11, and rebelled against his government in 1912 under the Orozquistas Plan, before joining Huerta's army in February 1913.

informal miners, known as gambusinos, and eventually anything of value had been removed from the mine and process plant site.

6.2 HISTORY OF EL TIGRE OWNERSHIP

In the late 1960s, the El Tigre mining concessions and the tailings were acquired by Sr. Higenio Garcia of Agua Prieta, Sonora, and were subsequently incorporated into a Mexican mining company known as Cia. Jaleros del Tigre, S.A. de C.V. ("Jaleros del Tigre"). In 1972, the Property was optioned to a US financed company known as Cia. Minera Sonrisa, S.A. de C.V. ("Sonrisa"). Sonrisa carried out a major evaluation of the El Tigre tailings. However, due to the untimely death of one of its principals, Sonrisa did not exercise its option.

Talaman was formed specifically to acquire the El Tigre properties and tailings. In 1978, Talaman optioned the Property from Jaleros del Tigre and continued the evaluation of the tailings and commenced preliminary work to put the tailings into production.

In 1981, Anaconda Minerals Company ("Anaconda") entered into an option agreement with Talaman to acquire the Property. Anaconda assumed and fulfilled Talaman's contractual obligations to Jaleros del Tigre. The Property position was expanded and consolidated. In 1984, citing unsatisfactory exploration results, Anaconda withdrew from the Talaman agreement. Talaman maintained the Property after 1984. Pacemaker Silver Mining S.A. de C.V. ("Pacemaker") then acquired Talaman.

On January 28, 2010, Herdron Capital Corp. ("Herdron") a capital pool company listed on the TSX-V exchange, agreed to acquire 100% of the issued and outstanding shares of Pacemaker. Upon completion of the transaction in February 2010, Pacemaker became a wholly-owned subsidiary of Herdron, and Herdron changed its name to El Tigre Silver Corporation ("El Tigre Silver").

In November 2015, Oceanus acquired the Property, through the acquisition of all the issued and outstanding common shares of El Tigre Silver whereby each outstanding El Tigre Silver share was exchanged for 0.2839 of one common share of Oceanus.

On May 14, 2020, Oceanus announced that it was changing its name to Silver Tiger Metals Inc. ("Silver Tiger").

6.3 MODERN EXPLORATION PROGRAMS

Modern exploration in the El Tigre area commenced with Anaconda in the 1980s and continued with Minera de Cordilleras 1995 and particularly El Tigre Silver from 2008 to 2013 and Oceanus Exploration 2016-2017. The work by each of these companies is summarized below, starting with Anaconda.

6.3.1 Anaconda

Modern exploration was initiated in 1981 by Anaconda through its wholly-owned subsidiary Cobre de Hercules ("Cobre"). These exploration efforts lasted 29 months and ended around the time Anaconda shut down all mining and exploration activities.

The 29-month exploration program included surface geological mapping at 1:10,000 and 1:2,000 scales, underground prospect surveying, underground geological mapping at 1:500 scale, diamond drilling of the vein structures with 22 drill holes totalling 7,812 m, 352 m of exploration drifting at the Fundadora Vein, road rebuilding from Esqueda, drill pad road construction, aerial photography, petrographic studies, tailings surveying, sampling and metallurgical testwork of the tailings, maintenance of the legal land status of the concessions, and production of land-controlled photogrammetric base maps.

Anaconda's exploration program was designed to achieve three main objectives:

1. Identify extensions of known veins for a high-grade underground operation;
2. Explore the lower-grade silver mineralization for its bulk tonnage potential; and
3. Evaluate the economic viability of reprocessing the tailings.

Objective two was abandoned early in the program and attention was focused on objectives one and three. Activities were halted in 1984 "due to a lack of sufficiently encouraging results." However, during this time silver prices were fluctuating downward and almost all of Anaconda's exploration activities were halted and Atlantic Richfield Company disbanded Anaconda Minerals Company shortly thereafter.

Much of the technical information produced by Cobre de Hercules, and other related information held by Anaconda Minerals Company, was recovered by Pacemaker.

6.3.1.1 Anaconda Mapping

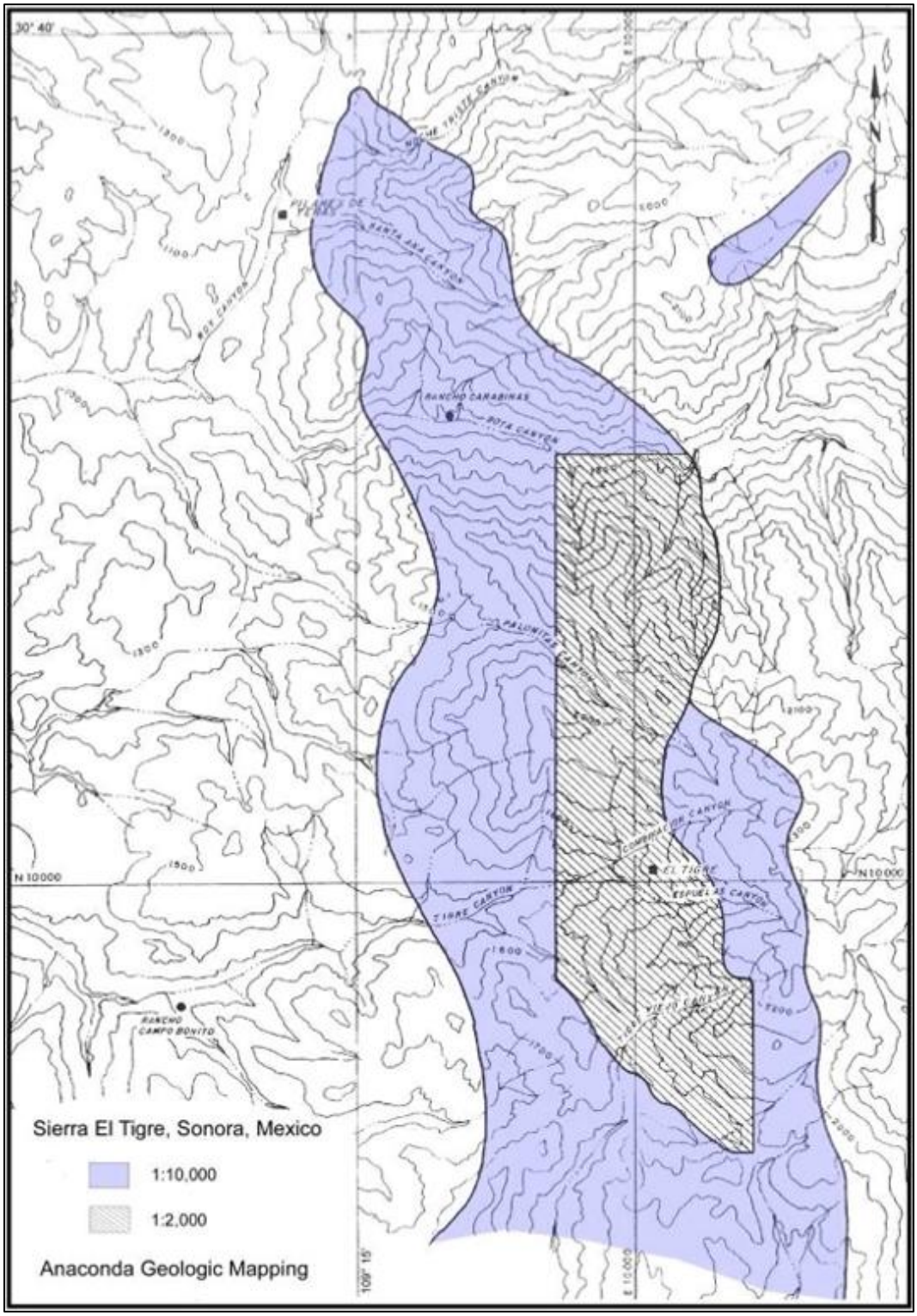
Anaconda completed district scale mapping of the area between 1981 and 1984. Surface geologic maps of the Project area were made at scales of 1:10,000 and 1:2000, covering an area from slightly south of the El Tigre Suertudo concession and north to the Pilares de Teras area (Figure 6.4). Pacemaker recovered excellent copies of Anaconda's mapping from the Anaconda Collection at the University of Wyoming and maintains them in project files. Pacemaker converted scans of both the Anaconda 1:10,000 and 1:2000 scale geologic maps into precise AutoCAD drawings.

6.3.1.2 Anaconda Geochemical Data

Anaconda also completed surface soil and stream sediment geochemical surveying over the Project area. Unfortunately, most of the records related to that work were not recovered. The only document relating to the work that has been recovered is a short progress report (Gee, 1982). In the summary, 10 geochemical anomalies were identified in the larger El Tigre region (Figure 6.5). The spacing of soil and stream sediment samples is unknown; however, the progress report suggests that the sampling was sparse, and it is assumed to be a first-pass sampling program. It is

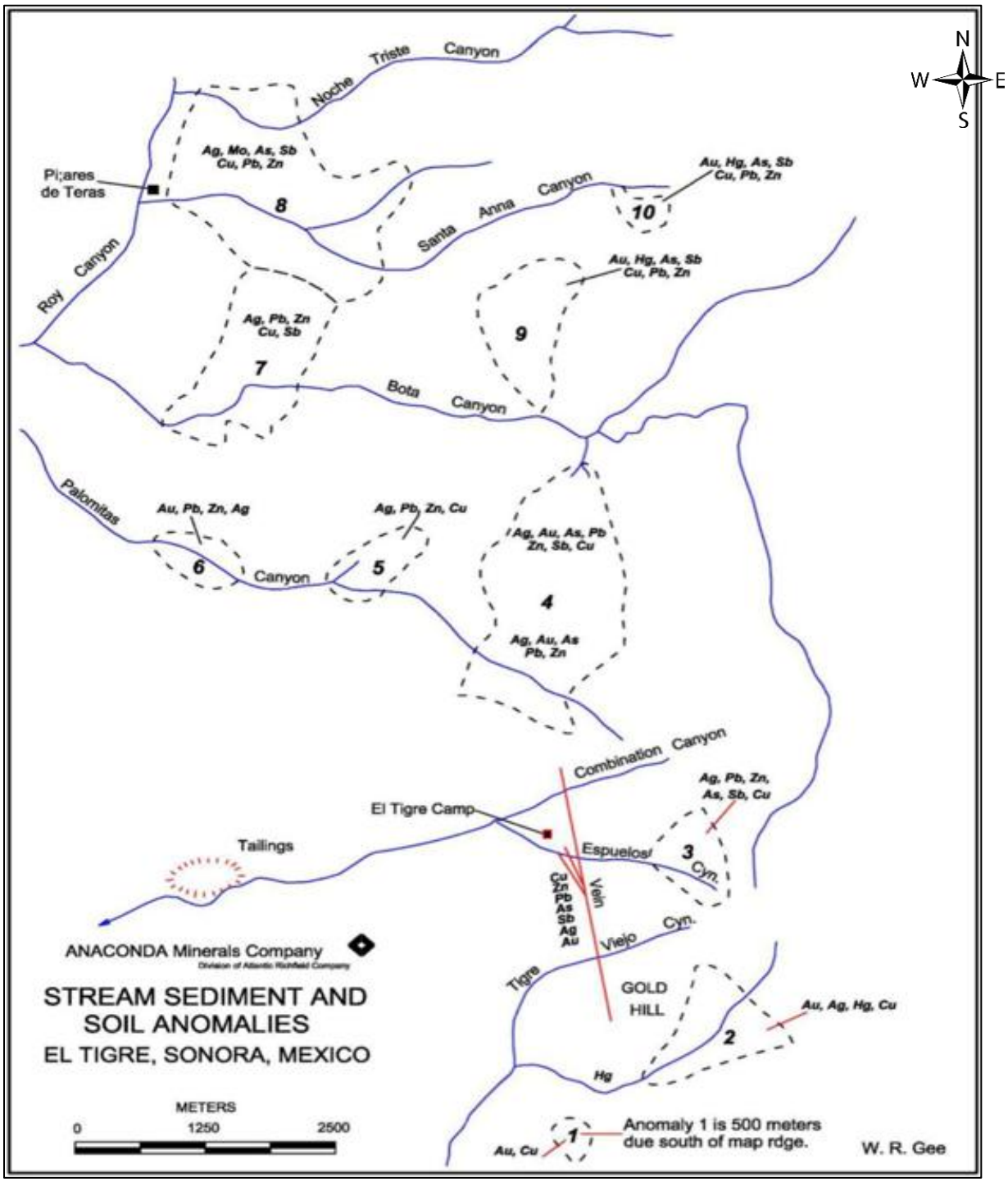
unknown whether Anaconda did any follow-up work based on the results of the initial geochemical survey.

FIGURE 6.4 ANACONDA GEOLOGICAL MAPPING COVERAGE, EL TIGRE PROJECT



Source: *Black and Choquette (2013)*

FIGURE 6.5 ANACONDA GEOCHEMICAL SAMPLING COVERAGE, EL TIGRE PROJECT



Source: *Black and Choquette (2013)*

As of the publication date for the 2017 P&E Technical Report, Oceanus had recovered all the maps and multi-element assay records for the Anaconda geochemical soil and silt surveys from the Anaconda Collection at the University of Wyoming Records that contained analytical results for 850 samples for Au, Ag, Cu, Pb, Zn, Sb, Mo, Hg, Mn and Fe.

6.3.1.3 Anaconda Drill Data

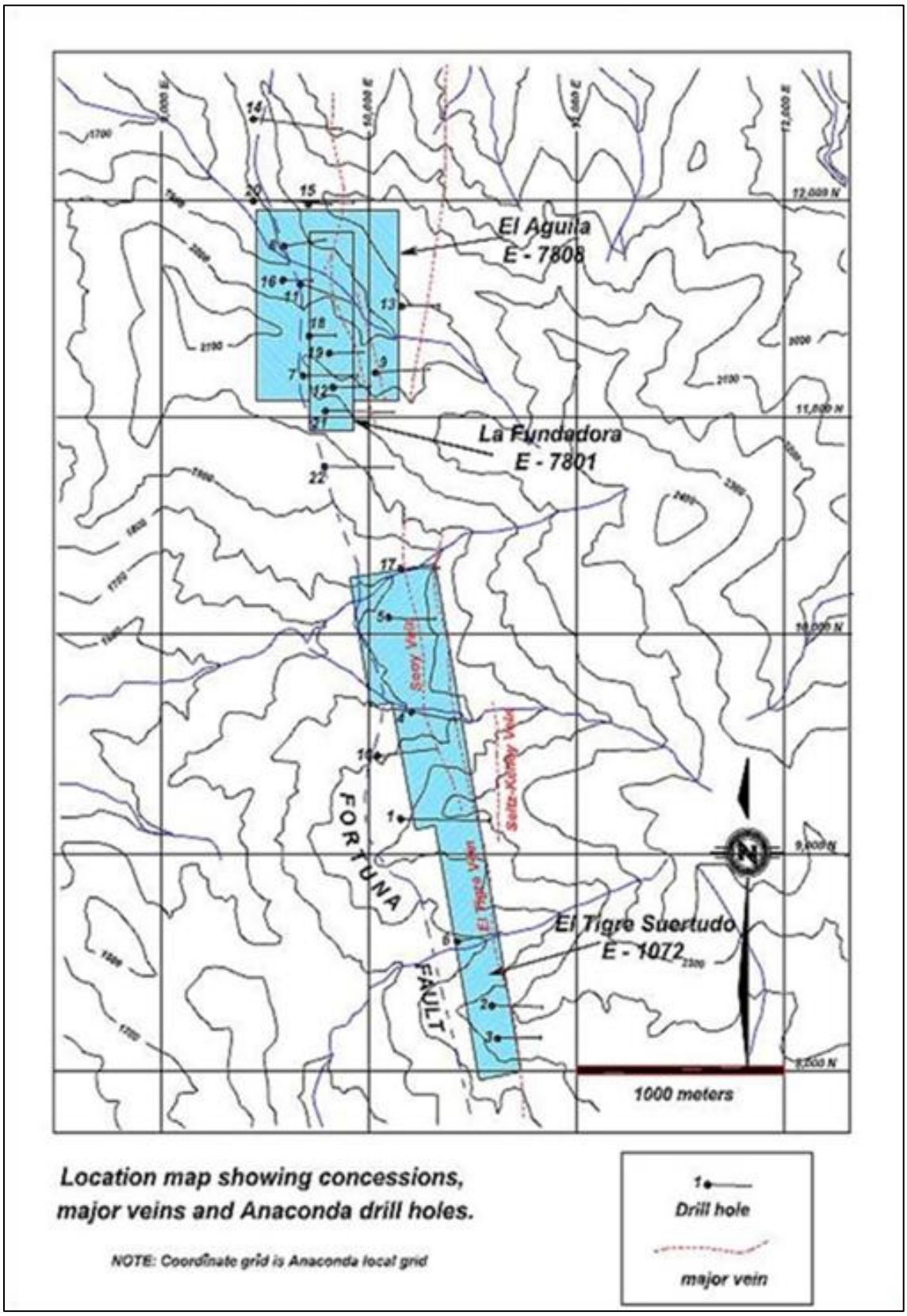
The database of historical Anaconda drilling compiled by Pacemaker contains collar locations, geological drill hole logs and down-hole assay information for 22 diamond drill holes totalling 7,813 m. All drill holes were surveyed in a local mine grid with down-the-hole Sperry-Sun instrument surveys to determine the location of vein intercepts and other geologic features at depth. All drill holes were inclined, between -40 and -61°, and drill hole lengths varied from 140 to 650 m. A summary of Anaconda's significant drill hole assay intervals is provided in Table 6.1. The locations of the Anaconda drill holes are shown in Figure 6.6.

TABLE 6.1
SUMMARY OF THE SIGNIFICANT INTERSECTIONS FROM THE ANACONDA DRILLING PROGRAM

Drill Hole ID	From (m)	To (m)	Length (m)	Ag (g/t)	Au (g/t)	Cu (%)	Pb (%)	Zn (%)	Target
T-1	166.85	168.85	2	316	0.03	NR	NR	NR	Tigre Vein
T-2	70	80	94	0.04	0.04	NR	NR	NR	Tigre Vein
T-3	125	134	9	55	1.50	*	*	*	Tigre Vein
T-4	59.55	59.80	0.25	292	0.06	0.14	0.47	*	Sooy Vein
and	167.45	169.66	2.21	98	0.10	*	0.22	0.32	Tigre Vein
T-5	144.15	144.25	0.10	350	0.19	0.20	0.70	4.65	Sooy Vein
T-6	30	33	3	*	0.91	*	*	*	Iron stained zone
T-7	282.0	282.2	0.2	250	0.89	0.54	6.80	1.01	Aguila Vein
and	368.00	368.25	0.25	258	0.07	1.65	9.80	22.50	Escondida Vein
T-8	217.7	217.9	0.2	29	0.21	*	2.26	7.44	Fundadora Vein
T-9	127.3	130.0	2.7	28	2.19	*	*	*	Unknown Vein
T-10	275.80	276.15	0.35	147	0.07	0.48	0.90	1.60	Sooy Vein
T-11	112.00	113.45	1.45	303	0.08	0.22	1.20	1.80	Fundadora Vein
and	364.4	364.7	0.3	34	0.05	*	0.14	2.20	Escondida Vein
T-12	114.0	116.5	2.5	256	1.10	*	*	*	Aguila Vein
and	212.1	212.3	0.2	1,700	0.30	0.39	0.89	1.10	Escondida Vein
T-14	473.00	473.15	0.15	408	1.80	4.00	8.40	9.30	Fundadora Vein
T-15	172.0	173.5	1.5	570	13.70	*	0.23	0.76	Fundadora Vein
T-16	192.65	192.80	0.15	153	4.60	5.40	*	0.40	Fundadora Vein
T-17	70.35	70.95	0.6	103	0.10	*	0.22	0.26	Unknown Vein
T-18	179.22	179.37	0.15	375	2.00	*	*	*	
and	193.5	193.6	0.1	2,902	43.40	*	0.40	*	Aguila Vein
T-19	86.35	87.40	1.05	773	2.70	*	*	0.11	Unknown Vein
T-21	167.10	167.32	0.22	23	0.05	*	0.21	0.65	Aguila Vein
T-22	98.1	98.2	0.2	275	*	*	1.19	0.45	Unknown Vein

Source: *P&E (2017)*
Note: ** = not assayed*

FIGURE 6.6 LOCATION MAP FOR THE ANACONDA DRILL HOLES



Source: *P&E (2023)*

6.3.2 Minera de Cordilleras 1995

In June 1995, consulting firm Minera de Cordilleras S.A. de R.L. de C.V. ("Minera de Cordilleras") completed a four-hole RC drilling program for a total of 890 m on behalf of a third party. These drill holes were intended to test the concept that the deeper part of the vein system was faulted, which brought the veins closer to the surface. Assays are available for those drill holes; however, the drill hole collar locations are unknown.

6.3.3 El Tigre Silver Exploration 2008 to 2012

El Tigre exploration programs summarized from Micon (2023) and updated where appropriate.

6.3.3.1 2008 - 2009 Exploration Programs

In 2008 and 2009, El Tigre Silver's subsidiary Pacemaker recovered many of the old Anaconda exploration files from the Anaconda Collection at the University of Wyoming. In the fall of 2007, James A. Bradbury, P.E. (Bradbury, 2007), analyzed the Anaconda and other data and proposed a low-grade silver mineralization target between the El Tigre and Seitz-Kelly Veins. Bradbury stated that the Anaconda reports had proposed that a low-grade mineral potential may exist and could be considered an exploration target. Bradbury analyzed sample data from Anaconda surface sampling in the Espuelas Canyon area, and pre-1939 sample results from underground sampling on the 400 levels and 700 levels of the El Tigre Mine. This target was the focus of El Tigre Silver exploration that involved surface rock chip sampling and drilling.

Data synthesis and field work completed by El Tigre Silver identified five exploration targets on the El Tigre Property that warranted additional detailed field work. The five targets are listed below and are prioritized in order of El Tigre Silver's expected potential:

1. El Tigre-Seitz-Kelly Veins and stockwork mineralization;
2. Gold Hill disseminated gold in altered El Tigre Formation;
3. Fundadora–Aguila Veins and breccia pipes;
4. Porvenir Canyon Vein target on south side of Gold Hill; and
5. Main El Tigre high-grade vein target.

Exploration commenced on the two highest priority targets, El Tigre-Seitz-Kelly and Gold Hill, in 2010 and culminated with a drilling program that ended in May 2013.

Satellite Imagery

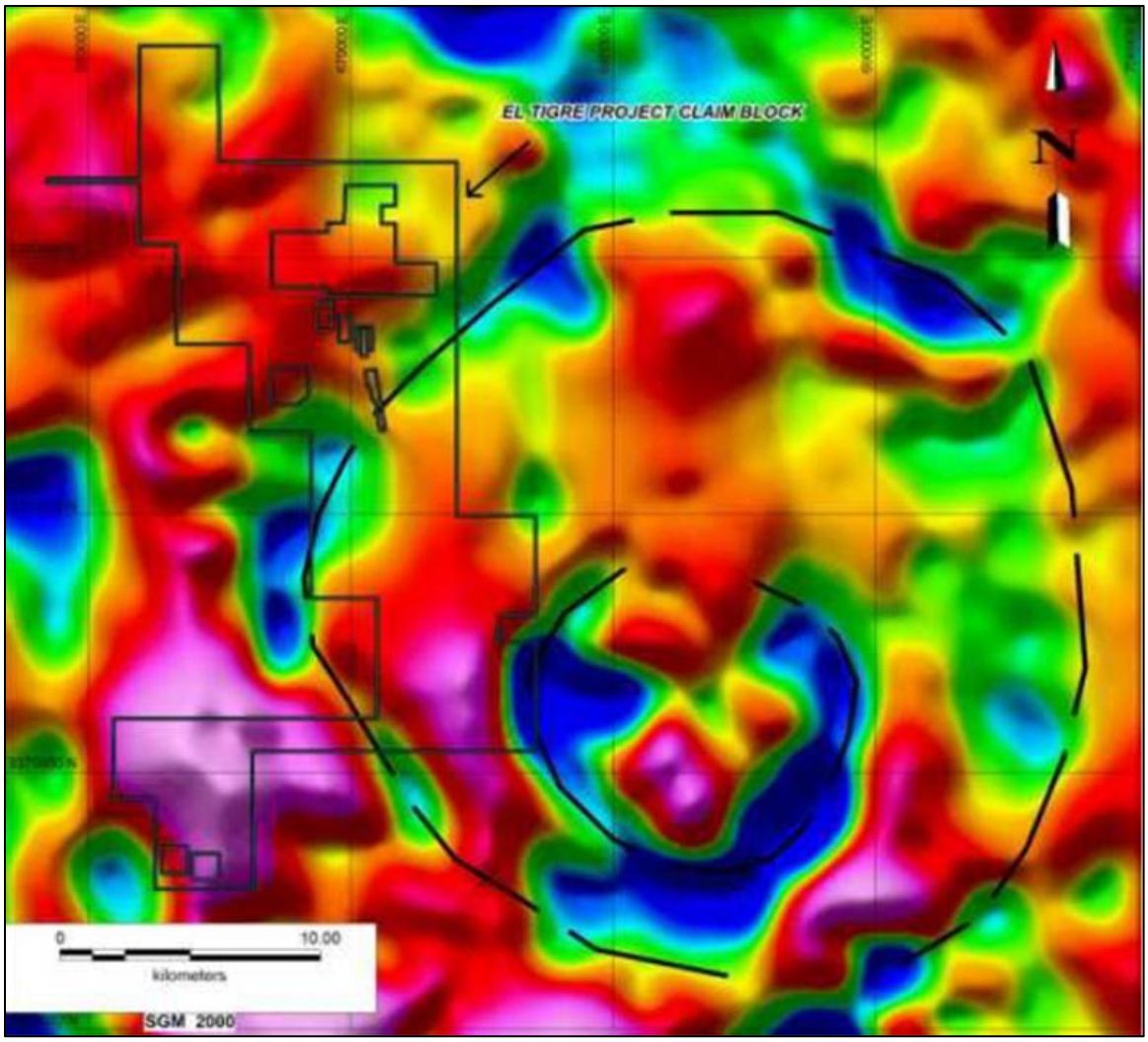
El Tigre Silver contracted Photosat Information, Ltd ("Photosat") of Vancouver, British Columbia, to generate a series of base maps for El Tigre. Using data from the Geoeye satellite, Photosat produced 100 km^2 of digital imagery with a 0.5 m pixel resolution and 45 km^2 of topographic coverage with a 10 cm vertical accuracy and 0.5 m x 0.5 m pixel size Digital Terrain Model ("DTM"). Contour maps with 1 m, 5 m and 10 m contour intervals were produced in Mapinfo™ GIS format. All El Tigre work by El Tigre Silver following receipt of the digital products from Photosat was completed using the World Geodetic System ("WGS") 84 UTM Zone 12 projection in metres.

In addition, Hard Rock Consulting ("HRC") was provided with a 1 m resolution aerial topography of the tailings from historical production. The provided topography covered ~495 ha. HRC created a 3-D topography surface for the entire area and draped the images over the topography to assist in delineating the boundaries of the tailings impoundment.

Geophysical Interpretation

Regional magnetics indicate two circular features possibly related to collapsed calderas to the east of the El Tigre Property. The larger one trends through the camp area and the faults that host mineralization may be associated with or be caldera collapse structures. A second smaller potential caldera resides inside the larger one and is represented by a strong magnetic low. Several circular tectonic features (possibly collapsed calderas) and associated north-trending linear fractures, are readily observed on band five TM Landsat images and on regional magnetics (Figure 6.7).

FIGURE 6.7 SGM REGIONAL MAGNETIC MAP (SGM 2000)



Source: *Black and Choquette (2013)*

Geologic Mapping Program

During October 2009, Lucas Ochoa Landin (2009) completed a field review that included mapping and sampling. Mr. Landin mapped the Espuelas Canyon area, where the veins cross the canyon bottom, and a portion of Mula Mountain and Gold Hill, to better identify the geology and sources of mineralization.

El Tigre geologists observed intense silicification, sericitization, iron oxides and fine-grained dark sulphide mineralization in faults, veinlets and shears semi-concordant to the El Tigre, Sooy and Seitz-Kelly Veins. This alteration extends southward for 2 km to Gold Hill and offered a perspective on the potential size of the mineralized system.

El Tigre Silver's Sampling Method and Approach

In order to establish exploration drill hole targets and support Mineral Resource Estimates within the 2 km strike length of the El Tigre Vein system, El Tigre Silver collected surface outcrop, dump, underground channel and surface channel samples. Additionally, test pit and trench samples were collected on the historical tailings impoundment.

Surface chip channel samples were marked by a line at each end of the channel and collected across zones of mineralization, alteration and structure by taking continuous (~10 cm width) chips from a saw cut made during a geologically defined traverse. The sample was chipped from the face with a mallet and chisel and captured by a large canvas. The canvas was cleaned after each sample had been taken and a lithologic description recorded. The samples ranged from 1 m to 2 m long, depending on degree of mineralization and weighed ~3 to 6 kg. Their location was recorded by a Garmin hand-held GPS unit.

Underground channel samples were marked by a line at each end of the channel and separated by structure and rock type. The sample site was cleaned with a wire brush to remove any dust and a 3 to 6 kg sample was chipped from the face with a mallet and chisel and captured by a large canvas. The canvas was cleaned after each sample has been taken and a lithologic description recorded. The sample bags were numbered and sealed with a sample tag inside. Individual samples were placed into numbered sacks of 10 samples each along with the appropriate blanks and certified reference materials and stored in a locked warehouse at the camp site until shipped. Samples were transported by El Tigre Silver's personnel to Hermosillo, where they were shipped by a contractor to the assay lab facility. The samples were located on underground maps and generally associated with a surface point by GPS.

Similar to the channel samples, single point rock chip samples were collected from an area of 1 to 2 m in diameter. Multiple chips were collected from different points in the sampling area with a resulting weight from 1 to 3 kg. The chips were bagged and the same protocol applied as with the channel samples. The location was recorded with a handheld Garmin GPS unit.

Six test pits in the tailings impoundment were either hand dug or a vertical channel of an eroded gully were sampled. These six samples were generally short, due to poor access to either higher levels of the gully or limited depth to be hand dug. Samples were collected and placed into a large plastic bag. Samples generally weighed between 10 to 30 kg. Due to the size, they were taken to

the camp storage facility and split with a Gilson splitter down to 5 kg. These smaller sample bags were then placed into a large transport bag along with the other tailing samples derived from the long channel sampling effort.

Twenty-two channel samples were collected down the flanks of the tailings pile. A total of 37 samples were collected. The channel sampling consisted of digging a 10 cm wide by 10 cm deep channel from the top of the pile to the bottom. The sample interval was applied to the coloured layer that the channel exposed. Essentially, most of the channel samples crossed the three-colour layers, such that each layer could be analyzed separately. The samples were taken to the camp storage facility and split down with a Gilson splitter to approximately 5 kg if they were too large. When split, they were placed into a transport bag for delivery to the ALS sample prep lab in Hermosillo.

6.3.3.2 2010 Exploration Program

El Tigre Silver completed a detailed, alteration and mineralization mapping program focused on the Gold Hill and Johnny Crosscut areas from July through September 2010. Gold Hill area is an area of iron-stained, pyritized and weakly silicified wall rocks along the El Tigre Vein that extends 1 km along strike and is up 0.7 km in exposed thickness. A high density of surface pits and shallow workings on Gold Hill, six assays ranging from 3.4 to 34.2 g/t Au in the Johnny Crosscut Mine, and anomalous 0.315 to 0.412 g/t Au in 80 m to 118 m intervals in historical Anaconda drill core holes T-2 and T-3 were combined to support a strong gold target in the Gold Hill area.

Rock chip samples were collected of individual altered and mineralized zones. Lithology, alteration, and mineralization were noted on maps and sample cards for geochemical analysis. Approximately 170 rock chip samples were collected in July 2010 and sent to Skyline Laboratory in Tucson, Arizona. Reconnaissance sampling tested numerous targets over a 1.5 km x 2.0 km area covering Mula Mountain and Gold Hill. Four smaller targets within the Gold Hill area were identified from the 2010 exploration program:

1) Johnny Crosscut Mine;
2) Gold Hill – El Tigre Vein hanging and footwall mineralization;
3) Porvenir Crosscut - Tabular and El Tigre Formation contact on the southern end of Gold Hill; and
4) Mula Mountain dome (Tabular Formation) just west of the Browns Shaft area.

Johnny Crosscut

Mine records and level maps from the Lucky Tiger Combination Gold Mining Company show that the southern half of the El Tigre Vein contains four mined-out historical mineralized shoots with a reported 1 oz/t average gold grade. The four areas extend over 550 m through Mula Mountain from Brown's shaft on the northern side to the Johnny Crosscut on the south. Seven surface samples were collected in the Johnny Crosscut area during the 2010 sampling program. Both the hanging wall and footwall of the El Tigre Vein were sampled, with grades ranging from 0.144 to 1.465 g/t Au and 3 to 74 g/t Ag. The samples were not representative of the thickness or average grades encountered at specific sample locations. This mineralization has been postulated to be the

northern extension of the intercepts in the historical Anaconda holes T-2 and T-3, 400 m to the south.

Gold Hill

Surface overburden cover obscures the outcrop of the projected mineralization of historical Anaconda drill holes T-2 and T-3. However, three samples collected from the limited surface rock outcrops in the hanging wall of the El Tigre Vein yielded assays from 0.169 to 0.284 g/t Au and 0.7 to 163 g/t Ag. The samples were not representative of the thickness or average grades encountered at specific sample locations.

6.3.3.3 2011 Exploration Program

Field work in 2011 continued to target the hanging wall alteration zone of the El Tigre Vein structure at Gold Hill. A total of 215 rock chip samples were collected along this prospective zone and returned assays grading from 0.01 to 3.50 g/t Au and 1.0 to 412 g/t Ag. The samples were not representative of the thickness or average grades encountered at specific sample locations. Additionally, a large outcrop of stockwork fractured and brecciated rock in the El Tigre Vein hanging wall tuffs was sampled. Assay results from this surface zone are listed in Table 6.2.

TABLE 6.2
SELECTED GOLD HILL ROCK CHIP SAMPLING ASSAY RESULTS

Sample ID	Surface Width (m)	Au (g/t)	Ag (g/t)
ET – 286	5	0.32	1.6
ET – 287	5	0.59	1.6
ET – 288	5	0.69	4.5
ET – 289	5	1.37	20.3
ET – 290	5	0.45	9.0
ET – 291	5	0.97	10.6
ET – 292	5	1.66	10.9

Source: *Black and Choquette (2013)*

In the summer of 2011, El Tigre Silver collected 43 channel and pit samples. The sample material consisted of very fine crushed rock with the consistency of coarse flour. Channel sampling was completed by excavating a total of 410 m down the sides of the tailings impoundment at 25 m spacing with 1.5 m intervals from the top of the impoundment to the base. Sample intervals were broken at material colour changes that correspond to different levels of oxidation from the original mined material. These layers, from the bottom to the top, were red (fully oxidized, mined first), grey (partially oxidized), and yellow (sulphide, mined last). Silver values ranged from 54 to 157 g/t Ag and gold ranged from 0.164 to 0.988 g/t Au. The average gold and silver grades are listed in Table 6.3.

TABLE 6.3 AVERAGE SAMPLE GRADES OF THE TAILINGS SAMPLING PROGRAM			
Metal	**Average**	**Maximum**	**Minimum**
Copper (ppm)	100	809	2
Lead (ppm)	2,780	24,300	97
Zinc (ppm)	1,620	9,920	192
Antimony (ppm)	275	1,010	22
Arsenic (ppm)	58	121	27
Cadmium (ppm)	15	103	1.6
Molybdenum (ppm)	5	17	1
Iron (%)	1.52	2.87	0.77
Manganese (ppm)	250	786	59
Sulphur (%)	0.58	3.00	0.18

Source: *Black and Choquette (2013)*

6.3.3.4 2012 Exploration Program

The 2012 exploration program was again focused on the 1 km long zone centred on south Mula Mountain to Gold Hill. New sampling was done on several waste dumps to determine their average gold and silver grades. These dumps included the Level 7 main haulage dump, a second Level 7 waste dump past the old process plant area, the level four waste dump, and Tigre Viejo Canyon waste dump. Some of the sampling also included rock-sawed channel samples that were collected within the Johnny Crosscut Mine, along some road cuts, and the level four portal area. The total number of samples collected in 2012 was 645 samples. These samples were sent to Inspectorate laboratory in Hermosillo, Mexico for sample preparation and Reno, Nevada for gold and silver assays.

The Level 7 dump was first constructed with material from a portal designed to intersect the El Tigre Vein and was subsequently used as the main haulage level to transport mineralization from the Gold Hill area to the process plant. Waste material was discarded near the portal entrance on the flanks of the canyon. The Level 7 dump contains considerable vein material that was mined, and either too low grade to go to the process plant or was mixed with waste material. The Level 7 dump measured 145 m long and 45 m wide across the top and down to the creek drainage.

Rock chip sampling focused on the better exposed material on the top edge and down the flanks of the dump. The sampling program of the Level 7 dump was performed over the top and front face of the dump and spaced every 10 to 20 m. Each of the 44 samples consisted of 10 kg of representative material at each site. The average silver assay of the 44 samples was 230 g/t (6.7 oz/ton) Ag and ranged from 16.3 to 937 g/t Ag. Gold grades averaged 0.89 g/t (0.026 oz/ton) Au within a range of 0.064 to 5.30 g/t Au. The samples were not representative of the thickness or average grades encountered at specific sample locations. Precious metal contents of the 44 samples assayed in the rock chip sampling program are listed by colour in Table 6.4. The trace element contents of the historical channel sampling program are given in Table 6.5.

TABLE 6.4 AVERAGE GRADES OF HISTORICAL TAILINGS IMPOUNDMENT CHANNEL SAMPLE PROGRAM			
Metal	**Red Layer**	**Grey Layer**	**Yellow Layer**
Ag (g/t)	13.6	81.0	87.7
Au (g/t)	0.425	0.336	0.264

Source: *Black and Choquette (2013)*

TABLE 6.5 AVERAGE SAMPLE GRADES OF HISTORICAL TAILINGS IMPOUNDMENT CHANNEL SAMPLING PROGRAM			
Metal	**Average**	**Maximum**	**Minimum**
Copper (ppm)	185	370	100
Lead (ppm)	1,190	2,590	720
Zinc (ppm)	1,380	4,860	260
Antimony (ppm)	115	220	70
Arsenic (ppm)	58	100	-50
Cadmium (ppm)	17	50	-10
Molybdenum (ppm)	32	50	-10
Iron (%)	8,100	14,700	5,200
Manganese (ppm)	317	1,120	60
Sulphur (%)	3,300	6,800	1,600

Source: *Black and Choquette (2013)*

6.3.4 El Tigre Silver Drilling Programs 2011 to 2013

6.3.4.1 El Tigre 2011 Drilling

Based on the results of the 2010 exploration program, El Tigre Silver completed 10 drill holes on four targets along the southern projection of the El Tigre Vein system in early 2011 (Figure 6.8). Mineralization in the veins was projected at least 300 m vertically below the surface to Level 7 of the El Tigre Mine.

The results confirmed that the El Tigre Vein, along with other intercepted veins and stockwork zones, continued both down-dip and along strike of the overall mineralized system. The individual drill targets were as follows:

- **Espuelas Canyon:** Disseminated argentite-galena mineralization encountered on the surface and in the Level four cross-cut;

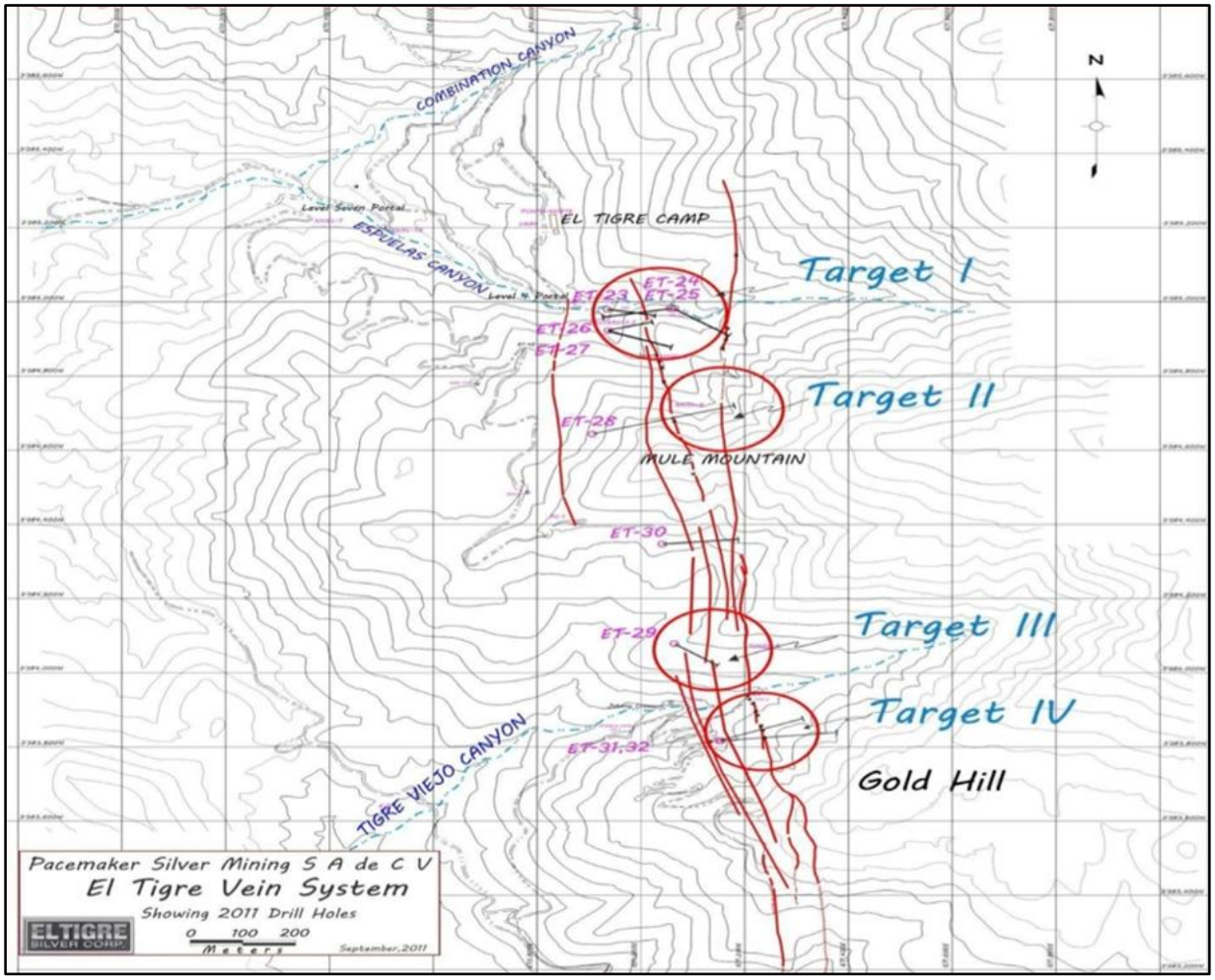
- **Mule Mountain:** Quartz-sericite-pyrite-galena stockwork, veins and veinlets in the hanging wall of the Seitz Kelly Vein on Level 7;

- **Tigre Viejo Canyon**: Quartz-sericite-pyrite-galena stockwork, veins and veinlets near the intersection of the El Tigre, Seitz Kelly, and Sooy Veins on Level 7; and

- **Gold Hill**: Disseminated and quartz veinlet-controlled gold mineralization outcropping in the Tigre Viejo Canyon and mined in stopes on Gold Hill.

The 2011 drilling was completed by Major Drilling de Mexico SA de CV of Hermosillo, Sonora, from January 27, 2011 to March 7, 2011. The drilling program consisted of 11 HQ diameter core drill holes totalling 2,313 m. Select significant intercepts from the 2011 drilling program, >1 m true thickness and >100 g/t Ag or 1.50 g/t Au, are presented in Table 6.6.

FIGURE 6.8 2011 DRILL HOLE LOCATION MAP



Source: *Black and Choquette (2013).*

TABLE 6.6 EL TIGRE SILVER 2011 DRILLING SIGNIFICANT INTERCEPTS						
Drill Hole ID	**From (m)**	**To (m)**	**Length (m)**	**True Thickness (m)**	**Ag (g/t)**	**Au (g/t)**
ET-23	142.00	144.00	2.00	1.28	121	0.580
ET-24	69.00	78.25	9.25	5.92	136	0.220
	124.00	128.00	4.00	2.56	108	0.120
ET-25	40.00	50.00	10.00	5.70	188	0.350
	43.00	45.00	2.00	1.14	713	0.680
ET-26	86.70	88.45	1.75	1.13	133	0.365
	90.15	91.90	1.75	1.13	142	0.230
	102.60	120.10	15.50	11.69	180	0.217
ET-27	107.00	111.00	4.00	2.80	242	3.170
ET-28	221.60	226.00	4.40	3.11	165	0.702
ET-31	85.00	102.50	17.5	11.24	129	0.861
	104.00	111.00	7.00	4.49	295	0.274
ET-32	104.00	108.50	4.50	2.88	146	0.076

Source: *P&E (2023)*

6.3.4.2 El Tigre 2012 Drilling

The 2012 El Tigre Silver drilling program was focused entirely on Gold Hill, in order to follow-up on the wide lower-grade gold intercepts in drill holes ET-31 and ET-32 and the historical drill holes T-2 and T-3 (see Table 6.1). Land Drill International Mexico, SA de CV of Hermosillo, Sonora, Mexico was commissioned to complete the HQ diameter diamond core drill holes. The drill program began in March and concluded in mid-May 2012. The drilling program consisted of 10 HQ diameter core drill holes totalling 2,236 m. Select significant intercepts from the 2012 drilling program >1.0 m true thickness and >100 g/t Ag or 1.50 g/t Au are presented in Table 6.7.

TABLE 6.7 EL TIGRE SILVER 2012 DRILLING SIGNIFICANT INTERCEPTS						
Drill Hole ID	**From (m)**	**To (m)**	**Length (m)**	**True Width (m)**	**Ag (g/t)**	**Au (g/t)**
ET-33	9.0	11.5	2.5	1.6	135	*
	67.9	85.0	17.1	11.0	1,697	1.11
ET-34	111.4	113.0	1.6	1.0	152	6.82
ET-37	175.5	185.5	10.0	1.7	336	0.22
includes	176.42	177.00	0.58	0.10	2,014	1.15
includes	183	184	1	0.2	506	0.22
ET-38	46.4	51.0	4.6	2.9	*	6.38
includes	48.0	50.0	2.0	1.3	*	5.76

TABLE 6.7 EL TIGRE SILVER 2012 DRILLING SIGNIFICANT INTERCEPTS						
Drill Hole ID	**From (m)**	**To (m)**	**Length (m)**	**True Width (m)**	**Ag (g/t)**	**Au (g/t)**
ET-39	116.8	118.4	1.6	1.0	*	1.55
ET-42	107.0	108.9	1.9	1.2	134	1.00

Source: *P&E (2023)*
Note: ** Below Detection Limit*

6.3.4.3 El Tigre 2013 Exploration Drilling

An expanded drilling program was undertaken over the northern portion of Gold Hill, with most of the effort expended on the southern flank of Mule Mountain, both north and south of the Brown Shaft. The last drill hole of the 2013 campaign was completed near the summit of Mula Mountain, overlooking the Camp area. The 2013 drilling program was completed by Major Drilling de Mexico SA de CV of Hermosillo, Sonora, from January to April 2013.

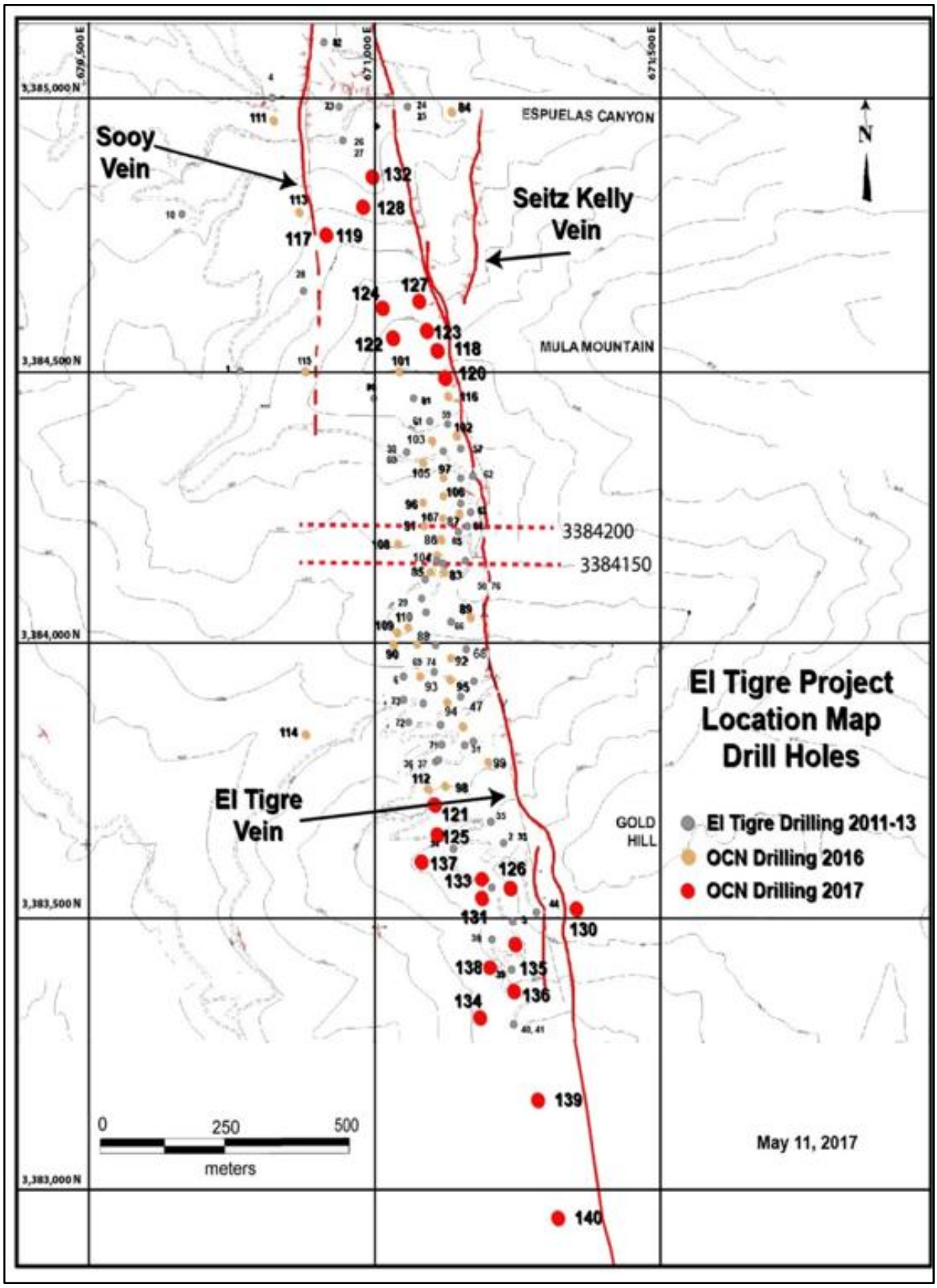
The drilling program consisted of 38 HQ diameter core drill holes totalling 4,862 m. Select significant intercepts from the 2013 drilling program, >1 m true width and >100 g/t Ag or 1.50 g/t Au, and other select high-grade intersections, are presented in Table 6.8 and drill hole locations are presented in Figure 6.9.

TABLE 6.8 EL TIGRE SILVER 2013 DRILLING SIGNIFICANT INTERCEPTS						
Drill Hole ID	**From (m)**	**To (m)**	**Length (m)**	**True Thickness (m)**	**Ag (g/t)**	**Au (g/t)**
ET13-45	54.25	55.75	1.50	1.04	*	4.59
and	60.05	61.50	1.45	1.00	*	1.58
ET13-46	27.00	30.00	3.00	2.12	11.75	3.28
includes	28.50	30.00	1.50	1.06	12.30	5.86
and	56.50	58.00	1.50	1.06	*	1.51
and	71.50	74.50	3.00	2.12	117.70	0.27
ET13-48	116.00	117.50	1.50	1.07	106.50	0.14
ET13-49	3.00	22.50	19.50	13.55	29.52	1.21
includes	4.50	13.67	9.17	6.37	58.77	1.06
includes	15.15	16.50	1.35	0.94	4.90	2.82
and	33.00	35.00	2.00	1.39	4.00	2.49
ET13-50	60.60	68.50	7.90	5.63	103.34	1.91
includes	60.60	63.10	2.90	2.07	220.40	5.55
ET13-51	48.50	73.00	25.50	10.79	20.90	2.93
includes	55.50	60.50	5.00	2.11	38.10	5.58

TABLE 6.8 EL TIGRE SILVER 2013 DRILLING SIGNIFICANT INTERCEPTS						
Drill Hole ID	**From (m)**	**To (m)**	**Length (m)**	**True Thickness (m)**	**Ag (g/t)**	**Au (g/t)**
ET13-53	64.05	66.70	2.65	1.29	9.54	1.79
and	71.60	81.60	10.00	4.85	109.55	1.26
includes	75.50	80.15	4.65	2.25	427.00	2.42
ET13-55	41.50	46.35	4.85	2.05	201.82	0.76
ET13-56	35.00	51.30	16.30	10.04	129.79	0.61
and	63.00	69.40	6.40	3.94	231.20	0.54
ET13-58	54.05	59.00	4.95	3.18	204.70	2.02
and	69.50	72.50	3.00	1.93	35.80	2.77
and	117.70	122.15	4.45	2.86	107.40	0.24
ET13-60	180.30	190.95	10.65	7.14	187.50	0.60
includes	181.30	185.10	3.80	2.55	519.90	0.81
ET13-62	0.00	7.00	7.00	4.99	164.90	0.20
ET13-64	13.00	18.70	5.70	3.89	60.70	1.89
ET13-66	91.60	93.00	1.40	0.78	2,057.50	74.19
includes	92.10	92.50	0.40	0.22	3,030.00	235.00
includes	92.50	93.00	0.50	0.28	2,920.00	5.34
ET13-67	19.00	45.00	26.00	17.89	6.70	1.65
and	81.00	83.50	2.50	1.72	235.60	0.45
and	108.25	110.20	1.95	1.34	133.30	0.23
ET13-69	92.00	102.00	10.00	7.40	157.20	0.17
includes	98.50	100.65	2.15	1.59	571.00	0.45
ET13-71	101.25	104.65	3.40	2.14	270.70	0.22
includes	103.30	104.65	1.35	0.85	590.40	0.47
includes	103.60	103.90	0.30	0.19	1,940.00	1.88
and	120.00	137.30	17.30	10.88	109.50	*
includes	124.95	125.40	0.45	0.28	291.00	0.86
includes	129.90	130.50	0.60	0.38	1,320.00	0.17
ET13-72	72.50	74.50	2.00	1.38	*	6.03
ET13-75	65.50	68.50	3.00	1.74	7.93	2.05
and	137.65	138.65	1.00	0.58	767.45	3.95
and	137.65	138.10	0.45	0.26	1,340.00	7.67
ET13-76	100.70	114.35	13.65	4.32	504.77	1.05
includes	102.85	104.25	1.40	0.44	4,143.21	5.41
includes	103.80	104.25	0.45	0.14	8,660.00	6.97
and	112.15	112.85	0.70	0.22	1,122.14	1.50
ET13-81	79.60	82.80	3.20	2.32	193.80	2.90

Source: *P&E (2023)* ***Note:*** **Below Detection Limit*

FIGURE 6.9 UPDATED DRILL HOLE LOCATION MAP



Source: *P&E (2023)*

6.3.4.4 Tailings Impoundment Drilling 2011 and 2013

As part of El Tigre Silver's 2011 exploration program, a full examination of the tailings impoundment was completed. El Tigre Silver drilled hollow stem auger holes to obtain samples in the thicker area of the impoundment. The tailings consist of three colour types, which from bottom to top are red oxide, grey mixed oxide-sulphide, and yellow sulphide. The drilling program was designed to obtain sufficient samples from each of the three colour types to be composited for metallurgical testwork.

At the end of November 2011, a 46-hole hollow stem auger drilling program was completed on the tailings impoundment. The drill holes were completed with a 30 cm diameter auger bit and averaged ~7.5 m in depth.

Assays from the auger drilling program (silver, gold and other minor elements) were completed by ALS Worldwide Labs, North Vancouver, Canada. This program produced 212 samples from the 46 auger drill holes totalling 315 m. Due to the homogenous nature of the tailings impoundment, the results from drill hole to drill hole were similar. The average grades from the 212 samples submitted for analysis are presented in Table 6.9.

TABLE 6.9
AUGER HOLE DRILL PROGRAM ANALYTICAL SUMMARY

Element	Red Layer	Grey Layer	Yellow Layer
Ag (g/t)	1,046	81	88
Au (g/t)	0.43	0.34	0.26

Source: *P&E (2023)*

At the completion of the 2013 in-situ vein area drilling, El Tigre Silver moved the diamond drill core rig down to the tailings impoundment to obtain additional samples for assaying and to intercept the contact of the tailings to the underlying soil horizon. This work was done to assist in building a more reliable volume calculation for the Mineral Resource model of the tailings impoundment. Seven HQ diameter diamond core drill holes totalling 132 m were completed.

Three auger drill holes were also completed into the Level 7 dump in 2013, in a line 15 m apart along the top edge. The drill holes were 1.5 m, 3.0 m and 4.5 m deep, with each sample collected representing a 1.5 m interval. The average silver assay of the nine auger samples was 259 g/t (7.5 oz/t) Ag within a range of 124 to 465 g/t Ag. Gold averaged 0.71 g/t (0.021 oz/t) Au within a range of 0.26 to 1.26 g/t Au.

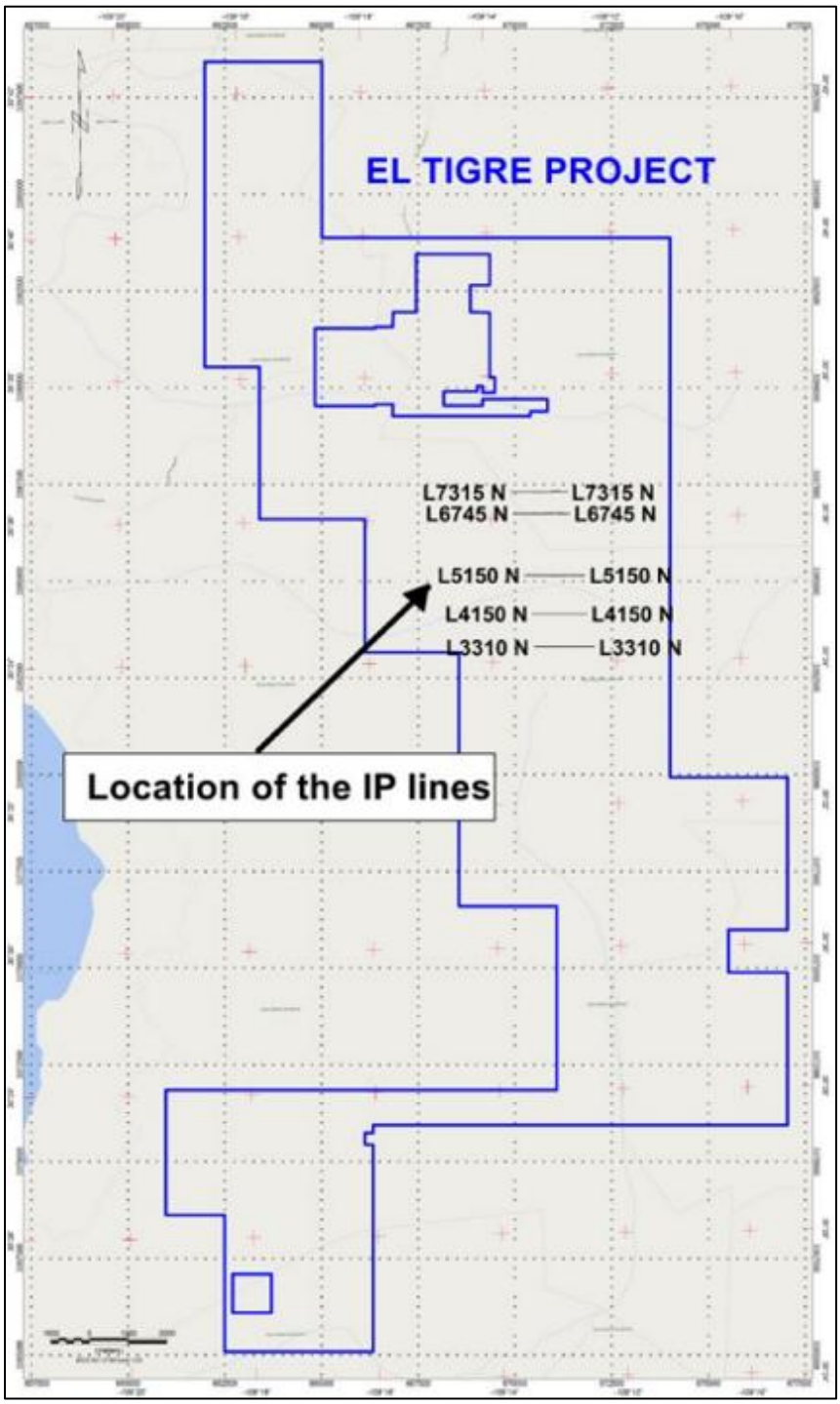
6.3.5 Oceanus Exploration 2016 to 2019

6.3.5.1 2016 IP Geophysical Survey Program

In December 2016, Oceanus retained Geofisica TMC to complete an orientation IP survey at El Tigre. A total of 7.4-line km of pole: dipole survey was completed on five lines crossing the vein, stockwork and fracture system. The surveys were completed in January 2017. Lines 7315N and

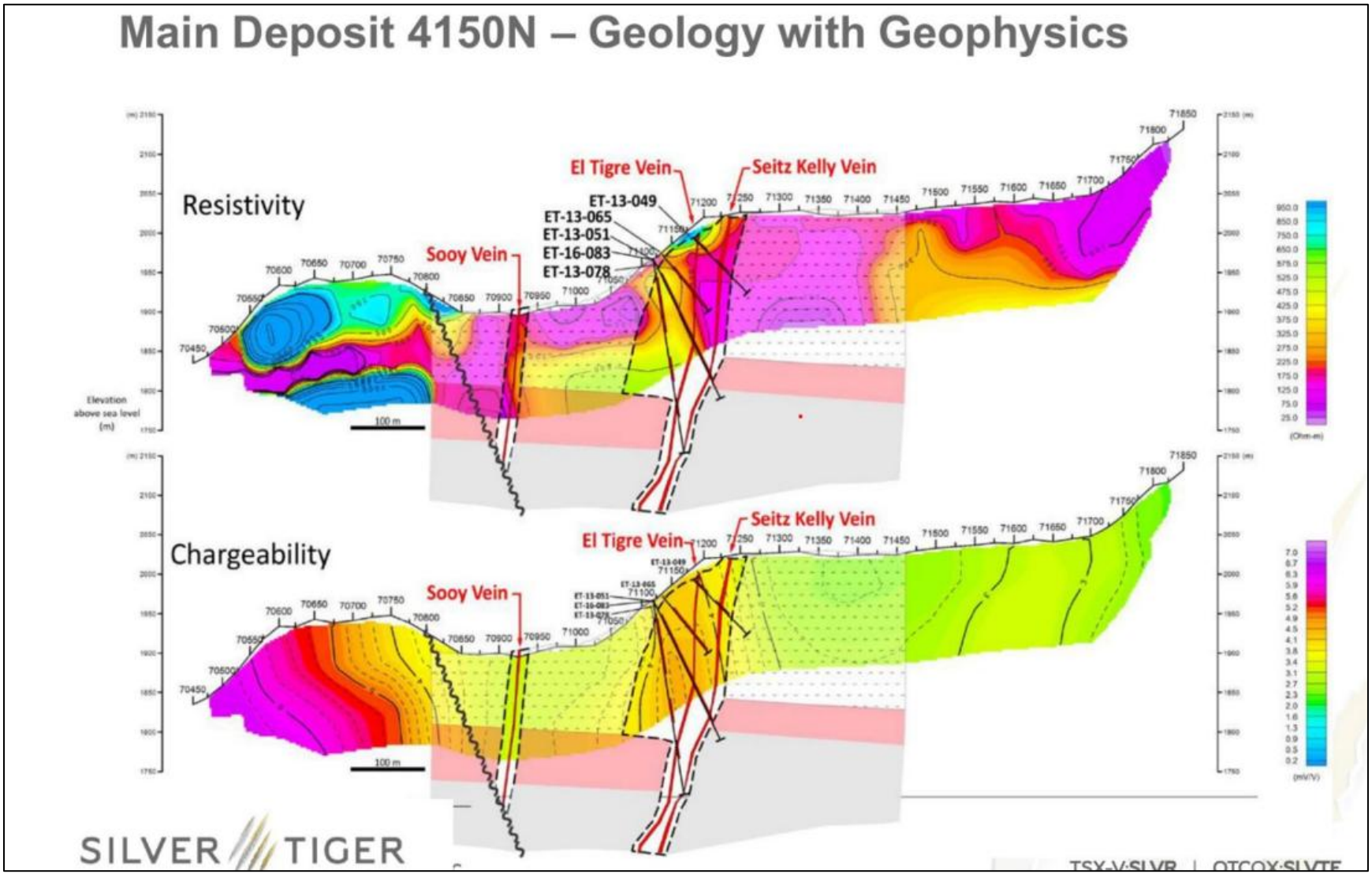
6745N tested the Fundadora and Protectora Veins located several km to the north of the El Tigre Mine, whereas Lines 5150N, 4150N and 3310N tested the Camp, Mula Mountain and Gold Hill Zones, respectively. All five surveyed lines showed chargeability highs and resistivity lows associated with the vein and stockwork/fracture zones (Figure 6.10 and Figure 6.11).

FIGURE 6.10 LOCATION OF 2016 IP SURVEY LINES



Source: *Simard (2017)*

FIGURE 6.11 IP CROSS-SECTION PROJECTION L4150N INTERPRETATION



Source: *Silver Tiger website (2023)*

6.3.5.2 2017 Underground Channel Program

Oceanus mapped and sampled the historical workings north of the El Tigre Mine during the summer 2017 exploration program. The goals of that work program were two-fold:

1) Map and identify the styles of mineralization and alteration exposed in the historical workings; and

2) Collect a suite of channel samples to document the gold, silver and base metal grades.

This work combined with field mapping would facilitate selecting drill targets for the fall 2017 drilling program. A total of 990 channel samples were collected.

Channel samples were collected from Level 9 underground exploration drift over a 2.0 km strike length of the Protectora Vein between cross-section 5650 and cross-section 7600 North and from additional exploration drifts on the Fundadora and Caleigh Veins. None of these veins have been mined. The first exploration drift on the Protectora Vein is located 650 m north along strike from the northern end of the historical El Tigre Mine workings.

After surveying the drifts, geologists mapped the quartz veins, and then collected channel samples across the back (roof) of the drifts. The majority of the mine openings are between 1 m and 2 m wide, and therefore the channel widths are limited to this approximate length.

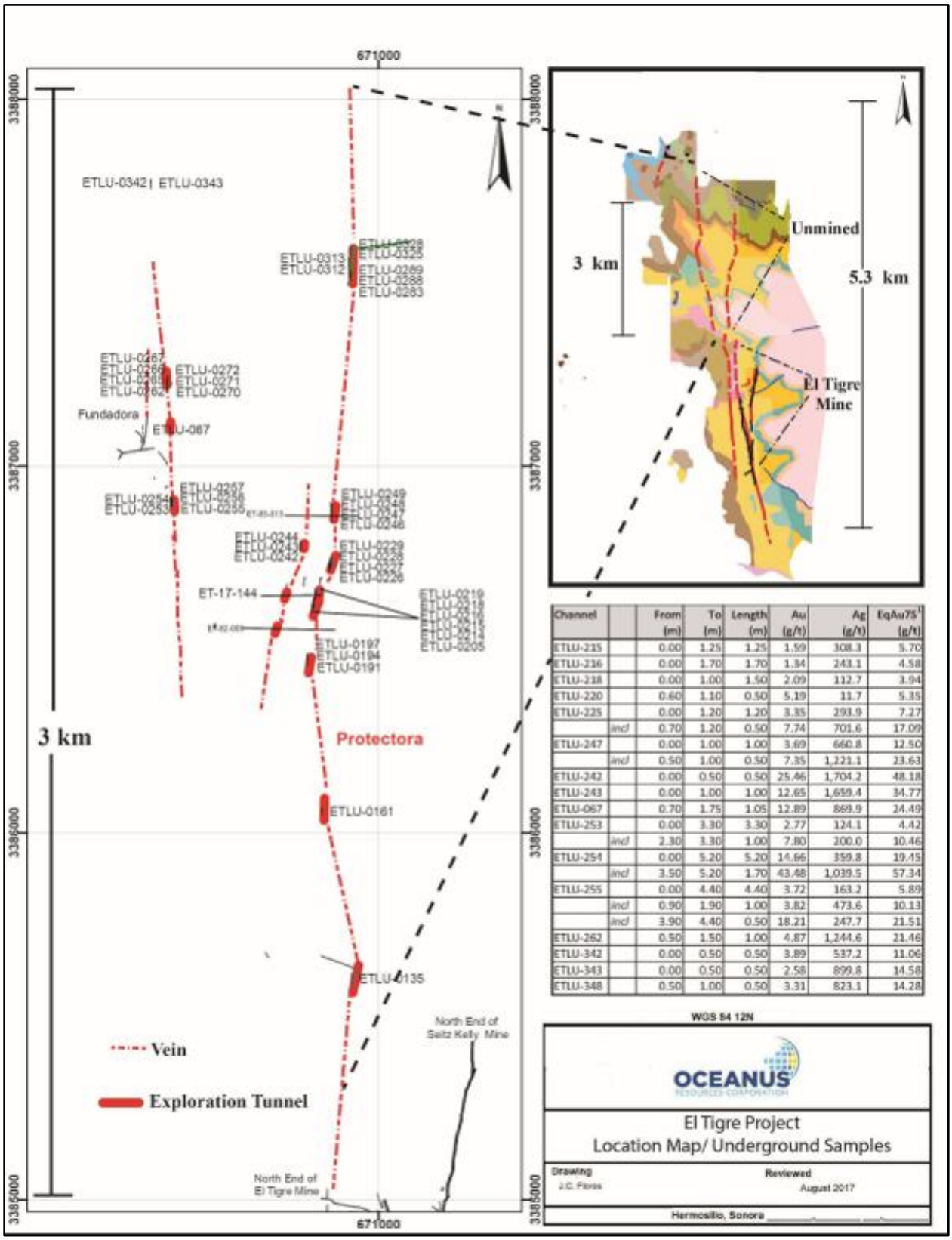
Geologists also collected samples of the hanging wall alteration zone, the quartz vein material, and the footwall alteration zones. The reported result is the weighted average grade across the width of the mine opening. The high-grade silver values are related to the quartz vein material. The quartz veins and alteration zones exposed in these exploration workings resemble those found in the historical El Tigre Mine workings.

A channel sample location map is shown in Figure 6.12 and significant channel sample assay results are listed in Table 6.10.

6.3.5.3 2018 Exploration Program

The 2018 exploration consisted of a three-phase prospecting and mapping program. The phase one prospecting and mapping program was completed to the south of Gold Hill and demonstrated that the favourable El Tigre Formation continues along strike to the southwest for an additional 5 km to the Lluvia de Oro Prospect. The geology, vein locations and the geochemical anomalies representing samples with a grade >0.75 g/t AuEq are shown in Figure 6.13.

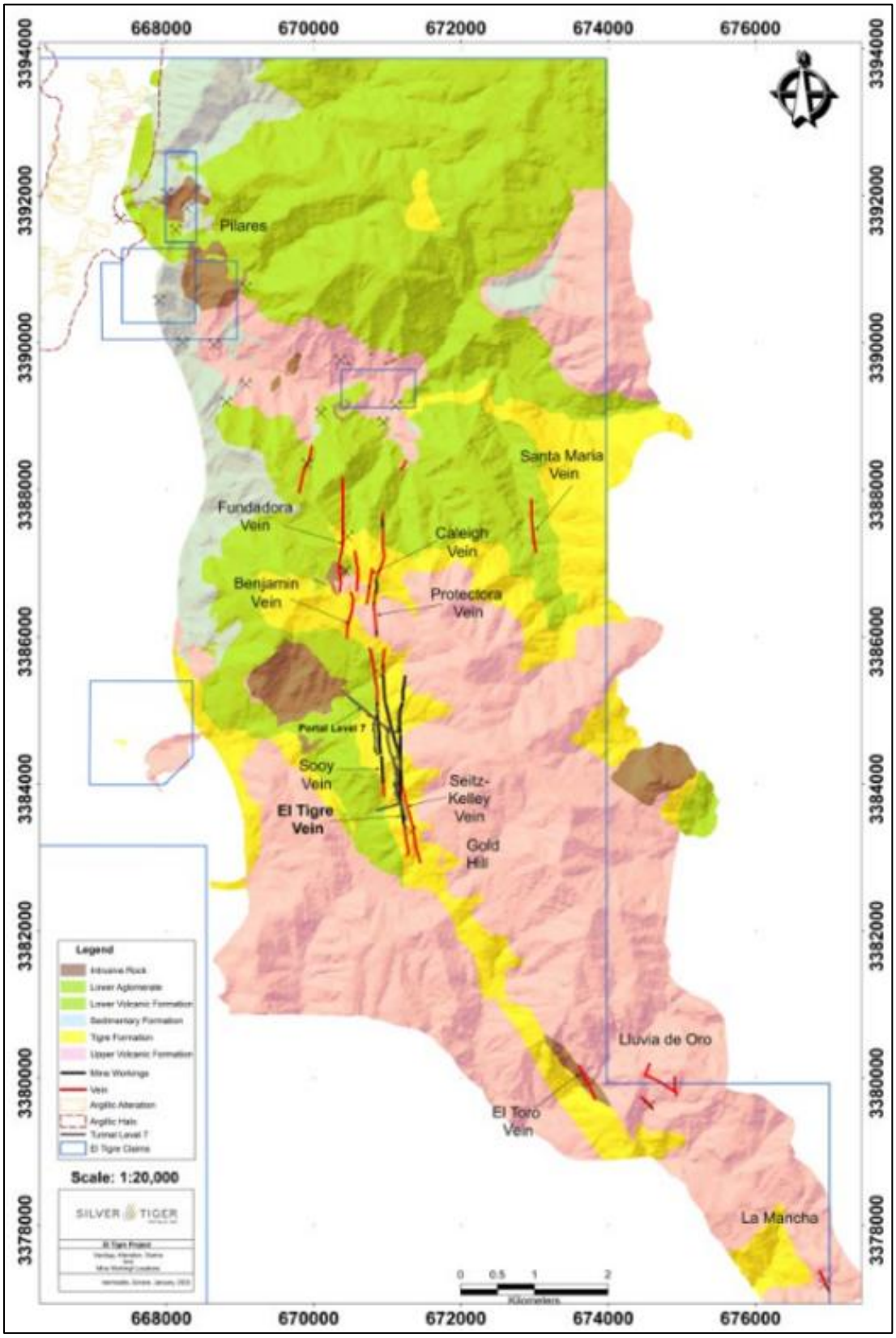
FIGURE 6.12 UNDERGROUND EXPLORATION DRIFT LOCATION MAP



Channel		From (m)	To (m)	Length (m)	Au (g/t)	Ag (g/t)	EqAu75[1] (g/t)
ETLU-215		0.00	1.25	1.25	1.59	308.3	5.70
ETLU-216		0.00	1.70	1.70	1.34	243.1	4.58
ETLU-218		0.00	1.00	1.50	2.09	112.7	3.94
ETLU-220		0.60	1.10	0.50	5.19	11.7	5.35
ETLU-225		0.00	1.20	1.20	3.35	293.9	7.27
	incl	0.70	1.20	0.50	7.74	701.6	17.09
ETLU-247		0.00	1.00	1.00	3.69	660.8	12.50
	incl	0.50	1.00	0.50	7.35	1,221.1	23.63
ETLU-242		0.00	0.50	0.50	25.46	1,704.2	48.18
ETLU-243		0.00	1.00	1.00	12.65	1,659.4	34.77
ETLU-067		0.70	1.75	1.05	12.89	869.9	24.49
ETLU-253		0.00	3.30	3.30	2.77	124.1	4.42
	incl	2.30	3.30	1.00	7.80	200.0	10.46
ETLU-254		0.00	5.20	5.20	14.66	359.8	19.45
	incl	3.50	5.20	1.70	43.48	1,039.5	57.34
ETLU-255		0.00	4.40	4.40	3.72	163.2	5.89
	incl	0.90	1.90	1.00	3.82	473.6	10.13
	incl	3.90	4.40	0.50	18.21	247.7	21.51
ETLU-262		0.50	1.50	1.00	4.87	1,244.6	21.46
ETLU-342		0.00	0.50	0.50	3.89	537.2	11.06
ETLU-343		0.00	0.50	0.50	2.58	899.8	14.58
ETLU-348		0.50	1.00	0.50	3.31	823.1	14.28

Source: *Oceanus (September 2017)*

FIGURE 6.13 DISTRIBUTION OF THE PROSPECTIVE EL TIGRE FORMATION



Source: *Oceanus (March 5, 2018)*

TABLE 6.10 2017 UNDERGROUND CHANNEL SAMPLE ASSAYS								
Vein	**Channel**	**Comment**	**From (m)**	**To (m)**	**Length (m)**	**Au (g/t)**	**Ag (g/t)**	**AuEq (g/t)***
Protectora	ETLU – 215		0.0	1.3	1.3	1.59	308	5.70
Protectora	ETLU – 216		0.0	1.7	1.7	1.34	243	4.58
Protectora	ETLU – 218		0.0	1.5	1.5	2.09	113	3.94
Protectora	ETLU – 220		0.6	1.1	0.5	5.19	12	5.35
Protectora	ETLU – 225		0.0	1.2	1.2	3.35	294	7.27
		including	0.7	1.2	0.5	7.74	702	17.09
Protectora	ETLU – 247		0.0	1.0	1.0	3.69	661	12.50
		including	0.5	1.0	0.5	7.35	1,221	23.63
Caleigh	ETLU – 242		0.0	0.5	0.5	25.46	1,704	48.18
Caleigh	ETLU – 243		0.0	1.0	1.0	12.65	1,659	34.77
Fundora	ETLU – 067		0.7	1.8	1.1	12.89	870	24.49
Fundora	ETLU – 253		0.0	3.3	3.3	2.77	124	4.42
		including	2.3	3.3	1.0	7.80	200	10.46
Fundora	ETLU – 254		0.0	5.2	5.2	14.66	3,608	19.45
		including	3.5	5.2	1.7	43.48	1,040	57.34
Fundora	ETLU – 255		0.0	4.4	4.4	3.72	163	5.89
		including	0.9	1.9	1.0	3.82	474	10.13
		including	3.9	4.4	0.5	18.21	248	21.51
Fundora	ETLU – 262		0.5	1.5	1.0	4.87	1,245	21.46
Fundora	ETLU – 342		0.0	0.5	0.5	3.89	537	11.06
Fundora	ETLU – 343		0.0	0.5	0.5	2.58	900	14.58
Fundora	ETLU – 348		0.5	1.0	0.5	3.31	823	14.28

Source: *Oceanus press release (September 7, 2018).*
Note: ** Gold Equivalent (AuEq) ratio based on gold to silver price ratio of 75:1 (Au:Ag).*

The Phase two prospecting and mapping program was completed on the eastern side of the mountain. The field team located several historical underground workings in this area (Santa Maria) that followed mineralized quartz veins similar to the historical El Tigre Mine, and outcrops of vein mineralization. As a result of these exploration outcomes, Oceanus decided to acquire additional concessions in this area.

The Phase three prospecting and mapping program was completed on the new concessions and identified outcrops of the El Tigre Formation in several areas.

Oceanus had identified >10 km of favourable host stratigraphy with several areas of mineralization identified to the south, east and northeast of the historical El Tigre Mine. The objective of the 2019 program was to establish targets for drill testing.

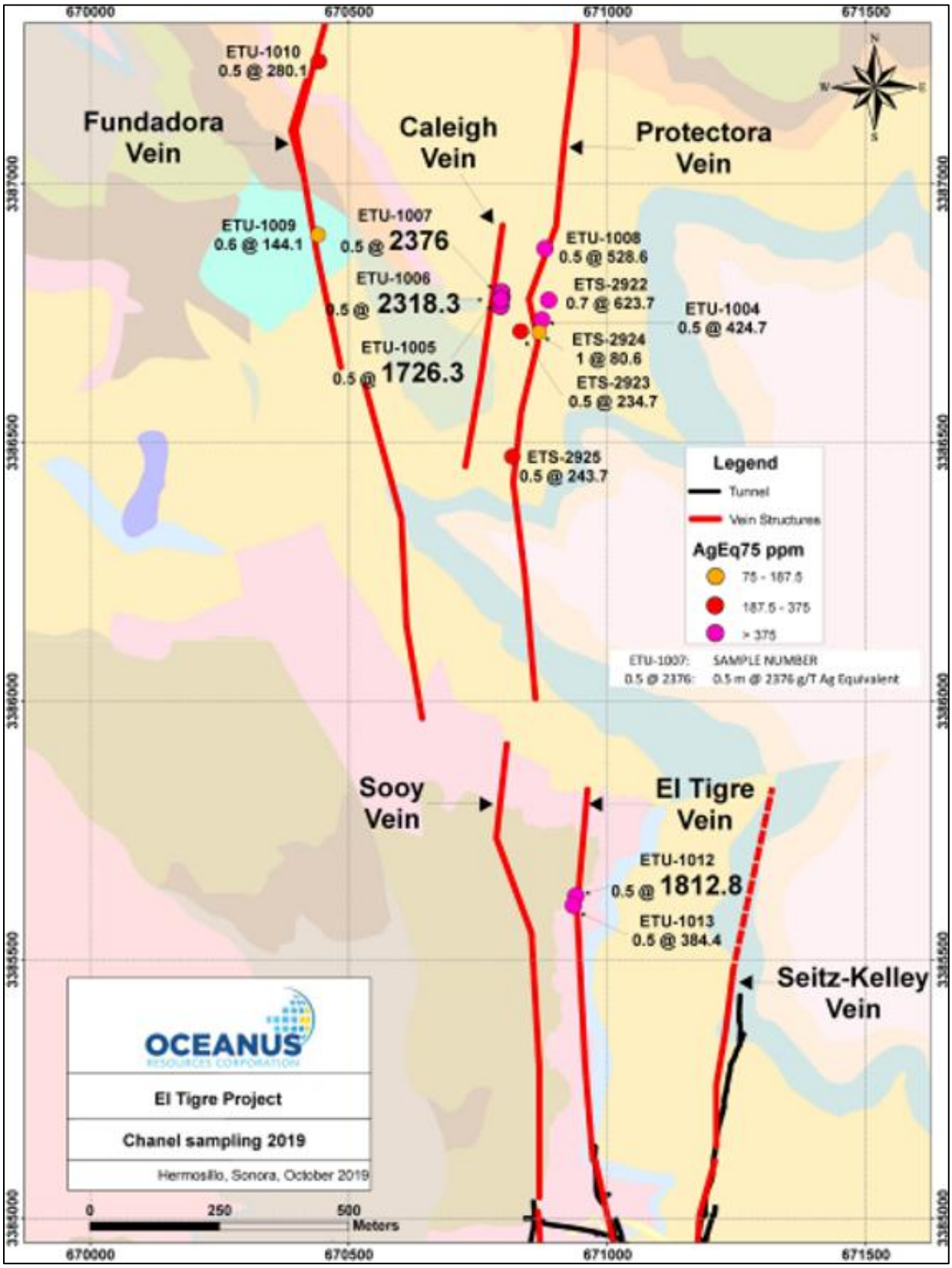
In 2018, Oceanus also received an updated 3-D model of the historical El Tigre Mine underground workings that incorporated assay data from 2,500 underground channel samples collected from the drifts, stopes and raises of the El Tigre, Sooy and Seitz Kelly Veins. While the Company was conducting a review of the historical maps and files of the Anaconda Geological Documents Collection archived at the American Heritage Center at the University of Wyoming in Laramie, Wyoming, historical data were discovered that enabled Oceanus to update its 3-D model of the El Tigre Mine's historical underground workings.

The plans discovered date from 1912 and present the Ag and Au assays of channel samples collected across the working face (typically 3.5 ft) every 5 ft along the drift. The data were digitized, converted to metric units, and geo-referenced for the 3-D Minesight™ model by SPM Mineria ("SPM") of Hermosillo, Mexico. These assay results are not compliant with NI 43-101; however, they are indicative of the gold and silver grades (Mishler, 1926).

6.3.5.4 2019 Underground Channel Sample Program

Encouraging assay results of channel samples collected in September 2019, from historical underground exploration drifts and surface samples on the Caleigh, Canon Combination (unmined portion of the El Tigre Vein), Protectora and Aguila Veins presented a series of high priority drill targets over a 2 km strike length north of the El Tigre Mine. A channel sample location map and assay highlights are presented in Figure 6.14. Significant channel sample assay results are listed in Table 6.11.

FIGURE 6.14 2019 CHANNEL SAMPLE LOCATION AND ASSAYS



Source: *Silver Tiger (November 6, 2019)*

TABLE 6.11 SIGNIFICANT 2019 CHANNEL SAMPLES ASSAY RESULTS								
Vein	**Sample ID***	**Length (m)**	**Easting ****	**Northing ****	**Elevation (masl)**	**Au (g/t)**	**Ag (g/t)**	**AgEq (g/t) *****
Caleigh	ETU – 1005	0.5	670,794	3,386,777	2,023	10.13	967	1,726
Caleigh	ETU – 1006	0.5	670,795	3,386,780	2,023	4.64	1,971	2,318
Caleigh	ETU – 1007	0.5	670,796	3,386,781	2,023	6.40	1,896	2,376
Cañon Combination	ETU – 1012	0.5	670,939	3,385,623	1,833	0.25	1,794	1,813
Cañon Combination	ETU – 1013	0.5	670,934	3,385,605	1,833	0.26	365	384
Protectora	ETU – 1008	0.5	670,879	3,386,875	2,092	2.01	378	529
Protectora	ETU – 1004	0.5	670,873	3,386,737	2,016	1.12	340	425
Aguila	ETU – 1009	0.5	670,440	3,386,901	1,943	0.94	73	144
Aguila	ETU – 1010	0.5	670,441	3,387,237	2,082	0.90	213	280
Level 4	ETU – 1011	0.5	670,862	3,385,002	1,798	0.08	41	48
Protectora	ETS – 2922	0.7	670,886	3,386,775	2,060	3.32	374	624
Protectora	ETS – 2923	0.5	670,832	3,386,715	2,030	0.57	192	235
Protectora	ETS – 2924	1.0	670,867	3,386,713	2,024	0.66	31	81
Protectora	ETS – 2925	0.5	670,816	3,386,473	2,075	1.00	169	244

Source: *Silver Tiger (November 6, 2023)*
Notes: **ETU = underground channel samples; ETS = surface channel samples.*
*** coordinates are in WGS 1984 UTM Zone 12.*
**** Silver Equivalent (AgEq) ratio based on gold to silver price ratio of 75:1 (Au:Ag)*

Selected assay highlights from this underground channel sampling program are listed below:

- In the Caleigh Vein, ~2 km north of the El Tigre Mine, underground channel sample ETU-1007 returned 1,896 g/t Ag and 6.40 g/t Au, or 2,376 g/t silver equivalent ("AgEq"), over 0.50 m (true thickness). Underground channel sample ETU-1006 returned 1,971 g/t Ag and 4.64 g/t Au, or 2,318 g/t AgEq, over 0.50 m (true thickness);

- In the Canon Combination Vein (unmined portion of the El Tigre Vein) underground channel sample ETU-1012 returned 1,794 g/t Ag and 0.25 g/t Au, or 1,813 g/t AgEq, over 0.50 m (true thickness); and

- In the Protectora Vein, ~2 km north of the El Tigre Mine, underground channel sample ETU-1008 returned 378 g/t Ag and 2.01 g/t Au, or 529 g/t AgEq, over 0.50 m (true thickness).

6.3.6 Oceanus Drilling 2016-2017

Oceanus completed the 2016-2017 infill drilling program at El Tigre in May 2017, having completed at least 62 diamond drill holes totalling 11,923 m of HQ size. The purpose of this program was to support an NI 43-101 Mineral Resource estimation for the El Tigre Property.

The initial phase of the drill program consisted of: 1) completing several new drill holes near drill holes ET-13-051 and ET-13-064 to cross the entire width of the mineralized zone and end in barren footwall rock; 2) completing several drill holes to test the extension of the high-grade clavos; and 3) completing a fence of drill holes across the entire mineralized zone, consisting of the Sooy Vein in the hanging wall, the central El Tigre Vein, and the Seitz-Kelly Vein in the footwall. Drill hole locations are shown in Figure 6.15 and listed in Table 6.12. Select significant mineralized intersections are presented in Table 6.13.

Several step-out drill holes completed at the end of the 2016-2017 drill program returned encouraging results 400 m to the south and 800 m to the north of the historical El Tigre Mine. Highlights from the step-out drilling to the south of the historical El Tigre Mine included:

- **Drill hole ET-17-133**: 67.6 m of 1.49 g/t AuEq from 78.5 to 146.1 m (consisting of 1.24 g/t Au and 19.1 g/t Ag), including 23.4 m of 3.31 g/t AuEq (consisting of 2.77 g/t Au and 40.5 g/t Ag);

- **Drill hole ET-17-139**: 5.2 m of 0.98 g/t AuEq from 10.6 to 15.8 m (consisting of 0.96 g/t Au and 1.7 g/t Ag); and

- **Drill hole ET-17-140**: 9.0 m of 1.86 g/t AuEq from 35.0 to 44.0 m (consisting of 0.18 g/t Au and 125.5 g/t Ag), including 1.5 m of 9.54 g/t AuEq (consisting of 0.43 g/t Au and 683.2 g/t Ag), in a step- out drill hole ~400 m to the south of the Mine.

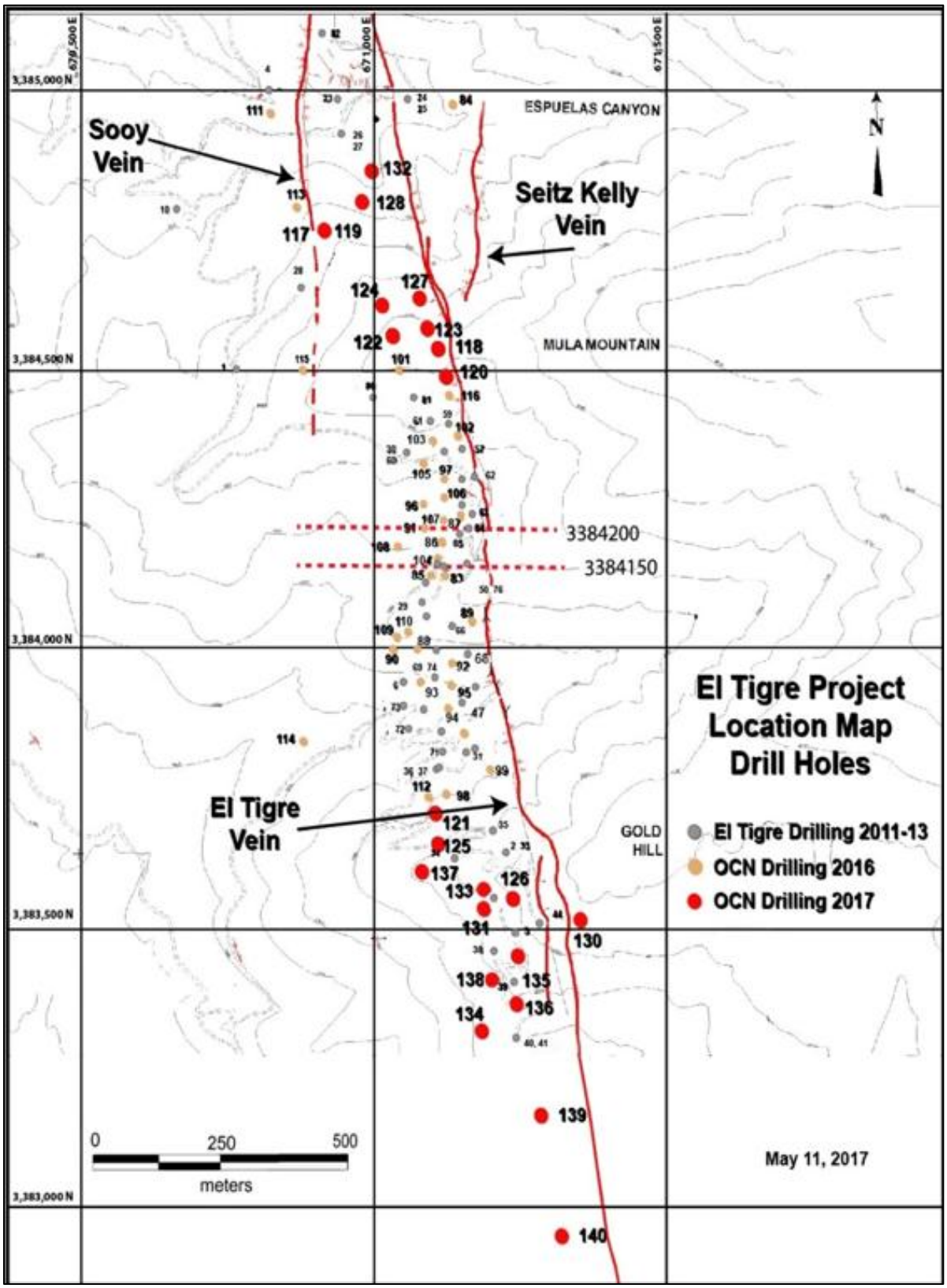
Step-out drill hole ET-17-144 intersected high-grade gold and silver mineralization in the Protectora Vein located 800 m north of the historical El Tigre Mine (Figure 6.16). Highlights from this step-out drilling included:

- **Drill hole ET-17-144**: 3.15 m of 36.6 g/t AuEq from a depth of 88.25 to 91.40 m (consisting of 10.1 g/t Au and 1,990.9 g/t Ag). This intercept included 0.85 m of 135.1 g/t AuEq (consisting of 37.2 g/t Au and 7,338.9 g/t Ag). The 0.85 m intercept also returned 2.84% Cu, 4.06% Zn and 1.38% Pb; and

- **Drill hole ET-17-144**: also returned 1,107.36 g/t Ag and 0.024 g/t Au over 1.5 m from a depth of 188.65 to 190.15 m.

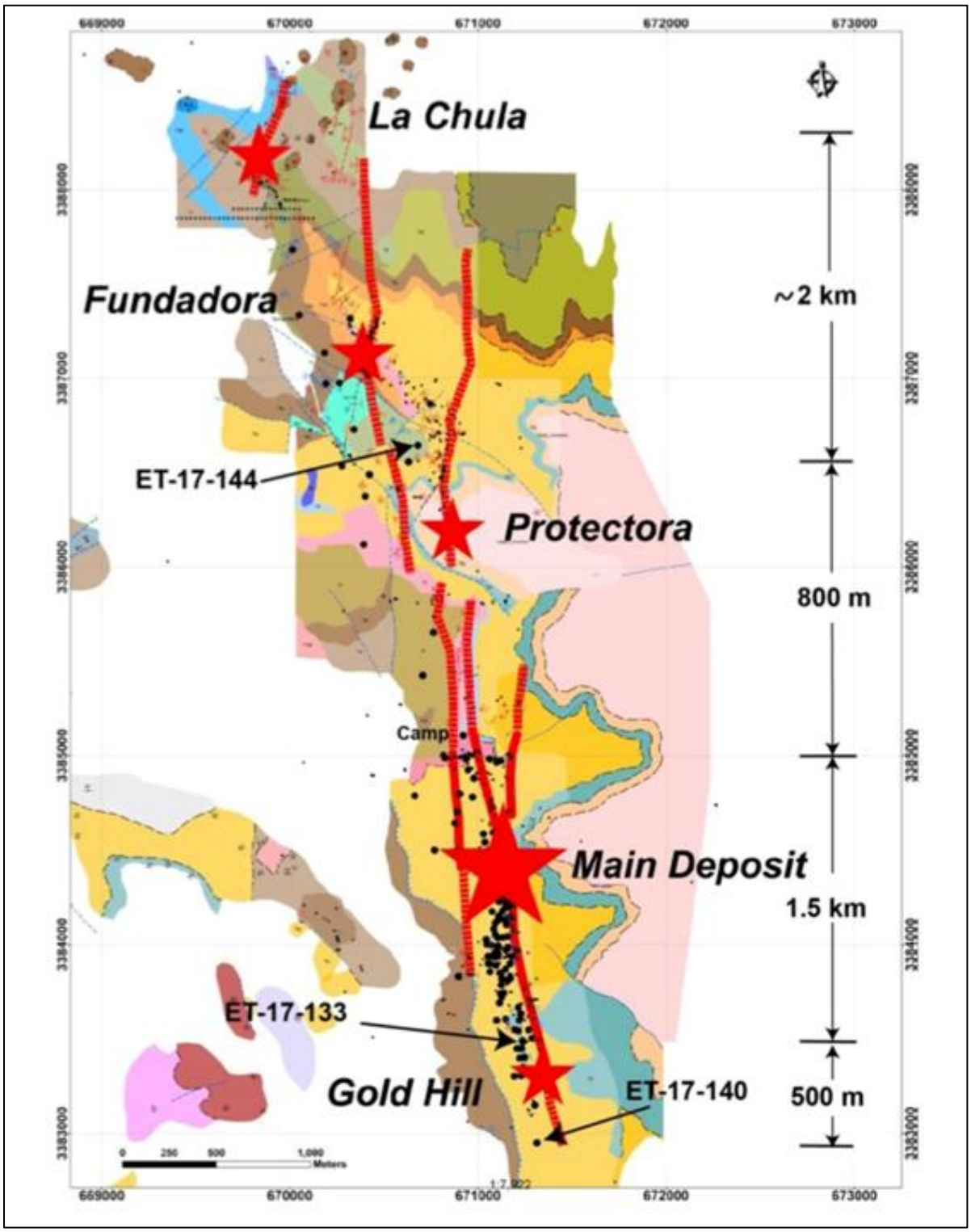
These drill results demonstrated wide oxidized zones of precious-metals mineralization at El Tigre that outcrop at surface.

FIGURE 6.15 UPDATED 2016 - 2017 DRILL HOLE LOCATION MAP



Source: *by P&E (2023)*

FIGURE 6.16 OCEANUS 2016 - 2017 DRILL HOLE LOCATIONS



Source: *P&E (2017)*

TABLE 6.12 OCEANUS DRILLING 2016 TO 2017						
Drill Hole ID	**Northing***	**Easting***	**Elevation (masl)**	**Azimuth (°)**	**Dip (°)**	**Length (m)**
ET-16-083	3,384,150	671,110	1,862	90	-65	197.10
ET-16-084	3,384,975	671,113	1,885	90	-45	142.30
ET-16-085	3,384,125	671,097	1,958.2	90	-45	260.70
ET-16-086	3,384,175	671,122	1,961	90	-45	151.10
ET-16-087	3,384,225	671,149	1,965.6	90	-50	199.90
ET-16-087A	3,384,225	671,149	1,965.6	90	-45	62.30
ET-16-088	3,384,000	671,074	1,905.7	90	-48	178.90
ET-16-089	3,384,050	671,164	1,951	90	-55	144.90
ET-16-090	3,384,000	671,051	1,904	90	-45	220.70
ET-16-091	3,384,200	671,104	1,904	90	-50	157.10
ET-16-092	3,383,975	671,141	1,900	90	-45	150.90
ET-16-093	3,383,950	671,080	1,872	90	-45	137.00
ET-16-094	3,383,900	671,136	1,876	90	-45	144.50
ET-16-095	3,383,975	671,091	1,893	90	-45	199.50
ET-16-096	3,384,250	671,112	1,949	90	-50	178.30
ET-16-097	3,384,300	671,129	1,964	90	-45	150.10
ET-16-098	3,383,750	671,157	1,942	90	-45	153.70
ET-16-099	3,383,800	671,187	1,934	90	-45	150.50
ET-16-100	3,383,875	371,151	1,986	90	-45	148.30
ET-16-101	3,384,500	671,055	2,074	90	-45	239.00
ET-16-102	3,384,375	671,140	2,007	90	-45	150.30
ET-16-103	3,384,375	671,121	1,976	90	-45	212.65
ET-16-104	3,384,125	671,145	1,986	90	-45	138.45
ET-16-105	3,384,325	671,121	1,976	90	-45	129.60
ET-16-106	3,384,275	671,130	1,959	90	-55	160.10
ET-16-107	3,384,225	671,108	1,944	90	-45	150.65
ET-16-108	3,384,175	671,086	1,935	90	-45	180.95
ET-16-109	3,384,025	671,035	1,925	90	-50	254.45
ET-16-110	3,384,025	671,115	1,930	90	-45	169.10
ET-16-111	3,384,975	670,858	1,819	90	-45	252.05
ET-16-112	3,383,750	671,130	1,942	90	-45	244.50
ET-16-113	3,384,800	670,902	1,954	90	-45	224.65
ET-16-114	3,383,850	670,900	1,814	90	-45	342.45
ET-16-115	3,384,500	670,898	2,049	90	-45	313.05
ET-16-116	3,384,425	671,144	2,027	90	-45	177.75
ET-17-117	3,384,704	670,888	1,970	90	-45	205.10
ET-17-118	3,384,500	671,120	2,065	90	-45	211.60

TABLE 6.12 OCEANUS DRILLING 2016 TO 2017						
Drill Hole ID	**Northing***	**Easting***	**Elevation (masl)**	**Azimuth (°)**	**Dip (°)**	**Length (m)**
ET-17-119	3,384,704	670,888	1,970	90	-60	50.00
ET-17-120	3,384,445	671,118	2,036	90	-45	200.70
ET-17-121	3,384,900	670,987	1,884	90	-45	221.00
ET-17-122	3,384,550	671,037	2,099	90	-45	147.60
ET-17-123	3,384,550	671,096	2,097	90	-45	184.30
ET-17-124	3,384,600	671,021	2,096	90	-45	150.60
ET-17-125	3,383,725	671,155	1,960	90	-45	215.00
ET-17-126	3,383,600	671,217	2,034	90	-45	149.00
ET-17-127	3,384,600	671,112	2,077	90	-45	182.60
ET-17-128	3,384,800	670,994	1,938	90	-45	156.60
ET-17-129	3,384,527	667,178	1,300	0	-90	275.50
ET-17-130	3,383,550	671,269	2,041	90	-45	101.90
ET-17-131	3,383,350	671,212	2,013	90	-68	259.10
ET-17-132	3,384,900	670,987	1,884	90	-45	80.00
ET-17-133	3,383,500	671,236	2,020	90	-45	169.10
ET-17-134	3,383,300	671,185	1,940	90	-50	302.00
ET-17-135	3,383,450	671,232	1,994	90	-45	223.70
ET-17-136	3,383,350	671,239	1,960	90	-45	221.00
ET-17-137	3,383,600	671,099	1,988	90	-55	144.20
ET-17-138	3,383,400	671,220	1,973	90	-60	275.40
ET-17-139	3,383,150	671,303	1,830	90	-45	116.00
ET-17-140	3,382,950	671,313	1,797	90	-45	212.00
ET-17-141	3,387,684	670,011	1,927	90	-45	299.00
ET-17-142	3,387,700	670,277	1,939	90	-45	247.40
ET-17-143	3,385,650	670,740	1,840	90	-45	232.00
ET-17-144	3,386,645	670,680	2,040	90	-45	224.00

Source: *P&E (2023)*
Notes: ** coordinates are in WGS 1984 UTM Zone 12.*

TABLE 6.13 OCEANUS DRILLING 2016 TO 2017 ASSAYS								
Drill Cross Section	**Drill Hole ID**	**Comment**	**From (m)**	**To (m)**	**Length(1) (m)**	**Au (g/t)**	**Ag (g/t)**	**AuEq(2) (g/t)**
4150	ET-16-083		12.40	133.50	121.10	1.02	27.0	1.38
		includes	16.70	74.40	57.80	1.51	28.9	1.90
		includes	38.20	57.90	19.70	2.63	40.3	3.17

TABLE 6.13 OCEANUS DRILLING 2016 TO 2017 ASSAYS								
Drill Cross Section	**Drill Hole ID**	**Comment**	**From (m)**	**To (m)**	**Length(1) (m)**	**Au (g/t)**	**Ag (g/t)**	**AuEq(2) (g/t)**
		includes	68.80	74.40	5.70	1.87	10.5	2.01
4975	ET-16-084		13.40	53.00	39.70	0.25	1.0	0.26
		includes	13.40	25.00	11.70	0.40	1.0	0.41
		and	64.30	68.00	3.70	0.14	120.5	1.75
4125	ET-16-085		39.60	129.30	89.70	0.62	30.3	1.02
		includes	80.50	129.30	48.80	0.74	48.9	1.40
		includes	97.30	118.40	21.10	1.38	73.6	2.36
4175	ET-16-086		0.00	6.20	6.20	1.21	37.1	1.71
		includes	41.40	49.00	7.60	1.28	30.0	1.68
		includes	60.30	71.80	11.50	1.14	27.7	1.51
4225	ET-16-087		0.00	79.20	79.20	0.80	16.7	1.02
		includes	14.60	42.10	27.50	1.14	38.7	1.66
		includes	52.50	59.00	6.50	2.86	5.1	2.92
4000	ET-16-088		22.60	30.00	7.50	0.82	1.3	0.84
		and	64.70	126.30	61.60	0.49	12.5	0.66
		includes	98.20	107.30	9.10	1.15	19.5	1.41
		includes	146.60	154.10	7.50	1.18	1.1	1.19
4050	ET-16-089		0.00	60.80	60.80	0.31	21.8	0.60
		includes	46.60	54.80	8.30	0.74	47.9	1.37
4000	ET-16-088		22.60	30.00	7.50	0.82	1.3	0.84
		and	64.70	154.10	89.40	0.48	9.5	0.61
		includes	98.20	107.30	9.10	1.15	19.5	1.41
		and	166.70	168.10	1.40	0.07	397.3	5.37
4000	ET-16-090		43.10	51.80	8.70	0.62	0.6	0.62
		and	96.20	125.30	29.10	0.46	3.0	0.50
4200	ET-16-091		33.40	146.30	112.90	0.39	9.9	0.52
3975	ET-16-092		0.00	95.60	95.60	1.17	13.2	1.35
		includes	0.00	42.10	42.10	2.40	17.1	2.62
3950	ET-16-093		39.40	57.80	18.40	0.37	3.3	0.41
		includes	40.60	44.00	3.40	0.99	4.2	1.04
3900	ET-16-094		0.00	94.70	94.70	0.35	11.7	0.51
		includes	60.00	73.80	13.90	1.00	2.7	1.03
		and	114.00	118.70	4.70	0.09	77.9	1.13
3975	ET-16-095		27.80	123.20	95.50	0.42	26.3	0.77
		includes	48.70	57.00	8.30	1.59	8.0	1.70
		includes	69.90	80.40	10.50	0.40	67.5	1.30
		includes	106.50	109.50	3.00	0.25	410.4	5.72
4250	ET-16-096		9.00	34.00	25.00	0.33	5.2	0.38
		and	54.00	59.40	5.40	1.71	29.4	2.11
		includes	81.00	109.70	28.70	1.06	15.1	1.26

TABLE 6.13 OCEANUS DRILLING 2016 TO 2017 ASSAYS								
Drill Cross Section	**Drill Hole ID**	**Comment**	**From (m)**	**To (m)**	**Length(1) (m)**	**Au (g/t)**	**Ag (g/t)**	**AuEq(2) (g/t)**
		and	140.20	142.80	2.70	1.16	0.7	1.16
4300	ET-16-097		0.00	25.00	25.00	0.23	17.7	0.46
		and	85.20	91.80	6.60	0.45	192.4	3.01
		and	110.00	125.00	15.00	0.44	2.2	0.47
3800	ET-16-099		21.90	36.80	14.90	0.76	12.4	0.92
		and	50.40	70.50	20.20	0.22	20.3	0.49
		and	80.40	98.60	18.30	0.36	92.3	1.60
		includes	81.00	86.20	5.20	0.74	292.6	4.64
3875	ET-16-100		3.80	28.10	24.30	0.60	11.0	0.74
		and	66.70	100.10	33.40	0.33	26.1	0.68
		includes	97.50	98.80	1.30	0.30	476.0	6.64
4500	ET-16-101		41.50	62.00	20.60	0.59	3.9	0.64
		and	72.00	95.30	23.30	0.80	6.7	0.89
		and	136.80	138.40	1.60	1.08	3.8	1.13
4375	ET-16-102		15.50	32.50	17.00	0.38	2.7	0.41
		and	39.90	57.50	17.70	0.28	6.8	0.37
		and	78.00	92.00	14.10	0.32	0.8	0.33
4375	ET-16-103		57.70	88.40	30.70	0.44	5.8	0.52
		and	153.20	164.00	10.80	0.60	1.1	0.61
		and	173.00	179.60	6.60	0.65	60.8	1.46
4125	ET-16-104		22.60	138.50	115.90	0.43	11.4	0.58
		includes	35.80	102.80	67.00	0.56	18.3	0.81
		includes	54.00	70.70	16.80	0.63	48.8	1.28
			95.30	98.20	2.90	5.01	10.1	5.15
4325	ET-16-105		14.50	93.50	79.00	0.54	10.6	0.68
		includes	41.50	58.50	17.00	0.64	29.4	1.04
		includes	54.20	58.50	4.40	0.84	79.5	1.90
		includes	81.00	93.50	12.50	1.25	3.1	1.29
4275	ET-16-106		0.00	54.90	54.90	0.30	14.4	0.49
		includes	32.30	42.30	10.00	0.45	42.8	1.02
		and	64.50	66.00	1.50	2.35	4.2	2.41
4225	ET-16-107		2.30	9.50	7.20	0.62	3.3	0.67
		and	18.30	81.90	63.70	0.36	34.9	0.83
		includes	19.40	30.60	11.20	0.67	33.3	1.11
		includes	59.50	65.90	6.50	1.04	129.9	2.77
		includes	71.10	74.40	3.40	0.27	117.0	1.83
		and	101.90	120.70	18.80	0.54	6.9	0.64
4175	ET-16-108		42.70	152.70	110.00	0.60	14.5	0.79
		includes	49.90	55.00	5.10	2.16	3.1	2.20
		includes	74.10	86.00	11.90	1.11	7.1	1.20

TABLE 6.13 OCEANUS DRILLING 2016 TO 2017 ASSAYS								
Drill Cross Section	**Drill Hole ID**	**Comment**	**From (m)**	**To (m)**	**Length(1) (m)**	**Au (g/t)**	**Ag (g/t)**	**AuEq(2) (g/t)**
		includes	102.50	118.50	16.00	0.82	64.7	1.69
		includes	136.50	144.00	7.50	1.20	2.6	1.23
4025	ET-16-109		111.90	140.70	28.80	0.70	3.1	0.75
		includes	117.20	124.20	7.00	1.57	4.4	1.63
		and	160.90	181.30	20.40	0.40	212.0	3.23
		includes	163.60	167.60	4.00	0.82	981.2	13.90
		includes	163.60	164.30	0.70	2.12	2,964.5	41.65
		and	196.50	199.90	3.30	0.30	6.0	0.38
		and	210.90	215.00	4.10	0.19	14.2	0.38
4550	ET-17-123		76.40	80.60	4.20	0.42	0.8	0.43
		and	133.50	149.50	16.00	0.20	13.5	0.38
4600	ET-17-124		64.50	82.70	18.20	0.23	87.3	1.39
		and	94.80	120.60	25.80	0.41	20.9	0.69
4600	ET-17-124		64.50	82.70	18.20	0.23	87.3	1.39
3700	ET-17-125		13.90	19.10	5.30	0.74	0.5	0.75
		and	58.70	62.00	3.40	0.42	33.0	0.86
		and	134.00	142.20	8.20	0.37	37.1	0.87
3600	ET-17-126		4.50	23.00	18.50	0.17	72.1	1.13
		includes	15.80	21.50	5.80	0.48	182.8	2.92
		and	87.00	105.00	18.00	0.35	32.3	0.78
		and	112.00	118.00	6.00	0.19	11.3	0.34
4600	ET-17-127		35.00	58.60	23.60	0.35	27.9	0.72
		includes	51.10	52.70	1.60	1.30	395.9	6.57
4775	ET-17-128		86.40	112.20	25.80	0.63	28.0	1.00
		includes	100.40	105.10	4.70	1.06	106.6	2.48
3550	ET-17-130		53.80	55.70	1.90	0.34	11.9	0.49
3550	ET-17-131		58.00	67.30	9.30	0.74	9.1	0.86
		and	77.90	86.30	8.40	0.27	2.7	0.30
		and	142.50	147.00	4.50	0.80	74.6	1.80
		and	178.00	202.20	24.20	0.35	22.0	0.65
4900	ET-17-132		37.50	48.20	10.70	0.20	22.1	0.50
		and	53.00	68.00	15.00	0.33	10.9	0.47
3500	ET-17-133		65.40	68.40	3.00	0.98	1.0	0.99
		and	78.50	146.10	67.60	1.24	19.1	1.49
		includes	97.50	120.90	23.40	2.77	40.5	3.31
		and	137.00	146.10	9.10	0.29	4.7	0.35
		and	156.90	160.00	3.10	0.48	0.7	0.49
3300	ET-17-134		98.20	105.50	7.30	0.62	9.6	0.75
		and	133.20	147.50	14.30	1.01	0.5	1.02
		includes	134.00	135.80	1.80	6.33	2.1	6.36

TABLE 6.13 OCEANUS DRILLING 2016 TO 2017 ASSAYS								
Drill Cross Section	**Drill Hole ID**	**Comment**	**From (m)**	**To (m)**	**Length(1) (m)**	**Au (g/t)**	**Ag (g/t)**	**AuEq(2) (g/t)**
		and	223.40	226.10	2.70	0.50	0.6	0.51
		and	239.90	242.00	2.10	1.06	80.6	2.13
3450	ET-17-135		71.90	109.60	37.70	0.62	12.4	0.78
		includes	77.60	96.40	18.80	0.91	18.9	1.16
		and	121.40	134.30	13.00	0.60	12.9	0.77
		and	140.30	154.60	14.30	0.68	4.7	0.74
		and	215.30	223.70	8.40	1.52	32.4	1.95
3350	ET-17-136		26.00	44.00	18.00	0.94	3.5	0.99
		and	137.00	146.50	9.50	1.57	3.1	1.62
		and	155.40	164.80	9.40	0.40	0.5	0.41
		and	174.00	180.00	6.00	0.35	0.8	0.36
		and	195.50	206.00	10.50	0.33	0.9	0.34
3600	ET-17-137		98.50	129.50	31.00	0.41	1.3	0.43
		and	145.20	175.30	30.10	0.38	13.0	0.55
		and	268.60	276.10	7.50	0.32	1.3	0.33
3400	ET-17-138		20.00	25.00	5.00	0.42	0.7	0.43
		and	66.70	83.40	16.80	0.21	6.9	0.31
		and	103.10	104.60	1.50	0.46	1.2	0.48
		and	178.00	185.00	7.00	0.24	1.2	0.26
		and	238.00	246.30	8.30	0.28	5.1	0.35
3150	ET-17-139		10.60	15.80	5.20	0.96	1.7	0.98
2950	ET-17-140		35.00	44.00	9.00	0.18	125.5	1.86
		includes	36.50	38.00	1.50	0.43	683.2	9.54

Source: *P&E (2023)*
Notes: *(1) True thickness has not been calculated for each individual intercept. It is generally estimated to be 75 to 90% of drill core interval length. Metallurgical recoveries and net smelter returns are assumed to be 100%.*
(2) Gold equivalent ration is based on gold to silver price ration of 75:1 (Au:Ag).

6.4 HISTORICAL RESOURCE ESTIMATES

Several mineral resource estimates have been completed on the Silver Tiger Project dating back to the early 1900s. However, these historical estimates were completed prior to the implementation of current Mineral Resource estimation standards and definitions, and will not be discussed further in this Technical Report. These historical mineral resources have been superseded by subsequent Mineral Resource Estimates prepared in accordance with NI 43-101.

The more recent Mineral Resource Estimates were completed in 2013 by Hard Rock Consulting Inc. and in 2017 and 2023 by P&E. These Mineral Resource Estimates have been superseded by the 2024 updated Mineral Resource Estimate by P&E and will not be discussed further in this Technical Report. The 2024 P&E Mineral Resource Estimate is described in Section 14 of this Technical Report.

6.5 HISTORICAL MINING

The El Tigre Mine began production in 1903 and continued operation until 1931, with the period between 1910 and 1917 seeing intermittent mining due to the Mexican Revolution. After 1931, the operations briefly resumed with the introduction of unionized labourers. However, in 1938, the Mine closed permanently due to low silver prices, increased union demands, and a new 11% production royalty that caused the Mine to become uneconomic.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

This section is based on Section 7 from P&E (2023) and has been updated where applicable.

7.1 REGIONAL GEOLOGY

El Tigre is located on the eastern flank of the Sierra El Tigre within the Basin and Range physiographic province, which extends from northern Nevada to Zacatecas and Jalisco in Mexico. The Sierra El Tigre is part of the massif of the Sierra Madre Occidental. The Sierra Madre Occidental Belt ("SMOB") is a 1,200 by 300 km northwest-trending volcanic plateau composed of thick accumulations of andesite to rhyolite volcanic rocks that extend from southeastern Sonora to Queretaro (Figure 7.1). The Basin and Range Province hosts many of Mexico's most historically important mineral deposits. The SMOB is characterized by a northwest-trending broad anticline.

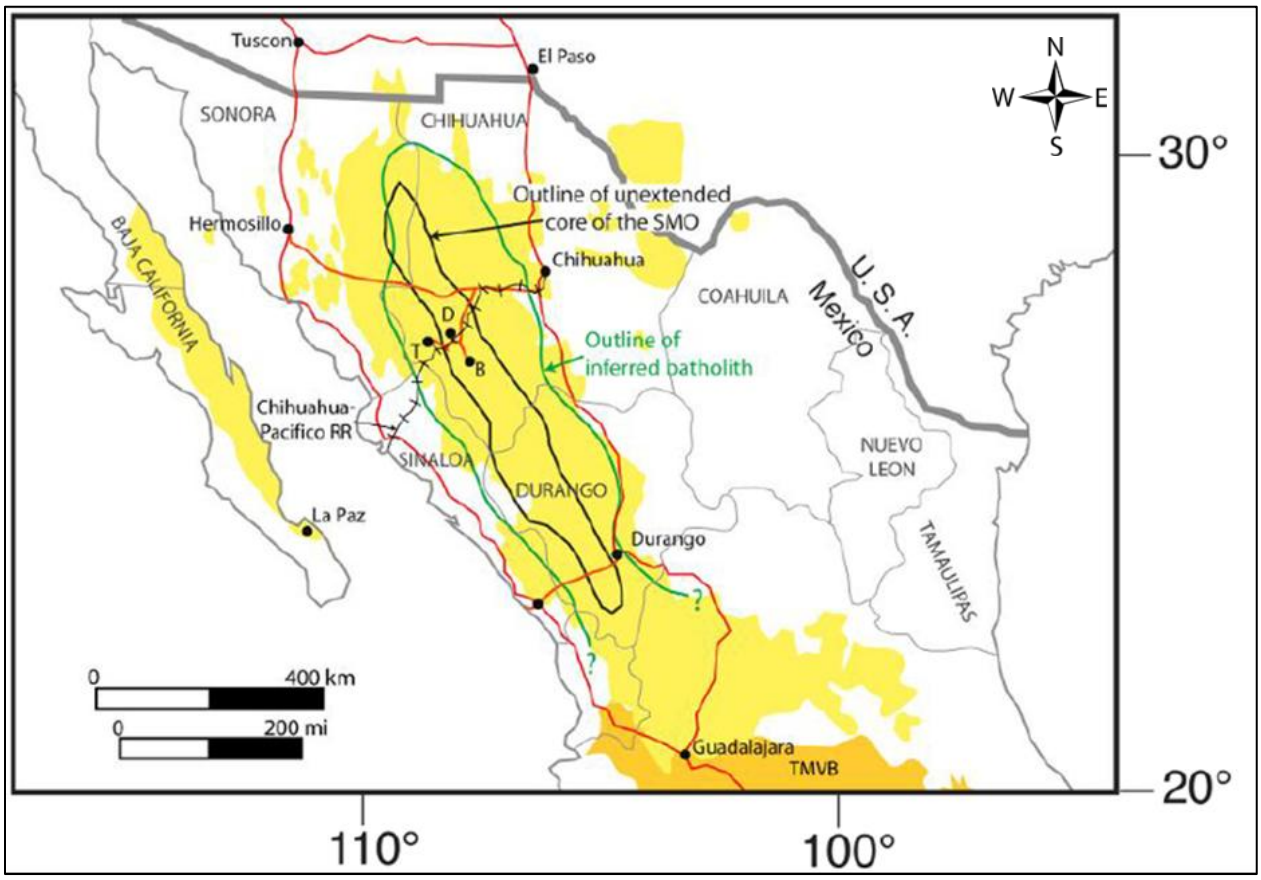
The geology of the SMOB is characterized by units of volcanic rocks known as the Upper Volcanic Series ("UVS") and the Lower Volcanic Series ("LVS"). The UVS and LVS are considered to reflect subduction-related continental arc magmatism that slowly migrated eastward during the early Tertiary, and then retreated more quickly westward, reaching the western margin of the continent by the end of the Oligocene (Sedlock *et al*., 1993). The eastward migration is represented in the SMOB by the LVS. The LVS is composed primarily of andesite with interlayered felsic ash flow deposits (46 Ma to 35 Ma), which are >2,000 m thick with local intrusions.

The westward retreat of the subduction-related continental arc magmatism is represented by the UVS of caldera-related, large-volume rhyolitic ash flow tuffs of Oligocene age (35 Ma to 27 Ma) lying unconformably on the LVS. The UVS generally consists of calc-alkalic rhyolite ignimbrites with minor andesite, dacite and basalt (Overbay *et al*., 2001) and is as much as 1,600 m thick.

Cenozoic extensional faulting, which consists of northerly-trending horsts and grabens, exposes Precambrian granite and Paleozoic limestone, the oldest rocks in the range. The Teras Fault Zone was the locus of the 7.5 magnitude Sonoran earthquake on May 3, 1887, when dip-slip movements of as much as 14 m were measured on scarps in the Sierra El Tigre (Suter, 2008). This same fault system transects the El Tigre mining district and mineralization appears to be hosted in associated graben bounding faults.

The fault zone forms the eastern boundary of the central horst block of the Sierra El Tigre. The horst block is an anomalous structural high in the region, exposing Paleozoic limestone and Precambrian granite. The presence of high-grade, epithermal precious metals veins in graben bounding faults is a common occurrence in many major epithermal Au-Ag districts worldwide.

FIGURE 7.1 TERTIARY VOLCANIC ROCKS OF THE SIERRA MADRE OCCIDENTAL



Source: *Busby (2008)*
Note: *SMOB = Sierra Madre Occidental Belt (outlined green).*

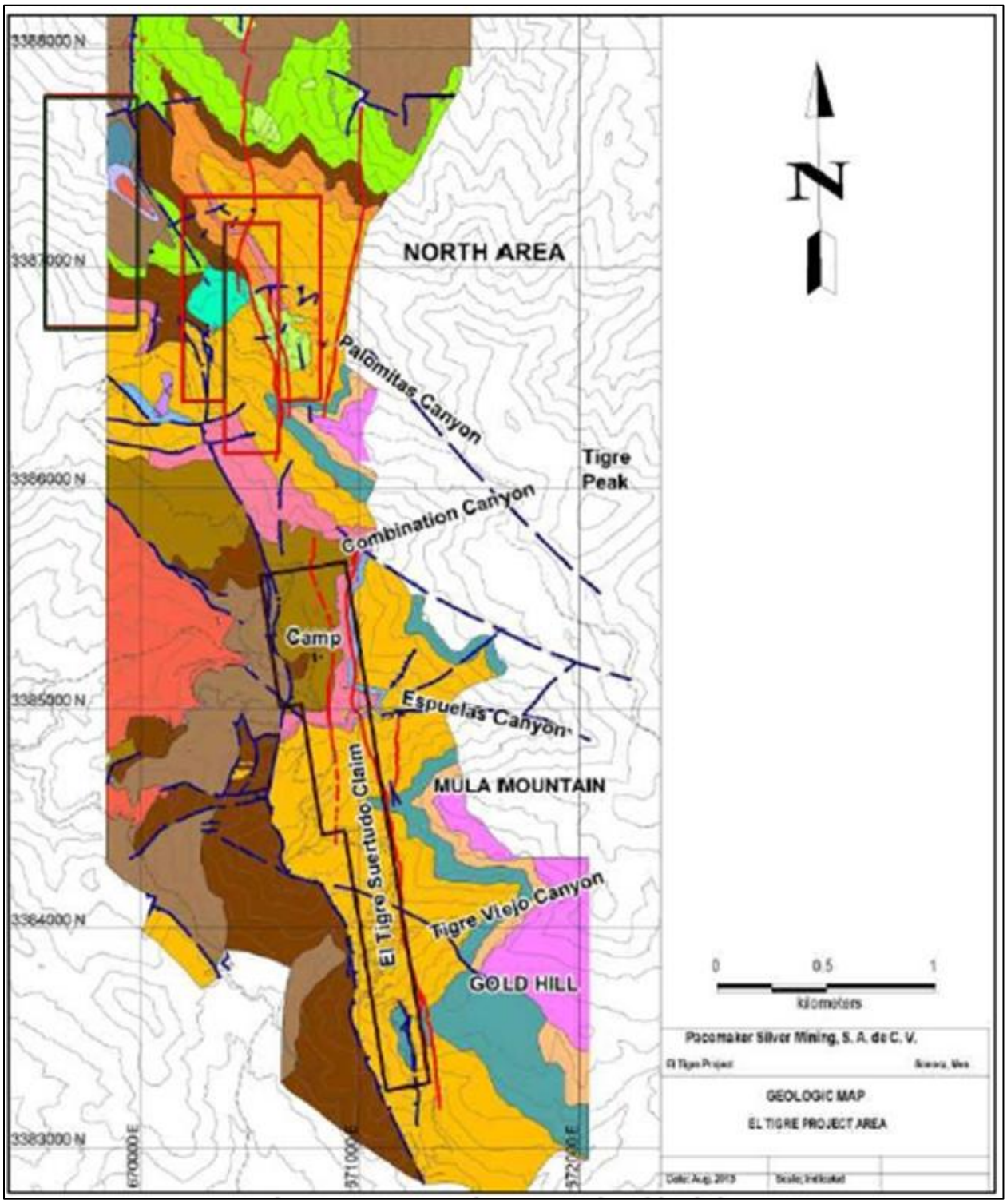
7.2 LOCAL GEOLOGY

The central Sierra El Tigre consists of a thick sequence of Tertiary volcanic rocks overlying granitic basement in the south, Pre-Cenozoic alluvial fanglomerates in the west, and Paleozoic bedded limestones in the north. Block faulting and the intrusion of several andesitic and rhyodacite stocks and dykes have broken up much of the original volcanic stratigraphy.

The entire volcanic sequence in the central portion of the El Tigre mining district is folded into a gentle anticline, in which the southern limb is tilted approximately 15° to the south. The axis of the anticline is approximately east-west and passes halfway between the El Tigre camp area and the northern veins.

Stratigraphic relationships indicate that the tilting occurred during or prior to deposition of the volcanic sequence (Figure 7.2 and Figure 7.3).

FIGURE 7.2 ANACONDA GEOLOGIC MAP OF THE EL TIGRE MINING DISTRICT



Source: *Black and Choquette (2013)*

FIGURE 7.3 GEOLOGIC MAP LEGEND

Source: *Black and Choquette (2013)*

The El Tigre area is underlain by a major, complex normal fault zone (the Teras Fault Zone) that forms the boundary between a horst block to the west and a graben block to the east. The fault zone runs north-south through the entire Sierra El Tigre mountain range. The Teras Fault is visibly identifiable at ground surface by an abrupt change in both rock formations and topographic relief at the mouth of the El Tigre canyon. On the footwall (eastern side) of the structure, the El Tigre Formation occurs 300 m above the base of the canyon. On the hanging wall (western side), very young andesite agglomerates and breccias occur 20 to 30 m above the stream bed.

7.3 LOCAL AND PROPERTY GEOLOGY

The results of original exploration of the district by Mishler (1920) defined the basic geological framework that was used to great advantage by the Lucky Tiger Combination Gold Mining Company to locate and develop mineralization in the El Tigre Mine.

7.3.1 Stratigraphy

Pre-Cenozoic basement rocks in the Sierra El Tigre include massive limestones and a coarse-grained granite intrusive of presumed Precambrian age. Mishler (1920) describes the granite as "consisting mainly of microcline, sanidine, quartz and phlogopite mica, the last now largely changed to serpentine and iron oxide".

The Cenozoic volcanic stratigraphy of the Sierra El Tigre in the El Tigre Property area was first described by Mishler (1920). The Mishler work became the basis of Anaconda's exploration work from 1981 to 1984. That field work extended the known volcanic stratigraphy away from the main district, and added several previously unknown units that lie outside the main mining area.

The following volcanic and intrusive units, described in stratigraphic order, are found in the main El Tigre area, which is an area of approximately six square miles (1,555 ha) represented in Figure 7.2 and Figure 7.3 and are described below in stratigraphic order.

7.3.1.1 Granite (PCgr)

The oldest lithologic unit at the El Tigre area is a dark reddish-brown, medium-to coarse-grained, hypidiomorphic-granular, biotite quartz monzonite to granite of Precambrian age. Exposures are found along the bottom of Tigre Canyon. The granite is strongly chloritized in many places and may have undergone regional metamorphism. Outcrops are of extensively weathered, crumbly rocks weakly resistant to erosion and are covered by a 3 to 5 m thick paleo-soil of coarse arkosic sandstone with limestone cobbles.

7.3.1.2 Nodular Formation (Tn)

In outcrop, the Nodular Formation forms nearly vertical cliffs up to 150 m high. Within the central part of the El Tigre area, outcrops commonly contain numerous spherulites (nodules) varying between 3 mm and 35 cm in diameter (Park, 1982). The Nodular Formation is found throughout much of the El Tigre area, varying in thickness up to 200 m in the vicinity of Mula Mountain.

The Nodular Formation is a light grey, coarse- to fine-ash, rhyolitic, welded, crystal-vitric tuff containing 2 to 10% K-feldspar and quartz crystals in a homogeneous, aphanitic matrix. The crystals average 1 mm to 3 mm in diameter and are generally anhedral or broken. Compaction layering, defined by the parallel alignment of flattened lenses filled with spherulites and quartz crystals, is also visible in thin section.

7.3.1.3 Fragmental Andesite (Tfa)

The Fragmental Andesite conformably overlies the Nodular Formation and is exposed in the Tigre Viejo Canyon west of the Fortuna Fault and in a fault slab in the Combinación Canyon. In both locations, the thickness is estimated to be ≤10 m. The Fragmental Andesite is a light greenish grey, rhyodacite, block-lapilli-ash, crystal-lithic agglomerate. Rock fragments are 0.2 to 15.0 cm in size and make up 35 to 45% in volume. The matrix is a fine-grained, rhyodacite ash tuff with 0.5 to 2.0 mm sericitized plagioclase crystals.

7.3.1.4 Flat Formation (Tf)

The Flat Formation outcrops in an area bounded by Mula Mountain to the south and Palomitas Canyon to the north. Thick exposures of the Flat Formation are observed in Combination Canyon and Espuelas Canyon, however, are not observed south of Mule Mountain, due to faulting and the general southerly dip of the volcanic rocks.

Drill holes have intersected varying thicknesses of the Flat Formation, ranging from 40 to 100 m. The formation may be interfingered with the nearby Mula Mountain flow dome unit (Tabular Formation), as evidenced in Espuelas Canyon in exposures adjacent to the level four main dump. The unit is very prominently bedded, with individual beds averaging 10 to 15 cm in thickness. Sedimentary features such as graded bedding ripple marks and flame structures are common.

The upper part of the Flat Formation is composed of gravel to fine-sand sized, angular to sub-angular fragments of white, siliceous volcanic rock set in a light tan or green, clay-rich matrix. Other matrix constituents include calcite, chlorite, silica, and hydro-biotite (Lujan *et al.*, 1984). The lower part of the Flat consists of thinly bedded, calcareous black shale and is a noticeable change in deposition environment from the upper portion of the Flat Formation. In surface exposures, the shale occurs either as discontinuous, ripple-marked beds 2 to 5 mm thick or, more frequently, as reworked, cornflake-shaped clasts within sandy, poorly graded beds.

The Flat Formation is interpreted as a water-laid tuff, which together with volcaniclastic sediments, was probably deposited in a lacustrine environment. The absence of crossbedding and stream channel features, the angularity and relatively small size of the lithic clasts, and the thinness and continuity of individual beds suggests that the depositional medium was quiet water. When the unit was formed, the tuffs were deposited into a lake adjacent to the Tabular flow dome.

There are three different time units that have been mapped together as Flat Formation. The thickest is the "true" Flat Formation as described by Mishler (1920) which outcrops in the northern portion of the El Tigre Suertudo concession in the Espuelas and Combinacion Canyons and appears to be syn- to post-Tabular in age. Alternatively, these exposures are found in the deepest portion of the lake and were physically removed from the Tabular eruption. The other two-time units are

lithologically similar, however, are only a few tens of metres thick. The older of these two units is pre-Tabular, post-Nodular in age. The younger unit is contemporaneous with the Tabular Formation. Both older Flat Formation units are also water-laid tuff and probably formed in shallow lakes that existed intermittently throughout the time required for the deposition of the Tabular Formation.

7.3.1.5 Cliff Formation (Tc)

The Cliff Formation is a coarse-to fine-ash, moderately welded, crystal-vitric rhyolite tuff that outcrops in the central part of the El Tigre area, in the northern portion of the El Tigre Suertudo Concession. The Cliff Formation is relatively thin in Palomitas Canyon in the northern portion of the El Tigre area and pinches out entirely before reaching Bota canyon. It conformably overlies Flat Formation south of Palomitas Canyon.

The Cliff Formation forms massive cliffs up to 50 m high, commonly with moderately well-developed, pseudo-columnar jointing. The massive nature of the Cliff Formation, together with the abundance of glass observed in petrographic thin sections, suggests that the Cliff was deposited as a pyroclastic ash flow or ignimbrite.

In hand specimen, the Cliff Formation is a greyish-white, homogeneous, very fine-grained rock with few visible crystals and no lithic fragments. Dark crystallites are locally abundant in some drill intercepts. Drill core is commonly stained with reddish-purple iron oxides. In petrographic thin section, the unit rock contains 10% to 12% coarse-ash-sized (1 to 2 mm) crystals of quartz and strongly sericitized feldspars. The matrix is composed of fine-ash-sized crystals and abundant glass spicules.

7.3.1.6 Tuff Formation (Ttf)

The Tuff Formation is a thin and restricted unit that outcrops only in the northeast part of the El Tigre Suertudo concession and is found in drill holes as far south as Tigre Viejo Canyon. It conformably overlies the Cliff Formation and has a maximum thickness of 18.3 m (Park, 1982). The rock is thinly foliated or stratified, and contains 25 to 30% angular lithic clasts, averaging approximately 2 cm in diameter. The clasts are set in a light grey, fine-ash matrix. The Tuff Formation is probably a lapilli to fine-ash lithic tuff deposited as a pyroclastic ash fall.

7.3.1.7 Tigre Formation (Ttg)

The Tigre Formation crops out east of the Fortuna Fault, from southern Bota Canyon southward to the southern boundary of the El Tigre area. In addition, it crops out in several small grabens and plateaus west of the Fortuna Fault. Its thickness varies from 250 m in the central portion of the El Tigre Suertudo Concession to 180 m in Palomitas Canyon. The Tigre Formation conformably overlies the Cliff Formation or the Tuff Formations. Unaltered biotite from the Tigre ash-flow tuff was dated radiometrically at 31.7 (±1.3) Ma (Thoms, 1988).

The Tigre Formation consists of a lower ash unit and an upper ash unit. The lower unit is lavender coloured and massive; the upper Tigre unit is a light tan massive ash flow that is similar in texture and composition to the Cliff Formation. The composition of the lower Tigre unit varies from

rhyodacite to quartz latite and contains 15 to 40% subhedral, coarse-ash-sized (1 to 3 mm) crystals, mostly plagioclase and K-feldspar, 2 to 4% anhedral quartz, and 4 to 5% subhedral biotite are typically present.

The crystals are set in a bluish-grey, glass-rich, fine-ash matrix. The rock also commonly contains 10 to 20% lapilli-sized (1 to 3 cm) fragments of flattened pumice. These fiamme define a crude foliation within the Tigre that is especially prominent in the upper part of the lower unit.

The upper part of the Tigre Formation is massive lapilli-to fine-ash, moderately welded, lithic-vitric-crystal tuff that was probably deposited as several separate, however, compositionally similar, ash flows. Although vertical changes in texture and composition are not strong, they are sufficiently prominent to suggest that the Tigre was not deposited by a single ash flow.

7.3.1.8 Quartz Rhyolite Formation (Tgr)

The Quartz Rhyolite Formation is 30 to 80 m thick. This unit is a coarse-to fine-ash, rhyolite, crystal tuff that conformably overlies the Tigre Formation. Its main outcrop is in the eastern portion of the Property, notably capping Gold Hill.

The Quartz Rhyolite contains 10 to 15% anhedral, coarse-ash-sized (2 to 3 mm) crystals of quartz and K-feldspar. Although slightly less than half of the crystals are quartz, the quartz stands out much more clearly than the K-feldspar and gives the rock the appearance of quartz-eye porphyry. The crystals are set in a yellowish-tan, poorly welded, faintly banded matrix of glassy fine-ash. The banding within the matrix is defined by 2 to 3 mm thick yellow-gold layers alternating with 5 to 10 mm-thick light tan layers. In petrographic thin section, thin quartz lenses parallel this layering.

7.3.1.9 Agglomerate Formation

The Agglomerate Formation conformably overlies the Quartz Rhyolite Formation and crops out in essentially the same areas. Its thickness can vary from 30 to 110 m, however, is typically only 30 to 50 m.

The Agglomerate Formation is a red-brown, block-to fine-ash, vitric-crystal-lithic tuff with a quartz latite composition. It contains 30 to 40% mostly lapilli-sized (0.5 to 1.5 cm), angular, grey lithic fragments set in a red-brown, crystal-rich matrix. The crystals in the matrix are ash-sized (0.2 to 0.5 mm), irregular fragments of quartz (10% of total rock), plagioclase (10%), and K-feldspar (8%). The remainder of the matrix is dominantly glass shards, with some clay.

7.3.1.10 Quartz Mica Rhyolite Tuff (Tgmr)

The Quartz-Mica Rhyolite Tuff covers most of the tops of Tigre Peak and Gold Hill. This unit conformably overlies the Agglomerate Formation and is ~140 m thick.

In both hand specimen and petrographic thin section, the texture of the Quartz-Mica Rhyolite closely resembles that of the Tigre Formation. Both formations are dark lavender or greyish-purple and contain abundant coarse-ash-sized crystals of feldspar. The crystals are set in a glass-rich, fine-

ash matrix. The Quartz-Mica Rhyolite typically contains slightly more crystals than the Tigre Formation, averaging approximately 30 to 35%. Also, the Quartz-Mica Rhyolite is more felsic (rhyolite) in composition with 7 to 10% quartz, 15 to 20% K-feldspar, and only 5 to 6% plagioclase crystals. Mafics totalling 2 to 4%, (mostly biotite) are also present. The rock is a coarse-to-fine ash, crystal-vitric rhyolite tuff.

The Quartz-Mica Rhyolite, the Agglomerate Formation, and the Quartz Rhyolite Formation are all generally massive in outcrop, although the Agglomerate exhibits a poorly developed foliation similar to that of the Tigre Formation. The three formations are also very glass-rich and contain many broken crystals. All three units were probably deposited as pyroclastic ash-flow tuffs, with each from a different source or from a single source with changing magmas over time.

All the volcanic rocks in the El Tigre district appear to have been deposited before the mineralization event took place that formed the veins.

7.3.1.11 Intrusive Rocks

There are two types of intrusive rocks within the El Tigre area. One is the flow banded Tabular Formation (Tta) found on Mula Mountain and along the Fortuna Fault. The other intrusive type is a nearly aphanitic greenish-black andesite (Ta) found as dykes throughout the El Tigre area.

7.3.1.12 Tabular Formation

The Tabular Formation averages approximately 120 m in thickness (ranging from zero to 180 m). The unit alternately thickens and thins from Gold Hill northward to Bota canyon. The Tabular Formation has previously been labelled as a coarse-to fine-ash, vitric-crystal rhyolite tuff containing 10 to 15% anhedral crystals of quartz, K-feldspar, and minor plagioclase in a fine-ash matrix. However, work by Lujan (2010) recognized it as a flow dome rhyolite.

The most conspicuous feature of the Tabular Formation is a well-developed tabular parting or foliation that allows the rock to be cleaved into 1 to 2 cm thick, wavy plates. Each parting is marked by 0.5 mm thick planar lenses of quartz that parallel the foliation. The foliation is probably the result of multiple intrusions of the viscous rhyolite into previously injected magma.

At several localities, the Tabular Formation is marked by a 2 to 3 m thick, laterally discontinuous breccia containing blocks of foliated Tabular Formation up to 0.5 m in diameter. These blocks could indicate dome debris breccias falling down the side of a growing dome. In Tigre Viejo Canyon near the Fortuna Fault the Tabular is assimilating Nodular Formation, which suggests that the Tabular Formation may have been extruded onto a surface consisting of Nodular Formation. Outcrops of Tabular are located near major faults, which also suggests that underlying magmas utilized these older fault zones as magma conduits.

7.3.1.13 Andesite

The other intrusive is the Andesite unit (Ta), which is mainly aphanitic and rarely contains more than 1 to 2% megascopic phenocrysts. In thin section, 5% pyroxene and 30% plagioclase crystals are set in a finer groundmass of feldspar. The rock is commonly propylitized and may contain 5 to

10% chlorite and equal amounts of secondary calcite. The Andesite in the Palomitas Canyon is rimmed by an intrusion breccia of cobble-sized andesite fragments in a clayey, weathered matrix. The Andesite typically outcrops in the bottoms of canyons, or else forms low, steep-sided ridges covered with thick, clayey soil. Andesite is found to intrude all the lower volcanic units up through the Tigre Formation. Commonly, the Andesite is found as narrow dykes found within vein structures. An example of this occurs in Espuelas Canyon.

7.3.2 Structure

The dominant structural feature in the El Tigre District is a north-northwest-trending, south-pointing, wedge-like horst limited by two large fault systems. The larger, the Corral Fault, cuts through 7 km of the El Tigre area in a northwesterly direction. The block west of the fault has been downthrown 450 to 950 m, depressing the entire flank of the Sierra El Tigre (Mishler 1920). The second largest fault, the Fortuna Fault, traverses the centre of the El Tigre area in a north-northwesterly direction for 7 km, where its vertical displacement ranges from 190 to 330 m. The combination of these faults has given the southern portion of the horst block a maximum topographic elevation to expose the Precambrian Granite, the oldest rock in the region.

The El Tigre Vein mineralization is lodged in the eastern hanging wall graben block.

The entire lower portion of the volcanic sequence is tilted to the south with a dip of 15°. The upper units are all thicker toward the south, which suggests that the tilting was completed before the end of the volcanic period.

The veins are hosted in minor, north-trending faults that represent the first fracturing in the region. Secondary, steeper faults parallel to the El Tigre Mine Vein and contemporaneous with the mineralization are indicated by Mishler (1920) to have contributed to form the high-grade mineralized bodies in the southern half of the Mine. In the northern portion of the Mine, east-west faults correlate with the high-grade mineralized bodies (Mishler, 1920). Possibly associated with the Corral and Fortuna Faults described above, are several northwest-trending normal faults that have affected the horst blocks and, with fewer incidences, the vein-bearing block on the east side.

Second order fault structures splay off the main faults and host the veins as sigmoid-loop-type structures. Abundant evidence suggests that the vein structures underwent both right-lateral strike-slip and dip-slip displacement at different periods of regional stress. Both directions of displacement developed areas of widening in the veins, which prepared the rock for mineralization. Historical mining records suggest that normal stoping widths along discrete veins were approximately 1 m in width, whereas some wider mineralized shoots were mined up to 3 to 5 m wide (Mishler, 1925).

7.4 DEPOSIT GEOLOGY AND MINERALIZATION

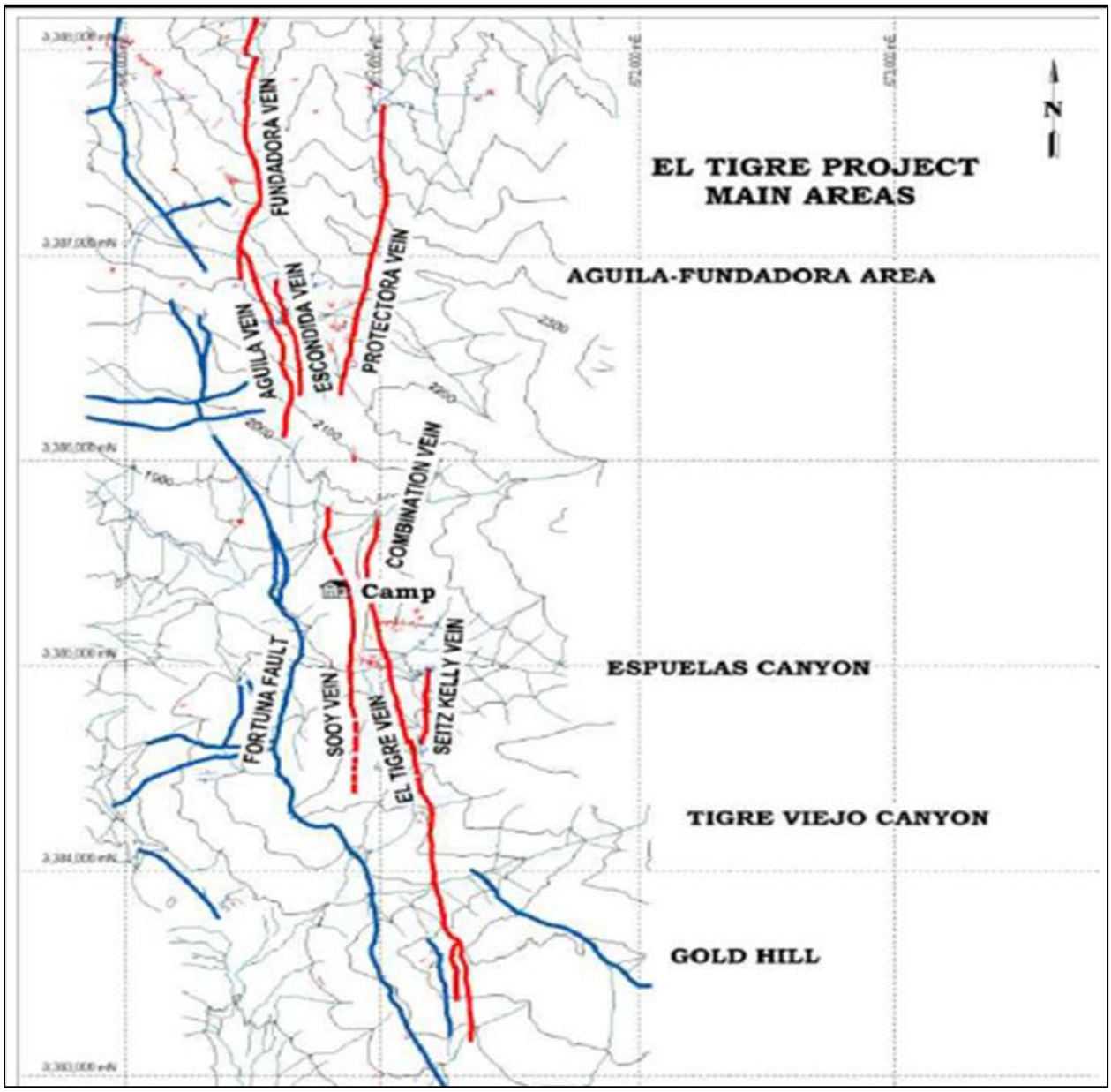
The El Tigre silver and gold deposit is related to a series of high-grade epithermal veins controlled by a north-south trending faults, which cut across the andesite and rhyolite tuffs of the Sierra Madre Volcanic Complex within a propylitic alteration zone, as much as 150 m in width, in the El Tigre Formation. The veins dip steeply to the west (although steep dip reversals occur locally) and are typically 0.5 m thick (locally up to 5 m thick). The veins, structures and mineralized zones outcrop

on surface and have been traced for 5.3 km along strike. Historical mining and exploration activities focused on the 1.5 km portion at the southern end of the Deposit, principally on the El Tigre, Seitz Kelly and Sooy Veins, whereas the Caleigh, Benjamin, Protectora and Fundadora Veins to the north remain under-explored. The location of these mineralized veins is shown in Figure 7.4. Exploration work by Silver Tiger at El Tigre has identified four mineralization styles:

1. Epithermal veins;
2. Stockwork Zone;
3. Black Shale Zone; and
4. Sulphide Zone.

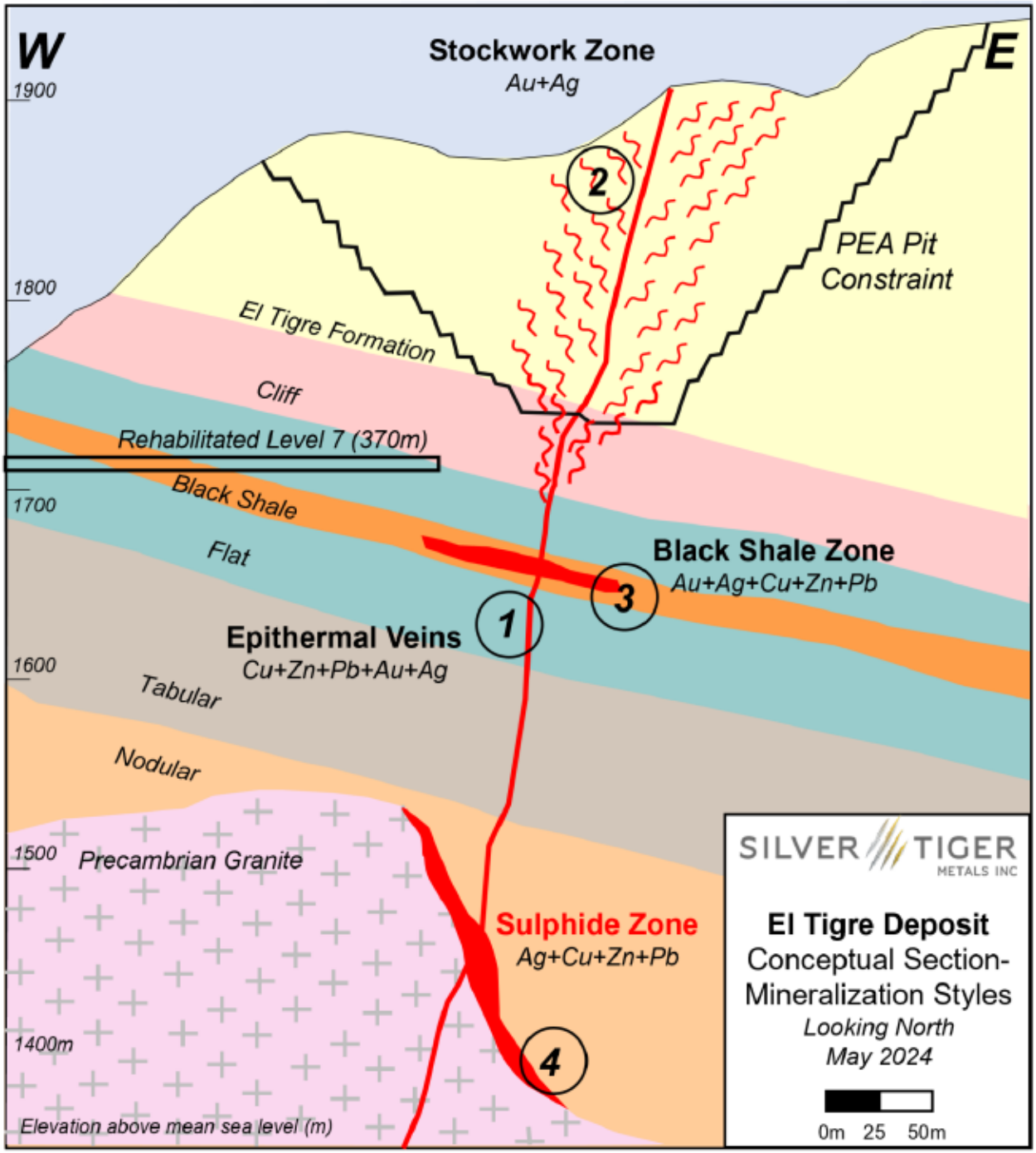
The relative conceptual distribution of these zones in cross-section and longitudinal section is shown in Figure 7.5 and Figure 7.6, respectively.

FIGURE 7.4 MINERALIZED VEINS AND POST-MINERAL FAULTS OF EL TIGRE



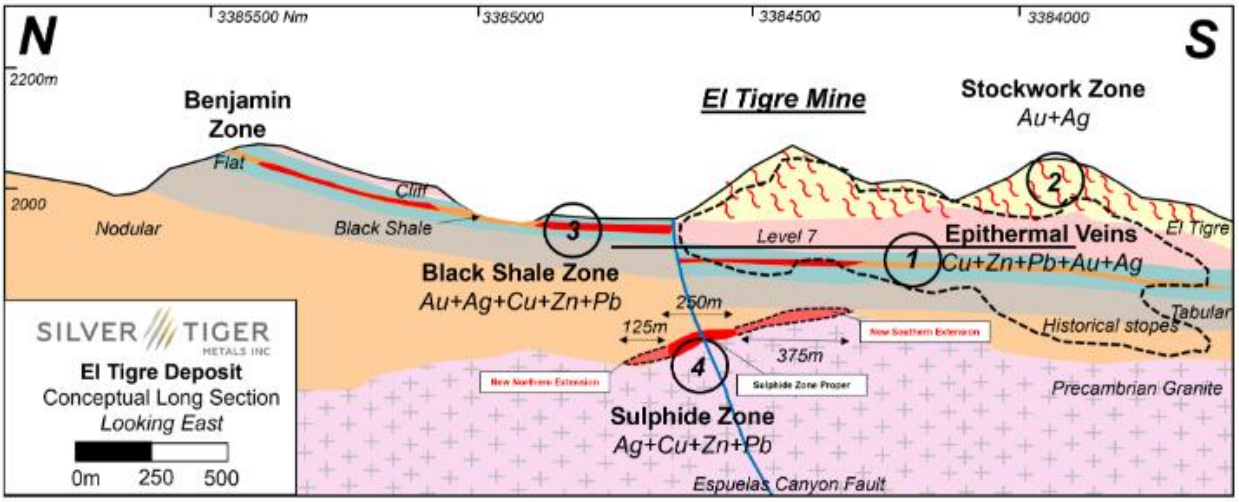
Source: *Black and Choquette (2013)*

FIGURE 7.5 MINERALIZATION STYLES AT EL TIGRE IN CONCEPTUAL VERTICAL CROSS-SECTION



Source: *Silver Tiger (August 27, 2024)*
Note: *Conceptual cross section showing the styles of silver-gold mineralization at El Tigre.*

FIGURE 7.6 MINERALIZATION STYLES AT EL TIGRE IN CONCEPTUAL LONGITUDINAL SECTION



Source: *Silver Tiger website (October 2024)*

Vein mineralization consists of quartz and varying proportions of zinc, iron, lead, copper, and silver sulphides with silicified or argillized fragments of host rock. Gold is associated with copper-silver sulphides. The mineralization occurs in discontinuous lenses of elongated, high-grade sulphides along the veins and as low-grade impregnations in the vein gangue material. A common feature of many of the mineralized bodies in the historical Mine was that they were much more extensive along strike than down-dip. Dilatancy was identified as one of the primary mineralization controls in the Mine and deflections of the vein gave rise to the characteristic horizontal elongation of the higher-grade mineralized bodies (Mishler, 1920). Intense alteration and fracturing of the brittle volcanic units along the veins host oxidized disseminated stockwork mineralization.

Metal zoning data collected during Anaconda's investigation suggest that the upper portions of the veins, which are at higher elevations on the Property (specifically on Gold Hill, where the original high-grade gold discovery was made), host bonanza-grade gold mineralization in discrete veins and disseminated lower-grade material in the altered stockwork zones.

The principal veins consist predominantly up to 80 to 90% gangue material, including silicified rock fragments, quartz, gouge, rock flour, clays and minor calcite, in order of abundance. The silicified fragments are angular to subangular and range in size from a few mm to 15 cm to 20 cm across. Larger blocks or slabs detached from the walls by faulting, occur in places and are criss-crossed by hairline fractures, with or without quartz or sulphide filling. Quartz occurs in lenses, bands, fragments, dissemination, and breccia matrix, and is the major gangue mineral in the vein. Rock flour, partially indurated, gouge, and clays occur throughout the vein in minor amounts as breccia matrix and fault linings. Minor calcite occurs in irregular veinlets and is locally associated with mineralized sulphides.

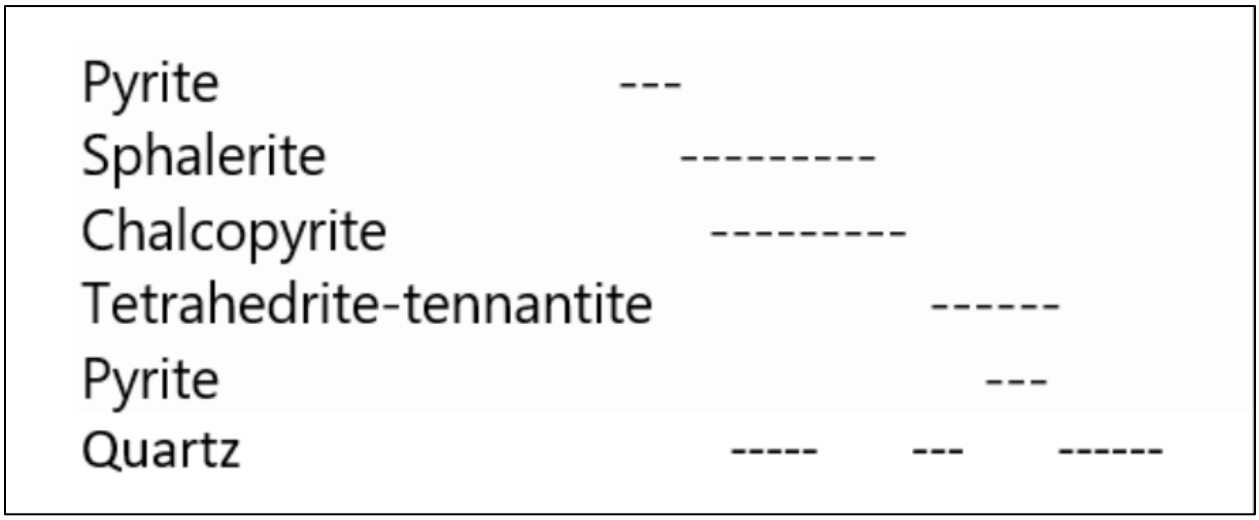
Mineralization consists, in order of abundance, of pyrite, sphalerite, galena, argentiferous galena, chalcopyrite, tetrahedrite, and covellite. Tetrahedrite occurs as its argentian variety, freibergite. Gold occurs in the native state as µm-sized specks, or as inclusions in galena and chalcopyrite.

Sulphides occur in small amounts in the veins, averaging 5 to 8%, although locally may reach 60% in banded lenses. Massive, coarse-grained, sphalerite and galena intergrowths are observed locally in those lenses, with subordinate amounts of coarse-grained chalcopyrite and pyrite. Tetrahedrite is associated mainly with chalcopyrite and, to a smaller extent, with the other sulphide phases.

Fine-grained argentiferous galena occurs associated with pyrite and quartz with little or no sphalerite. Pyrite occurs with quartz and hematite, or with other sulphides in lenses and in clusters or in strongly disseminated patches. Pyrite also fills numerous irregular veinlets in large rock fragments and slabs in the vein and in the wall rock. Quartz occurs in substantial amounts in all the occurrences noted above. A significant amount of sulphides occur as vein fragments and crushed material. Grain size varies from virtually pulverized to fragments ranging in size from a few mm to a few cm. Larger fragments preserve their textures, however, are subordinate in volume to crushed sulphides. Pulverized sulphides, mainly pyrite, occur along the walls of the vein. Sulphide dissemination is, except for pyrite, restricted to rock fragments or massive quartz in the vein. Minor drusy structures near the centre of the vein are typically lined with pyrite.

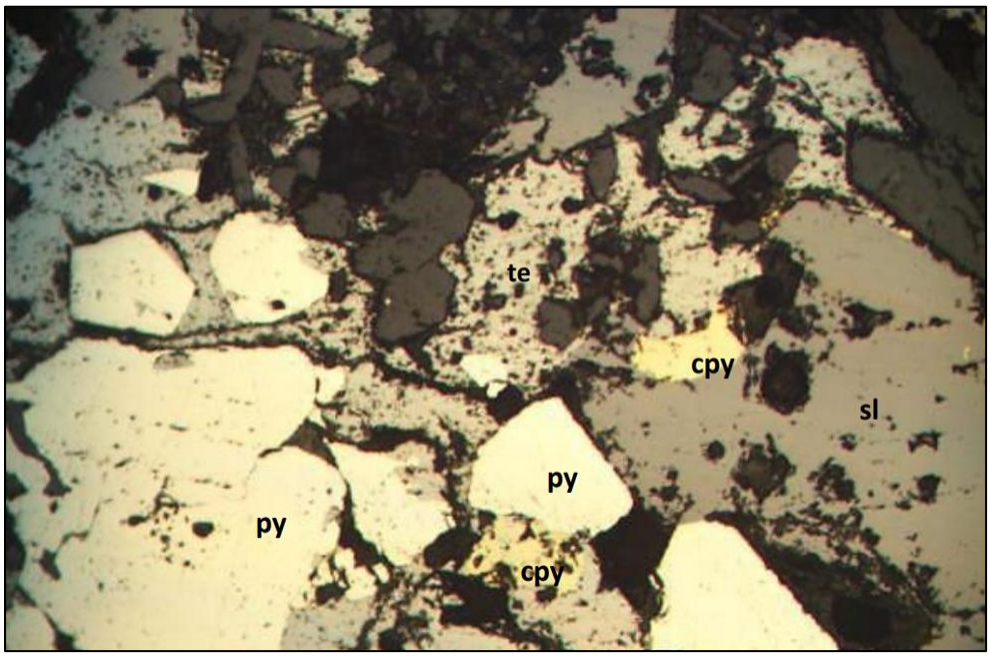
The sulphide mineralization was studied in reflected light and analyzed with a scanning electron microscope ("SEM") by Landin (2022). The mineralization in 16 drill core samples consisted mainly of sphalerite, galena, chalcopyrite, pyrite, and tetrahedrite-tennantite (Figure 7.7 to Figure 7.9). EDS analyses confirmed that tetrahedrite-tennantite and galena are the main silver-bearing minerals. In some samples, tetrahedrite-tennantite occurs including and cutting the other sulphide phases, which suggest that it was a relatively late-forming phase during mineralization.

FIGURE 7.7 TENTATIVE PARAGENETIC SEQUENCE



Source: *Landin (2022)*

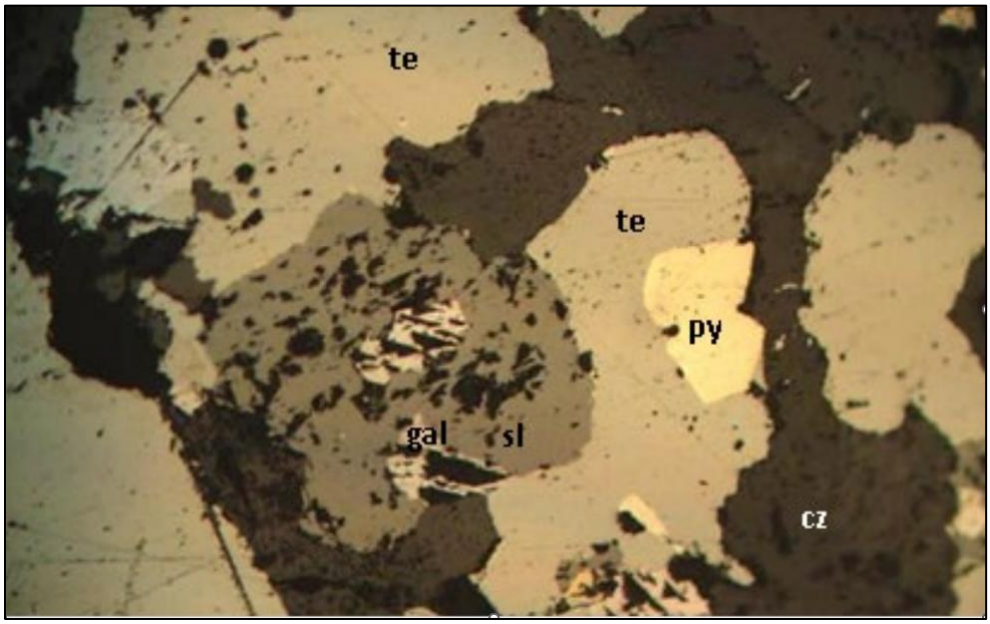
FIGURE 7.8 PHOTOMICROGRAPH OF TETRAHEDRITE AND OTHER SULPHIDES



Source: *Landin (2022)*

Figure 7.8 Description: *Photomicrograph of the sample ETC-66782 in reflected light showing the presence of pyrite (py), sphalerite (sl), tetrahedrite (te) and chalcopyrite (cpy, 0.1 mm) in contact with each other (10x magnification).*

FIGURE 7.9 PHOTOMICROGRAPH OF TETRAHEDRITE AND GALENA WITH OTHER SULPHIDES AND QUARTZ



Source: *Landin (2022)*

Figure 7.9 Description: *Photomicrograph of the sample ETC-68666 showing in reflected light the presence of sphalerite (sl) of 0.7 mm in diameter, assimilating galena crystals (gal) with tetrahedrite in contact and lower pyrite (py). Note the presence of quartz (cz) filled gaps between the sulphide phases (10x magnification).*

7.4.1 Geological Controls

Mineralization in the El Tigre District is controlled almost entirely by secondary structural features: faults and their concomitant breccias, fissures, fractures, and fracture zones. All the veins described in the district host mineralization in one or more of the structural features listed above. Lithologies of the volcanic sequence appear to have had little or no chemical control on mineralization. On the other hand, the various physical properties of volcanic sequence have influenced the nature and extent of openings available for mineralization.

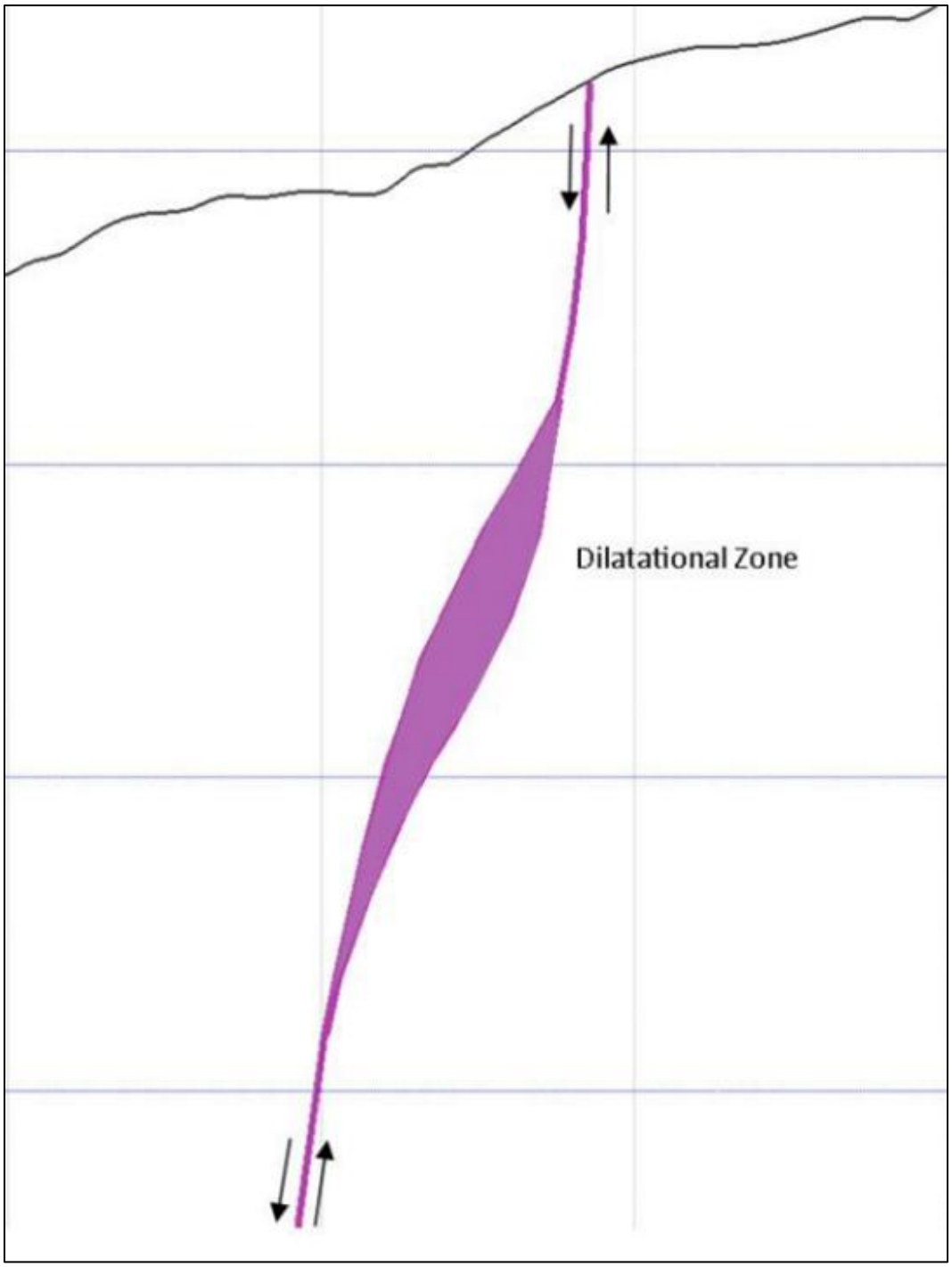
A structural analysis of the El Tigre Vein has assisted in understanding the structure and its relationship with mineralized shoot locations. The El Tigre Vein, developed over 1,950 m laterally and 450 m vertically, is a composite structure that comprises two alternating sets of faults with varying dips. The main set consists of three long segments striking 8° to 342°, which are interconnected by two shorter segments striking 3° to 358°.

The dislocations, or variations in strike, occur only within the lower-level workings of the Mine. The vein is relatively consistent at a strike of ~352° over the entire length of Mine levels two and three (1,380 m). Previously mined mineralized shoots, defined as "a pipelike, ribbonlike, or chimney-like mass of mineralization within a deposit (usually a vein) representing the more valuable part of the deposit", occur largely on the north-northwest portions of the vein, in the lower levels of the historical mine workings, where dilation prior to mineralization resulted in greater vein width and increased potential for fluid flow. Where strike of the vein deflects to the north, in the upper levels of the Mine, vein width (and width of mineralization) decreases significantly, representing 'tighter' portions of the vein structure that likely prevented the flow of mineralizing fluids.

A favourable portion of the vein in cross section looks like a wide asymmetric curve with shallow dip at the top and gradually increasing to become vertical at the bottom (Figure 7.10). Above the curved portion of the vein, a vertical segment contains less mineralization. The greatest width and distribution of high-grade material occurs in crest of the curve.

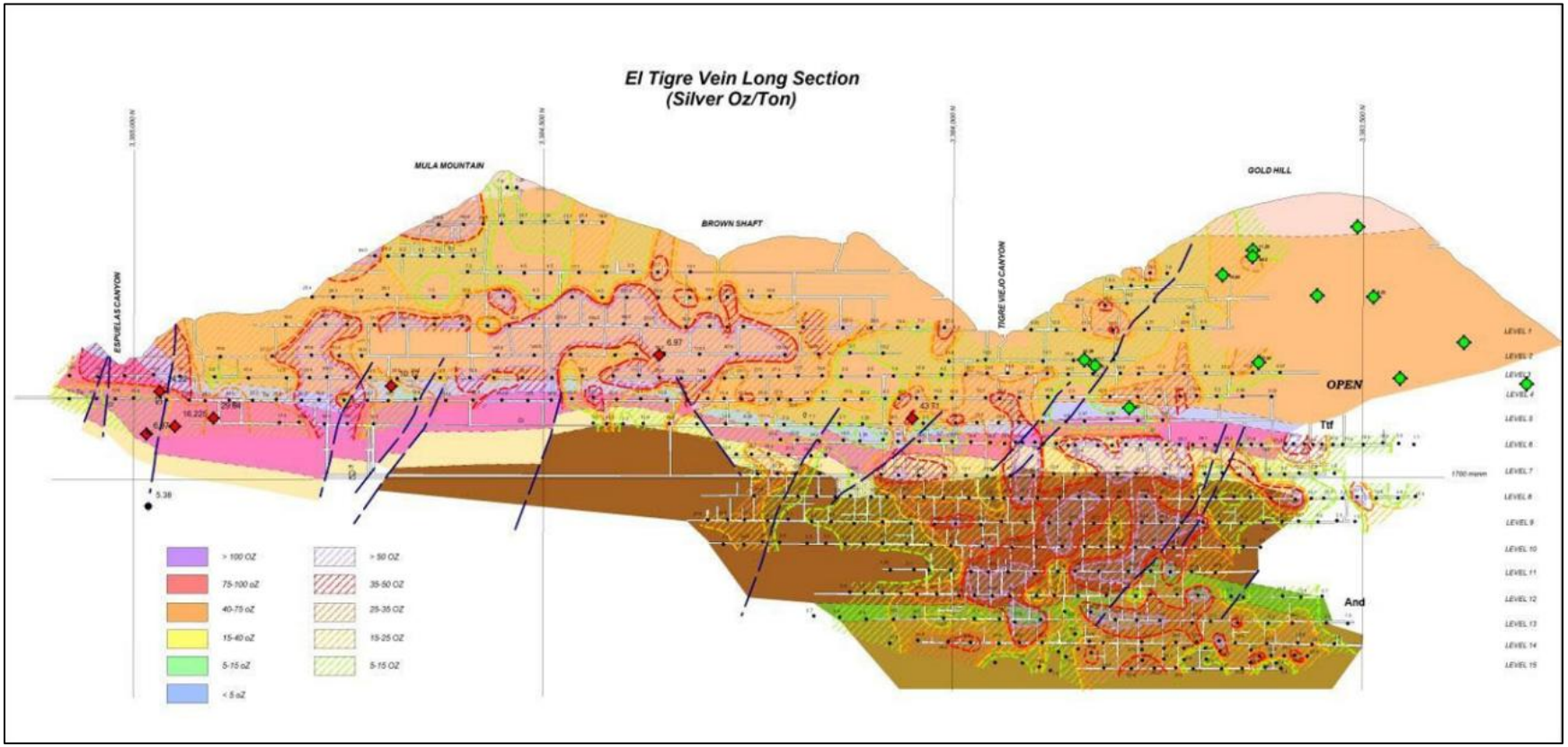
Mineralized shoots occur in El Tigre Vein in the entire lower volcanic series within the Nodular to Tigre Formations. There appears to be no definite correlation of high-grade mineralization with one particular rock unit that could be interpreted as chemical control (Figure 7.11). However, the character of the wall rock has affected the local shape and extent of the mineralized shoots. The structure is fairly uniform in the Tigre Formation and extremely variable in Tuff Formation (Mishler, 1920). Mineralization occurs in two or more veins in the brittle Cliff Formation, whereas the more ductile Flat Formation hosts irregular veins and widths.

FIGURE 7.10 CONCEPTUAL VERTICAL CROSS-SECTION OF A TYPICAL EL TIGRE DILATION ZONE



Source: *Black and Choquette (2013)*

FIGURE 7.11 ANACONDA GEOLOGICAL LONGITUDINAL PROJECTION OF THE EL TIGRE VEINS



Source: *Black and Choquette (2013)*
Looking East

7.4.2 Alteration

Adularia replacement, minor silicification, argillization and propylitization are alteration styles that affect the wall rocks of the veins of the district. Although there is a general alteration zoning pattern outward from the vein in the order listed above, the distribution and width of alteration types appear to be controlled by the nature of the host rock.

In the level four mine area at the northern end of the southern vein system, the Cliff Formation stands out prominently, due to the intense adularization of the rock. In this area, veins containing quartz and mineralization show pink adularia rims on rock fragments that have also been adularized. In the El Tigre Vein, evidence of some intense silicification is found adjacent to vein.

Farther to the south along the vein system, adularization declines and a broad argillic halo becomes evident. The internal character of the veins also changes as mineralization is found in crushed host rock and minor quartz vein material. Oxidation becomes dominant since the rocks are broken and brecciated.

Fine-grained pyritization is widespread and stronger immediately adjacent to the veins. The complete alteration assemblage is found in silicified rock fragments inside the vein. Some of the fine-grained silicification is due to adularia flooding of a receptive rock type. Argillization occurs as wide, bleached envelopes around the veins, and consists of illite, kaolinite, and montmorillonite.

Propylitization is typically observed outside the argillic zones, although it may occur adjacent to the veins. Propylitic alteration consists of a mixture of quartz, chlorite, calcite, sericite, and illite and gives the rock a characteristic greenish light-grey colour. Medium-grained pyrite, slightly coarser than in the silicified zone, invariably accompanies both argillic and propylitic alteration.

8.0 DEPOSIT TYPES

This section is summarized from Section 8 from Micon (2023) and has been updated where applicable.

Epithermal systems may be classified as high, intermediate, and low sulphidation styles. They are characterized by the sulphidation state of the hypogene sulphide mineral assemblage, and show general relations in volcano-tectonic setting, precious and base metal content, igneous rock association, proximal hypogene alteration, and sulphide abundance (Sillitoe and Hedenquist, 2003).

The veins at El Tigre closely resemble those that form quartz-adularia, low sulphidation epithermal deposits. Epithermal deposits, as classically defined, are the products of igneous-related hydrothermal activity at shallow depths and low temperatures, with deposition normally taking place within ~1 km of the surface in the temperature range of 50° to 300°C. Most deposits are in the form of quartz veining and related stockworks and breccias. These open-space fillings are common and, in most deposits, are the dominant mode of mineralization.

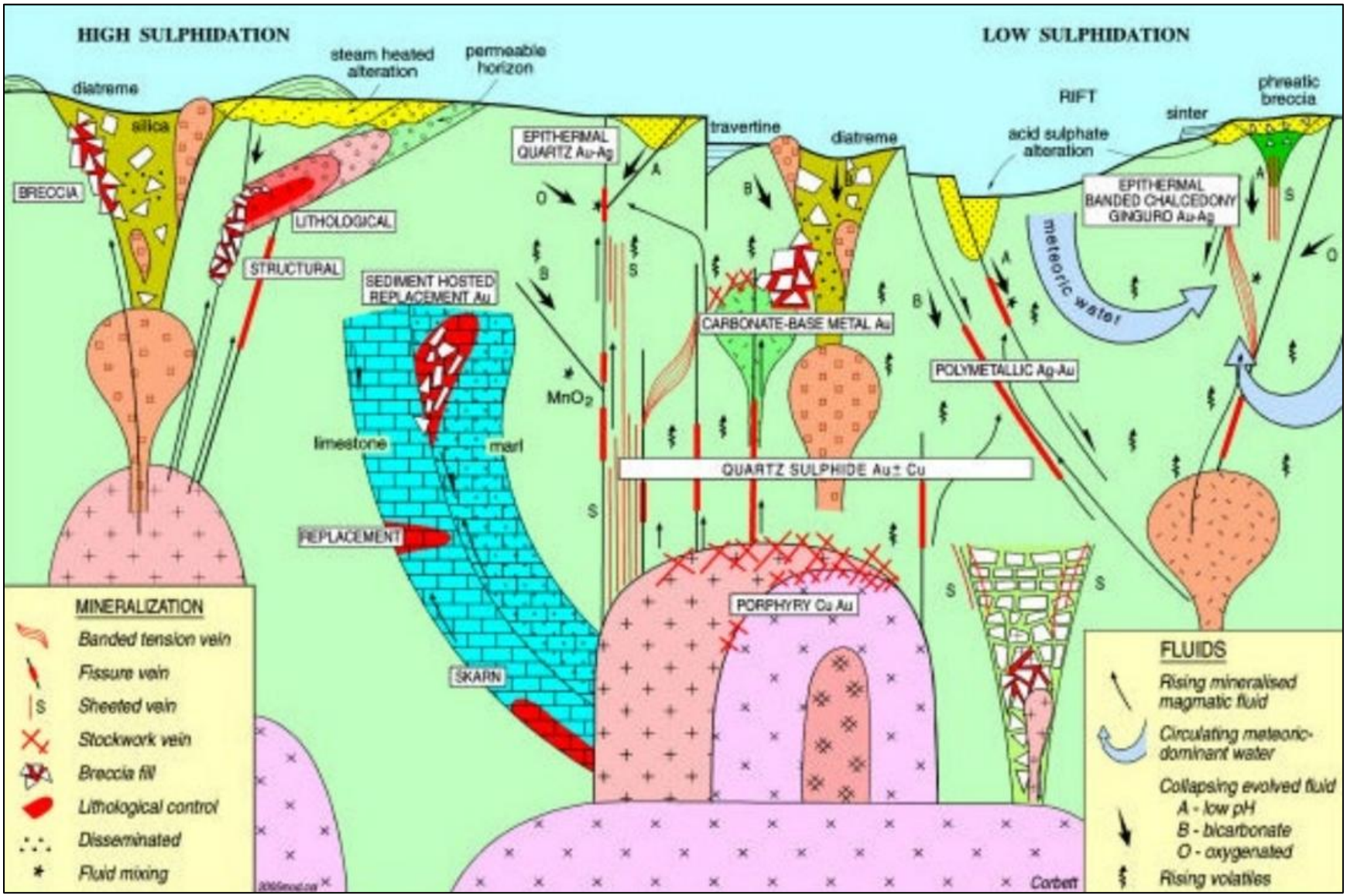
Drusy cavities, cockade structures, crustifications, and symmetrical banding are generally conspicuous. Colloform textures characteristic of epithermal environments presumably reflect relatively low temperatures (e.g., shallow depths) and hydrothermal fluid circulation through open spaces formed by mechanical anisotropies, such as networks of fractures, contacts between units with dissimilar mechanical properties, and (or) cross-cutting structures, intrusive bodies and shears (summarized from Guilbert and Park, 1986).

There are two types or styles of silver and gold mineralization found in the El Tigre area. The first and best-known are the fissure veins that host silver, lead, zinc, copper, and gold mineralization within a narrow, 5.3 km long, north-trending belt. The second is the undeveloped low-grade stockwork halo near the veins. This mineralization is associated with fractured volcanic rocks and occurs as stockwork veinlets containing minor quartz, pyrite, chalcopyrite, sphalerite and galena. These systems generally have basic to neutral pH fluids enriched in potassium and silica. Very little evidence of boiling has been found in the El Tigre Vein, as it appears that the quartz and sulphides were deposited in a passive, low energy environment.

The veins occur along fissures that generally dip steeply to the west, although steep dip reversals to the east occur in some sections of the veins. Vein mineralization consists of quartz and varying proportions of zinc, iron, lead, copper, and silver sulphides; silicified/adularized or argillized fragments of host rock are generally part of the vein material. Gold in minor amounts is associated with copper-silver sulphides. The mineralization occurs in discontinuous lenses of high-grade sulphides along the veins and as low-grade impregnations in the vein material.

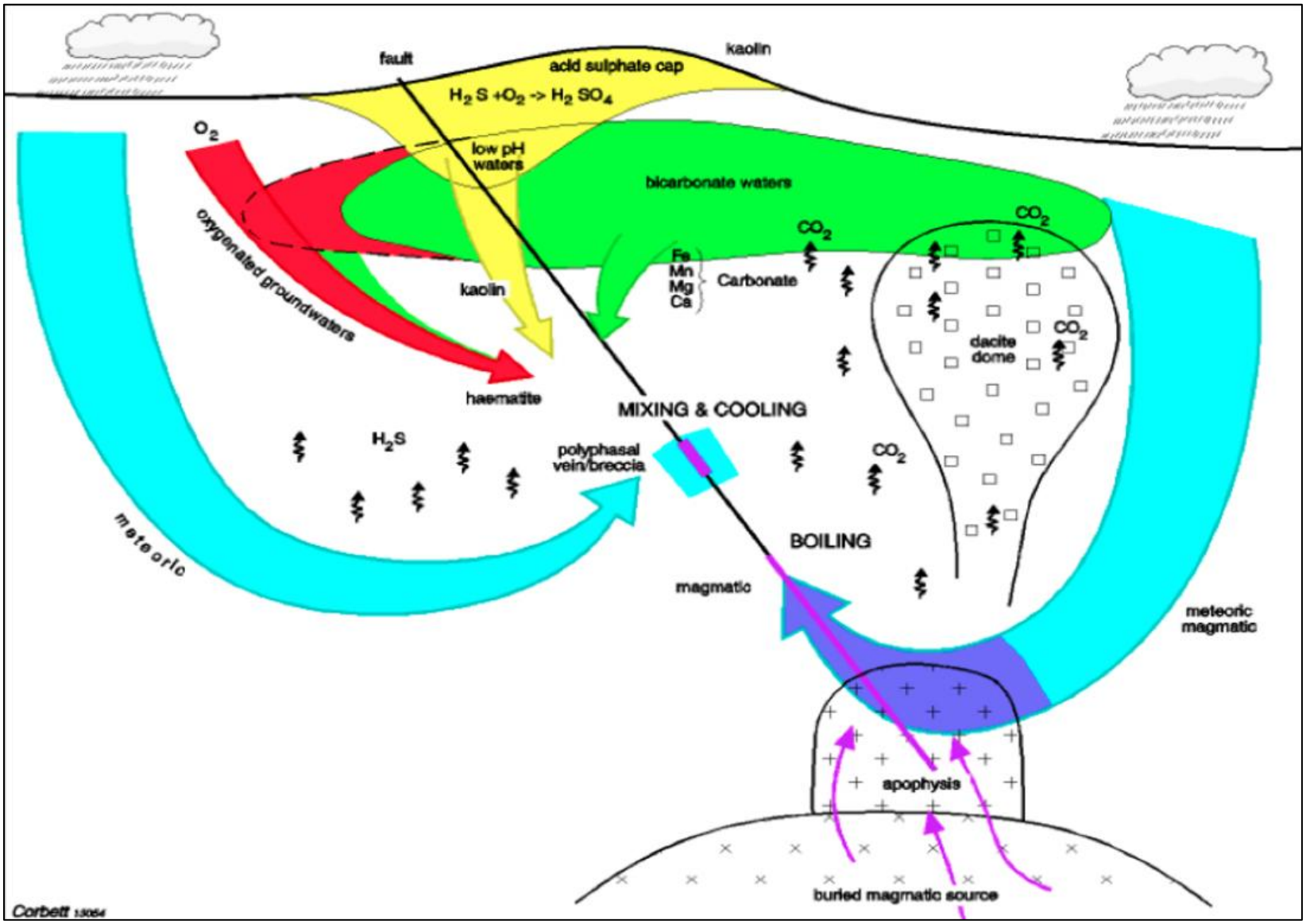
Schematic models of epithermal systems are shown in Figure 8.1 and Figure 8.2.

FIGURE 8.1 EPITHERMAL MINERALIZATION MODEL



Source: *Corbett (2009)*

FIGURE 8.2 MODEL FOR LOW-SULPHIDATION EPITHERMAL AU-AG VEIN FORMATION



Source: *Corbett (2009)*

9.0 EXPLORATION

This section is summarized from Micon (2023) Section 9 and has been updated where applicable. The following section addresses the exploration work completed by Silver Tiger, beginning in 2020.

All drill hole survey and assay values are expressed in metric units, with grid coordinates reported in Universal Transverse Mercator (UTM) coordinate space relative to WGS 1984 Zone 12.

9.1 SILVER TIGER EXPLORATION PROGRAM

In the summer of 2020, Silver Tiger's exploration included a channel sampling program of historical underground exploration drifts and surface sampling located on the 3 km of vein extensions that outcrop north of the historical El Tigre Mine. The areas of focus include the Caleigh, Canon Combination, Protectora, and Fundadora Veins (Figure 9.1).

9.1.1 Underground Channel Sampling

The 2020 channel sampling program was planned to generate additional drill targets and followed-up on the success of the underground channel sampling completed in the same vein extensions in 2019, which returned multiple high-grade values.

The 2020 sampling program returned multiple high-grade values, including:

- Fundadora Vein, ~2 km north of the historical El Tigre Mine, channel sample ETU-1042 returned 3,064 g/t Ag and 4.44 g/t Au, or 3,397 g/t AgEq, over 0.5 m (true thickness); channel sample ETU-1026 returned 2,500 g/t Ag and 8.6 g/t Au, or 3,145 g/t AgEq, over 0.4 m (true thickness); and channel sample ETU-1016 returned 606 g/t Ag and 22.3 g/t Au, or 2,279 g/t AgEq, over 1.2 m (true thickness);

- Protectora Vein, ~1.6 km north of the El Tigre Mine, underground channel sample ETU-1022 returned 2,283 g/t Ag and 32.99 g/t Au, or 2,474 g/t AgEq, over 0.4 m (true thickness);

- Caleigh Vein, ~1.6 km north of the El Tigre Mine, sample ETU-1025 returned 1,679 g/t Ag and 26.65 g/t Au, or 1,999 g/t AgEq, across 0.4 m (true thickness); and

- Aquilas Norte area, ~2.5 km north of the El Tigre Mine, channel sample ETU-1038 returned 1,709 g/t Ag and 4.71 g/t Au, or 2,062 g/t AgEq, over 0.3 m (true thickness). Channel sample ETU-1036 returned 1,843 g/t Ag and 1.37 g/t Au, or 1,946 g/t AgEq, over 0.3 m (true thickness).

Note that the AgEq values are based on a gold to silver price ratio of 75:1 (Au:Ag). A sample location map showing these assay results is provided below (Figure 9.1), along with a listing of the significant underground channel sample assay results (Table 9.1).

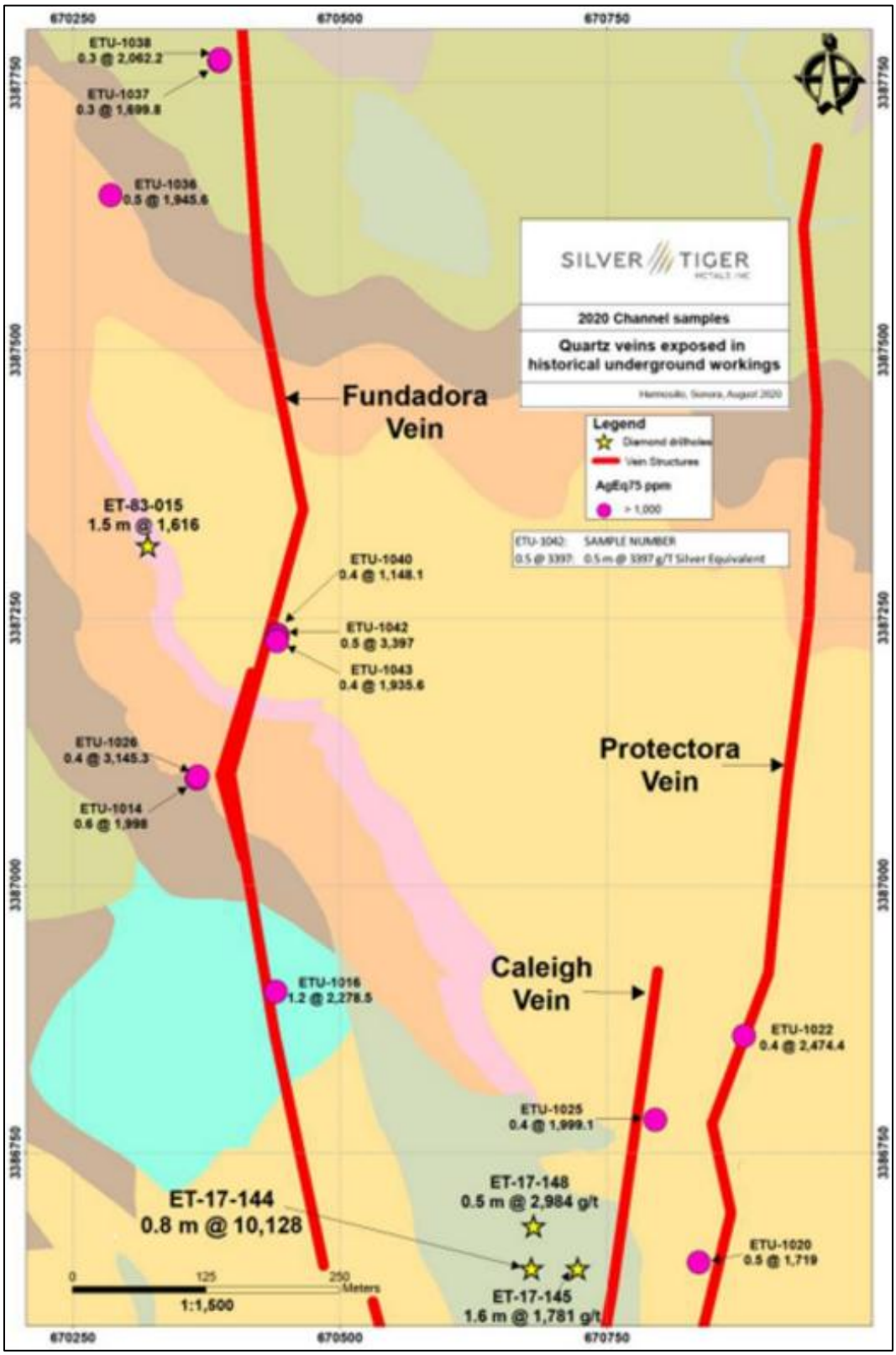
Silver Tiger's exploration focus for the remainder of 2020 and in 2021 through 2023 was drilling, the results of which are described in Section 10 of this Technical Report.

TABLE 9.1
2020 SIGNIFICANT CHANNEL SAMPLE ASSAY RESULTS

Vein / Area	Sample ID	Length (m)	Easting*	Northing*	Elevation (masl)	Ag (g/t)	Au (g/t)	AgEq (g/t)	AuEq (g/t)
Fundadora	ETU-1042	0.5	670,441	3,387,229	2,082	3,064	4.44	3,397	45.29
Fundadora	ETU-1026	0.4	670,367	3,387,101	1,903	2,500	8.60	3,145	41.94
Protectora	ETU-1022	0.4	670,878	3,386,859	2,078	2,283	2.55	2,474	32.99
Fundadora	ETU-1016	1.2	670,440	3,386,901	1,943	606	22.30	2,279	30.38
Aguilas Norte	ETU-1038	0.3	670,387	3,387,772	1,822	1,709	4.71	2,062	27.50
Caleigh	ETU-1025	0.4	670,795	3,386,781	2,023	1,679	4.27	1,999	26.66
Fundadora	ETU-1014	0.6	670,366	3,387,100	1,903	768	16.40	1,998	26.64
Aguilas Norte	ETU-1036	0.5	670,285	3,387,645	1,935	1,843	1.37	1,946	25.94
Fundadora	ETU-1043	0.4	670,441	3,387,228	2,081	1,585	4.68	1,936	25.81
Protectora	ETU-1020	0.5	670,835	3,386,648	2,040	729	13.20	1,719	22.92
Aguilas Norte	ETU-1037	0.3	670,386	3,387,770	1,822	1,277	5.64	1,700	22.67
Fundadora	ETU-1039	0.4	670,441	3,387,234	2,082	988	5.93	1,433	19.10
Fundadora	ETU-1040	0.4	670,441	3,387,233	2,082	815	4.44	1,148	15.31

Source: *Silver Tiger (August 12, 2020)*
Notes: ** coordinates are in WGS 1984 UTM Zone 12.*
ETU = underground channel samples,
Silver Equivalent ("AgEq") ratio based on gold to silver price ratio of 75:1 (Au:Ag).

FIGURE 9.1 2020 CHANNEL SAMPLES AND ASSAY HIGHLIGHTS LOCATION MAP



Source: *Silver Tiger (August 12, 2020)*
Note: *The assay results shown are AgEq values, which are based on a gold to silver price ratio of 75:1 (Au:Ag).*

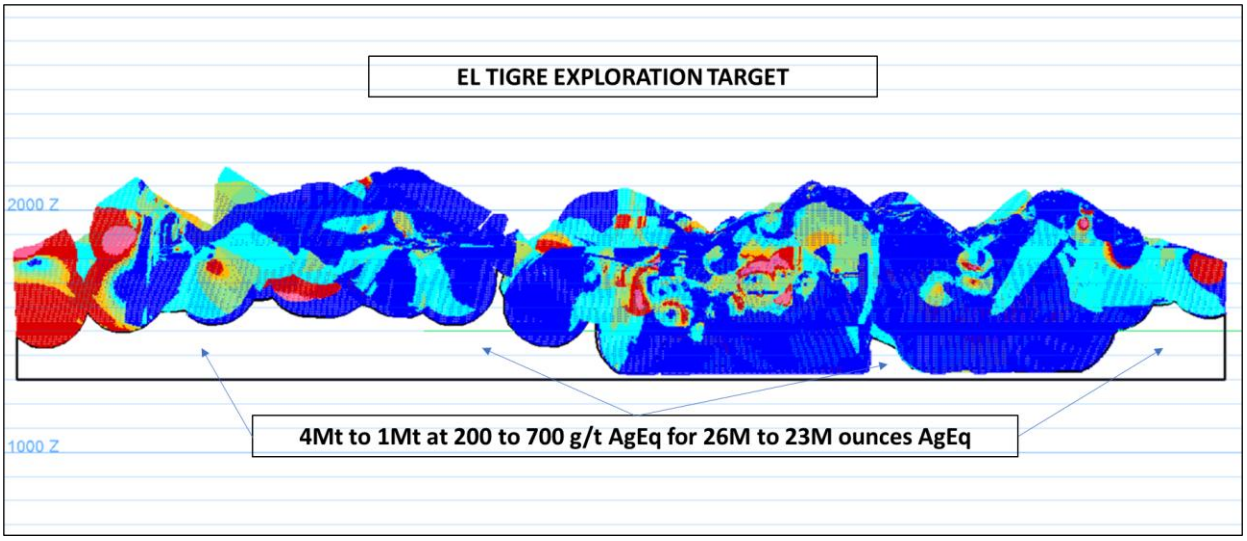
9.2 EXPLORATION POTENTIAL

High-grade mineralization at the El Tigre Project is open down-dip and along strike. In addition to the exploration work completed, and based on widely spaced drill holes and average width, length and depth geometry, the Authors established that the El Tigre mineral deposits contain an additional Exploration Target as follows:

- 1 to 4 million tonnes at 200 to 700 g/t AgEq for 23 to 26 Moz AgEq.

The Exploration Target is shown in Figure 9.2 (looking east).

FIGURE 9.2 EXPLORATION TARGET BELOW THE UPDATED MINERAL RESOURCES



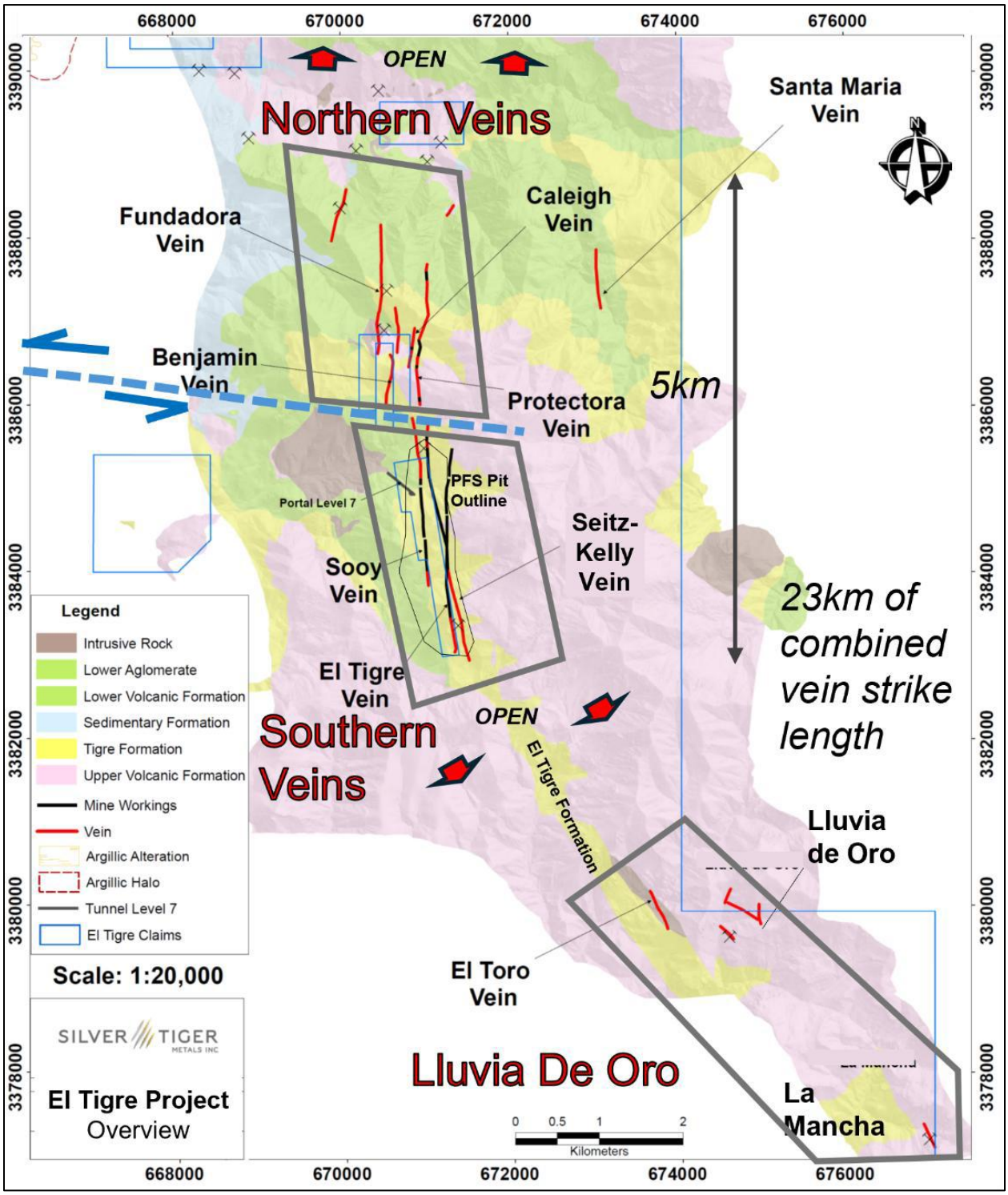
Source: *P&E (2025)*
Looking East

The potential quantities and grades of the Exploration Targets are conceptual in nature. There has been insufficient work done by a Qualified Person to define these estimates as Mineral Resources. The Company is not treating these estimates as Mineral Resources, and readers should not place undue reliance on these estimates. Even with additional work, there is no certainty that the estimates will be classified as Mineral Resources. In addition, there is no certainty that these estimates will ever prove to be economically recoverable.

9.3 2025 PRELIMINARY EXPLORATION RESULTS

In a Company press release dated August 13, 2025, Silver Tiger announced preliminary results from its 2025 summer exploration program on the prospective Northern Veins, located 2 km north of the historical El Tigre Mine and outlined initial exploration plans to test several undrilled greenfield areas on the Southern Veins (Figure 9.3). Current exploration work consists of mapping and sampling surface and underground workings to define drill targets for testing in Q4 2025.

FIGURE 9.3 LOCATION OF THE NORTHERN VEINS AND THE SOUTHERN VEINS AT EL TIGRE



Source: *Silver Tiger press release dated August 13, 2025.*

The Company's exploration objectives for 2025 are three-fold:

1. Commence drilling on the Northern Veins in Q4 2025;
2. Generate drill targets on the Southern Veins at Lluvia de Oro, La Mancha and El Toro through analysis of old mining works; and
3. With 25 km of mapped vein strike complete, which is 50% of the Project area, the plan is to increase mapping coverage to 70%.

9.3.1 Northern Vein Area

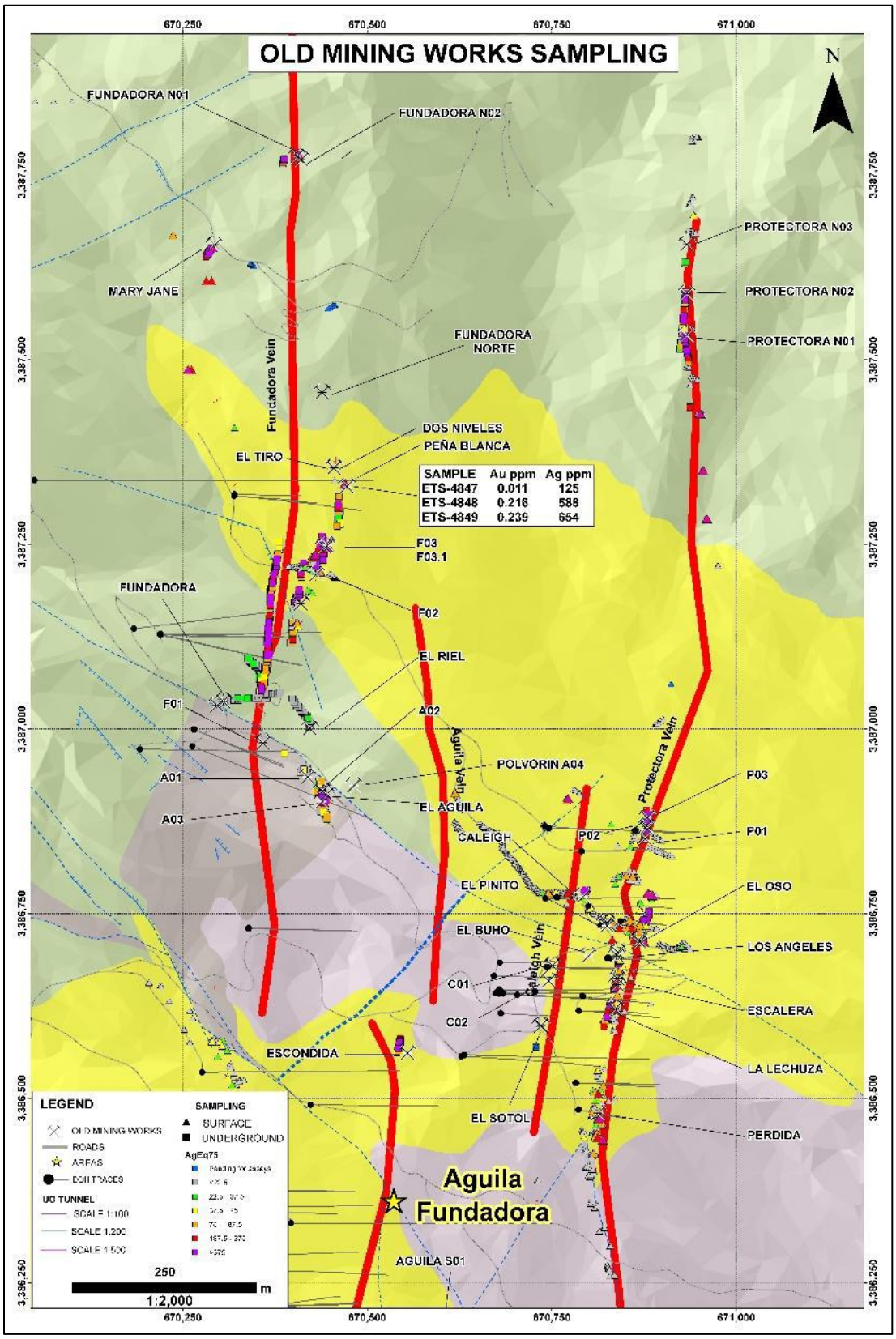
The Northern Vein Area has over 10 km of combined strike length confirmed by surface mapping and sampling, drilling and underground workings, mainly on the Fundadora, Protectora and Caleigh Veins. This area is considered brownfields with 15% of the mapped strike of veins being drilled. These veins are the faulted offset of the El Tigre Mine Veins and exhibit the same structure, mineralogy, grades, orientations, strike length and width (1 to 2 m) as the El Tigre Veins.

Exploration crews have begun to locate, catalog, map and sample historical adits and drifts in the Northern Veins (Figure 9.4). This work is aimed at confirming the geological similarities of the Northern Vein mineralized systems to the well-established El Tigre, Seitz-Kelly and Sooy Veins. When assessment of the historical workings is complete all adits and drifts will be scanned with a LiDAR-capable drone. For the remainder of the summer, Silver Tiger will continue to define targets to drill in Q4 2025.

After surveying the tunnels, the geologists mapped the veins and collected chip samples across the veins at intervals between 3 to 5 m along the length of the tunnel. The samples were chipped with a rock hammer across the face of the vein exposed in the tunnel. The minimum length of the samples was 0.50 m, and the samples weighed approximately 3 to 5 kg. Sampling stopped at either the end of the tunnel or where access was restricted by ground conditions. Of the 110 chip samples collected, the chip sample assay results received to date are shown below (Figure 9.4 and Table 9.2).

Reconnaissance drilling was completed on the Northern Veins by the Company in 2017 and 2020 (Table 9.3), before focus shifted to development of the El Tigre Mine Veins. Significant mineralized veins were consistently intercepted in the North (Table 9.4), consistent with the continuity and grade observed in the underground historical workings. Anaconda (circa. 1982-1983) completed a 250 m exploration drift on the Northern Veins (Figures 9.5 and 9.6).

FIGURE 9.4 2025 SAMPLING OF HISTORICAL WORKINGS IN NORTHERN VEIN AREA



Source: *Silver Tiger press release dated August 13, 2025*

TABLE 9.2 SIGNIFICANT CHANNEL SAMPLE ASSAY INTERVALS IN THE NORTHERN VEINS AREA									
Vein	**Channel ID**	**Samples**	**From (m)**	**To (m)**	**Interval[1] (m)**	**Sample No.**	**Au (g/t)**	**Ag (g/t)**	**AgEq[2] (g/t)**
Fundadora	ETL-2078	Composite: ETS-4847/4848/4849	0	1.6	1.6	3	0.15	464	476
Fundadora	ETL-2078	Including: ETS-4849	1.1	1.6	0.5	1	0.24	654	672
Fundadora	ETL-2079	Composite: ETS-4853/4853	0	0.8	0.8	2	0.40	95	118

Source: *Silver Tiger press release dated August 13, 2025*
Notes: *1) Not true thickness; 2) Silver Equivalent ("AgEq") ratio is based on a silver to gold price ratio of 75:1 (Au:Ag).*

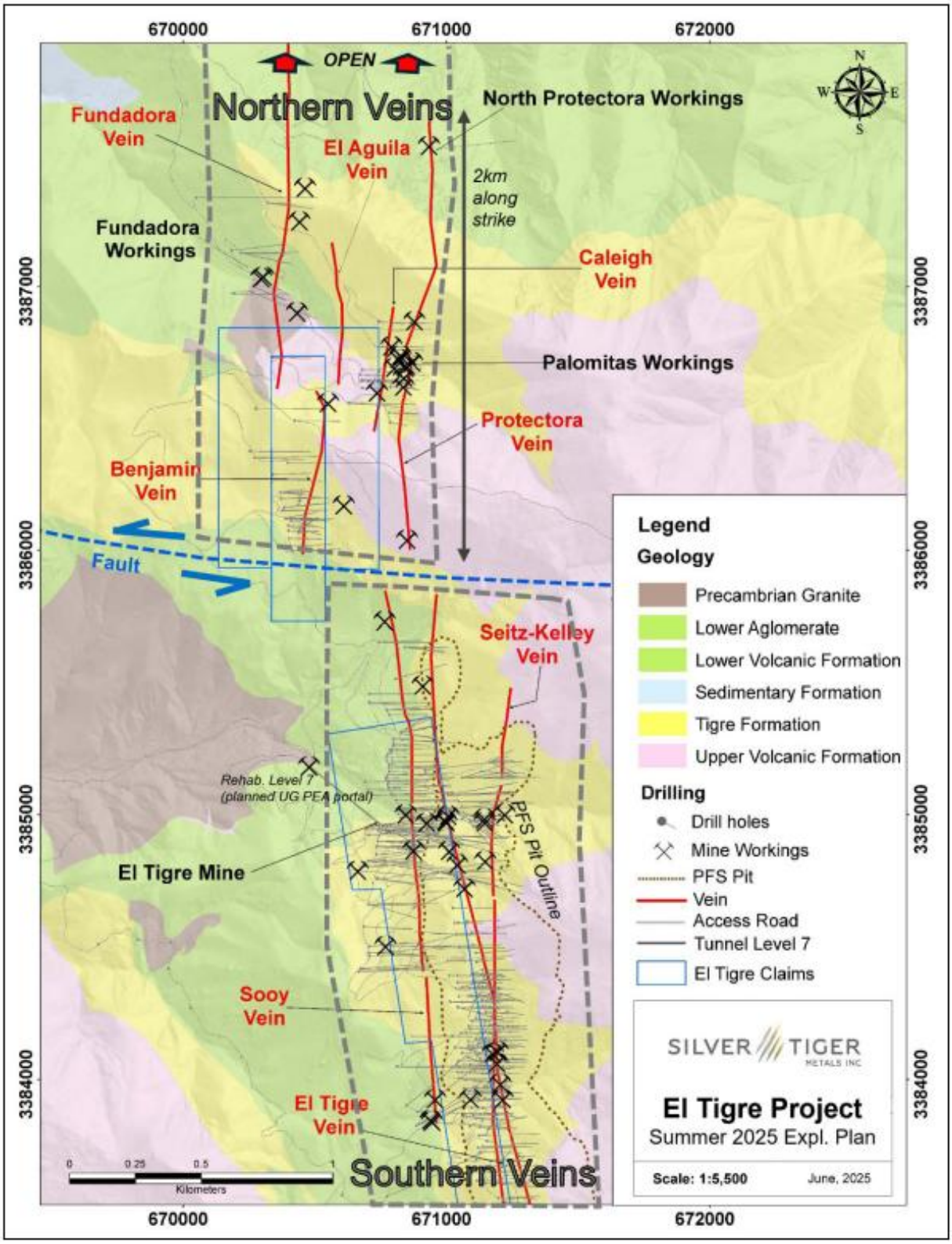
TABLE 9.3 DRILL HOLE AND CHANNEL LOCATIONS FOR SIGNIFICANT INTERCEPTS - NORTHERN VEINS						
Drill Hole/ Channel	**Easting***	**Northing***	**Elevation (m)**	**Azimuth (°)**	**Dip (°)**	**Length (m)**
ET-83-015	670,320	3,387,318	2,055	91	-61	187.7
ET-17-144	670,679	3,386,642	2,002	90	-45	224.0
ET-20-158	670,679	3,386,647	1,999	90	-45	131.2
ET-20-163	670,828	3,386,690	2,020	90	-25	61.0
ET-20-164	670,826	3,386,690	2,020	90	-45	114.4
ET-20-193	670,390	3,386,121	1,936	90	-50	186.1
ETL-2078	670,471	3,387,332	2,124	270	-14	2.7
ETL-2079	670,471	3,387,330	2,123	230	15	0.8

Source: *Silver Tiger press release dated August 13, 2025*
Note: ** coordinates are in WGS 1984 UTM Zone 12.*

TABLE 9.4 DRILL HOLE LOCATIONS FOR THE SIGNIFICANT INTERCEPTS IN THE NORTHERN VEINS AREA							
Drill Hole ID	**Comment**	**From (m)**	**To (m)**	**Length[1] (m)**	**Au (g/t)**	**Ag (g/t)**	**AgEq[2] (g/t)**
ET-83-015	Fundadora Vein	172.4	173.5	1.1	14.38	537.7	1,616.1
ET-83-015	Including	172.4	173.0	0.60	24.40	913.0	2,743.0
ET-17-144	Caleigh Vein	88.3	91.4	3.15	10.10	1,990.9	2,748.4
ET-17-144	Including	88.3	89.1	0.85	37.20	7,338.9	10,128.9
ET-20-158	Caleigh Vein	89.1	91.2	2.10	1.46	280.5	390.1
ET-20-158	Including	90.0	90.7	0.70	4.09	815.0	1,121.6
ET-20-163	Protectora Vein	10.0	18.4	8.40	0.42	165.5	197.1
ET-20-163	Including	16.9	17.4	0.50	3.56	1,782.0	2,049.1
ET-20-164	Protectora Vein	17.0	28.4	11.40	0.66	87.1	136.8
ET-20-164	Including	17.0	17.5	0.50	10.50	805.0	1,592.5
ET-20-193	Benjamin Vein	114.0	119.5	5.50	0.08	726.1	732.2
ET-20-193	Including	116.5	119.5	3.00	0.09	1,303.2	1,310.1

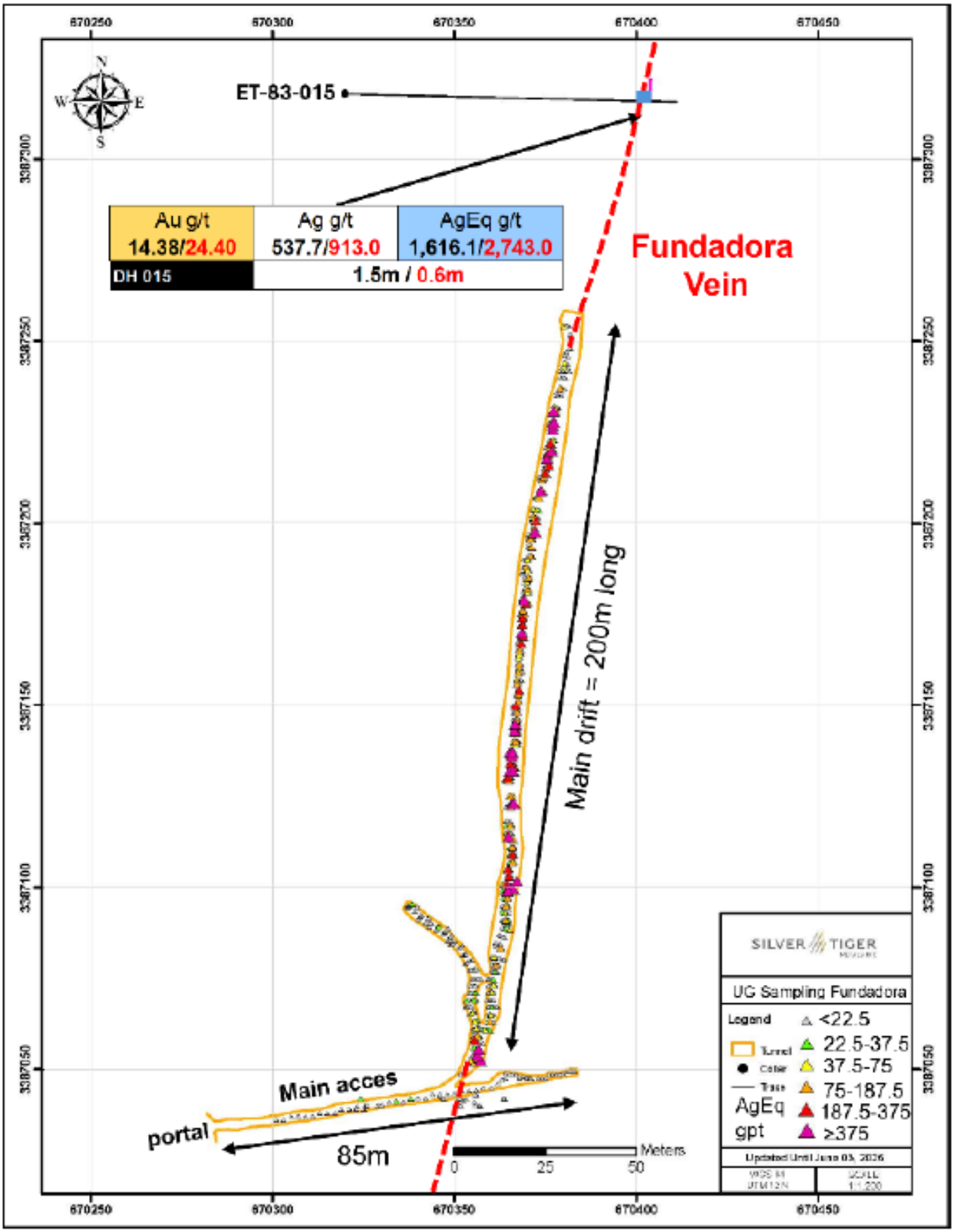
Source: *Silver Tiger press release dated August 13, 2025*
Notes: *1. Not true thickness; 2. Silver Equivalent ("AgEq') ratio is based on a silver to gold price ratio of 75:1 (Au:Ag).*

FIGURE 9.5 PLAN VIEW OF THE NORTHERN VEINS AND SOUTHERN VEINS AREAS OF THE EL TIGRE PROJECT



Source: *Silver Tiger press release dated August 13, 2025*

FIGURE 9.6 PLAN VIEW OF HISTORICAL UNDERGROUND SAMPLING AND MAPPING AT THE FUNDADORA WORKINGS



Source: *Silver Tiger press release dated August 13, 2025*

9.3.1.1 Caleigh Vein

Discovered in 2020 by the Company, the Caleigh Vein is the highest grade of the Northern Veins. For example, drill core sampling of diamond drill hole ET-17-144 returned assays at 10,129 g/t AgEq over 0.85 m (Table 9.4). The Caleigh Vein is mapped on surface for 1.2 km with only 25% of the vein drill tested.

9.3.1.2 Benjamin Vein and Black Shale

The newly discovery Benjamin Vein has been mapped on surface for 1 km, with only 25% drill tested. Drill hole ET-20-193 returned assays at 1,310 g/t AgEq over 3.0 m (Table 9.4). The Benjamin Vein also hosts significant Black Shale style mineralization that is observed in the El Tigre Mine Veins.

9.3.1.2 Fundadora Vein

This vein was mined intermittently during the time of the El Tigre Mine (circa. 1892 to 1930), and therefore hosts many small workings, including a 250+ m exploration drift constructed by Anaconda Mining in 1982 (Figure 9.6). Drilling assays confirm the high grade observed in channel sampling during a 2020 program. Diamond drill hole ET-83-015 returned assays of 2,743 g/t AgEq over 0.6 m (Table 9.4). The Fundadora Vein has been mapped over 2 km on surface with only 15% of the vein drill tested.

9.3.1.3 Protectora Vein

The Protectora Vein extends along strike for 2 km and from its position, dip, and strike it can be considered the north extension of the Sooy Vein. This vein has only been drill tested for 15% of the mapped strike length. Like the Fundadora Vein, Protectora was mined historically as the Palomita Mine and the North Tigre Mine. Drilling assays confirm the high-grade observed in channel sampling during the 2020 program. Diamond drill hole ET-20-163 returned assays of 2,049 g/t AqEq over 0.5 m (Table 9.4).

9.3.2 Southern Vein Area

The Southern Vein area, which includes Lluvia de Oro, La Mancha and El Toro, is considered greenfields with veins striking for over 10 km and no drilling ever completed on this area. This prospective area features historical workings and high-grade channel sampling. The El Toro, Lluvia de Oro and La Mancha Veins are hosted in the El Tigre Formation, a permissive formation that is the host for the Stockwork Zone, which is the basis of Silver Tiger's 2024 PFS (Company press release dated October 22, 2024).

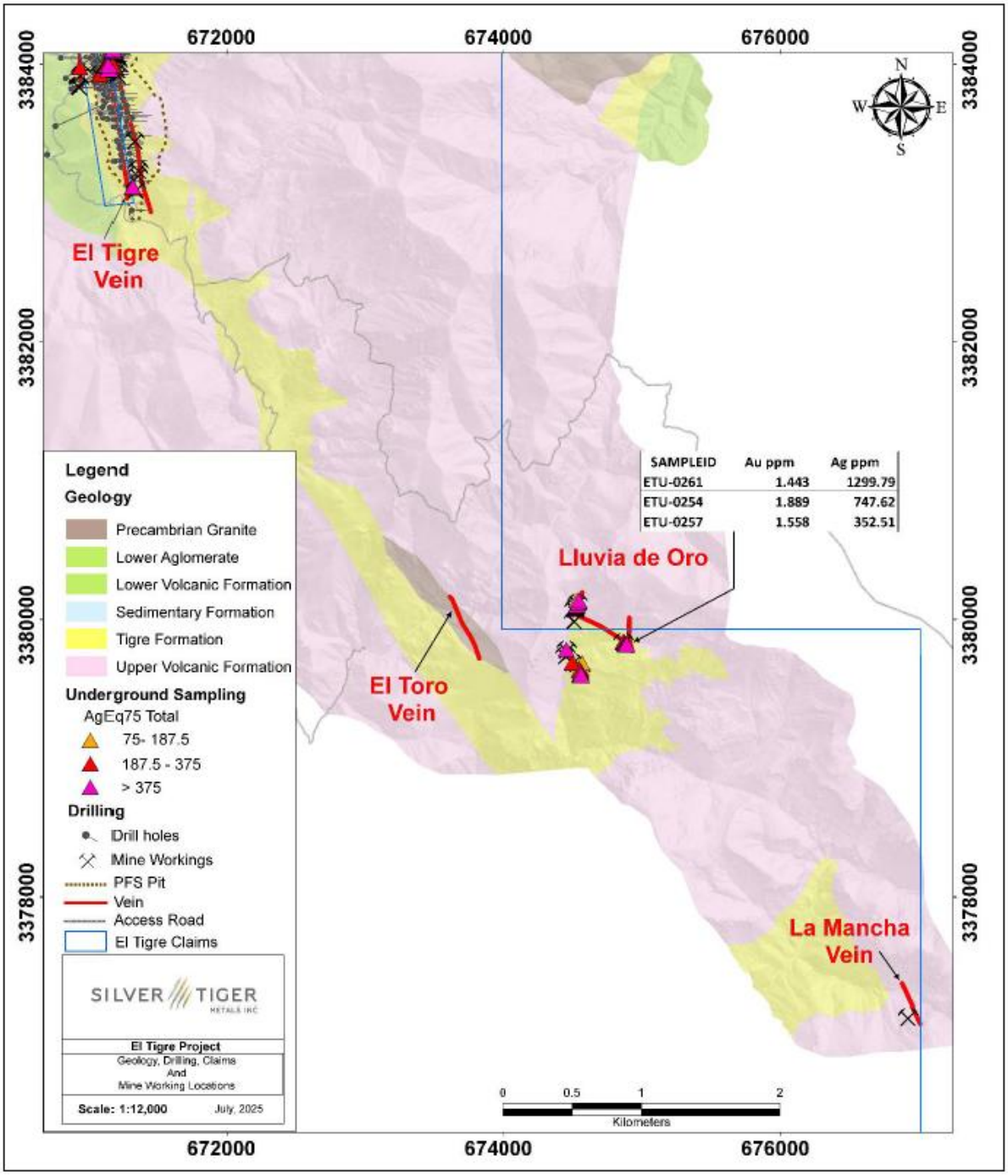
Reconnaissance mapping and sampling was conducted in the Lluvia de Oro Area by the Company in 2016 and 2020. The El Toro and La Mancha Veins, although mapped, remain unsampled.

Significant historical underground workings (>500 m long) were found in Lluvia de Oro with grab sampling completed in a 2020 program returning assays up to 1.90 g/t Au and 1,300 g/t Ag (Figure 9.7). Importantly, the Lluvia de Oro Vein system is contained within the El Tigre Formation, the

host to the Stockwork Zone of which the Company's 2024 PFS is centered. There is potential to extend surface mineralization from the far south of the proposed surface mine south towards Lluvia de Oro.

As work progresses on the Northern Veins, exploration crews will begin analysing Lluvia de Oro, La Mancha and El Toro on the Southern Veins.

FIGURE 9.7 PLAN VIEW EL TIGRE PROJECT– LLUVIA DE ORO - SOUTHERN VEINS



Source: *Silver Tiger press release dated August 13, 2025.*

10.0 DRILLING

The information covering drilling between 2011 and 2013 is summarized from the 2013 Feasibility Report (Black and Choquette 2013). Details on drill programs completed by other companies are provided in Section 6 and the Silver Tiger drilling programs prior to 2020 are summarized from Micon (2023). All drill hole survey and assay values are expressed in metric units, with grid coordinates reported in Universal Transverse Mercator ("UTM") coordinate space relative to WGS 1984 Zone 12.

10.1 NATURE AND EXTENT OF WORK

Many drilling programs have tested various targets and veins on the El Tigre Project Property between 1982 and 2024 (Tables 10.1 and 10.2). The first two drill programs were completed prior to Silver Tiger's acquisition of the Property. In the 1980s, Anaconda (Cobre de Hercules) completed 22 HQ and NQ diamond core drill holes and in 1995 Minera de Cordillaras completed a four-hole reverse circulation ("RC") drilling program for a total of 890 m on behalf of a third party.

Four programs were completed by El Tigre Silver in 2011, 2012 and 2013 and Oceanus in 2016 to 2017. El Tigre Silver completed two additional drilling programs on the material within the historical tailings impoundment. The first program was carried out in 2011 and consisted of 46 straight stem auger holes totalling 315.4 m. The second campaign included seven diamond drill core holes completed in 2013 for 129.9 m. Silver Tiger completed five drilling programs in 2020, 2021, 2022, 2023 and 2024, and an underground drill program in late-2024. A detailed breakdown of the Oceanus and Silver Tiger drill holes by year is presented in Table 10.2.

Silver equivalent ("AgEq") grades in this section are based on a silver to gold price ratio of 75:1 (Au:Ag). Copper, lead and zinc were converted using US$3.66/lb copper, $0.90/lb lead, $1.26/lb zinc at 100% metal recoveries based on a silver price of $26.00/oz.

Collar locations for 2020 to 2024 are tabulated in Appendix F and select significant intercepts are presented in Tables 10.3 to 10.10. A drill hole plan showing the mineralized domains is in Appendix A.

TABLE 10.1
EL TIGRE PROJECT LIST OF DRILLING PROGRAMS

Company	Year	Number of Drill Holes	Metres Drilled
Anaconda	1982 to 1983	22	7,812.65
Minera de Cordilleras	1995	4 (RC)	890.00
El Tigre Silver	2011	10	2,313.35
El Tigre Silver	2012	11	2,235.77
El Tigre Silver	2013	38	4,861.90
Oceanus / Silver Tiger	2016 to 2024	547	132,766.70
Total		**632**	**150,883.10**

TABLE 10.2 BREAKDOWN OF OCEANUS AND SILVER TIGER DRILLING PROGRAMS					
Company	**Year**	**Number of Drill Holes**	**Metres Drilled**	**Drill Holes From**	**Drill Holes To**
Oceanus	2016	34	6,448.6	ET-16-083	ET-16-116
Oceanus	2017	35	6,311.9	ET-17-117	ET-17-151
Silver Tiger	2020	51	8,323.2	ET-20-152	ET-20-202
Silver Tiger	2021	104	33,596.5	ET-21-203	ET-21-305
Silver Tiger	2022	139	38,004.5	ET-21-306	ET-22-444
Silver Tiger	2023	119	29,338.8	ET-22-445	ET-23-563
Silver Tiger	2024	65	10,743.5	ET-23-564	ET-24-625
Total		**547**	**132,766.7**		

10.2 2020 DRILLING

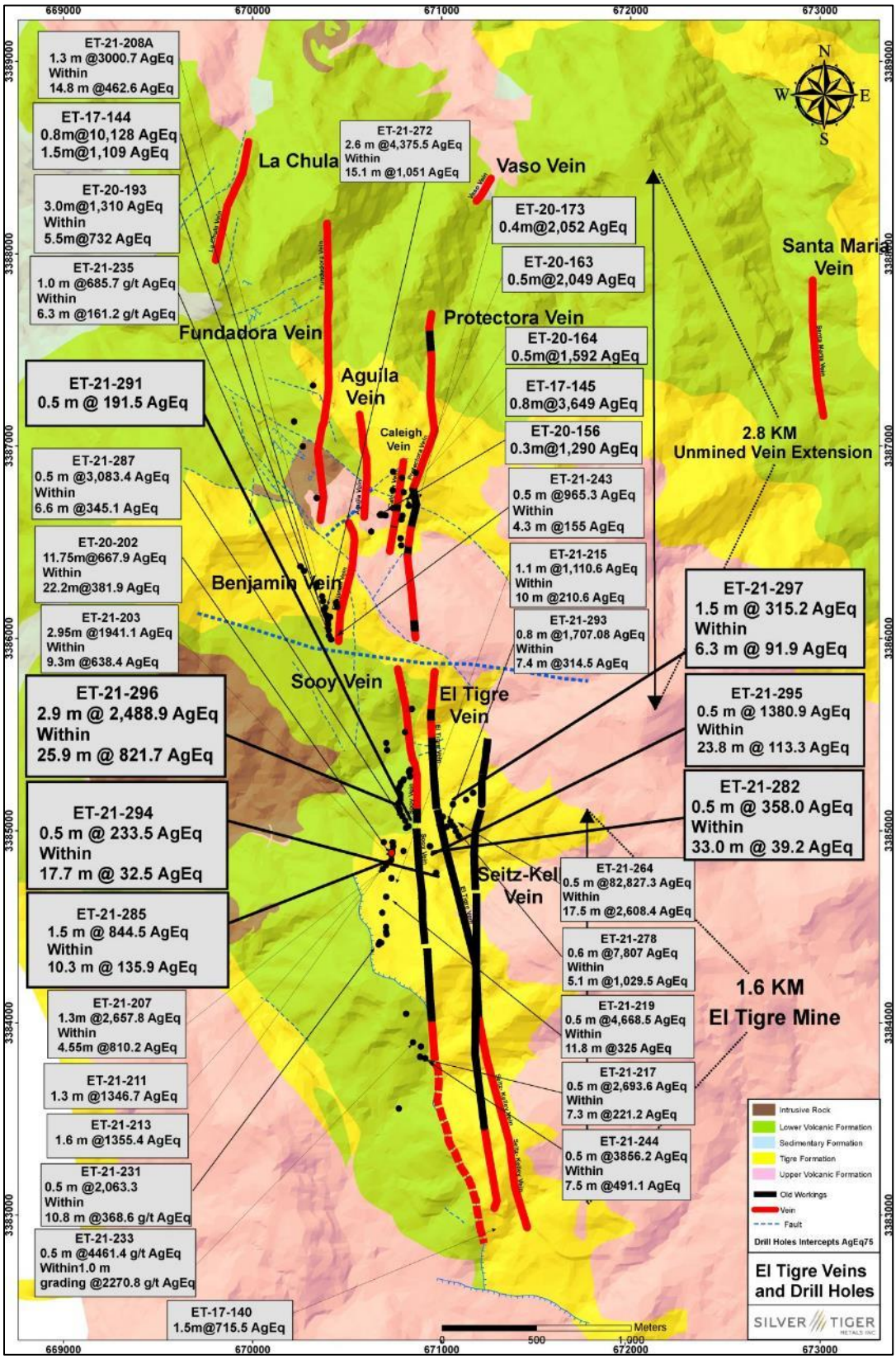
Silver Tiger restarted its exploration program in June 2020 with channel sampling and drilling commencing in August 2020. Fifty-one drill holes totalling 8,323.2 m were completed.

The drilling program utilized two rigs drilling HQ-sized drill core targeting the 3 km of vein extensions north of the historical El Tigre Mine. The drilling focused on the Caleigh, Protectora and Fundadora Veins (Figure 10.1). Cross-sectional projections 6650 N (Figure 10.2) and 6675 N (Figure 10.3) show the Silver Tiger drill holes begin ~1.7 km north of the end of the historical El Tigre Mine, in approximately the middle of the vein extensions north of the Mine. Cross-sectional projection 6675 N is located 25 m north of cross-sectional projection 6650 N.

Significant mineralized intersections from the 2020 drilling are listed in Table 10.3 below.

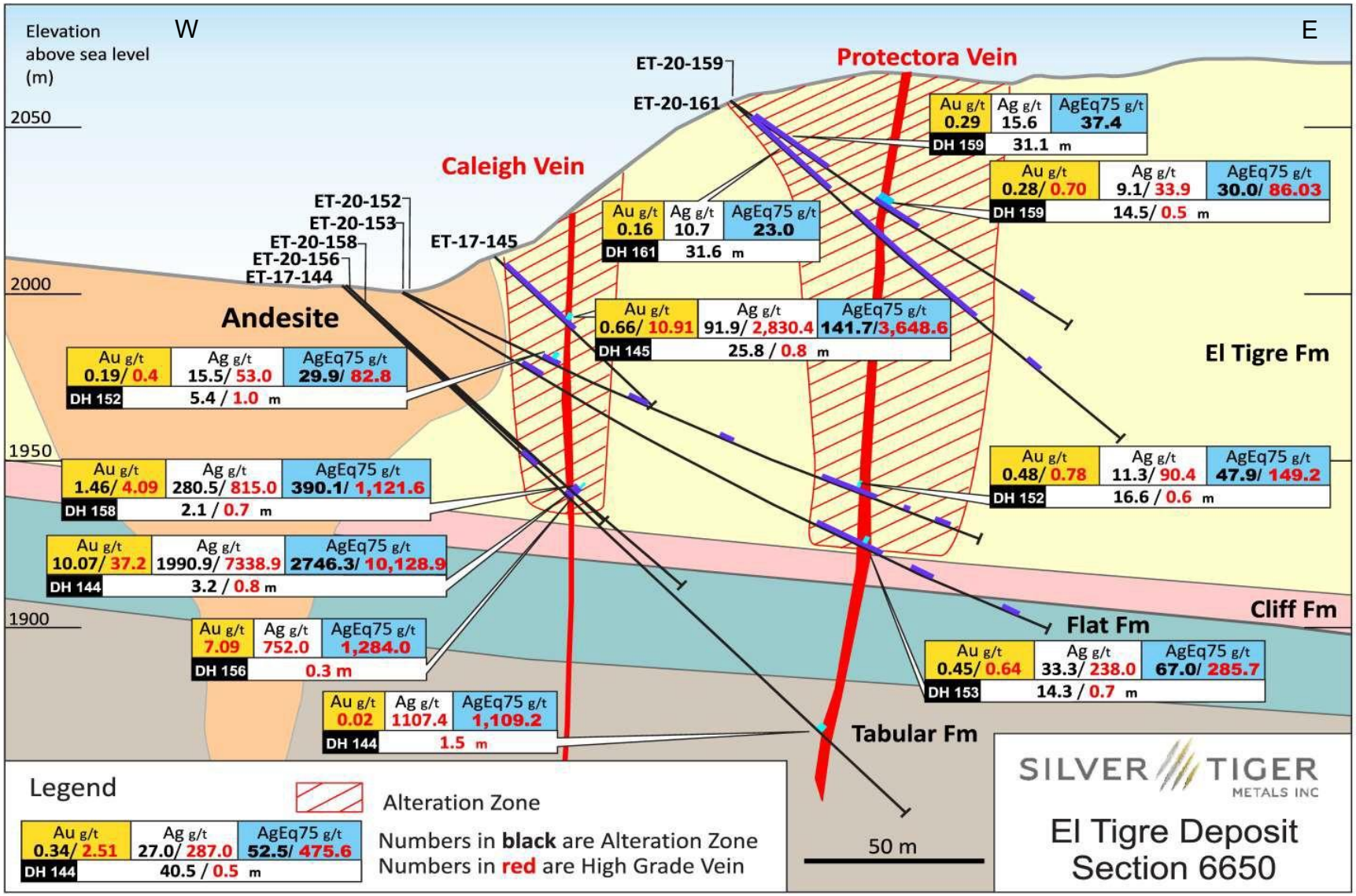
Exploration in 2020 also resulted in the discovery of the Benjamin Vein (see Figure 10.1). Select assay highlights from this drilling are listed in Table 10.4 below.

FIGURE 10.1 PLAN MAP OF EL TIGRE WITH SELECT 2020 AND 2021 DRILL HOLES



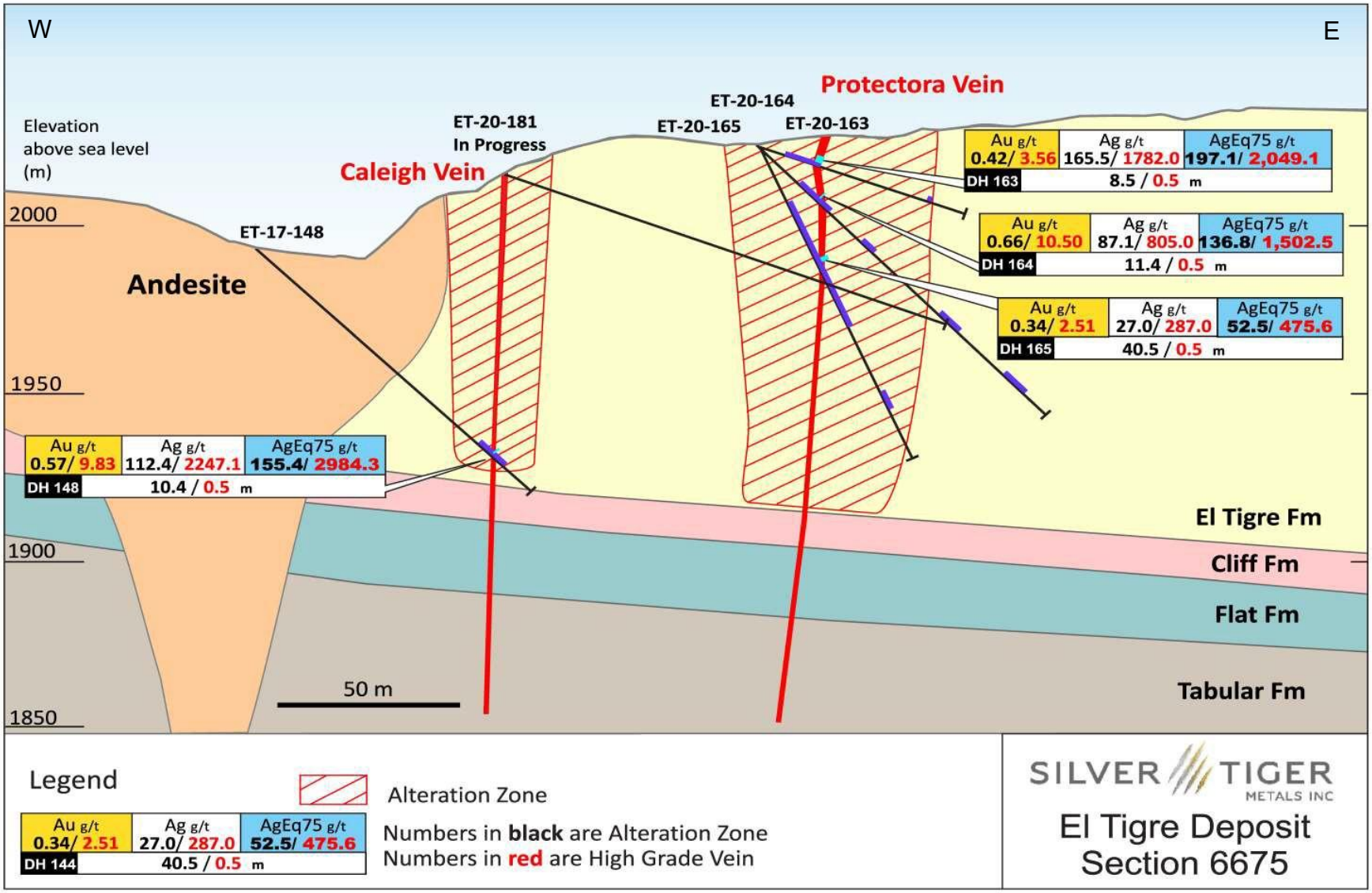
Source: *Silver Tiger website (2023)*

FIGURE 10.2 EL TIGRE CROSS-SECTIONAL PROJECTION 6650 N



Source: *Silver Tiger website (2023)*

FIGURE 10.3 EL TIGRE CROSS-SECTIONAL PROJECTION 6675 N



Source: *Silver Tiger website (2023)*

TABLE 10.3
SIGNIFICANT 2020 MINERALIZED DRILL INTERCEPTS IN THE PROTECTORA, CALEIGH AND BENJAMIN VEINS

Drill Hole ID	Section	Target / Comment[3,4]	From (m)	To (m)	Length (m)[1]	Au (g/t)	Ag (g/t)	AuEq (g/t)[2]	AgEq (g/t)[2]
ET-20-163	6675N	HW Alteration Zone[3]	8.00	18.40	8.50	0.42	165.5	2.63	197.1
ET-20-163	6675N	Protectora Vein Void[5]	11.30	13.20	1.90	*	*	*	*
ET-20-163	6675N	Including	15.20	18.40	3.20	0.74	335.1	5.21	390.7
ET-20-163	6675N	Protectora Vein	16.90	17.40	0.50	3.56	1,782.0	27.32	2,049.1
ET-20-163	6675N	FW Vein	51.40	52.40	1.00	0.48	727.3	10.17	762.9
ET-20-163	6675N	Including	51.90	52.40	0.50	0.89	1,374.0	19.21	1,440.6
ET-20-164	6675N	HW Alteration Zone	17.00	28.40	11.40	0.66	87.1	1.82	136.8
ET-20-164	6675N	Including	17.00	18.20	1.20	4.59	370.4	9.52	714.4
ET-20-164	6675N	Protectora Vein	17.00	17.50	0.50	10.50	805.0	21.23	1,592.5
ET-20-164	6675N	FW Alteration Zone	41.60	46.10	4.50	0.17	8.7	0.29	21.4
ET-20-164	6675N	FW Alteration Zone	71.20	78.80	7.60	0.12	15.8	0.33	24.9
ET-20-164	6675N	FW Alteration Zone	96.90	104.70	7.90	0.21	2.7	0.25	18.7
ET-20-164	6675N	FW Alteration Zone	113.00	114.40	1.30	0.28	0.9	0.30	22.2
ET-20-156	6650N	HW Alteration Zone	67.80	69.30	1.50	0.08	10.5	0.22	16.3
ET-20-156	6650N	Caleigh Vein	82.00	82.30	0.30	7.09	752.0	17.12	1,284.0
ET-20-158	6650N	HW Alteration Zone	89.10	91.20	2.10	1.46	280.5	5.20	390.1
ET-20-158	6650N	Caleigh Vein	90.00	90.70	0.70	4.09	815.0	14.95	1,121.6
ET-20-189	6125N	Benjamin Vein	73.00	78.00	5.00	0.12	140.9	2.00	150.1
ET-20-189	6125N	including	77.50	78.00	0.50	0.13	474.0	6.45	483.9
ET-20-189	6125N	and	108.50	110.00	1.50	0.05	50.6	0.73	54.4
ET-20-189	6125N	and	127.70	128.20	0.50	0.02	91.4	1.24	92.8
ET-20-193	6125N	Benjamin Vein	114.00	119.50	5.50	0.08	726.1	9.76	732.2
ET-20-193	6125N	Including	116.50	119.50	3.00	0.09	1,303.2	17.47	1,310.1
ET-20-193	6125N	Including	117.50	118.80	1.30	0.11	1,967.7	26.34	1,975.8
ET-20-195	6125N	Benjamin Vein	122.00	123.00	1.00	1.02	241.6	4.24	318.4
ET-20-195	6125N	and	131.00	144.70	13.70	0.05	25.6	0.39	29.3

TABLE 10.3
SIGNIFICANT 2020 MINERALIZED DRILL INTERCEPTS IN THE PROTECTORA, CALEIGH AND BENJAMIN VEINS

Drill Hole ID	Section	Target / Comment[3,4]	From (m)	To (m)	Length (m)[1]	Au (g/t)	Ag (g/t)	AuEq (g/t)[2]	AgEq (g/t)[2]
ET-20-195	6125N	including	142.70	143.40	0.70	0.11	107.5	1.54	115.4
ET-20-195	6125N	Benjamin Vein	170.00	171.50	1.50	0.11	326.3	4.46	334.6
ET-20-195	6125N	including	170.50	171.00	0.50	0.12	625.0	8.45	634.0

Notes:

1. *Not true thickness.*
2. *Silver Equivalent ("AgEq") ratios are based on a silver to gold price ratio of 75:1 (Au:Ag). Copper, lead and zinc are converted using $3.66/lb copper, $0.90/lb lead, $1.26/lb zinc at 100% metal recoveries based on a silver price of $26.00/oz.*
3. *HW = hanging wall.*
4. *FW = footwall.*
5. *Void from historical mine workings.*

TABLE 10.4
SIGNIFICANT 2020 MINERALIZED DRILL INTERCEPTS OF THE SOOY VEIN

Drill Hole ID	Target / Comment	From (m)	To (m)	Length (m)[1]	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)[2]
ET-20-202	Sooy Vein	234.10	256.30	22.20	1.12	176.9	0.23	1.43	1.96	381.9
ET-20-202	including	235.10	246.90	11.80	2.02	305.0	0.39	2.34	3.56	667.9
ET-20-202	including	239.90	246.90	7.00	3.34	344.7	0.33	2.05	3.36	787.5
ET-20-202	including	241.90	243.50	1.60	0.13	657.9	0.46	4.25	7.64	1,065.8
ET-20-202	including	239.90	240.40	0.50	0.14	1,496.0	1.43	3.90	6.69	1,958.1
ET-20-202	including	241.90	242.50	0.60	0.22	1,096.0	0.47	5.79	14.25	1,767.4
ET-20-202	including	245.85	246.90	1.00	22.50	28.9	0.01	0.49	0.37	1,741.4

Notes:

1. *Not true thickness*
2. *Silver Equivalent ("AgEq") ratios are based on a silver to gold price ratio of 75:1 (Au:Ag). Copper, lead and zinc are converted using $3.66/lb copper, $0.90/lb lead, $1.26/lb zinc at 100% metal recoveries based on a silver price of $26.00/oz.*

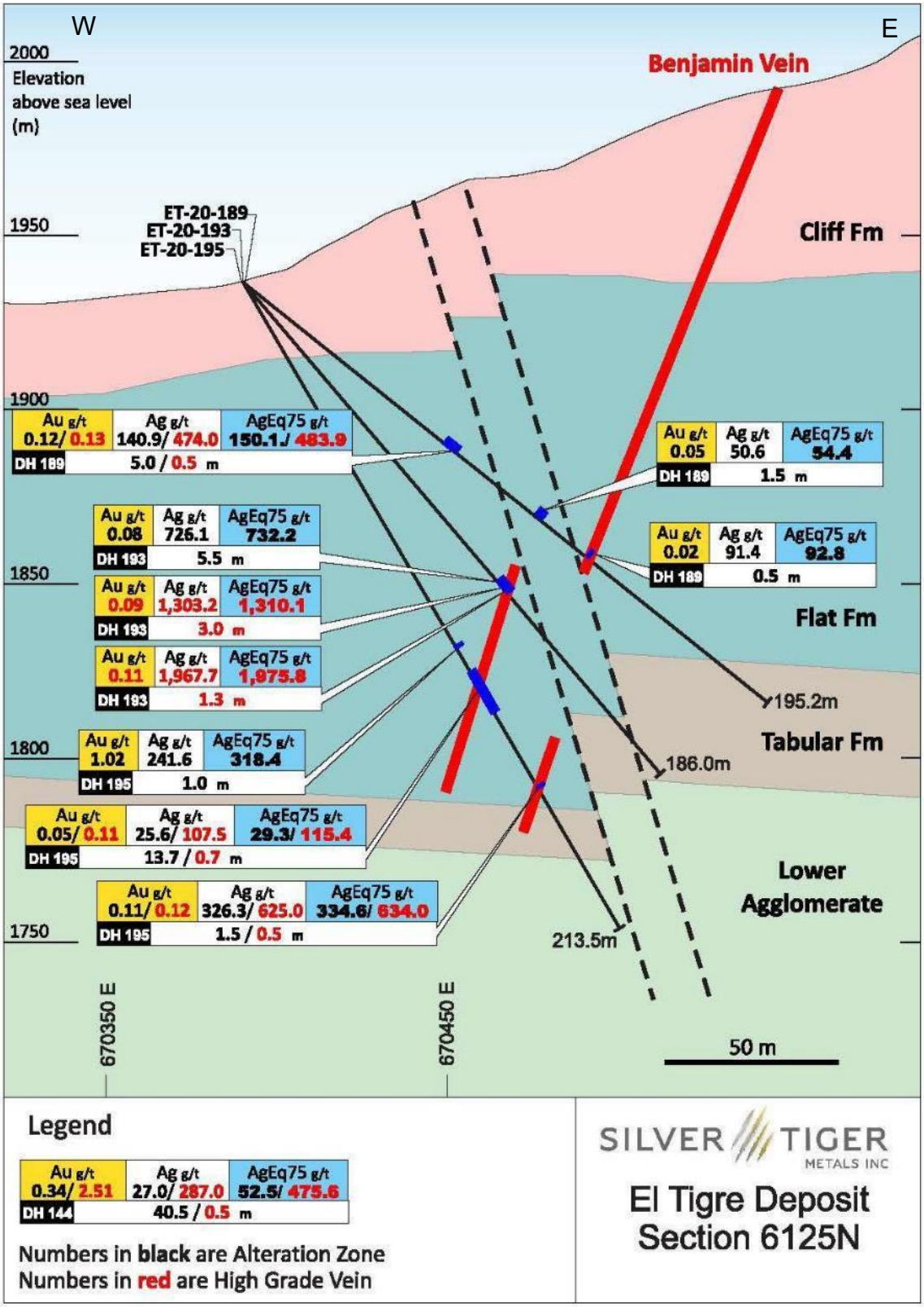
In addition, drill hole ET-20-199 intersected 1.1 m grading 1,015.9 g/t AgEq from 113.6 to 114.7 m, consisting of 699.4 g/t Ag, 0.11 g/t Au, 0.2% Cu, 2.08% Pb and 7.01% Zn within 7.7 m grading 304.6 g/t AgEq from 107.0 to 114.7 m, consisting of 222.3 g/t Ag, 0.05 g/t Au, 0.09% Cu, 0.51% Pb and 1.69% Zn.

A cross-sectional projection showing the intersection of drill holes ET-20-189, ET-20-193 and ET-20-195 with the Benjamin Vein is shown in Figure 10.4.

Drill Hole ET-20-202 (see Figure 10.1) was completed to test the down-dip potential of the Sooy Vein targeting just under the lowest mine level, ~150 m from surface where mining ceased in 1930. The drill hole passed through mine workings on the Sooy Vein and continued beyond the footwall of the Sooy Vein and discovered a new style of wide, high-grade mineralization in the Flat Formation that is not the traditional quartz vein mineralization which had previously been mined at El Tigre. El Tigre cross-sectional projection 4900N is presented in Figure 10.5.

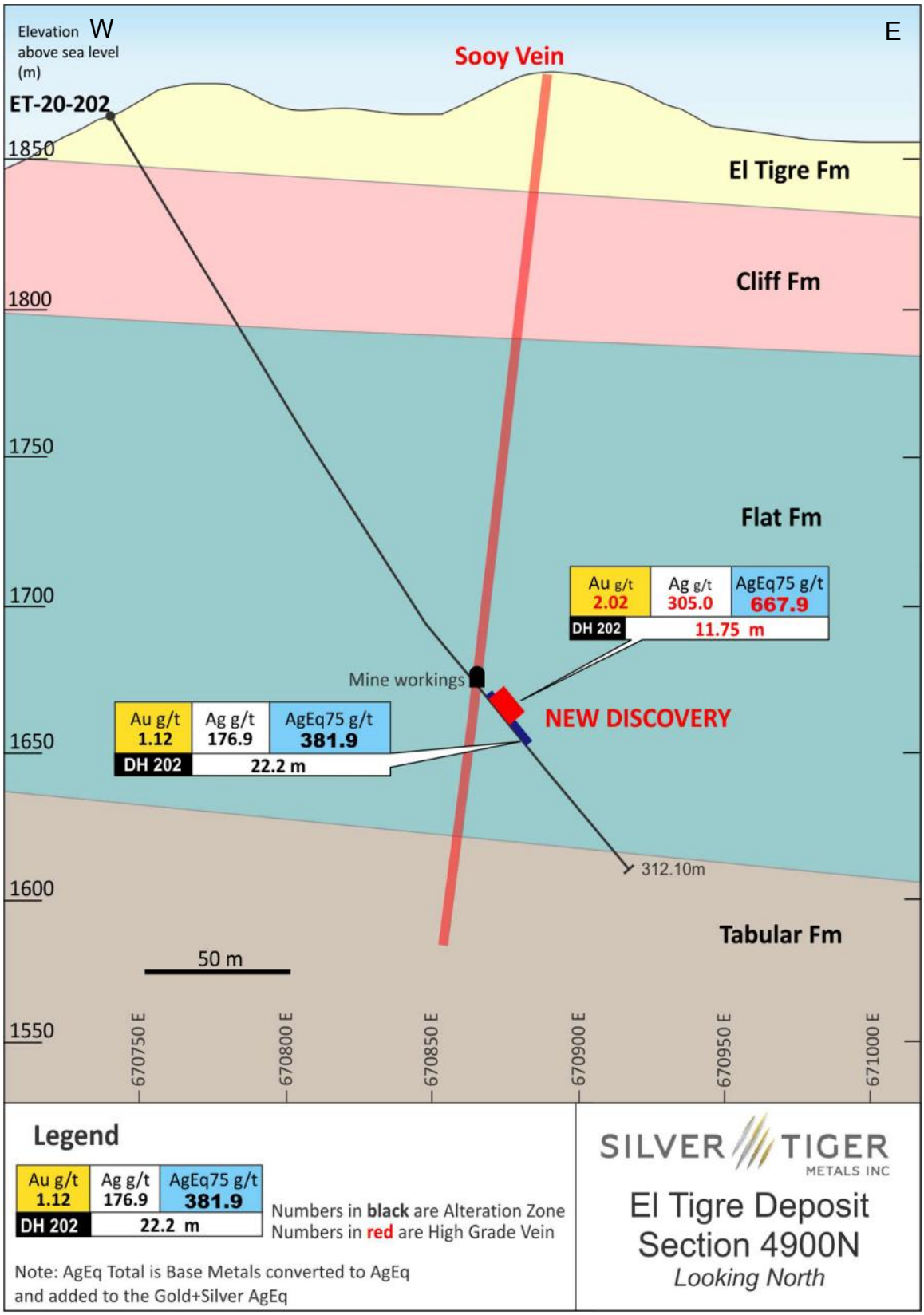
Significant mineralized intersections in drill hole ET-20-202 are listed in Table 10.4.

FIGURE 10.4 EL TIGRE CROSS-SECTIONAL PROJECTION 6125 N



Source: *Silver Tiger website (2023)*

FIGURE 10.5 EL TIGRE CROSS-SECTIONAL PROJECTION 6125 N



Source: *Silver Tiger website (2023)*

10.3 2021 DRILLING

Silver Tiger continued drilling in 2021, a third drill rig was mobilized to the site and increased to six drill rigs over the course of the year. A total of 104 drill holes were completed for 33,597 m of drilling. Significant intersections from the 2021 drilling are presented in Table 10.5.

Drilling followed-up on the mineralization discovered in drill hole ET-20-202 in the Sooy Vein. A plan view of the Seitz Kelly Vein and the Sooy Vein with 2021 drilling highlights is presented in Figure 10.6. Drill hole ET-21-203 was completed on cross-sectional projection 4875N to test the Footwall Zone ~25 m to the south along strike from the drill hole ET-20-202. Drill hole ET-21-207 was collared on cross-sectional projection 4825N to test the Footwall Zone ~75 m to the south along strike from drill hole ET-20-202. Drill Hole ET- 21-219 was completed on cross-sectional projection 4650N (Figure 10.7), 250 m to the south along strike from drill hole ET-20-202, whereas drill hole ET-21-217 was located 1,025 m to the south of drill hole ET-20-202 on cross-sectional projection 3875N (Figure 10.8). Drill hole ET-21-236 is located 25 m along strike to the north from drill hole ET-20-202. A plan view of the drill holes is presented in Figure 10.1 above.

In addition, drill hole ET-21-221 intersected 1.5 m grading 799.8 g/t Ag/Eq from 239.5 to 241.0 m, consisting of 465.7 g/t Ag, 4.22 g/t Au, 0.07% Cu, 0.20% Pb and 0.21% Zn (Figure 10.6).

TABLE 10.5
SIGNIFICANT 2021 MINERALIZED DRILL INTERCEPTS IN THE SOOY, BENJAMIN, EL TIGRE AND SEITZ KELLY VEINS

Drill Hole ID	Target / Comment	From (m)	To (m)	Length (m)[1]	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)[2]
ET-21-203	Sooy Vein	263.00	264.20	1.20	0.07	154.3	0.21	0.09	0.01	182.6
ET-21-203	and	274.60	276.00	1.40	0.06	148.8	0.06	0.11	0.08	164.0
ET-21-203	Footwall Zone	290.90	300.20	9.30	0.22	555.7	0.51	0.45	0.20	638.4
ET-21-203	including	297.30	300.20	2.90	0.56	1,708.8	1.53	1.18	0.44	1,941.1
ET-21-207		285.70	290.30	4.60	0.18	669.8	1.04	0.68	0.30	810.2
ET-21-207	including	289.00	290.30	1.30	0.60	2,210.7	3.40	2.18	0.69	2,657.8
ET-21-207	and	299.30	301.00	1.80	0.08	290.7	0.67	0.06	0.18	368.7
ET-21-204	Benjamin Vein	111.80	126.00	14.20	0.12	140.5	0.08	2.98	8.54	519.5
ET-21-204	including	120.40	126.00	5.60	0.14	272.7	0.18	5.53	16.95	1,010.3
ET-21-206	Benjamin Vein	106.20	118.80	12.60	0.07	181.3	0.05	0.57	1.79	266.6
ET-21-206	including	106.20	114.20	8.00	0.10	278.0	0.08	0.87	2.74	407.1
ET-21-206	including	107.50	111.80	4.30	0.12	502.3	0.14	1.55	4.93	730.3
ET-21-206	including	109.80	111.80	2.00	0.18	1,034.5	0.29	2.74	8.83	1,443.0
ET-21-208	Benjamin Vein	114.10	120.50	6.40	0.04	115.1	0.07	0.39	0.93	166.4
ET-21-208	including	119.50	120.50	1.00	0.13	512.5	0.25	1.02	4.89	737.8
ET-21-217	Sooy Vein	103.70	111.00	7.30	0.08	210.5	0.04	0.03	0.01	221.2
ET-21-217	including	108.90	109.50	0.50	0.66	2,595.0	0.41	0.26	0.05	2,693.6
ET-21-219	Sooy Vein	179.00	190.80	11.80	0.29	288.3	0.05	0.11	0.20	325.0
ET-21-219	including	186.90	190.80	4.20	0.11	648.8	0.14	0.28	0.55	696.2
ET-21-219	including	190.30	190.80	0.50	0.49	4,375.0	0.88	2.00	3.57	4,668.5
ET-21-236	Sooy Vein	193.90	199.60	5.70	0.34	324.4	0.47	0.73	0.45	427.8
ET-21-236	including	193.90	196.30	2.30	0.74	676.7	1.06	1.51	1.04	905.2
ET-21-236	including	195.00	195.80	0.80	0.27	894.0	0.93	0.78	0.51	1,039.3

TABLE 10.5
SIGNIFICANT 2021 MINERALIZED DRILL INTERCEPTS IN THE SOOY, BENJAMIN, EL TIGRE AND SEITZ KELLY VEINS

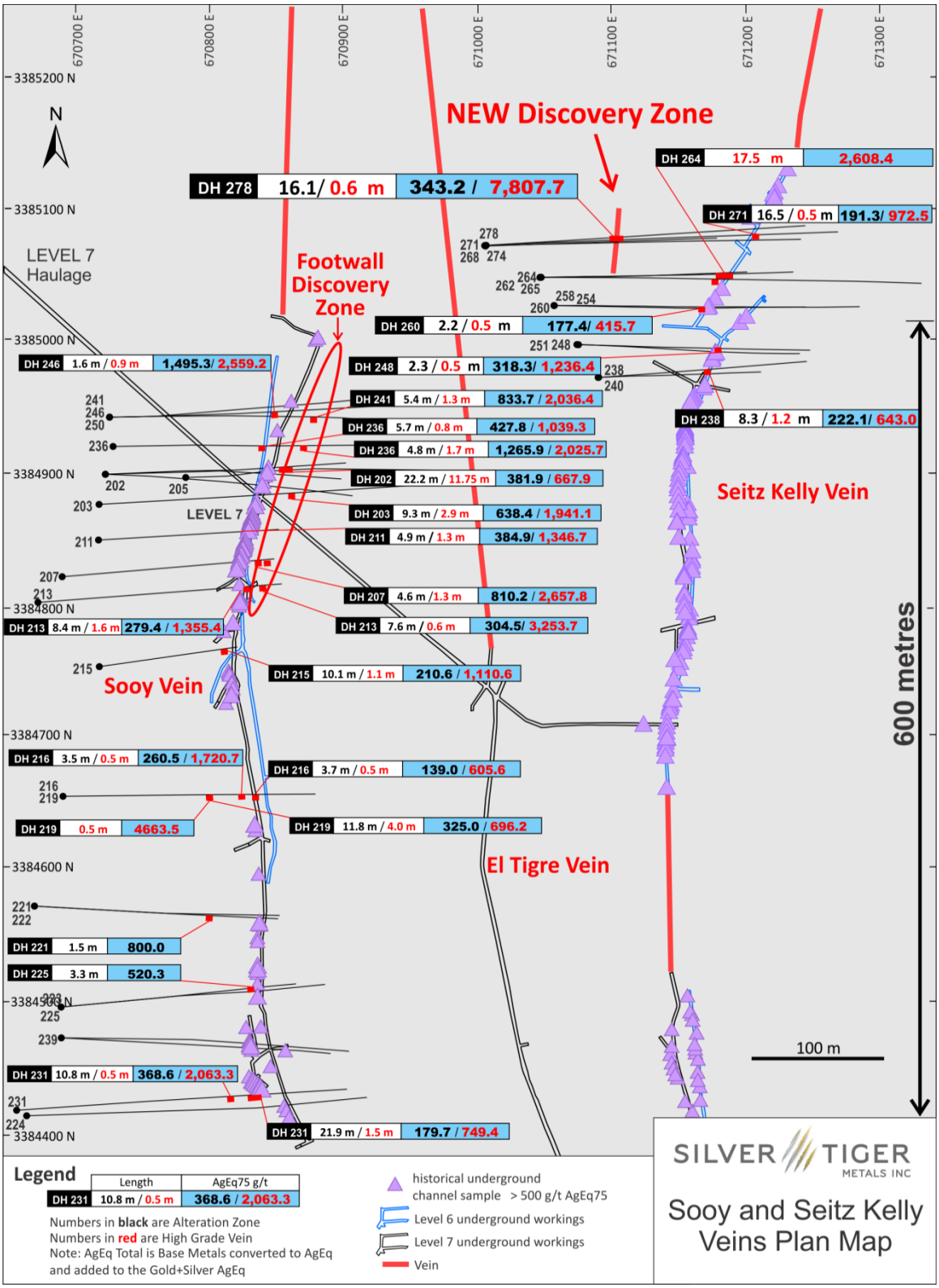
Drill Hole ID	Target / Comment	From (m)	To (m)	Length (m)[1]	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)[2]
ET-21-236	FW Discovery Zone	246.60	253.80	7.20	0.16	480.7	0.69	3.81	6.65	870.3
ET-21-236	including	246.60	251.50	4.80	0.17	702.9	1.02	5.59	9.65	1,265.9
ET-21-236	including	248.10	250.80	2.70	0.24	1,069.5	1.25	6.43	9.34	1,669.8
ET-21-236	including	249.10	250.80	1.70	0.29	1,381.9	1.55	7.01	9.24	2,025.7
ET-21-236	including	249.60	250.10	0.50	0.32	2,093.0	2.61	11.02	14.07	3,096.3
ET-21-236	and	265.20	267.50	2.30	0.06	105.8	0.06	0.10	0.45	133.2
ET-21-236	and	306.10	307.30	1.30	0.03	26.1	0.02	0.28	0.63	58.7
ET-21-244		159.00	160.00	1.00	0.08	304.3	0.08	0.05	0.16	323.9
ET-21-244	Sooy Vein	244.00	251.50	7.50	0.16	454.8	0.12	0.33	0.16	491.1
ET-21-244	including	249.00	251.50	2.50	0.33	1,073.1	0.3	0.82	0.30	1,156.4
ET-21-244	including	251.00	251.50	0.50	1.30	3,531.0	1.06	3.64	1.18	3,856.2
ET-21-244	FW Zone	273.00	274.80	1.80	0.13	593.4	0.13	0.28	0.06	624.5
ET-21-244	including	274.30	274.80	0.50	0.40	2,030.0	0.46	0.99	0.16	2,132.6

Notes:

1. Not true thickness.

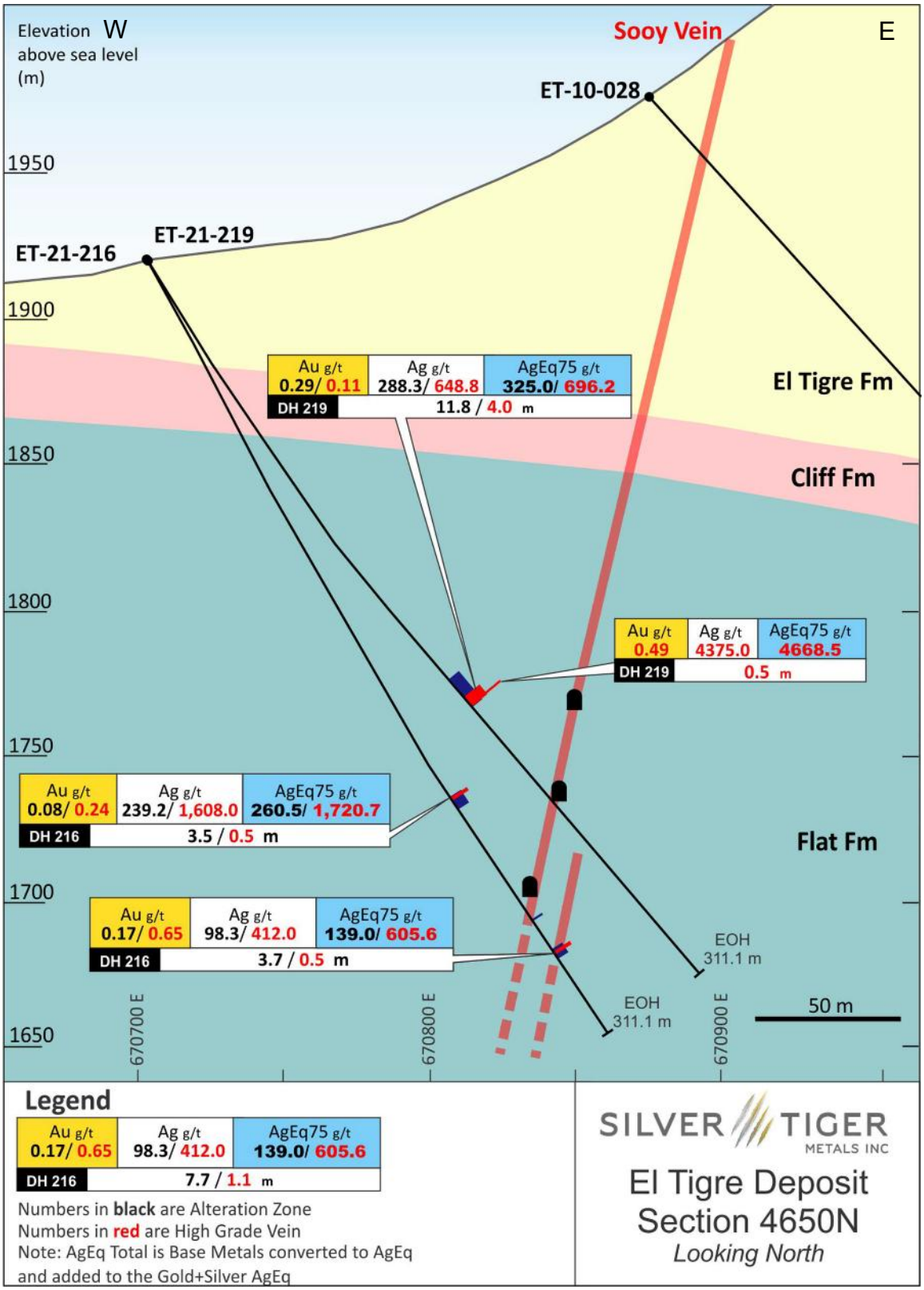
2. Silver Equivalent ("AgEq") ratios are based on a silver to gold price ratio of 75:1 (Au:Ag). Copper, lead and zinc are converted using $3.66/lb copper, $0.90/lb lead, $1.26/lb zinc at 100% metal recoveries based on a silver price of $26.00/oz.

FIGURE 10.6 2021 PLAN MAP OF SOOY AND SEITZ KELLY VEIN WITH THE NEW DISCOVERY ZONE



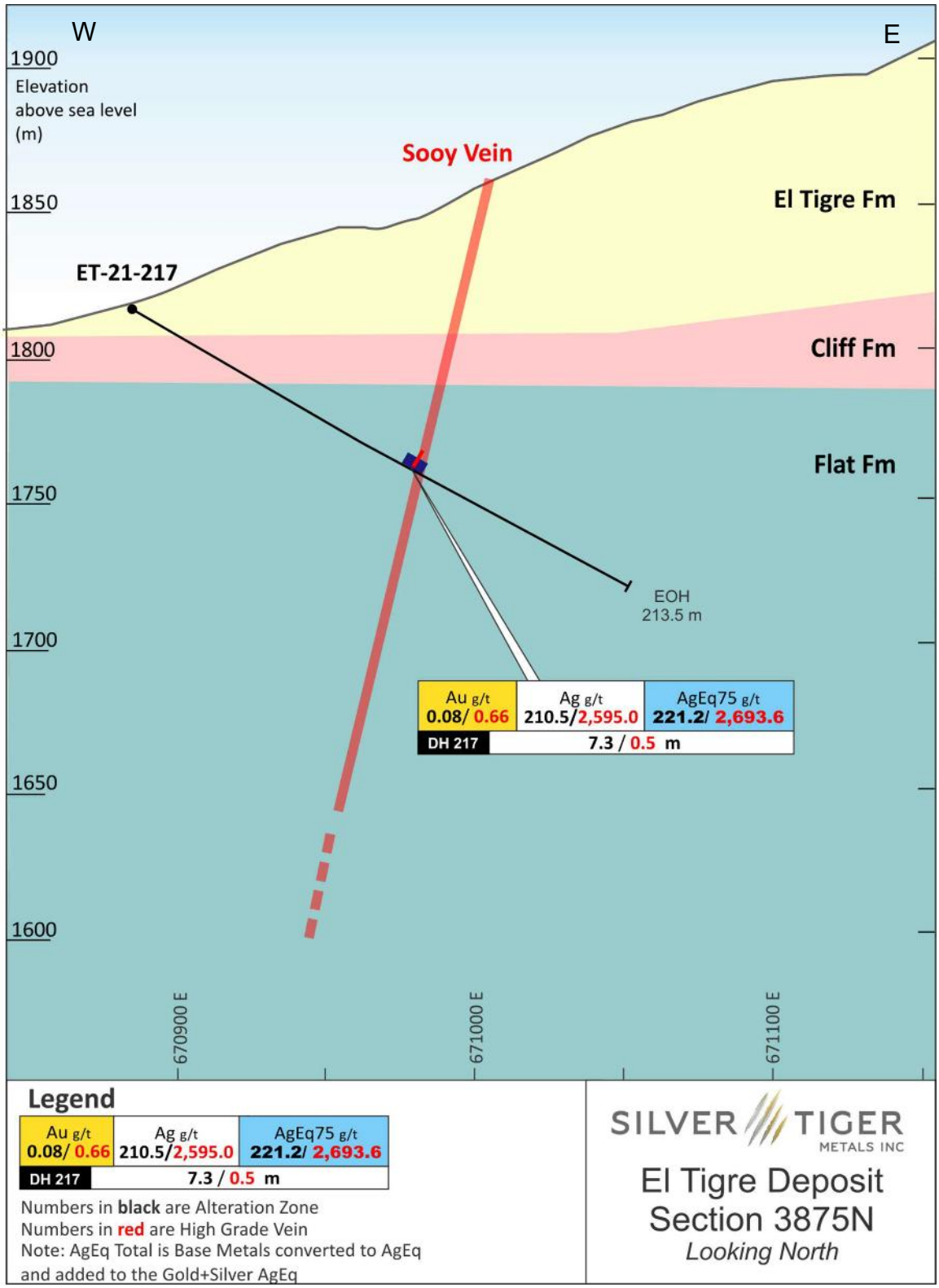
Source: *Silver Tiger website (2023)*

FIGURE 10.7 EL TIGRE CROSS-SECTIONAL PROJECTION 4650 N



Source: *Silver Tiger website (2023)*

FIGURE 10.8 EL TIGRE CROSS-SECTIONAL PROJECTION 3875 N



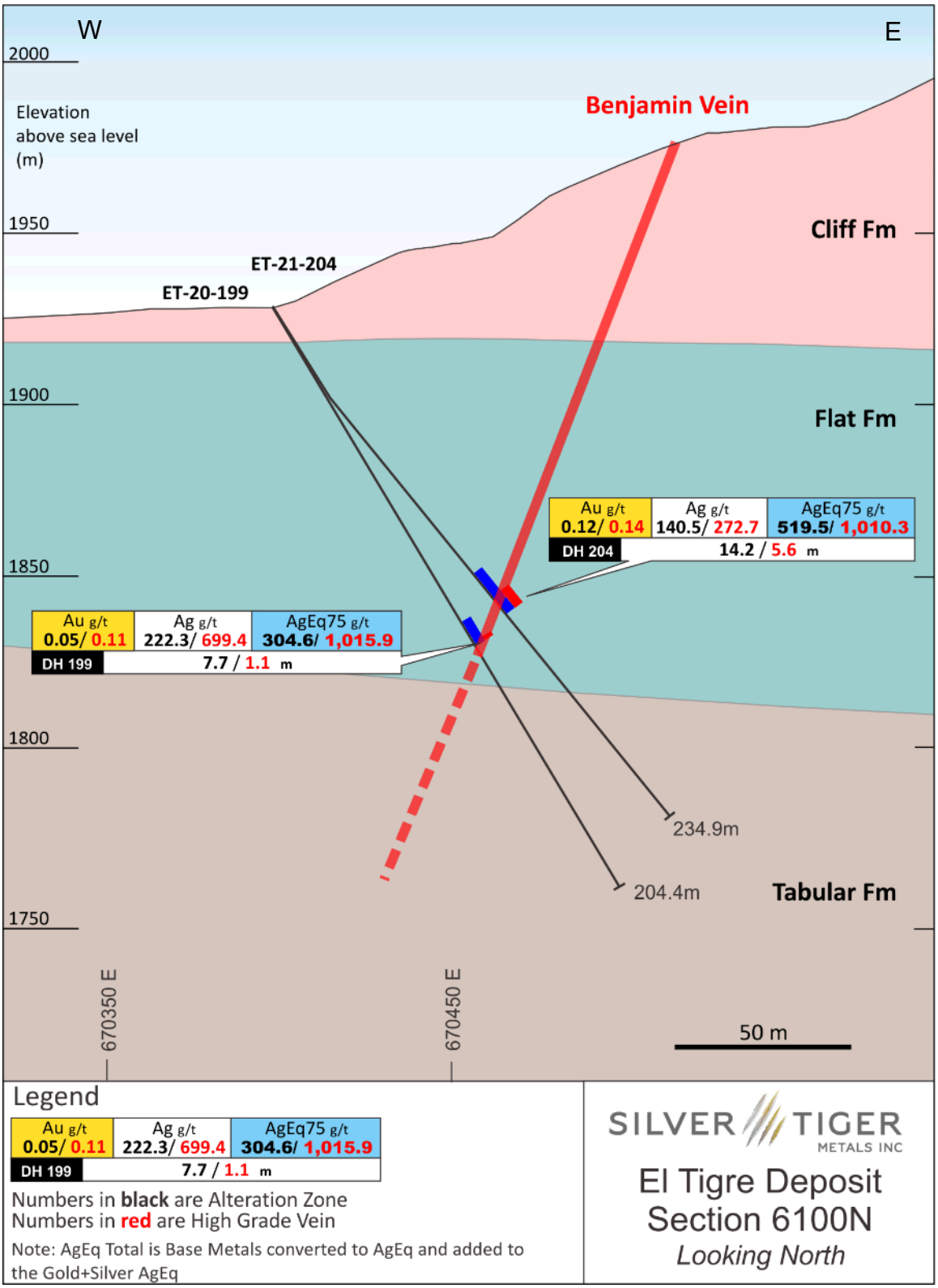
Source: *Silver Tiger website (2023)*

Drilling was also undertaken on the Benjamin Vein, following-up on the discovery drill hole ET-20-193. Drill holes ET-21-204 and ET-21-206 are located on cross-sections ~25 and 50 m to the south of drill hole ET-20-193. Drill hole ET-20-208A was collared on the set-up for drill hole ET-20-208. Cross-sectional projections are presented in Figures 10.9 and 10.10. Select highlights are listed above in Table 10.5.

In addition**,** drill hole ET-21-272 intersected 4,375.5 g/t AgEq over 2.6 m from 119.9 to 122.5 m within a broader mineralized interval of 15.1 m grading 1,051.0 g/t AgEq from 110.7 to 125.8 m. Drill hole ET-21-272, the northernmost drill hole on the Benjamin Vein, intersected a wide zone of semi-massive to massive sphalerite, galena and chalcopyrite mineralization on the footwall side of the Benjamin Vein (Figure 10.12).

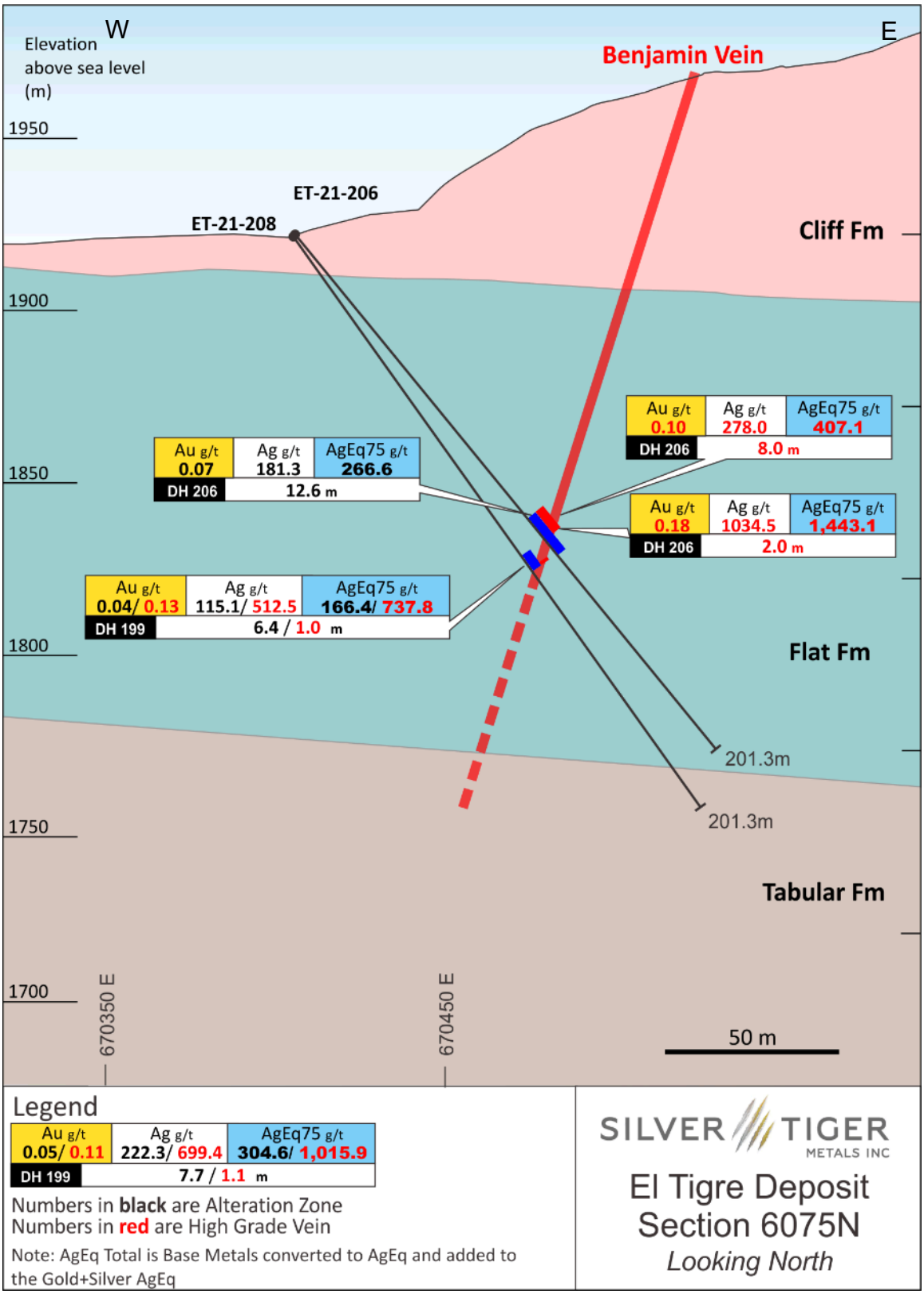
Drill hole ET-21-264, completed in the Seitz Kelly Vein, returned the highest metal grades encountered in 2021, intersecting 82,827.3 g/t AgEq over 0.5 m from 181.3 to 181.8 m within a broader mineralized interval of 17.5 m grading 2,608.4 g/t AgEq from 181.3 to 198.8 m. A new discovery zone, labelled Vein 4, was identified between the Tiger Vein and Seitz Kelly Vein. Drill hole ET-21-278 intersected 7,807 g/t AgEq over 0.6 m within a broader interval of 5.1 m grading 1,029.5 g/t AgEq. A cross-sectional projection is presented in Figure 10.11.

FIGURE 10.9 EL TIGRE CROSS-SECTIONAL PROJECTION 6100 N



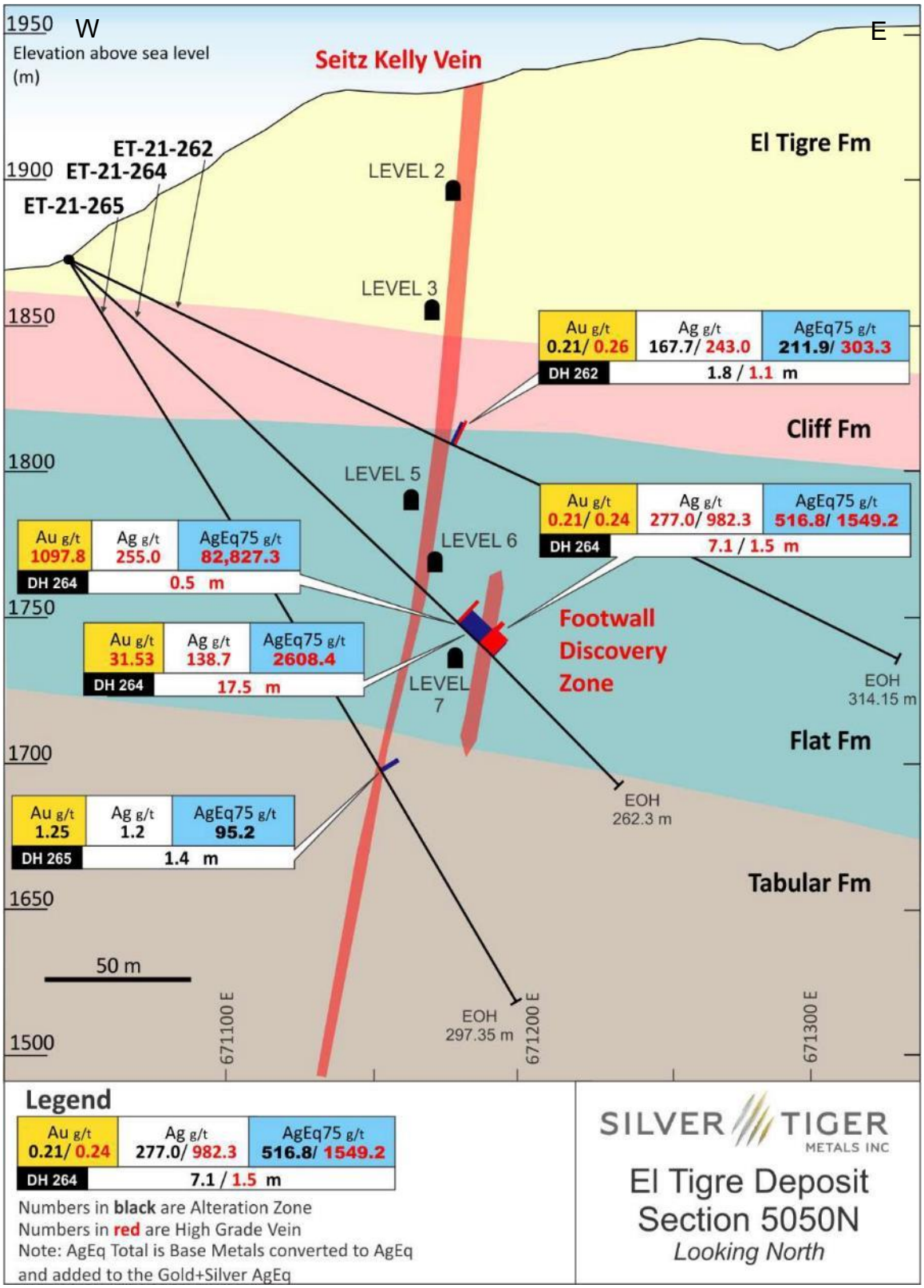
Source: *Silver Tiger website (2023)*

FIGURE 10.10 EL TIGRE CROSS-SECTIONAL PROJECTION 6075 N



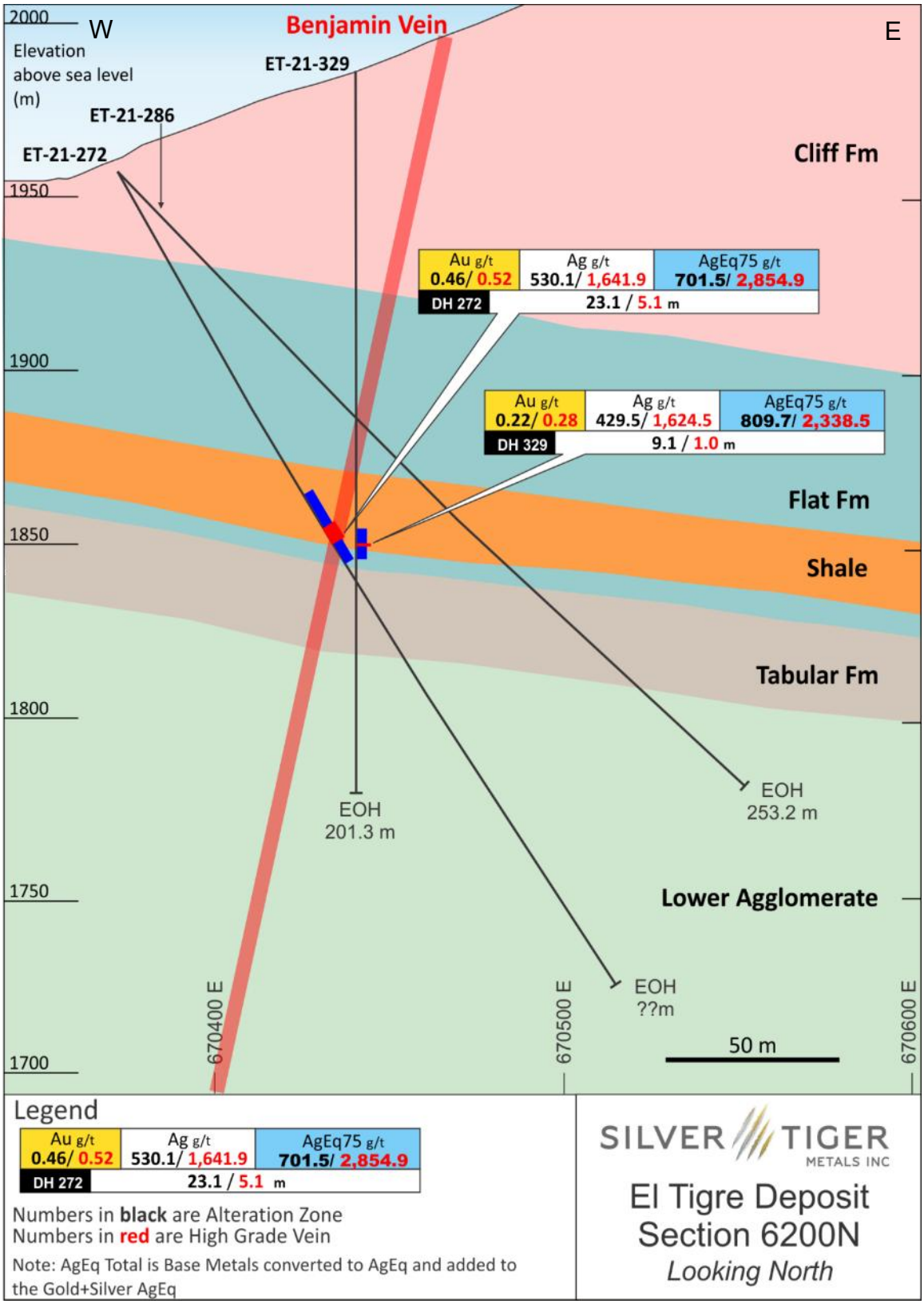
Source: *Silver Tiger website (2023)*

FIGURE 10.11 EL TIGRE CROSS-SECTIONAL PROJECTION 5050 N



Source: *Silver Tiger website (2023)*

FIGURE 10.12 EL TIGRE CROSS-SECTIONAL PROJECTION 6200 N



Source: *Silver Tiger website (2023)*

10.4 2022 DRILLING

Six drill rigs were working on El Tigre when the program resumed in 2022. A total of 139 drill holes were completed in 2022 for 38,005 m. Significant intersections from the 2022 drilling are listed in Table 10.6.

Definition drilling continued on the Sooy Vein and the El Tigre Vein. Drill hole ET-431 targeted the El Tigre Vein under the northern unmined portion of the historical El Tigre Mine. Selected additional highlights include:

- **Drill Hole ET-21-395:** intersected 2.0 m grading 1,653.8 g/t AgEq from 397.7 to 399.7 m, consisting of 1,270.3 g/t Ag, 0.2 g/t Au, 1.66% Cu, 2.85% Pb and 4.26% Zn, within a broader mineralized interval of 8.1 m grading 834.5 g/t AgEq from 394.8 to 402.9 m, consisting of 554.2 g/t Ag, 0.17 g/t Au, 0.92% Cu, 2.48% Pb and 3.63% Zn at the El Tigre Vein;

- **Drill Hole ET-21-417:** intersected 0.5 m grading 2,473.2 g/t AgEq from 303.7 to 304.2 m, consisting of 2,010.0 g/t Ag, 0.27 g/t Au, 2.51% Cu, 1.53% Pb and 4.97% Zn at the El Tigre Vein;

- **Drill Hole ET-21-427:** intersected 1.1 m grading 3,331.9 g/t AgEq from 56.4 to 57.5 m, consisting of 3,096.0 g/t Ag, 1.79 g/t Au, 0.68% Cu, 1.25% Pb and 0.22% Zn at the Sooy Vein;

- **Drill Hole ET-21-428:** intersected 0.6 m grading 3,362.6 g/t AgEq from 48.8 to 49.4 m, consisting of 3,097.0 g/t Ag, 2.47 g/t Au, 0.32% Cu, 0.97% Pb and 0.81% Zn at the Sooy Vein; and

- **Drill Hole ET-31:** intersected 0.5 m grading 1,010.9 g/t AgEq from 104.3 to 104.8 m, consisting of 835.0 g/t Ag, 2.30 g/t Au, 0.03% Cu, 0.02% Pb and 0.02% Zn in the El Tigre Vein.

A new sulphide zone, named Sulphide Zone, was also discovered below the Sooy Vein. A plan view of the Sulphide Zone is presented in Figure 10.13 and a cross-sectional projection is presented in Figure 10.14.

TABLE 10.6
SIGNIFICANT 2022 MINERALIZED DRILL INTERCEPTS IN THE EL TIGRE, SEITZ KELLY VEINS AND SULPHIDE ZONE

Drill Hole ID	Target / Comment	From (m)	To (m)	Length (m)[1]	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)[2]
ET-308	El Tigre Au Zone	5.70	33.60	27.90	0.50	25.7	0.01	0.02	0.02	64.8
ET-308	including	15.80	33.60	17.80	0.57	40.0	0.01	0.03	0.03	85.3
ET-308	including	24.40	28.00	3.60	1.45	144.3	0.03	0.14	0.06	260.9
ET-308	including	24.40	25.40	1.00	1.82	366.0	0.05	0.27	0.16	518.9
ET-308	Seitz Kelly Vein	141.90	143.40	1.50	0.12	497.4	0.27	1.50	0.52	585.5
ET-308	including	142.70	143.40	0.70	0.23	1,022.0	0.55	3.16	1.08	1,203.8
ET-308	and	255.50	259.50	4.00	0.02	67.1	0.03	0.01	0.02	73.0
ET-308	including	259.00	259.50	0.50	0.02	376.0	0.19	0.01	0.09	398.9
ET-429	Sooy Vein Zone	63.00	63.50	0.50	1.46	1,320.0	0.32	0.16	0.07	1,465.9
ET-429	El Tigre Vein	247.80	253.20	5.40	0.06	219.6	0.18	0.24	0.17	252.9
ET-429	including	249.90	251.00	1.10	0.07	707.0	0.39	0.37	0.40	771.9
ET-429	El Tigre Vein	265.20	275.90	10.70	0.04	120.2	0.17	0.3	0.22	154.6
ET-429	including	274.30	275.40	1.10	0.04	549.7	0.74	1.02	0.90	677.9
ET-431	HW Gold Vein Zone	10.10	46.50	36.40	0.13	41.9	0.00	0.01	0.01	52.6
ET-431	including	10.10	11.60	1.50	0.11	731.0	0.00	0.05	0.00	740.9
ET-431	Sooy Vein Zone	83.90	86.70	2.80	0.59	137.4	0.03	0.19	0.24	196.5
ET-431	and	339.50	342.60	3.10	0.06	211.5	0.08	0.02	0.03	225.2
ET-431	El Tigre Vein	409.10	418.50	9.40	0.19	641.3	0.65	3.32	6.51	1,013.3
ET-431	including	413.50	415.60	2.10	0.19	1,536.1	1.62	7.71	13.66	2,341.9
ET-431	HW Gold Zone	10.10	46.50	36.40	0.13	41.9	0.00	0.01	0.01	52.6
ET-431	including	10.10	11.60	1.50	0.11	731.0	0.00	0.05	0.00	740.9
ET-431	Sooy Vein Zone	83.90	86.70	2.80	0.59	137.4	0.03	0.19	0.24	196.5
ET-431	and	339.50	342.60	3.10	0.06	211.5	0.08	0.02	0.03	225.2
ET-431	El Tigre Vein	409.10	418.50	9.40	0.19	641.3	0.65	3.32	6.51	1,013.3

TABLE 10.6
SIGNIFICANT 2022 MINERALIZED DRILL INTERCEPTS IN THE EL TIGRE, SEITZ KELLY VEINS AND SULPHIDE ZONE

Drill Hole ID	Target / Comment	From (m)	To (m)	Length (m)[1]	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)[2]
ET-431	including	413.50	415.60	2.10	0.19	1,536.1	1.62	7.71	13.66	2,341.9
ET-22-432	HW Gold Zone	4.60	14.00	9.40	0.10	39.0	0.01	0.02	0.02	48.5
ET-22-432	Sooy Vein Zone	191.20	201.60	10.40	0.16	399.0	0.43	2.84	5.16	690.6
ET-22-432	including	195.60	197.70	2.10	0.24	1,153.6	1.29	9.76	16.82	2,084.8
ET-22-432	Sulphide Zone	348.40	383.20	34.80	0.13	257.4	0.47	1.18	2.02	407.4
ET-22-432	including	372.40	380.60	8.20	0.13	956.6	1.69	3.58	7.01	1,446.2
ET-22-432	including	378.50	380.60	2.10	0.17	1,663.5	4.32	6.28	11.5	2,622.5
ET-22-434	HW Gold Zone	6.50	15.00	8.50	0.39	96.7	0.01	0.08	0.01	129.8
ET-22-434	including	10.10	11.60	1.50	2.04	336.0	0.02	0.17	0.04	496.6
ET-22-434	Sooy Vein Zone	184.60	197.60	13.00	0.16	172.5	0.36	2.08	2.56	354.0
ET-22-434	including	190.60	195.80	5.20	0.19	326.4	0.84	4.89	5.97	734.8
ET-22-434	including	192.30	194.40	2.10	0.24	496.1	1.76	7.17	8.62	1,139.2
ET-22-434	Sulphide Zone	306.70	313.30	6.60	0.11	376.8	0.32	0.14	0.09	421.9
ET-22-434	including	311.30	313.30	2.00	0.20	1,116.5	0.89	0.20	0.11	1,225.9
ET-22-434	including	312.30	313.30	1.00	0.25	1,859.0	1.44	0.33	0.19	2,030.4
ET-22-434	Sulphide Zone	361.70	381.60	19.90	0.22	605.6	1.13	4.04	7.43	1,072.9
ET-22-434	including	370.10	380.60	10.50	0.20	914.0	1.68	5.92	12.42	1,642.4
ET-22-438	HW Gold Zone	9.20	52.70	43.50	0.16	18.7	0.00	0.01	0.01	31.3
ET-22-438	including	29.00	29.60	0.60	2.26	566.0	0.02	0.02	0.02	739.5
ET-22-438	Sooy Vein Zone	321.30	324.50	3.20	0.52	300.5	0.18	0.73	0.33	384.8
ET-22-438	including	322.10	323.00	0.90	1.51	773.0	0.48	2.08	0.77	1,006.7
ET-22-438	Sulphide Zone	393.50	413.00	19.50	0.27	408.4	0.53	0.88	0.83	527.5
ET-22-438	including	394.80	398.50	3.70	0.24	879.4	0.76	0.86	1.36	1,035.8
ET-22-438	including	396.00	396.50	0.50	0.62	2,796.0	1.75	1.85	4.16	3,193.1
ET-22-438	including	403.80	411.80	8.00	0.51	564.3	0.90	1.68	1.31	771.9

TABLE 10.6
SIGNIFICANT 2022 MINERALIZED DRILL INTERCEPTS IN THE EL TIGRE, SEITZ KELLY VEINS AND SULPHIDE ZONE

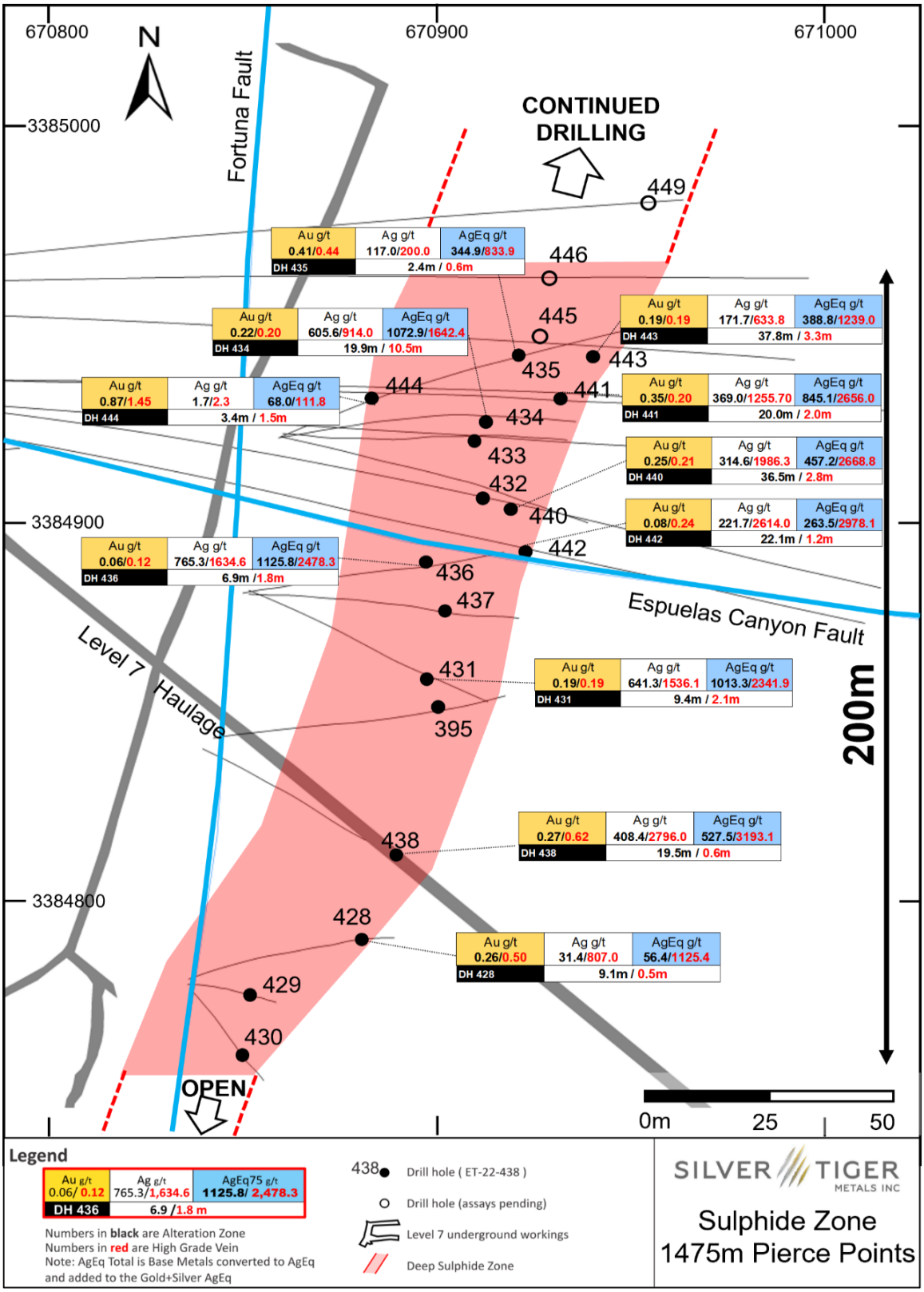
Drill Hole ID	Target / Comment	From (m)	To (m)	Length (m)[1]	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)[2]
ET-22-438	including	406.30	407.00	0.70	1.47	1,148.0	1.40	1.52	0.71	1,452.3
ET-22-441	Sulphide Zone	415.30	435.30	20.00	0.35	369.0	0.85	4.11	8.15	845.1
ET-22-441	including	432.50	434.50	2.00	0.20	1,255.7	2.02	12.67	26.87	2,656.0
ET-22-441	including	432.50	433.80	1.30	0.21	1,484.1	2.71	14.67	29.82	3,097.9
ET-22-442	Sooy Vein	363.50	364.50	1.00	0.01	453.0	0.56	0.00	0.11	511.6
ET-22-442	Sulphide Zone	370.00	392.10	22.10	0.08	221.7	0.21	0.28	0.28	263.5
ET-22-442	including	378.20	379.40	1.20	0.24	2,614.0	2.67	1.82	1.38	2,978.1

Notes:

1. Not true thickness

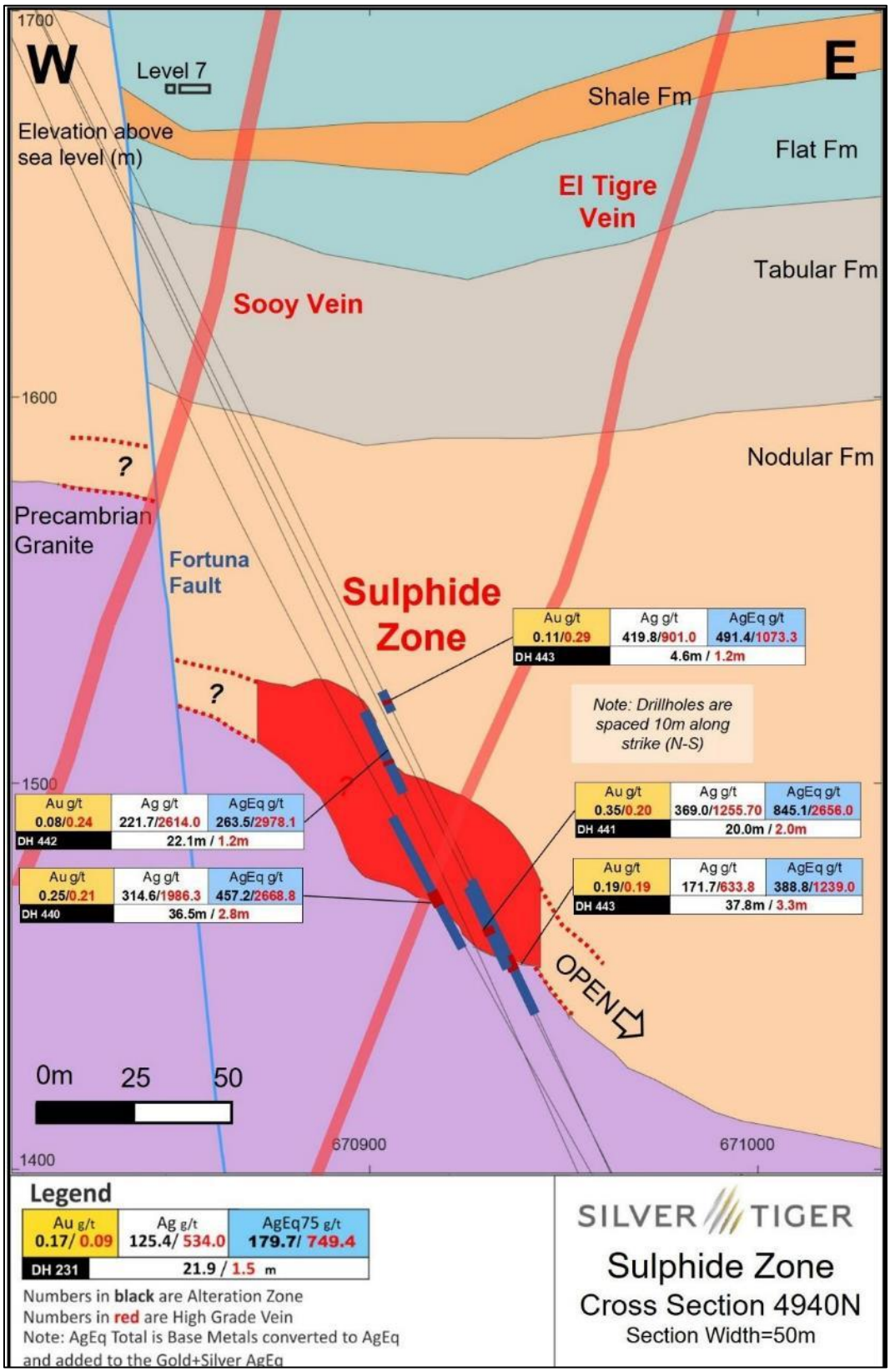
2. Silver Equivalent ("AgEq") ratios are based on a silver to gold price ratio of 75:1 (Au:Ag). Copper, lead and zinc are converted using $3.66/lb copper, $0.90/lb lead, $1.26/lb zinc at 100% metal recoveries based on a silver price of $26.00/oz.

FIGURE 10.13 SULPHIDE ZONE PLAN



Source: *Silver Tiger website (2023)*

FIGURE 10.14 SULPHIDE ZONE CROSS-SECTIONAL PROJECTION 4940 N



Source: *Silver Tiger website (2023)*

Select intersection highlights are listed in Table 10.6 above.

In addition, drill hole ET-21-329 was a vertical drill hole designed to test the width of disseminated mineralization in the black shale away from Benjamin Vein (see Figure 10.12). Drill hole ET-22-378 also tested the Black Shale Zone. Select highlight intersections are as follows:

- **Drill Hole ET-21-329**: 1.0 m grading 2,338.5 g/t AgEq from 128.9 to 129.9 m, consisting of 1,624.5 g/t Ag, 0.28 g/t Au, 0.10% Cu, 1.36% Pb and 19.63% Zn within 9.1 m grading 809.7 g/t AgEq from 124.0 to 133.1 m, consisting of 429.5 g/t Ag, 0.22 g/t Au, 0.22% Cu, 4.82% Pb and 6.89% Zn; and

- **Drill Hole ET-22-378**: 1.0 m grading 1,095.8 g/t total AgEq from 89.9 to 90.9 m, consisting of 1,002.0 g/t Ag, 0.25 g/t Au, 0.39% Cu, 0.60% Pb and 0.69% Zn within 6.5 m grading 524.9 g/t AgEq from 89.9 to 96.4 m consisting of 469.4 g/t Ag, 0.12 g/t Au, 0.16% Cu, 0.50% Pb and 0.58% Zn.

Drilling also continued on the Seitz Kelly Vein. Drill hole ET-22-349 intersected the Sietz Kelly Vein and a shale horizon beyond the main vein. Select highlights include:

- **Drill Hole ET-308**: 0.7 m grading 1,203.8 g/t AgEq from 142.7 to 143.4 m, consisting of 1,022.0 g/t Ag, 0.23 g/t Au, 0.55% Cu, 3.16% Pb and 1.08% Zn; and

- **Drill Hole ET-22-349**: 0.7 m grading 6,182.0 g/t AgEq from 236.3 to 237.0 m, consisting of 6,063.0 g/t Ag, 0.22 g/t Au, 1.00% Cu, 0.14% Pb and 0.09% Zn within 10.1 m grading 474.2 g/t AgEq from 228.3 to 238.4 m consisting of 429.5 g/t Ag, 0.46 g/t Au, 0.07% Cu, 0.04% Pb, and 0.06% Zn in the Shale Horizon at the Seitz Kelly Vein.

Two drill hole composites were sampled and shipped to SGS Lakefield for flotation scoping testwork. Details of this work are provided in Section 13. Drill hole ET-22-436 was advanced in the Black Shale Zone and ET-22-443 was advanced in the Sulphide Zone.

10.5 2023 DRILLING

Drilling in 2023 continued with five drill rigs, three normal drill rigs and two portable drill rigs operated by Energold and Role. A total of 119 drill holes for 29,339 m of drilling were completed in 2023. Significant intersections from the 2023 drilling are listed in Tables 10.7 and 10.8.

Drilling continued on the Sulphide Zone, originally discovered in 2022. Drilling on the Sulphide Zone in 2023 expanded the known mineralization. A longitudinal projection is presented Figure 10.15. Drilling was also focused on converting the first two to three years of mineralized material in the potential "Starter Stockwork Zone" from Indicated Mineral Resource to Measured, and converting portions of the 2 km strike length "ultimate" PEA pit-constrained Inferred Mineral Resource to Indicated (Figures 10.16, 10.17 and 10.18). Drilling was also completed on the Keel Zone (Figure 10.18).

TABLE 10.7
SELECTED 2023 MINERALIZED DRILL INTERCEPTS

Drill Hole ID	Target / Comment	From (m)	To (m)	Length (m)[1]	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)[2]
ET-23-457	Sulphide Zone	441.00	479.70	38.70	0.11	297.5	0.28	1.42	2.19	438.9
ET-23-457	including	441.00	456.80	15.80	0.17	692.6	0.60	2.99	4.61	986.8
ET-23-457	including	445.00	451.30	6.30	0.15	1,100.1	0.87	5.10	8.01	1,581.4
ET-23-457	including	445.00	448.40	3.40	0.13	1,245.7	1.00	6.38	13.43	1,948.8
ET-23-457	including	447.80	448.40	0.60	0.15	1,943.0	1.58	7.29	10.95	2,642.2
ET-23-464	HW Gold Zone	487.00	490.00	3.00	0.04	63.1	0.08	0.24	0.03	80.5
ET-23-464	including	488.30	489.00	0.70	0.04	167.0	0.18	0.62	0.06	204.0
ET-23-464	HW Gold Zone	497.00	509.00	12.00	0.07	31.1	0.13	0.48	0.37	72.4
ET-23-464	including	497.00	500.00	3.00	0.08	69.2	0.09	0.73	1.16	139.9
ET-23-464	Sulphide Zone	538.00	551.90	13.90	0.23	27.4	0.02	1.16	6.22	280.5
ET-23-464	including	541.50	548.20	6.70	0.33	43.1	0.03	1.72	11.61	495.9
ET-23-464	including	547.00	547.70	0.70	0.21	38.9	0.04	2.55	24.64	936.4
ET-23-464	and	565.10	568.20	3.10	0.26	14.4	0.02	0.18	0.71	63.8
ET-23-456	Sulphide Zone	280.80	285.50	4.70	0.15	344.3	0.44	0.65	0.99	446.2
ET-23-456	including	282.50	284.90	2.40	0.24	653.1	0.85	1.26	1.94	847.4
ET-23-456	including	284.10	284.90	0.80	0.48	1,145.0	1.57	2.76	1.63	1,452.0
ET-23-456	El Tigre Vein Zone	403.70	404.20	0.60	0.22	30.4	0.03	1.75	1.68	146.8
ET-23-456	Sulphide Zone	616.70	617.30	0.50	0.26	1,294.0	0.62	3.76	6.30	1,670.4

Notes:

1. *Not true thickness*
2. *Silver Equivalent ("AgEq") ratios are based on a silver to gold price ratio of 75:1 (Au:Ag). Copper, lead and zinc are converted using $3.66/lb copper, $0.90/lb lead, $1.26/lb zinc at 100% metal recoveries based on a silver price of $26.00/oz.*

TABLE 10.8
SELECTED 2023 MINERALIZED INTERCEPTS - PRECIOUS METALS ONLY

Drill Hole ID	Target / Comment	From (m)	To (m)	Length (m)[1]	Au (g/t)	Ag (g/t)	AuEq (g/t)[2]	AgEq (g/t)[2]
ET-23-479	Stockwork Zone	22.00	76.00	54.00	0.71	10.1	0.85	63.7
ET-23-479	Stockwork Zone	96.50	150.00	53.50	1.21	19.5	1.47	110.0
ET-23-479	including	97.90	120.00	22.10	1.98	5.9	2.06	154.4
ET-23-479	including	132.00	147.00	15.00	0.96	29.9	1.36	101.8
ET-23-479	and	166.90	173.50	6.60	0.18	172.5	2.48	186.0
ET-23-479	Stockwork	44.00	120.00	76.00	0.30	14.8	0.50	37.3
ET-23-486	Stockwork Zone[3]	104.00	183.00	79.00	0.44	31.3	0.86	64.5
ET-23-486	Mining Void	155.00	158.00	3.00	*	*	*	*
ET-23-486	Including	158.00	184.00	26.00	0.46	85.2	1.60	119.7
ET-23-489	Stockwork Zone[4]	101.00	190.50	89.50	0.60	22.3	0.89	67.1
ET-23-489	Including	110.00	120.00	10.00	1.45	4.1	1.50	112.6
ET-23-489	Including	142.50	151.40	8.90	1.69	14.3	1.88	141.0
ET-23-489	Mining Void	173.90	174.60	0.70	*	*	*	*
ET-23-494	Stockwork Zone	0.00	78.10	78.10	0.45	22.9	0.75	56.5
ET-23-494	including	3.00	18.30	15.30	0.56	28.1	0.93	70.1
ET-23-494	including	67.00	75.00	8.00	0.67	69.8	1.60	120.2
ET-23-494	Mining Void	78.10	81.70	3.60	*	*	*	*
ET-23-497	Stockwork Zone	37.00	153.00	116.00	0.40	21.1	0.68	51.2
ET-23-497	including	72.00	134.00	62.00	0.50	28.5	0.88	65.7
ET-23-497	including	72.00	96.00	24.00	0.76	13.9	0.94	70.8
ET-23-497	including	107.00	132.00	25.00	0.37	46.8	0.99	74.4
ET-23-504	Stockwork Zone	4.00	11.10	7.10	0.72	1.8	0.75	56.2
ET-23-504	Stockwork Zone	37.20	153.00	115.80	0.49	9.2	0.61	46.1
ET-23-504	including	37.20	60.70	23.50	0.89	15.2	1.09	81.9
ET-23-506	Stockwork Zone[5]	6.00	86.00	80.00	0.75	25.6	1.09	82.0

TABLE 10.8
SELECTED 2023 MINERALIZED INTERCEPTS - PRECIOUS METALS ONLY

Drill Hole ID	Target / Comment	From (m)	To (m)	Length (m)[1]	Au (g/t)	Ag (g/t)	AuEq (g/t)[2]	AgEq (g/t)[2]
ET-23-506	including[5]	34.50	82.00	47.50	1.08	34.7	1.55	115.9
ET-23-506	Mining Void	69.20	71.60	2.40	*	*	*	*
ET-23-506	Stockwork Zone	98.00	124.40	26.40	0.37	1.5	0.39	29.3
ET-23-506	Mining Void	124.40	127.90	3.50	*	*	*	*
ET-23-514	Stockwork Zone	69.00	74.00	5.00	0.34	1.9	0.36	27.2
ET-23-514	Stockwork Zone	102.00	126.00	24.00	0.46	4.9	0.52	39.2
ET-23-514	including	115.00	124.00	9.00	0.91	3.1	0.95	71.1
ET-23-514	El Tigre & SK Keel	167.00	201.40	34.40	0.72	78.0	1.76	131.9
ET-23-514	El Tigre Keel	168.00	172.50	4.50	1.48	172.0	3.77	283.0
ET-23-514	SK Keel	188.60	194.00	5.40	2.84	330.4	7.25	543.6
ET-23-514	including	188.60	189.40	0.80	16.6	1,018.0	30.17	2,263.0
ET-23-527	Stockwork Zone	0.00	74.00	74.00	0.55	17.0	0.78	58.5
ET-23-527	including	5.40	14.60	9.20	0.90	96.3	2.19	164.0
ET-23-527	including	48.80	74.00	25.20	1.04	5.3	1.11	83.4
ET-23-529	Stockwork Zone	107.00	149.50	42.50	0.91	5.6	0.99	73.9
ET-23-529	including	107.00	140.00	33.00	1.09	6.6	1.18	88.7
ET-23-529	including	107.00	115.00	8.00	1.57	1.3	1.58	118.7
ET-23-529	including	122.00	140.00	18.00	1.27	11.2	1.42	106.2
ET-23-530	Stockwork Zone	28.50	39.40	10.90	0.51	2.4	0.54	40.6
ET-23-530	Stockwork Zone	44.30	101.20	56.90	0.68	3.7	0.73	54.9
ET-23-530	including	73.50	101.20	27.70	0.86	4.6	0.92	69.3
ET-23-530	including	82.50	101.20	18.70	0.93	6.1	1.01	75.8
ET-23-541	Stockwork Zone	92.50	172.00	79.50	0.52	17.4	0.75	56.1
ET-23-541	including	94.00	120.50	26.50	1.08	8.0	1.19	89.3
ET-23-541	Seitz Kelly	187.00	197.20	10.20	0.14	26.8	0.50	37.6
ET-23-543	Stockwork Zone[6]	148.00	188.20	40.20	0.84	27.9	1.21	90.7

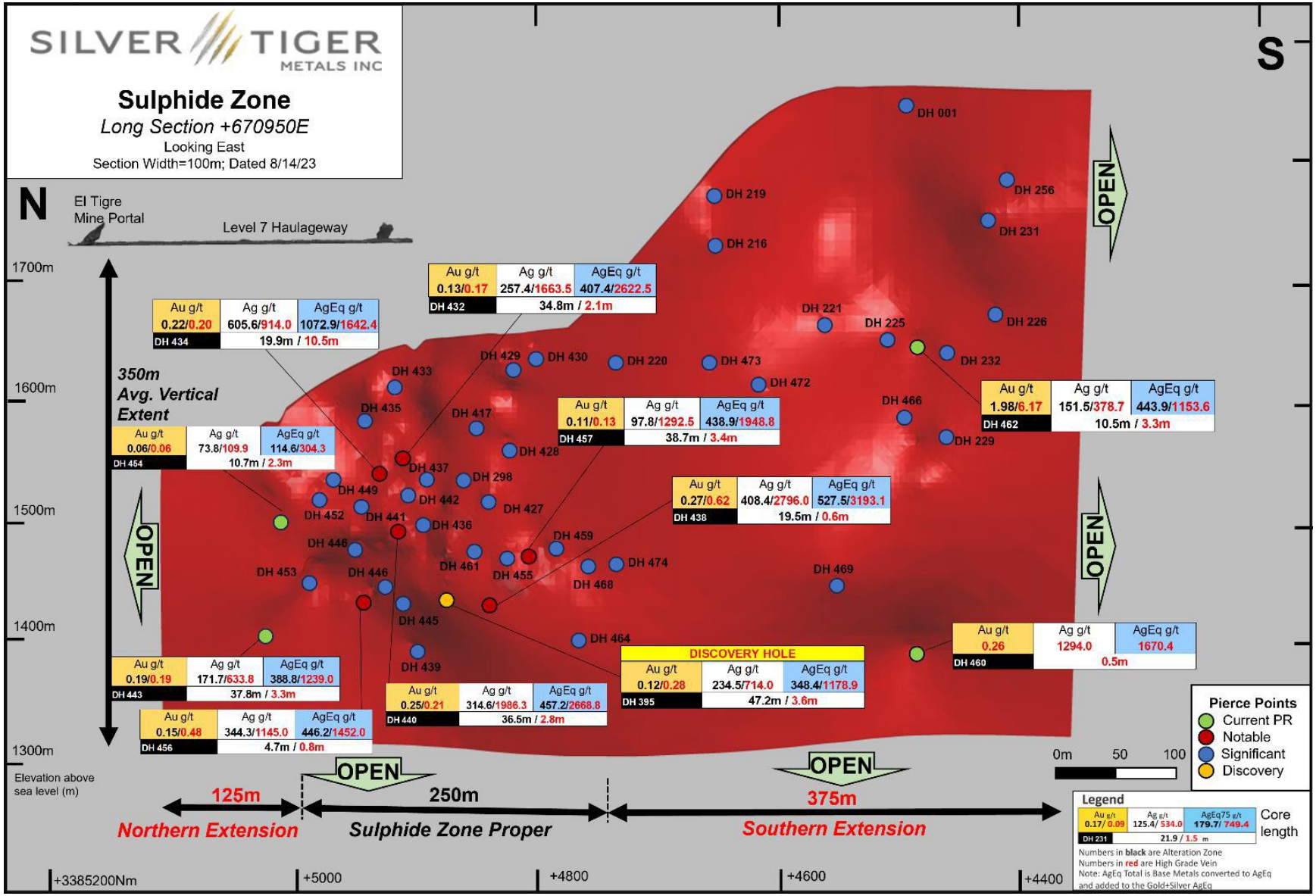
TABLE 10.8
SELECTED 2023 MINERALIZED INTERCEPTS - PRECIOUS METALS ONLY

Drill Hole ID	Target / Comment	From (m)	To (m)	Length (m)[1]	Au (g/t)	Ag (g/t)	AuEq (g/t)[2]	AgEq (g/t)[2]
ET-23-543	including	152.00	175.00	23.00	1.22	37.9	1.72	129.3
ET-23-543	Mining Void	183.00	184.70	1.70	*	*	*	*
ET-23-548	Stockwork Zone	0.00	71.00	71.00	0.62	3.9	0.67	50.6
ET-23-548	including	39.50	58.00	18.50	1.32	2.4	1.36	101.7
ET-23-548	Stockwork Zone	86.30	97.50	11.30	0.68	0.7	0.69	52.0
ET-23-550	Stockwork Zone	0.00	21.00	21.00	0.53	10.5	0.67	50.1
ET-23-550	Stockwork Zone	43.50	124.50	81.10	0.90	3.0	0.94	70.3
ET-23-550	including	54.00	86.00	32.00	1.31	3.6	1.36	102.1
ET-23-552	Stockwork Zone[7]	107.00	142.50	35.50	0.83	6.2	0.92	68.7
ET-23-552	Mining Void	121.70	123.80	2.10	*	*	*	*
ET-23-552	including	135.50	142.50	7.00	1.66	1.9	1.68	126.3
ET-23-552	Stockwork Zone	156.50	176.00	19.50	0.44	0.5	0.45	33.4
ET-23-563	Stockwork Zone	4.00	49.00	45.00	0.70	35.0	1.17	87.6
ET-23-563	including	11.00	26.00	15.00	1.01	89.0	2.2	164.7
ET-23-563	including	24.00	26.00	2.00	1.43	634.0	9.89	741.5
ET-23-563	Stockwork Zone	64.00	70.00	6.00	0.68	2.0	0.71	53.1

Notes:

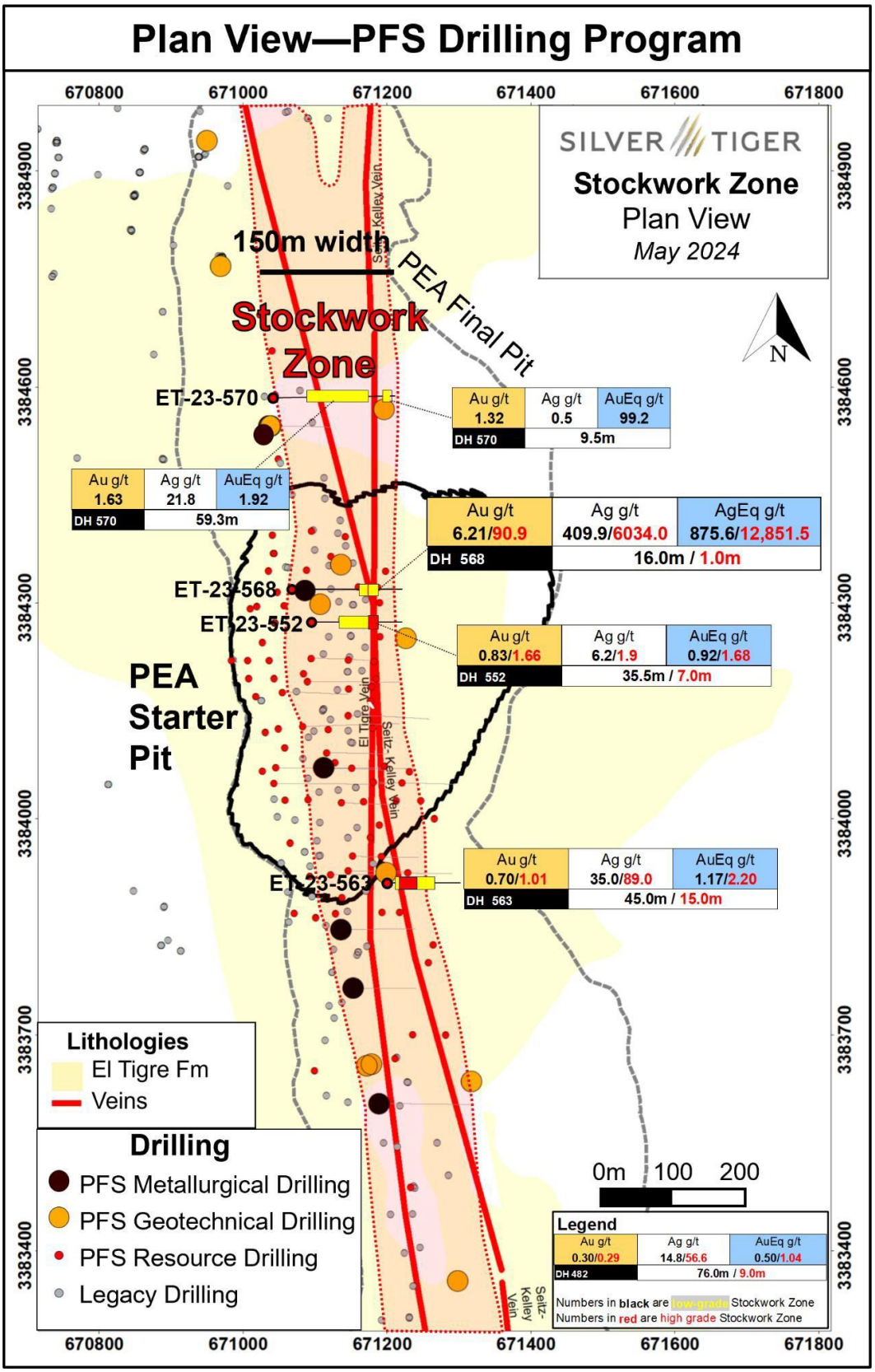
1. *Not true thickness.*
2. *Silver Equivalent ("AgEq") ratios are based on a silver to gold price ratio of 75:1 (Au:Ag).*
3. *Excludes 3.0 m of mining void.*
4. *Excludes 0.7 m of mining void.*
5. *Excludes 3.0 m mining void.*
6. *Excludes 2.0 m of mining void.*
7. *Excludes 2.1 m of mining void.*

FIGURE 10.15 SULPHIDE ZONE LONGITUDINAL PROJECTION



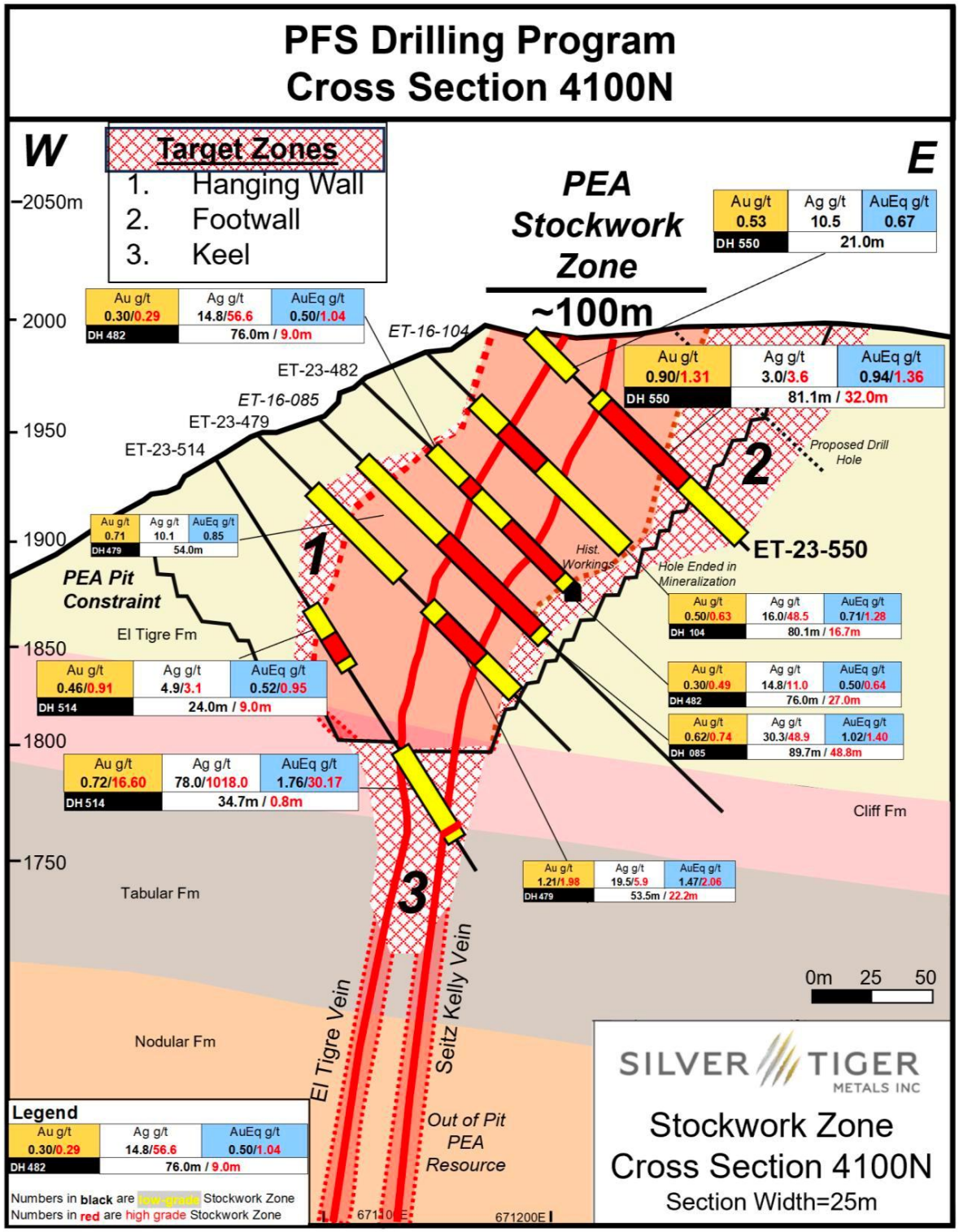
Source: *Silver Tiger website (2023)*

FIGURE 10.16 STOCKWORK ZONE PLAN VIEW



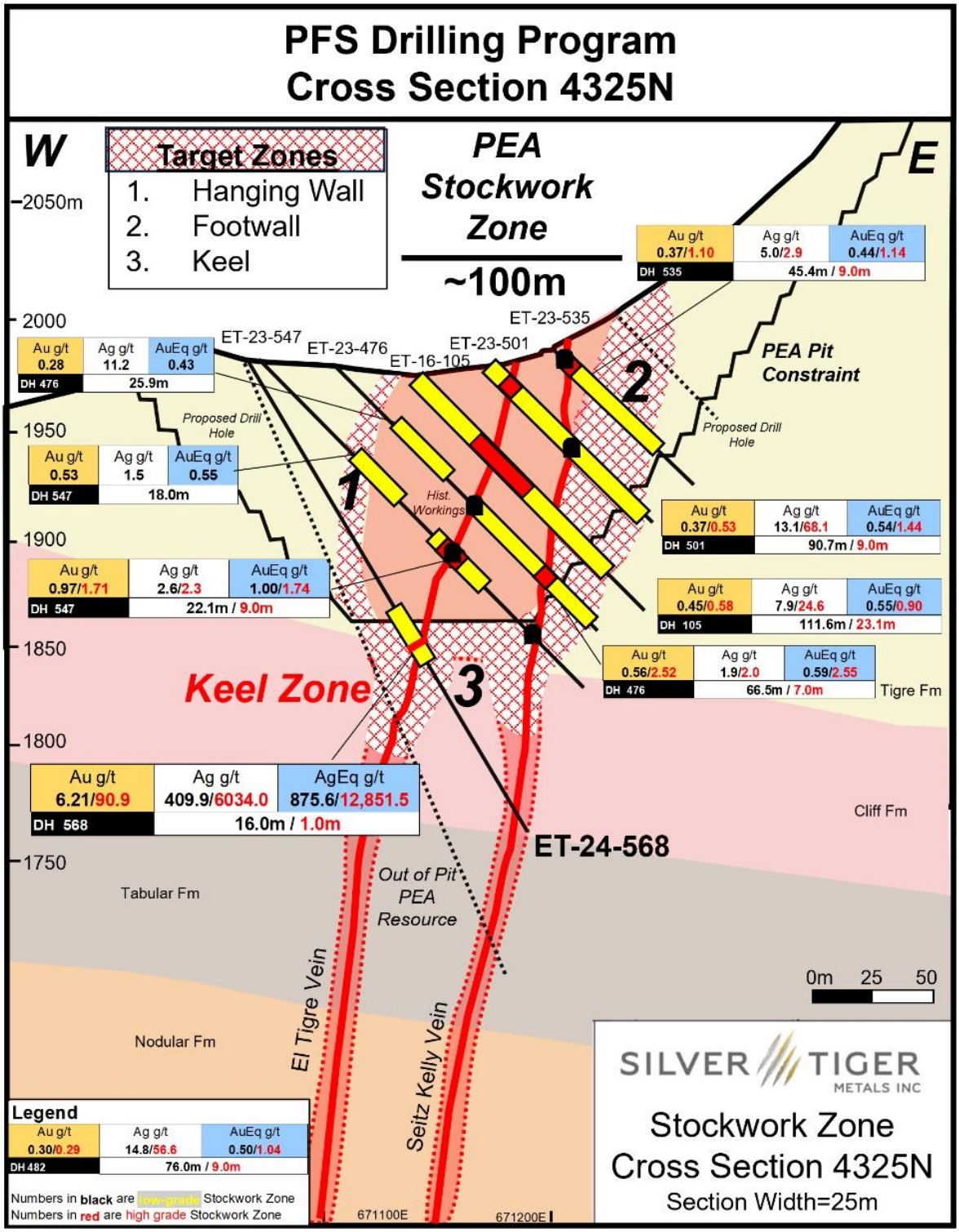
Source: *Silver Tiger website (2023)*

FIGURE 10.17 STOCKWORK ZONE CROSS-SECTIONAL PROJECTION 4100 N



Source: *Silver Tiger website (2023)*

FIGURE 10.18 STOCKWORK ZONE CROSS-SECTIONAL PROJECTION 4325 N



Source: *Silver Tiger website (2023)*

Select additional highlight drill hole intersections from the 2023 program are as follows:

- **Drill Hole ET-23-486:** 79.0 m grading 0.86 g/t AuEq or 64.5 g/t AgEq from 104.0 to 183.0 m, consisting of 31.3 g/t Ag and 0.44 g/t Au including 26.0 m grading 1.60 g/t AuEq or 119.7 g/t AgEq from 158.0 to 184.0 m consisting of 85.2 g/t Ag and 0.46 g/t Au in the Stockwork Zone;

- **Drill Hole ET-23-489**: 89.5 m grading 0.89 g/t AuEq or 67.1 g/t AgEq from 101.0 to 190.5 m, consisting of 22.3 g/t Ag and 0.60 g/t Au including 10.0 m grading 1.50 g/t AuEq or 112.6 g/t AgEq from 110.0 to 120.0 m consisting of 4.1 g/t Ag and 1.45 g/t Au in the Stockwork Zone;

- **Drill Hole ET-23-494**: 78.1 m grading 0.75 g/t AuEq or 56.5 g/t AgEq from 0.0 to 78.1 m, consisting of 22.9 g/t Ag and 0.45 g/t Au including 15.3 m grading 0.93 g/t AuEq or 70.1 g/t AgEq from 3.0 to 18.3 m consisting of 28.1 g/t silver and 0.56 g/t gold in the Stockwork Zone;

- **Drill Hole ET-23-497:** 116.0 m grading 0.68 g/t AuEq or 51.2 g/t AgEq from 37.0 to 153.0 m, consisting of 0.40 g/t Au and 21.1 g/t Ag including 25.0 m grading 0.99 g/t AuEq or 74.4 g/t AgEq from 107.0 to 132.0 m consisting of 0.37 g/t Au and 46.8 g/t Ag in the Stockwork Zone; and

- **Drill Hole ET-23-518:** 16.9 m grading 0.96 g/t AuEq or 72.3 g/t AgEq from 97.7 to 114.6 m, consisting of 0.07 g/t Au and 66.9 g/t Ag including 2.0 m grading 6.7 g/t AuEq or 502.5 g/t AgEq from 108.0 to 110.0 m consisting of 0.20 g/t Au and 487.5 g/t Ag in the Keel Zone.

10.6 2024 DRILLING

Drilling in 2024 continued with three drill rigs, two normal drill rigs and a portable drill rig operated by Role. A total of 62 drill holes, totalling 9,712 m of surface drilling were completed (collar locations in Appendix F). Significant intersections from the 2024 drilling are listed in Tables 10.9 and 10.10.

In 2024, drilling continued to focus on the expansion of mineralization along the Footwall of the PEA "starter pit" and the Stockwork Zone comprised of Footwall, Hanging Wall and Keel Zones. Drill hole ET-24-568 expanded the depth and strike of the Keel Zone over the 2023 Mineral Resource Estimate. Plan views and cross-sections are presented in Figures 10.19 to 10.22.

TABLE 10.9
SELECTED 2024 MINERALIZED INTERCEPTS

Drill Hole ID	Target / Comment	From (m)	To (m)	Length (m)[1]	Au (g/t)	Ag (g/t)	AuEq (g/t)[2]	AgEq (g/t)[2]
ET-24-571	Stockwork Zone	0.00	44.20	44.20	0.49	38.4	1.00	75.1
ET-24-571	including	24.00	36.70	12.70	0.89	84.1	2.01	151.0
ET-24-571	Stockwork Zone	55.50	112.50	57.00	0.31	1.1	0.32	24.0
ET-24-571	Stockwork Zone	122.50	127.00	4.50	1.55	61.5	2.37	177.8
ET-24-576	Stockwork Zone	88.00	95.50	7.50	0.43	0.5	0.44	32.9
ET-24-576	Stockwork Zone[3]	129.00	170.10	41.10	0.92	8.5	1.03	77.5
ET-24-576	including	140.90	155.10	14.20	1.52	7.5	1.62	121.8
ET-24-576	Mining Void	166.00	167.20	1.20	*	*	*	*
ET-24-580	Stockwork Zone	25.90	53.00	27.10	0.68	4.6	0.74	55.5
ET-24-580	Stockwork Zone	84.00	93.00	9.00	1.60	11.5	1.75	131.4
ET-24-586	Stockwork Zone	1.00	19.20	18.20	0.53	6.6	0.62	46.2
ET-24-586	Mining Void	32.00	33.70	1.70	*	*	*	*
ET-24-586	El Tigre Vein	33.70	39.50	5.80	0.43	328.3	4.81	360.6
ET-24-586	including	33.70	35.50	1.80	1.10	1,041	14.98	1,123.4
ET-24-586	Stockwork Zone	49.50	57.00	7.50	0.52	4.8	0.59	44.1
ET-24-586	Seitz Kelly	66.00	78.40	12.40	0.54	13.7	0.72	54.1
ET-24-588	Stockwork Zone[4]	76.50	98.50	22.00	0.62	54.7	1.35	101.5
ET-24-588	Stockwork Zone	114.00	122.50	8.50	0.37	1.8	0.39	29.2
ET-24-588	Stockwork Zone	146.50	157.50	11.00	0.16	127.7	1.86	139.4
ET-24-588	including	147.50	152.00	4.50	0.25	268.8	3.83	287.4
ET-24-590	Stockwork Zone	148.00	220.00	72.00	0.62	14.5	0.81	60.8
ET-24-590	including	181.50	205.50	24.00	1.02	14.0	1.20	90.2
ET-24-600	Stockwork Zone	19.50	66.00	46.50	0.37	34.3	0.83	62.1
ET-24-600	including	19.50	32.80	13.30	0.49	105.6	1.90	142.5
ET-24-600	and	107.00	111.00	4.00	0.21	207.0	2.97	222.6
ET-24-600	and	118.00	123.00	5.00	0.05	71.8	1.01	75.7

TABLE 10.9
SELECTED 2024 MINERALIZED INTERCEPTS

Drill Hole ID	Target / Comment	From (m)	To (m)	Length (m)[1]	Au (g/t)	Ag (g/t)	AuEq (g/t)[2]	AgEq (g/t)[2]
ET-24-602	Stockwork Zone	49.50	142.00	92.50	0.45	15.5	0.66	49.2
ET-24-602	including	78.00	103.00	25.00	0.88	15.9	1.09	82.1
ET-24-602	including	130.40	141.00	10.60	0.57	86.6	1.72	129.4
ET-24-607	Stockwork Zone[5]	7.90	128.40	120.50	0.58	8.1	0.69	51.9
ET-24-607	and	87.80	95.40	7.60	1.46	15.4	1.67	125.1
ET-24-609	Stockwork Zone	3.70	27.90	24.20	0.76	0.8	0.77	58.1
ET-24-609	Stockwork Zone	51.40	105.20	53.80	0.56	2.8	0.60	44.7
ET-24-609	El Tigre Vein Keel	124.40	164.40	40.00	0.09	74.1	1.08	80.7
ET-24-609	including	145.10	162.40	17.30	0.12	134.7	1.92	143.8
ET-24-609	Seitz Kelly Vein Keel	178.40	229.10	50.70	0.17	142.1	2.07	155.0
ET-24-609	including	198.40	212.70	14.30	0.20	319.5	4.46	334.3
ET-24-615	Stockwork Zone[6]	0.00	18.40	18.40	1.90	122.2	3.53	264.5
ET-24-615	Stockwork Zone	34.00	38.40	4.40	0.61	56.7	1.37	102.4
ET-24-615	Stockwork Zone	66.30	88.20	21.90	0.40	35.2	0.86	64.8
ET-24-615	Stockwork Zone	96.20	118.40	22.20	0.22	18.2	0.46	34.7
ET-24-615	Stockwork Zone	129.30	135.30	6.00	0.29	22.2	0.58	43.7
ET-24-616	Stockwork Zone	61.40	93.30	31.90	0.14	128.5	1.85	138.7
ET-24-616	including	69.40	78.70	9.30	0.27	399.4	5.60	419.9
ET-24-616	including	77.40	78.70	1.30	1.07	2,698	37.04	2,778.3
ET-24-619	Stockwork Zone[7]	3.30	28.00	24.70	0.39	50.6	1.06	79.9
ET-24-619	Mining Void	17.60	20.60	3.00	*	*	*	*
ET-24-619	Mining Void	21.20	25.00	3.80	*	*	*	*
ET-24-619	Stockwork Zone	41.40	60.30	18.90	0.31	7.4	0.41	30.6

Notes:

1. *Not true thickness.*
2. *Silver Equivalent ("AgEq") ratios are based on a silver to gold price ratio of 75:1 (Au:Ag).*
3. *Excludes 1.2 m historical mining void.*
4. *Excludes 1.8 m historical mining void.*

5. *Excludes 2.5 m historical mining void.*
6. *Excludes 1.1 m historical mining void.*
7. *Excludes 6.8 m historical mining void.*

TABLE 10.10
GEOTECHNICAL DRILL HOLE RESULTS

Drill Hole ID	Target / Comment	From (m)	To (m)	Length (m)[1]	Au (g/t)	Ag (g/t)	AuEq (g/t)[2]	AgEq (g/t)[2]
ET-23-513	Stockwork Zone	7.40	134.00	126.60	0.55	16.6	0.77	57.5
ET-23-513	including	32.00	56.20	24.20	0.77	39.1	1.29	96.5
ET-23-513	including	94.00	113.60	19.60	1.03	48.8	1.68	126.2
ET-23-522	Margin - Stockwork	No Significant Assays						
ET-23-531	Stockwork Zone	35.00	53.50	18.50	0.47	0.9	0.48	36.1
ET-23-531	Stockwork Zone	101.00	102.50	1.50	6.10	8.9	6.22	466.3
ET-23-531	Stockwork Zone	121.40	144.80	23.40	0.29	75.7	1.30	97.6
ET-23-531	including	140.00	142.20	2.20	1.20	791.4	11.75	881.5
ET-23-531	Stockwork Zone	187.90	189.50	1.60	3.47	28.7	3.85	288.8
ET-23-531	Stockwork Zone	200.00	201.70	1.70	1.59	8.2	1.70	127.5
ET-23-542	Margin - Stockwork	No Significant Assays						
ET-23-544	Margin - Stockwork	4.60	6.10	1.50	0.52	6.6	0.61	45.7
ET-23-544	and	82.50	86.00	3.50	0.24	0.6	0.25	18.6
ET-23-544	and	120.50	122.50	2.00	0.70	2.5	0.74	55.3
ET-23-544	and	143.30	144.00	0.70	3.63	1.0	3.65	273.6
ET-23-546	Margin - Stockwork	No Significant Assays						
ET-23-549	Margin - Stockwork	No Significant Assays						
ET-23-551	Margin - Stockwork	105.50	108.00	2.50	0.63	1.1	0.64	48.4
ET-23-551	and	119.00	122.00	3.00	0.41	0.5	0.41	30.9
ET-23-555	Stockwork Zone	11.00	19.80	8.80	0.30	16.2	0.51	38.6
ET-23-555	Stockwork Zone	79.90	149.00	69.10	0.57	12.2	0.73	54.8
ET-23-555	including	82.00	89.00	7.00	2.12	0.7	2.13	160.0

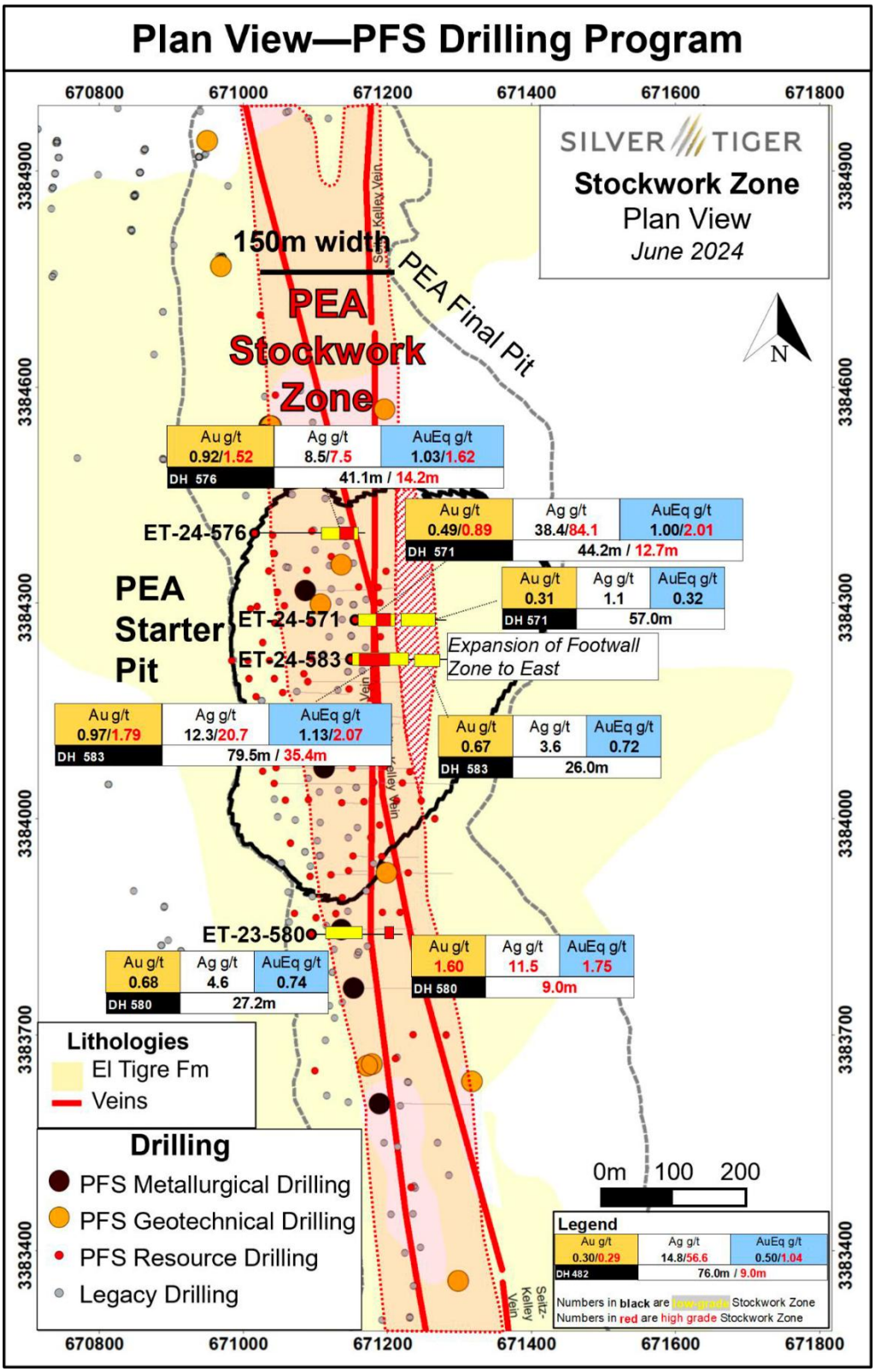
TABLE 10.10
GEOTECHNICAL DRILL HOLE RESULTS

Drill Hole ID	Target / Comment	From (m)	To (m)	Length (m)[1]	Au (g/t)	Ag (g/t)	AuEq (g/t)[2]	AgEq (g/t)[2]
ET-23-555	SK Vein	203.90	206.20	2.30	4.16	32.9	4.59	344.5
ET-23-555	Mining Void	208.20	211.80	3.60	*	*	*	*
ET-23-555	and	211.80	213.10	1.30	0.09	7.0	0.18	13.4
ET-23-555	Mining Void	213.10	218.20	5.10	*	*	*	*
ET-23-556	Margin - Stockwork	No Significant Assays						
ET-23-559	Stockwork Zone	2.40	29.50	27.10	0.31	30.7	0.72	54.1
ET-23-559	including	13.90	16.90	3.00	1.49	239.3	4.68	351.2
ET-23-559	Seitz Kelly Vein	74.70	79.00	4.30	0.29	21.1	0.57	42.7
ET-23-561	Margin - Stockwork	No Significant Assays						
ET-23-562	Stockwork Zone	25.30	31.00	5.70	0.59	0.5	0.60	44.8
ET-23-562	Stockwork Zone	55.90	174.60	118.70	0.46	3.5	0.51	38.3
ET-23-562	Seitz Kelly Vein	165.00	174.60	9.60	0.80	28.3	1.18	88.6
ET-23-562	Mining Void	174.60	179.20	4.60	*	*	*	*

Notes:

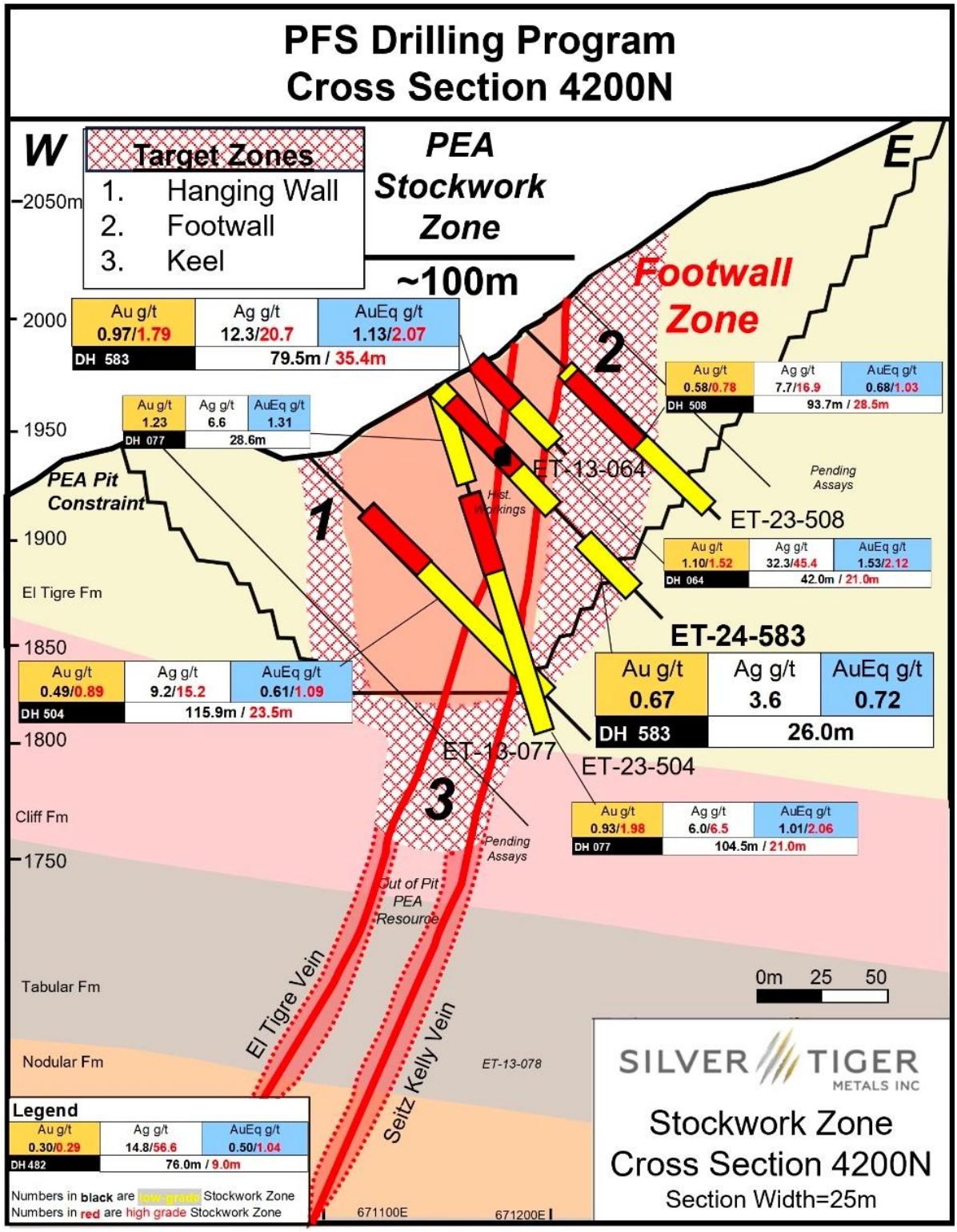
1. *Not true thickness.*
2. *Silver Equivalent ("AgEq") ratios are based on a silver to gold price ratio of 75:1 (Au:Ag).*

FIGURE 10.19 STOCKWORK ZONE PLAN VIEW



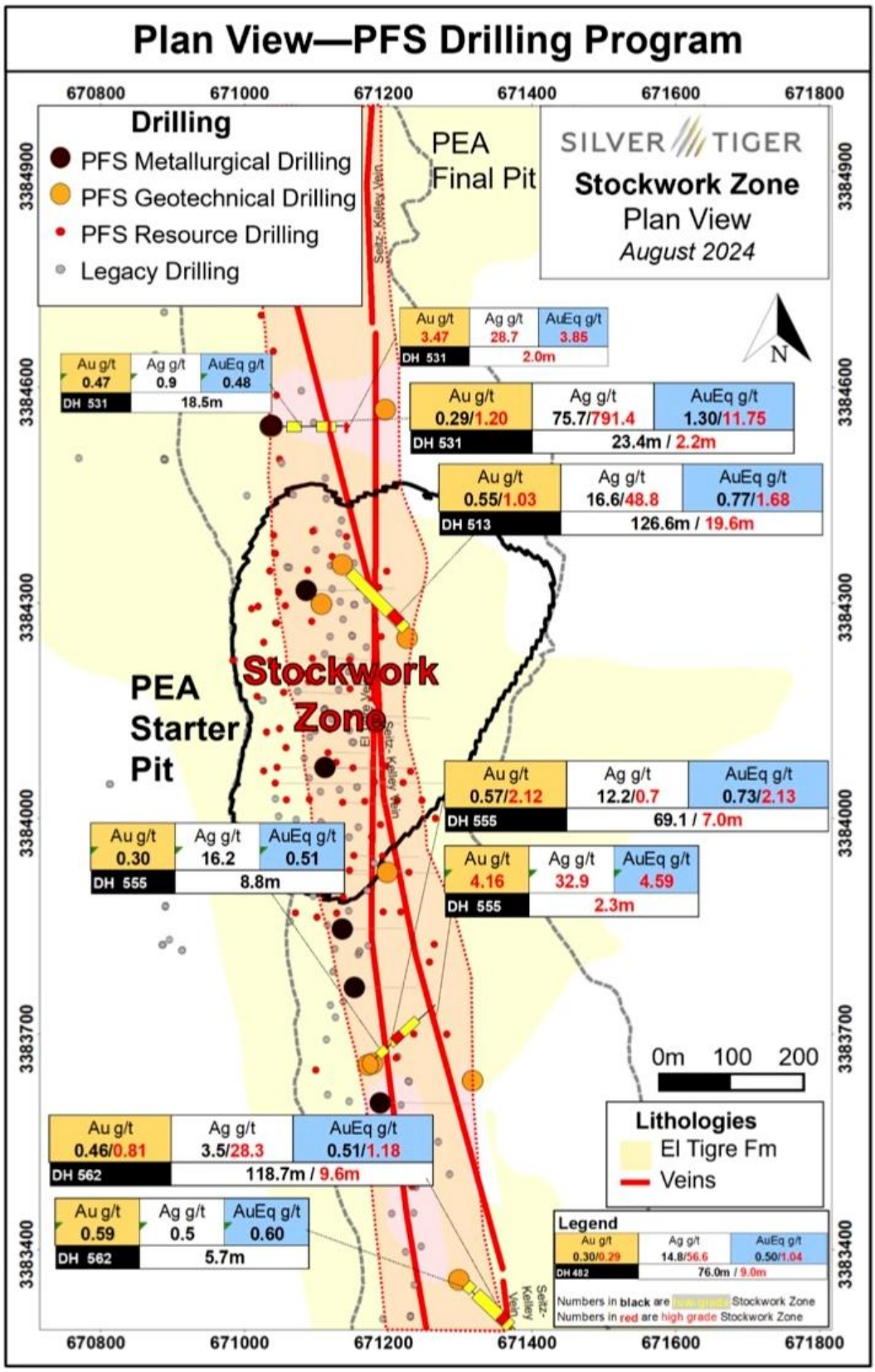
Source: *Silver Tiger website (2024)*

FIGURE 10.20 STOCKWORK ZONE CROSS-SECTIONAL PROJECTION 4200 N



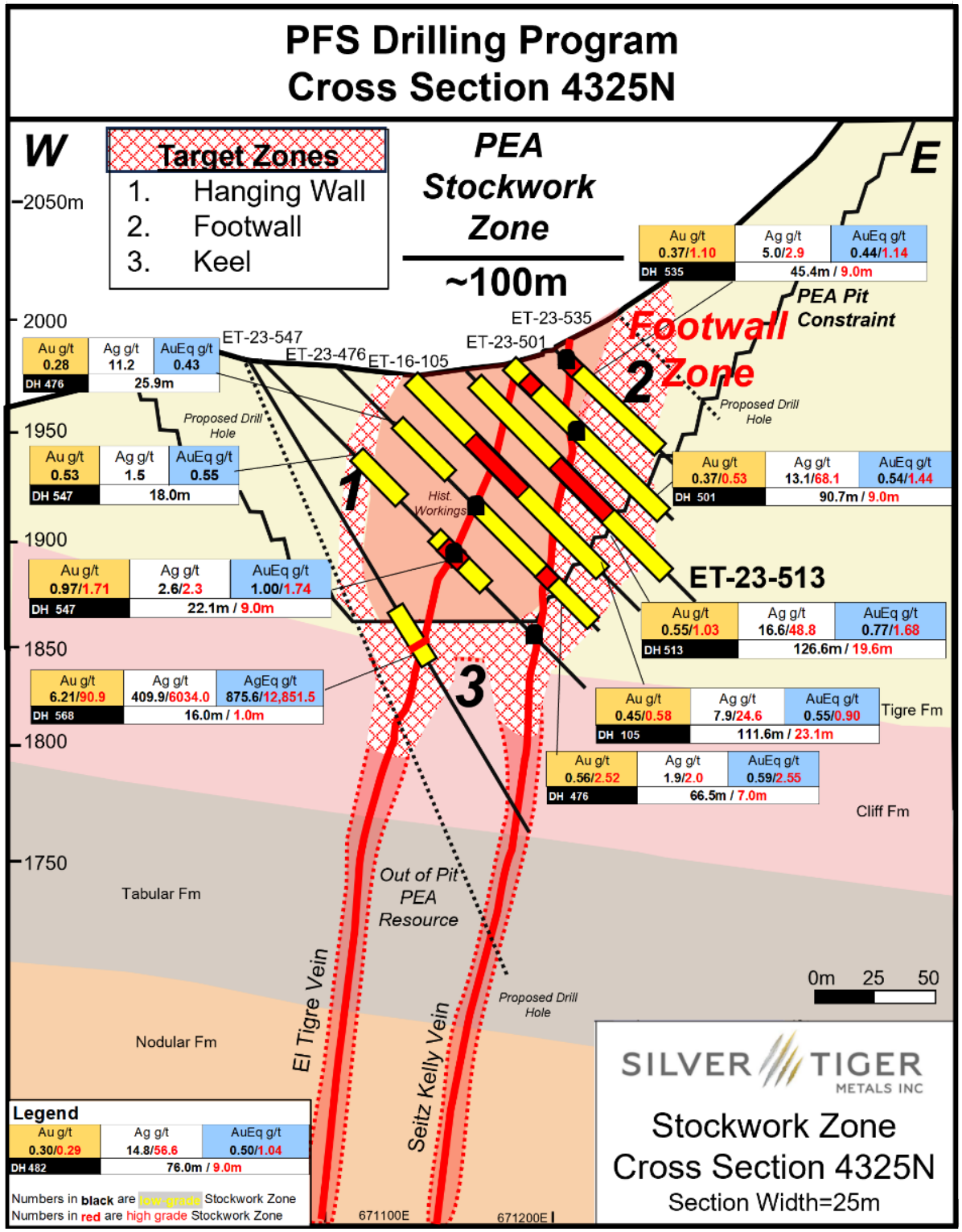
Source: *Silver Tiger website (2024)*

FIGURE 10.21 STOCKWORK ZONE PLAN VIEW



Source: Silver Tiger website (2024)

FIGURE 10.22 STOCKWORK ZONE CROSS-SECTIONAL PROJECTION 4325 N



Source: *Silver Tiger website (2024)*

In addition, drill hole ET-24-570 intersected 59.3 m grading 1.92 g/t AuEq or 144.1 g/t AgEq from 100.3 to 159.6 m, consisting of 1.63 g/t Au and 21.8 g/t Ag and 9.5 m grading 1.32 g/t AuEq or 99.2 g/t AgEq from 173.5 to 183.0 m consisting of 1.32 g/t Au and 0.5 g/t Ag. This drill hole is located ~150 m past the northern end of the 2023 PEA Starter Pit.

10.7 LATE-2024 UNDERGROUND DRILLING

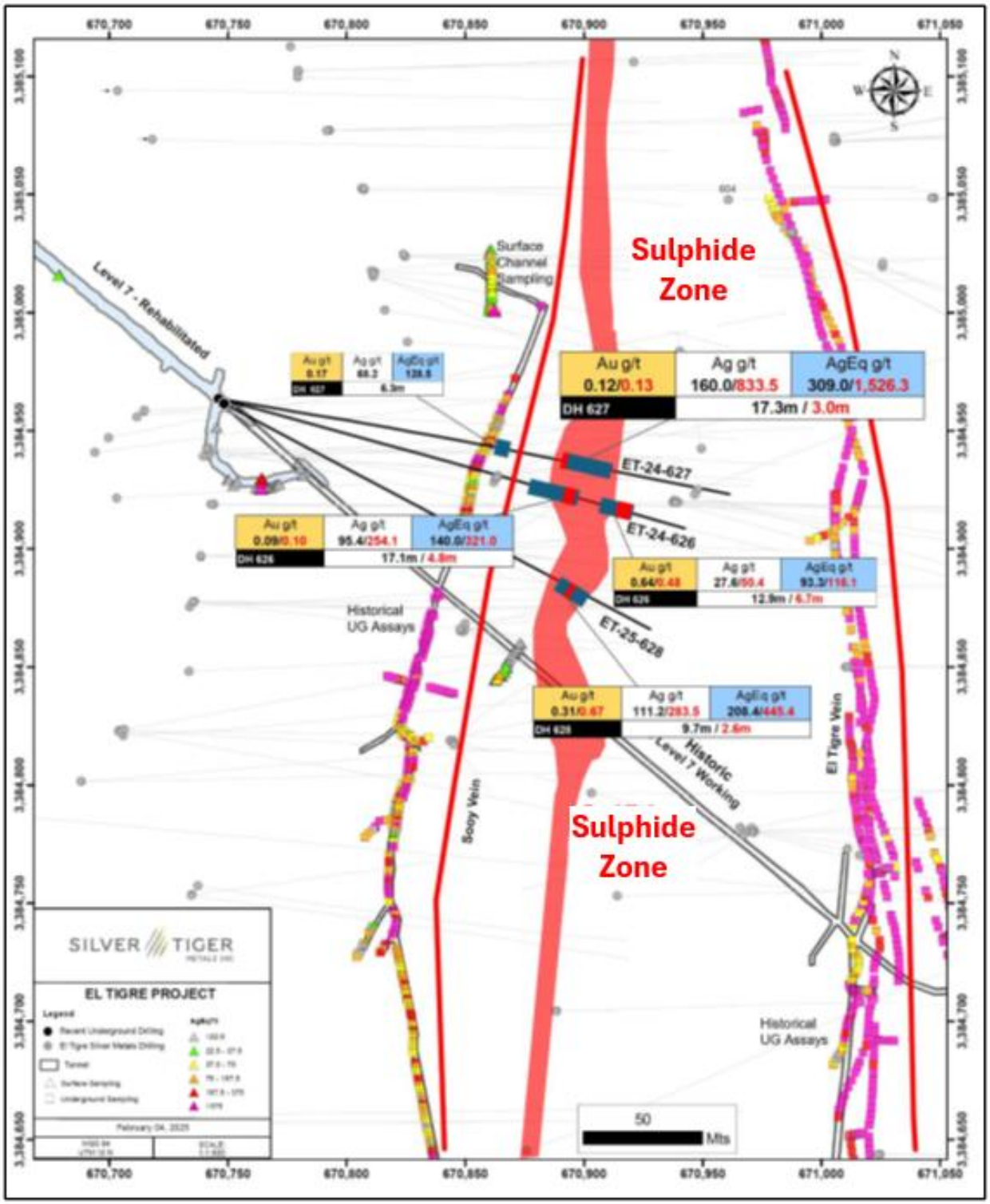
In late-2024, Silver Tiger started a drilling program from underground pads that targeted the high-grade Vein, Sulphide and Shale Zones. Highlights of two drill holes from the program are as follows:

- **Drill Hole ET-24-627**: 3.0 m grading 1,526 g/t AgEq from 297.5 m to 300.5 m, consisting of 833.5 g/t Ag, 0.13 g/t Au, 0.87% Cu, 6.22% Pb and 13.24% Zn, within 17.3 m grading 309 g/t AgEq from 297.5 m, consisting of 160.0 g/t Ag, 0.12 g/t Au, 0.18% Cu, 1.34% Pb and 2.68% Zn in the Sulphide Zone (Figures 10.23 and 10.24); and

- **Drill Hole ET-24-626**: 4.8 m grading 321.0 g/t AgEq from 253.5 m to 258.3 m, consisting of 254.1 g/t Ag, 0.10 g/t Au, 0.33% Cu, 0.33% Pb and 0.61% Zn, within 17.1 m grading 140.0 g/t AgEq from 242.0 m, consisting of 95.4 g/t Ag, 0.09 g/t Au, 0.13% Cu, 0.29% Pb and 0.54% Zn in the Sulphide Zone (Figure 10.23).

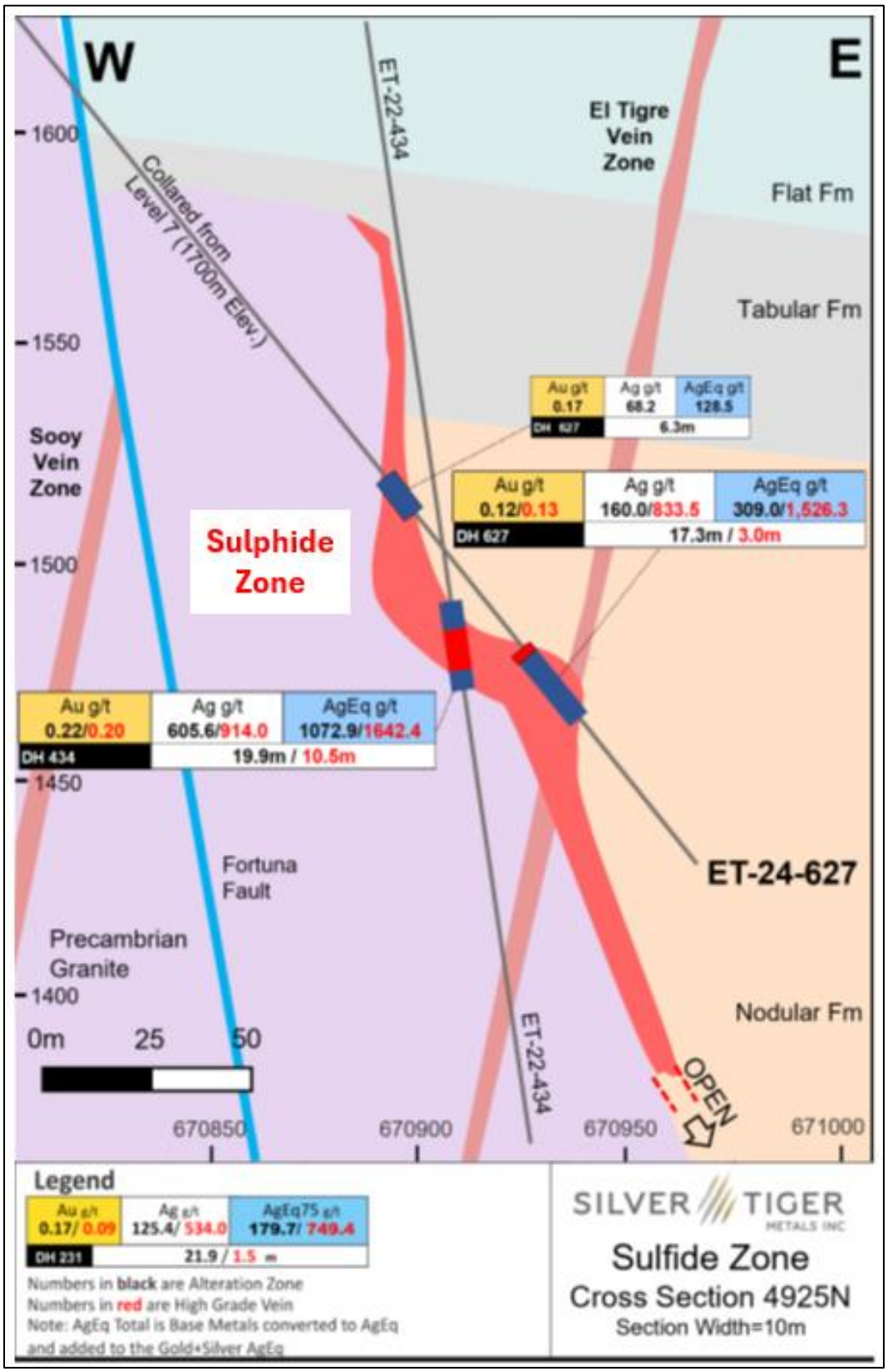
Details of three drill hole locations, orientations, lengths and significant mineralized intersections are presented in Table 10.11 and 10.12 below.

FIGURE 10.23 PLAN VIEW OF 2024 UNDERGROUND DRILLING



***Source**: Company press release dated February 11, 2025*

FIGURE 10.24 SULPHIDE ZONE CROSS-SECTION 4925 N



***Source**: Company press release dated February 11, 2025*

TABLE 10.11 2024 UNDERGROUND DRILL HOLE LOCATIONS						
Drill Hole ID	**Easting***	**Northing***	**Elevation (m)**	**Azimuth (°)**	**Dip (°)**	**Length (m)**
ET-24-626	670,749.1	3,384,961.5	1,705.5	104	-54	330.5
ET-24-627	670,749.2	3,384,962.0	1,705.4	98	-53	350.2
ET-24-628	670,748.4	3,384,961.4	1,705.3	118	-53	350.5

Source: *Company press release dated February 11, 2025*
Note: *** *coordinates are in WGS 1984 Zone 12.*

TABLE 10.12 SELECTED 2024 UNDERGROUND DRILL HOLE MINERALIZED INTERCEPTS										
Drill Hole ID	**Target**	**From (m)**	**To (m)**	**Length[1] (m)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Pb (%)**	**Zn (%)**	**AgEq Total[2] (g/t)**
ET-24-626	Sulphide Zone	242.00	259.05	17.05	0.09	95.4	0.13	0.29	0.54	140.0
ET-24-626	including	253.50	258.30	4.80	0.10	254.1	0.33	0.33	0.61	321.0
ET-24-626	and	276.50	289.36	12.86	0.64	27.6	0.06	0.15	0.24	93.3
ET-24-626	including	282.70	289.36	6.66	0.48	50.4	0.10	0.26	0.42	116.1
ET-24-626	and	323.00	327.10	4.10	0.26	14.4	0.08	0.85	1.53	112.7
ET-24-627	Sulphide Zone	245.50	251.80	6.30	0.17	68.2	0.10	0.50	0.70	128.5
ET-24-627	and	297.50	314.80	17.30	0.12	160.0	0.18	1.34	2.68	309.0
ET-24-627	including	297.50	300.50	3.00	0.13	833.5	0.87	6.22	13.24	1,526.3
ET-24-628	Sulphide Zone	287.90	297.60	9.70	0.31	111.2	0.26	0.67	1.00	208.4
ET-24-628	including	287.90	290.45	2.55	0.67	283.5	0.48	0.75	1.44	445.4

Source: *Company press release dated February 11, 2025*
Notes: *[1]Not true width.*
[2] Silver Equivalent ("AgEq") ratios are based on a silver to gold price ratio of 75:1 (Au:Ag). Copper, lead and zinc are converted using $3.66/lb copper, $0.90/lb lead, and $1.26/lb zinc at 100% metal recoveries based on a silver price of $26/oz.

10.8 DRILLING PROCEDURES

10.8.1 Drilling, Logging, Sampling

Drill core for the Silver Tiger and Oceanus campaigns were mainly HQ diameter. Certain situations required a reduction in drill core size to NQ diameter, such as drilling through fault

zones or intersecting historical mine workings. Also, the portable drill rigs struggled to drill HQ beyond 300 m and the drill core size was reduced to NQ at that depth.

All drilling activities were monitored by an on-site geologist. Drilling sites are prepared by a contractor with heavy equipment suitable for making and maintaining exploration roads. Drill hole orientation was marked out with wooden stakes for the drill crew. When the drill rig was in position, the on-site geologist verified the azimuth and inclination of the drill hole with the contractor and drilling commenced.

Drill core was retrieved as needed by Silver Tiger technicians under the direction of the geologist. The drill core was placed at the drill in wax impregnated cardboard boxes holding 2 m of drill core. An El Tigre technician transported the drill core boxes to a dedicated on-site drill core facility, where it was processed for assaying. At the drill core shed, the drill core was washed and the technician wrote the beginning and ending depths in metres of each drill core box on the front of each box with a marking pen. Next, the technician examined measures and records geotechnical information such as recovery and rock quality designation ("RQD") >10 cm. Drill core recoveries were generally ≥90%. Geologists described the drill core on paper logs with graphic and text entry methods. The paper log had sections for lithology and alteration description and for mineralization, veins, and structure. There was also an area for sample interval and number.

The geologist selected intervals for analytical sampling. Sample length varied with changes in lithology, alteration, and mineralization. The geologist marked each sample interval in the drill core box and writes the sample number at the end of each interval. A sample tag was also stapled in the drill core box at the end of each sample interval.

10.8.2 Drill Hole Collar Surveys

All drill hole collars were surveyed following completion of the drilling program. The surveys were completed by various registered surveyors using a high-quality and accurate GPS system that locates a point within a few centimetres.

10.8.3 Downhole Surveys

Every diamond drill hole was down-hole surveyed at the end of drilling each hole. The readings were taken every 50 m, beginning at the first 50 m below the collar. Each drill hole had at least two survey measurements, depending on the depth. The downhole instruments recorded azimuth and declination of the drill holes and have been used to confirm the orientation of the drill rig at the surface. This information is recorded in an Excel™ worksheet.

10.9 COMMENTS ON SECTION 10

It is the Author's opinion that Silver Tiger used industry standards in completing its drilling and logging programs.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

This section is summarized on Section 11 of P&E (2023) and has been updated where applicable.

The following section of this Technical Report reviews the recent sample preparation, analyses, and security measures undertaken during drilling and channel sampling at the El Tigre Property by Silver Tiger between 2016 and January 2025, and drilling undertaken by El Tigre Silver between 2010 and 2013. Information relating to the historical drilling carried out by Anaconda Minerals on the Property between 1982 and 1983 (forming almost 4% of the current Mineral Resource Estimate drill hole assay data.) is limited, with laboratory assay certificates unavailable, and sparse information on sampling, security and quality control measures taken.

The Author has partially relied on publicly available information from previous Property reports and internal documentation supplied by Silver Tiger to assess the reliability of Anaconda's historical data, and on verification sampling undertaken by the site visit Qualified Person in 2017 (as described in Section 12 of this Technical Report).

11.1 HISTORICAL SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1.1 1982 to 1983 Anaconda Sampling

The Mexican subsidiary of Anaconda Minerals, Cobre de Hercules S.A., completed 22 diamond drill holes at the Property between 1982 to 1983, totalling 7,813 m. The drilling was undertaken by Boyles Brothers Drilling Co. using Longyear 44 core drills (Wood, 2009). Drill holes were collared with NC-size drill core. All drill holes were surveyed with a down-the-hole Sperry-Sun instrument to determine the exact location of vein intercepts and other geologic features at depth. All drill holes were inclined between -40° and -61° and drill hole lengths varied from 140 to 650 m. After the drill core had been retrieved from the drill hole, it was logged and selected intervals were split, with ½ sent for assay and the remaining ½ drill core returned to the drill core tray for future reference.

Sample preparation and analyses were carried out at the Anaconda Geoanalytical Laboratory in Tucson, Arizona. Samples were oven-dried at 100°C if required, jaw-crushed to -¼ inch, roller-crushed to -10 mesh, and then homogenized before taking a 100 g split and pulverizing to -80 mesh or -200 mesh. After preparation, pulps were assayed for Ag, Au, Cu, Pb, and Zn by Atomic Absorption or Inductively Coupled Plasma Emission Spectrometer. Pulps returning values >2 ppm Au and (or) 50 ppm Ag were re-analyzed by fire assay.

Information relating to quality control procedures used by Anaconda during the 1982-1983 drilling have not been found and there remains sparse documentation regarding the drilling, sampling and assaying procedures employed, and no original assay documentation. Drill core from Anaconda's drilling is reported to have been stored in an historical mine building at the 700 Level portal of the El Tigre Mine and the majority of drill core boxes were scattered and destroyed by vandals. Nevertheless, a total of 698 m of the Anaconda drill core (~9% of the total drilled) from 18 drill holes were rescued intact in the original drill core trays from the collapsed historical drill core storage shed at El Tigre in 2011 by the El Tigre Silver Corp geologists.

11.1.2 2010 to 2013 El Tigre Silver Corp Sampling

11.1.2.1 2010 to 2013 Sampling and Security

The following information relating to sample preparation, analyses and security undertaken by El Tigre Silver Corp., has largely been taken from the 2009 (Wood, 2009), 2011(Gibson, 2011), and 2013 (Black and Choquette, 2013) Reports on the El Tigre Property.

The Author has not reviewed information relating to quality assurance/quality control ("QA/QC" or "QC") prior to 2010, due to a lack of information relating to protocol, sampling methods and analytical procedures. Additionally, the Author has not reviewed the raw data relating to QA/QC for the 2010 through 2013 exploration work and has relied on the aforenoted 2009, 2011 and 2013 reports. Information relating to sampling procedures, analyses, and security for El Tigre Silver' 2010 to 2013 exploration programs is discussed herein.

El Tigre Silver completed extensive geochemical rock chip sampling and three core-drilling campaigns from 2010 to 2013. Two additional drilling campaigns were also undertaken at the historical tailings impoundment in 2011 and 2013.

Drill core was collected from the drill rig site by authorized company personnel and taken to the on-site, fenced drill core storage facility behind the main office/camp building. Drill core samples were marked with a unique sample number during collection and logging and the marked intervals cut with a diamond drill core saw. Half of the drill core sample was placed in a plastic sample bag, then tagged (with its unique sample number), labelled and sealed. The remaining half of the drill core was returned to the drill core boxes and stored in the steel drill core shed on-site.

Sample shipments consisted of 60 to 200 samples and details of all samples were documented on a lab submittal sheet, which was included with the sample shipment. The bagged samples were then taken by truck to Hermosillo to either the Pacemaker (a subsidiary of El Tigre Silver) office storage area in Hermosillo for temporary storage or taken directly to the lab. Samples were under the control of authorized El Tigre Silver personnel at all times, or securely stored at the on-site, fenced drill core storage facility behind the main office/camp building, from the time of collection at the drill rig or field site, until received by the lab.

Samples were analyzed at three different laboratories throughout 2010 to 2013. El Tigre Silver used Skyline Assayers and Laboratories Inc., of Tucson, Arizona ("Skyline") in the first half of 2010, ALS Minerals ("ALS") of Vancouver, British Columbia from mid-2010 through 2011, Skyline again in 2012, and Inspectorate of Reno, Nevada.

When required, ALS personnel transported the samples from El Tigre Silver Hermosillo office to the ALS sample preparation facility in Hermosillo. The prepared sample pulps were subsequently sent for gold and silver analysis to the ALS assaying facility in either Canada or the United States.

11.1.2.2 2010 to 2013 Sample Preparation and Analysis

El Tigre Silver used three different assay labs: Skyline in 2010 and 2012; ALS Global in 2010 and 2011; and Inspectorate in 2012. Sample preparation by each of these labs is generally similar.

When a sample arrives at the assay laboratory, it is given a unique bar code for tracking, weighed and dried. Samples were then prepared for assaying by crushing with a jaw crusher to >80% minus 10 mesh, split; and pulverized to >90% minus 150 mesh. This pulped material is then bagged, and the samples are assayed from a 250 g sample split. Gold is assayed by 30 g fire assay followed by an atomic absorption ("AA") analysis. Values over 10 ppm gold are fire assayed with gravimetric finish. For silver and other trace elements, the pulp is digested either in an aqua regia solution or a four-acid total digestion and leach followed by either an AA scan for silver and (or) an ICP-AES ICP Scan for 33 elements. Silver over-limits are fire assayed. Pulps and rejects are returned to the El Tigre Silver office and stored.

Skyline was accredited in accordance with the recognized International Standard ISO/IEC 17025:2017. This accreditation demonstrates technical competence for a defined scope and the operation of a laboratory quality management system (refer to joint ISO-ILAC-IAF Communique dated April 2017, available at https://ilac.org/about-ilac/partnerships/international-partners/iso/).

ALS has developed and implemented strategically designed processes and a global quality management system at each of its locations. The global quality program includes internal and external inter-laboratory test programs and regularly scheduled internal audits that meet all requirements of ISO/IEC 17025:2017 and ISO 9001:2015. All ALS geochemical hub laboratories are accredited to ISO/IEC 17025:2017 for specific analytical procedures.

Inspectorate (rebranded as Bureau Veritas on October 1, 2018) is a leading provider of laboratory testing, inspection, and certification, operating in more than 1,600 offices and laboratories in 140 countries. Bureau Veritas is ISO 9001 compliant and for selected methods, ISO 17025 compliant and has an extensive QA/QC program to ensure that clients receive consistently high-quality data.

11.1.2.3 2010 to 2013 Quality Assurance / Quality Control

El Tigre Silver's exploration program in 2010 included transporting 300 rock chip samples to Skyline laboratory, in the first half of the year. El Tigre Silver relied on Skyline laboratory's own internal QA/QC protocol. In the second half of 2010, El Tigre Silver submitted additional rock chip samples for analysis and again relied on the laboratory's own internal QA/QC protocol.

In 2011, El Tigre Silver completed its first drilling program, which consisted of 10 drill holes. El Tigre Silver began inserting a limited number of certified reference materials ("CRM"), duplicates and blanks into the sample stream for analysis. In total, 20 blanks, 11 duplicates and three CRMs were submitted with the drill samples. ALS completed all analyses. Performance of the QC samples was largely acceptable.

The 2012 drill program consisted of 11 drill holes and the QA/QC program was greatly improved over earlier efforts. A total of 18 blanks, 23 duplicates and seven CRMs were submitted with the drill samples. Analysis was carried out by Inspectorate, with sample preparation completed in Hermosillo. Again, for the most part, these samples passed quality control assessment.

The 2013 drill program consisted of 38 drill holes and the volume of QA/QC samples tripled compared to the previous drilling programs. The number of CRMs and blanks increased for each

submittal of drill samples. Field duplicate sampling ceased soon after the drill program commenced and by the end of the sampling program, 65 blanks, four duplicates and 65 CRMs were submitted with the drill samples to Skyline for analysis. Performance of the QC samples was largely acceptable.

The Author did not review the original QA/QC data, however, did assess the QC performance charts available in the historical Preliminary Feasibility Study for the El Tigre Property (Zachary *et al.*, 2013). Commercially available CRMs were utilized during the 2010 to 2013 sampling at the Project (summary of CRMs used listed in Table 11.1). El Tigre Silver used one main CRM for the drilling programs and submitted 82 CRM samples for analysis at Skyline, ALS or Inspectorate. The CDN Resource Laboratories Ltd. ("CDN") CRM used was CDN-ME-12, which is certified for use with the following techniques:

- FA using a 30 g charge and AAS finish for gold; and
- AAS with a four-acid digestion for silver.

TABLE 11.1
SUMMARY OF CERTIFIED REFERENCE MATERIAL USED BY EL TIGRE SILVER AT EL TIGRE

CRM	Element (ppm)	Mean	Size	Std Dev
CDN-ME-12	Au_Sel	0.348	13	0.04
CDN-ME-12	Ag_Sel	52.5	13	4.3
MEG-Au.12.25	Au_Sel	0.71	47	0.059
MEG-Au.12.25	Ag_Sel	4.442	40	0.9
OxG83	Au_Sel	1.002	37	0.027

Source: *Zachary et al. (2013)*
Note: *Std Dev = standard deviation.*

El Tigre Silver used one main CRM for the drilling programs and submitted 82 CRM samples for analysis at Skyline, ALS, or Inspectorate. The individual CRMs were plotted against ±2 and ±3 standard deviations of the expected CRM mean. The CRM CDN-ME-12 performed as expected for silver with most data plotting within ±2 standard deviations of the mean, and two failures only that fall just below -3 standard deviations. Gold data for this CRM are more scattered and with a slight high bias. A total of 14 samples fails for gold, giving a failure rate of 19% for gold.

The silver blanks performed extremely well, with all except one sample reporting below 2.5 g/t Ag. The single failure in sample ET-10595 of drill hole ET-27 was from the early 2011 drilling program and is assumed to represent a sample mix up by El Tigre Silver. The gold blanks performed reasonably well, with the majority of results reporting below a 0.025 g/t Au. However, there is a large sample population in the Skyline analysis results that are reporting at or above the 0.025 g/t Au limit. This could be a result of the assaying method, a systematic failure, or a problem with the blank being submitted. Each of the failures is similar in value and probably represents a problem with the blank material.

El Tigre Silver undertook field duplicate sampling during all drilling programs. In general, silver duplicates performed well with most of the data falling close to the 1:1 line when scatter graphed. There was noticeably more deviation in the gold duplicate pairs.

El Tigre Silver selected a total of 40 of the Inspectorate prepared coarse reject drill samples from the 2012 drilling program in late 2012 for umpire assaying at a secondary lab, specifically ALS in Hermisillo for pulp preparation and ALS in Vancouver for assaying. Control samples (four CRMs and four duplicates) were included with the umpire assays to monitor performance at ALS and returned values within normal limits. Significant deviation was shown in the paired silver data, however, the labs utilized different analytical methods. The gold pairs performed within the expected ±30% for a coarse duplicate analysis.

In late 2012, mineralized drill hole pulps from eight separate assay certificates were selected from the 2011 drilling program originally analyzed by ALS. These were submitted to Inspectorate for re-analysis to check the ALS assay results for gold and silver. As a result of this study, the use of a contaminated container in four-acid digestion analysis for silver was discovered, and based on the severity of the issue, El Tigre Silver made the decision to no longer use Inspectorate.

El Tigre Silver also submitted 83 of the ALS-prepared pulp samples from the 2011 drilling program to Skyline for umpire assaying. Both laboratories reported results for both gold and silver within the expected ±15% for a pulp duplicate analysis, with only two deviations from the ±15% noted in the gold assays.

11.2 BULK DENSITY

Bulk density measurements were determined by Silver Tiger using the water immersion method on diamond drill core samples. Company authorized geo-technicians select an intact cylinder of drill core 10 to 20 cm in length, record the weight, coat the sample in paraffin wax, and dry and record the weight with wax, submerge the sample in the graduated cylinder filled with water, record the change in volume in water, and divide the weight with paraffin by the volume displaced to determine the bulk density. Measurements for each box of drill core are made from the top to the bottom of the drill hole, thereby providing excellent representative coverage through the hanging wall units to the footwall units. The average bulk density is 2.56 t/m^3 and 2.72 t/m^3 for the mineralized veins only. The Tailings average dry bulk density of 1.39 t/m^3 was derived from 37 nuclear density measurements. For the Low-Grade Stockpiles a bulk density of 1.60 t/m^3, the bulk density of dry sand, was assigned.

Independent verification sampling carried out between 2017 and 2025 has confirmed Silver Tiger's on-site bulk density measurements. A total of 88 due diligence samples were measured independently at either ALS or Actlabs, returning a mean bulk density value of 2.61 t/m^3, median value of 2.58 t/m^3, minimum value of 2.35 t/m^3, and a maximum value of 3.79 t/m^3.

11.3 2016 TO JANUARY 2025 SILVER TIGER SAMPLING

11.3.1 Sample Preparation and Security

11.3.1.1 Channel Sampling

Silver Tiger has carried out a general surface and underground sampling program on the El Tigre Property. Sampling included chip channel samples and grab samples following a protocol of sampling procedures that included:

- Channel sampling controls including keeping records of the sample type, size, number and location using GPS;
- The sample locations were photographed;
- One every 40 sample were duplicated and sent for analysis;
- Every 40 samples one blank sample was inserted; and
- Every 40 samples one control sample CRM was inserted.

Identical procedures were used for sampling in the mine workings. Samples were taken by local crews under the supervision of a geologist from SPM. Chip samples were cut with a hammer and chisel, collected on a tarp and placed in a plastic bag to be labelled and sent to the laboratory for precious metal assay and ICP multi-element analysis.

11.3.1.2 Drill Core Sampling

The protocol for handling, sampling and assaying diamond drill core samples was developed in 2011 by David Duncan, P.Geo., for Silver Tiger's San Diego project (in Durango State) during 2012 to 2014 and Silver Tiger's Santa Gertrudis program in 2015 to 2017. These same protocols are used for the El Tigre drilling program and are summarized as follows:

- The drill core was placed in labelled drill core boxes by the drilling contractor with metreage blocks inserted in the trays at the end of each run. The lids were placed on and subsequently fastened to the drill core boxes;

- Silver Tiger geologists and geo-technicians were present at the drill rig to ensure that drill core handling, drill core accommodation, box number and depth recording was done properly by the drilling contractor;

- The drill core was transferred from the drill rig to the Company's core logging, sampling and storage facilities, where the trays were placed in order on the logging tables and the first inspection is made prior to cleaning and washing the drill core of any drilling muds;

- All depth marker tags were checked for completeness and accuracy with special attention paid to possible intersection of mining voids;

- The SPM geo-technicians aligned the drill core pieces, assess and measure drill core recoveries and RQD, and photographed the drill core;

- Bulk density measurements were reported for all diamond drill holes by Silver Tiger geo-technicians using the water immersion method;

- The SPM geologists logged the drill and lay out the areas to be sampled by the geo-technicians;

- Boxes of drill core were transferred to the sampling room, where the drill core was sawn in half by a diamond saw;

- The half drill core samples were placed in plastic bags along with a sample tag ID and tied closed with zip locks under the supervision of the SPM geologists. Sample tags had three portions; one for the drill core tray, the sample bag and one left in the sample book;

- Up to 10 sample bags were placed in larger rice bags, which were tied closed with zip ties and labelled;

- The remainder of the drill core sample was returned to the drill core box, the lids replaced, and the boxes were transferred to drill core racks at the Company's secure drill core storage facility;

- All samples were collected by SPM personnel and delivered to the Actlabs laboratory in Zacatecas (2016-2017), ALS in Hermisillo (2022 tailings auger samples and 2023-2024 umpire samples), or Bureau Veritas in Hermisillo all 2020 to 2025 samples, excluding 2022 auger and 2023-2024 umpire samples. The drill core and samples were under Silver Tiger's or SPM's supervision, from the time of pick-up of the drill core at the drill site until they were delivered to laboratory staff. All drill core and sample splits are kept in a secure drill core storage facility. SPM use their own vehicles to transport the samples to the lab and the samples are generally received by the lab within two days; and

- Assay data was reported electronically from Actlabs, ALS and Bureau Veritas to Silver Tiger and SPM.

11.3.1.3 Sample Analysis

When a sample arrives at the assay laboratory, it is given a unique bar code for tracking, weighed and dried. Representative 200 to 300 g pulp samples are then prepared for assaying by crushing the whole sample with a jaw crusher to >70% minus 10 mesh (2 mm), mechanically splitting (riffle) to obtain a representative sample, and then pulverizing to >90% minus 140 mesh (105 µm) at ActLabs, >85% minus 200 mesh (75 µm) at ALS and >90% minus 200 mesh (75 µm) at Bureau Veritas.

Samples at ActLabs (2016/2017) were analyzed for gold and silver, and an array of other elements. Gold analysis was carried out by fire assay with atomic absorption spectroscopy ("AAS") finish. Reporting limits for this test method were 0.005 to 10 g/t Au. Results exceeding 10 g/t Au were

reanalyzed using fire assay with a gravimetric finish and reported in g/t Au. Silver analysis was carried out by total digestion with ICP finish. Reporting limits for this test method were 0.3 to 100 g/t Ag. Results exceeding 100 ppm Ag were re-analyzed using fire assay with a gravimetric finish, and reported in g/t Ag. The Actlabs' Quality System is accredited to international quality standards through ISO/IEC 17025:2017 and ISO 9001:2015. The accreditation program includes ongoing audits, which verify the QA system and all applicable registered test methods. Actlabs is also accredited by Health Canada. Actlabs is independent of Silver Tiger and SPM.

Samples at ALS (2022 tailings auger samples and 2023-2024 umpire samples) were analyzed for gold, silver, copper, lead and zinc, and an array of other elements. The pulps are assayed for gold using a 30 g charge by fire assay with AAS finish (Code AA23) and reporting limits of 0.005 to 10 g/t. Over limits >10 g/t are re-assayed using a gravimetric finish (Code ME-GRAV21). Silver, copper, lead, zinc, and multi-element analysis is completed using total digestion and ICP on a 0.25 g sample (Code ME-ICP61). Reporting limits for silver were 0.5 to 100 g/t Ag, copper was 1 ppm to 10,000 ppm Cu, and lead and zinc were 2 to 10,000 ppm Pb and Zn. Silver assay over-limits >100 g/t are re-assayed by "ore" grade four-acid with ICP-AES or AAS finish on a 0.4-g sample (Ag-AA62 or ME-ICP61) for the 2022 tailings auger samples and 2023-2024 umpire samples, or by fire assay with gravimetric finish for the 2023-2024 umpire assaying program. ALS is independent of Silver Tiger and SPM and has developed and implemented strategically designed processes and a global quality management system at each of its locations. The global quality program includes internal and external inter-laboratory test programs and regularly scheduled internal audits that meet all requirements of ISO/IEC 17025:2017 and ISO 9001:2015. All ALS geochemical hub laboratories are accredited to ISO/IEC 17025:2017 for specific analytical procedures.

Samples at Bureau Veritas (all 2020 to 2025 samples, excluding 2022 auger and 2023-2024 umpire samples) were analyzed for gold, silver, copper, lead and zinc, and an array of other elements. The pulps are assayed for gold using a 30 g charge by fire assay with AA finish (Code FA630 or FA430) and reporting limits of 0.005 to 10 g/t Au. Over-limits >10 g/t Au are re-assayed using a gravimetric finish (Code FA530). Silver, copper, lead, zinc, and multi-element analysis is completed using total digestion and ICP-MS on a 0.25-g sample (Code MA200). Reporting limits for silver were 0.1 to 200 ppm Ag, copper and lead were 0.1 ppm to 10,000 ppm Cu and Zn, and zinc 1 to 10,000 ppm. Silver assay over-limits >200 g/t are re-assayed using a gravimetric finish (Code FA530). Copper, lead, and zinc over-limits are re-assayed using total digestion and ICP-ES/MS on a >1 g sample (Code MA270). Lead assays returning >100,000 ppm was also assayed by "ore" grade lead titration on a >5 g sample (Code GC817). Bureau Veritas is independent of Silver Tiger and SPM and is ISO 9001 compliant and, for selected methods, ISO 17025 compliant and has an extensive QC program to ensure that clients receive consistently high-quality data.

Silver Tiger and SPM are also independent of the laboratories (Anaconda Geoanalytical Laboratory, Skyline and Inspectorate) utilized in earlier sampling campaigns at the Property prior to the Company's involvement.

11.3.2 2016 to 2017 Quality Assurance / Quality Control (Actlabs)

QA/QC procedures monitor the chain-of-custody of the samples and include the systematic insertion and monitoring of appropriate reference materials (CRMs, blanks and duplicates) into

the sample stream. The results of the assaying of the QC material included in each batch are tracked to ensure the integrity of the assay data.

A total of 3,129 samples were sent for analysis during the 2016 and 2017 drill program, 2,672 of which were drill core samples. A total of 253 CRMs, 126 blanks and 78 field duplicates (¼ drill core duplicate) were inserted routinely into the samples stream.

11.3.2.1 Performance of Certified Reference Materials

Four different CRMs were used during the 2016 and 2017 drill program to monitor gold accuracy: OxG83, OXD108 and CDN-GS-P7E and SQ47. The Author has also reviewed Actlab's internal QA/QC data to determine the quality of silver assay results throughout the drill program.

Criteria for assessing CRM performance are based as follows. Data falling within ±3 standard deviations from the accepted mean value pass. Data falling outside ±3 standard deviations from the accepted mean value fail.

A summary of results for the gold CRMs are presented in Table 11.2**Error! Reference source not found.**.

TABLE 11.2
SUMMARY OF CERTIFIED REFERENCE MATERIALS USED AT EL TIGRE

Reference Material	Certified Mean Value (ppm)	±1 SD (ppm)	±2 SD (ppm)	ActLabs Results			
				No. Results	No. (-3 SD) Failures	No. (+3 SD) Failures	Average Result (ppm)
OxG83	1.002	0.027	0.054	125	12	0	0.943
OxD108	0.414	0.012	0.024	20	0	0	0.412
CDN-GS-P7E	0.766	0.043	0.086	107	2	0	0.768
SQ47	39.88	0.850	1.70	4	0	0	39.49

Source: *P&E (2017)*
Note: *SD = standard deviation.*

There were no failures for either the OxD108 or SQ47 gold CRMs.

There were total of 12 failures for the OxG83 gold CRM, and a low bias was noted for this CRM. All 12 failures plotted below minus three times the standard deviation from the mean and the Author does not consider these to be of material impact to the current Mineral Resource Estimate.

There were two low failures recorded for the CDN-GS-P7E CRM. However, these values correspond with the OxD108 CRM and are likely to be misallocations. Both results pass for the OxD108 CRM and no further action is required.

The Author reviewed the performance of Actlabs' internal silver CRMs for the 2016 to 2017 drill program. Of the 368 CRMs inserted by the lab, there were three high failures and 20 low failures. The majority of the low failures were for the GXR-6 CRM, likely due to the mean CRM value

being too close to the lower detection limit, and the Author does not consider these to have a significant impact on the current Mineral Resource Estimate. All failures, except one low failure in batch Z16-293, had multiple other CRMs in the same batch and no further action was considered necessary. The single CRM in batch Z16-293 failed low and was also not considered to be of material impact to the current Mineral Resource Estimate. A summary of Actlabs' internal silver CRM performance is presented in Table 11.3.

TABLE 11.3
SUMMARY OF ACTLABS' INTERNAL CERTIFIED REFERENCE MATERIALS USED AT EL TIGRE

Certified Reference Material	Certified Mean Value (ppm)	±1 SD (ppm)	±2 SD (ppm)	ActLabs Results			
				No. Results	No. (-3 SD) Failures	No. (+3 SD) Failures	Average Result (ppm)
CDN-GS-P5D	66	2.85	5.7	3	0	0	64.7
CDN-ME-1201	37.6	1.7	3.4	1	0	0	38.3
CDN-ME-1301	26.1	1.1	2.2	3	2	0	18.4
CDN-ME-1305	231	6	12	41	0	0	231.6
CDN-ME-1306	104	3.5	7	19	0	0	101.4
CDN-ME-1408	396	6.5	13	15	0	0	392.3
CDN-ME-16	30.8	1.1	2.2	1	0	1	48.9
CDN-ME-1602	137	3	6	14	0	0	136.4
CDN-ME-19	103	3.5	7	14	0	0	102.0
GXR-1	31	1.2*	2.4	91	0	1	31.2
GXR-4	4	0.4*	0.7	64	2	0	3.4
GXR-6	1.3	0.4	0.7	95	16	1	0.5
PM1145	811	36.5	73.0	1	0	0	792.2
PM1146	1,586	68.98	137.96	6	0	0	1,578.0

Source: *P&E (2017)*
Notes: ** SD = Standard deviation has been calculated from Actlabs data.*

The Author is of the opinion that CRM performance throughout the 2016/2017 drill program demonstrates reasonable accuracy.

11.3.2.2 Performance of Blank Material

Blank material was used by the Company to monitor for both gold and silver contamination. All blank data for gold were assessed by the Author. If the assayed value in the certificate was indicated as being less than detection limit, the value was assigned the value of half the detection limit for data treatment purposes. An upper tolerance limit of three times the detection limit was set. There were total of 126 data points to examine.

The majority of the data plots below the set tolerance limit, with only three points falling above. Two of the data points that returned results of 0.021 and 0.190 ppm plot just outside of the set tolerance limit and the Author does not consider these to be significant to the integrity of the data.

A third data point (sample ETC-640 returning a value of 0.400 ppm) plots high above the upper tolerance limit. On review of this certificate, the high blank result appears to be carry-over contamination from preceding high-grade samples and the Author considers this to be within reasonable limits of carry-over contamination and of no impact to the current Mineral Resource Estimate. One other blank sample, and eight internal laboratory blanks return values around the lower detection limit within this batch of samples and no further action is deemed necessary.

Blank data for silver were also assessed by the Author. If the assayed value in the certificate was indicated as being less than detection limit the value was assigned the value of half the detection limit for data treatment purposes. An upper tolerance limit of 0.9 ppm (three times the detection limit) was set. There was a total of 117 data points to examine.

The majority of the data plotted below the set tolerance limit, with only eight points plot above. Seven of the data points returned results plotting just outside of the set tolerance limit and the Author does not consider these to be significant to the integrity of the data. An eighth data point (sample ETC-3920 returning a value of 16.4 ppm) plots higher above the upper tolerance limit. On review of this certificate, the high blank result follows several high-grade samples and appears to be carry-over contamination that the Author considers within reasonable limits and of no impact to the current Mineral Resource Estimate. Another two blank samples, and 10 internal laboratory blanks return values around the lower detection limit within this batch of samples and no further action is deemed necessary.

The Author does not consider contamination to be a material issue for the 2016 to 2017 drill data.

11.3.2.3 Performance of Field Duplicates

Field duplicate data from 2016 and 2017 were examined for both gold and silver. There were totals of 78 duplicate pairs for Au and 67 for Ag in the data set. Data were scatter graphed and the Coefficient of Determination (R^2) for Au and Ag found to be 0.6152 and 0.8788, respectively, and 0.8482 for Au data excluding seven of the largest outliers. Aside from a small number of outliers, both data sets were found to have reasonable precision at the field level for this type of mineralization.

11.3.2.4 Performance of Laboratory Duplicate Samples

The Author reviewed Actlabs' internal prep duplicate (coarse or crusher duplicate) and pulp duplicate samples for the two different types of gold and silver analyses performed. Actlabs inserted duplicate samples into the sample stream throughout the QC program at El Tigre to monitor precision for gold and silver.

Duplicate data reviewed for gold included 230 prep duplicate samples analyzed by FA-AAS method and one by FA-GRAV method, and 412 pulp duplicate samples analyzed by FA-AAS method and six by FA-GRAV method.

Duplicate data reviewed for silver included 232 prep duplicate samples analyzed by TD-ICP method and one by FA-GRAV method, and 598 pulp duplicate samples analyzed by TD-ICP method and nine by FA-GRAV method.

Original versus duplicate sample data were graphed on scatter plots and all data plot closely to the 1:1 parity line. The R^2 values for Au were estimated at 0.9987, 0.9954 and 0.9973 for the FA-AAS preparation duplicates, FA-AAS pulp duplicates and FA-GRAV pulp duplicates, respectively. The R^2 values for Ag were estimated at 0.995, 0.9913 and 1 for the TD-ICP prep duplicates, TD-ICP pulp duplicates and FA-GRAV pulp duplicates, respectively.

The Author considers that precision is of an acceptable level for both Au and Ag.

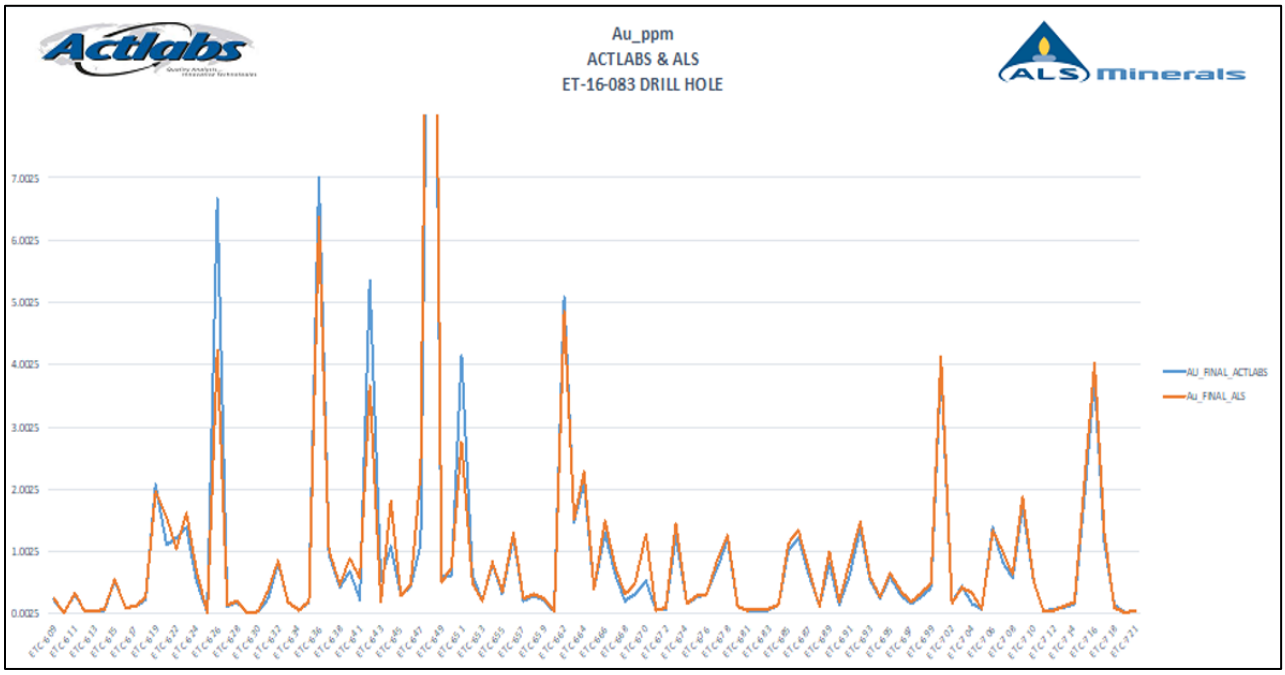
11.3.2.5 Silver Tiger 2017 Check Assaying Actlabs Versus ALS

Silver Tiger undertook check assaying at ALS in Hermosillo, Mexico in 2017 and selected 201 representative pulp samples over varying grades from three El Tigre drill holes. Samples were selected from drill holes ET-16-083, ET-16-101, and ET-17-133 and analyzed for gold by fire assay method with an AA finish and silver by fire assay with ICP or gravimetric finish.

Results from ALS were compared to the original Actlabs results to confirm the original reported grades. Silver Tiger plotted results for the three drill holes on individual line graphs for both gold and silver (Figures 11.1 to 11.6) and comparison between the original and check assays is excellent for all drill holes and both elements.

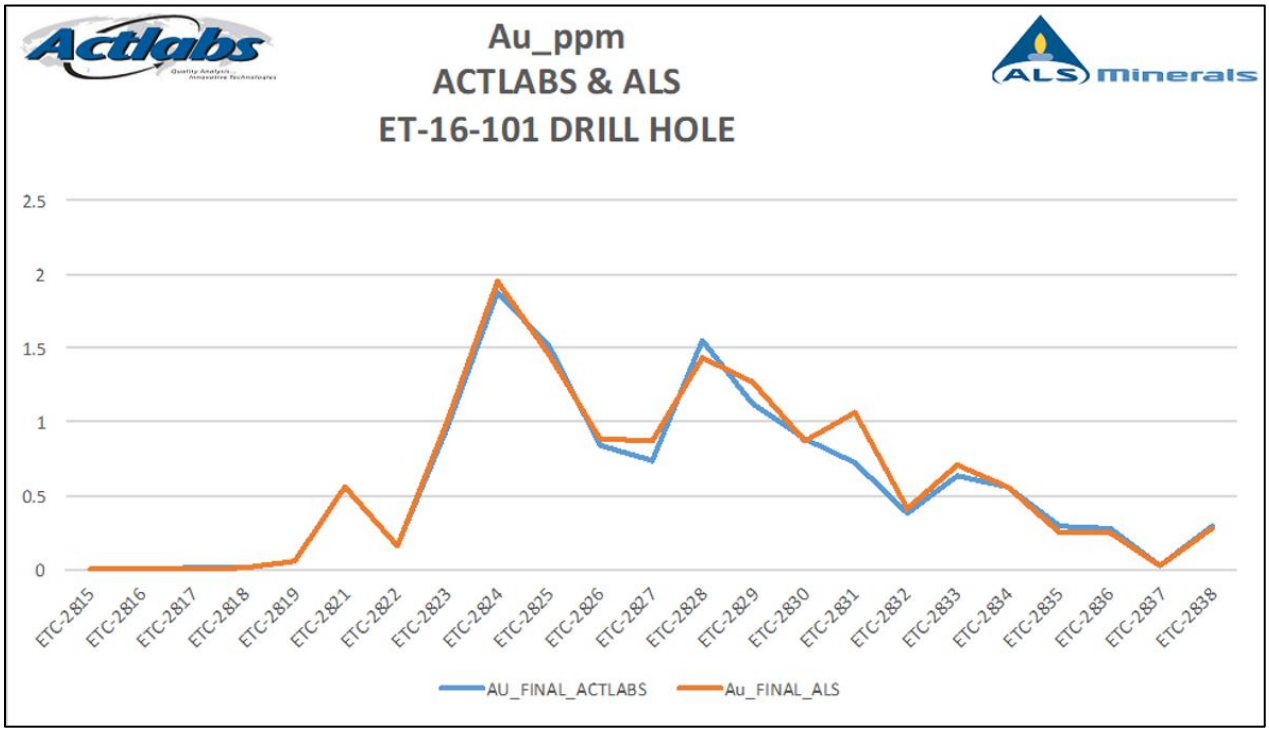
Some of the charts have been intentionally truncated along their vertical axis, due to extremely high-grade results that make interpretation of the majority of data difficult if shown. In the Author's opinion, comparison of all duplicate samples not displayed on the charts is also acceptable.

FIGURE 11.1 2017 ACTLABS VERSUS ALS CHECK ASSAY: ET-16-083 FOR GOLD



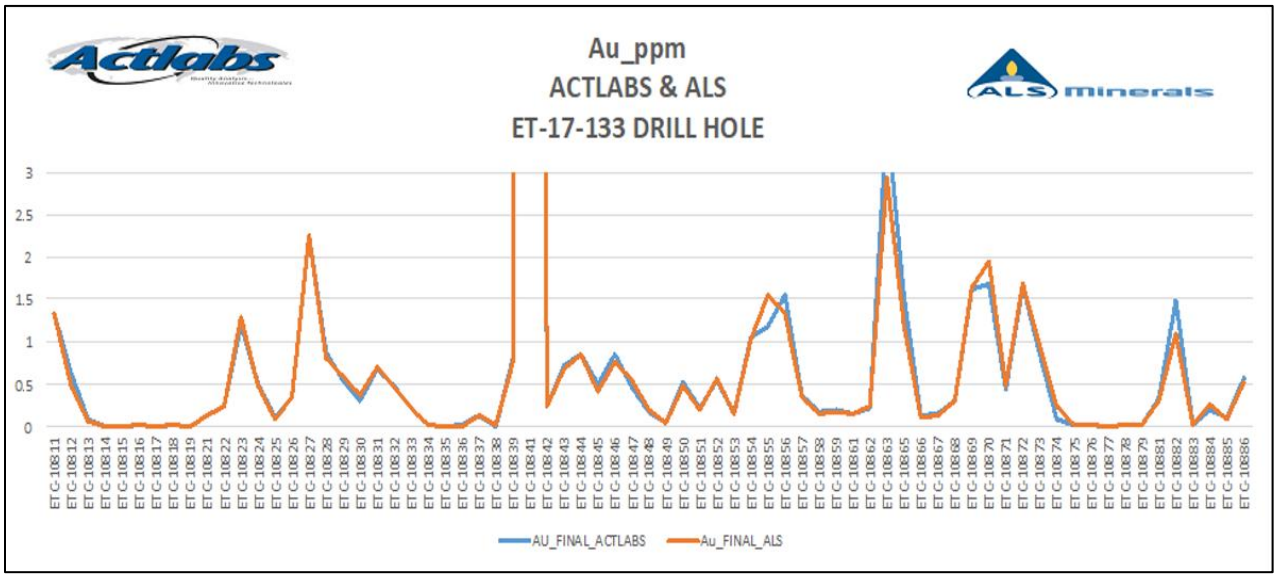
Source: *P&E (2017)*

FIGURE 11.2 2017 ACTLABS VERSUS ALS CHECK ASSAY: ET-16-101 FOR GOLD



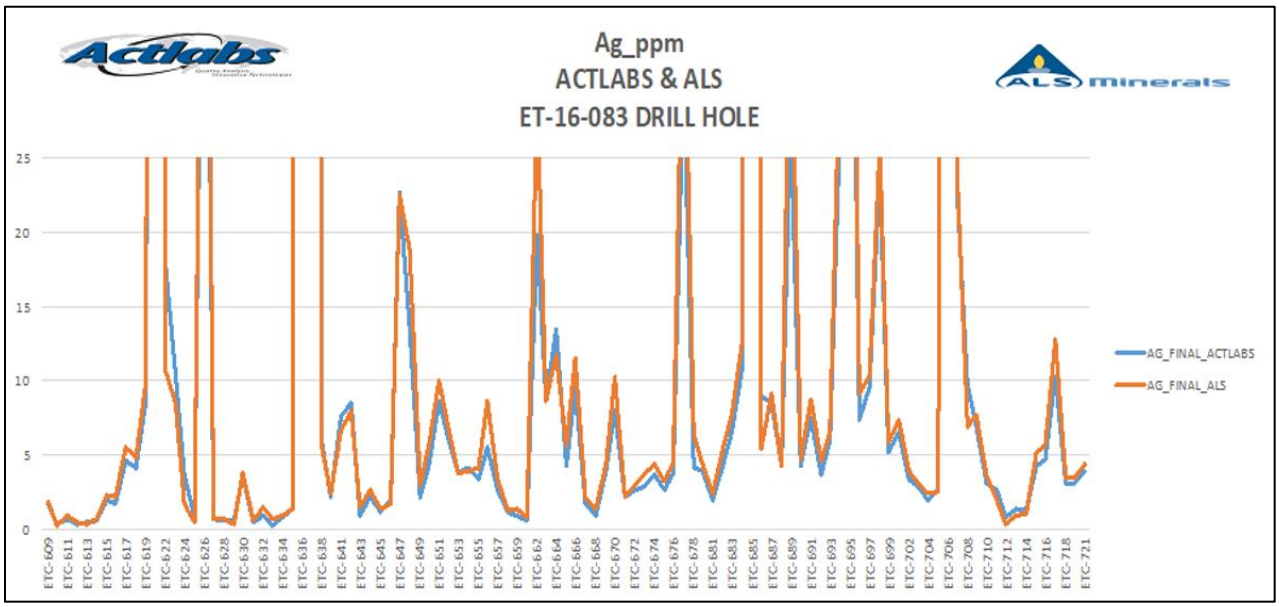
Source: *P&E (2017)*

FIGURE 11.3 2017 ACTLABS VERSUS ALS CHECK ASSAY: ET-17-133 FOR GOLD



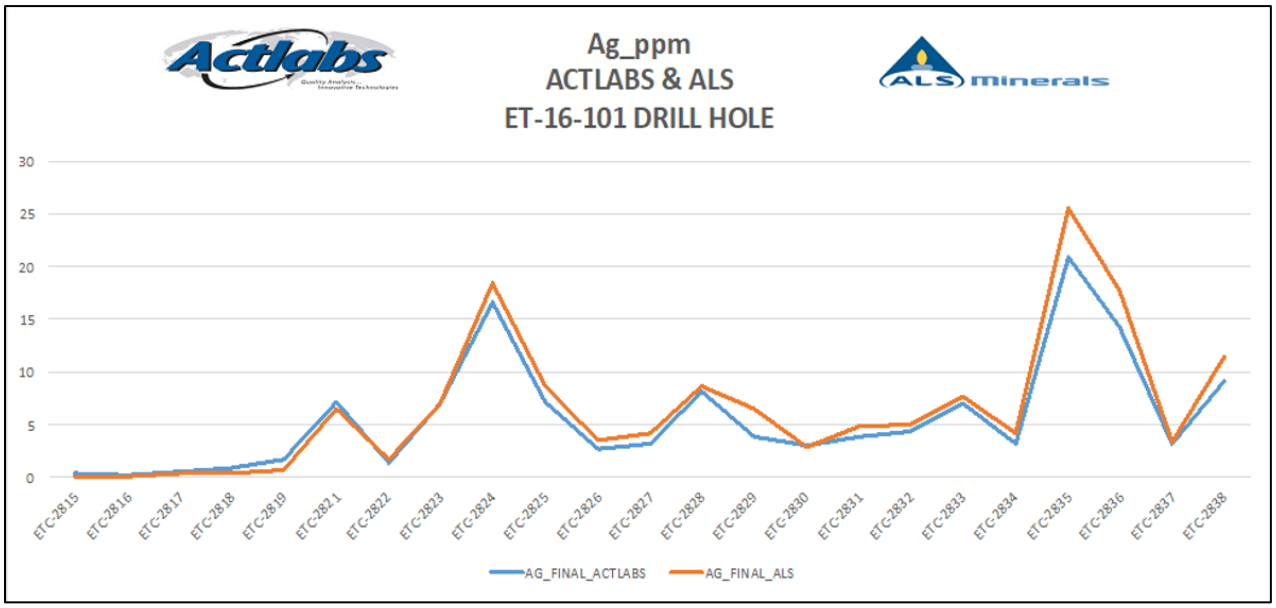
Source: *P&E (2017)*

FIGURE 11.4 ACTLABS VERSUS ALS CHECK ASSAY: ET-16-083 FOR SILVER



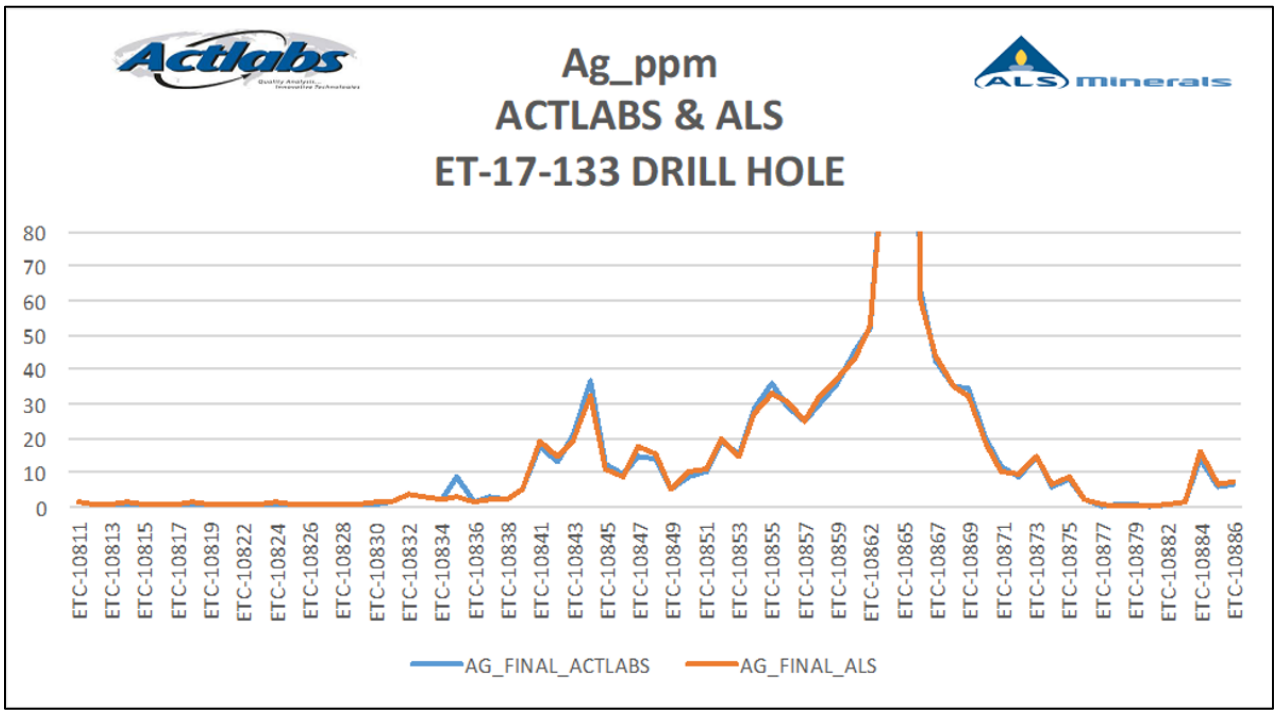
Source: *P&E (2017)*

FIGURE 11.5 ACTLABS VERSUS ALS CHECK ASSAY: ET-16-101 FOR SILVER



Source: *P&E (2017)*

FIGURE 11.6 2017 ACTLABS VERSUS ALS CHECK ASSAY: ET-17-133 FOR SILVER



Source: *P&E (2017)*

11.3.3 2022 to June 2023 Quality Assurance / Quality Control (ALS)

QA/QC procedures at El Tigre for the 2022 tailings auger sampling program continued in accordance with the 2016 and 2017 protocols. A total of 712 samples were sent for analysis during the 2022 auger drilling program, 599 of which were drill core samples. QC samples were routinely inserted into the sample stream and included: 56 CRMs, 29 blanks and 28 field duplicates (¼ drill core duplicate).

11.3.3.1 Performance of Certified Reference Materials

Two different CRMs were used during the 2022 drill program to monitor gold accuracy: the OxC168 and the Oxide CRMs. The OxC168 CRM was supplied by Rocklabs of Auckland, New Zealand and the Oxide CRM by Klen International Pty Ltd., of Neerabup, Western Australia. The Author has also reviewed ALS' internal QA/QC data to determine the quality of silver results throughout the drill program. Criteria for assessing CRM performance are described in Section 11.3.2.1.

There were no failures for either the OxC168 or the Oxide CRM. The Author also reviewed the performance of ALS' internal silver CRMs for the 2022 auger drilling program. Of the 188 CRMs inserted by the lab there were no material issues with accuracy found.

The Author is of the opinion that CRM performance throughout the 2022 auger drilling program demonstrates acceptable accuracy.

11.3.3.2 Performance of Blank Material

The blank material used by the Company was used to monitor gold and silver contamination. All blank data for gold were assessed by the Author. If the assayed value in the certificate was indicated as being less than detection limit, the value was assigned the value of half the detection limit for data treatment purposes. An upper tolerance limit of three times the detection limit was set. There was a total of 29 data points to examine. All samples for both gold and silver returned values at less than the detection limit, except for a single sample that returned 0.08 ppm Au.

The Author does not consider contamination to be an issue for the 2022 auger drilling data.

11.3.3.3 Performance of Field Duplicates

The 2022 field duplicate data were examined for gold and silver. There was a total of 28 duplicate pairs in the data set. Data were scatter graphed and the R^2 for Au and Ag found to be 0.9856 and 0.9991, respectively. Both data sets were found to have acceptable precision.

11.3.3.4 Performance of Laboratory Duplicate Samples

The Author reviewed ALS' internal pulp duplicate samples for the two different types of gold and silver analyses performed. There were 164 duplicate pairs to examine for the gold AA23 and silver AA62 analyses. Original versus duplicate sample data were graphed on scatter plots and all data plot closely to the 1:1 parity line. The R^2 values for gold were estimated at 0.995 and 0.997 for the gold and silver duplicates, respectively. The Author considers that precision is of an acceptable level for both gold and silver at the pulp level.

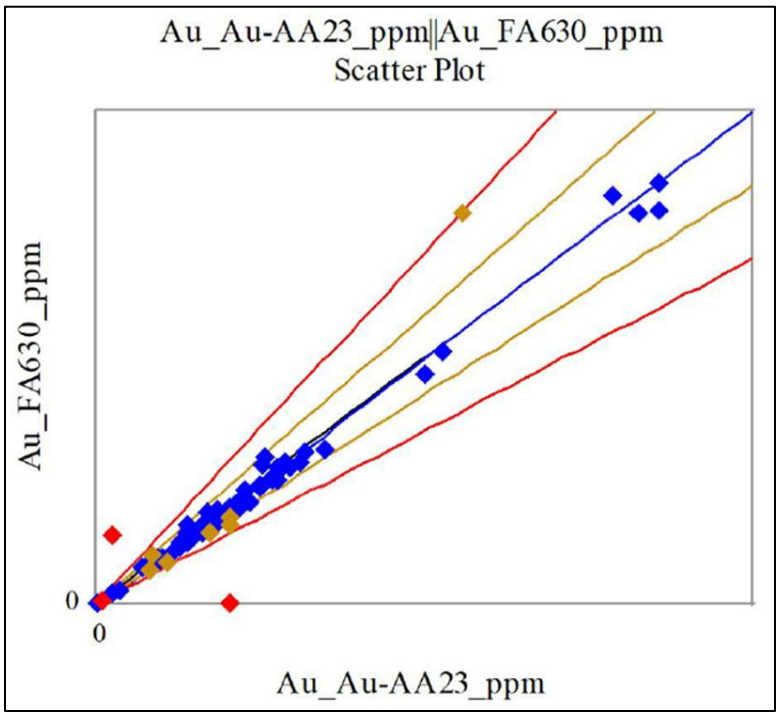
11.3.3.5 Silver Tiger 2022 Check Assaying ALS Versus Bureau Veritas

Silver Tiger undertook check assaying at Bureau Veritas in Hermosillo, Mexico in 2022 and selected 70 representative reject samples over varying grades from four El Tigre Tailing drill holes. Samples were selected from drill holes ETT-22-004, ETT-22-016, ETT-22-022 and ET-17-033. Samples at Bureau Veritas were pulverized to 85% passing 200 mesh. The pulps were then assayed for gold using a 30 g charge by fire assay with AAS finish and for silver by fire assay with gravimetric finish.

Bureau Veritas is independent of Silver Tiger and is ISO 9001 compliant and for selected methods, ISO 17025 compliant and has an extensive Quality Assurance/Quality Control ("QA/QC" or "QC") program to ensure that clients receive consistently high-quality data.

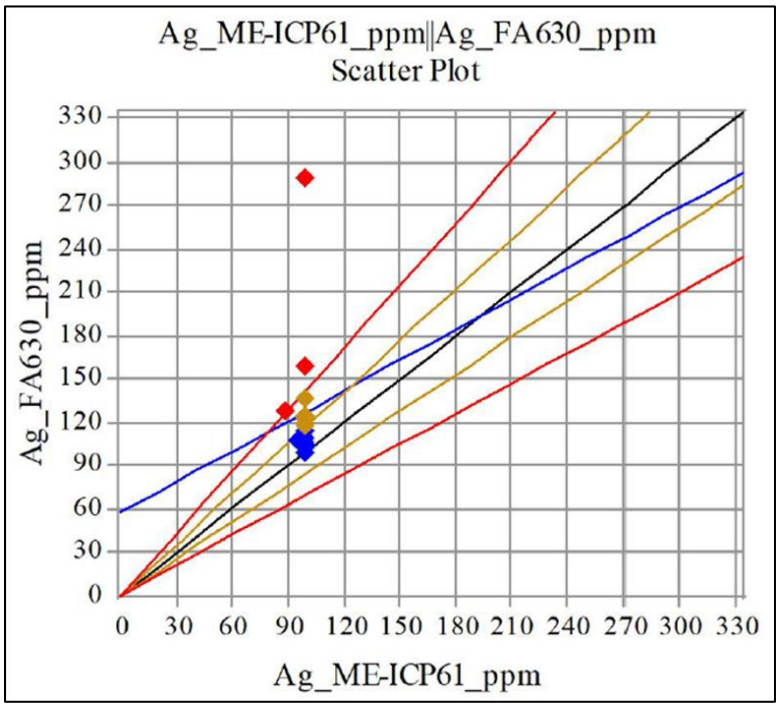
Results from Bureau Veritas were compared to the original ALS results to confirm the original reported grades. Silver Tiger plotted gold and silver results for the four drill holes on scatter graphs (see Figure 11.7 to Figure 11.9) and the Author considers comparison between the original and check assays to be acceptable for all test methods. The expected reduced precision in the silver results, due to the proximity of the original ME-ICP61 results to the 100 g/t threshold of this test limit, is shown in Figure 11.8.

FIGURE 11.7 PERFORMANCE OF CHECK ASSAYS FOR GOLD (AA23 VERSUS FA A630)



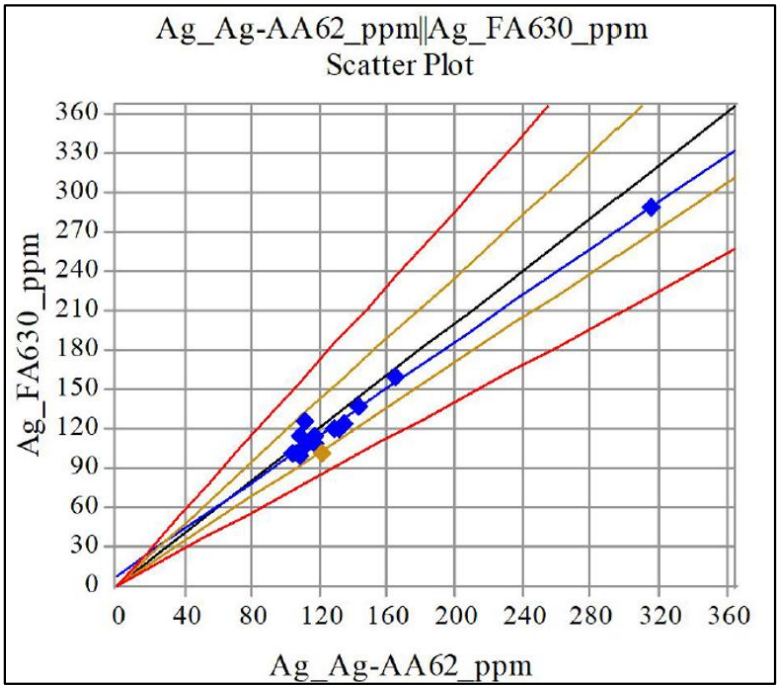
Source: *P&E (2023)*

FIGURE 11.8 PERFORMANCE OF CHECK ASSAYS FOR SILVER (ME-ICP61 VERSUS FA630)



Source: *P&E (2023)*

FIGURE 11.9 PERFORMANCE OF CHECK ASSAYS FOR SILVER (AA62 VERSUS FA630)



Source: *P&E (2023)*

11.3.3.6 2020 to June 2023 Quality Assurance / Quality Control (Bureau Veritas)

Silver Tiger continued implementing similar QC protocol for the next phases of drilling at El Tigre from 2020 to June 2023. CRMs were inserted approximately one in 34 samples for gold, blanks approximately every one in 76 samples, and field duplicates consisting of ¼ drill core were collected approximately every 76 samples. Silver CRMs were inserted at a lower rate, where silver mineralization was expected.

11.3.3.7 Performance of Certified Reference Materials

CRMs to monitor gold performance were inserted into the analysis stream approximately every 34 samples. Silver CRM insertion rate was considerably less, with a focus on insertion only where higher levels of mineralization were anticipated, due to the difficulty encountered in replenishing silver CRM stock during the pandemic. The following gold-only CRMs were utilized throughout the 2020 to June 2023 drill core sampling at the Project: Oxide CRM, OxC145, OxC152, OxC168, OxF85, OxG83, OxG140 and OxG141. The following gold- and silver-certified CRMs were also utilized: SQ47, SQ88 and SQ130. All CRMs, except the CRM Oxide, were supplied by Rocklabs of Auckland, New Zealand, and the CRM Oxide was supplied by Klen International Pty Ltd., of Neerabup, Western Australia. Criteria for assessing CRM performance remained as described in Section 11.2.2.1 of this Technical Report.

A summary of the CRMs used and their performance throughout the 2020 to June 2023 sampling at the Project is presented in Table 11.4. Performance charts for each CRM are available in P&E (2023).

TABLE 11.4 SUMMARY OF CERTIFIED REFERENCE MATERIALS USED AT EL TIGRE FROM 2020 TO JUNE 2023								
Certified Reference Material	**Certified Mean Value (ppm)**	**±1 SD (ppm)**	**±2 SD (ppm)**	**Lab Results**				
				No. Results	**No. (-) Failures**	**No. (+) Failures**	**% Failures**	**Average Result (ppm)**
Monitoring Gold								
CRM Oxide	1.24	0.046	0.092	393	12	23	8.9	1.232
OxC145	0.212	0.007	0.014	83	0	3	3.6	0.215
OxC152	0.216	0.008	0.016	351	1	3	1.1	0.216
OxC168	0.213	0.006	0.012	283	2	13	5.3	0.216
OxF85	0.805	0.025	0.05	46	0	0	0.0	0.801
OxG83	1.002	0.027	0.054	70	2	1	4.3	0.995
OxG140	1.019	0.022	0.044	158	3	3	3.8	1.023
OxG141	0.930	0.016	0.032	46	3	1	8.7	0.920
SQ47	39.88	0.850	1.700	42	2	1	7.1	38.824
SQ88	39.723	0.947	1.894	93	7	1	8.6	38.030
SQ130	39.47	0.885	1.770	9	1	0	11.1	39.58
Total				**1,574**	**33**	**49**	**5.2**	
Monitoring Silver								
SQ47	122.3	5.7	11.4	42	2	2	9.5	120.86
SQ88	160.8	5.1	10.2	93	33	9	45.2	148.71
SQ130	158.6	5.8	11.6	9	1	1	22.2	153.71
Total				**144**	**36**	**12**	**33.3**	

Source: *P&E (2023)*
Note: *SD = standard deviation.*

The Author considers that gold CRM performance is acceptable, with an overall failure rate of ~5% for all gold data. As indicated in Table 11.4, the SQ130 CRM shows the highest failure rate for gold, at 11.1%, followed by the Oxide CRM, OxG141, SQ88 and SQ47, with failure rates between 7.1% and 8.9%. All other CRMs recorded failure rates of ~5% and below. The Author reviewed all CRM failures and found all failures to be minority failures within a particular batch (with several other CRMs passing within each respective batch), very minor failures of no material impact, or failures in batches with negligible mineralization. The Author considers that the CRM data demonstrate acceptable accuracy in the 2020 to June 2023 El Tigre gold assay data.

As observed in Table 11.4, the number of silver CRMs inserted throughout the program is deficient and, as a result, the Author's assessment of silver data accuracy is based on the lab's own QC data performance and the umpire sampling program undertaken on the 2020 to 2023 drill core samples by Silver Tiger. The umpire sampling program is discussed in more detail in section 11.2.3.10 and

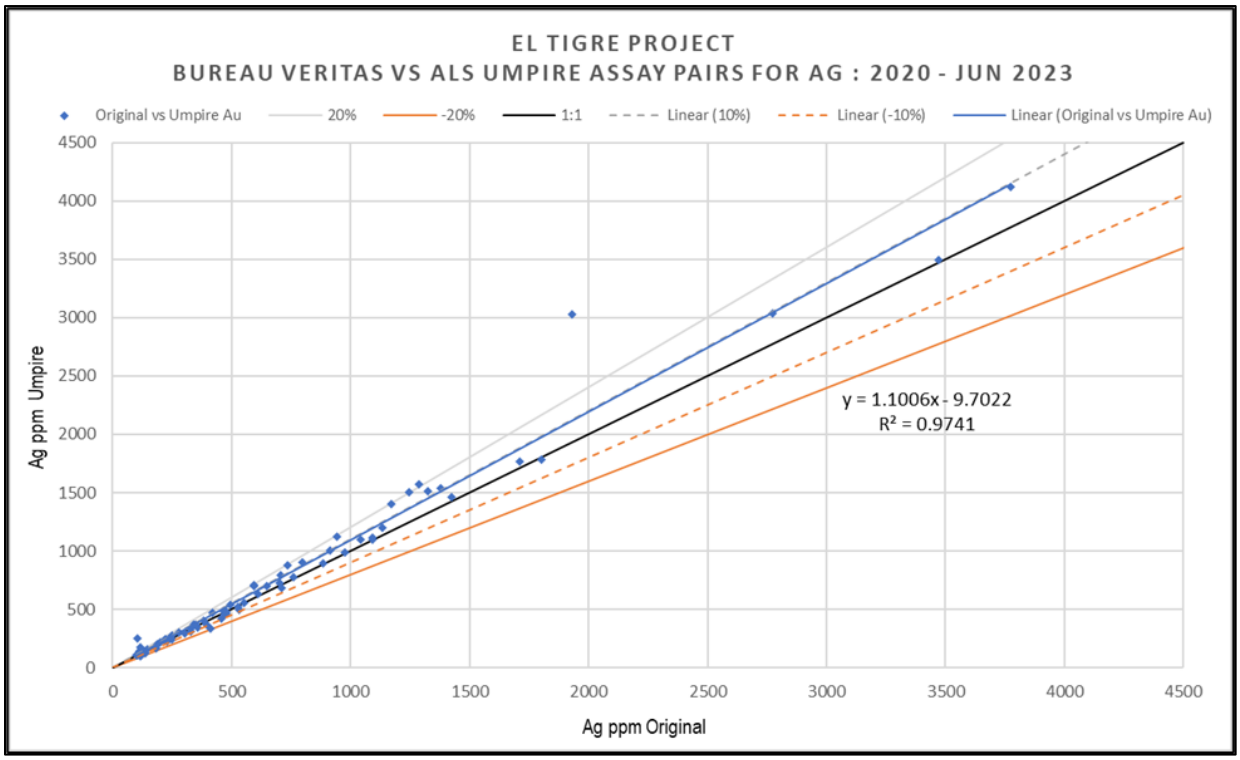
is also discussed in the current section regarding the need to verify at least 10% of the significantly mineralized silver assay data included in the current Mineral Resource Estimate.

The Author assessed Bureau Veritas' internal QC data for the drill core sample assaying throughout 2020 to June 2023. Silver CRM data were reviewed, and sufficient and appropriate CRMs have been used to assess assay results throughout the program. Failures within batches were observed to be more common earlier in the program, and a sufficient number of passing CRMs can be observed within each batch.

The Author reviewed drill core sample batches for the 2020 to June 2023 program and flagged all batches with significant silver mineralization that did not contain field-inserted silver CRMs. Within this subset of data, all samples returning grades of 100 g/t Ag and greater were further filtered, leaving 382 samples. Of these 382 samples, 75 Bureau Veritas-prepared pulp samples formed part of the 2023 umpire sampling program, giving a total of 19.6% of the target sample group verified. The original Bureau Veritas samples, all of which returned values of 100 g/t Ag or greater, were assayed by fire assay with gravimetric finish. The umpire samples were assayed by ALS in Hermosillo, Mexico, and the majority of those samples were also analyzed by fire assay with gravimetric finish. A total of four samples returning values <100 g/t Ag were analyzed by 4-Acid method with ICP-AES finish.

Direct comparison of the umpire samples was made via scatter graph, as shown in Figure 11.10. A distinct bias between the two labs can be observed in the scattered data, with the majority of data returning higher grades at the umpire laboratory, thereby confirming the original Bureau Veritas silver grades. Given the high percentage of CRM failures -3 standard deviations from the mean, and the bias detected in the umpire samples, there is the potential that the Bureau Veritas silver grades may be understated by ~5%. The Author is satisfied that the original Bureau Veritas silver assays have been confirmed and are suitable for use in the current Mineral Resource Estimate.

FIGURE 11.10 COMPARISON OF BV VERSUS ALS CONSTRAINED DATA >100 G/T AG (FA/GRAV VERSUS FA/GRAV OR 4-ACID/ICP-AES)



Source: *P&E (2023)*

11.3.3.8 Performance of Blank Material

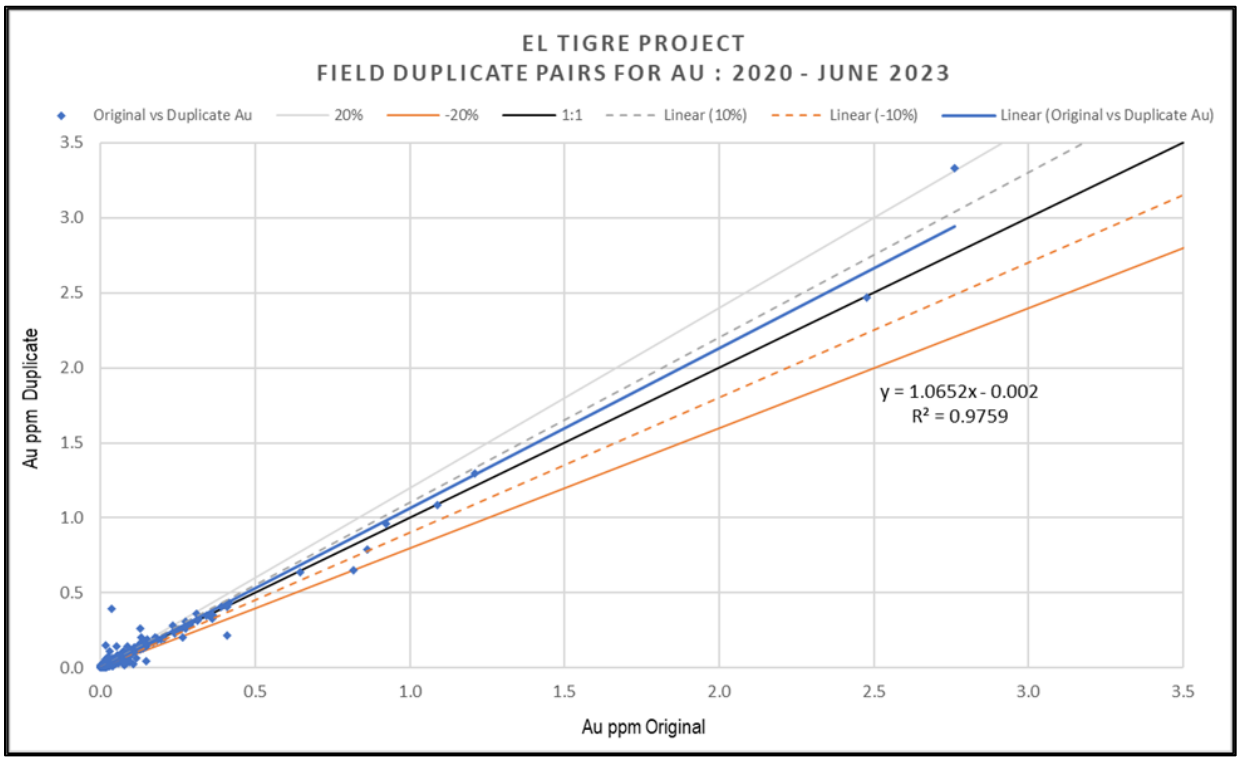
All blank data for gold and silver were reviewed by the Author. If the assayed value in the certificate was indicated as being less than detection limit, the value was assigned the value of one-half the detection limit for data treatment purposes. An upper tolerance limit of five times the detection limit was set. There were 710 data points to examine.

The vast majority of gold and silver data plot at or below the set tolerance limits and the Author does not consider the very few outliers to be significant to the integrity of the data. Performance charts for the blank material are available in P&E (2023).

11.3.3.9 Performance of Field Duplicates

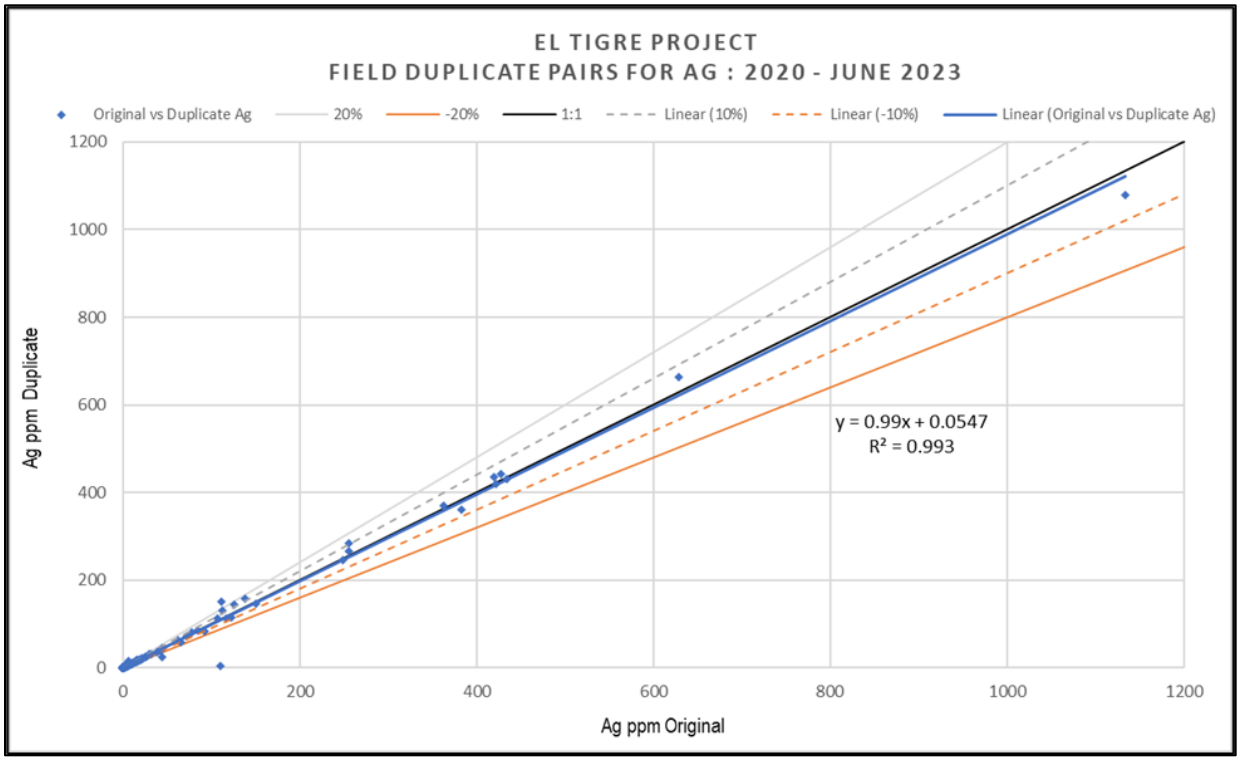
Field duplicate data for gold and silver were examined for the 2020 to June 2023 drilling at El Tigre. There were 710 duplicate pairs in the dataset. Data were scatter graphed (Figure 11.11 and Figure 11.12) and found to have acceptable precision at the field level for gold and silver, with R^2 values of 0.976 and 0.993, respectively, and the majority of the data plotting close to the 1:1 line (except where expected at grades close to lower detection limit).

FIGURE 11.11 PERFORMANCE OF FIELD DUPLICATES FOR GOLD



Source: *P&E (2023)*

FIGURE 11.12 PERFORMANCE OF FIELD DUPLICATES FOR SILVER



Source: *P&E (2023)*

11.3.3.10 Silver Tiger 2020 to June 2023 Umpire Assaying Bureau Veritas Versus ALS

Silver Tiger carried out a comprehensive umpire sampling program of a selection of the 2020 to June 2023 El Tigre drill core samples, to verify the primary lab (Bureau Veritas) results. A total of 10 drill holes from the 2021 program, 15 from the 2022, and one from the 2023 program (26 holes in total) were chosen to verify, ensuring that the selected holes were spread out along the length of the Deposit, extended to depth and temporally represented the 2021 to 2023 drilling.

A total of 1,055 pulp samples from the original drill core samples were assayed at ALS in Hermosillo, representing around 2% of the primary samples. When perusing the constrained data included in the current Mineral Resource Estimate, however, the percentage of primary samples umpire assayed is just under 10%. Each batch assayed contained a range of QC samples, including CRMs and blanks and, where possible, samples were assayed by the same method as the original primary laboratory analysis.

Samples at Bureau Veritas and ALS were analyzed for gold by fire assay with either AA or gravimetric finish. Samples at Bureau Veritas were analyzed for silver by multi-acid digest with ICP-MS finish or by fire assay with gravimetric finish, and by multi-acid digest with AAS finish or by fire assay with gravimetric finish at ALS. A summary of drill holes selected for umpire sampling and pulp samples sent for assaying is presented in Table 11.5.

TABLE 11.5
UMPIRE SAMPLING PROGRAM DRILL HOLES

Drill Hole ID	Area	Umpire Lab	Sample From	Sample To	Pulp Samples
ET-21-204	Fundadora	ALS	ETC-22941	ETC-22960	20
ET-21-215	Sooy	ALS	ETC-21967	ETC-21998	32
ET-21-229	Sooy	ALS	ETC-28291	ETC-28313	23
ET-21-231	Sooy	ALS	ETC-26311	ETC-26385	75
ET-21-246	Sooy	ALS	ETC-30161	ETC-30199	39
ET-21-250	Sooy	ALS	ETC-30396	ETC-30415	20
ET-21-264	Seitz Kelley	ALS	ETC-35346	ETC-35419	74
ET-21-271	Seitz Kelley	ALS	ETC-37176	ETC-37232	57
ET-21-287	Sooy	ALS	ETC-40490	ETC-40509	20
ET-21-295	Seitz Kelley	ALS	ETC-46061	ETC-46104	44
ET-22-309	Seitz Kelley	ALS	ETC-43505	ETC-43552	48
ET-22-335	Seitz Kelley	ALS	ETC-50416 ETC-50493 ETC-50530 ETC-50551	ETC-50443 ETC-50502 ETC-50537 ETC-50554	50
ET-22-349	Seitz Kelley	ALS	ETC-50724 ETC-50755	ETC-50748 ETC-50779	50
ET-22-357	Seitz Kelley	ALS	ETC-51901	ETC-51929	29
ET-22-367	Sooy	ALS	ETC-58033	ETC-58051	19
ET-22-377	Sooy	ALS	ETC-58205	ETC-58224	20

TABLE 11.5 UMPIRE SAMPLING PROGRAM DRILL HOLES					
Drill Hole ID	**Area**	**Umpire Lab**	**Sample From**	**Sample To**	**Pulp Samples**
ET-22-386	Benjamin	ALS	ETC-54728	ETC-54761	34
ET-22-395	Sooy	ALS	ETC-62086	ETC-62161	76
ET-22-405	Sooy	ALS	ETC-60925	ETC-60948	24
ET-22-410	Benjamin	ALS	ETC-61707	ETC-61732	26
ET-22-428	Sooy	ALS	ETC-63661 ETC-63714	ETC-63685 ETC-63746	58
ET-22-433	Sooy	ALS	ETC-67383	ETC-67475	93
ET-22-440	Sooy	ALS	ETC-69869	ETC-69957	89
ET-23-457	Sooy	ALS	ETC-74241	ETC-74307	67
ET-22-319	Sooy	ALS	ETC-48267	ETC-48285	19
ET-22-326	Sooy	ALS	ETC-49967	ETC-49983	17
Total					**1123**

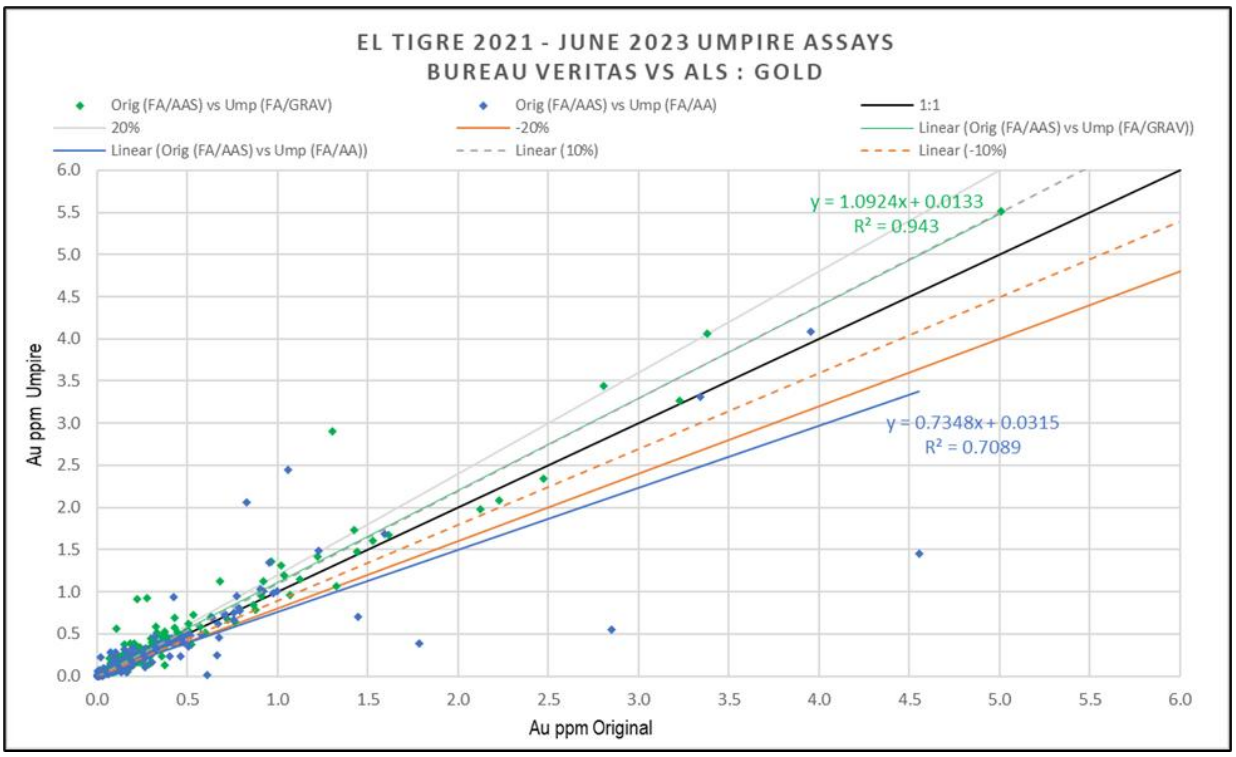
Source: *P&E (2023)*

The Author has reviewed the umpire assay results for the 2023 umpire assaying program at El Tigre, and comparisons made between the primary lab results and the umpire lab results, with the aid of line graph and scatter plots (Figure 11.13 and Figure 11.14). Comparison between Bureau Veritas and ALS gold grades by analytical method in Figure 11.13 reveals some dispersion between the primary and umpire samples. However, the Author considers the over-all between-lab correlation to be reasonable with minimal bias evident.

The subset of data comparing Bureau Veritas FA/AAS assays to ALS FA/GRAV assays (shown in green) returned an R^2 value of 0.943 and the subset comparing the FA/AA data (shown in blue) returned an R^2 value of 0.709, primarily due to a few gross outliers. The Author considers the between-lab gold assay comparison to be acceptable.

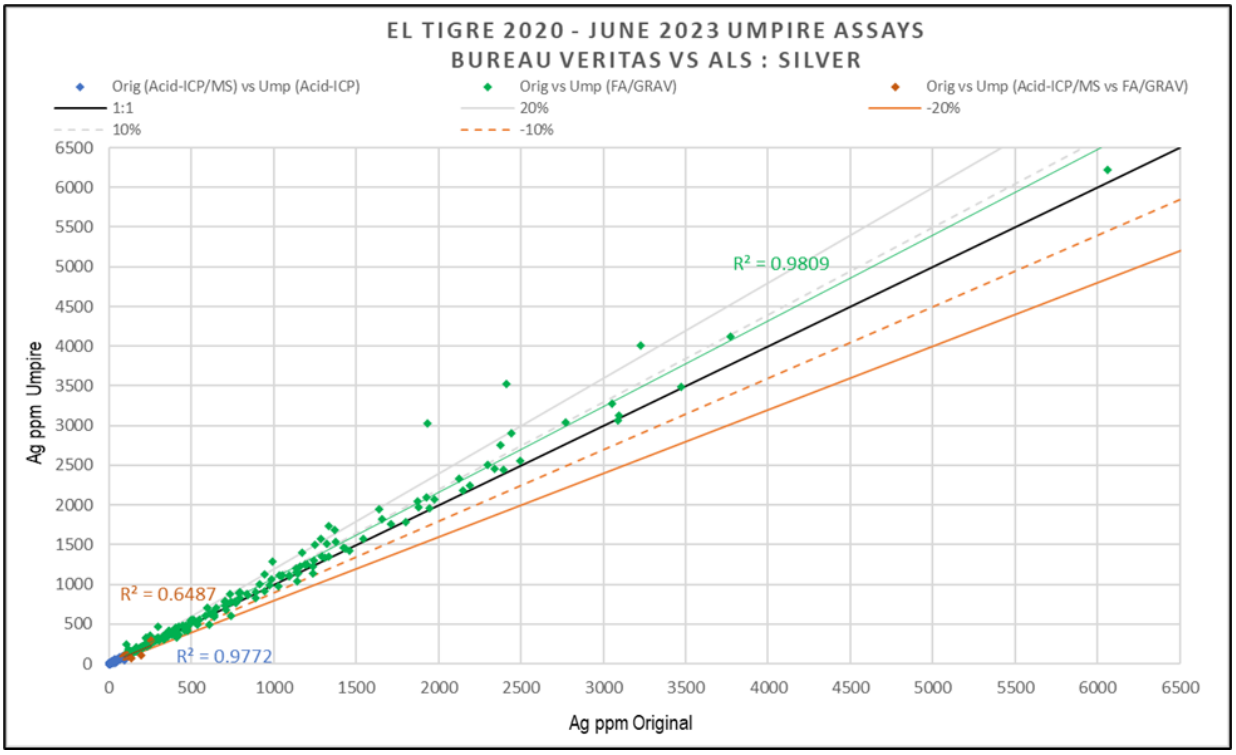
The umpire assays for silver (Figure 11.14) include three separate datasets: FA/GRAV (green data), Bureau Veritas Multi-Acid-ICP-MS assays versus ALS Multi-Acid-ICP assays (blue data) and Multi-Acid-ICP-MS or Multi-Acid-ICP versus FA/GRAV (rust data). The green and blue silver data show excellent between-lab correlation with R^2 values of 0.981 and 0.977, respectively, whereas the rust-coloured data, comparing different analytical methods, show less correlation with an R^2 value of 0.649. The data confirms the primary lab's silver results and indicates a slight bias between the two labs, with the ALS assays consistently higher than those analyzed at Bureau Veritas. There is the potential that the Bureau Veritas silver grades may be understated by ~5%.

FIGURE 11.13 PERFORMANCE OF UMPIRE ASSAYS FOR GOLD - BUREAU VERITAS VERSUS ALS



Source: *P&E (2023)*

FIGURE 11.14 PERFORMANCE OF UMPIRE ASSAYS FOR SILVER - BUREAU VERITAS VERSUS ALS



Source: *P&E (2023)*

11.3.4 July 2023 to May 2024 Quality Assurance / Quality Control (Bureau Veritas)

Silver Tiger continued implementing similar QC protocol for the next phases of drilling at El Tigre from July 2023 to May 2024. CRMs were inserted approximately one in seven samples for gold, blanks approximately every one in 20 samples, and field duplicates consisting of ¼ drill core were collected approximately every 20 samples. Silver CRMs were inserted at a lower rate of approximately 1 in 33 samples, where silver mineralization was expected.

11.3.4.1 Performance of Certified Reference Materials

CRMs to monitor gold performance were inserted into the analysis stream approximately every seven samples. Silver CRM insertion rate was considerably less (at a rate of approximately one every 33 samples), with a focus on insertion only where higher levels of mineralization were anticipated. The following gold-only CRMs were utilized throughout the July 2023 to May 2024 drill core sampling at the Project: Oxide CRM, OxC168, OxC185, OxE150, OxE182 and OxH163. The following gold- and silver-certified CRMs were also utilized: CDN-GS-2AD, CDN-MS-1707 and SQ130. All CRMs, except the CRM Oxide, were supplied by either Rocklabs of Auckland, New Zealand or CDN Resources Laboratories of Langley BC, and the CRM Oxide was supplied by Klen International Pty Ltd., of Neerabup, Western Australia. Criteria for assessing CRM performance remained as described in Section 11.3.2.1 of this Technical Report.

A summary of the CRMs used and their performance throughout the July 2023 to May 2024 sampling at the Project is presented in Table 11.6.

TABLE 11.6
SUMMARY OF REFERENCE MATERIALS USED AT EL TIGRE FROM JULY 2023 TO MAY 2024

Reference Material	Certified Mean Value (ppm)	±1 SD (ppm)	±2 SD (ppm)	Lab Results				
				No. Results	No. (-) Failures	No. (+) Failures	% Failures	Average Result (ppm)
Monitoring Gold								
CRM Oxide	1.24	0.046	0.092	128	0	1	0.8	1.238
CDN-GS-2AD	1.925	0.091	0.182	267	0	11	4.1	1.971
CDN-MS-1707	2.020	0.107	0.214	230	2	7	3.9	2.047
OxC168	0.213	0.006	0.012	858	0	28	3.3	0.219
OxC185	0.218	0.006	0.012	82	0	1	1.2	0.222
OxE150	0.658	0.016	0.032	378	0	0	0.0	0.660
OxE182	0.663	0.012	0.024	559	0	0	0.0	0.671
OxH163	1.313	0.026	0.052	318	1	0	0.3	1.311
SQ130	39.47	0.885	1.770	90	0	1	1.1	39.697
Total				**2,910**	**3**	**49**	**1.8**	
Monitoring Silver								
CDN-GS-2AD	12	0.6	1.2	267	8	1	3.4	11.79
CDN-MS-1707	27.9	1.45	2.9	230	10	0	4.3	26.74
SQ130	158.6	5.8	11.6	90	2	0	2.2	156.41
Total				**587**	**20**	**1**	**3.6**	

Source: *P&E (2024)*

The Author considers that gold CRM performance is acceptable, with an overall failure rate of below 2% for all gold data. As indicated in Table 11.7**Error! Reference source not found.**, the C DN-GS-2AD CRM shows the highest failure rate for gold, at 4.1%, followed by the CDN-MS-1707 and OxC168, with failure rates between 3.3% and 3.9%. All other CRMs recorded failure rates of ~1% and below. Silver CRM performance is also considered acceptable, with an overall failure rate of 3.6% for all silver data. The CDN-MS-1707 CRM shows the highest failure rate for gold, at 4.3%, followed by the CDN-GS-2AD at 3.4% and the SQ130 at 2.2%. The Author reviewed all CRM failures and found all to be minority failures within a particular batch (with several other CRMs passing within each respective batch), very minor failures of no material impact, or failures in batches with negligible mineralization. The Author considers that the CRM data demonstrate acceptable accuracy in the July 2023 to May 2024 El Tigre gold assay data.

11.3.4.2 Performance of Blank Material

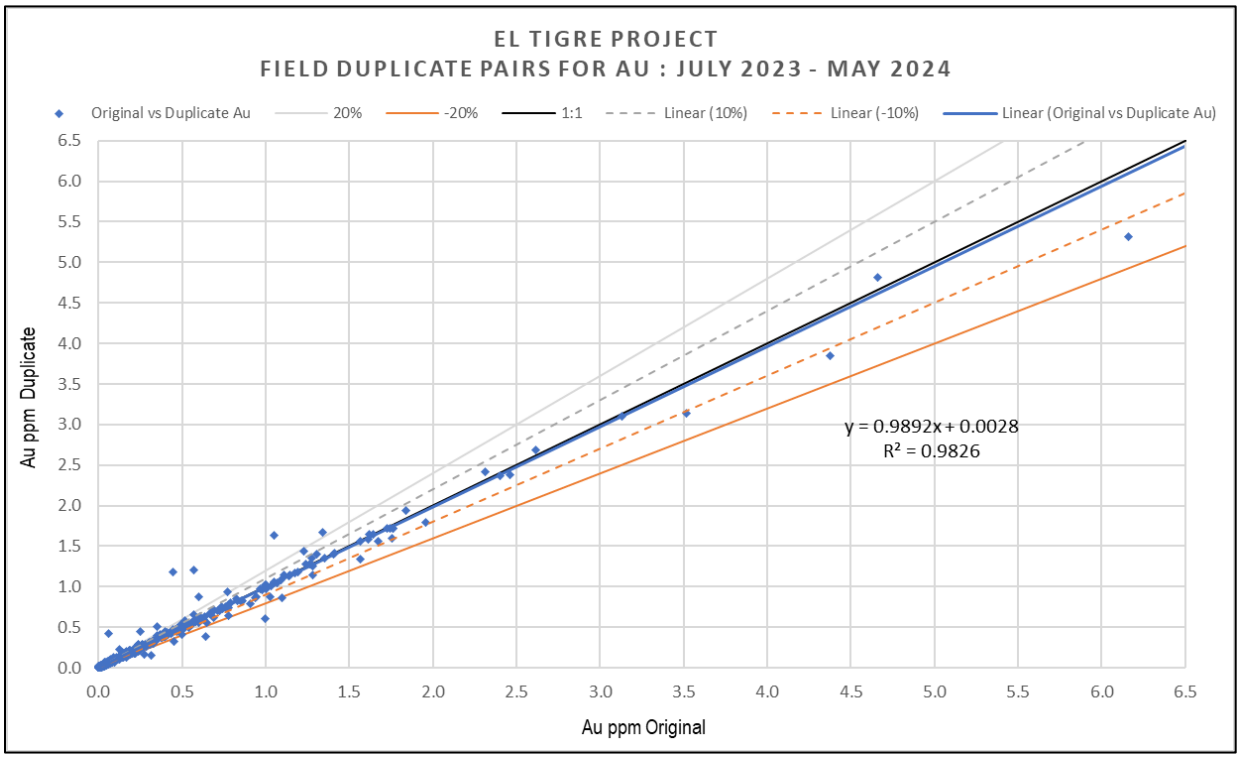
All blank data for gold and silver were reviewed by the Author. If the assayed value in the certificate was indicated as being less than detection limit, the value was assigned the value of one-

half the detection limit for data treatment purposes. An upper tolerance limit of five times the detection limit was set. There were 939 data points to examine and the vast majority of gold and silver data plot at or below the set tolerance limits. The Author does not consider the very few outliers to be significant to the integrity of the data.

11.3.4.3 Performance of Field Duplicates

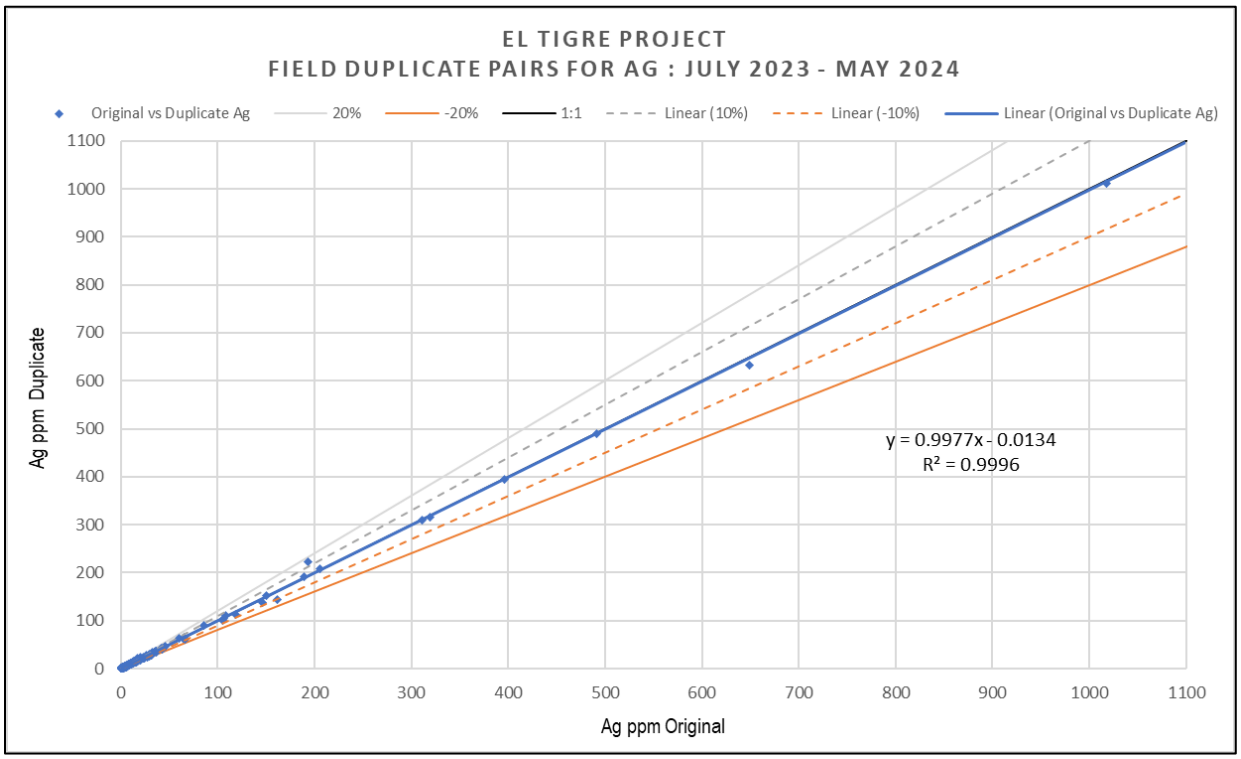
Field duplicate data for gold and silver were examined for the July 2023 to May 2024 drilling at El Tigre. There were 939 duplicate pairs in the dataset. Data were scatter graphed (Figures 11.15 and 11.16) and found to have acceptable precision at the field level for gold and silver, with R^2 values of 0.983 and 1, respectively, and the majority of the data plotting close to the 1:1 line (except where expected for gold at grades close to lower detection limit).

FIGURE 11.15 PERFORMANCE OF FIELD DUPLICATES FOR GOLD



Source: *P&E (2024)*

FIGURE 11.16 PERFORMANCE OF FIELD DUPLICATES FOR SILVER



Source: *P&E (2024)*

11.3.4.4 Silver Tiger July 2023 to May 2024 Umpire Assaying Bureau Veritas Versus ALS

Silver Tiger carried out a comprehensive umpire sampling program of a selection of the July 2023 to May 2024 El Tigre drill core samples, to verify the primary labs' (Bureau Veritas) results. A total of 14 drill holes from the 2023 program and six from the 2024 program (20 drill holes in total) were chosen to verify, ensuring that the selected drill holes were spread out along the length of the Deposit, extended to depth and temporally represented the July 2023 to May 2024 drilling.

A total of 1,192 pulp samples from the original drill core samples were assayed at ALS in Hermosillo, representing around 6% of the primary samples. When looking at the constrained data included in the current Mineral Resource Estimate, however, the percentage of primary samples umpire assayed is just under 10%. Each batch assayed contained a range of QC samples, including CRMs and blanks and, where possible, samples were assayed by the same method as the original primary laboratory analysis.

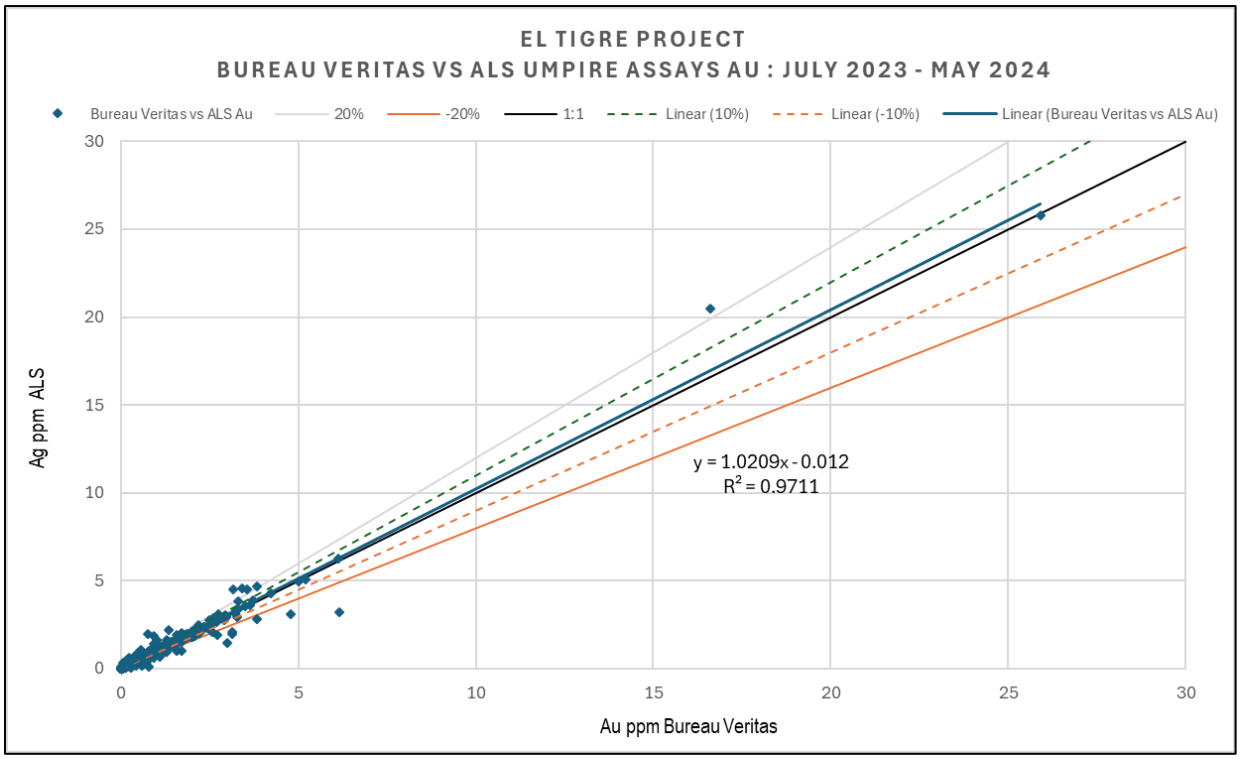
Samples at Bureau Veritas and ALS were analyzed for gold by fire assay with either AA or gravimetric finish. Samples at Bureau Veritas were analyzed for silver by multi-acid or aqua regia digest with AA or ICP-ES/MS finish or by fire assay with AA or gravimetric finish, and by multi-acid digest with ICP-AES finish or by fire assay with gravimetric finish at ALS. A summary of drill holes selected for umpire sampling and pulp samples sent for assaying is presented in Table 11.7.

TABLE 11.7 UMPIRE SAMPLING PROGRAM DRILL HOLES					
Drill Hole ID	**Area**	**Umpire Lab**	**Sample From**	**Sample To**	**Pulp Samples**
ET-23-446	Sooy	ALS	ETC-72098	ETC-72134	37
ET-23-455	Sooy	ALS	ETC-74048	ETC-74069	22
ET-23-463	Sooy	ALS	ETC-74885	ETC-74929	45
ET-23-479	La_Mula	ALS	ETC-64525	ETC-64659	135
ET-23-486	La_Mula	ALS	ETC-80128	ETC-80230	103
ET-23-496	La_Mula	ALS	ETC-81557	ETC-81639	83
ET-23-506	La_Mula	ALS	ETC-82921 ETC-85001	ETC-83000 ETC-85009	80 9
ET-23-513	La_Mula	ALS	ETC-101797	ETC-101932	136
ET-23-514	La_Mula	ALS	ETC-84588	ETC-84623	36
ET-23-529	La_Mula	ALS	ETC-87251	ETC-87305	55
ET-23-539	La_Mula	ALS	ETC-88565	ETC-88616	52
ET-23-543	La_Mula	ALS	ETC-87965 ETC-90001	ETC-88000 ETC-90007	36 7
ET-23-553	La_Mula	ALS	ETC-89798	ETC-89832	35
ET-23-563	La_Mula	ALS	ETC-92291	ETC-92350	60
ET-24-576	La_Mula	ALS	ETC-101797	ETC-101932	136
ET-24-586	La_Mula	ALS	ETC-93951	ETC-94009	59
ET-24-594	La_Mula	ALS	ETC-95514	ETC-95604	91
ET-24-602	La_Mula	ALS	ETC-96619	ETC-96679	61
ET-24-607	La_Mula	ALS	ETC-97883	ETC-97975	93
ET-24-615	La_Mula	ALS	ETC-100092	ETC-100203	112
			ETC-102295 ETC-102374	ETC-102337 ETC-102432	43 59
Total					**1,585**

Source: *P&E (2024)*

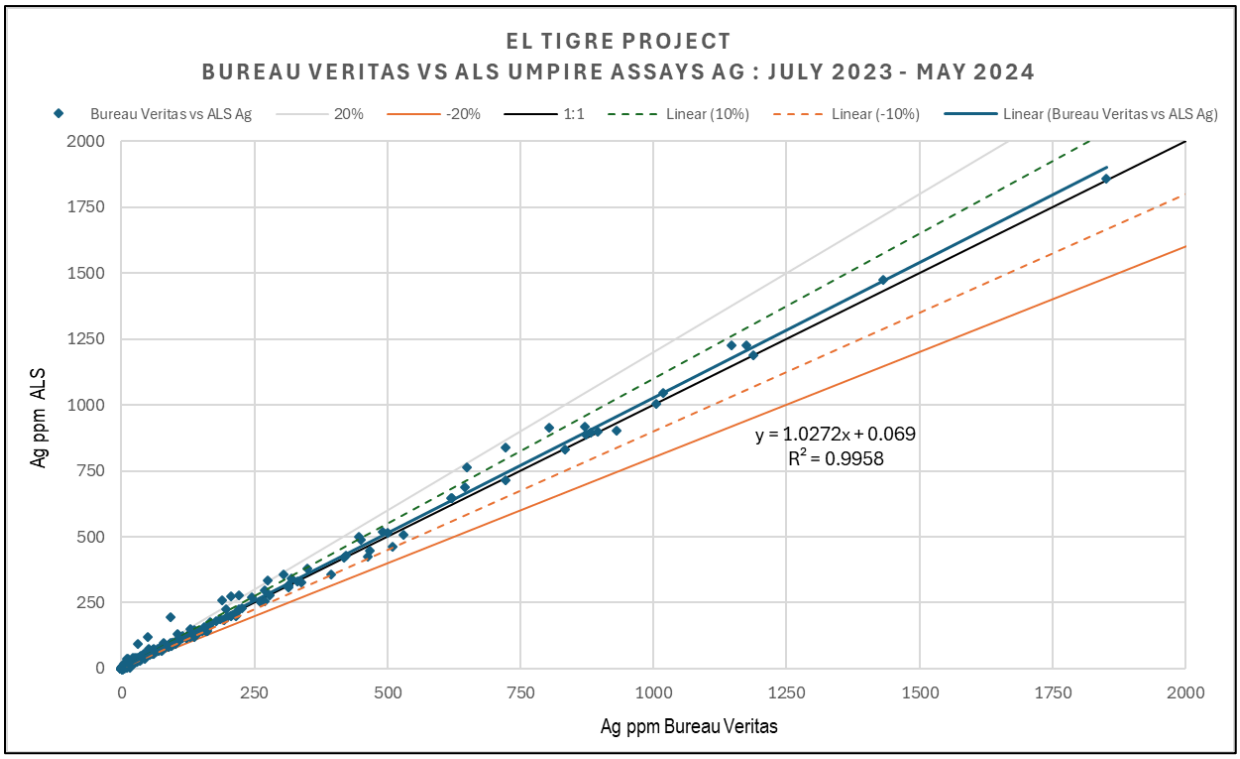
The Author has reviewed the umpire assay results for the July 2023 to May 2024 umpire assaying program at El Tigre, and comparison was made between the primary lab results and the umpire lab results with the aid of line graph and scatter plots (Figure 11.17 and Figure 11.18). Comparison between Bureau Veritas and ALS gold and silver grades reveals some dispersion between the primary and umpire samples. However, the Author considers the over-all between-lab correlation to be reasonable with minimal bias evident.

FIGURE 11.17 PERFORMANCE OF UMPIRE ASSAYS FOR GOLD - BUREAU VERITAS VERSUS ALS



Source: *P&E (2024)*

FIGURE 11.18 PERFORMANCE OF UMPIRE ASSAYS FOR SILVER - BUREAU VERITAS VERSUS ALS



Source: *P&E (2024)*

11.3.5 June 2024 to January 2025 Quality Assurance / Quality Control (Bureau Veritas)

Silver Tiger continued implementing the same QC protocol for the next phase of drilling at El Tigre from June 2024 to January 2025. CRMs were inserted approximately one in six samples for gold, blanks approximately every one in 20 samples, and field duplicates consisting of ¼ drill core were collected approximately every 20 samples. Silver CRMs were inserted at a lower rate of approximately 1 in 25 samples, where silver mineralization was expected.

11.3.5.1 Performance of Certified Reference Materials

CRMs to monitor gold performance were inserted into the analysis stream approximately every six samples. Silver CRM insertion rate was considerably less (at a rate of approximately one every 25 samples), with a focus on insertion only where higher levels of mineralization were anticipated. The following gold-only CRMs were utilized throughout the June 2024 to January 2025 drill core sampling at the Project: OxC185, OxE182 and OxH163. The following gold- and silver-certified CRMs were also utilized: CDN-GS-2AD, CDN-ME-1707 and CDN-ME-2301. All CRMs were supplied by either Rocklabs of Auckland, New Zealand or CDN Resources Laboratories of Langley BC. Criteria for assessing CRM performance remained as described in Section 11.3.2.1 of this Technical Report.

A summary of the CRMs used and their performance throughout the June 2024 to January 2025 sampling at the Project is presented in Table 11.8.

TABLE 11.8
SUMMARY OF REFERENCE MATERIALS USED AT EL TIGRE FROM JUNE 2024 TO JANUARY 2025

Reference Material	Certified Mean Value (ppm)	±1 SD (ppm)	±2 SD (ppm)	Lab Results				
				No. Results	No. (-) Failures	No. (+) Failures	% Failures	Average Result (ppm)
Monitoring Gold								
CDN-GS-2AD	1.925	0.091	0.182	9	0	0	0	1.967
CDN-ME-1707	2.02	0.107	0.214	7	0	0	0	2.105
CDN-ME-2301	1.329	0.665	1.329	4	0	0	0	1.317
OxC185	0.218	0.006	0.012	25	0	1	4	0.229
OxE182	0.663	0.012	0.024	25	0	0	0	0.677
OxH163	1.313	0.026	0.052	9	0	0	0	1.320
Total				**79**	**0**	**1**	**1.3**	
Monitoring Silver								
CDN-GS-2AD	12	0.6	1.2	9	0	1	11.1	12.8
CDN-ME-1707	27.9	1.45	2.9	7	0	0	0.0	27.4
CDN-ME-2301	318	7	14	4	1	0	25.0	305.3
Total				**20**	**1**	**1**	**10.0**	

Source: *P&E (2025)*

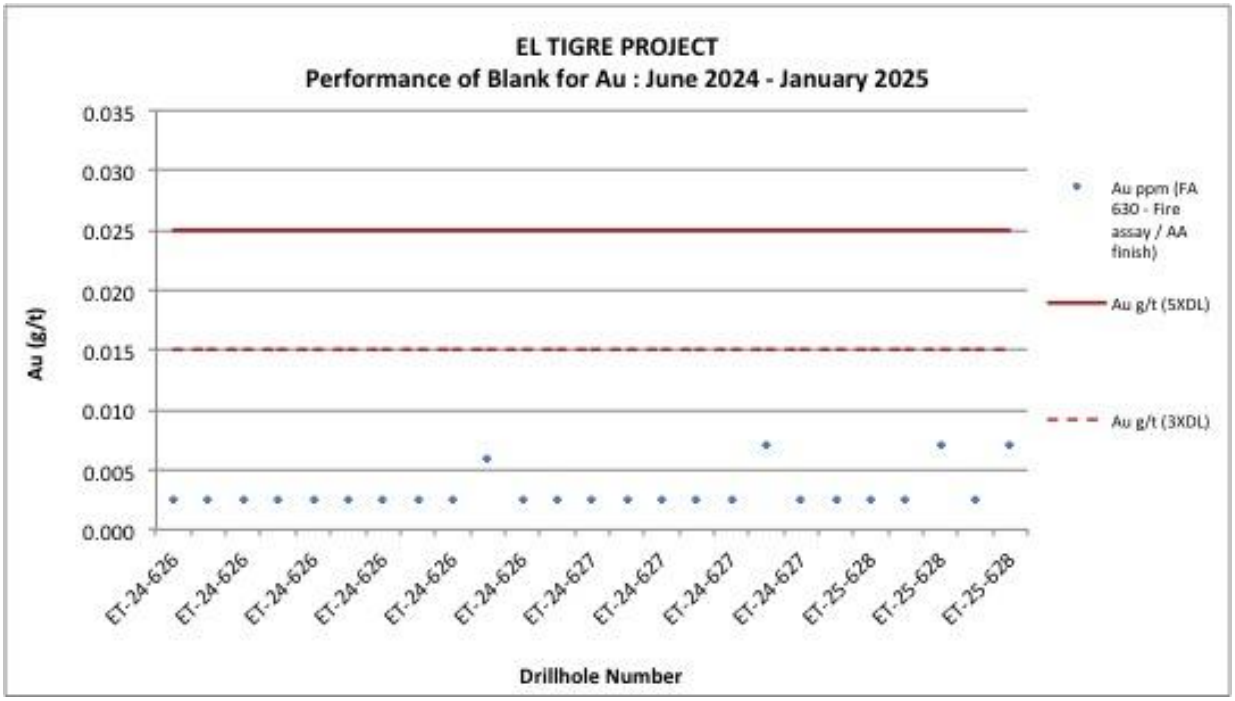
The Author considers that gold CRM performance is acceptable, with very few CRM failures recorded for the program and an overall failure rate of approximately 1% for all gold data and 10% for all silver data. The Author reviewed all CRM failures and found all to be minority failures within a particular batch, with several other CRMs passing within each respective batch. The Author considers that the CRM data demonstrate acceptable accuracy in the June 2024 to January 2025 El Tigre gold assay data.

11.3.5.2 Performance of Blank Material

All blank data for gold and silver were reviewed by the Author. If the assayed value in the certificate was indicated as being less than detection limit, the value was assigned the value of one-half the detection limit for data treatment purposes. An upper tolerance limit of five times the detection limit was set. There were 25 data points to examine.

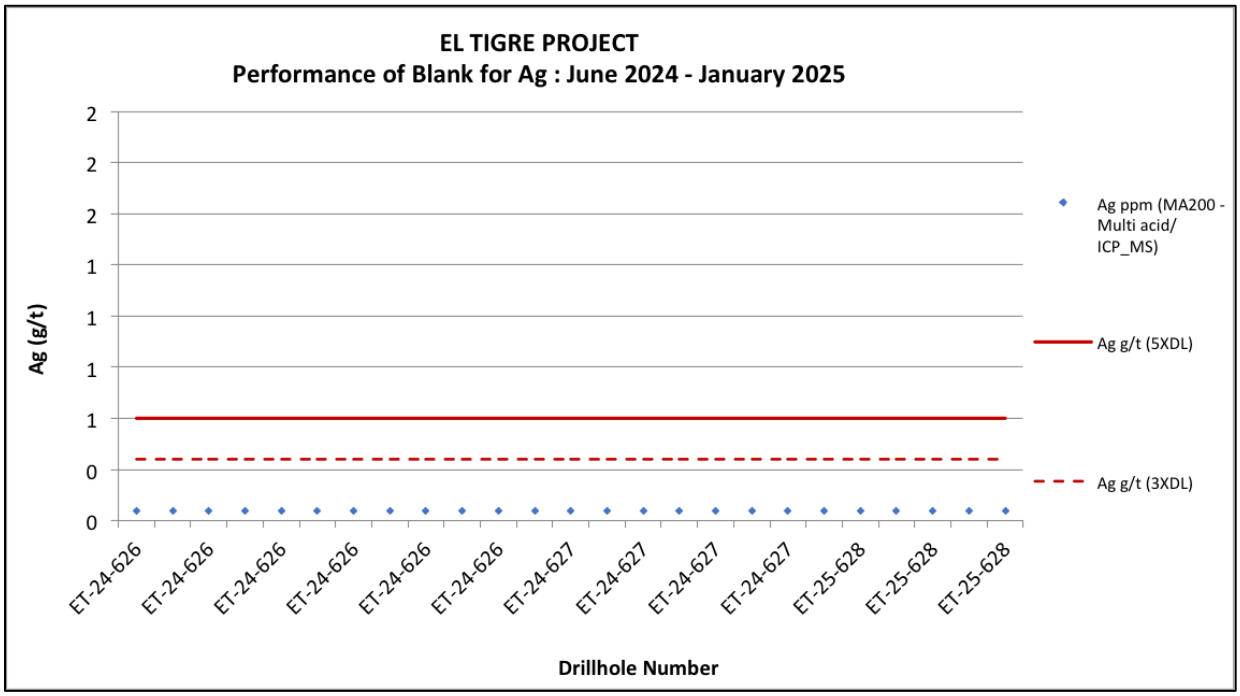
All gold and silver data plot below the set tolerance limits (Figures 11.19 and 11.20) and the Author does not consider contamination to be an issue with the data.

FIGURE 11.19 PERFORMANCE OF BLANKS FOR GOLD



Source: *P&E (2025)*

FIGURE 11.20 PERFORMANCE OF BLANKS FOR SILVER

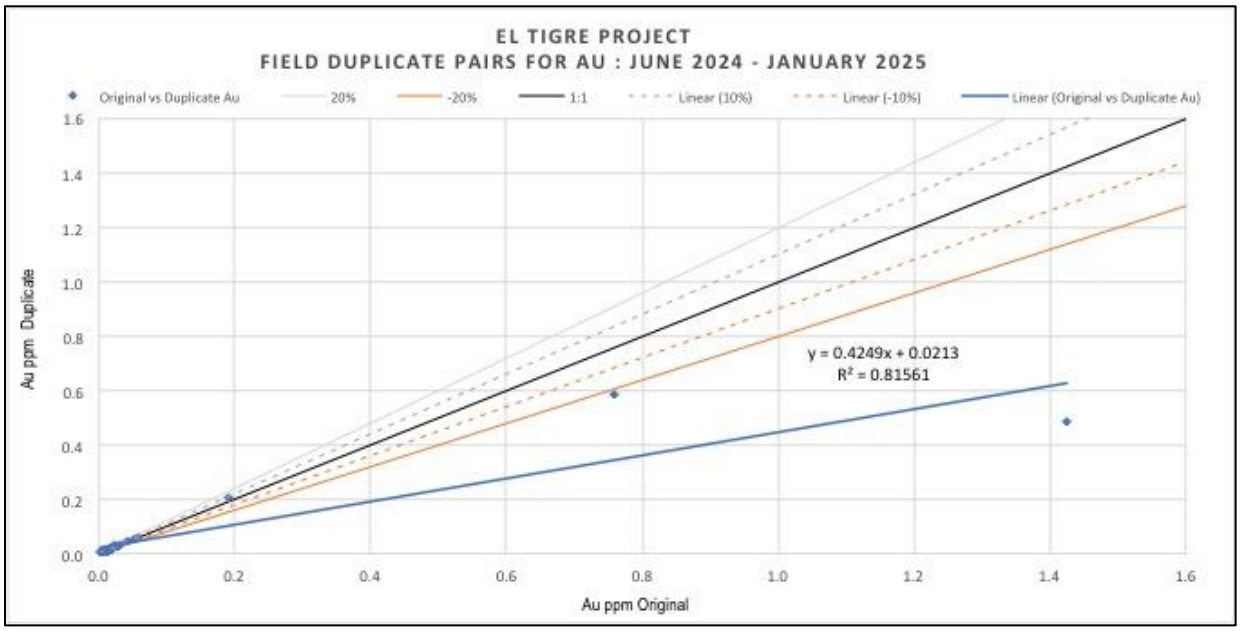


Source: *P&E (2025)*

11.3.5.3 Performance of Field Duplicates

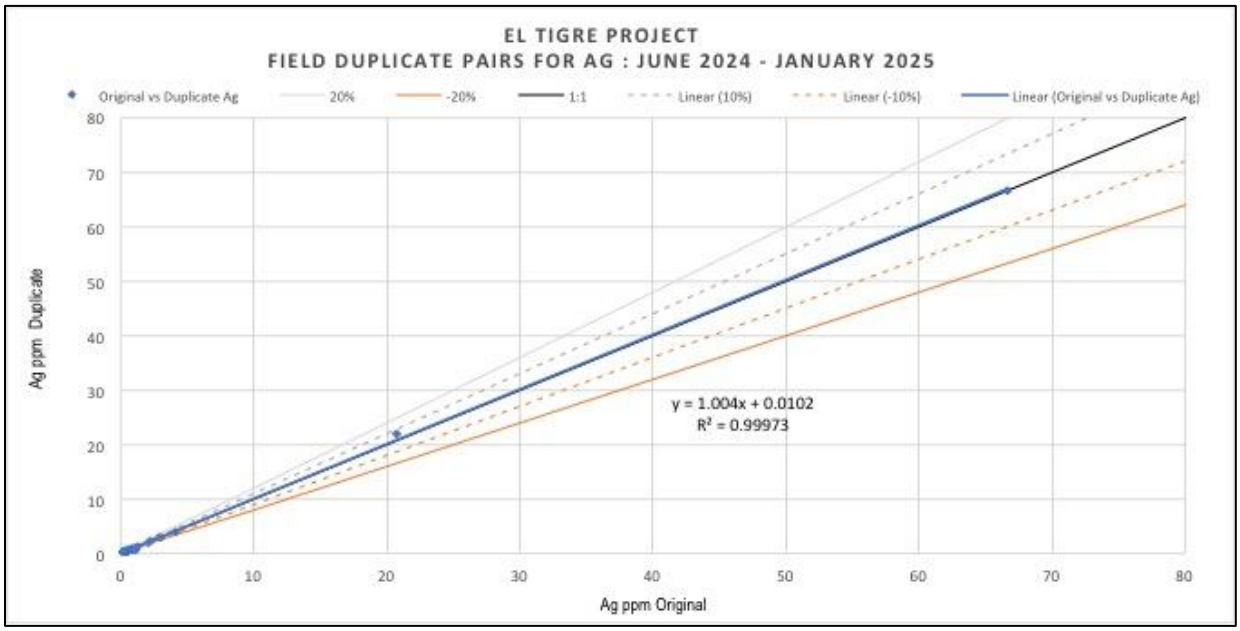
Field duplicate data for gold and silver were examined for the June 2024 to January 2025 drilling at El Tigre. There were 25 duplicate pairs in the dataset. Data were scatter graphed (Figures 11.21 and 11.22) and found to have acceptable precision at the field level for gold and silver, with R^2 values of 0.816 and 1, respectively, and the majority of the data plotting close to the 1:1 line.

FIGURE 11.21 PERFORMANCE OF FIELD DUPLICATES FOR GOLD



Source: *P&E (2025)*

FIGURE 11.22 PERFORMANCE OF FIELD DUPLICATES FOR SILVER



Source: *P&E (2025)*

11.4 CONCLUSION

The Author is of the opinion that sample preparation, security, and analytical procedures for the El Tigre Project 2010 to January 2025 drilling were adequate and examination of QA/QC results for all recent sampling indicates no significant issues with accuracy, contamination, or precision of the data. The Author considers the data to be of satisfactory quality and suitable for use in the current Mineral Resource Estimate.

12.0 DATA VERIFICATION

12.1 ASSAY VERIFICATION

12.1.1 2016 to 2017 Drill Hole Assay Data

The Authors completed verification of the drill hole assay database by comparison of the database entries with the assay certificates. The assay certificates were obtained in digital format directly from the assay laboratory and compiled.

Assay data ranging from 2016 through 2017 were verified for the El Tigre Project. A total of 48% (2,752 out of 5,742) of the constrained drilling assay data were checked for Au and 47% (2,670 out of 5,742) were checked for Ag, against the original laboratory certificates from Actlabs of Hermosillo. One minor error only, in the Ag data, was observed and corrected, with the overall impact to the database being negligible.

12.1.2 2011 to 2017 Low-Grade Stockpile Assay Data

Verification of the 2011 to 2017 low-grade stockpile assay data was performed by the Authors on 146 assay intervals for Au, Ag, Cu, Pb and Zn. The 146 verified intervals were checked against the signed ALS and Actlabs assay certificates, obtained from Silver Tiger in PDF format. The checked assays represent 100% of the entire low-grade stockpile assay database. No errors were encountered.

12.1.3 2022 Tailings Assay Data

Verification of the 2022 tailings assay data entry was performed by the Authors on 594 assay intervals for Au and Ag. The 594 verified intervals were checked against original digital assay laboratory certificates downloaded directly from the ALS Webtrieve™ website by the Authors. The checked assays represent 83.9% of the entire updated database. No errors were encountered.

12.1.4 2020 to 2023 Drill Hole, Surface and Adit Assay Data

The Authors completed verification of the updated drill hole, surface and adit assay databases by comparison of the database entries with the assay certificates. The assay certificates were downloaded directly from the Bureau Veritas WebAccess website by the Authors in digital format and compiled.

Assay data ranging from 2020 through 2023 were verified for the El Tigre Project. Exactly 98% (54,087 out of 55,215) of the drilling assay data were checked for Au, Ag, Cu, Pb and Zn and 9% (580 out of 6,474) of the surface and adit assay data were checked for Au, Ag, Cu, Pb and Zn, against the original laboratory certificates from Bureau Veritas. No errors were observed in the data.

12.1.5 July 2023 to May 2024 Drill Hole, Surface and Adit Assay Data

The Authors completed verification of the updated drill hole assay databases by comparison of the database entries with the assay certificates. The assay certificates were downloaded directly from the Bureau Veritas WebAccess website by the Authors in digital format and compiled.

Assay data ranging from July 2023 through May 2024 were verified for the El Tigre Project. Approximately 98% (20,225 out of 20,615) of the drilling assay data and 99% (11,099 out of 11,214) of the constrained data were checked for Au and Ag. Approximately 86% (998 out of 1,154) of the drilling assay data and 98% (792 out of 809) of the constrained data were checked for Cu, Pb and Zn. No errors were encountered in the data during the verification process.

12.1.6 June 2024 to January 2025 Drill Hole, Surface and Adit Assay Data

The Authors conducted verification of the updated drill hole assay database by comparison of the database entries with the assay certificates. The assay certificates were downloaded directly from the Bureau Veritas WebAccess website by the Authors in digital format and compiled.

Assay data ranging from June 2024 through January 2025 were verified for the El Tigre Project. Approximately 95% (490 out of 515) of the drilling assay data were checked for Au, Ag, Cu, Pb and Zn, with approximately 22% of the overall Au and Ag data checked (22,084 out of a total of 100,052 samples in the current MRE database). No errors were observed in the data.

12.2 DRILL HOLE DATA VERIFICATION

The Authors also validated the supplied Mineral Resource database by checking for inconsistencies in analytical units, duplicate entries, interval, length or distance values less than or equal to zero, blank or zero-value assay results, out-of-sequence intervals, intervals or distances greater than the reported drill hole length, inappropriate collar locations, survey and missing intervals and coordinate fields. A few minor errors were identified and corrected in the database where necessary.

12.3 2016 TO 2024 P&E SITE VISITS AND INDEPENDENT SAMPLING

The El Tigre Property was visited by David Burga, P.Geo., of P&E, from January 20 to 21, 2016, by Fred Brown, P.Geo., of P&E, from June 19 to 20, 2016, and on May 22, 2017, by Yungang Wu, P. Geo., of P&E, from July 13 to 14, 2017, and again by Mr. Burga from August 5 to 6, 2023, and May 20 to 21, 2024, for the purpose of completing site visits and due diligence sampling. During the site visits, Mr. Burga, Mr. Brown and Mr. Wu viewed drilling sites and outcrops, undertook GPS location verifications, discussed and reviewed data acquisition procedures, drill core logging procedures and QA/QC protocols.

Mr. Burga collected 13 verification samples from 13 diamond drill holes completed between 2012 and 2013 by El Tigre Silver and eight tailings samples (four El Tigre Silver and four Anaconda) in January 2016.

In May 2017, Mr. Brown collected 12 verification samples from five diamond drill holes completed by Silver Tiger in 2016 and 2017 and four samples from three Anaconda drill holes completed between 1982 and 1983. Mr. Wu collected 25 verification samples in July 2017. One diamond drill hole was sampled, and the remainder of the samples were to verify historical underground channel sampling. During Mr. Burga's second site visit to the Property in August 2023, he collected 20 verification samples from 20 diamond drill holes completed at the Property between 2020 and 2022. During Mr. Burga's May 2024 visit, he collected 15 verification samples from 12 diamond drill holes completed on the Property in 2023 and 2024. Drill core samples were collected by taking the half drill core remaining in the drill core box to independently confirm the presence and tenor of mineralization. The underground samples were taken directly beneath the client channels, which were marked by tags. When the drill core samples were collected, they were placed in a large bag and taken by Messrs. Burga, Brown and Wu to ALS in Hermosillo (2016, 2017 and 2024), or by Mr. Burga to the Activation Laboratories Ltd. (Actlabs) facility in Ancaster, Ontario (2023) for preparation and analysis.

All samples at ALS and Actlabs were analyzed for gold and silver by fire assay with a gravimetric finish and bulk densities were determined.

ALS has developed and implemented strategically designed processes and a global quality management system at each of its locations. The global quality program includes internal and external inter-laboratory test programs and regularly scheduled internal audits that meet all requirements of ISO/IEC 17025:2017 and ISO 9001:2015. All ALS geochemical hub laboratories are accredited to ISO/IEC 17025:2017 for specific analytical procedures.

The Actlabs' Quality System is accredited to international quality standards through ISO/IEC 17025:2017 and ISO 9001:2015. The accreditation program includes ongoing audits, which verify the QA system and all applicable registered test methods. Actlabs is also accredited by Health Canada. Both ALS and Actlabs are independent of Silver Tiger and SPM.

12.4 MAY 2025 P&E SITE VISIT AND INDEPENDENT SAMPLING

The El Tigre Property was visited by Mr. Greg Robinson, P.Eng., of P&E, from May 5 to 9, 2025, for the purpose of completing a site visit and due diligence sampling. During the site visit, Mr. Robinson accessed the Property via a 45 km-long dirt road off a main highway. Transit time to the site is approximately six hours from Hermosillo, with the final 45 km taking approximately two hours. Mr. Robinson noted that the road was in good condition, one to one and a half lane for light vehicles the entire way, and crossing local ranches (Figure 12.1). A 'ghost town' exists from a previous operation, with historical tailings storage outside of the town, and the foundations for the historical mill are visible from the road as progress is made through the old town.

The Project camp/office complex is located in the old Company store, with new buildings in place for the core shack/core cutting area (Figure 12.1). The core shack is well organized. Overflow core is stacked outside and covered from the elements. The camp comprises a main office area, a kitchen/eating area, approximately four to eight private rooms, bathroom and shower facilities for both genders, and a large open bunkhouse with approximately eight sets of bunk beds that open to the main corridor/common area. Total sleeping capacity is approximately 20. The camp is located across a valley from the planned surface mine/underground mining area situated on the El

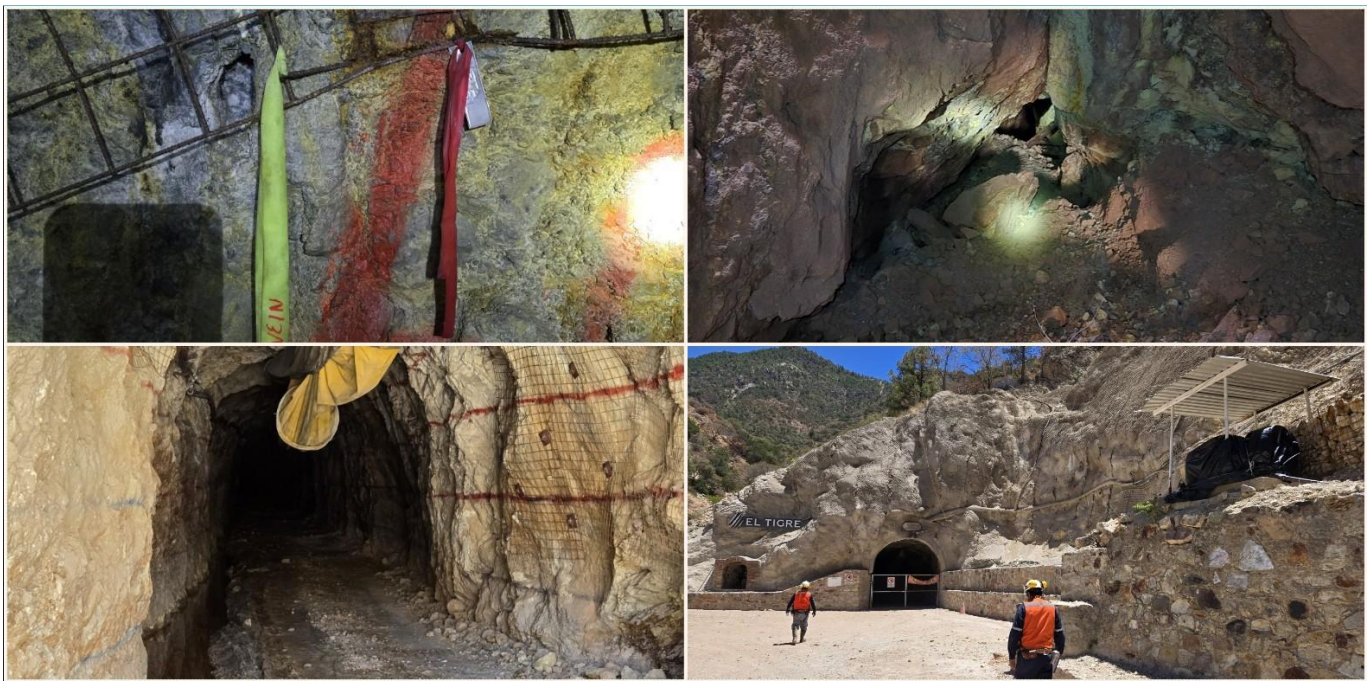
Tigre/Sooy/SK Veins, and to the south of the Protectora/Fundadora area on a flat plateau, and may be too close to the projected operations to remain in use during production.

FIGURE 12.1 EL TIGRE SITE PHOTOS

Source: *P&E (2025)*
Description of Images *– Clockwise from top left: (A) Access roads over surface mine area; (B) Historical process plant foundations; (C) Historical tailings; (D) View over camp facilities, looking north.*

Site inspection included general views of the area planned for surface mining, and an investigation of various underground openings. Extensive historical underground mining has taken place on the site, and numerous adits and crosscuts intersect surface. The "Level 7" adit (Figure 12.2) has been rehabilitated by contractor Cominvi, and has infrastructure suitable for initial portal development, although no room for ore storage or a good parking/laydown area. Drainage in the adit is poor, even though ditches are in place, indicative of insufficient grade for good drainage. Ground support is acceptable, however, split sets were used in the back and are degrading. Rebar would have been a better choice. A ground fall exists at the end of the primary adit, and occurs again in a bypass drift, as a result of intersecting a fault structure. The Angela Vein is visible, if small, in the wall of the bypass drift. If Level 7 is used again for primary access, it is recommended that a second bypass be driven and the fault structure intersected only after probe drilling and additional ground support plans are in place.

FIGURE 12.2 EL TIGRE LEVEL 7 UNDERGROUND WORKINGS PHOTOS



Source: *P&E (2025)*
Description of Images *– Clockwise from top left: (A) Angela Vein; (B) Level 7 bypass rockfall; (C) Level 7 main adit; (D) Level 7 portal.*

The Johnny Crosscut was also investigated (Figure 12.3) and terminates in a ground fall. This area contains old rail (appears to be badly corroded 10-20 lb rail). Rail cars and mucking machines at camp appear to be from a more recent time period. Sample tags from recent sampling are visible on the walls, and channel sample cuts are visible nearby on surface. Other unnamed adits were also investigated and found to contain old rail. One adit also appears to contain a raise to surface, as there is a visible opening above the level, and boards on the sill indicative of the raise continuing downwards: a structure is visible in the wall of the raise and may be an ore vein. Overall, the rock quality in the old tunnels is relatively good, however, the ground falls observed in Johnny and Level 7 indicate that quality is definitely structurally controlled, and that any areas around faults are likely to be collapsed. At least two stopes are open to surface on the north side of the main valley (camp side).

FIGURE 12.3 EL TIGRE JOHNNY CROSSCUT UNDERGROUND WORKINGS PHOTOS

Source: *P&E (2025)*
Description of Images *– Clockwise from top left: (A) Sample tag in Johnny Crosscut; (B) Johnny Crosscut drift; (C) Rockfall at Johnny Crosscut.*

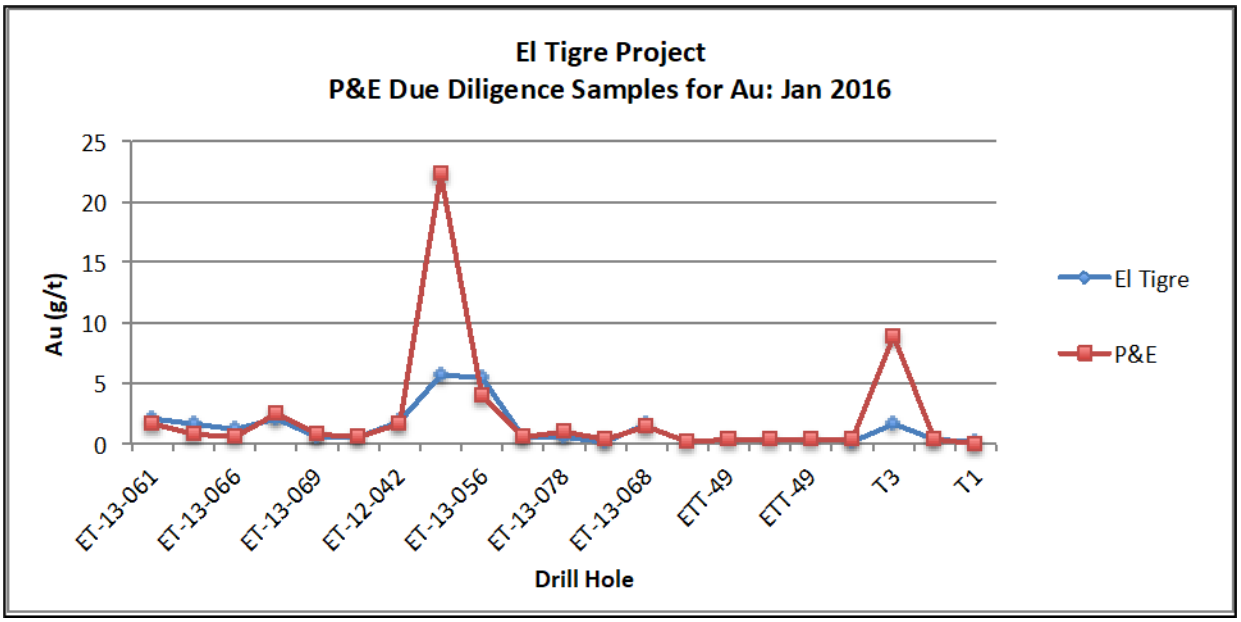
Extensive surface access roads to drill pads and site infrastructure criss-cross the site, and are generally in passable, if rough, condition.

Mr. Robinson considers that, overall, the site shows all the hallmark signs of a previous underground producer, and should be suitable for brownfields exploitation using either or both of surface mining or underground mining methods.

During Mr. Robinson's May 2025 visit, he collected 17 verification samples, targeting a range of low-, medium- and high-grade mineralization, from nine diamond drill holes completed on the Property in 2024. Drill core samples were collected by taking the half drill core remaining in the drill core box to independently confirm the presence and tenor of mineralization. Once collected, the drill core samples were placed in a large bag and taken by Mr. Robinson to ALS in Hermosillo for preparation and analysis. Samples were kept in Mr. Robinson's possession until delivered to ALS in Hermosillo. All samples at ALS were analyzed for gold and silver by fire assay with a gravimetric finish and bulk densities were determined.

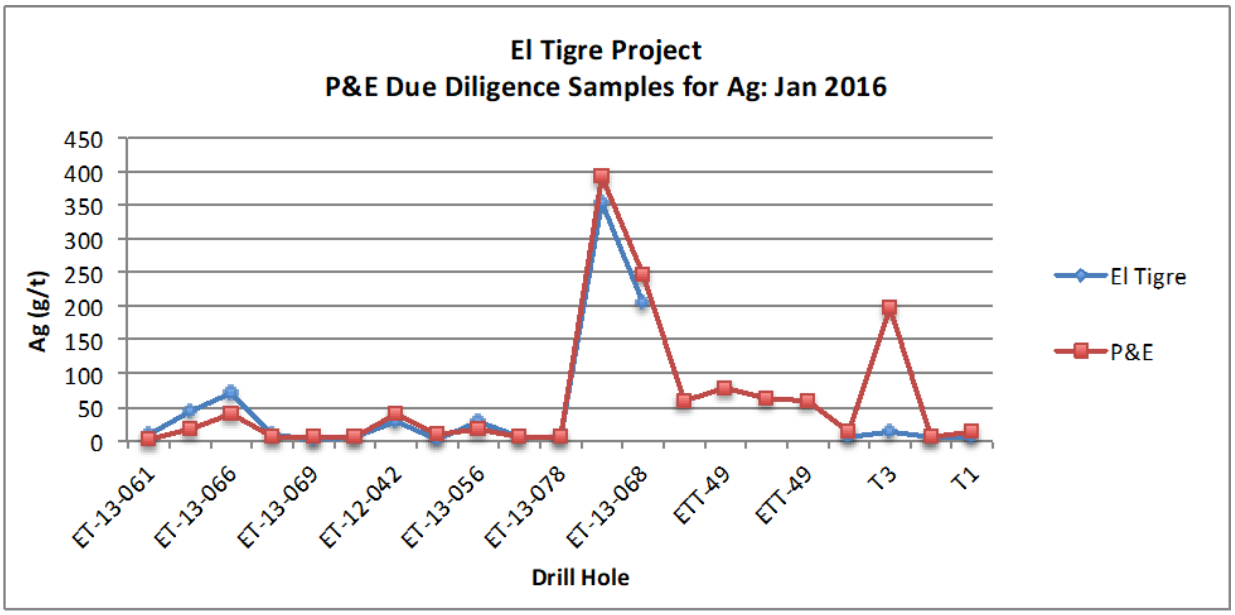
Results of the El Tigre site visit verification samples for both Au and Ag are presented in Figure 12.4 to Figure 12.18.

FIGURE 12.4 RESULTS OF JANUARY 2016 GOLD VERIFICATION SAMPLING BY P&E



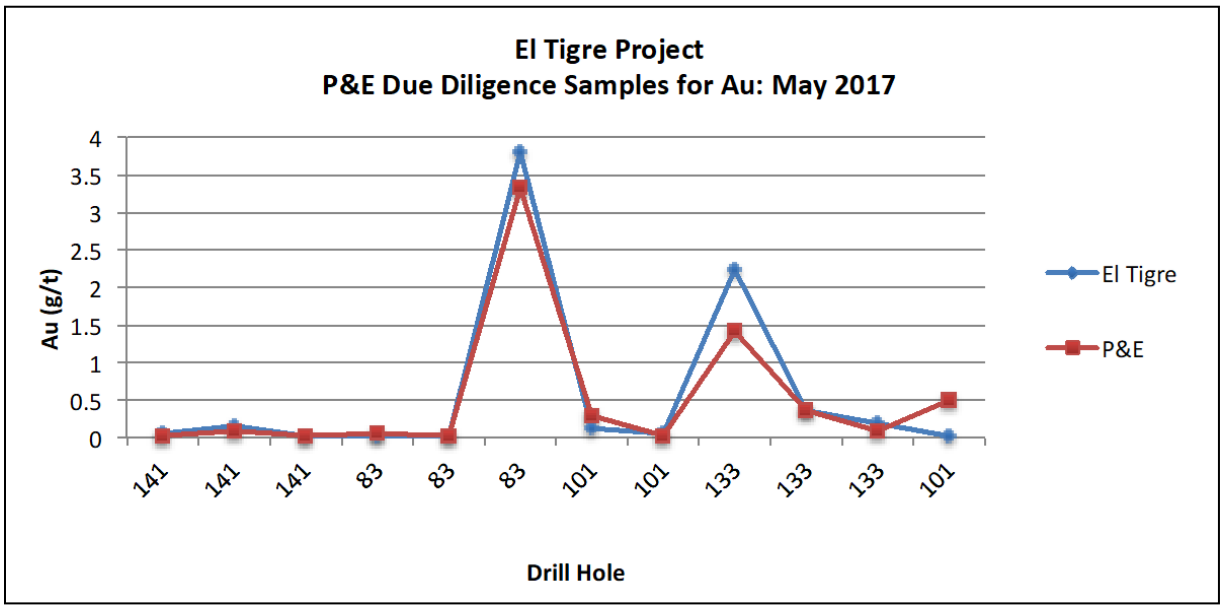
Source: *P&E (2016)*

FIGURE 12.5 RESULTS OF JANUARY 2016 SILVER VERIFICATION SAMPLING BY P&E



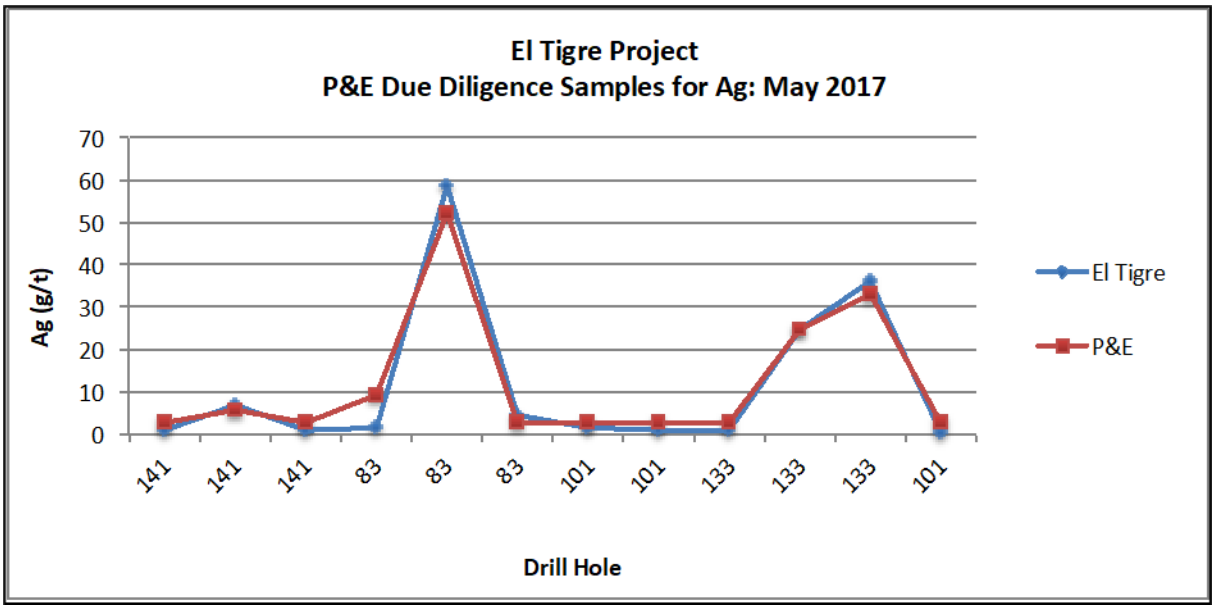
Source: *P&E (2016)*

FIGURE 12.6 RESULTS OF MAY 2017 GOLD VERIFICATION SAMPLING BY P&E



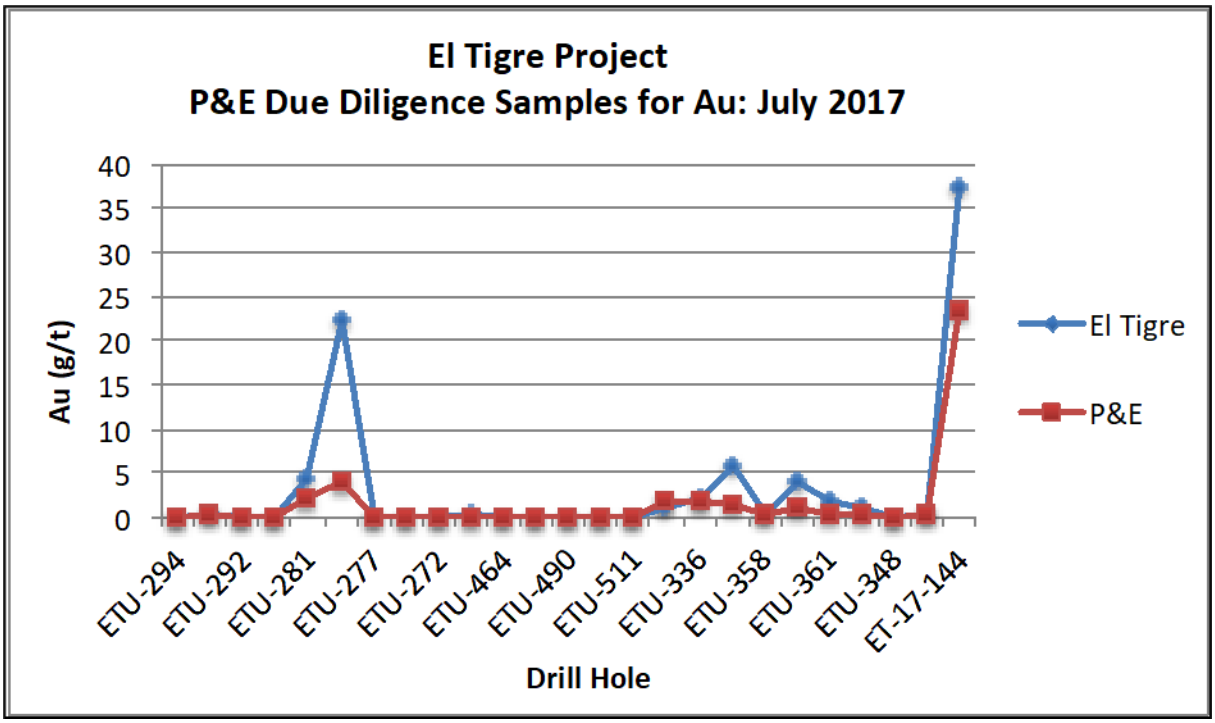
Source: *P&E (2017)*

FIGURE 12.7 RESULTS OF MAY 2017 SILVER VERIFICATION SAMPLING BY P&E



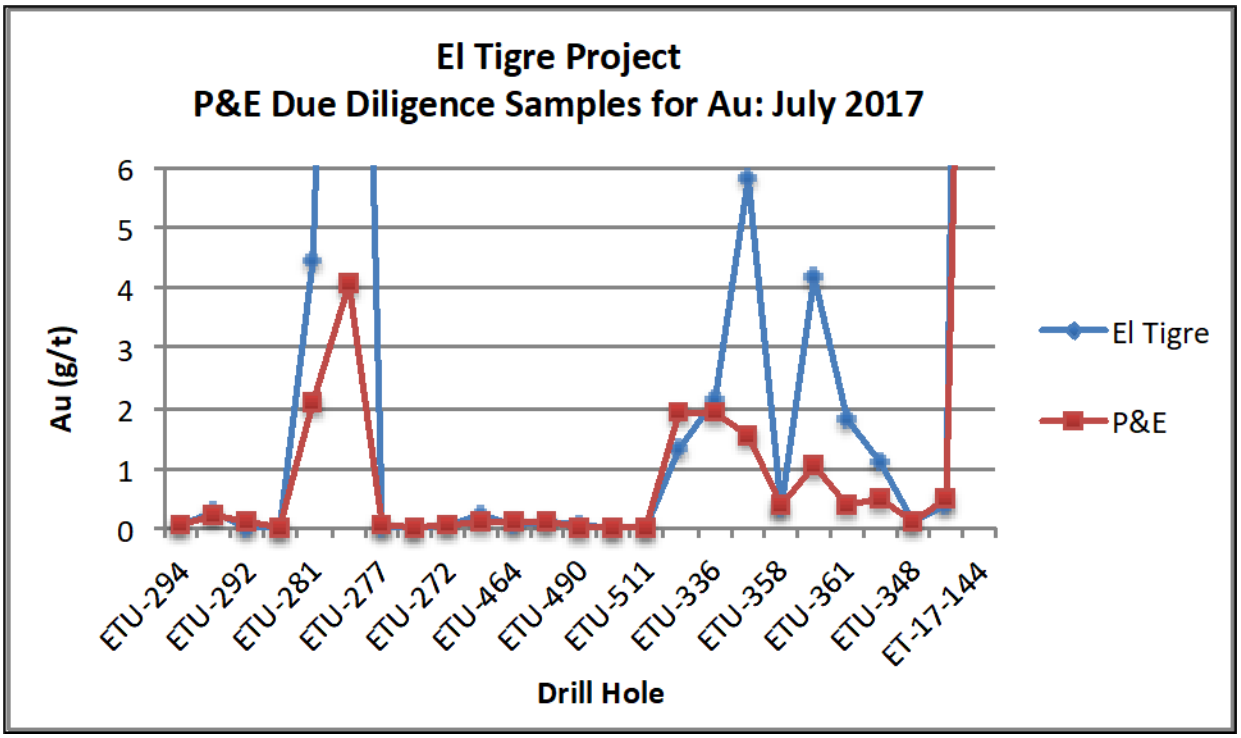
Source: *P&E (2017)*

FIGURE 12.8 RESULTS OF JULY 2017 GOLD VERIFICATION SAMPLING BY P&E



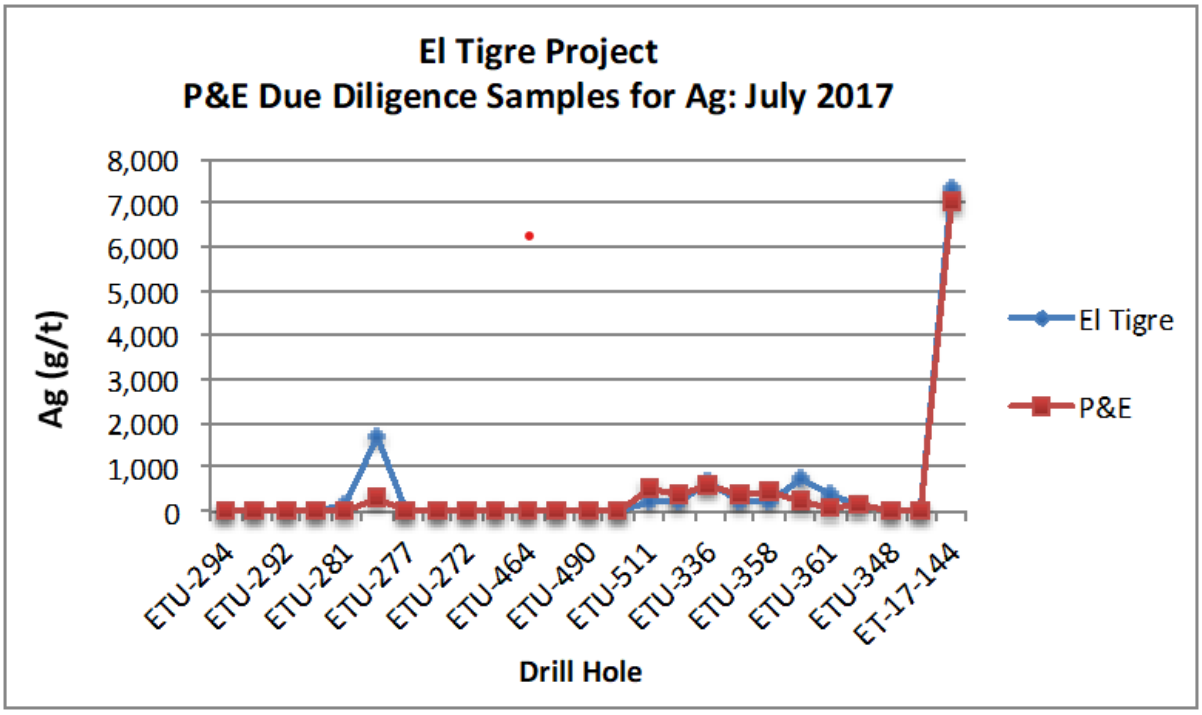
Source: *P&E (2017)*

FIGURE 12.9 RESULTS OF JULY 2017 GOLD VERIFICATION SAMPLING BY P&E (CLOSE-UP)



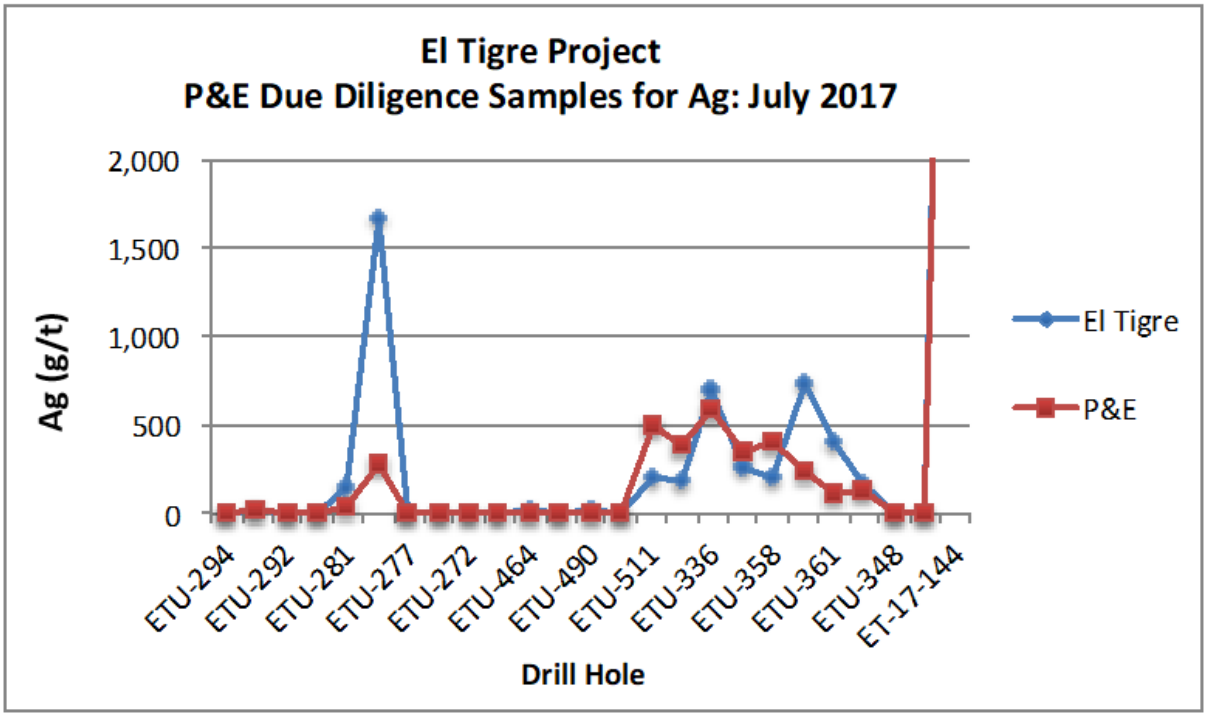
Source: *P&E (2017)*

FIGURE 12.10 RESULTS OF JULY 2017 SILVER VERIFICATION SAMPLING BY P&E



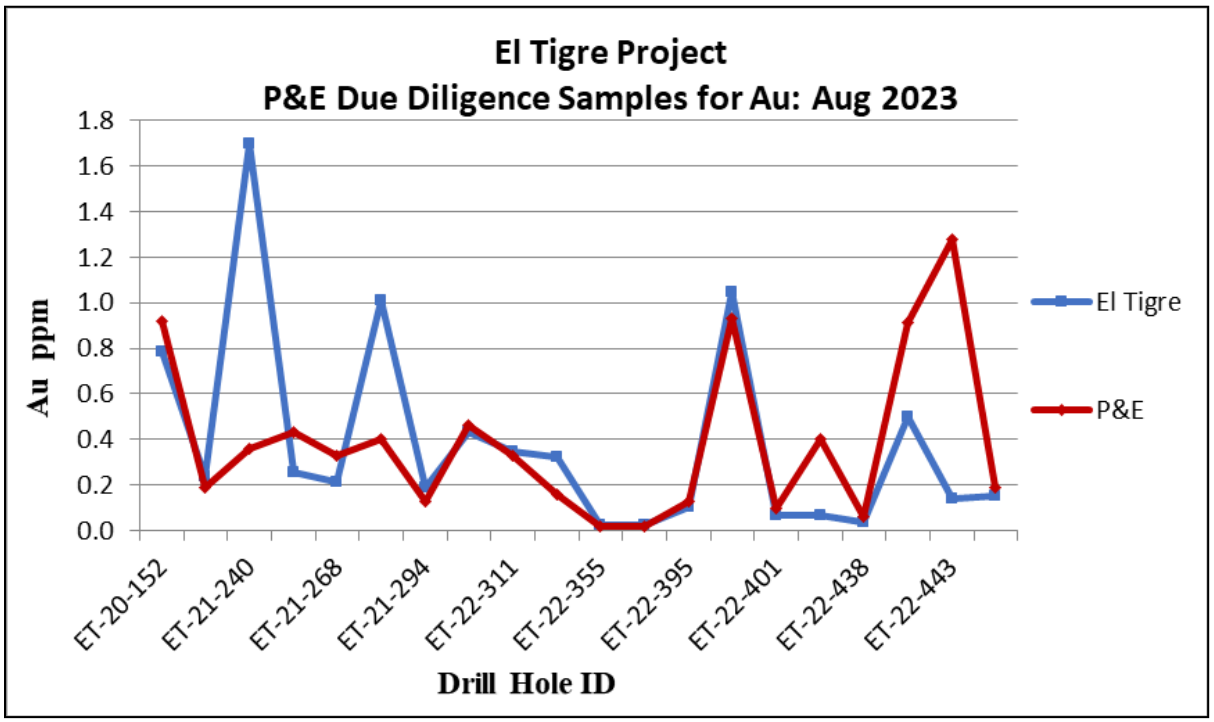
Source: *P&E (2017)*

FIGURE 12.11 RESULTS OF JULY 2017 SILVER VERIFICATION SAMPLING BY P&E (CLOSE-UP)



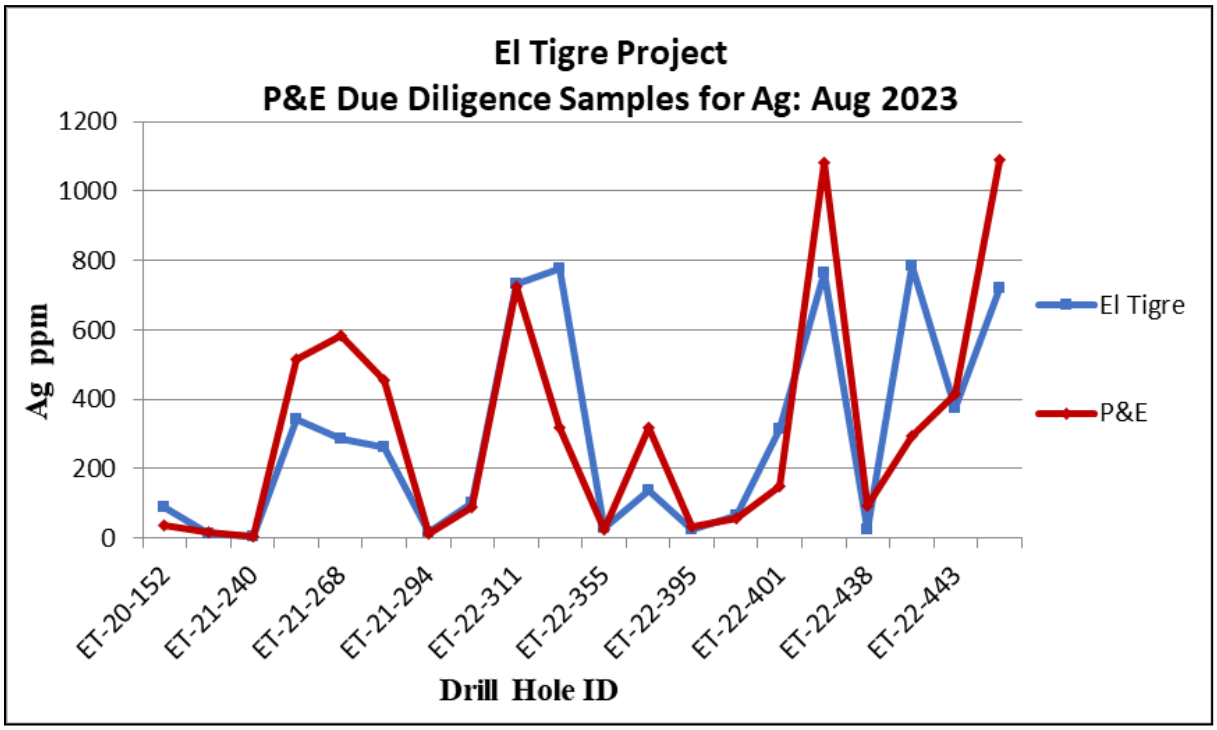
Source: *P&E (2017)*

FIGURE 12.12 RESULTS OF AUGUST 2023 GOLD VERIFICATION SAMPLING BY P&E



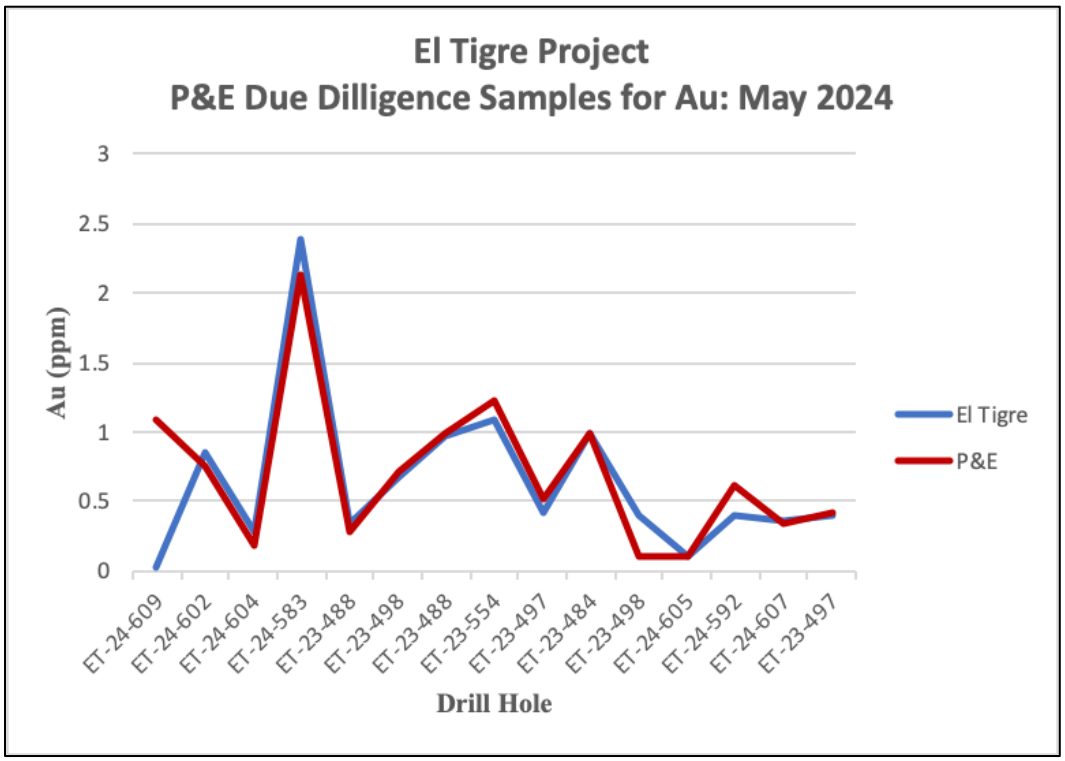
Source: *P&E (2023)*

FIGURE 12.13 RESULTS OF AUGUST 2023 SILVER VERIFICATION SAMPLING BY P&E



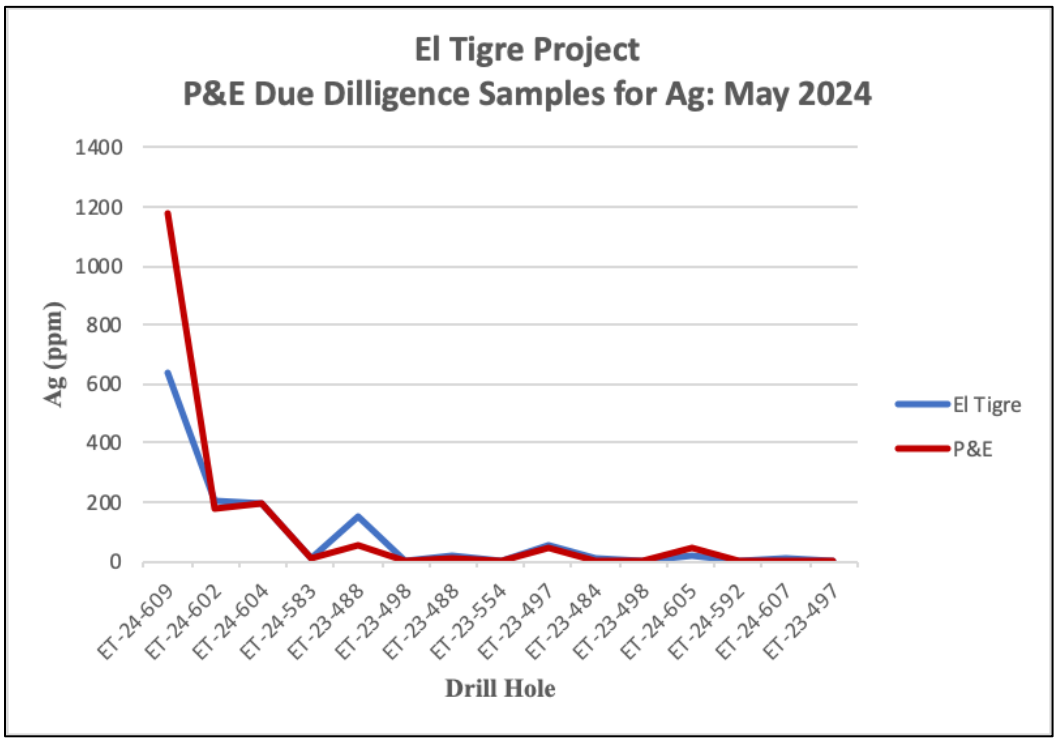
Source: *P&E (2023)*

FIGURE 12.14 RESULTS OF MAY 2024 GOLD VERIFICATION SAMPLING BY P&E



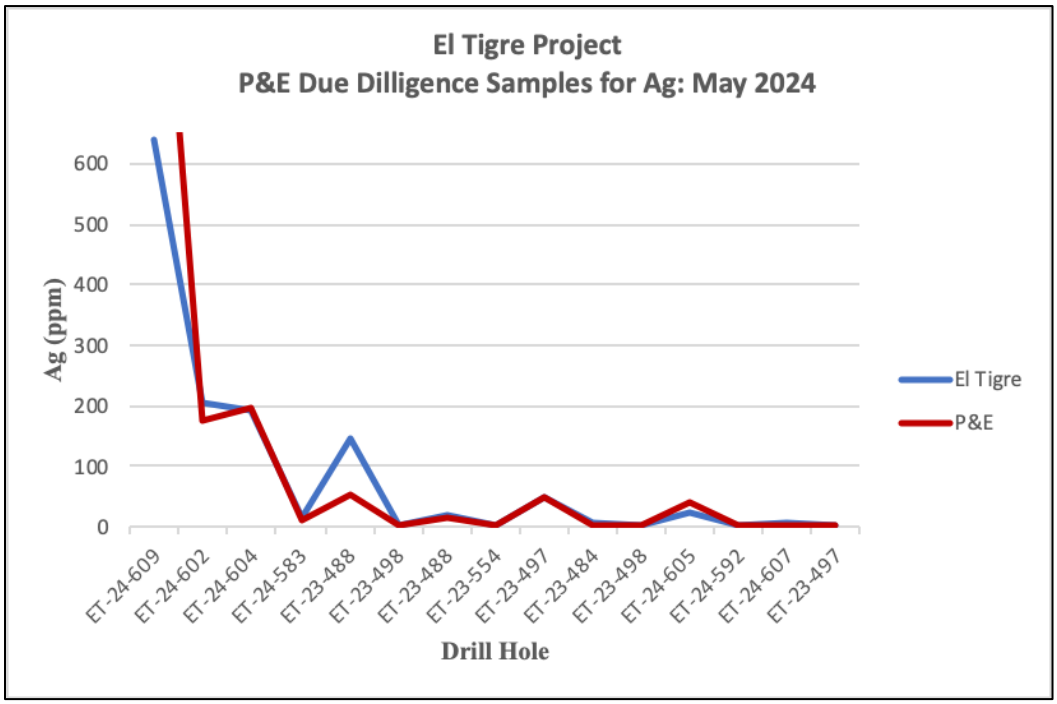
Source: *P&E (2024)*

FIGURE 12.15 RESULTS OF MAY 2024 SILVER VERIFICATION SAMPLING BY P&E



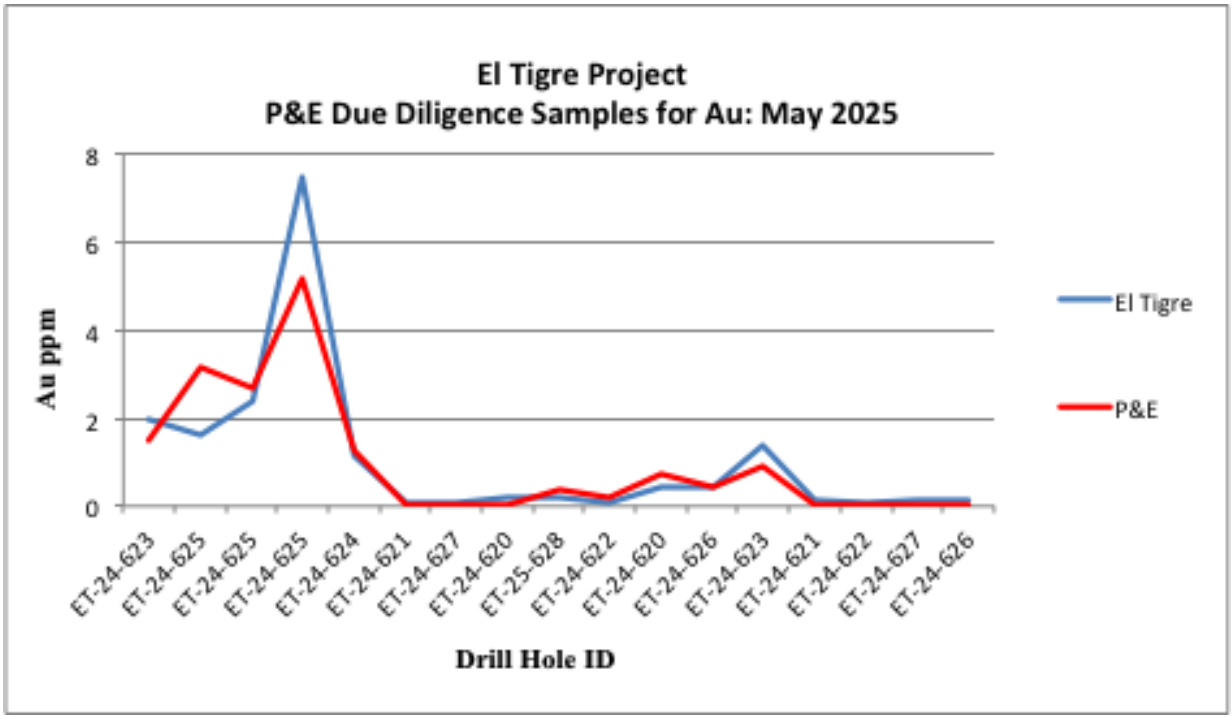
Source: *P&E (2024)*

FIGURE 12.16 RESULTS OF MAY 2024 GOLD VERIFICATION SAMPLING BY P&E (CLOSE UP)



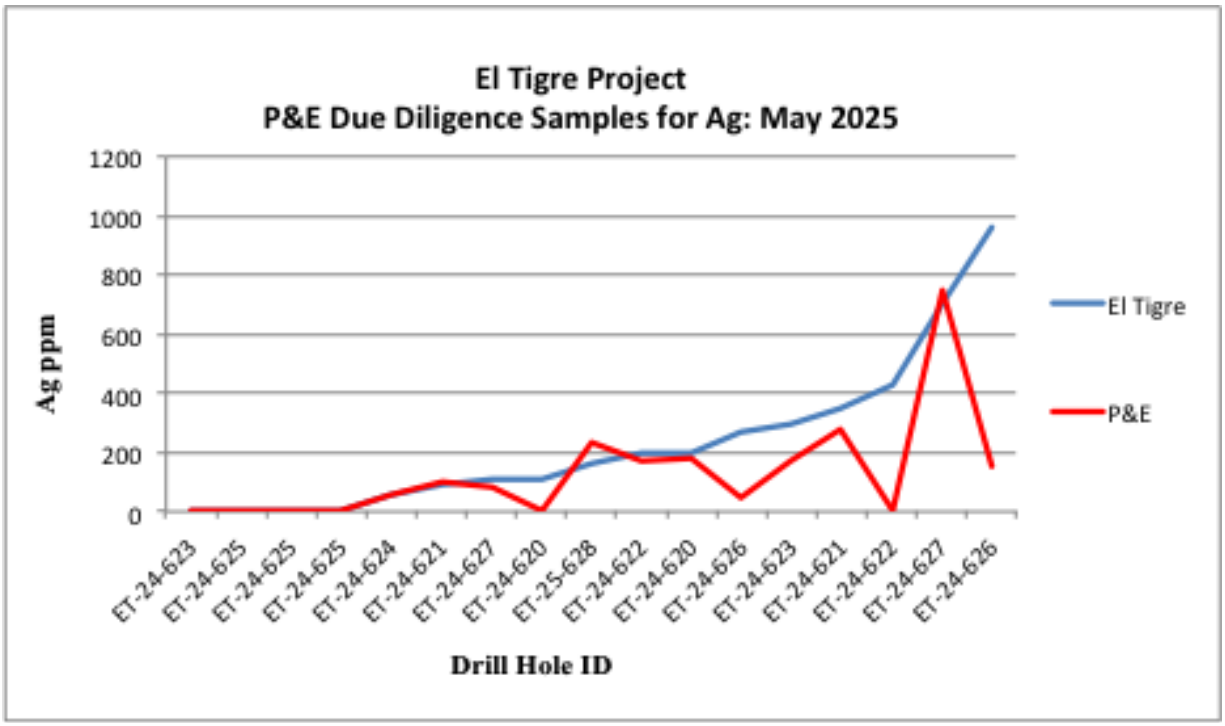
Source: *P&E (2024)*

FIGURE 12.17 RESULTS OF MAY 2025 GOLD VERIFICATION SAMPLING BY P&E



Source: *P&E (2025)*

FIGURE 12.18 RESULTS OF MAY 2025 SILVER VERIFICATION SAMPLING BY P&E



Source: *P&E (2025)*

12.5 LIMITATION ON DATA VERIFICATION

Information relating to the sample preparation, analyses, and security measures for the historical drilling carried out by Anaconda Minerals at the Property between 1982 and 1983 is limited, with laboratory assay certificates unavailable, and sparse information on sampling, security and quality control measures taken. Although the majority of the original drill core from Anaconda's drilling campaign was destroyed, 698.22 m of drill core (~9% of the original drill core) from 18 drill holes has survived, some of which has been independently verified by the Authors. The Authors have also relied on publicly available information from previous reports on the Property to further assess the reliability of Anaconda's historical data, which forms just under 6% of the current Mineral Resource Estimate drill hole assay data.

12.6 ADEQUACY OF DATA

Verification of the El Tigre Project data used for the current Mineral Resource Estimate, has been undertaken by the Authors, including verification of drilling assay data and four separate site visits and due diligence sampling of both recent drill core and historical Anaconda drill core.

The Authors consider that there is good correlation between gold and silver assay values in Silver Tiger's database and the independently collected verification samples analyzed at ALS and Actlabs. Grade variation is evident in some samples. However, the Authors consider the due diligence results to be acceptable.

The Authors have also relied on internal documentation supplied by Silver Tiger and publicly available information from previous reports on the Property to assess the reliability of Anaconda's historical data, which forms approximately 6% of the current Mineral Resource Estimate drill hole assay data. Upon assessment of El Tigre Silver's historical drill core assaying program, the twin holes and the Qualified Person's 2017 due diligence samples of Anaconda drill core, the Authors conclude that sufficient verification has been undertaken to support the validity of the historical Anaconda data.

The Authors are satisfied that sufficient verification of the drill hole data has been undertaken and that the supplied data are of satisfactory quality and suitable for use in the current Mineral Resource Estimate of the El Tigre Project.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 EL TIGRE METALLURGICAL PROGRAMS SUMMARY

Initial preliminary metallurgical testwork of the El Tigre District Deposit was completed in 2012 and summarized in a Technical Report titled "Preliminary Feasibility Study for the El Tigre Silver Project", dated August 15, 2013. The selected process included direct cyanidation followed by Merrill Crowe recovery of Au and Ag at an initial throughput of 200 tpd with future expansion to 400 tpd. The limited amount of cyanidation testwork was undertaken on three composite tailings samples representing visually distinguishable characteristics, however, testwork details are not available.

In August 2022, an initial scoping-level metallurgical testwork program was commenced at SGS Lakefield ("SGS") located in Ontario, Canada. The objectives of testwork were to develop metallurgical data to evaluate and optimize various processes for the recovery of gold and silver, including whole ore cyanide leaching, Merrill Crowe precipitation, flotation, and heap leach amenability. Mineralogical, environmental, and solid/liquid separation and rheology examinations of both fresh ore and leach tailing samples were completed to support the testing program. The subsequent results of this program were presented in the Technical Report titled "NI 43-101 Technical Report and Preliminary Economic Assessment ("PEA") for the El Tigre Project, Pilares – El Tigre Mining District, Sonora, Mexico" Effective Date: October 27, 2023, Report Date: December 14, 2023.

Metallurgical work outlined in the PEA focused on the surface mine heap leach mineralized zone to process material in two development phases. Phase I would process 7,500 tpd during the initial three years of production and Phase II would expand production in year four to process 15,000 tpd. Details of the 2022-2023 SGS program are summarized below and referenced in the report titled, "An Investigation into the El Tigre Deposit," dated October 6, 2023.

As part of this PFS, additional metallurgical testing was completed at McClelland Laboratories, Inc. ("MLI") in Sparks, Nevada on mineralization from the proposed starter pit at the El Tigre Project site. Testwork from six locations within the starter pit was undertaken in 2024 to determine variability recovery by heap leach cyanidation processing and to further analyze work completed under the 2023 PEA. The testwork results are the foundation of this PFS section and support project process design criteria and process equipment as well as capital and operating costs and financial modelling. Variability in testwork results indicate starter pit mineralization is comprised of multiple geological zones including oxide, transition, and sulphide. This resulted in determination of variable recoveries versus a global recovery assumed in the PEA.

MLI presented the results of the program in a report titled "Report on Metallurgical Testing Program - El Tigre Drill Core Composites" dated August 28, 2024. Details of the MLI metallurgical program are summarized below.

Further testwork to coincide with the MLI testing was completed at Laboratorio Tecnológico de Metalurgia ("LTM"), Hermosillo, Mexico, in a report titled "Reporte Metalurgico Lixiviacion Dinamica en Botelas Proyecto El Tigre" dated September 20, 2024. Testwork investigated the

zone variability of the proposed mined material for processing. Sample locations were determined by the Silver Tiger geological group.

13.2 MLI METALLURGY PROGRAM SUMMARY 2024

13.2.1 Sample Receipt and Preparation

A total of 310 interval samples from six drill holes were received and utilized to prepare six composites identified as 4977-001 through 4977-006. Composites 4977-001 and 4977-006 were identified as representing low-grade and high-grade mineralization, respectively. The MLI sample locations are shown in Figure 13.1 (in conjunction with the SGS PEA sample location).

Hand-selected drill core samples were removed from each composite for comminution testing and a Bond low-energy crusher work index ("CWi") test. The remaining 4977-001 composite was stage crushed to 80% -38 mm and blended by repeated coning and quartered to obtain 440 kg for a column test, 35 kg for a head screen, 8 kg for a bottle roll test and 8 kg for a static leach test.

Composites 4977-002 through 4977-005 were crushed to just passing 50 mm and blended by repeated coning and quartered to obtain 250 kg for finer crushing. Each 250 kg split and the entire composite 4977-006 were stage crushed to 80% -12.7 mm then blended and split to obtain 75 kg for a column leach test, 5 kg for a static leach test, 15 kg for an abrasion index test and 100 kg for finer crushing.

Each 100 kg split was stage crushed to 80% -9.5 mm then blended and split to obtain 75 kg for a column leach test, 1 kg for a bottle roll test and 5 kg for a static leach test. Each bottle roll test split was stage crushed to 80% -1.7 mm for a bottle roll test. Composite 4977-001 sample was blended and split to obtain 1 kg for the bottle roll test.

The abrasion ("Ai") index test splits were crushed to -19 mm and screened on a 12.7 mm sieve to generate 19 mm x 12.7 mm product.

13.2.2 Head Analysis

Conventional fire assay fusion procedures were used to determine total gold content while a four-acid digestion/AA finish geochemical assay procedure was used to determine total silver content. Cyanide soluble content was determined using a standard hot cyanide shake procedure.

Composite average head grades for gold ranged from 0.28 to 1.19 g/t Au with an average of 0.56 g/t Au. Composite average head grades for silver ranged from 7 to 13 g/t Ag with an average of 9 g/t Ag for five composites an average grade of 24 g/t Ag for the remaining composite.

Cyanide shake analysis indicates average cyanide soluble/fire assay ("CN/FA") gold ratios of 72.0 to 74.7% for three composites and 83.7 and 92.0% for two composites. CN/FA ratio averaged 56.8% for the higher-grade composite 4977-006. Average cyanide soluble silver content ranged 58.7 to 91.0% with an average of 79.3%.

Sulphur speciation analyses performed on each composite was variable and results averaged from 0.31 to 1.83%.

13.2.3 Comminution Testing

Bond crusher work index and abrasion index tests were undertaken for hardness and abrasiveness, with respect to crushing. Crusher work indices ranged from 7.86 to 9.75 kilowatt-hour per tonne ("kWh/t") of ore, classifying the material as very soft. Abrasion indices ranged from 0.0935 to 0.3047 grams, classifying the material as moderately abrasive to abrasive.

13.2.4 Static Leach and Bottle Roll Tests

Preliminary direct cyanidation tests were completed on each composite at the column leach test feed sizes ("static leach tests") of 80% -12.7 mm and 80% -9.5 mm to obtain preliminary information concerning leachability. The bottle roll tests were completed at an 80% -1.7 mm feed size to generate preliminary data concerning heap leach amenability, lime requirement data for the static and column leach tests and feed size sensitivity.

Results indicate that El Tigre composites were amenable to direct cyanidation treatment at the feed sizes evaluated. Gold recovery rates for the static leach tests were slow and extraction progressed slowly when leaching terminated. Gold recovery rates for the 1.7 mm feed size bottle roll tests were rapid, and extraction was complete in 24 to 48 hours of leaching. Longer leach cycles would improve gold recoveries from the static leach tests and decrease the indicated feed size sensitivity.

Silver recovery rates for the bottle roll tests ranged from 28.6 to 66.7% and averaged 47.6%. Static leach test silver recoveries were lower. The difference in silver recoveries observed with changing feed size would be expected to decrease somewhat with longer leach cycles. Table 13.1 shows the static leach and bottle roll tests results.

TABLE 13.1
STATIC LEACH AND BOTTLE ROLL TESTS RESULTS

Composite	Extraction		NaCN (kg/t)	CaO (kg/t)	Final pH
	Au (%)	Ag (%)			
Bottle Roll Test 80% -1.7 mm (72-hour leach time)					
4977-001	70.0	50.0	0.04	1.10	10.9
4977-002	65.2	66.7	0.15	1.10	10.7
4977-003	54.8	42.1	0.20	1.30	11.1
4977-004	86.8	42.9	0.14	1.70	10.9
4977-005	81.4	28.6	0.19	1.30	10.7
4977-006	49.4	55.6	0.23	1.50	11.0
Static Leach Test 80% -38 mm (14-day leach time)					
4977-001	50.0	11.1	0.35	1.10	9.9

TABLE 13.1 STATIC LEACH AND BOTTLE ROLL TESTS RESULTS					
Composite	Extraction		NaCN (kg/t)	CaO (kg/t)	Final pH
	Au (%)	Ag (%)			
Static Leach Test 80% -12.7 mm (14-day leach time)					
4977-002	66.1	28.6	0.59	1.10	9.4
4977-003	44.4	30.0	0.67	1.30	9.4
4977-004	77.8	28.6	0.66	1.70	9.6
4977-005	74.2	14.3	0.67	1.30	9.2
4977-006	12.0	11.8	0.78	1.50	9.2
Static Leach Test 80% -9.5 mm (14-day leach time)					
4977-002	66.0	25.0	0.72	1.10	9.6
4977-003	48.3	33.3	0.69	1.30	9.4
4977-004	80.6	37.5	0.58	1.70	9.3
4977-005	72.7	14.3	0.78	1.30	9.2
4977-006	41.8	17.4	0.71	1.50	9.5

Source: *MLI (2024)*

Cyanide consumptions were low and ranged from 0.04 to 0.23 kg NaCN/t for the 1.7 mm feeds and were higher for the static leach tests due to the longer test duration and the frequent change-out of the leach solution.

Lime requirements were low and ranged from 1.1 to 1.7 kg/t for the 1.7 mm feeds. The same lime additions were used for the static leach tests.

13.2.5 Column Leach Tests

Two column tests were completed on five composites at 80% -12.7 mm and 80% -9.5 mm feed sizes and on one composite at 80% -38 mm feed size (low grade coarse). Results showed that the El Tigre Drill Core composites generally were amenable to simulated heap leach cyanidation treatment at both the 80% -12.7 mm and 80% -9.5 mm feed sizes.

Gold recoveries obtained from four composites at the 80% -12.7 mm feed size, were 68.0, 50.0, 86.2, and 75.7% in 64 to 114 days of leaching and rinsing. Respective gold recoveries obtained at the 80% -9.5 mm feed size were 67.3, 50.0, 89.1, and 82.4% in 64 to 113 days of leaching and rinsing. The difference in gold recovery at the two feed sizes were relatively low.

Gold recovery rate from one composite at 80% -38 mm feed size was slow and extraction progressed slowly when leaching terminated. A longer leach cycle would incrementally increase gold recovery from that feed.

Silver recovery rates were slow and would increase incrementally with longer leaching cycles. Table 13.2 lists the column leach tests results.

TABLE 13.2 COLUMN LEACH TESTS RESULTS					
Composite	**Extraction**		**NaCN (kg/t)**	**CaO (kg/t)**	**Final pH**
	Au (%)	**Ag (%)**			
Column Leach Test 80% -38 mm (114-day leach time)					
4977-001	58.1	25.0	0.35	1.10	9.6
Column Leach Test 80% -12.7 mm (64 to 114-day leach time)					
4977-002	68.0	42.9	0.47	1.10	10.0
4977-003	50.0	58.3	0.33	1.30	10.3
4977-004	86.2	41.7	0.73	1.70	10.1
4977-005	75.7	28.6	0.38	1.30	10.4
4977-006	31.8	34.8	0.64	1.50	9.9
Column Leach Test 80% -9.5 mm (14-day leach time)					
4977-002	67.3	40.0	0.67	1.10	9.6
4977-003	50.0	50.0	0.34	1.30	10.2
4977-004	89.1	40.0	0.82	1.70	10.0
4977-005	82.4	25.0	0.35	1.30	10.3
4977-006	47.9	40.9	0.82	1.50	9.8

Source: *MLI (2024)*

Gold recoveries for the higher-grade composite 4977-006 were more variable at the 80% -12.7 mm and 80% -9.5 mm feed sizes. The gold occurrence was "spotty", and the material was not as amenable to simulated heap leaching treatment as were the other composites. Significant grade and recovery variability was also observed during static leach and bottle roll testing of this composite.

Column test cyanide consumptions were low with consumption at 0.35 kg NaCN/t for the 38 mm feed and at 0.33 to 0.73 kg NaCN/t for the 12.7 mm feed with an averaged of 0.51 kg NaCN/t. Consumptions for the 9.5 mm feeds ranged from 0.34 to 0.82 kg NaCN/t and averaged 0.60 kg NaCN/t.

The hydrated lime added before leaching was generally sufficient for maintaining protective alkalinity during leaching.

Domain characterization of the six composites and related drill holes are presented in Table 13.3 drill hole composites and related domain characterization.

TABLE 13.3 DRILL HOLE COMPOSITES AND DOMAIN CHARACTERIZATION		
Composite	**Drill Hole ID**	**Domain Characterization**
4977-001	ET-23-475	Oxide
4977-002	ET-23-476	Transition – Sulphide with minor oxide
4977-003	ET-23-477	Transition – Sulphide with minor oxide
4977-004	ET-23-478	Oxide
4977-005	ET-23-480	Oxide
4977-006	ET-23-481	Tiger Vein – Sulphide

Source: *D.E.N.M. (2024)*

13.2.6 Environmental Characterization

Environmental characterization testing was completed on each of the column leach residues.

Nevada Modified Acid Base Accounting ("NV-ABA") and Nevada Modified Net Acid Generating ("NV-NAG") tests were completed on all the samples to assess potential of the solids to generate or neutralize acid in a natural weathering and oxidizing environment.

The ABA results displayed a greater potential to generate acid than to neutralize acid in a natural weathering and oxidizing environment. The results are summarized below:

- Paste pH: 8.29 to 9.43.
- Pyritic sulphur: 0.29 to 1.83 wt %.
  - AGP: 9.1 to 59.1 t $CaCO_3$ per 1,000 t of solids.
- ANP: 1.1 to 6.1 t $CaCO_3$/1,000 t of solids.
- NNP: -52.9 to -5.4 t $CaCO_3$/1,000 t of solids.
- Ratio (ANP/AGP) <0.3 to 0.4.

The NAG results displayed a greater potential to generate acid than to neutralize acid in a natural weathering and oxidizing environment. The results are summarized below:

- NAG values ranged from 14.9 to 50.9 kg H_2SO_4/t.

13.2.7 MLI Report Conclusions

Bottle roll test gold recovery results were well correlated with expected heap leach recoveries and metallurgical testwork overall confirmed that El Tigre mineralization was generally amenable to simulated heap leach cyanidation treatment at feed sizes of 80% -12.7 mm and 80% -9.5 mm.

While the higher-grade mineralization identified in composite 4977-006 was shown not as amenable to simulated heap leaching treatment as the other composites, lower-grade mineralization identified in composite 4977-001 was showed to be amenable to simulated heap leach cyanidation treatment at an 80% -38 mm feed size.

Heap leaching reagent consumptions are expected to be low. Good ore permeability assumes that agglomeration pre-treatment should not be required for heap leaching of the mineralization tested.

Although the composites were generally not particularly sensitive to feed size, with respect to gold recovery, testwork showed that there was a general tendency for gold recovery to decrease with increasing pyritic sulphur grade. The ore contains significant quantities of potentially acid generating pyritic sulphur. Adequate base addition to ensure long-term pH control will be important for successful heap leaching.

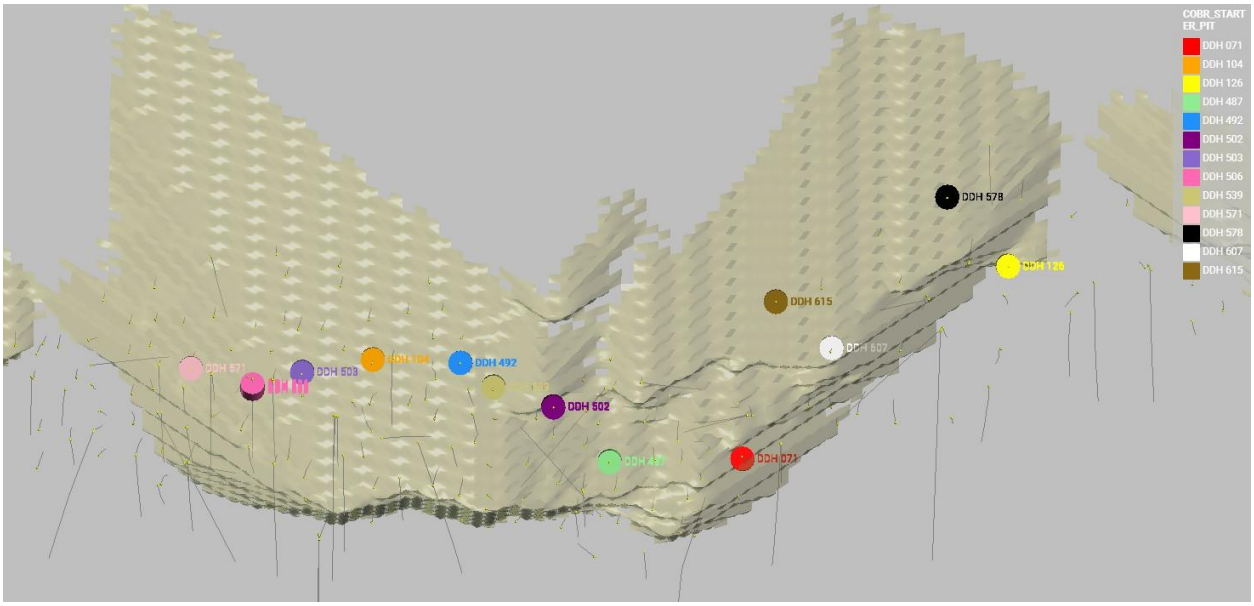
13.3 LTM METALLURGICAL PROGRAM SUMMARY 2024

13.3.1 Sample Receipt and Preparation

A total of 140 drill core interval samples were received and utilized to prepare 13 composites for head analysis and bottle leach testing. The composites were crushed to 100% -9.5 mm size and 1.0 kg ore charge was used for both tests.

The sample locations within the starter pit are shown in Figure 13.1.

FIGURE 13.1 MLI SAMPLE LOCATIONS



Source: *Pacemaker (2024)*

13.3.2 Head Analysis

Testwork was completed to determine gold, silver and copper content in the 13 composites. Fire assay and atomic absorption procedures were used to determine gold content while fire assay gravimetry, four-acid digestion and ICP-OES analysis were used to determine silver content. For copper content, four-acid digestion and ICP-OES analysis were used. Table 13.4 summarizes the results of the head analysis testwork.

TABLE 13.4 HEAD ANALYSIS TESTWORK RESULTS			
Composite	**Au (g/t)**	**Ag (g/t)**	**Cu (g/t)**
ET-16-104	0.300	18.073	46.087
ET-17-126	0.026	20.971	9.216
ET-23-492	0.547	25.217	25.026
ET-23-487	0.387	12.778	23.834
ET-23-503	1.539	93.250	25.347
ET-23-506	0.552	54.563	17.092
ET-23-502	0.548	25.250	30.423
ET-23-539	0.346	36.000	30.853
ET-24-571	0.563	52.444	4.667
ET-24-578	0.600	72.762	13.559
ET-24-607	0.621	20.141	18.267
ET-24-615	1.681	92.225	53.574
ET-13-071	0.617	19.985	17.946

Source: *LTM (2024)*

Tests of gold, silver and copper soluble in hot cyanide were also completed. Only one composite returned a soluble gold grade below levels of detection.

13.3.3 Bottle Leach Tests

Testwork was performed to determine gold and silver extraction as well as consumption of consumption of NaCN and CaO. Leach time was 72 hours with sample testing done periodically for silver and gold analysis. At the end of the leaching cycle, the loaded solution was measured and sampled while the leaching residues were analyzed by fire assay and atomic absorption procedures for gold and fire assay gravimetry, four-acid digestion and ICP-OES analysis for silver. Table 13.5 summarizes the results of the bottle leach testwork.

TABLE 13.5 BOTTLE LEACH TESTS				
Composite	**Extraction**		**NaCN (kg/t)**	**CaO (kg/t)**
	Au (%)	**Ag (%)**		
Bottle Leach Tests 100% - 9.5 mm (72-hour leach time)				
PB1	97.96	23.17	1.159	3.690
PB2	57.25	8.10	1.045	3.492
PB3	89.05	96.71	1.870	5.786
PB4	84.42	29.14	1.181	2.775

TABLE 13.5 BOTTLE LEACH TESTS				
Composite	Extraction		NaCN (kg/t)	CaO (kg/t)
	Au (%)	Ag (%)		
PB5	41.86	48.71	1.487	3.678
PB6	60.43	63.08	1.554	4.437
PB7	68.45	93.31	1.728	4.662
PB8	59.87	62.96	1.463	5.636
PB9	95.48	49.34	1.371	5.800
PB10	11.31	5.32	1.020	2.540
PB11	85.05	56.90	1.197	3.363
PB12	49.82	8.79	1.032	3.082
PB13	84.12	37.76	1.427	3.319

Source: *LTM (2024)*

Six of the composites reached extraction rates above 80% for gold while two composites reached extraction rates above 85% for silver. Testwork did demonstrate that reagent intakes of NaCN and CaO were within typical normal range.

13.3.4 LTM Report Conclusions

Testwork on almost half the composites demonstrated extraction rates above 80% for gold and above 85% for silver on two composites. The varied indicate a possible Preg-Robbing effect that can inhibits gold recovery due to potential presence of clays and carbonaceous materials. Testwork also identified the presence of arsenic (As), antimony (Sb) and lead (Pb) in the composites.

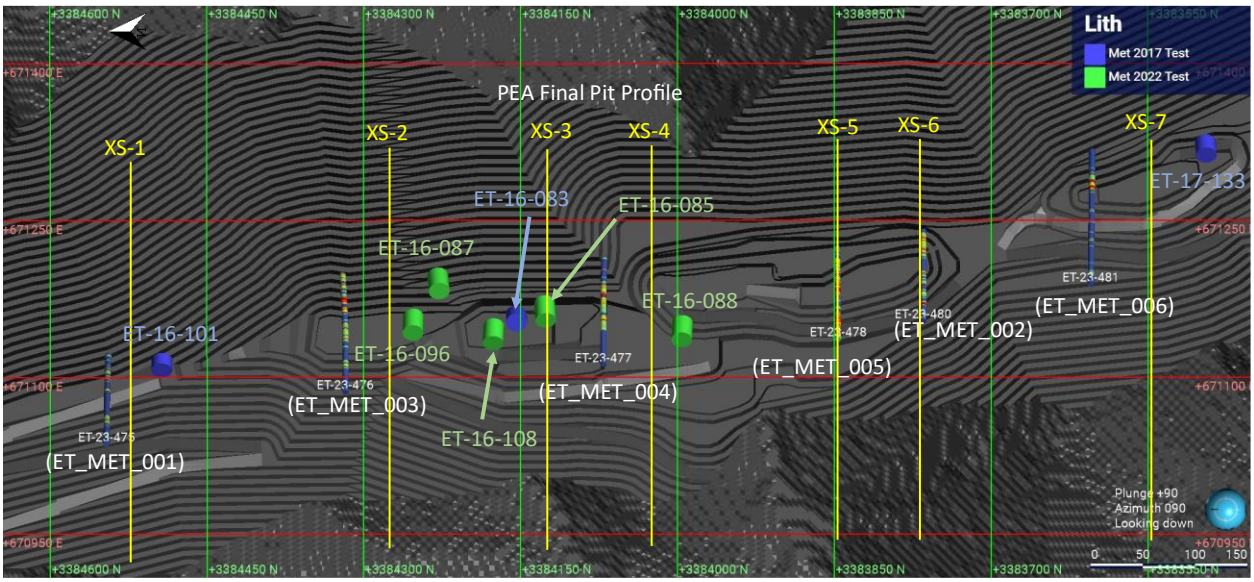
13.4 SGS METALLURGY PROGRAM SUMMARY 2022 – 2023

13.4.1 Sample Receipt and Preparation

One shipment of Heap Leach ore samples was received and utilized to form a Heap Leach Composite and five heap leach variability composites. 110 interval samples from five drill holes were crushed to 0.5 inch and blended. From the blended material for each hole, a 14 kg and a 1 kg charge were riffled out. The 14 kg charge material from the five-hole samples was combined and blended to form 70 kg of Heap Leach Composite.

The sample locations are illustrated in Figure 13.2 below identifying five drill hole ET-16-085, 087, 088, 096, and 108.

FIGURE 13.2 HEAP LEACH SAMPLE LOCATIONS (SGS – PEA AND MLI – PFS)



Source: *Silver Tiger (2024)*

From the 70 kg material, 5 x 10 kg and a single, 6 kg, 2 kg, 1 kg and 500 g charge were riffled out. The 6 kg charge of ½ inch material was crushed to 3/8 inch, blended and a 2 kg and a 500 g charge were riffled out. Finally, the remaining ~3.5 kg was then crushed to ¼ inch, blended and a 2 kg and 500 g charge were riffled out.

13.4.2 Sample Characterization

Head sample analyses were completed on the composites and variability samples to determine the precious metal concentration as well as the content of other elements. Assays included gold and silver in duplicate by fire assay, sulphur speciation for total sulphur (S_T) and sulphide sulphur (S=) by Leco, copper by atomic absorption, and a semi-quantitative Inductively Coupled Plasma ("ICP") scan analysis.

13.4.2.1 Gold and Silver Analysis

Duplicate sample cuts of 30 g of ~75 µm pulverized material for the samples were submitted for both gold and silver by fire assay. Heap leach samples returned gold head grades ranging from 0.38 to 0.75 g/t Au with composite being 0.61 g/t Au. Silver head grades ranged from 3.1 to 28.6 g/t Ag with the composite being 20.1 g/t Ag.

13.4.2.2 Sulphur Speciation

Representative pulverized (<75 µm) subsamples were submitted for sulphur speciation including total sulphur (S_T), and sulphide sulphur ($S^=$) by Leco analysis.

The analysis determined that the total sulphur for the heap leach samples ranged from 0.46 to 1.64% with the composite being 0.95% while the sulphide sulphur grades ranged from 0.34 to 1.40% with the composite being 0.87%.

13.4.2.3 Individual Copper Analysis

Representative pulverized (<75 µm) subsamples were submitted for individual copper analysis by atomic absorption. The analysis determined that the copper for the for the heap leach samples ranged from 0.001 to 0.004% Cu, with the composite being 0.003% Cu.

13.4.3 Metallurgical Testing

Metallurgical testwork included whole ore cyanidation, bulk leach cyanidation, Merrill Crowe, rougher flotation, cyanidation of flotation products, selective flotation, coarse ore bottle roll ("COBR") cyanidation, percolation, column heap leaching and environmental evaluation of feed and leach tailing samples.

13.4.3.1 Heap Leach Composite Whole Ore Cyanidation

A single whole ore bottle roll cyanidation test was completed on the Heap Leach Composite to examine/evaluate potential gold and silver extractions and to compare the results to the heap leach amenability testing.

From the test on the Heap Leach Composite, the gold and silver extractions were 89 and 92% respectively. The test produced tailing grades of 0.08 g/t Au and 2 g/t Ag. Copper extraction was 29.6% Cu, giving a tailing grade of less than the detection limit of 0.01%. The test resulted in NaCN addition and consumption of 11.8 and 4.27 kg/t, respectively, as well as lime addition and consumption of 0.13 and 0.05 kg/t, respectively.

13.4.3.2 Coarse Bottle Roll (COBR) Testing

COBR cyanidation heap leach amenability testing was completed on 2 kg charges of the Heap Leach Composite at crush sizes of ¼ inch (6.25 mm), 3/8 inch (9.50 mm) and ½ inch (12.5 mm).

The results from the testing on the Heap Leach Composite at day 21 showed gold extractions of 76% for the ¼ inch material testing, 73% for the 3/8-inch material testing, and 85% for the ½ inch material testing. Silver extractions over the same period were 56, 53 and 58%, respectively. The reagent additions of sodium cyanide and lime ranged from 3.77 to 3.83 kg/t and 2.22 to 2.71 kg/t, respectively, while the reagent consumptions of sodium cyanide and lime ranged from 3.10 to 3.16 kg/t and 1.90 to 2.48 kg/t, respectively.

13.4.3.3 Percolation Testing

The percolation ability of the Heap Leach Composite was examined to determine if agglomeration would be required for the heap column leaching. Testing was performed on 500 g charges at the crush sizes of ¼ inch, 3/8 inch, and ½ inch in percolation columns. Water flowed through the column at various solution rates and into a collection vessel over a period of several days.

Flowrate for heap leaching is typically 10 L/h/m^2 and during percolation testing, flowrates were doubled every two days, ranging from 10 to 40 L/h/m^2 and based on flowrates calculated for the 6-inch diameter column. Initial flowrate was 10 L/h/m^2 for all tests and after two days the flowrate was increased to 20 L/h/m^2, and after another two days the flowrate was increased to 40 L/h/m^2. At four times the typical heap flowrate, the percolation columns experienced no flow issues indicating any of the three crush sizes could be selected for heap column leach testing with no agglomeration required.

13.4.3.4 Heap Leach Column Leaching

Utilizing results from the COBR and percolation testing, crush sizes of 3/8 and 1/2 inch were selected for column leach testing to evaluate gold and silver extraction as a function of leach time. After 70 days of cyanide leaching and three days of washing, total gold and silver extractions for the 3/8-inch material test, was 83 and 64%, respectively. After 77 days of cyanide leaching and seven days of washing, total gold and silver extractions for the ½ inch material test was 78 and 47%, respectively.

The sodium cyanide and lime additions went from 1.35 to 1.47 kg/t and 2.33 to 2.73 kg/t, respectively, while the sodium cyanide and lime consumptions went from 0.68 to 0.81 kg/t and 2.33 to 2.73 kg/t, respectively, when comparing 3/8-inch crush size to ½ inch crush size.

From the shape of the extraction curves, it is evident that gold extraction in both tests slows significantly after approximately one week for both tests, silver extraction slowed significantly after approximately five weeks for 3/8-inch crush size and four weeks for ½ inch crush size.

13.4.4 Environmental Testing

13.4.4.1 Modified Acid Base Accounting

Modified acid-base accounting ("ABA") testing was completed on the cyanide leach tailing samples for the Heap Leach composites to help evaluate tailings/waste rock propensity to generate acidic conditions and provide input parameters for kinetics testwork.

ABA testwork also provides quantification of the total sulphur, sulphide sulphur, and sulphate concentrations present, and the potential acid generation ("AP") related to the oxidation of the sulphide sulphur concentration. Carbonate concentrations were also be analyzed, and carbonate neutralization potential ("NP") values were determined. The balance between the AP and NP assists in defining the potential of the sample to generate acid drainage.

Results for the samples showed the solids are acid generating for the heap leach composite and have no net acid consumption potential as evidenced by the AP/NP ratio's of less than one and ranging from 0.07 to 0.57. The net neutralization potential ("Net NP") results ranged from -4.74 to -25.39t $CaCO_3$ /1,000 tonnes of material, respectively, for the three acid producing samples.

13.4.4.2 Net Acid Generation Testing

Net acid generation ("NAG") testing was completed on the cyanide leach tailing samples for the Heap Leach composites to determine the balance between the acid producing and acid consuming components of the samples. NAG results confirm acid rock drainage characteristics of each sample based on oxidation of the samples sulphide content (as well as ferrous iron from siderite dissolution).

NAG testing uses hydrogen peroxide to react with the sample's sulphides and the produced acid is consumed by carbonates and/or other acid consuming components of the material. The pH of the solution is measured (NAG pH), and the remaining acid is titrated with standardized NaOH to determine the net acid generated by the reaction.

Results for the four leach tailing samples shows the net acid production potential (NAG) at pH 4.5 is 0.00 kg H_2SO_4/t for the Tailing Composite sample and ranges from 2.8 to 19 kg H_2SO_4/t for the remaining three samples. At NAG pH 7, the net acid production potential ranges from 0.6 to 19 kg H_2SO_4/t for all four samples.

Results indicate that on complete oxidation of the samples, acid is detected as the net result and thus is acid generating and has no net acid consumption potential.

13.4.5 SGS Conclusions and Recommendations – 2022-2023

The concentrations in the Heap Leach Composite as well as the five-hole variability samples ranged from 0.38 to 0.75 g/t Au and 3 to 29 g/t Ag. Heap leach amenability testing was selected as the main process for examination for these samples.

COBR cyanidation testing performed on the Heap Leach Composite sample at various crush sizes produced high gold and moderate silver respective extractions ranging from 75 to 85% and 53 to 58%, confirming this sample would likely be amenable to the heap leach process.

Percolation testing completed on the Heap Leach Composite indicated no agglomeration is required for heap column leach testing, regardless of the crush size selected.

Heap column leach testing performed on the Heap Leach Composite determined the sample is amenable to the heap leach process, producing respective gold and silver extractions of 83 and 64%, respectively, at 3/8 inch crush size and 78 and 47%, at ½ inch crush size after 10 to 11 weeks of leaching. Gold leach kinetics were very fast during the first one to two weeks and then slowed severely, while the silver leach kinetics were more gradual over time.

13.4.6 SGS Testing Addendum – 2024

Due to the variability in the leaching results from the MLI metallurgical program, SGS was engaged to review the MLI sampling and preparation procedures including logging and observations. SGS was also engaged to redo the coarse bottle roll testwork on five composites that SGS completed during its 2023 metallurgical program.

To determine heap leach amenability of the composites for gold and silver cyanide leaching, a 3/8 inch crush size feed was transferred to a bottle and pulped with water to the proper percent solids. Testing parameters included a pulp density of 45%, a pH target at 10.5, a NaCN concentration of 1.0 g/L and a leaching time of 21 days.

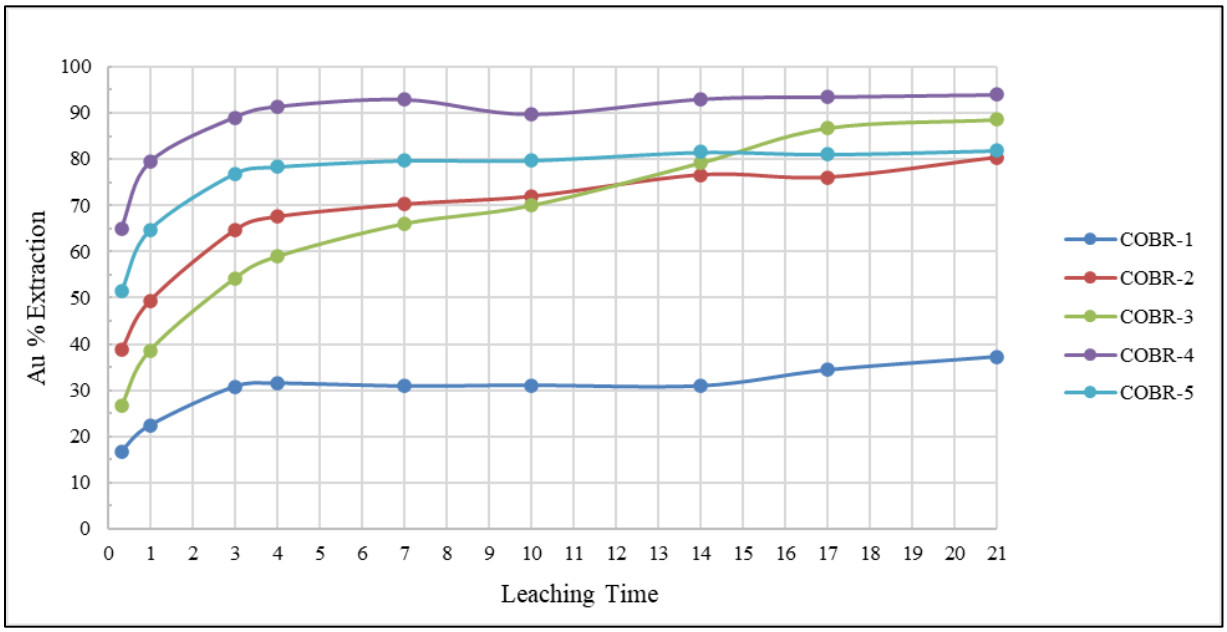
Upon completion of the leaching cycle, the pulp was filtered, and the residue was washed. The final products were submitted for the required assays. Testwork results are summarized in Table 13.6, Figure 13.3 and Figure 13.4.

TABLE 13.6
COARSE BOTTLE ROLL TEST RESULTS

Composite	Extraction		NaCN (kg/t)	CaO (kg/t)
	Au (%)	Ag (%)		
Coarse Bottle Roll Cyanidations 3/8 inch (21-day leach time)				
COBR-1	37	38	3.07	1.41
COBR-2	80	44	3.51	1.64
COBR-3	89	41	3.03	1.21
COBR-4	94	81	3.39	1.92
COBR-5	82	58	3.08	1.45

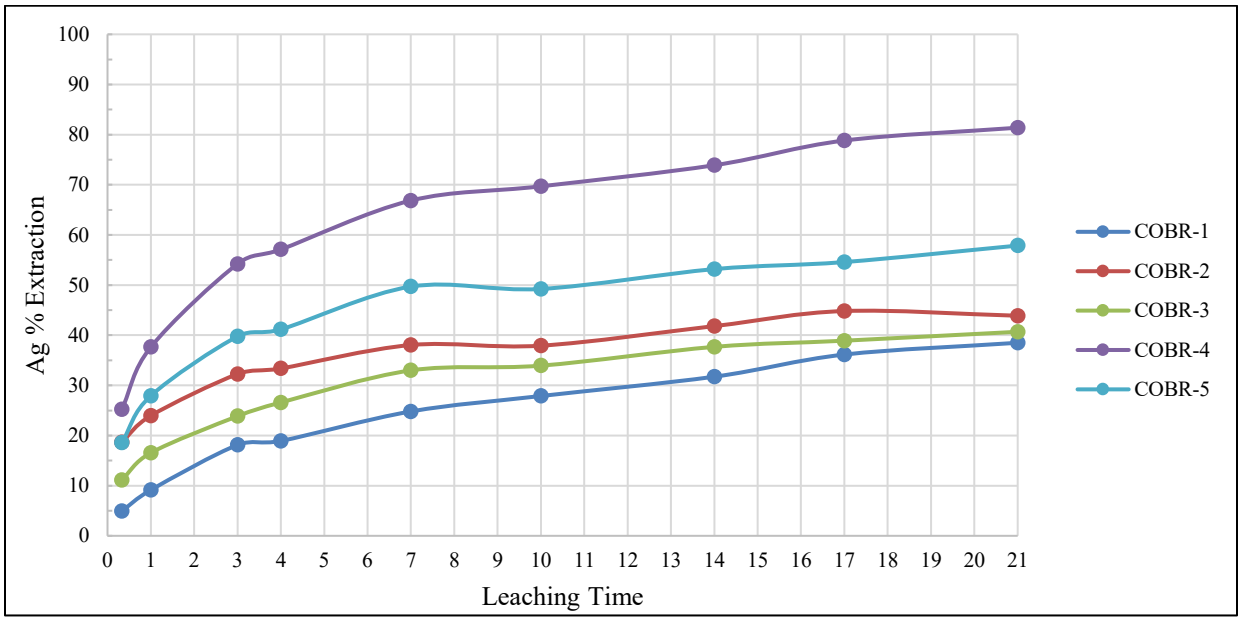
Source: *SGS (2024)*

FIGURE 13.3 GOLD EXTRACTION PLOT (3/8 INCH)



Source: *SGS (2024)*

FIGURE 13.4 SILVER EXTRACTION PLOT (3/8 INCH)



Source: *SGS (2024)*

13.4.6.1 Conclusions SGS Testing Addendum – 2024

These testwork results further enforces the presence of transition and sulphide material in the starter pit and this will need to be addressed in the subsequent mine plan.

13.5 SUMMARY

Table 13.7 summaries metallurgical testwork and Process Design Criteria for the current PFS.

TABLE 13.7
PROCESS DESIGN CRITERIA

Criteria	Units	Value
Ore Characteristics		
Specific Gravity	g/cm³	2.6
Plant Availability/Utilization		
Overall Process Plant Feed-Nominal – Years 1-3	tpy	2,737,500
Overall Process Plant Feed- Nominal – Years 1-3	tpd	7,500
Overall Process Plant Feed-Nominal – Years 4 +	tpy	5,475,000
Overall Process Plant Feed- Nominal – Years 4 +	tpd	15,000
Crushing Plant Availability	%	75
Heap and Processing Plant Availability	%	95
Crushing Product (to pad)	P_{80} inch (mm)	3/8 (9.0)
Process Plant Production		
Process Plant Feed Characteristics (Life of Mine Average)		

TABLE 13.7 PROCESS DESIGN CRITERIA		
Criteria	**Units**	**Value**
Gold Head Grade	g/t	0.40
Silver Head Grade	g/t	14.9
Metal Recoveries and Process Details		
Anticipated Overall Gold Oxide Recovery- design[1]	%	83
Anticipated Overall Silver Oxide Recovery- design[1]	%	45
Anticipated Overall Gold Sulphide Recovery – design[1]	%	56.0
Anticipated Overall Silver Sulphide Recovery – design[1]	%	40.0
Cyanide Addition Rates – design[2]	kg/t	0.25
Lime Addition Rates – design[2]	kg/t	1.5
Merrill Crowe Phase I – design	m^3/h	590
Merrill Crowe Phase II – design	m^3/h	1,180
Merrill Crowe Recovery – Au and Ag	%	99

Source: *D.E.N.M. (2024)*

Note [1] *- Column testing indicated higher oxide gold from the PEA recovery at a 3/8 in. crush size. In the process design and financial model, these have been discounted by 3% for leaching in the field versus optimum conditions in the laboratory. As noted, recent column testing has indicated sulphide zones are present (15% of the total) that will affect the overall recovery. This variation should be noted in the mine schedule and financial modelling.*

[2] *- Cyanide Consumption is also discounted for the process design, operating costs, and financial model. Lime is not discounted from the recent column test data.*

14.0 MINERAL RESOURCE ESTIMATE

The Mineral Resource Estimate presented herein for the El Tigre Project has been prepared following the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1, and in conformity with generally accepted "CIM Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines. Mineral Resources have been classified in accordance with the "CIM Standards on Mineral Resources and Reserves: Definition (2014) and Best Practices (2019)" as adopted by CIM Council.

> *A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.*
>
> *An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.*
>
> *An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.*

Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no guarantee that all or any part of the Mineral Resource will be converted into a Mineral Reserve. Confidence in the estimate of an Inferred Mineral Resource is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure.

Although the Authors are not experts with respect to environmental, permitting, legal, title, taxation, socio-economic, marketing, or political matters, the Authors are not aware of any unusual

factors relating to these matters that may materially affect the estimated Mineral Resources as of the effective date of this Technical Report.

The Authors consider that the Mineral Resource Estimate and Mineral Resource classification represent a reasonable estimation of the global mineralization for the El Tigre Project with regard to compliance with generally accepted industry standards and guidelines, the methodology used for estimation, the classification criteria used and the actual implementation of the methodology in terms of Mineral Resource estimation and reporting.

The Authors responsible for this Technical Report section are Messieurs Fred Brown, P.Geo., and Eugene Puritch, P.Eng., FEC, CET, both of P&E, and independent Qualified Persons in terms of National Instrument 43-101 by reason of education, affiliation with a professional association, and past relevant work experience. A draft copy of this Technical Report has been reviewed by Silver Tiger for factual errors.

14.1 DATA SUPPLIED

The El Tigre Project includes the El Tigre Main and North Veins, El Tigre Tailings and the El Tigre Low-Grade Stockpile.

Silver Tiger supplied exploration data as csv format tables for the El Tigre Project. The supplied exploration tables contain collar, survey, assay, lithology and bulk density data. The assay table contains Au g/t, Ag g/t, Cu %, Pb % and Zn % grades. The combined databases contain a total of 5,875 collar records that contribute directly to the Mineral Resource Estimate (Table 14.1). The drilling and deposits extend ~5 km along strike (Figure 14.1 and Appendix A).

TABLE 14.1
EXPLORATION SUMMARY

Type	Count	Total (m)
Vein Drilling		
Diamond drill holes	592	143,792.5
Geology drill holes	8	192.2
Historical underground channels	4,689	7,879.3
Metallurgical drill holes	6	167.0
Underground channels	465	978.0
Subtotal	**5,760**	**153,009.0**
Stockpiles		
Channel samples	67	41.0
Tailings		
Auger	46	315.4
Diamond drill holes	7	132.9
Pit	73	85.0

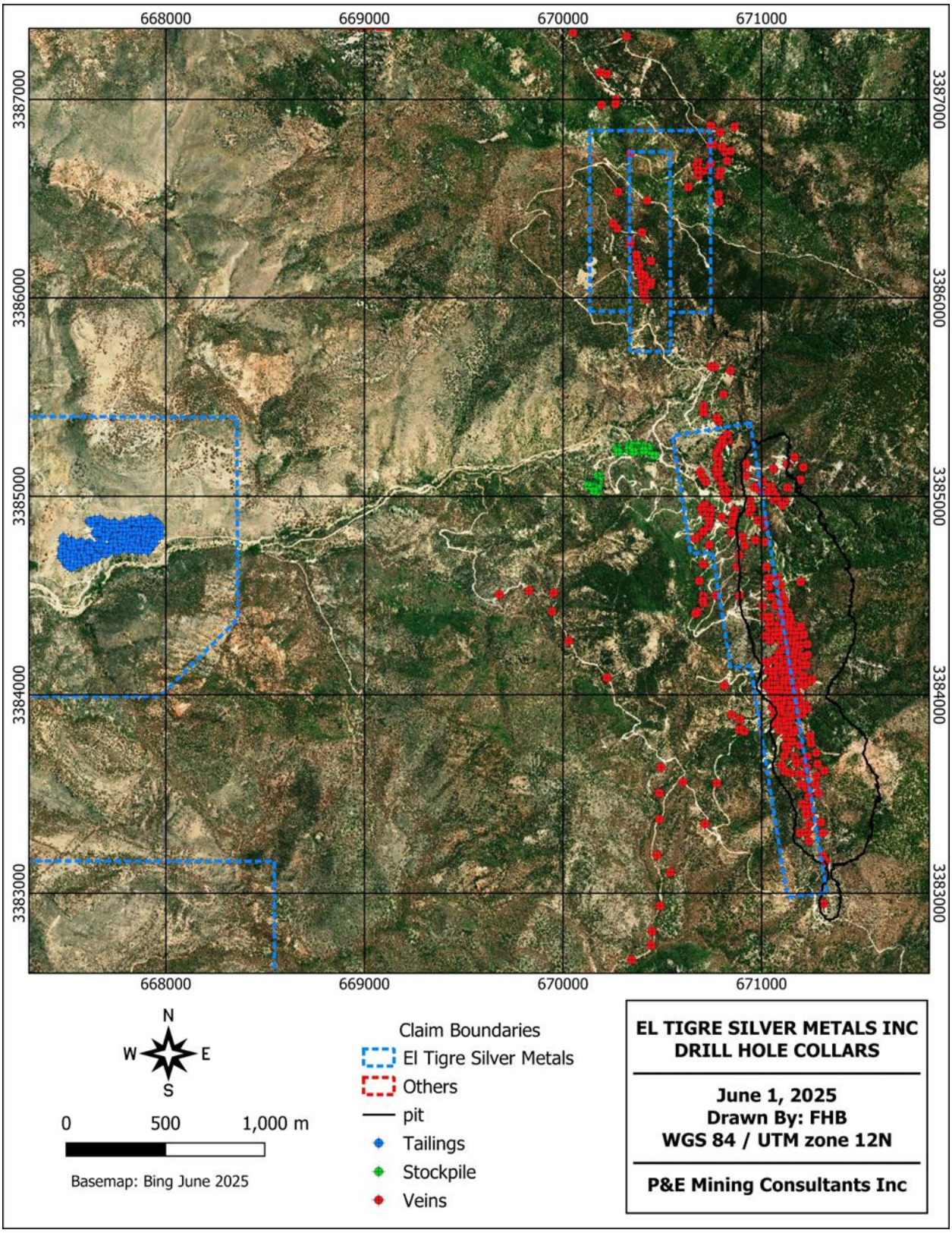
TABLE 14.1 EXPLORATION SUMMARY		
Type	**Count**	**Total (m)**
Channel samples	22	462.2
Subtotal	**148**	**995.5**
Total	**5,875**	**154,046.5**

The coordinate system used is WGS84 UTM Zone 12. Silver Tiger supplied Digital Terrain Models and AutoCAD™ format wireframes of the historical workings. Silver Tiger geologists also supplied interpreted wireframes of the mineralized domains at El Tigre.

Industry standard validation checks were carried out by the Authors on the supplied databases, and minor corrections made where necessary. The Authors typically validate a Mineral Resource database by checking for inconsistencies in naming conventions or analytical units, duplicate entries, interval, length or distance values less than or equal to zero, blank or zero-value assay results, out-of-sequence intervals, intervals or distances greater than the reported drill hole length, inappropriate collar locations, and missing interval and coordinate fields.

No significant errors were found with the supplied databases. The Authors consider that the exploration database is suitable for Mineral Resource Estimation.

FIGURE 14.1 EL TIGRE DRILLING AND DEPOSIT LOCATIONS



Source: *P&E (2025)*

14.2 ECONOMIC CONSIDERATIONS

For the Mineral Resource models, Silver Tiger selected the economic parameters listed in Table 14.2. The Authors reviewed these economic parameters and consider that they are suitable for Mineral Resource estimation.

TABLE 14.2
ECONOMIC PARAMETERS (US$)

Metal Prices			
Metal	**Units**	**Price**	
Gold Price	$/oz	2,400	
Silver Price	$/oz	28.00	
Copper Price	$/lb	4.25	
Lead Price	$/lb	0.97	
Zinc Price	$/lb	1.30	
Pit-Constrained			
Item	**Units**	**Oxide**	**Sulphide**
Au Recovery	%	83	56
Ag Recovery	%	45	40
Mining Cost	$/t	2.00	2.00
Process Cost	$/t	4.25	4.25
G&A Cost	$/t	1.00	1.00
Ag:Au	Ratio	178	120
Cut-off	AuEq g/t	0.08	0.12
Out-of-Pit			
Item	**Units**	**Amount**	
Au Recovery	%	89	
Ag Recovery	%	93	
Cu Recovery	%	85	
Pb Recovery	%	95	
Zn Recovery	%	80	
Mining Cost	$/t	60.00	
Process Cost	$/t	25.00	
G&A Cost	$/t	3.00	
Ag:Au	Ratio	82	
Cu:Au	Ratio	116	
Pb:Au	Ratio	30	
Zn:Au	Ratio	33	
Cut-off	AuEq g/t	1.28	

TABLE 14.2 ECONOMIC PARAMETERS (US$)			
Tailings			
Item	**Units**	**Amount**	
Au Recovery	%	82	
Ag Recovery	%	83	
Process Cost	$/t	20.62	
G&A Cost	$/t	8.10	
Ag:Au	Ratio	85	
Cut-off	AuEq g/t	0.45	
Stockpile			
Item	**Units**	**Amount**	
Au Recovery	%	85	
Ag Recovery	%	85	
Mining Cost	$/t	2.00	
Process Cost	$/t	25.00	
G&A Cost	$/t	3.00	
Ag:Au	Ratio	86	
Cut-off	AuEq g/t	0.46	

14.3 MINERALIZED DOMAINS

Interpreted mineralized wireframes were developed by Silver Tiger geologists for the El Tigre Veins based on logged drill hole lithology, assay grades and historical records. Silver Tiger identified continuous zones of mineralization from assay grades equal to or greater than 0.30 g/t AuEq with observed continuity along strike and down-dip, using a calculated Ag:Au equivalent factor of 75:1. The selected intervals include lower-grade material where necessary to maintain continuity between drill holes. Three-dimensional wireframes linking drill hole cross-sections were subsequently constructed using the Leapfrog™ Radial Basis Function, with hanging wall and footwall surfaces snapped directly to the selected drill hole intercepts.

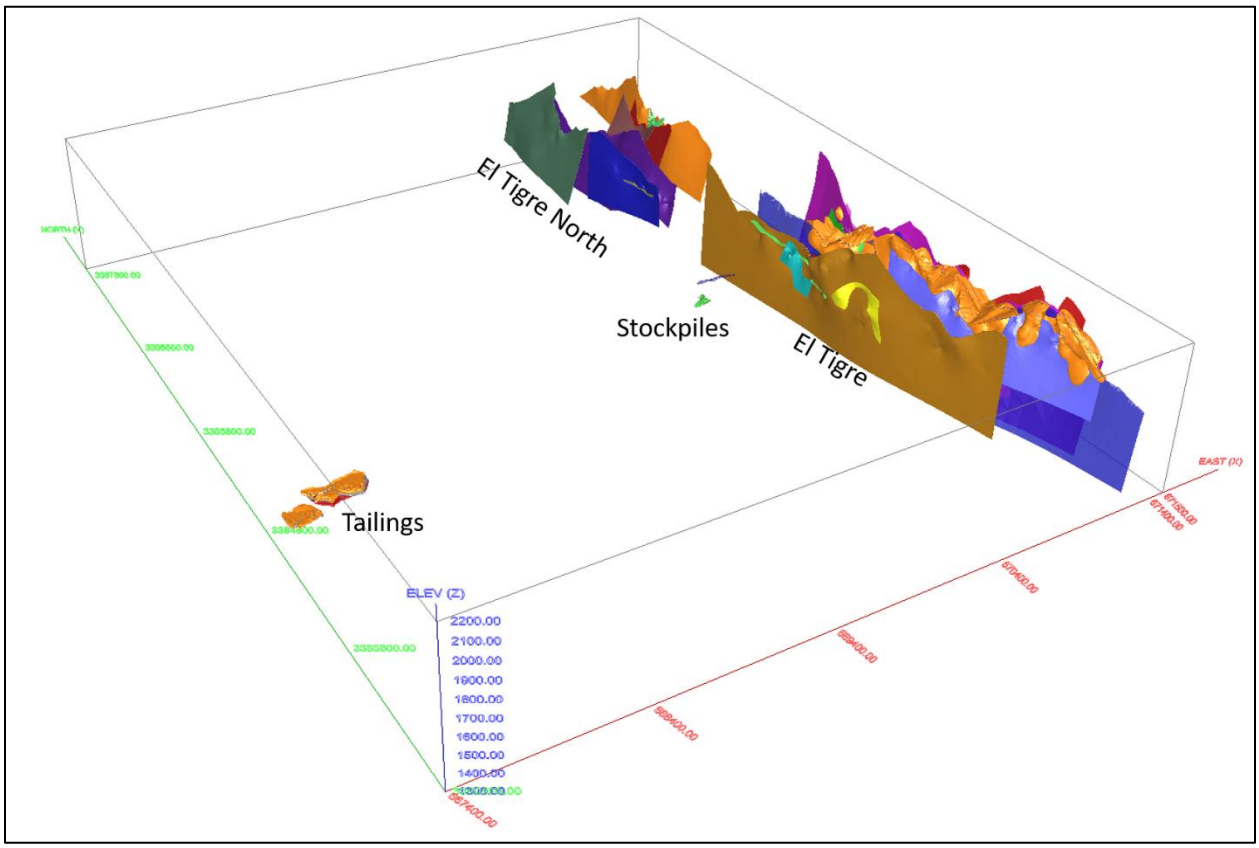
The Authors developed mineralized domains for the El Tigre Low-Grade Stockpile and the El Tigre Tailings based on lithological logging and LiDAR surface topography.

A total of 28 individual mineralized domains were defined (Figure 14.2 and Appendix B). The Authors reviewed the mineralized domains and confirm that they are suitable for Mineral Resource estimation. The mineralized domain wireframes were used to back-tag the assay, bulk density and composite tables with unique rock codes (Table 14.3).

Silver Tiger also supplied interpreted oxide and sulphide boundaries for the El Tigre Main Veins based on the logging of unoxidized pyrite.

TABLE 14.3 MINERALIZED DOMAINS			
Domain	**Zone**	**Rock Code**	**Strike Length (m)**
El Tigre	Main Veins	100	2,750
East Tigre	Main Veins	105	400
West Tigre	Main Veins	110	240
BS Tigre	Main Veins	115	510
Sooy	Main Veins	200	2,410
East Sooy	Main Veins	205	1,350
Angi	Main Veins	210	350
BS Sooy	Main Veins	215	740
Seitz-Kelly	Main Veins	300	2,350
East Seitz-Kelly 2	Main Veins	310	180
West Seitz-Kelly	Main Veins	315	330
BS Seitz-Kelly	Main Veins	320	500
East Seitz-Kelly	Main Veins	350	950
Protectora	North Veins	400	1,770
Fundadora	North Veins	405	1,160
Caleigh	North Veins	410	630
Benjamin	North Veins	415	610
BS Benjamin	North Veins	420	260
Aquila	North Veins	425	1,300
Esc	North Veins	430	530
Protectora Halo	North Veins	440	260
Sulphide	Main Veins	500	770
Stockwork	Main Veins	600	2,120
Stockpile 1	Stockpile	600	230
Stockpile 2	Stockpile	605	80
Grey	Tailings	100	490
Red	Tailings	200	350
Yellow	Tailings	300	490

FIGURE 14.2 MINERALIZED DOMAINS



Source: *P&E (2024)*

14.4 EXPLORATORY DATA ANALYSIS

The average nearest neighbour collar distances are as follows:

- Veins: 14.8 m.
- Low-Grade Stockpiles: 8.0 m.
- Tailings: 12.6 m.

The average length of the diamond drill holes for all veins is 242.9 m. The average length of the Low-Grade Stockpiles channel samples is 0.67 m, and the average length of the Tailing samples is 8.46 m. Summary statistics for the constrained assay data are listed in Table 14.4.

TABLE 14.4 ASSAY SUMMARY STATISTICS						
Ag (g/t)	**N**	**Mean**	**Std Dev**	**CoV**	**Minimum**	**Maximum**
Main Veins	28,312	168.70	784.30	4.65	0.05	32,845.70
North Veins	1,322	138.80	407.40	2.94	0.10	7,338.90
Stockpiles	70	157.50	121.00	7.68	2.40	655.30
Tailings	318	85.50	24.78	2.90	8.10	191
Au (g/t)	**N**	**Mean**	**Std Dev**	**CoV**	**Minimum**	**Maximum**
Main Veins	28,312	1.06	11.39	10.71	0.003	1,097.80
North Veins	1,322	0.62	2.72	4.40	0.003	58.67
Stockpiles	70	1.37	3.23	2.37	0.03	21.38
Tailings	319	0.30	0.15	5.03	0.08	1.27
Cu (%)	**N**	**Mean**	**Std Dev**	**CoV**	**Minimum**	**Maximum**
Main Veins	8,392	0.06	0.30	5.20	0.001	7.27
North Veins	1,142	0.05	0.27	5.14	0.001	5.40
Stockpiles	70	0.02	0.02	1.00	0.0001	0.100
Tailings	NA	NA	NA	NA	NA	NA
Pb (%)	**N**	**Mean**	**Std Dev**	**CoV**	**Minimum**	**Maximum**
Main Veins	11,270	0.16	0.90	5.73	0.001	17.48
North Veins	1,196	0.60	1.93	3.22	0.001	20.74
Stockpiles	70	0.20	0.25	1.26	0.01	1.66
Tailings	NA	NA	NA	NA	NA	NA
Zn (%)	**N**	**Mean**	**Std Dev**	**CoV**	**Minimum**	**Maximum**
Main Veins	11,573	0.26	1.61	6.19	0.001	0.01
North Veins	1,193	1.10	3.73	3.39	0.001	0.03
Stockpiles	70	0.10	0.15	1.45	0.01	1.18
Tailings	NA	NA	NA	NA	NA	NA

Note: *N = sample population size, Std Dev = standard deviation, CoV = coefficient of variation.*

Bulk density measurements for the veins were determined by Silver Tiger using the water immersion (Archimedes) method on diamond drill core samples. Samples were collected at regular downhole intervals and data from all the major units in the hanging wall, mineralized zones and footwall were logged. The average bulk density is 2.56 t/m^3 and for only the mineralized veins is 2.72 t/m^3 (Table 14.5).

The Tailings average dry bulk density of 1.39 t/m^3 was derived from 37 nuclear density measurements. For the Low-Grade Stockpiles a bulk density of 1.60 t/m^3, the bulk density of dry sand, was assigned.

TABLE 14.5 SUMMARY OF BULK DENSITY STATISTICS			
Vein	**Count**	**Avg. Bulk Density (t/m^3)**	**Median Bulk Density (t/m^3)**
El Tigre	62	2.66	2.56
East Tigre	13	2.57	2.52
West Tigre	3	2.78	2.88
Sooy	94	2.72	2.68
East Sooy	4	2.51	2.53
Angi	11	2.50	2.52
BS Sooy	113	2.96	2.88
Seitz-Kelly	38	2.56	2.55
East Seitz-Kelly 2	17	3.04	2.91
West Seitz-Kelly	5	2.68	2.62
BS Seitz-Kelly	18	2.72	2.67
East Seitz-Kelly	10	2.55	2.49
BS Benjamin	71	2.88	2.72
Sulphide	461	3.01	2.89
Stockwork	829	2.52	2.51
Total Veins	**1,749**	**2.72**	**2.59**
Unassigned	5,542	2.56	2.54

14.5 COMPOSITING

Constrained assay sample lengths within the defined mineralized veins range from 0.01 to 6.90 m, with an average sample length of 1.19 m and a median of 1.00 m. Based on the median of 1.00 m, all constrained assay samples were composited to this length in order to ensure equal sample support. Length-weighted composites were calculated within the defined mineralized vein domains. A small number of un-sampled intervals in the data were treated as null intervals due to the presence of mining voids, backfill and surface stockpiles. Residual composites less than 0.50 m were merged with the preceding interval.

No compositing was utilized for the Low-Grade Stockpiles since 91% of the sample lengths are 0.50 m. Each sample location was treated as a single point for grade estimation.

Assay sample lengths within the Tailings area average 1.61 m. A total of 71% of the samples are exactly 1.50 m in length, and Tailing assays were therefore not composited prior to grade estimation.

The composite data were visually validated against the mineralized wireframes and used to back-tag a rock code into the drill hole workspace. Composite summary statistics are listed in Table 14.6.

TABLE 14.6 COMPOSITE SUMMARY STATISTICS						
Ag (g/t)	**N**	**Mean**	**Std Dev**	**CoV**	**Minimum**	**Maximum**
Main Veins	30,689	100.70	526.30	5.22	0.001	25,961.10
North Veins	1,034	105.07	291.50	2.77	0.001	4,898.01
Stockpiles	70	157.50	121.00	7.68	2.40	655.30
Tailings	318	85.50	24.78	2.90	8.10	191
Au (g/t)	**N**	**Mean**	**Std Dev**	**CoV**	**Minimum**	**Maximum**
Main Veins	30,689	0.58	4.35	7.49	0.001	547.86
North Veins	1,034	0.40	1.45	3.64	0.001	24.83
Stockpiles	70	1.37	3.23	2.37	0.03	21.38
Tailings	319	0.30	0.15	5.03	0.08	1.27
Cu (%)	**N**	**Mean**	**Std Dev**	**CoV**	**Minimum**	**Maximum**
Main Veins	26,613	0.01	0.12	9.38	0.001	5.73
North Veins	994	0.04	0.23	5.71	0.001	5.40
Stockpiles	70	0.02	0.02	1.00	0.0001	0.100
Tailings	NA	NA	NA	NA	NA	NA
Pb (%)	**N**	**Mean**	**Std Dev**	**CoV**	**Minimum**	**Maximum**
Main Veins	29,999	0.04	0.41	10.18	0.001	14.74
North Veins	1,034	0.44	1.39	3.17	0.001	16.38
Stockpiles	70	0.20	0.25	1.26	0.01	1.66
Tailings	NA	NA	NA	NA	NA	NA
Zn (%)	**N**	**Mean**	**Std Dev**	**CoV**	**Minimum**	**Maximum**
Main Veins	30,389	0.07	0.74	11.11	0.001	30.03
North Veins	1,034	0.83	2.90	3.49	0.001	33.31
Stockpiles	70	0.10	0.15	1.45	0.01	1.18
Tailings	NA	NA	NA	NA	NA	NA

Note: *N = sample population size, Std Dev = standard deviation, CoV = coefficient of variation.*

14.6 TREATMENT OF EXTREME VALUES

Capping thresholds were determined by the analysis of individual composite log-probability distributions. Composites are capped to the defined threshold prior to grade estimation (Table 14.7).

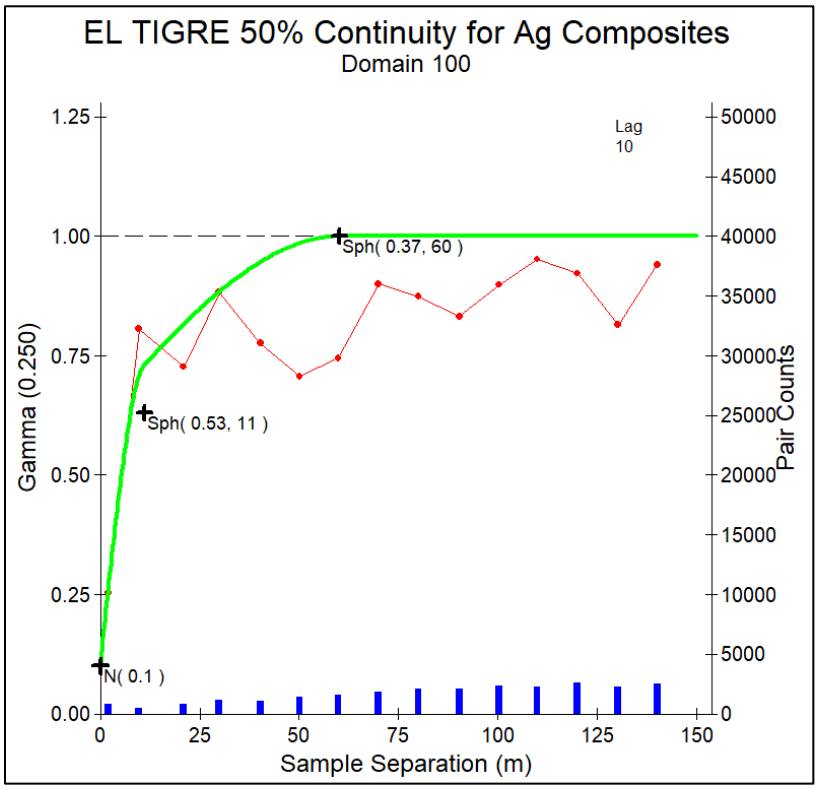
TABLE 14.7 CAPPING THRESHOLDS					
Domain	**Ag (g/t)**	**Au (g/t)**	**Cu (%)**	**Pb (%)**	**Zn (%)**
El Tigre 100	1,350	12.00	0.10	0.50	0.20
East Tigre 105	180	5.00	0.06	0.20	0.20
West Tigre 110	150	2.00	0.04	0.40	0.11
BS Tigre 115	150	NA	NA	NA	NA
Sooy 200	1,450	3.00	0.24	0.40	0.50
East Sooy 205	100	6.00	0.01	0.01	0.01
Angi 210	1,440	4.00	0.02	0.30	0.04
BS Sooy 215	3400	NA	NA	NA	NA
Seitz-Kelly 300	900	4.00	0.10	0.30	0.30
East Seitz-Kelly 2 310	1,000	0.35	0.04	0.05	0.30
West Seitz-Kelly 315	200	2.00	0.03	0.05	0.10
BS Seitz-Kelly 320	530	NA	NA	NA	NA
East Seitz-Kelly 350	190	3.00	0.03	0.10	0.10
Protectora 400	1,000	13.00	NA	11.00	NA
Fundadora 405	850	18.00	NA	NA	NA
Caleigh 410	1,420	30.00	NA	NA	NA
Benjamin 415	6,000	1.00	NA	NA	12.00
BS Benjamin 420	2,900	1.00	NA	NA	18.00
Aquila 425	500	4.00	NA	NA	NA
Esc 430	200	1.00	NA	NA	NA
Sulphide 500	2,000	1.40	NA	NA	NA
Stockwork 600	1,200	25.00	0.01	0.40	0.10
Stockpile 1	490	20.00	NA	NA	0.40
Stockpile 2	490	20.00	NA	NA	0.40
Grey	100	1.00	NA	NA	NA
Red	100	1.00	NA	NA	NA
Yellow	100	1.00	NA	NA	NA

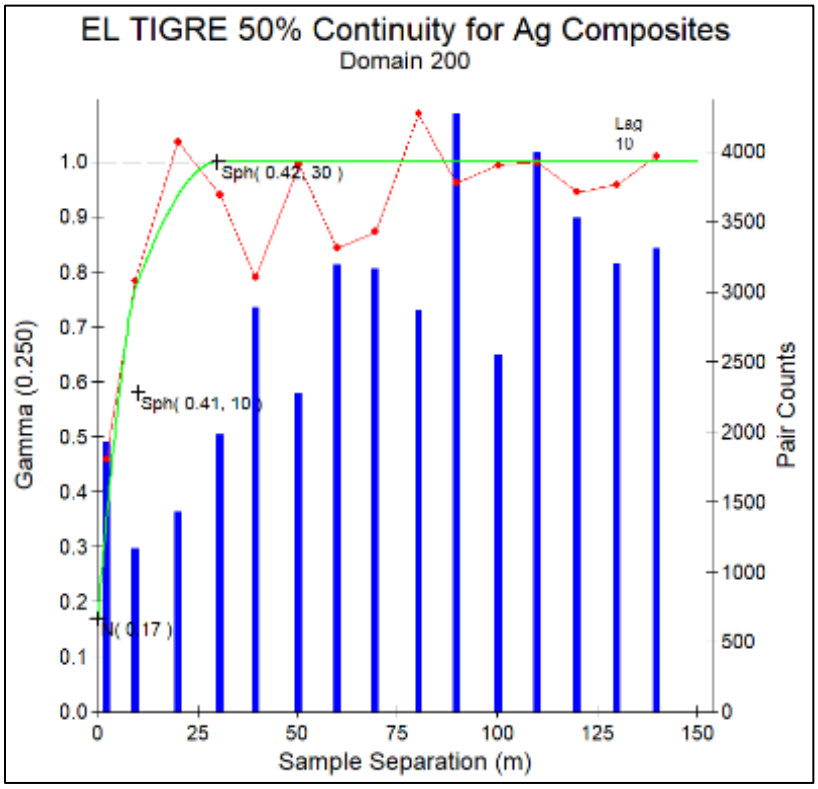
14.7 CONTINUITY ANALYSIS

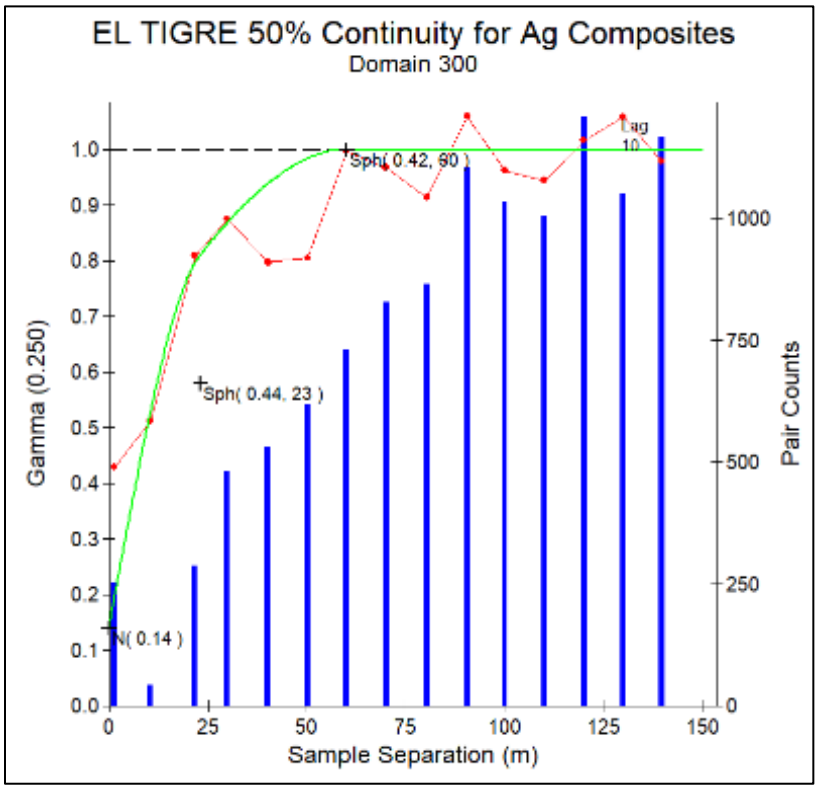
For the El Tigre Main Veins and the El Tigre Tailings, median indicator variograms of Ag uncapped composites within individual domains with sufficient informing data were used as a guide in determining grade interpolation search ranges and ellipse orientation strategy. The downhole variogram was viewed at a 1.0 m lag spacing (equivalent to the composite length) to assess the nugget variance contribution. Standardized spherical models were used to model the experimental semi-variograms (Figure 14.3). Continuity ellipses based on the observed ranges were subsequently generated and utilized as the basis for grade estimation search ranges, distance weighting calculations and Mineral Resource classification criteria.

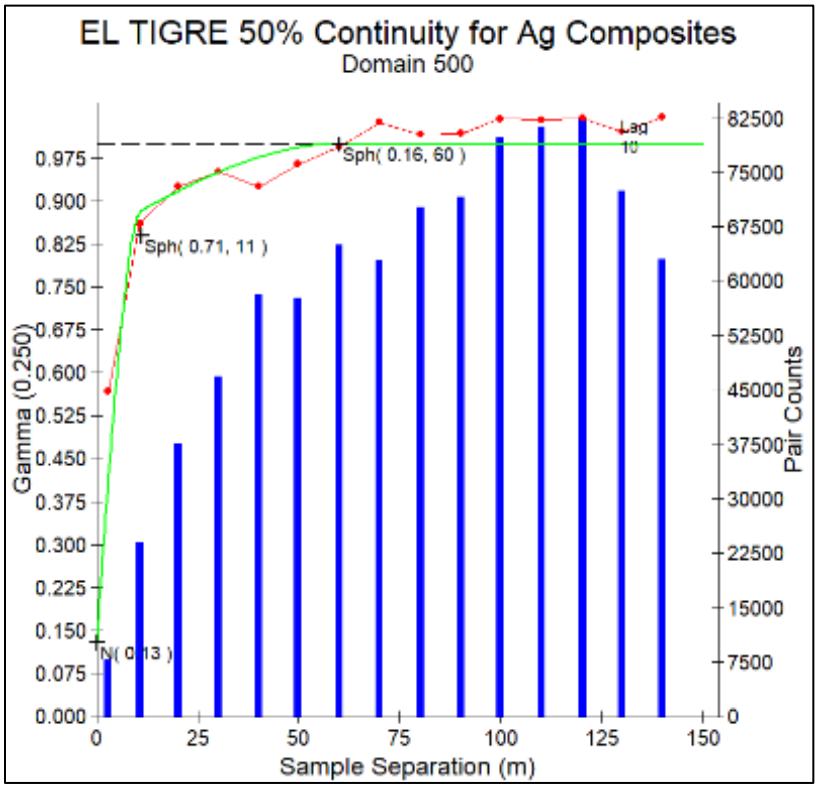
For El Tigre North, variography of Ag capped composites within individual domains with sufficient informing data was used as a guide in determining grade interpolation search ranges and ellipse orientation strategy. Continuity ellipses based on the observed ranges were subsequently generated and utilized as the basis for grade estimation search ranges, distance weighting calculations and Mineral Resource classification criteria.

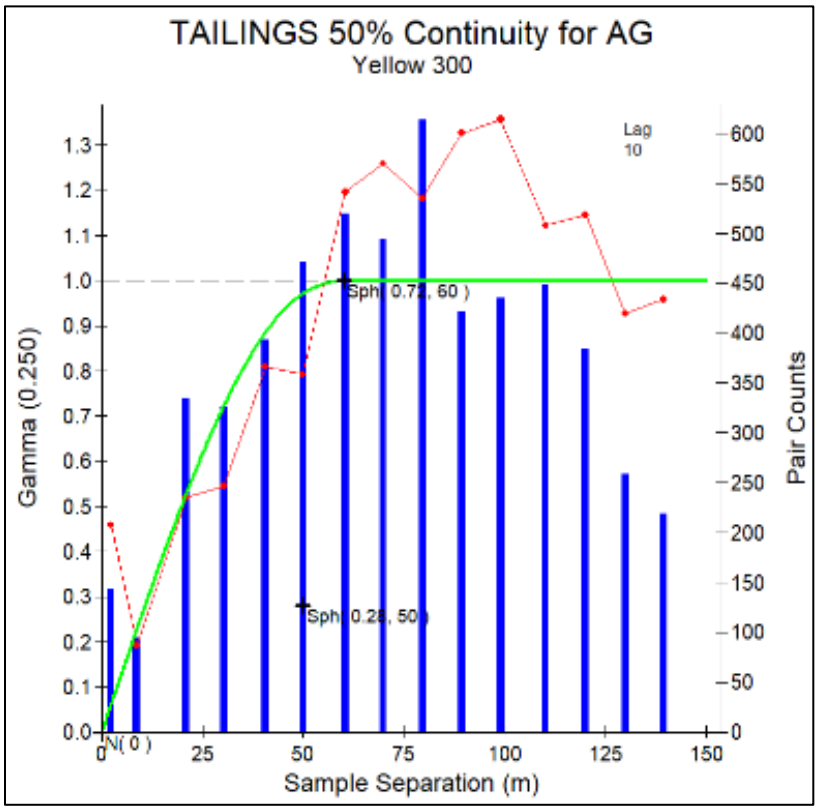
FIGURE 14.3 SEMI-VARIOGRAMS

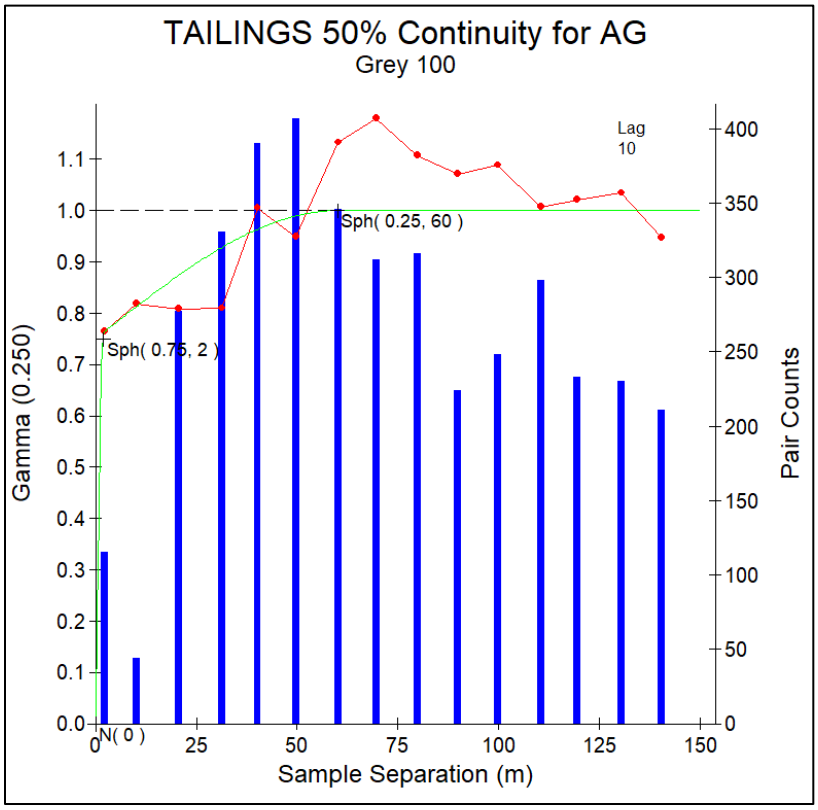



EL TIGRE 50% Continuity for Ag Composites
Domain 200
Lag
10
Sph(0.42, 30)
Sph(0.41, 10)
N(0.17)
Gamma (0.250)
Pair Counts
Sample Separation (m)


EL TIGRE 50% Continuity for Ag Composites
Domain 300
Lag
10
Sph(0.42, 80)
Sph(0.44, 23)
N(0.14)
Gamma (0.250)
Pair Counts
Sample Separation (m)


EL TIGRE 50% Continuity for Ag Composites
Domain 500
Gamma (0.250)
Pair Counts
Sample Separation (m)
Sph(0.16, 60)
Sph(0.71, 11)
Lag
10
N(0.13)


TAILINGS 50% Continuity for AG
Yellow 300
Gamma (0.250)
Pair Counts
Sample Separation (m)
Lag
10
Sph(0.28, 50)
N(0)



Source: *P&E (2024)*

14.8 BLOCK MODELS

Orthogonal block models were established with the block model limits selected to cover the extent of the mineralized domains (Table 14.8). Cross-sections and plans showing the AuEq block models are presented in Appendix C.

TABLE 14.8
VEINS BLOCK MODEL SETUP

Direction	Origin (m)	Extents (m)	Block Size (m)	Sub-Block Size (m)
Minimum X	670,100	1,800	2	1
Minimum Y	3,382,600	5,400	6	1
Minimum Z	1,270	1,200	7	1
Rotation	No Rotation			

LOW-GRADE STOCKPILE BLOCK MODEL SETUP

Direction	Origin (m)	Number of Blocks	Block Size (m)
Minimum X	670,100	80	5

TABLE 14.8 VEINS BLOCK MODEL SETUP			
Minimum Y	3,385,000	60	5
Minimum Z	1,660	75	1
Rotation	No Rotation		
TAILINGS BLOCK MODEL SETUP			
Direction	**Origin (m)**	**Number of Blocks**	**Block Size (m)**
Minimum X	667,400	134	5
Minimum Y	3,384,600	70	5
Minimum Z	1,320	30	5
Rotation	No Rotation		

14.9 BULK DENSITY, GRADE ESTIMATION AND MINERAL RESOURCE CLASSIFICATION

Mineralized bulk density values were assigned for each of the El Tigre Main Veins based on the median vein measurement. For the El Tigre North Veins, a bulk density of 2.65 t/m^3 was assigned for the veins. For the Low-Grade Stockpile a value of 1.60 t/m^3 was assigned, and for the Tailings a value of 1.39 t/m^3 was utilized.

Vein block grades for gold and silver were estimated by Inverse Distance Cubed ("ID^3") interpolation of capped composites using a minimum of four and a maximum of 12 composites. Vein block grades for copper, lead and zinc were estimated by Inverse Distance Squared ("ID^2") interpolation of capped composites using a minimum of four and a maximum of 12 composites.

Nearest-Neighbour grade interpolation was used for the Low-Grade Stockpiles. Tailings block grades were estimated by ID^2 estimation of capped assays using a minimum of four and a maximum of 12 samples.

The orientation of the search ellipsoids were defined by the modeled variography, observed grade trends and historical mining. Composites were selected within a search ellipsoid oriented parallel to the observed mineralization. Composite selection was restricted to a maximum of three composite samples from a single drill hole. Search and grade estimation were constrained within the individual mineralized domains, which define hard boundaries for grade estimation.

The parameters used to define the classification limits included the availability of bulk density and metallurgical information, spatial analysis, drill hole spacing, and the observed continuity of the mineralization. Mineral Resources were classified algorithmically based on the local drill hole spacing within each individual mineralized domain. For the purposes of classification, historical underground channel sampling and surface trench sampling were treated as drill holes, however, only results from drill holes and Silver Tiger contemporary channel sampling were used for block grade estimation.

The El Tigre Veins blocks within 30 m of three or more drill holes/channels were classified as Measured Mineral Resources, and blocks within 60 m of three or more drill holes/channels were classified as Indicated Mineral Resources. All additional estimated El Tigre blocks within 180 m of a drill hole were classified as Inferred Mineral Resources. Subsequent to the initial classification, blocks were re-classified using a maximum a-posteriori selection pass that corrected isolated classification artifacts and consolidated areas of similar classification into continuous shapes. Areas of known historical mining were depleted from the model.

Low-Grade Stockpiles with blocks within 15 m of two or more drill holes were classified as Indicated Mineral Resources. All additional estimated Stockpile blocks were classified as Inferred Mineral Resources.

Tailings blocks within 30 m of three or more auger or core drill holes were classified as Measured Mineral Resources. Blocks within 60 m of two or more auger/drill holes/pits or trenches were classified as Indicated Mineral Resources. All additional estimated Tailings blocks were classified as Inferred Mineral Resources. Subsequent to the initial classification, blocks were re-classified using a maximum a-posteriori selection pass that corrected isolated classification artifacts and consolidated areas of similar classification into continuous shapes.

Classification block model plans and cross-sections are located in Appendix D.

14.10 MINERAL RESOURCE ESTIMATE

National Instrument 43-101 incorporates by reference the definition of, among other terms, "Mineral Resource" from the Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards for Mineral Resources & Mineral Reserves (the "CIM Definition Standards (2014)" and Best Practices Guidelines (2019)). Under the CIM Definition Standards, a Mineral Resource must demonstrate "reasonable prospects for eventual economic extraction". The effective date of this Mineral Resource Estimate is June 20, 2025.

Pit-constrained Mineral Resources reported herein have been constrained within an optimized pit shell as shown in Appendix E. Pit optimization was performed using NPV Scheduler™ software. The results from the optimized pit shell are used solely for the purpose of reporting Mineral Resources and include Measured, Indicated and Inferred Mineral Resources. Historical mining has been depleted from the El Tigre Veins. Pit-constrained Mineral Resources are reported using a cut-off of 0.08 g/t AuEq for oxides and 0.12 g/t AuEq for sulphides.

Out-of-pit Mineral Resources have been reported beneath the pit shell for areas that exhibit historical continuity and reasonable potential for extraction by cut and fill and long hole mining methods. Optimized stope shapes suitable for long term planning were generated using Deswik Stope Optimizer™ software. Out-of-pit Mineral Resources are reported using a cut-off of 1.28 g/t AuEq within the optimized stope shapes.

The Low-Grade Stockpiles are reported using a cut-off of 0.46 g/t AuEq. The Tailings are reported using a cut-off of 0.45 g/t AuEq. The Mineral Resource Estimate is presented in Table 14.9.

TABLE 14.9
MINERAL RESOURCE ESTIMATE (1-14)

Classification	Cut-off AuEq (g/t)	Tonnes (k)	Ag (g/t)	Ag (koz)	Au (g/t)	Au (koz)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)	AgEq (koz)	AuEq (g/t)	AuEq (koz)
Pit-Constrained Oxide													
Measured	0.08	27,732	39	35,034	0.54	609	NA	NA	NA	136	120,923	0.78	693
Indicated	0.08	19,804	10	6,593	0.36	75	NA	NA	NA	74	46,979	0.42	266
Meas + Ind	**0.08**	**47,536**	**27**	**41,626**	**0.46**	**709**	**NA**	**NA**	**NA**	**110**	**167,902**	**0.63**	**959**
Inferred	0.08	4,475	10	1,453	0.38	18	NA	NA	NA	77	11,045	0.44	63
Pit-Constrained Sulphide													
Measured	0.12	6,031	31	6,042	0.54	105	NA	NA	NA	96	18,667	0.79	154
Indicated	0.12	1,723	14	756	0.46	25	NA	NA	NA	69	3,817	0.57	32
Meas + Ind	**0.12**	**7,755**	**27**	**6,798**	**0.52**	**131**	**NA**	**NA**	**NA**	**90**	**22,484**	**0.74**	**185**
Inferred	0.12	463	74	1,096	0.52	8	NA	NA	NA	136	2,034	1.11	17
Pit-Constrained Total													
Measured	0.08/0.12	33,763	38	41,076	0.54	588	NA	NA	NA	129	139,590	0.78	846
Indicated	0.08/0.12	21,528	11	7,348	0.36	252	NA	NA	NA	73	50,796	0.43	298
Meas + Ind	**0.08/0.12**	**55,291**	**27**	**48,425**	**0.47**	**840**	**NA**	**NA**	**NA**	**107**	**190,386**	**0.64**	**1,144**
Inferred	0.08/0.12	4,939	16	2,550	0.39	62	NA	NA	NA	82	13,078	0.50	79
Out-Of-Pit													
Measured	1.28	1,010	453	14,698	1.05	34	0.13	0.54	0.92	589	19,118	7.09	230
Indicated	1.28	1,374	398	17,588	1.05	47	0.08	0.24	0.41	509	22,482	6.13	271
Meas + Ind	**1.28**	**2,384**	**421**	**32,286**	**1.05**	**81**	**0.10**	**0.37**	**0.63**	**543**	**41,600**	**6.54**	**501**
Inferred	1.28	4,100	294	38,822	1.58	208	0.32	0.75	0.89	497	65,556	6.01	792

TABLE 14.9
MINERAL RESOURCE ESTIMATE (1-14)

Classification	Cut-off AuEq (g/t)	Tonnes (k)	Ag (g/t)	Ag (koz)	Au (g/t)	Au (koz)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)	AgEq (koz)	AuEq (g/t)	AuEq (koz)
Tailings													
Measured	0.45	146	79	371	0.29	1	NA	NA	NA	103	487	1.22	6
Indicated	0.45	706	77	1,752	0.26	6	NA	NA	NA	99	2,252	1.17	27
Meas + Ind	**0.45**	**852**	**77**	**2,118**	**0.27**	**7**	**NA**	**NA**	**NA**	**202**	**2,739**	**1.18**	**32**
Inferred	0.45	52	81	134	0.27	1	NA	NA	NA	104	173	1.23	2
Stockpile													
Measured	0.46	0	0	0	0	0	0	0	0	0	0	0	0
Indicated	0.46	101	181	586	0.92	3	0.02	0.23	0.10	256	826	3.17	10
Mea + Ind	**0.46**	**101**	**181**	**586**	**0.92**	**3**	**0.02**	**0.23**	**0.10**	**256**	**826**	**3.17**	**10**
Inferred	0.46	18	146	83	0.46	0.3	0.02	0.17	0.09	183	104	2.27	1
Total													
Measured	NA	34,920	50	56,145	0.55	623	0.13	0.54	0.92	142	159,193	0.96	1,082
Indicated	NA	23,708	36	27,275	0.40	308	0.07	0.24	0.39	100	76,350	0.79	605
Mea + Ind	**NA**	**58,628**	**44**	**83,415**	**0.49**	**931**	**0.09**	**0.36**	**0.61**	**125**	**235,543**	**0.90**	**1,688**
Inferred	NA	9,108	142	41,589	0.92	271	0.32	0.75	0.89	269	78,911	2.99	874

Notes:

1. *Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*
2. *The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.*
3. *The Mineral Resources were estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.*
4. *Historically mined areas were depleted from the Mineral Resource model.*

5. *Prices used are US$2,400/oz Au, US$28/oz Ag, US$4.25/lb Cu, US$0.97 lb Pb and US$1.30/lb Zn.*
6. *The pit-constrained respective oxide and sulphide AuEq cut-off grades of 0.08 and 0.12 g/t were derived from 45% Ag and 83% Au oxide process recovery, 40% Ag and 56% Au sulphide process recovery, US$5.25/t process and G&A cost. The constraining pit optimization parameters were $2.00/t mining cost and 45° pit slopes.*
7. *The out-of-pit AuEq cut-off grade of 1.28 g/t was derived from 93% Ag and 89% Au process recovery, US$28/t process and G&A cost, and a $60/t mining cost. The out-of-pit Mineral Resource grade blocks were quantified above the 1.28 g/t AuEq cut-off, below the constraining pit shell and within the constraining mineralized wireframes. Out–of-Pit Mineral Resources are restricted to the El Tigre Veins which exhibit historical continuity and reasonable potential for extraction by cut and fill and long hole mining methods.*
8. *The Stockpile AuEq cut-off grade of 0.46 g/t was derived from 85% Ag and 85% Au process recovery, US$28/t process and G&A cost, and a $2/t mining cost.*
9. *The Tailings AuEq cut-off grade of 0.45 g/t was derived from 83% Ag and 82% Au process recovery, US$28.72/t process and G&A cost.*
10. *AgEq and AuEq were calculated at an Ag/Au ratio of 178:1 (oxide) and 120:1 (sulphide) for pit-constrained Mineral Resources.*
11. *AgEq and AuEq were calculated at an Ag/Au ratio of 82:1 for out-of-pit Mineral Resources.*
12. *AgEq and AuEq were calculated at an Ag/Au ratio of 86:1 for Low-Grade Stockpile Mineral Resources.*
13. *AgEq and AuEq were calculated at an Ag/Au ratio of 85:1 for Tailings Mineral Resources.*
14. *Totals may not sum due to rounding.*

Table 14.10 presents the Out-Of-Pit (underground) Mineral Resources split into the South and North areas.

TABLE 14.10
OUT-OF-PIT MINERAL RESOURCE ESTIMATE (1-8)

Classification	Cut-off AuEq (g/t)	Tonnes (k)	Ag (g/t)	Ag (koz)	Au (g/t)	Au (koz)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)	AgEq (koz)	AuEq (g/t)	AuEq (koz)
Out-Of-Pit South													
Measured	1.28	822	479	12,661	1.06	28	0.14	0.50	0.87	615	16,242	7.41	196
Indicated	1.28	1,066	430	14,736	1.01	35	0.07	0.20	0.40	535	18,340	6.44	221
Meas + Ind	**1.28**	**1,887**	**452**	**27,397**	**1.03**	**63**	**0.10**	**0.33**	**0.60**	**570**	**34,582**	**6.86**	**416**
Inferred	1.28	1,370	322	14,193	0.63	28	0.04	0.11	0.14	385	16,943	4.63	204
Out-Of-Pit North													
Measured	1.28	189	336	2,037	1.01	6	0.08	0.71	1.12	474	2,876	5.72	35
Indicated	1.28	308	288	2,852	1.19	12	0.11	0.38	0.47	418	4,142	5.04	50
Meas + Ind	**1.28**	**497**	**306**	**4,890**	**1.12**	**18**	**0.10**	**0.51**	**0.72**	**439**	**7,018**	**5.30**	**85**
Inferred	1.28	2730	281	24,629	2.06	180	0.46	1.07	1.27	554	48,613	6.70	588
Out-Of-Pit Total													
Measured	1.28	1,010	453	14,698	1.05	34	0.13	0.54	0.92	589	19,118	7.09	230
Indicated	1.28	1,374	398	17,588	1.05	47	0.08	0.24	0.41	509	22,482	6.13	271
Mea + Ind	**1.28**	**2,384**	**421**	**32,286**	**1.05**	**81**	**0.10**	**0.37**	**0.63**	**543**	**41,600**	**6.54**	**501**
Inferred	1.28	4,100	294	38,822	1.58	208	0.32	0.75	0.89	497	65,556	6.01	792

Notes:

1. *Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.*
2. *The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.*
3. *The Mineral Resources were estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.*
4. *Historically mined areas were depleted from the Mineral Resource model.*

5. *Prices used are US$2,400/oz Au, US$28/oz Ag, US$4.25/lb Cu, US$0.97 lb Pb and US$1.30/lb Zn.*
6. *The out-of-pit AuEq cut-off grade of 1.28 g/t was derived from 93% Ag and 89% Au process recovery, US$28/t process and G&A cost, and a $60/t mining cost. The out-of-pit Mineral Resource grade blocks were quantified above the 1.28 g/t AuEq cut-off, below the constraining pit shell and within the constraining mineralized wireframes. Out–of-Pit Mineral Resources are restricted to the El Tigre Veins which exhibit historical continuity and reasonable potential for extraction by cut and fill and long hole mining methods.*
7. *AgEq and AuEq were calculated at an Ag/Au ratio of 82:1 for out-of-pit Mineral Resources.*
8. *Totals may not sum due to rounding.*

14.11 SENSITIVITY

The sensitivity of the Mineral Resource Estimate to changes in cut-off grade was examined by summarizing tonnes and grade at varying AuEq cut-off grades for Pit-Constrained (Tables 14.11 and 14.12) and Out-Of-Pit Mineral Resources (Table 14.13). The results suggest that the Mineral Resource models are relatively insensitive to small changes in cut-off grade.

TABLE 14.11
PIT-CONSTRAINED OXIDE MINERAL RESOURCE SENSITIVITY (1)

Cut-off AuEq (g/t)	Tonnes (k)	Ag (g/t)	Ag (koz)	Au (g/t)	Au (koz)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)	AgEq (koz)	AuEq (g/t)	AuEq (koz)
Measured												
1.00	3,844	236	29,179	1.93	239	NA	NA	NA	580	71,720	3.35	414
0.90	4,509	204	29,532	1.77	257	NA	NA	NA	519	75,304	3.00	435
0.80	5,358	174	29,939	1.61	277	NA	NA	NA	461	79,400	2.66	458
0.70	6,584	144	30,387	1.44	305	NA	NA	NA	400	84,601	2.30	487
0.60	8,046	119	30,912	1.28	331	NA	NA	NA	348	89,981	2.00	517
0.50	9,877	99	31,493	1.14	362	NA	NA	NA	301	95,685	1.73	549
0.40	12,376	81	32,147	0.99	394	NA	NA	NA	256	102,042	1.47	585
0.30	15,732	65	32,953	0.84	425	NA	NA	NA	215	108,658	1.23	622
0.20	20,375	52	33,893	0.70	459	NA	NA	NA	176	115,162	1.01	662
0.08	27,732	39	35,034	0.54	609	NA	NA	NA	136	120,923	0.78	693
0.05	29,705	37	35,212	0.51	487	NA	NA	NA	127	121,642	0.73	697
Indicated												
1.00	1,186	105	4,019	1.55	59	NA	NA	NA	382	14,559	2.19	83
0.90	1,451	89	4,133	1.43	67	NA	NA	NA	343	15,994	1.96	91
0.80	1,767	75	4,253	1.31	74	NA	NA	NA	308	17,517	1.76	100
0.70	2,409	57	4,393	1.15	89	NA	NA	NA	261	20,240	1.49	115
0.60	3,219	44	4,601	1.01	105	NA	NA	NA	225	23,245	1.28	132
0.50	4,258	35	4,804	0.89	122	NA	NA	NA	193	26,479	1.10	151
0.40	5,814	27	5,042	0.76	142	NA	NA	NA	163	30,442	0.93	174
0.30	8,434	20	5,399	0.63	171	NA	NA	NA	131	35,586	0.74	201
0.20	12,530	15	5,878	0.49	197	NA	NA	NA	103	41,354	0.58	234
0.08	19,804	10	6,593	0.36	75	NA	NA	NA	74	46,979	0.42	266
0.05	21,853	10	6,745	0.33	232	NA	NA	NA	68	47,726	0.39	274

TABLE 14.11
PIT-CONSTRAINED OXIDE MINERAL RESOURCE SENSITIVITY [1]

Cut-off AuEq (g/t)	Tonnes (k)	Ag (g/t)	Ag (koz)	Au (g/t)	Au (koz)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)	AgEq (koz)	AuEq (g/t)	AuEq (koz)
Measured and Indicated												
1.00	5,030	205	33,198	1.84	298	NA	NA	NA	534	86,279	3.08	498
0.90	5,960	176	33,665	1.69	323	NA	NA	NA	476	91,298	2.75	526
0.80	7,126	149	34,191	1.54	352	NA	NA	NA	423	96,918	2.44	558
0.70	8,993	120	34,780	1.36	394	NA	NA	NA	363	104,841	2.08	602
0.60	11,266	98	35,513	1.20	436	NA	NA	NA	313	113,226	1.79	650
0.50	14,135	80	36,297	1.06	484	NA	NA	NA	269	122,164	1.54	700
0.40	18,190	64	37,188	0.92	536	NA	NA	NA	227	132,484	1.30	759
0.30	24,167	49	38,352	0.77	596	NA	NA	NA	186	144,243	1.06	823
0.20	32,905	38	39,771	0.62	656	NA	NA	NA	148	156,515	0.85	895
0.08	47,466	27	41,622	0.47	710	NA	NA	NA	110	167,871	0.63	962
0.05	51,557	25	41,957	0.43	719	NA	NA	NA	102	169,368	0.59	971
Inferred												
1.00	481	53	819	1.27	20	NA	NA	NA	279	4,323	1.59	25
0.90	579	47	878	1.2	22	NA	NA	NA	261	4,853	1.48	28
0.80	640	44	911	1.16	24	NA	NA	NA	250	5,149	1.42	29
0.70	752	39	945	1.09	26	NA	NA	NA	232	5,621	1.32	32
0.60	890	35	1,009	1.01	29	NA	NA	NA	214	6,131	1.22	35
0.50	1,056	31	1,048	0.93	32	NA	NA	NA	196	6,647	1.11	38
0.40	1,357	25	1,105	0.81	35	NA	NA	NA	170	7,414	0.96	42
0.30	1,882	20	1,180	0.67	41	NA	NA	NA	140	8,446	0.79	48
0.20	2,881	14	1,285	0.52	48	NA	NA	NA	106	9,846	0.60	56
0.08	4,475	10	1,453	0.38	18	NA	NA	NA	77	11,045	0.44	63
0.05	4,999	9	1,490	0.34	55	NA	NA	NA	70	11,239	0.40	64

Note: *[1] See Table 14.9 notes for assumptions.*

TABLE 14.12
PIT-CONSTRAINED SULPHIDE MINERAL RESOURCE SENSITIVITY (1)

Cut-off AuEq (g/t)	Tonnes (k)	Ag (g/t)	Ag (koz)	Au (g/t)	Au (koz)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)	AgEq (koz)	AuEq (g/t)	AuEq (koz)
Measured												
1.00	738	225	5,337	1.90	45	NA	NA	NA	453	10,731	3.70	88
0.90	872	192	5,382	1.74	49	NA	NA	NA	400	11,222	3.27	92
0.80	1,058	160	5,427	1.57	53	NA	NA	NA	348	11,833	2.85	97
0.70	1,335	128	5,488	1.39	60	NA	NA	NA	294	12,630	2.41	103
0.60	1,739	100	5,583	1.20	67	NA	NA	NA	244	13,646	2.00	112
0.50	2,233	79	5,675	1.05	75	NA	NA	NA	205	14,699	1.68	121
0.40	2,822	63	5,755	0.92	83	NA	NA	NA	173	15,721	1.42	129
0.30	3,612	50	5,844	0.79	92	NA	NA	NA	145	16,787	1.19	138
0.20	4,754	39	5,949	0.65	99	NA	NA	NA	117	17,877	0.96	147
0.12	6,031	31	6,042	0.54	105	NA	NA	NA	96	18,667	0.79	153.6
0.05	7,142	27	6,094	0.47	108	NA	NA	NA	83	19,032	0.68	156
Indicated												
1.00	186	95	571	1.26	8	NA	NA	NA	247	1,476	2.02	12
0.90	234	77	581	1.19	9	NA	NA	NA	220	1,650	1.80	14
0.80	298	62	596	1.10	11	NA	NA	NA	194	1,863	1.60	15
0.70	396	48	615	1.00	13	NA	NA	NA	169	2,143	1.39	18
0.60	501	40	637	0.92	15	NA	NA	NA	150	2,408	1.23	20
0.50	636	33	665	0.83	17	NA	NA	NA	132	2,696	1.09	22
0.40	805	26	686	0.74	19	NA	NA	NA	116	2,991	0.95	25
0.30	1,012	22	705	0.66	21	NA	NA	NA	100	3,270	0.83	27
0.20	1,341	17	729	0.55	24	NA	NA	NA	83	3,581	0.69	30
0.12	1,723	14	756	0.46	25	NA	NA	NA	69	3,817	0.57	32
0.05	2,105	11	774	0.39	26	NA	NA	NA	58	3,945	0.48	32

TABLE 14.12
PIT-CONSTRAINED SULPHIDE MINERAL RESOURCE SENSITIVITY [1]

Cut-off AuEq (g/t)	Tonnes (k)	Ag (g/t)	Ag (koz)	Au (g/t)	Au (koz)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)	AgEq (koz)	AuEq (g/t)	AuEq (koz)
Measured and Indicated												
1.00	924	199	5,908	1.77	53	NA	NA	NA	411	12,206	3.36	100
0.90	1,106	168	5,963	1.62	58	NA	NA	NA	362	12,872	2.96	105
0.80	1,356	138	6,022	1.47	64	NA	NA	NA	314	13,696	2.57	112
0.70	1,730	110	6,103	1.30	72	NA	NA	NA	266	14,774	2.18	121
0.60	2,240	86	6,221	1.14	82	NA	NA	NA	223	16,054	1.83	132
0.50	2,869	69	6,341	1.00	92	NA	NA	NA	189	17,395	1.55	143
0.40	3,627	55	6,441	0.88	103	NA	NA	NA	160	18,712	1.32	153
0.30	4,624	44	6,549	0.76	113	NA	NA	NA	135	20,056	1.11	165
0.20	6,096	34	6,678	0.63	123	NA	NA	NA	109	21,458	0.90	176
0.12	7,755	27	6,798	0.52	131	NA	NA	NA	90	22,484	0.74	185
0.05	9,246	23	6,868	0.45	134	NA	NA	NA	77	22,977	0.63	189
Inferred												
1.00	127	242	984	0.73	3	NA	NA	NA	329	1,339	2.66	11
0.90	149	208	996	0.74	4	NA	NA	NA	296	1,420	2.40	12
0.80	194	161	1,006	0.75	5	NA	NA	NA	251	1,568	2.04	13
0.70	241	134	1,036	0.72	6	NA	NA	NA	220	1,704	1.79	14
0.60	272	121	1,055	0.69	6	NA	NA	NA	204	1,783	1.66	14
0.50	306	109	1,075	0.66	6	NA	NA	NA	189	1,857	1.53	15
0.40	341	99	1,085	0.63	7	NA	NA	NA	175	1,917	1.42	16
0.30	374	91	1,089	0.60	7	NA	NA	NA	163	1,961	1.33	16
0.20	427	80	1,094	0.56	8	NA	NA	NA	146	2,011	1.19	16
0.12	463	74	1,096	0.52	8	NA	NA	NA	136	2,034	1.11	17
0.05	496	69	1,098	0.50	8	NA	NA	NA	128	2,045	1.05	17

Note: *[1] See Table 14.9 notes for assumptions.*

TABLE 14.13
OUT-OF-PIT MINERAL RESOURCE SENSITIVITY (1)

Cut-off AuEq (g/t)	Tonnes (k)	Ag (g/t)	Ag (koz)	Au (g/t)	Au (koz)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)	AgEq (koz)	AuEq (g/t)	AuEq (koz)
Measured												
2.00	952	474	14,530	1.10	34	0.13	0.55	0.95	616	18,865	7.42	227
1.90	961	471	14,561	1.09	34	0.13	0.55	0.94	612	18,909	7.37	228
1.80	970	468	14,589	1.08	34	0.13	0.55	0.94	608	18,952	7.32	228
1.70	977	465	14,612	1.07	34	0.13	0.55	0.94	604	18,988	7.28	229
1.60	986	462	14,636	1.07	34	0.13	0.54	0.93	600	19,024	7.23	229
1.50	994	459	14,660	1.06	34	0.13	0.54	0.93	597	19,057	7.18	229
1.40	1,001	456	14,676	1.05	34	0.13	0.54	0.92	593	19,085	7.14	230
1.28	1,010	453	14,698	1.05	34	0.13	0.54	0.92	589	19,118	7.09	230
1.20	1,016	450	14,710	1.04	34	0.13	0.54	0.92	586	19,138	7.05	230
1.10	1,023	447	14,724	1.04	34	0.13	0.53	0.91	582	19,161	7.01	231
Indicated												
2.00	1,305	414	17,374	1.09	46	0.08	0.25	0.42	528	22,172	6.36	267
1.90	1,317	411	17,416	1.08	46	0.08	0.25	0.42	525	22,233	6.32	268
1.80	1,328	409	17,454	1.08	46	0.08	0.24	0.42	522	22,289	6.28	268
1.70	1,340	406	17,491	1.07	46	0.08	0.24	0.42	519	22,344	6.25	269
1.60	1,351	404	17,524	1.07	46	0.08	0.24	0.42	516	22,391	6.21	270
1.50	1,361	401	17,554	1.06	46	0.08	0.24	0.42	513	22,433	6.17	270
1.40	1,368	400	17,572	1.06	47	0.08	0.24	0.42	511	22,459	6.15	270
1.28	1,374	398	17,588	1.05	47	0.08	0.24	0.41	509	22,482	6.13	271
1.20	1,378	397	17,598	1.05	47	0.08	0.24	0.41	508	22,496	6.11	271
1.10	1,384	396	17,611	1.05	47	0.08	0.24	0.41	506	22,514	6.09	271

TABLE 14.13
OUT-OF-PIT MINERAL RESOURCE SENSITIVITY (1)

Cut-off AuEq (g/t)	Tonnes (k)	Ag (g/t)	Ag (koz)	Au (g/t)	Au (koz)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)	AgEq (koz)	AuEq (g/t)	AuEq (koz)
Measured and Indicated												
2.00	2,258	440	31,903	1.09	79	0.10	0.38	0.64	565	41,037	6.81	494
1.90	2,278	437	31,977	1.08	79	0.10	0.38	0.64	562	41,142	6.76	495
1.80	2,298	434	32,043	1.08	80	0.10	0.37	0.64	558	41,241	6.72	496
1.70	2,317	431	32,103	1.07	80	0.10	0.37	0.64	555	41,331	6.68	498
1.60	2,336	428	32,160	1.07	80	0.10	0.37	0.64	551	41,415	6.64	499
1.50	2,355	426	32,214	1.06	80	0.10	0.37	0.64	548	41,491	6.60	499
1.40	2,368	423	32,248	1.06	80	0.10	0.37	0.63	546	41,544	6.57	500
1.28	2,384	421	32,286	1.05	81	0.10	0.37	0.63	543	41,600	6.54	501
1.20	2,395	420	32,308	1.05	81	0.10	0.37	0.63	541	41,634	6.51	501
1.10	2,408	418	32,335	1.05	81	0.10	0.36	0.62	538	41,674	6.48	502
Inferred												
2.00	3,921	303	38,252	1.63	206	0.34	0.78	0.93	514	64,744	6.2	782
1.90	3,953	302	38,375	1.62	206	0.33	0.78	0.92	511	64,910	6.17	784
1.80	3,991	300	38,509	1.61	207	0.33	0.77	0.91	507	65,096	6.13	787
1.70	4,024	298	38,615	1.60	207	0.33	0.76	0.9	504	65,249	6.09	788
1.60	4,045	297	38,674	1.60	208	0.33	0.76	0.9	502	65,343	6.07	789
1.50	4,063	296	38,727	1.59	208	0.32	0.76	0.9	501	65,418	6.05	790
1.40	4,085	295	38,784	1.58	208	0.32	0.75	0.89	499	65,502	6.02	791
1.28	4,100	294	38,822	1.58	208	0.32	0.75	0.89	497	65,556	6.01	792
1.20	4,107	294	38,837	1.58	209	0.32	0.75	0.89	497	65,580	6.00	792
1.10	4,118	294	38,861	1.57	208	0.32	0.75	0.89	496	65,613	5.99	793

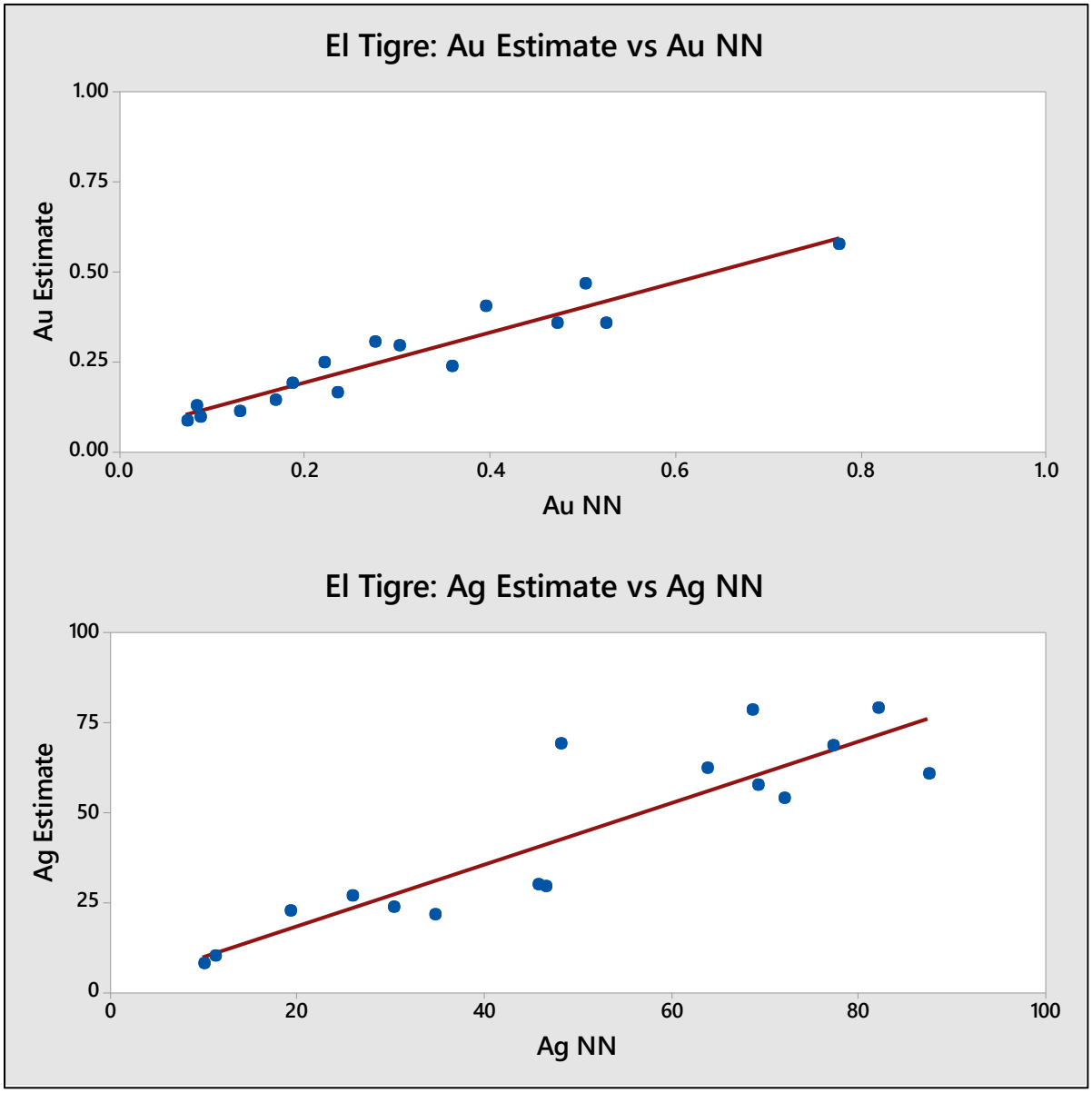
Note: *[1] See Table 14.9 notes for assumptions.*

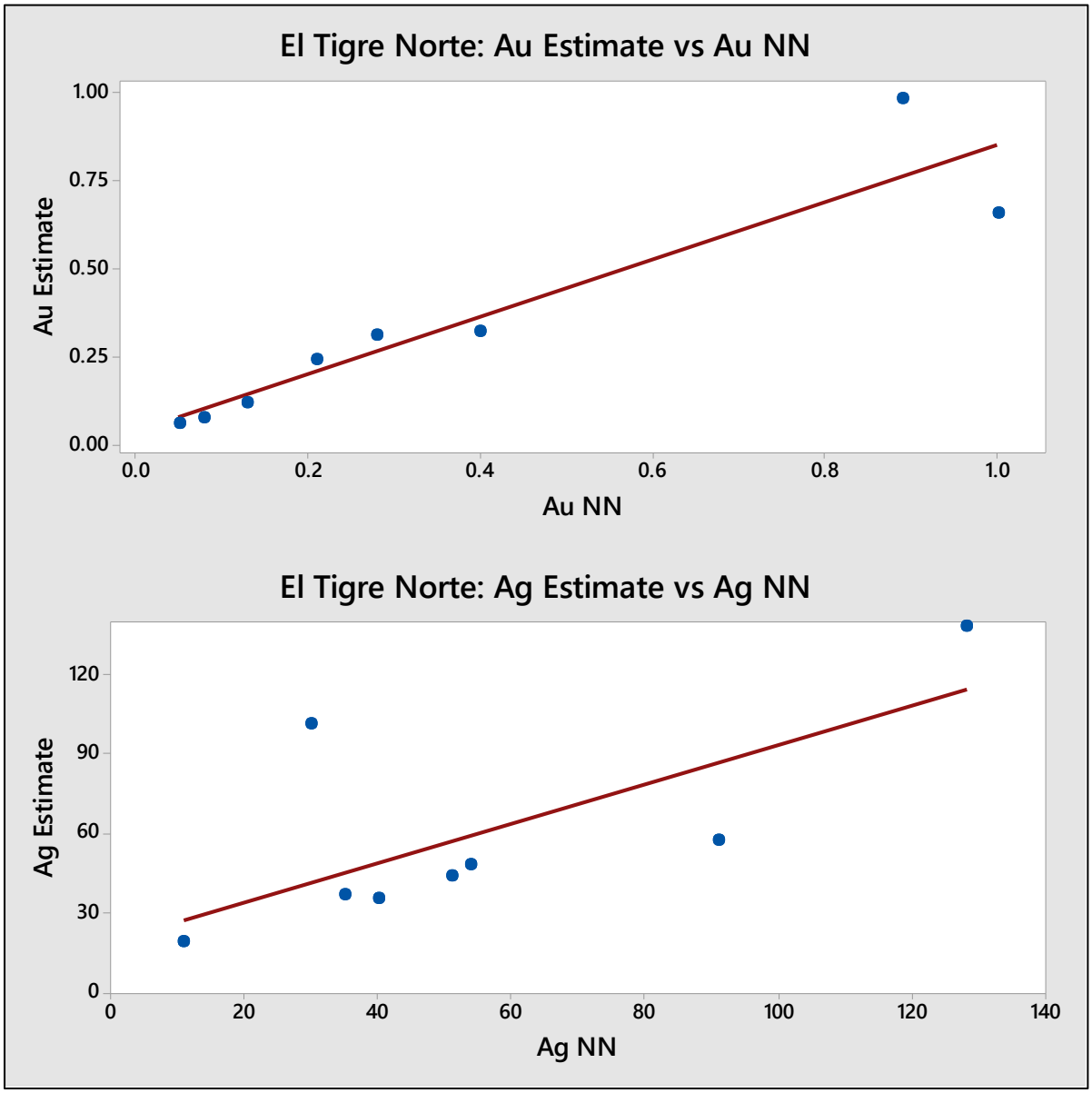
14.12 VALIDATION

The block models were validated visually by the inspection of successive cross-sections in order to confirm that the models correctly reflect the distribution of high-grade and low-grade values.

The average estimated block grades were compared to the average Nearest Neighbour block grade at a 0.001 g/t AuEq cut-off (Figure 14.4). The results fall within acceptable limits for linear grade estimation of partial block models (Table 14.14).

FIGURE 14.4 EL TIGRE NN CORRELATION





Source: *P&E (2024)*

TABLE 14.14 COMPARISON OF ID^3 BLOCK ESTIMATE AND NN AVERAGE BLOCK GRADES						
Domain	**Ag (g/t)**	**Ag NN (g/t)**	**ID^3/NN for Ag (%)**	**Au (g/t)**	**Au NN (g/t)**	**ID^3/NN for Au (%)**
El Tigre	61	87	70	0.36	0.52	68
East Tigre	23	19	118	0.30	0.28	110
West Tigre	27	26	103	0.24	0.36	66
BS Tigre	29	47	63	0.14	0.17	84
Sooy	62	64	97	0.11	0.13	84
East Sooy	8	10	77	0.57	0.78	74
Angi	59	48	122	0.12	0.08	152
BS Sooy	78	69	113	0.08	0.07	117
Seitz-Kelly	54	72	75	0.36	0.47	76
East Seitz-Kelly 2	79	82	96	0.19	0.19	101
West Seitz-Kelly	24	30	78	0.40	0.39	103
BS Seitz-Kelly	30	46	65	0.25	0.22	112
East Seitz-Kelly	21	35	62	0.47	0.50	93
Protectora	101	30	337	0.31	0.28	111
Fundadora	48	54	89	0.66	1.00	66
Caleigh	138	128	108	0.98	0.89	110
Benjamin	37	35	106	0.06	0.05	120
BS Benjamin	57	91	63	0.08	0.08	100
Aquila	44	51	86	0.32	0.40	80
Esc	35	40	88	0.12	0.13	92
Protectora Halo	19	11	173	0.24	0.21	114
Sulphide	56	69	81	0.15	0.23	65
Stockwork	10	11	91	0.28	0.30	93
Stockpile 1	177	177	NA	0.90	0.90	NA
Stockpile 2	146	146	NA	0.46	0.46	NA
Grey	79	80	99	0.30	0.31	97
Red	82	82	100	0.29	0.29	100
Yellow	78	80	98	0.27	0.27	100

Note: *ID^3 = Inverse Distance Cubed, NN = Nearest Neighbour*

The volume estimated was also checked against the reported volume of the individual mineralized veins. Estimated volumes are combined partial block volumes at 0.001 g/t AuEq cut-off (Table 14.15). The results fall within acceptable limits for linear grade estimation of partial models.

TABLE 14.15 VOLUME COMPARISON		
Domain	**Wireframe Volume ($k\text{-}m^3$)**	**Estimated Volume ($k\text{-}m^3$)**
North Veins	5,689	5,060
Main Veins	45,145	45,021
Stockpile	80	80
Tailings	650	650
Total	**51,564**	**50,811**

15.0 MINERAL RESERVE ESTIMATE

The El Tigre Project will consist of single large surface mining operation. While underground mining may be a future option, the Mineral Reserves stated herein are based solely on a surface scenario. The Mineral Reserve Estimate is summarized in Table 15.1.

The Mineral Reserve is the portion of the Mineral Resource that has been deemed economically mineable by the surface mine production plan and within a designed surface mine. The Mineral Reserve consists only of the portion of the Mineral Resource classified as Measured or Indicated. The Inferred Mineral Resource is considered as waste material in this PFS.

The surface mine design process is described in Section 15.1.

TABLE 15.1
MINERAL RESERVE SUMMARY (1-7)

Classification	Ore Tonnes (k)	Au (g/t)	Ag (g/t)	AuEq (g/t)	AgEq (g/t)	Contained Au (koz)	Contained Ag (koz)
Proven							
Oxide	19,564	0.40	16.4	0.50	83.4	252	10,316
Sulphide	4,655	0.52	7.6	0.58	65.2	78	1,137
Total Proven	**24,220**	**0.43**	**14.7**	**0.52**	**79.9**	**329**	**11,453**
Probable							
Oxide	14,800	0.37	13.7	0.45	74.3	176	6,519
Sulphide	1,272	0.43	33.8	0.73	82.1	18	1,382
Total Probable	**16,073**	**0.37**	**15.3**	**0.47**	**74.9**	**194**	**7,901**
Proven and Probable							
Oxide	34,365	0.39	15.2	0.48	79.5	428	16,834
Sulphide	5,928	0.50	13.2	0.61	68.8	95	2,520
Total Proven & Probable	**40,292**	**0.40**	**14.9**	**0.50**	**77.9**	**523**	**19,354**

Notes:

1. *The Mineral Reserve Estimate was prepared in accordance with 2014 CIM Definition Standards by Andrew Bradfield, P.Eng.*
2. *The effective date of the Mineral Reserve Estimate is October 22, 2024.*
3. *Au price of US$1,900/oz and Ag price of US$23/oz are assumed.*
4. *The Mineral Reserves are based on the surface mine production schedule.*
5. *Mineral Reserves include a combined ore loss and dilution of 2.9%.*
6. *Oxide and Sulphide Mineral Reserves are reported using a cut-off grade of 0.11 g/t AuEq and 0.16 g/t AuEq, respectively. Ore and waste rock mining cost of $2.60/t was assumed, with a process cost of $4.33/t (assumes 15,000 tpd production), G&A cost of $1.00/t, a royalty of 0.5% of revenue, and pit slopes ranging from 46 to 49 degrees. Heap leach recoveries were Oxide 83% Au and 45% Ag, Sulphide 56% Au and 40% Ag.*
7. *Rounding of some figures may lead to minor discrepancies in total.*

15.1 SURFACE MINE DESIGN

The upper portion of the El Tigre Deposit is near surface and lends itself to conventional surface mining methods. A single surface mine will be developed, although it will be subdivided into three Phases for production scheduling efficiency.

The design of the mine required several steps, as listed below:

- Create a diluted block model from the Mineral Resource model;
- Complete pit optimization to select the optimal pit shell to be used for the mine design;
- Design an operational surface mine (with ramps and benches) based on the optimal pit shell; and
- Develop a life-of-mine ("LOM") production schedule, based on supplying up to 15,000 tonnes of ore per day (5.5 million ore tonnes per year) to a heap leach facility.

15.1.1 Diluted Block Model

Dilution and ore losses will occur during mining. It is assumed that some waste rock surrounding the ore zones will be mixed with the ore during mining, thereby causing dilution.

To address dilution, the 2.5 m x 5 m x 5 m (X-Y-Z) Mineral Resource block model was regularized from a Geovia GEMS™ volume percent model to a regularized whole block selective mining unit ("SMU") model. This process resulted in waste and ore portions of a block being combined into a single block value. In some cases, the resulting block grade may be below the applied cut-off grade, in which case that block would be considered an "ore loss". In other cases, while the block grade may decrease due to the waste dilution, the entire block grade may remain above the cut-off grade resulting in an increase in ore tonnage albeit at a lower head grade. Hence the SMU model is considered a diluted model and was used for optimization, production scheduling, and is the basis for the Mineral Reserve Estimate.

The amount of dilution incorporated by the regularization process varies depending on the block size selectivity and the vein width. Table 15.2 compares the overall undiluted and diluted tonnes and grade and summarizes the calculated net dilution. The "net" dilution incorporates both the total dilution and ore loss, and is approximately 2.9%.

TABLE 15.2
NET DILUTION
(COMBINES DILUTION AND ORE LOSS)

Type	Ore (kt)	Au (g/t)	Ag (g/t)	AuEq (g/t)	AgEq (g/t)
Undiluted	39,158	0.41	15.3	0.51	80.0
Diluted	40,292	0.40	14.9	0.50	77.9
Dilution (%)	2.9	2.7	2.5	2.6	2.6

15.1.2 Surface Mine Optimization Inputs

A series of pit optimizations were completed on the diluted block model using Datamine NPV Scheduler™ software. The optimization process produces a series of nested pit shells each containing ore that is economically mineable according to a given set of physical and economic parameters.

The pit optimizations were undertaken using the parameters shown in Table 15.3.

For the pit optimization, a base case gold price of $US1,900/oz and a silver price of US$23.00/oz were used. The pit optimization analysis included only Measured and Indicated Mineral Resources as process plant feed. Inferred Mineral Resources were considered waste material in the pit optimization. Revenue factors from 2 to 110% were applied in the optimization, with the base case metal prices representing the 100% revenue factor.

Two ore types will be encountered in the surface mine: oxide ore and sulphide ore. The oxide ore ("OX") and the sulphide ore ("SU") will have differing gold and silver process plant metal recoveries. Hence each ore type will have a slightly different economic cut-off grade. The leaching recovery assumptions and cut-off grades are outlined in Table 15.3.

Although Table 15.3 outlines optimization criteria for both 7,500 and 15,000 tpd, the latter optimization result was used to select the optimal shell. This is since 15,000 tpd is the planned production rate when the surface mine would be developed outwards to the final pit walls.

TABLE 15.3
SURFACE MINE OPTIMIZATION PARAMETERS

Parameter	Units	7,500 tpd	15,000 tpd
Classifications		M & I only	M & I only
Gold Price (less refining, payable)	US$/oz	1,900	1,900
Silver Price	US$/oz	23.00	23.00
Government Royalty	%	0.5	0.5
Operating Costs			
Mining - Waste Cost	$/t	2.60	2.60
Mining - Ore Cost	$/t	2.60	2.60
Heap Leaching Cost (OX)	$/t	5.59	4.33
Heap Leaching Cost (SU)	$/t	5.59	4.33
G&A + Mining G&A	$/t	2.01	1.00
Heap Leach Recovery			
OX Gold recovery	%	83	83
OX Silver recovery	%	45	45
SU Gold recovery	%	56	56
SU Silver recovery	%	40	40
Cut-off Grades			

TABLE 15.3 SURFACE MINE OPTIMIZATION PARAMETERS			
Parameter	**Units**	**7,500 tpd**	**15,000 tpd**
OX Incremental Operating Cost	$/t	7.60	5.33
OX Cut-off Grade	AuEq	0.15	0.11
SU Incremental Operating Cost	$/t	7.60	5.33
SU Cut-off Grade	AuEq	0.22	0.16
Optimization Slope Angles		45 deg	45 deg

Note: *OX = oxide ore type, SU = sulphide ore type.*

15.1.3 Pit Optimization Results

The results of the pit optimization are shown in Figure 15.1. The NPV curve starts to flatten off above a revenue factor ("RF") of ~80%. To maximize ore recovered, the revenue factor 87% shell was selected as the basis for the El Tigre surface mine design.

Figure 15.2 is a chart showing the ore and waste tonnes with increasing revenue factor. At the higher revenue factors the rate of increase in waste tonnage is faster that rate of increase in ore tonnes. This indicates stripping ratios will increase rapidly in larger pit shells.

FIGURE 15.1 PIT OPTIMIZATION NPV CURVE

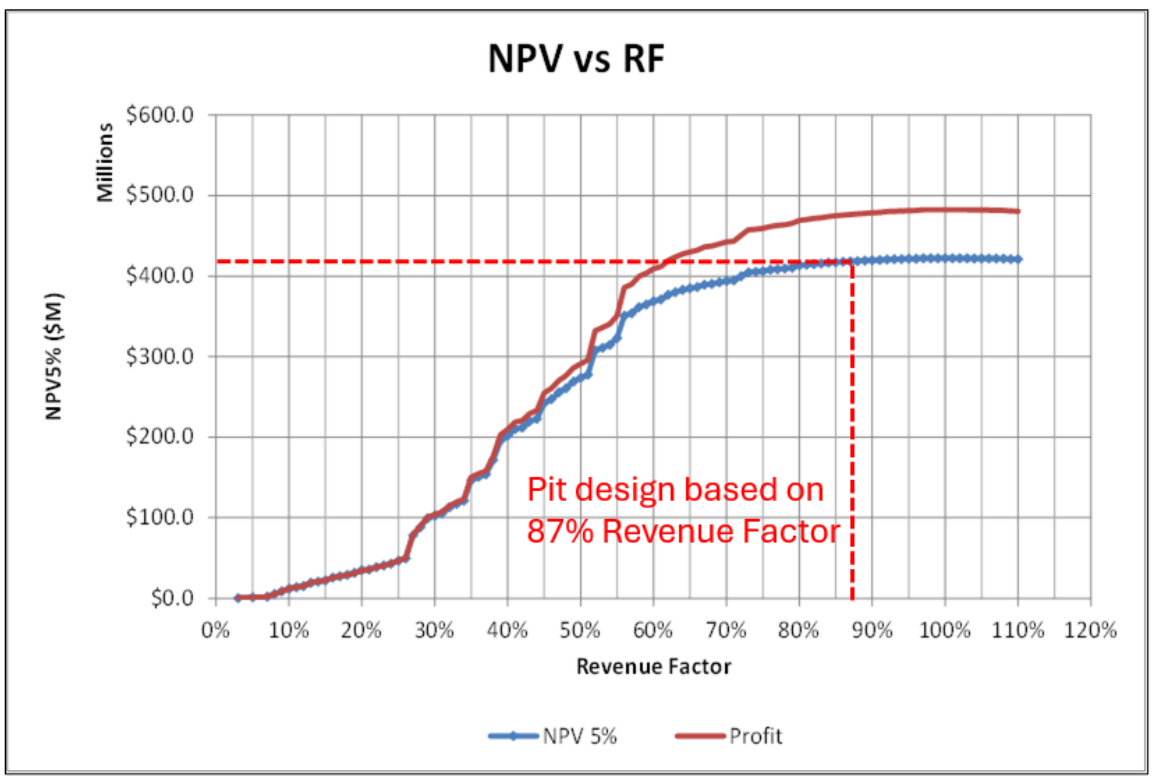
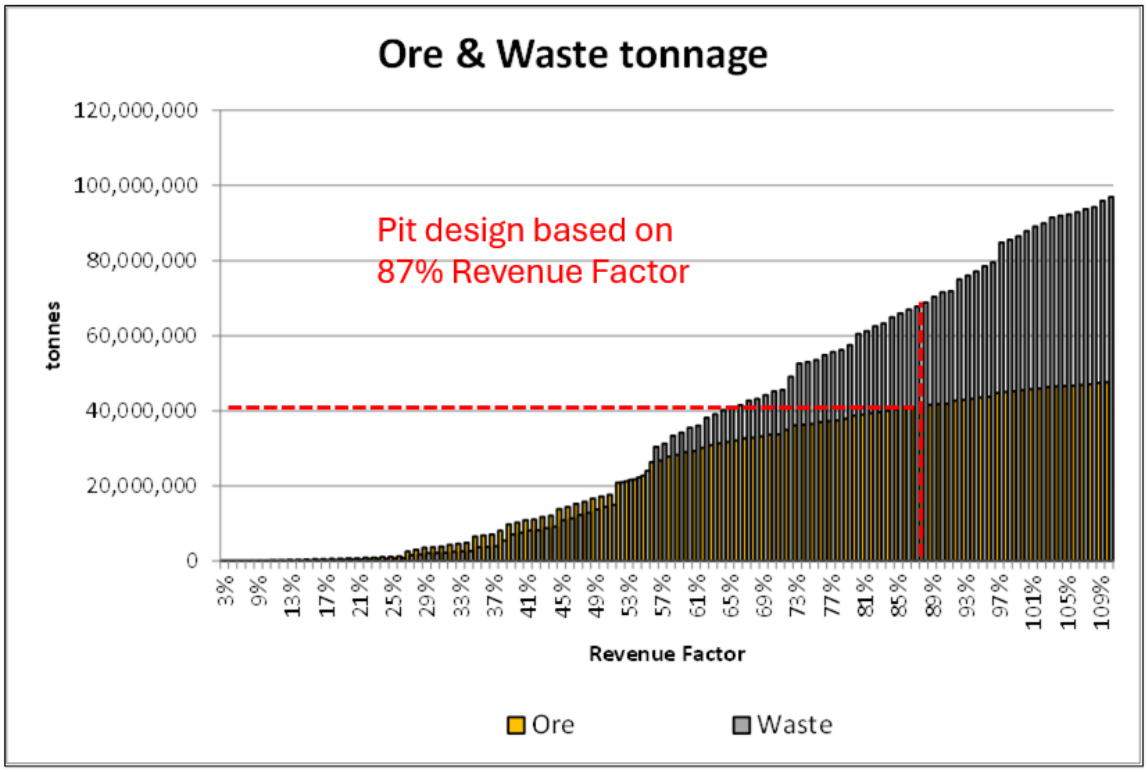


FIGURE 15.2 PIT OPTIMIZATION ORE AND WASTE TONNES



15.1.4 Surface Mine Design

The El Tigre surface mine was designed using the selected RF 87% optimized shell as a guide for sizing the pit.

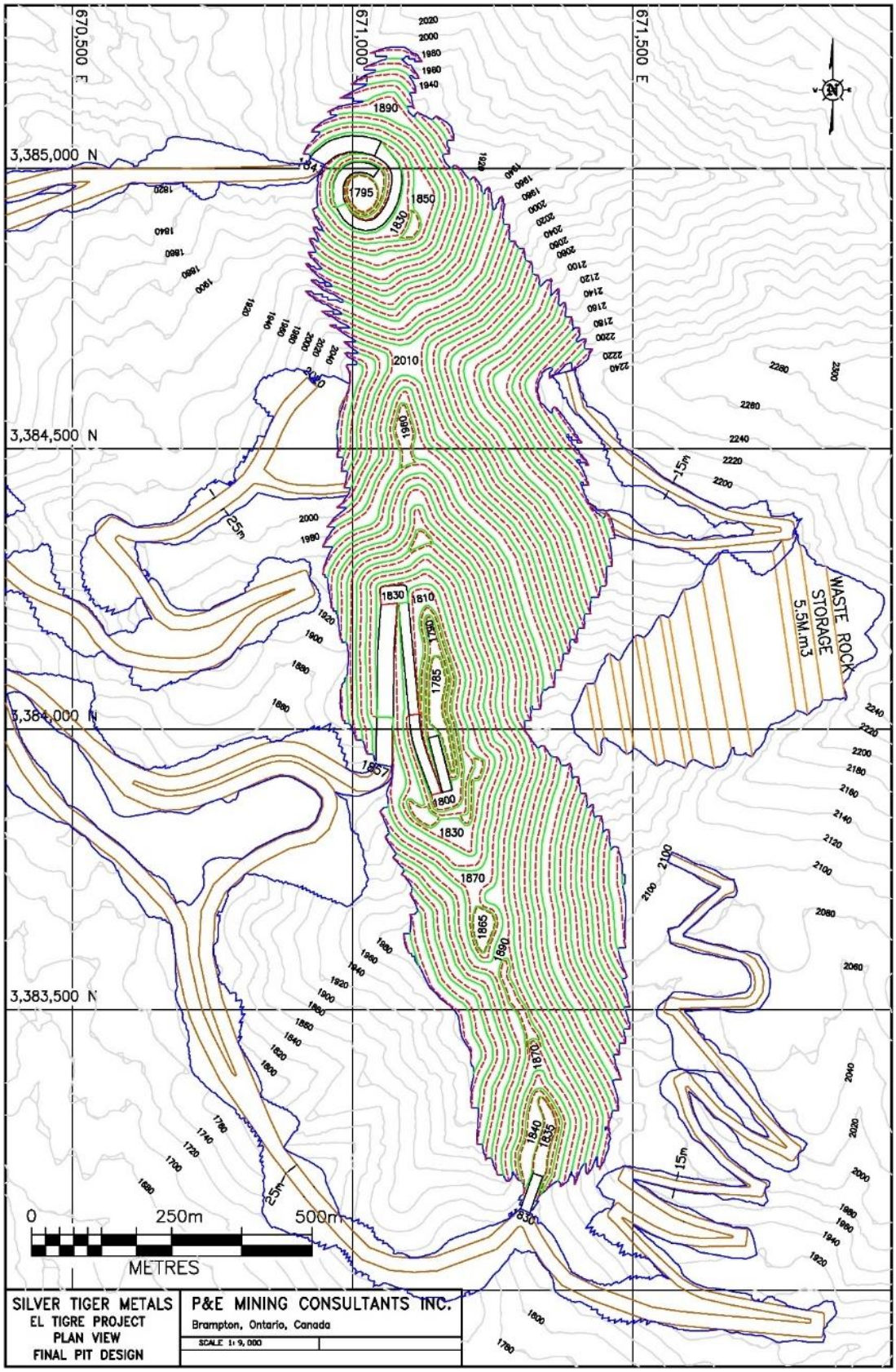
The design examined potential access points along the pit periphery nearest to the crushing plant, and then added benches, berms and haul ramps accordingly.

A bench height of 5 m was used with a final configuration of quadruple benches (20 m). Inter-ramp pit slope angles ranged from 46 to 49 degrees as described in section 15.2 of this Technical Report.

The surface mine design used an internal haul ramp width of 30 m for double lane traffic and 19 m for single lane traffic. Single lane haul roads were used in the bottom benches of the pits to minimize the addition of excess waste rock from expanding the pit walls outwards more than required.

The surface mine design is shown in Figure 15.3, and is planned to be mined in three phases, as described in Section 16.

FIGURE 15.3 SURFACE MINE DESIGN

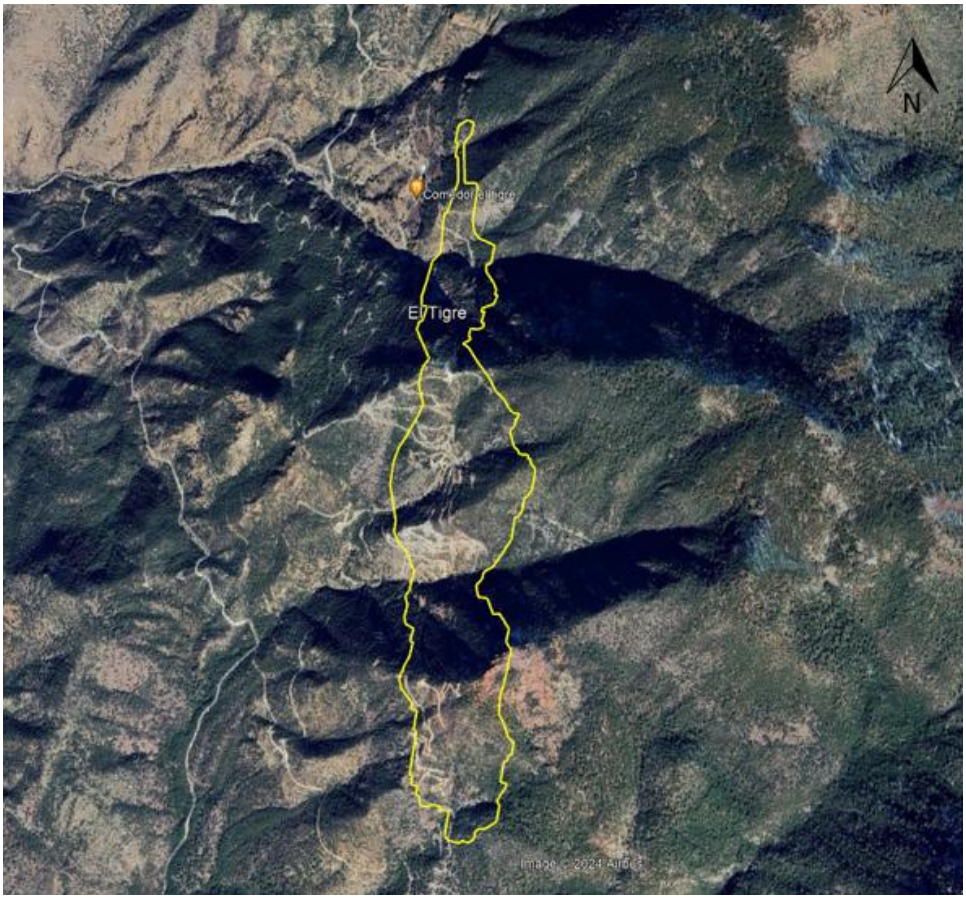


15.2 GEOTECHNICAL AND HYDROGEOLOGICAL STUDIES

The surface mine slope criteria and hydrogeological data assessments used in this PFS were provided by WSP in a report dated November 13, 2024 and titled "El Tigre Open Pit Prefeasibility Report" and are summarized below.

This Technical Report section contains a summary of the open pit slope geometry design recommendations based on the interpretation of geotechnical and hydrogeological character of the Project site. The interpretation follows a review of available and recently collected geotechnical and hydrogeological data, assessment of this data, development of rock mass characteristics at the El Tigre Property, and stability assessments and assignment of slope geometry recommendations to different design sectors in the proposed open pit (Figure 15.4).

FIGURE 15.4 EL TIGRE PROJECT OPEN PIT LAYOUT



Source: *Google Earth (2024)*
Note: *not to scale.*

15.2.1 Available Data

Geotechnical and hydrogeological data used to support this PFS includes historical datasets, interpretations prepared by Silver Tiger, and investigation work specifically planned to capture data for the PFS. The main data used for this PFS includes a geotechnical drill hole investigation

that was performed by Silver Tiger staff between November 8, 2023, and January 22, 2024. Fourteen HQ-3 drill holes totalling 2,587.6 m were completed and logged with drill core orientation. In addition to the PFS geotechnical drilling program, vibrating wire piezometer instrumentation was installed, and a geophysics campaign consisting of Acoustic Televiewer survey ("ATV") was completed. No geotechnically logged drill holes were surveyed with ATV.

15.2.2 Engineering Geology Model

The Engineering Geology Model ("EGM") examines the constituent components of rock mass characteristics to develop an understanding of potential rock mass behaviour during mining and to estimate rock mass geotechnical and hydrogeological parameters for interpretation and assessment. The EGM results in input parameters for rock mass strength and structural conditions for use in open pit design assessments.

15.2.3 Intact Rock Strength

Intact rock strength was measured by Point Load Tests ("PLTs"), uniaxial compressive strength ("UCS") tests, triaxial compressive strength ("TCS") tests, and indirect tensile strength ("ITS") tests. Estimates of intact strength are also captured during logging (ISRM field strength estimates). While the PLTs and field strength estimates can help understand strength variability, the laboratory strength testing was used to develop Hoek-Brown failure criteria for use as inputs for the slope stability assessments.

Hoek-Brown curves were developed for each lithological unit, for a hypothetical unit which combined every lithological unit, and for weathered rock mass (defined as the first 30 m of bedrock) (Table Table 15.415.4). The results of the intact strength interpretation suggest that the intact strengths vary from 52 to 113 MPa for σ_{ci} (unconfined compressive strength) and when every sample is combined the σ_{ci} is 107 MPa. While the σ_{ci} shows some variation, the UCS values are similar suggesting that the rock masses with higher σ_{ci} are likely due to relatively higher failure stresses for their triaxial testing specimen. The Lithic Tuff and Quartz Rhyolite lithological units represent the weakest intact strengths, and both are similar to the strength of the weathered ground.

TABLE 15.4
HOEK-BROWN STRENGTH PARAMETERS FOR INTACT ROCK BY LITHOLOGY

Lithology	No. of UCS Samples	No. of BTS Samples	No. of TCS Samples	Average UCS (MPa)	Hoek-Brown (Hoek et al., 2002)	
					Mean m_i	Mean σ_{ci} (MPa)
Combined	37	39	35	56	17	107
Weathered Rock Mass – Top 30 m	13	14	11	47	16	65
Crystal Tuff	8	9	9	63	18	97
Lithic Tuff	11	11	9	53	17	60
Quartz Rhyolite	11	7	9	45	16	52
Rhyolitic Flow	7	8	8	69	19	113

Note: *No. = number, MPa = megapascal, m_i = Hoek-Brown constant, σ_{ci} = unconfined compressive strength. Number of samples reflects those laboratory specimens whose failure type was identified as 'Homogeneous' or 'Combined'.*

15.2.4 Rock Mass Classification

An assessment of the rock mass quality was completed using the Rock Mass Rating (RMR_{76}, Bieniawski, 1976) rock mass classification system. A summary the RMR_{76} rock mass classification is shown in Table 15.5. This summary only includes lithologies in which more than 60 m of geotechnical drill core was logged, with the exception of Quartz Rhyolite, since it has associated intact rock strength testing. For each lithological unit, the average values, standard deviations, and percentiles are all within 10 RMR_{76}. This suggests that the lithological units within the El Tigre Property have a similar classification and rock mass character.

TABLE 15.5
STATISTICS OF RMR_{76} BY LOGGED LITHOLOGICAL UNIT

Lithology	No. of Runs	Average	Std Dev	33rd Percentile	50th Percentile	66th Percentile
Crystal Tuff	222	59	25	60	68	77
Latite	37	54	29	21	66	73
Lithic Tuff	745	65	23	64	75	80
Rhyolite Lithic Tuff	126	59	24	58	67	73
Rhyolitic Flow	392	60	24	61	69	75
Quartz Rhyolite	19	56	26	60	68	77

Note: *Std Dev = standard deviation.*

15.2.5 Rock Mass Input Parameters for Geotechnical Domains

The lithological units intersected by geotechnical drilling (which are almost entirely within the El Tigre Formation, where the proposed open pit is planned) will be treated as a single geotechnical domain called the El Tigre Formation with a sub-domain for the near surface weathered rock masses. The two geotechnical domains which are considered for the stability assessments are described below:

- El Tigre Formation: This unit include various volcanic and intrusive units with similar strength and rock mass classifications, these are Lithic Tuff, Rhyolitic Flow, Crystal Tuff, Rhyolite Lithic Tuff and several other minor volcanic units. This domain covers the fresh, fair to good quality rock mass zones with GSI > 40 and UCS > 40 MPa. Due to the broad generalization of the rock masses at the El Tigre Property, conservative intact strength parameters were assumed (mi = 10 and σci = 43) to be below the mean values of every lithological unit.

- Weathered Rock Mass: This unit corresponds to weathered/altered and fractured rock mass with GSI < 40 and UCS < 40 MPa, in the geotechnical drill holes, this area was typically within the first 30 m below the bedrock contact. Conservative intact strength parameters were assumed for the first 30 m (mi = 9 and σci = 35).

Table 15.6 shows the summary of rock mass input parameters for use in the design assessments for each of the geotechnical domains. The GSI considered for the El Tigre Formation was the lowest RMR_{76} average value from the lithological units in Table 15.5, and the intact rock mass parameters from the bullets above were used. For the weathered rock mass, a GSI of 16 was applied based on the 30th percentile of the rock masses located in the upper 30 m.

TABLE 15.6
ROCK MASS PARAMETERS

Geotechnical Unit	Hoek-Brown (Hoek *et al.*, 2002)		
	GSI	m_i	σ_{ci} (MPa)
Weathered Rock Mass	16	9	35
El Tigre Formation	54	10	43

Note: *GSI = geological strength index, m_i = Hoek-Brown constant, σ_{ci} = unconfined compressive strength, MPa = megapascal.*

15.2.6 Structural Characteristics

15.2.6.1 Structural Set Identification

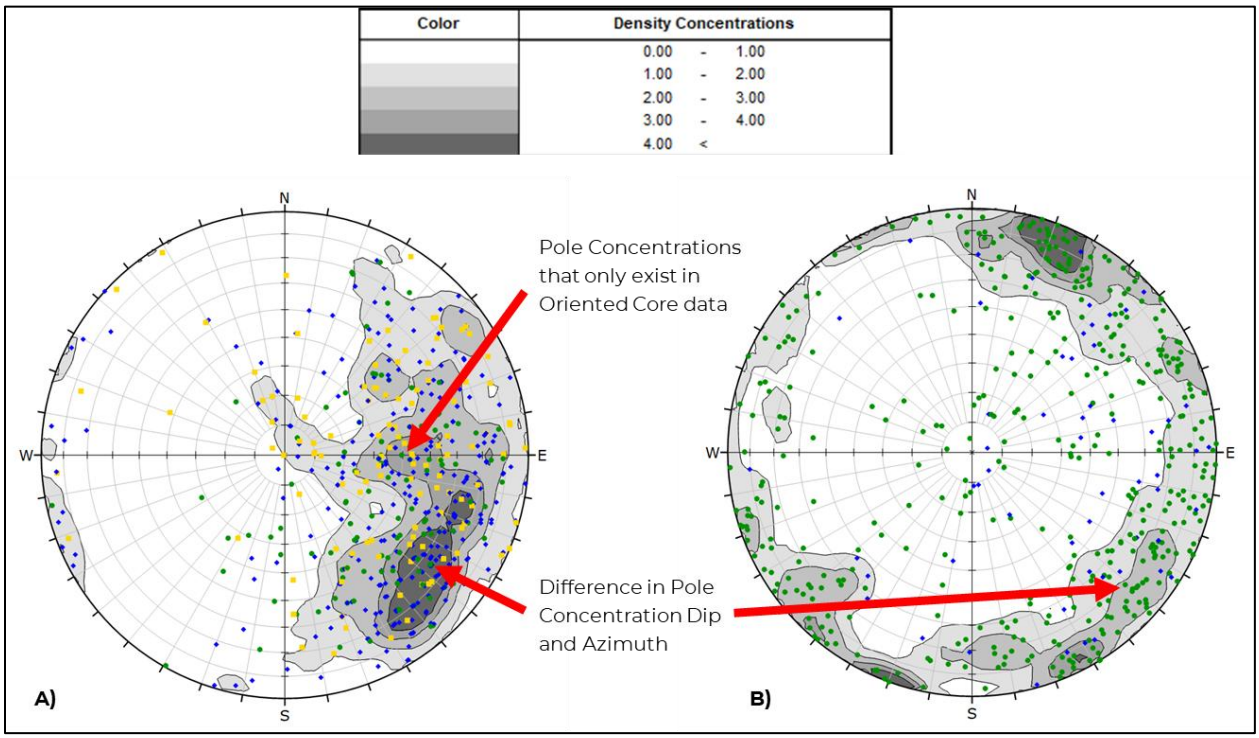
The oriented drill core and ATV data were considered, however, examined separately due to the differences in their recorded pole concentrations and drill holes in which their data was collected (televiewer was performed in exploration drill holes while drill core orientation was performed in geotechnical specific drill holes). Additionally, the televiewer drill holes were all completed along the same azimuth (trending 090°, directly East). The interpretation suggests the following:

- The structural sets are pervasive across logged lithology.
- The sets do have some variation across the site (± 20° azimuth and ± 15° dip), however, the variation cannot be spatially defined at this stage.
- There is variation, of up to 30°, between pole concentrations identified from oriented drill core and from the televiewer data, and there are pole concentrations identified in the oriented drill core that are not present in the televiewer dataset (Figure 15.5).
  - Generally, the televiewer data should have less uncertainty as it is a direct survey of the in-situ rock mass and open structures.
  - For the purposes of this assessment, the televiewer data was considered to be of higher confidence when defining set orientations where the set is present in both datasets.
- Two major structural sets (> 3% pole concentration density) were identified:
  - **Major Sub Vertical Set (JN1)**: This sub-vertical major joint set is present in all pit areas. The dip ranges from 60° to 88° and the dip direction ranges from 289° to 325°.

- **Major Sub Vertical Set (JN2)**: This sub-vertical major joint set is present in all pit areas. The dip ranges from 68° to 87° and the dip direction ranges from 185° to 031°.

- Some minor sets may exist although the prevalence of these sets is less certain due to the bias in the televiewer and oriented drill core data and the limited poles identified from oriented core:
  - ***Minor inclined Joint Set*** (jn3): It has an average dip of 57° and an average dip direction of 042°.
  - ***Minor inclined Joint Set*** (jn4): It has an average dip of 47° and an average dip direction of 271°.

FIGURE 15.5 ORIENTED AND TELEVIEWER POLE CONCENTRATIONS OF DRILL HOLE JOINTS DIAGRAMS



Source: *WSP (2024)*
Note: *A) Oriented drill core pole concentrations for joints from drill holes trending East.*
B) Televiewer pole concentrations from joints (all televiewer drill holes trend East).

15.2.6.2 Faults, Lithological Contacts, and Mineralized Vein Character

Drill core intersecting the faults was observed to be damaged and altered. Drill core intersecting lithological contacts were observed to have minimal alteration and damage present resulting in lithological contacts not being considered as planes of weakens for the stability assessments. Drill core intersecting the mineralized veins were observed to occasionally have an increase in structure at the vein boundary; however, a definite character could not be determined due to limited drill core intersecting the veins.

15.2.6.3 Shear Strength

Limited data was available to define shear strengths. Statistics on the surface characteristics of geotechnically logged structure and review of drill core intersections were considered when assuming the shear strength values outlined in Table 15.7Table 15.7 and Table 15.8. Pit scale shear strengths consider the influence of rock bridging and persistence.

TABLE 15.7
MOHR-COULOMB SHEAR STRENGTH PARAMETERS FOR STRUCTURE AT A BENCH SCALE

Discontinuity Type	Shear Strength at Bench Scale	
	Cohesion (kPa)	Phi (°)
Joints	0	30
Faults and Mineralized Veins	0	25

TABLE 15.8
MOHR-COULOMB SHEAR STRENGTH PARAMETERS FOR STRUCTURE AT A PIT SCALE

Geotechnical Unit	Shear Strength at Pit Scale	
	Cohesion (kPa)	Phi (°)
Joints	200	35
Faults and Mineralized Veins	20	25

15.2.7 Hydrogeological Characteristics

Groundwater data collected during the geotechnical investigations were limited to installation of fully grouted vibrating wire piezometers ("VWP") in three selected geotechnical drill holes. Continuous data collection from the VWPs using data loggers was initiated between March 16, 2024, and August 20, 2024. Hydraulic head elevation obtained from the installed VWPs on August 20, 2024 are presented below:

- GT-PFS-23-003 (ET-23-546) – 1,732.0 masl (VWP installation elevation – 1,718.9 masl).
- GT-PFS-23-010 (ET-23-559) – 1,742.5 masl (VWP installation elevation – 1,742.7 masl).
- GT-PFS-23-013 (ET-23-551) – 1,841.0 masl (VWP installation elevation – 1,833.2 masl).

Based on the above information, hydraulic head of approximately 13.1 m above the VW sensor was measured in drill hole GT-PFS-23-003 and approximately 7.8 m of head above the VW sensor was measured in drill hole GT-PFS-23-013. VWP data collected from the sensor installed in drill hole GT-PFS-23-010 indicate the sensor was installed above the water table and the drill hole is considered to be dry. Due to the limited available hydrogeological data, no groundwater model was developed as part of the existing scope of work, and dry conditions were assumed for all slope stability assessments.

15.2.8 Slope Design Assessments

15.2.8.1 Bench Scale Kinematics

Kinematic segments were oriented perpendicular to the dominant pit slope direction for the assessment sectors. In certain cases, due to large curvature in the pit wall the average slope orientation was selected for the kinematic segments. The purpose of the kinematic analysis is to determine optimum bench face angles, an essential part of the pit design for bench and inter-ramp stability based on the observed discontinuity orientation data for each geotechnical domain.

A summary of the results by failure mode is shown in Table 15.9. For each sector, the principal failure mode is identified. The likelihood of each failure mode is then discussed in the comments. Major sets are shown in **bold** and minor sets in *italic*.

TABLE 15.9
KINEMATIC ANALYSES RESULTS

Design Sector	KS	Wall Dip Direction (°)	Principal Failure Mode	Comments
South Wall	1	000	Wedge (JN1, *jn3*) and Toppling (JN2)	A potential wedge geometry involving a major and minor set and a topping geometry from a major set are both present in north dipping wall geometries. The El Tigre pit is expected to have no wall geometries facing north, limiting the impact of these kinematic concerns. In the event a wall is developed facing north, a BFA of 65° is recommended reduce the potential failure volume from the major toppling set; the wedge geometry is expected to intermittently occur based on its geometry involving a minor structural set.
West Wall	2	080	-	No consistent kinematic failure mode identified. Variations in the west wall dip direction are expected due to the valleys present in the topography. If the wall orientation varies down to 060°, a planar and wedge kinematic mode could occur. Both modes involve at least one minor set meaning they are less likely to be consistent stability concerns. Regardless, the BFA of this wall is limited to 70° to limit failure volumes based on pit wall orientation variations.

TABLE 15.9 KINEMATIC ANALYSES RESULTS				
Design Sector	**KS**	**Wall Dip Direction (°)**	**Principal Failure Mode**	**Comments**
North Wall	3	180	Planar (JN2)	A potential planar set is present in south dipping wall geometries. The El Tigre pit is expected to have no wall geometries facing south, limiting the impact of these kinematic concerns. In the event a wall is developed facing south, a BFA of 65° is recommended to undercut this set.
East Wall	4	250	Planar (*jn4*) and Wedge (JN1, JN2, *jn4*)	This wall has multiple wedge orientations (only one involving two major sets) and potential for planar failures from jn4 and JN1 (with wall orientation variation). JN1 has significant potential for variation on the east wall and may be steeper dipping than suggested by the televiewer data. To account for this uncertainty, the BFA was limited to 65° to undercut the majority of the structure identified in the televiewer data and to make the average pole concentrations in the drill core orientation data.

15.2.9 Slope Stability Assessment

To assess the potential for overall slope failure at the kinematically designed slope angles, stability analysis was completed using the limit equilibrium ("LE") approach using the Rocscience software package, Slide2. A total of five theoretical cross-sections were considered for the assessment, they were aligned perpendicular to local slope orientation, however, the slope was adjusted to match the recommended pit slope geometry.

The slopes were assessed under static and dry conditions; a sensitivity analysis was completed examining pseudo-static conditions considering a seismic coefficient of 0.025 g (50% of the potential peak ground acceleration of 0.05 g in the area of the El Tigre Project (Hynes-Griffin & Franklin, 1984) (Jonhson et al., 2023). The assessments included the fault orientations, vein locations, and joint anisotropy in the rock mass.

Based on the results it is not anticipated that deep seated slope failure will be a concern for the open pit under static and dry conditions. Pseudo-static results reduce the Factor of Safety ("FoS") by approximately 0.1, which still remains in line with the design criteria recommended by Reed and Stacey (2009) for pseudo-static stability. The worst-case scenarios are on the east wall; these conservatively assume that the minor set (jn4, only seen in oriented core) is prevalent in the east wall, even though this is not supported by the televiewer data. In this conservative scenario, sections 1 and 5 are controlled by the influence of the anisotropy caused by jn4; however, the FoS values are still greater than the minimum acceptance criteria of 1.3.

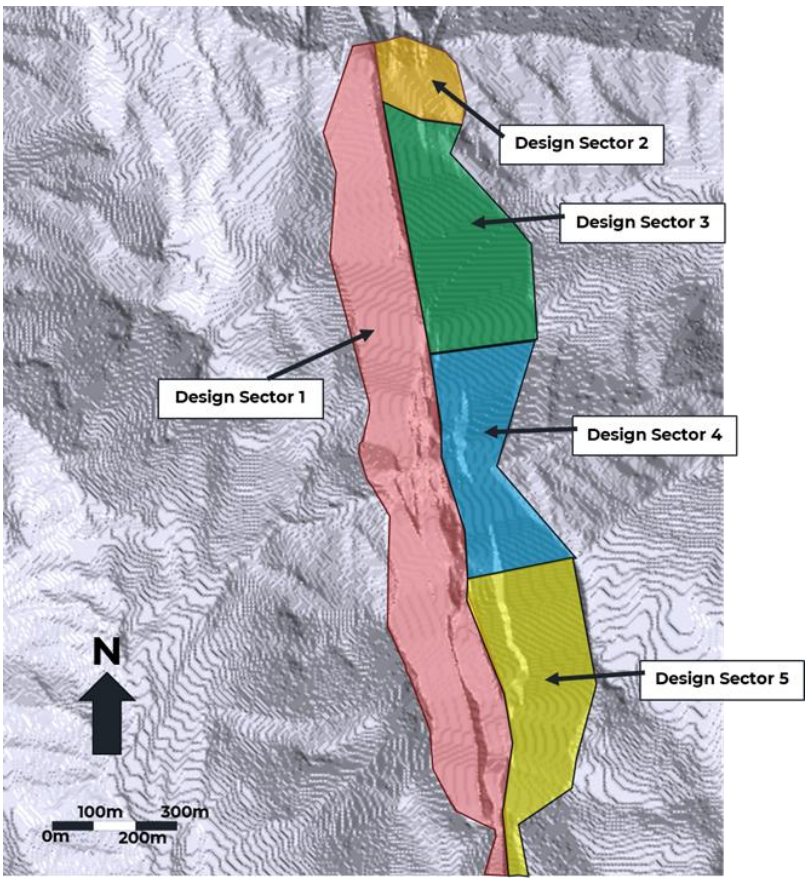
15.2.10 Slope Design Recommendations

Based on kinematic analysis and limit equilibrium approach, the updated design of the PFS pit was assessed considering slightly to moderately weathered rock with quadruple benches (20 m) for all sectors. For the west walls a BFA of 70° and IRA of 49° are suggested while for the East Walls a BFA of 65° and IRA of 46° were considered. Slopes of appreciable sizes are not anticipated on the North and South walls; however, if they are included in a future design, they should follow the East Wall geometry. The recommended open pit slope criteria at a PFS level are detailed in Figure 15.6 and Table 15.10. Additional considerations include:

- Historical underground workings are anticipated to be encountered in the El Tigre open pit. When mining through or around underground openings the bench height may need to be locally reduced to 5 m (single benches) to account for damaged ground.

- Where overburden is encountered (understood to typically be less than 5 m), it is recommended that slopes be laid back at a 3H:1V slope and a 10 m berm width be incorporated between the overburden and rock slopes. If overburden thickness is greater than 10 m a specific overburden slope design and investigation will be necessary.

FIGURE 15.6 DESIGN SECTORS FOR EL TIGRE OPEN PIT



Source: *WSP (2024)*

TABLE 15.10
RECOMMENDED PFS OPEN PIT SLOPE CRITERIA FOR EL TIGRE OPEN PIT

Design Sector	Geotechnical Unit	Wall Dip Dir. (°)[4]	Principal Failure Modes	Bench Height (m)	Max BFA (°)	Bench Width (m)[3]	IRA (°)	Max Inter-ramp height (m)	Geotech. Berm (m)
1	El Tigre Formation[1]	050 - 120	Kinematic	20	70	10	49	120	15
2-5		210 - 300	Kinematic	20	65	10	46		
All	Weathered or Faulted Rock Mass[2]	All	Rock mass	10	65	8	38	-	

1. *Only applied to fresh to slightly weathered rock with GSI > 40 and UCS > 40.*
2. *Near surface weathered rock mass is attributed to moderately to highly weathered and fractured rock with GSI < 40 and UCS < 40. Near surface weathering may be up to 30 m deep in places. Local bench height adjustments may be necessary through faulted areas if crest instability is identified during development of interim slopes.*
3. *Bench widths are determined based on the Modified Ritchie criteria and the spill berm width; additional buffer has been included due to structural uncertainty. This value may be adjusted later in the design process when more information is available regarding the structural data, persistence, and properties, and the rockfall performance of the benches.*
4. *The wall dip direction in the table above refers to the location you are facing if you were standing on the crest and looking downslope.*

15.3 SURFACE MINE MINERAL RESERVE

After the surface mine designs are completed, the ore tonnage from the diluted block model is reported inside the pit. The corresponding tonnes and grades are summarized by Mineral Reserve classification in Table 15.1. These Mineral Reserve tonnages and grades are used as the planning basis for the production schedule.

The total quantity of ore (i.e. Mineral Reserve) over the life-of-mine ("LOM") sent to the heap leach facility is 40.3 Mt.

There is a small quantity of Inferred Mineral Resource within the surface mine design, some of which may eventually be converted to either the Measured or Indicated Mineral Resource classifications. The Inferred Mineral Resource material is considered as waste in the mining schedule, and is approximately 1.4 million tonnes (Mt), summarized in Table 15.11.

TABLE 15.11
INFERRED MINERAL RESOURCE

Inferred Resource	Tonnes (k)	Au (g/t)	Ag (g/t)	AuEq (g/t)	AgEq (g/t)
Oxide	1,303	0.23	4.9	0.26	42.1
Sulphide	105	0.24	7.9	0.31	35.1
Total Inferred Resource	**1,408**	**0.23**	**5.1**	**0.26**	**41.6**

AuEq Cut-off grades: Oxide = 0.11 g/t, Sulphide = 0.16 g/t.

16.0 MINING METHODS

The El Tigre Project will consist of a single surface mine operation using conventional equipment. Mining will be initiated in Year -1 (pre-stripping period) and will be completed in Year 9. Reclamation and closure will be carried out in Year 10. Mining will occur in three development phases (pushbacks).

The initial ore heap leaching rate (Years 1 to 3) is 7,500 tpd (2.74 Mt per annum ("Mtpa")). In Year 4 the crushing rate will increase to 15,000 tpd (5.5 Mtpa) and this rate is maintained to the end of the mine life.

16.1 MINE PHASING

To access higher-grade ore and lower waste stripping quantities early in the production schedule, the pit has been subdivided into three phases. The phase designs are shown in Figure 16.1. The ore and waste tonnages for each phase are shown in Table 16.1. Process plant head grade is highest in Phase I and the strip ratio is the least.

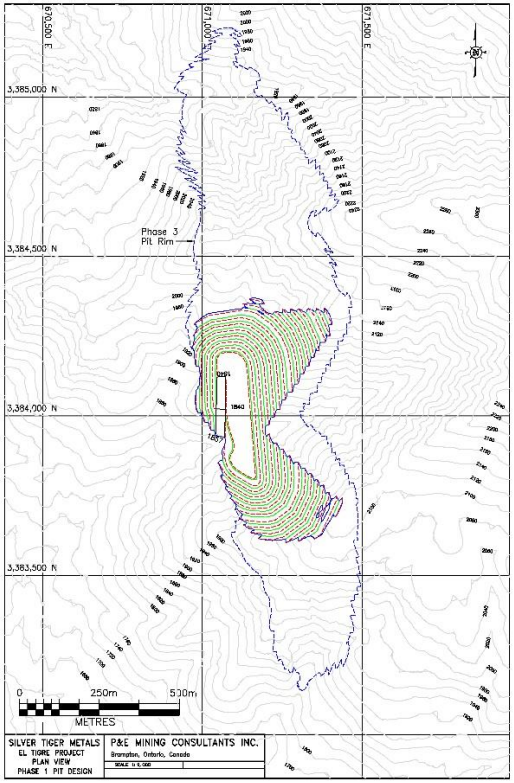
TABLE 16.1
PHASE TONNAGE SUMMARY

Material	Total	Phase I	Phase II	Phase III
Total Material (kt)	108,614	24,361	30,952	53,301
Waste Rock (kt)	68,321	12,820	20,211	35,290
Strip Ratio	1.7	1.1	1.9	2.0
OX-LG (kt)	3,575	676	1,287	1,612
Au g/t	0.10	0.10	0.11	0.10
Ag g/t	3.6	3.3	3.0	4.3
AuEq g/t	0.13	0.13	0.13	0.13
OX-HG (kt)	30,790	8,400	9,341	13,049
Au g/t	0.42	0.53	0.42	0.35
Ag g/t	16.6	13.0	14.1	20.6
AuEq g/t	0.52	0.61	0.51	0.47
SU-LG (kt)	910	257	11	642
Au g/t	0.16	0.15	0.16	0.16
Ag g/t	2.8	3.1	2.5	2.7
AuEq g/t	0.18	0.18	0.18	0.18
SU-HG (kt)	5,018	2,208	102	2,708
Au g/t	0.56	0.66	0.61	0.47
Ag g/t	15.1	9.1	16.2	19.9
AuEq g/t	0.69	0.74	0.76	0.65
Total Ore (kt)	**40,292**	**11,541**	**10,740**	**18,011**
Au g/t	**0.40**	**0.52**	**0.39**	**0.34**
Ag g/t	**14.9**	**11.5**	**12.7**	**18.4**
AuEq g/t	**0.50**	**0.60**	**0.46**	**0.46**

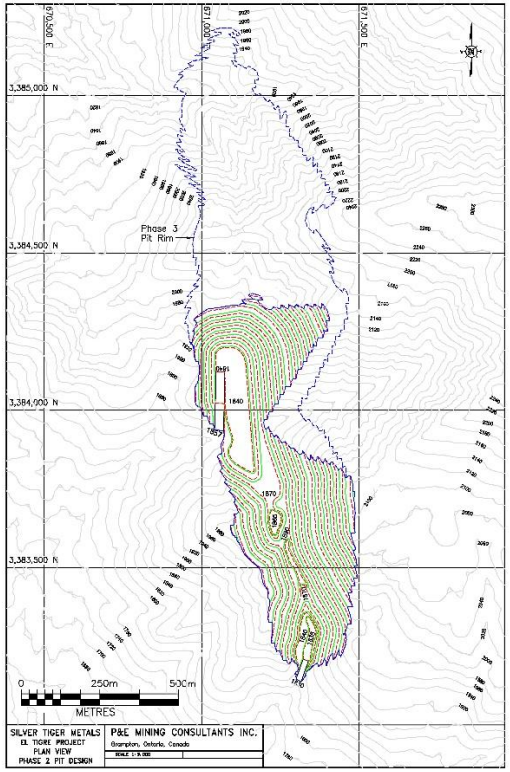
OX=oxide, SU=sulphide, LG=low grade, HG=high grade.

FIGURE 16.1 SURFACE MINE PHASES

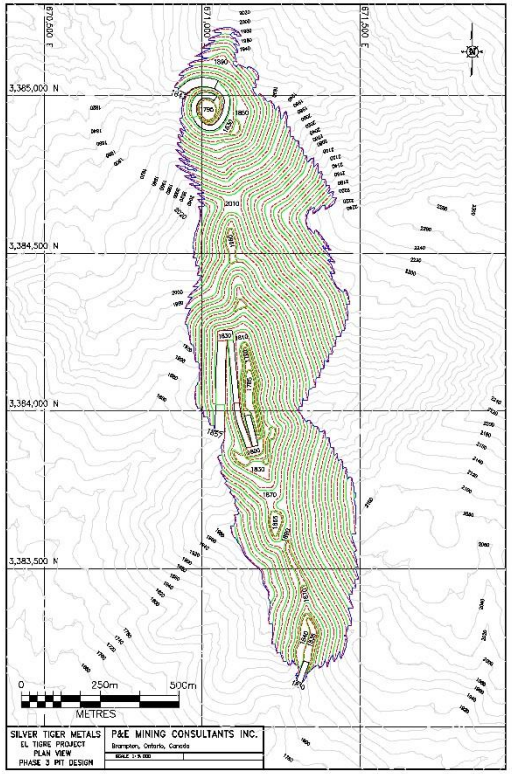
Phase I



Phase II



Phase III (Final Pit)



The operating cost, and hence the cut-off grade, when operating at 15,000 tpd is lower than when operating at 7,500 tpd. Hence in Years 1 to 3, ore between the 7,500 tpd cut-off and 15,000 tpd cut-off is designated as low grade ("LG") and is stockpiled for processing at the end of the mine life. The cut-off grades applied are as follows:

OX = Oxide ore	OX-LG = 0.11-0.15 g/t AuEq	OX-HG > 0.15 g/t AuEq
SU = Sulphide ore	SU-LG = 0.16-0.22 g/t AuEq	OX-HG > 0.22 g/t AuEq

16.2 MINING SCHEDULE

The surface mine production schedule consists of one year of pre-stripping (Year -1) and nine years of mine production (Year 1 through Year 9). The mining sequence is shown in Table 16.2. Mining often occurs in two phases simultaneously to blend waste stripping peaks.

TABLE 16.2
PHASE MINING SEQUENCE

Phase	No. Years	-1	1	2	3	4	5	6	7	8	9
1	5										
2	5										
3	6										

The life-of mine ("LOM") production schedule is shown in Table 16.3. The peak total mining rate (ore and waste) is 18 Mtpa occurring in Years 4, 5, 6 when strip ratios peak at 2.3:1. This is the period when Phase III is started, requiring significant waste rock removal. Gradually the total mining rate tapers off as the phases intersect more ore and less waste rock.

TABLE 16.3
SURFACE MINE PRODUCTION SCHEDULE

Material	Total	-1	1	2	3	4	5	6	7	8	9
Total LOM											
Total Material (kt)	108,614	1,500	7,500	7,500	9,000	18,000	18,000	18,000	14,000	9,000	6,114
Waste Rock (kt)	68,321	1,446	4,499	4,553	5,971	12,528	12,528	12,528	8,528	3,528	2,212
Strip Ratio	1.7	26.5	1.5	1.5	2.0	2.3	2.3	2.3	1.6	0.6	0.6
OX-LG (kt)	3,575	10	238	121	212	609	608	554	327	516	380
Au g/t	0.10	0.09	0.10	0.10	0.10	0.11	0.11	0.11	0.11	0.10	0.08
Ag g/t	3.6	5.1	3.1	3.8	4.1	3.3	2.6	2.3	2.1	4.3	7.8
AuEq g/t	0.13	0.12	0.12	0.13	0.13	0.13	0.13	0.13	0.12	0.13	0.12
OX-HG (kt)	30,790	45	2,107	1,820	2,129	4,748	4,863	4,413	3,323	4,084	3,259
Au g/t	0.42	0.35	0.56	0.57	0.48	0.45	0.43	0.41	0.35	0.34	0.33
Ag g/t	16.6	10.6	16.1	17.1	12.8	11.6	13.1	14.9	11.5	22.8	31.1
AuEq g/t	0.52	0.42	0.66	0.67	0.56	0.52	0.51	0.50	0.42	0.48	0.52
SU-LG (kt)	910		71	90	81	15		108	315	180	49
Au g/t	0.16	-	0.16	0.16	0.15	0.13	-	0.15	0.17	0.16	0.12
Ag g/t	2.8	-	2.5	3.2	3.5	2.1	-	2.5	1.9	3.0	6.3
AuEq g/t	0.18	-	0.18	0.18	0.18	0.15	-	0.17	0.18	0.19	0.18
SU-HG (kt)	5,018		584	916	607	101	0	397	1,507	692	213
Au g/t	0.56	-	0.57	0.73	0.66	0.49	0.22	0.42	0.53	0.45	0.32
Ag g/t	15.1	-	8.3	10.8	8.0	4.8	62.7	15.8	4.2	39.3	73.6
AuEq g/t	0.69	-	0.64	0.83	0.73	0.54	0.78	0.56	0.57	0.80	0.98
Total LOM Ore (kt)	**40,292**	**54**	**3,001**	**2,947**	**3,029**	**5,472**	**5,472**	**5,472**	**5,472**	**5,472**	**3,901**
Au g/t	0.40	0.31	0.52	0.59	0.48	0.41	0.39	0.37	0.37	0.33	0.30
Ag g/t	14.9	9.6	13.2	14.2	11.0	10.5	11.9	13.4	8.4	22.5	30.9
AuEq g/t	0.50	0.36	0.60	0.68	0.55	0.48	0.47	0.46	0.43	0.48	0.50

TABLE 16.3
SURFACE MINE PRODUCTION SCHEDULE

Material	Total	-1	1	2	3	4	5	6	7	8	9
Phase I											
Total Material (kt)	24,361	1,500	7,500	4,994	5,522	4,845					
Waste Rock (kt)	12,820	1,446	4,499	2,056	2,790	2,029					
Strip Ratio	1.1	26.5	1.5	0.7	1.0	0.7	-	-	-	-	-
OX-LG (kt)	676	10	238	118	140	171					
Au g/t	0.10	0.09	0.10	0.10	0.10	0.11	-	-	-	-	-
Ag g/t	3.3	5.1	3.1	3.7	3.5	2.8	-	-	-	-	-
AuEq g/t	0.13	0.12	0.12	0.13	0.13	0.13	-	-	-	-	-
OX-HG (kt)	8,400	45	2,107	1,814	1,904	2,530					
Au g/t	0.53	0.35	0.56	0.57	0.51	0.50	-	-	-	-	-
Ag g/t	13.0	10.6	16.1	17.1	12.4	8.1	-	-	-	-	-
AuEq g/t	0.61	0.42	0.66	0.67	0.58	0.55	-	-	-	-	-
SU-LG (kt)	257		71	90	81	15					
Au g/t	0.15	-	0.16	0.16	0.15	0.13	-	-	-	-	-
Ag g/t	3.1	-	2.5	3.2	3.5	2.1	-	-	-	-	-
AuEq g/t	0.18	-	0.18	0.18	0.18	0.15	-	-	-	-	-
SU-HG (kt)	2,208		584	916	607	101					
Au g/t	0.66	-	0.57	0.73	0.66	0.49	-	-	-	-	-
Ag g/t	9.1	-	8.3	10.8	8.0	4.8	-	-	-	-	-
AuEq g/t	0.74	-	0.64	0.83	0.73	0.54	-	-	-	-	-
Total Phase I Ore (kt)	**11,541**	**54**	**3,001**	**2,938**	**2,731**	**2,816**					
Au g/t	0.52	0.31	0.52	0.59	0.51	0.47	-	-	-	-	-
Ag g/t	11.5	9.6	13.2	14.2	10.7	7.6	-	-	-	-	-
AuEq g/t	0.60	0.36	0.60	0.69	0.58	0.52	-	-	-	-	-

TABLE 16.3
SURFACE MINE PRODUCTION SCHEDULE

Material	Total	-1	1	2	3	4	5	6	7	8	9
Phase II											
Total Material (kt)	30,952			2,506	3,478	12,155	8,422	4,391			
Waste Rock (kt)	20,211			2,497	3,181	9,499	3,493	1,541			
Strip Ratio	1.9	-	-	286.7	10.7	3.6	0.7	0.5	-	-	-
OX-LG (kt)	1,287			3	73	438	506	267			
Au g/t	0.11	-	-	0.09	0.09	0.10	0.11	0.11	-	-	-
Ag g/t	3.0	-	-	4.2	5.3	3.5	2.4	2.4	-	-	-
AuEq g/t	0.13	-	-	0.12	0.12	0.13	0.13	0.13	-	-	-
OX-HG (kt)	9,341			6	225	2,218	4,422	2,470			
Au g/t	0.42	-	-	0.17	0.25	0.40	0.45	0.41	-	-	-
Ag g/t	14.1	-	-	15.6	15.8	15.5	13.2	14.1	-	-	-
AuEq g/t	0.51	-	-	0.27	0.35	0.49	0.53	0.50	-	-	-
SU-LG (kt)	11							11			
Au g/t	0.16	-	-	-	-	-	-	0.16	-	-	-
Ag g/t	2.5	-	-	-	-	-	-	2.5	-	-	-
AuEq g/t	0.18	-	-	-	-	-	-	0.18	-	-	-
SU-HG (kt)	102							102			
Au g/t	0.61	-	-	-	-	-	-	0.61	-	-	-
Ag g/t	16.2	-	-	-	-	-	-	16.2	-	-	-
AuEq g/t	0.76	-	-	-	-	-	-	0.76	-	-	-
Total Phase II Ore (kt)	**10,740**			**9**	**298**	**2,656**	**4,929**	**2,850**			
Au g/t	0.39	-	-	0.15	0.21	0.35	0.41	0.39	-	-	-
Ag g/t	12.7	-	-	11.8	13.2	13.5	12.1	13.0	-	-	-
AuEq g/t	0.46	-	-	0.22	0.29	0.43	0.49	0.47	-	-	-

TABLE 16.3
SURFACE MINE PRODUCTION SCHEDULE

Material	Total	-1	1	2	3	4	5	6	7	8	9
Phase III											
Total Material (kt)	53,301					1,000	9,578	13,609	14,000	9,000	6,114
Waste Rock (kt)	35,290					1,000	9,035	10,987	8,528	3,528	2,212
Strip Ratio	2.0	-	-	-	-	-	16.6	4.2	1.6	0.6	0.6
OX-LG (kt)	1,612						102	287	327	516	380
Au g/t	0.10	-	-	-	-	-	0.10	0.11	0.11	0.10	0.08
Ag g/t	4.3	-	-	-	-	-	3.6	2.2	2.1	4.3	7.8
AuEq g/t	0.13	-	-	-	-	-	0.12	0.13	0.12	0.13	0.12
OX-HG (kt)	13,049						441	1,943	3,323	4,084	3,259
Au g/t	0.35	-	-	-	-	-	0.26	0.40	0.35	0.34	0.33
Ag g/t	20.6	-	-	-	-	-	11.7	15.9	11.5	22.8	31.1
AuEq g/t	0.47	-	-	-	-	-	0.33	0.49	0.42	0.48	0.52
SU-LG (kt)	642							97	315	180	49
Au g/t	0.16	-	-	-	-	-	-	0.15	0.17	0.16	0.12
Ag g/t	2.7	-	-	-	-	-	-	2.5	1.9	3.0	6.3
AuEq g/t	0.18	-	-	-	-	-	-	0.17	0.18	0.19	0.18
SU-HG (kt)	2,708						0	295	1,507	692	213
Au g/t	0.47	-	-	-	-	-	0.22	0.36	0.53	0.45	0.32
Ag g/t	19.9	-	-	-	-	-	62.7	15.6	4.2	39.3	73.6
AuEq g/t	0.65	-	-	-	-	-	0.78	0.50	0.57	0.80	0.98
Total Phase III Ore (kt)	**18,011**						**543**	**2,622**	**5,472**	**5,472**	**3,901**
Au g/t	0.34	-	-	-	-	-	0.23	0.35	0.37	0.33	0.30
Ag g/t	18.4	-	-	-	-	-	10.2	13.9	8.4	22.5	30.9
AuEq g/t	0.46	-	-	-	-	-	0.29	0.44	0.43	0.48	0.50

16.3 MINING PRACTICES

It is assumed that the El Tigre Mine will be a contract mining operation. While owner-operated mining may be an option, this was not considered in this PFS. Numerous other mines in Mexico rely on the use of contract mining, with many experienced contractors being available at competitive rates. Contractor budgetary quotations for mining the El Tigre deposits were obtained for this PFS.

The Mining Contractor will undertake all drilling, loading, hauling, and mine site maintenance activities. A separate Blasting Contractor will provide blast hole loading and charging services. The Owner will provide overall mine management and technical services, such as mine planning, grade control, geotechnical, and surveying services.

16.3.1 Contract Mining

It is anticipated that the mining operations would be undertaken 24 hours per day and seven days per week throughout the entire year.

The exact equipment fleet to be used by the Mining Contractor has not yet been defined in detail at this stage. This will depend on the types of equipment the contractor has in its fleet once a formal mining contract is enacted. However, based on current information, it is expected that mid-sized hydraulic excavators and diesel-powered front-end loaders will be used. The anticipated truck size is 91 tonnes, similar to the CAT 777 truck, although alternate truck sizes may eventually be used.

The primary mining operation will be supported by the Mining Contractor's fleet of equipment consisting of dozers, road graders, watering trucks, maintenance vehicles, and service vehicles. The projected mining equipment fleet is shown in Table 16.4.

TABLE 16.4
CONTRACTOR MINING FLEET

Mining Contractor Fleet	Peak Units
Atlas DM45 Drill	5
Komatsu PC1250 Excavator	2
CAT 993 Wheel Loader	2
CAT 777 Truck	13
CAT D8T Dozer	3
CAT 16M Grader	1
CAT 773 Water Truck	2

It is assumed that all materials mined will require drilling and blasting to some degree. The overburden layer is very thin, at less than 1.0 m in most places, with some areas in valley floors up to several metres thick. The Blasting Contractor will provide the blast hole charging services,

and it is anticipated that blasting of the rock will be carried out using an ammonium nitrate fuel oil mixture ("ANFO").

Portions of the deeper phase pits may experience groundwater seepage. No quantitative information was available to adequately predict the expected water inflow into the surface mine, however, the Mining Contractor will be responsible to supply and operate pit dewatering pumps and pipelines to keep the mine dry and operable.

16.3.2 Owner's Mining Team

The mine Owner will be responsible for providing contract management and overall supervision of the mining contractor. The owner will also provide technical services, such as mine planning and scheduling, geotechnical engineering, grade control, and surveying. Table 16.5 lists the personnel on the owner's mining team.

TABLE 16.5
OWNER'S MINING TEAM

Mining Team	Number
Mine Superintendent	1
Mine Superintendent Assistant	1
Mine Supervisor	1
Drill & Blast Supervisor	1
Technical Services Superintendent	1
Chief Mine Engineer	1
Surveyor	2
Surveyor Assistant	2
Geology Supervisor	2
Samplers	2
General Services Assistant	2
Total Team	**16**

16.4 WASTE ROCK STORAGE FACILITIES

Several waste rock storage facilities will be utilized depending on where the mining activity is occurring. The Main Waste Rock Storage Facility will be located to the west of the surface mine. Two smaller waste rock facilities will be required when stripping the upper benches since haul distances to the Main Facility will be lengthy. The location of the waste rock storage facilities are shown on the site general layout drawing in Section 18.

Some minor amounts of the waste rock will also be used to build new haul roads and widen existing roads as necessary. The waste rock material balance is shown in Table 16.6.

TABLE 16.6 WASTE ROCK VOLUMES		
Waste Rock	**Units**	**Volume**
Total Waste Tonnage Mined	kt	68,321
Waste Loose Density (30% swell)	t/m^3	1.92
Waste Volume for Storage	$k\text{-}m^3$	35,527
Placement		
Main Waste Facility	$k\text{-}m^3$	28,259
East Waste Facility	$k\text{-}m^3$	5,218
Southeast Waste Facility	$k\text{-}m^3$	2,050
Total Waste Rock Placed	**$k\text{-}m^3$**	**35,527**

Note: $k\text{-}m^3$ = thousands of cubic metres

16.5 MINE SUPPORT FACILITIES

The El Tigre surface mine will require mine offices, maintenance facilities, warehousing, and storage areas.

A truck maintenance shop area will be provided by the Mining Contractor. A fuel storage/distribution and lube station will be provided by a diesel fuel contractor for fuelling the mining fleet.

17.0 RECOVERY METHODS AND PROCESS PLANT

17.1 SUMMARY

This section describes the recovery methods implemented in the design of the crushing and process facilities for the El Tigre Project Phase I and II. Testwork results presented in Section 13 form the basis for flowsheet development and design criteria. The process plant design presented in this PFS is based on a nominal 7,500 tpd (Years 1 to 3) and 15,000 tpd (Years 4 to 9) of ore with average grades of 0.40 g/t gold and 14.9 g/t silver. The recovery methods are consistent with the crushing, pad loading, leaching, Merrill Crowe precipitation, refinery, and support processing methods presented in the 2023 PEA.

Revised process recovery methods included in this PFS are as follows:

- Process plant facility now located adjacent to the heap pond infrastructure to improve process operability and eliminate distant pumping and power demand. This is illustrated in Section 18.

- Water will be supplied by surface water wells.

- Selected equipment for crushing and recovery during Phase I has been "upsized" to lower Phase II direct capital requirements and related construction time. This includes the primary crusher system and material handling, discharge conveyors to the pad loading, and Merrill Crowe deaeration, vacuum and process pumps. The refinery has also been designed to treat Phase II metal production.

The process plant flowsheet design comprises of three-stage conventional crushing, material handling of crushed product and loading onto the lined heap pads. Solution ponds and pumping system allows irrigation of loaded mineralized material and subsequent collection of the pregnant solution. Solution ponds and pumping system allows irrigation of loaded ore and subsequent collection of the pregnant solution. The pregnant solution is pumped to the Merrill Crowe recovery facility for precipitation of the gold and silver in solution and filtered accordingly. The resultant precipitate will be smelted at an on-site refinery to produce doré bars for market. The barren solution from the recovery process is recirculated to heap leach pond (barren) for cyanide addition and pumping to the heap leaching pad.

Unit operations and support facilities includes the following:

- ROM material and primary crushing with internal stockpile;

- Secondary cone(s) crushing with screens;

- Tertiary cone(s) crushing and screens (Final crush size of P_{80} –3/8 in.);

- Material Handling and closed circuit crushing and heap leach pad loading;

- 80 mm lined (LLDPE) heap pads capable of the supporting the entire Mineral Reserve (two stages to allow placement of LOM 40.3 Mt). Phase I – 168,000 m^2 and Phase II – 248,000 m^2;

- 60 mm lined (LLDPE) Solution ponds – barren, pregnant, and emergency ponds complete with internal pumping, piping, and flow distribution system to the pads;

- Conventional Merrill Crowe (zinc precipitation) process plant;

- Doré Refinery with dust collection system;

- Regent preparation facilities (main plant and pad);

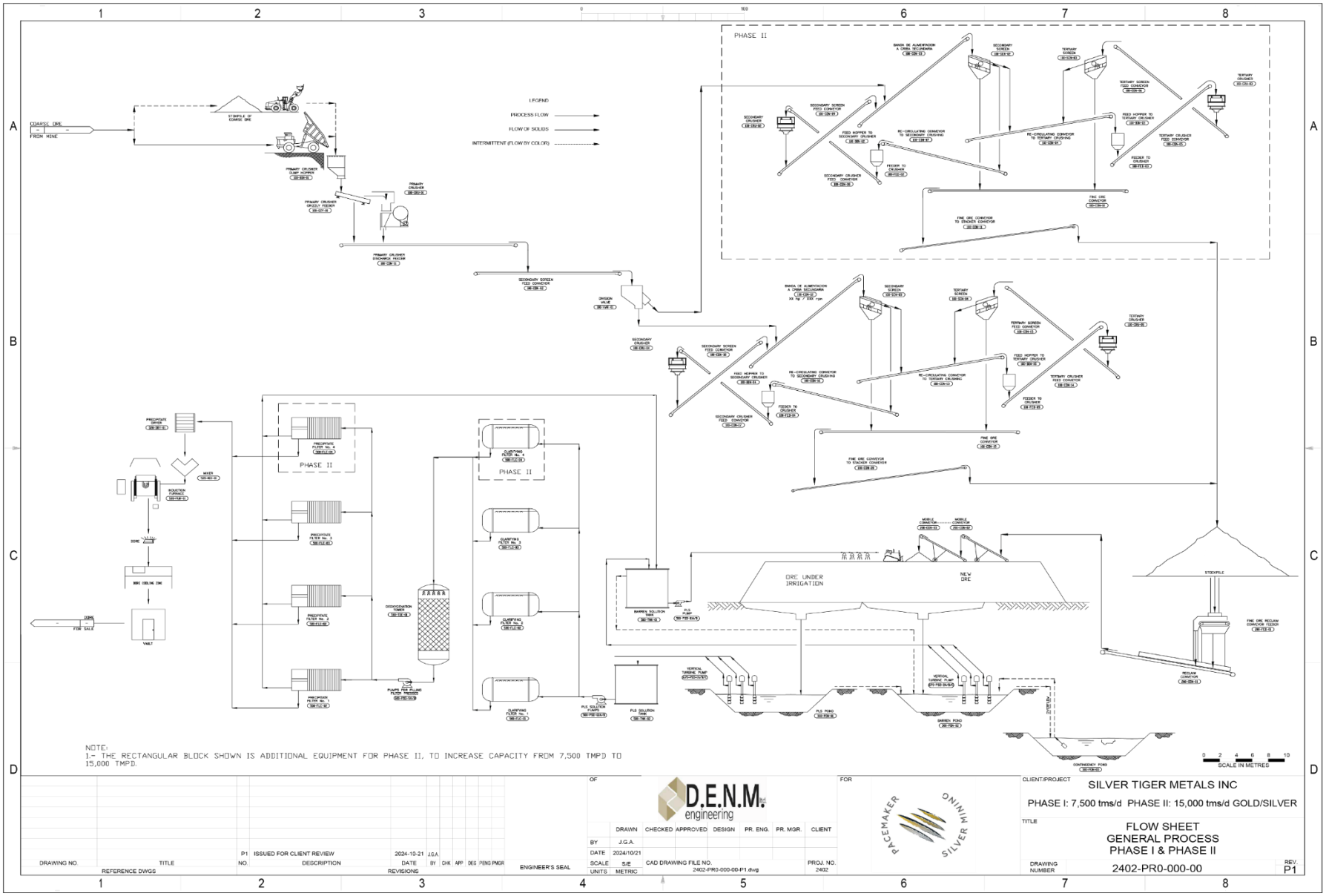
- Assay and metallurgical Laboratory; and

- Utilities (water, air, standby power).

Make-up water for reagent mixing along with water evaporation and general process requirements will be supplied by wells as noted above.

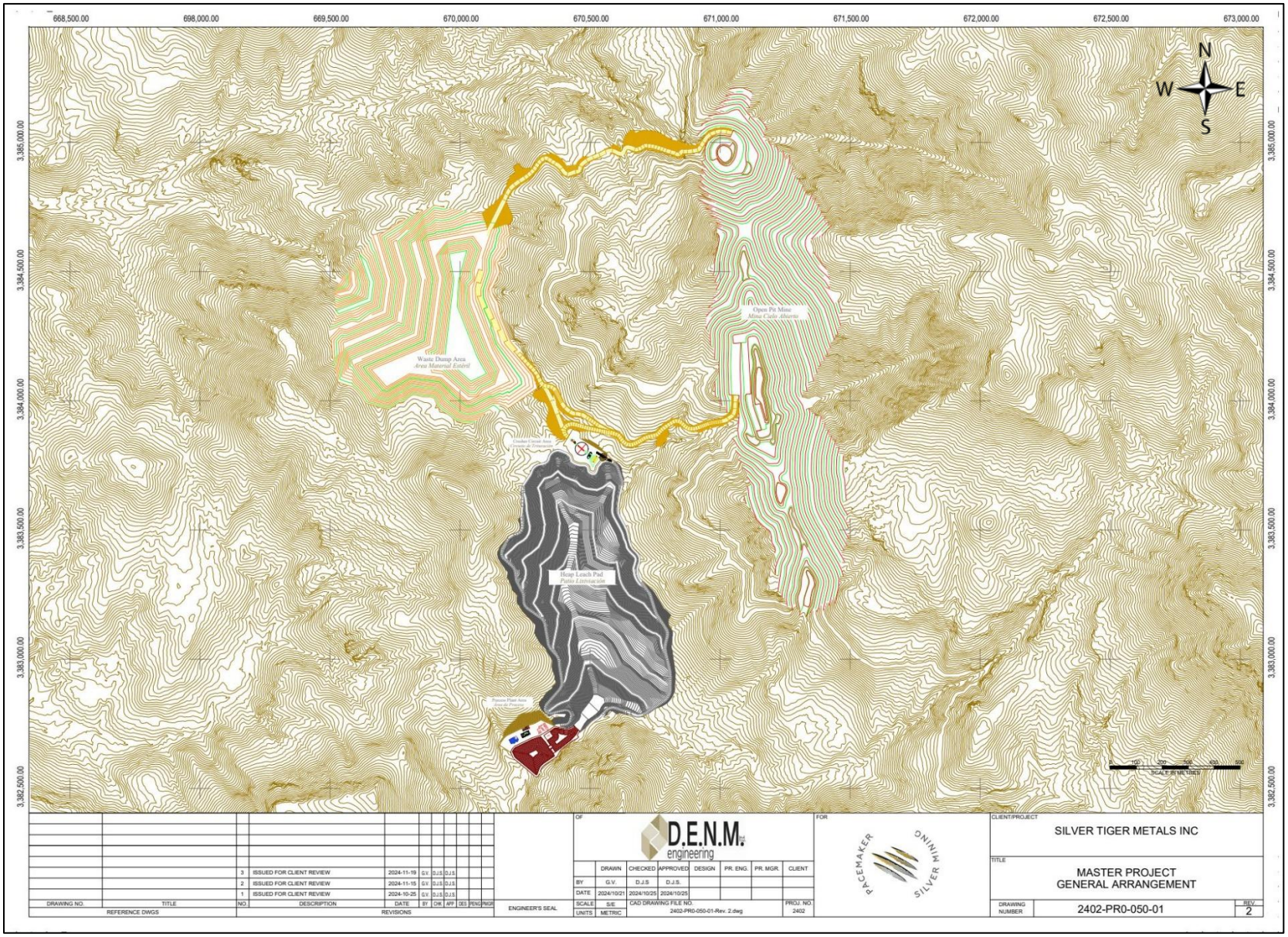
Figures 17.1 and 17.2 illustrate a simplified process flowsheet for Phase I and Phase II as well as the proposed location for the process components within the site plan.

FIGURE 17.1 SIMPLIFIED PROCESS FLOWSHEET FOR PHASE I AND PHASE II



Source: *D.E.N.M. (2024)*

FIGURE 17.2 SITE PLAN



Source: *D.E.N.M. (2024)*

17.2 PROCESS PLANT DESIGN

17.2.1 Design Criteria

The El Tigre process plant is designed to treat gold-silver bearing ore at a nominal rate of a nominal rate of 7,500 tpd for the initial three years and then increase to 15,000 tpd in year four for the remaining LOM.

- Phase I: Years 1 to 3 at 7,500 tpd (2,737,500 tpy, 416 t/h).
- Phase II: Years 4 to 9 at 15,000 tpd (5,475,000 tpy, 833 t/h).

PFS testwork results indicate the presence of oxide, transition, sulphide domains as well as associated variable recovery. Table 17.1 illustrates the preliminary key process design criteria.

TABLE 17.1
PROCESS DESIGN CRITERIA

Criteria	Units	Value
Ore Characteristics		
Specific Gravity	g/cm³	2.6
Process Plant Availability/Utilization		
Overall Process Plant Feed-Nominal – Years 1-3+	tpy	2,737,500
Overall Process Plant Feed- Nominal – Years 1-3+	tpd	7,500
Overall Process Plant Feed-Nominal – Years 3 +	tpy	5,475,000
Overall Process Plant Feed- Nominal – Years 3 +	tpd	15,000
Crushing Plant Availability	%	75
Heap and Processing Plant Availability	%	95
Crushing Product (to pad)	P_{80} inch (mm)	3/8 (9.0)
Process Plant Production		
Plant Feed Characteristics (Life of Mine Average)		
Gold Head Grade	g/t	0.40
Silver Head Grade	g/t	14.9
Metal Recoveries and Process Details		
Anticipated Overall Gold Oxide Recovery- design[1]	%	83
Anticipated Overall Silver Oxide Recovery- design[1]	%	45
Anticipated Overall Gold Sulphide Recovery- design[1]	%	56
Anticipated Overall Silver Sulphide Recovery- design[1]	%	40
Cyanide Addition Rates – design[2]	kg/t	0.25
Lime Addition Rates – design[2]	kg/t	1.5
Merrill Crowe Phase I – design	m^3/h	590
Merrill Crowe Phase II – design	m^3/h	1,180
Merrill Crowe Recovery – Au and Ag	%	99

Source: *D.E.N.M. (2024)*

Notes: [1] Column testing indicated both higher and gold and silver recovery (oxide and sulphide) at a 3/8 in. crush size. In the process design and financial model, these have been discounted by 3% for leaching in the field versus optimum conditions in the laboratory
[2] Cyanide Consumption is also discounted for the process design, operating costs, and financial model. Lime is not discounted from the column test data.

Considering 408 koz of gold and 8.57 Moz of silver recovered during Life of Mine, estimated doré metal content for market is 4% Au and 93% Ag with the remaining residual precipitation elements.

17.2.2 Operating Schedule and Availability

The El Tigre processing plant is designed to operate for two 12-hour shifts per day, 365 days per year. Utilization expected for the specific circuits is 75% for the primary crusher and 95% for the leaching and Merrill Crowe facility. The factors applied allow for sufficient downtime for maintenance, both scheduled and unscheduled, within the crushing and processing areas.

17.3 PROCESS PLANT DESCRIPTION

17.3.1 Primary Crushing Circuit

The proposed primary crushing circuit reduces the run-of-mine ore from a nominal top size of 600 mm to a product of (P_{80}) –3/8 in (9.0 mm) for the conveyor loading to the heap leach pads. As noted, the front end has been upsized to handle Phase 2 design.

The crushing circuit includes the following equipment:

- ROM feed hopper complete with feeder and vibrating grizzly screen;
- Primary jaw crusher;
- Associated conveyors belts to feed and discharge crushed material;
- Belt scale and magnet; and
- Feed control system.

The jaw crusher system processes a nominal 416 t/h (Phase I) and 833 t/h (Phase II post expansion) of oversized material based on the utilization factor noted in Table 17.1.

The selected 200 kW jaw crusher is Model # CJ815 (1,500 mm x 1,300 mm) and the crushed product discharge is conveyed to the crushed ore stockpile.

17.3.2 Primary Crushed Mineralized Material Stockpile and Reclaim

The stockpile provides production surge capacity to ensure a steady rate to the secondary crushing circuit.

The pan feeders discharge onto the secondary feed conveyor to feed crushed ore to the secondary screen unit. The feeders reclaim the material from the stockpile and ensure a controlled feed rate to the secondary circuit. Feed control to the feeders is ensured by the inline belt scale.

17.3.3 Secondary and Tertiary Crushing Circuit

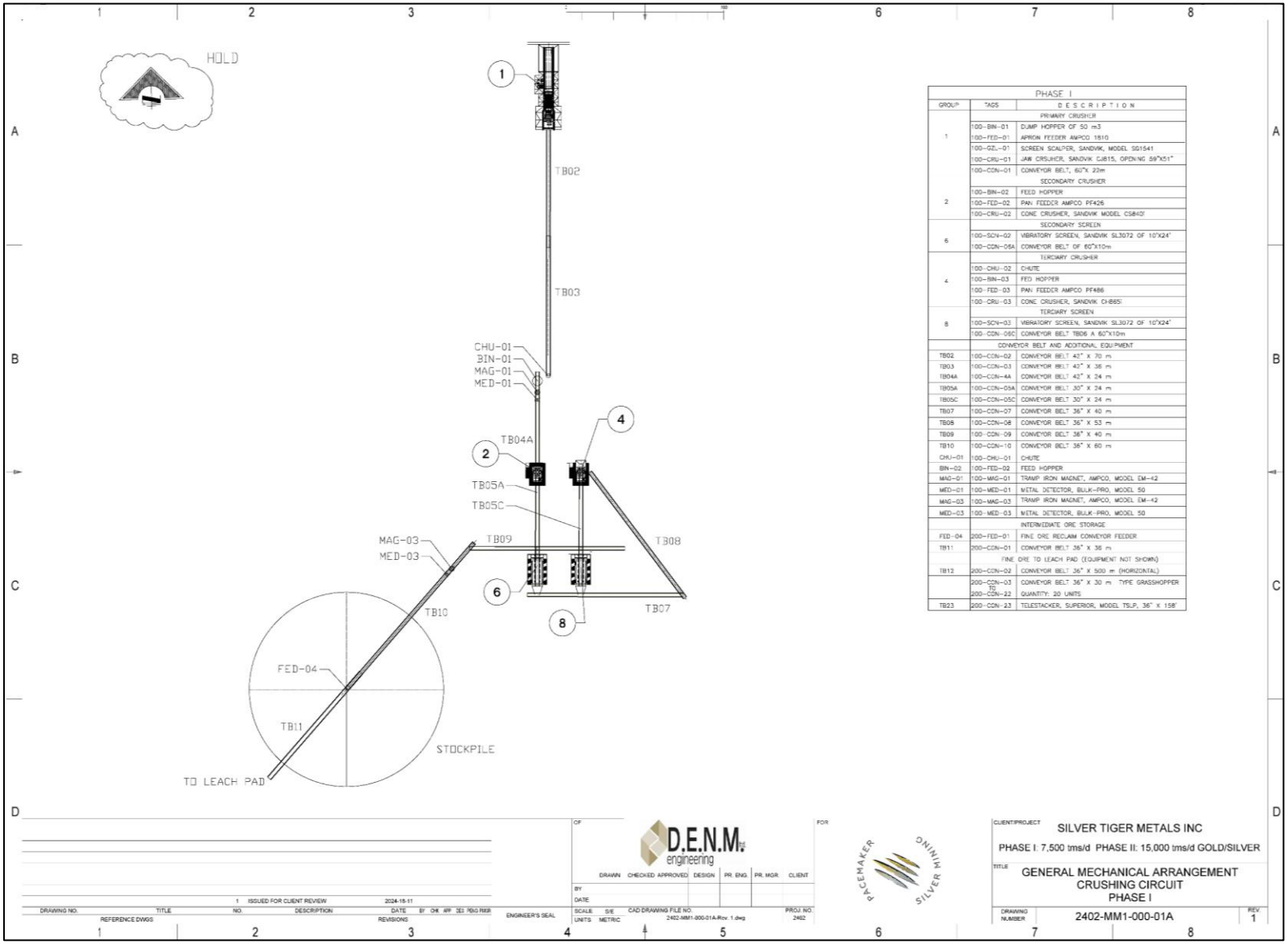
The equipment in this area includes:

- Secondary inclined linear double deck screen;
- Secondary crusher with closed side setting of 44 mm;
- Tertiary inclined screens operating in parallel;
- Tertiary cone crushers with closed side setting of 9 mm (or less); and
- Associated conveyors belt feeders and discharge system for recirculation and discharge to the crushed ore stockpile. Again, upsized to suit Phase II.

The crushing circuit is located upstream of the heap leach pad facility and process plant and ponds. The crushed material is reclaimed via a series of feeders and grasshopper conveyors and stacking units to systemically load the crushed ore onto the lined pads.

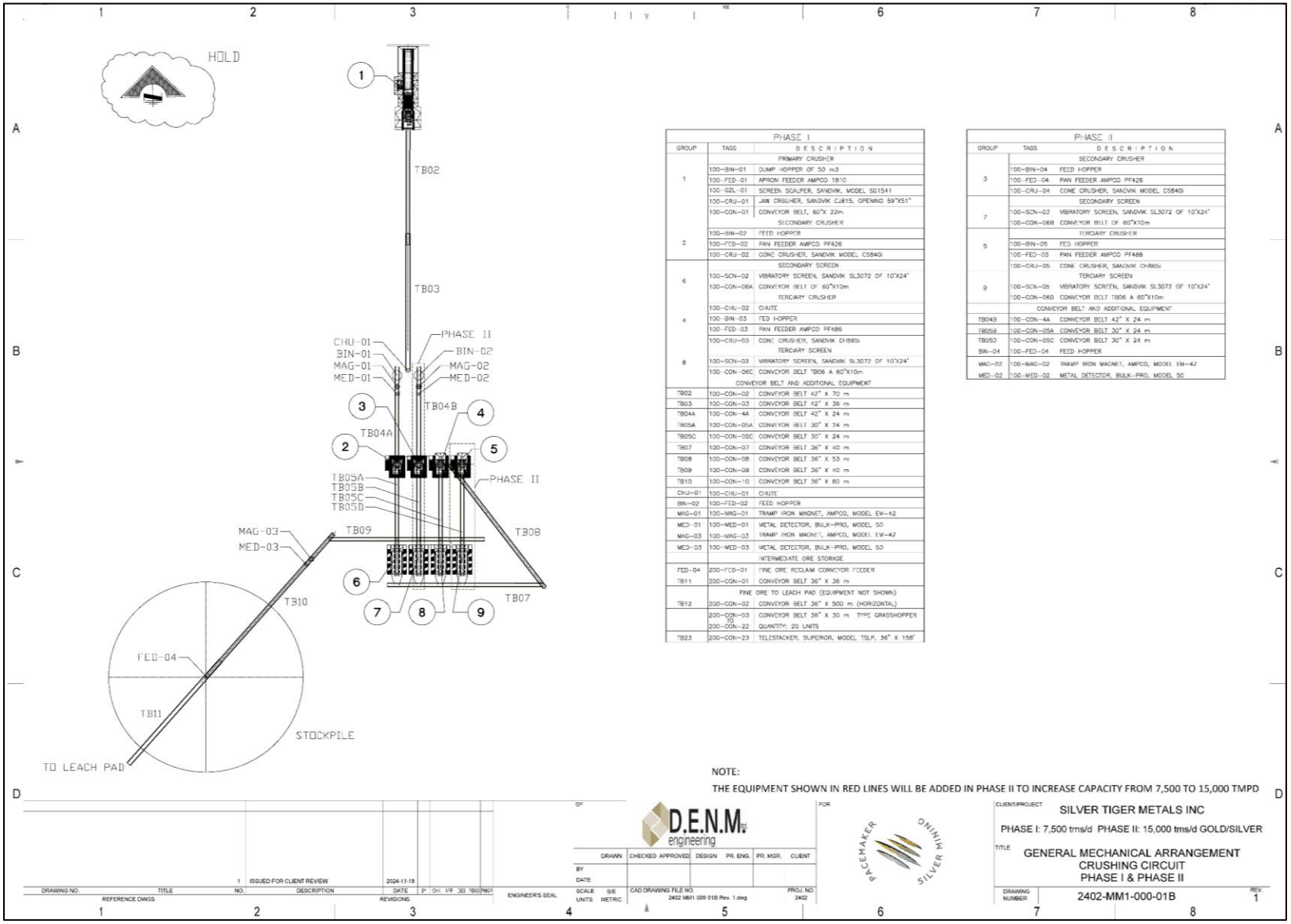
Details of Phase I and Phase II in this area are noted on Figures 17.3 through 17.5 that illustrate the crushing circuit layout and simulation.

FIGURE 17.3 CRUSHING CIRCUIT PHASE I



Source: *D.E.N.M. (2024)*

FIGURE 17.4 CRUSHING CIRCUIT PHASE II

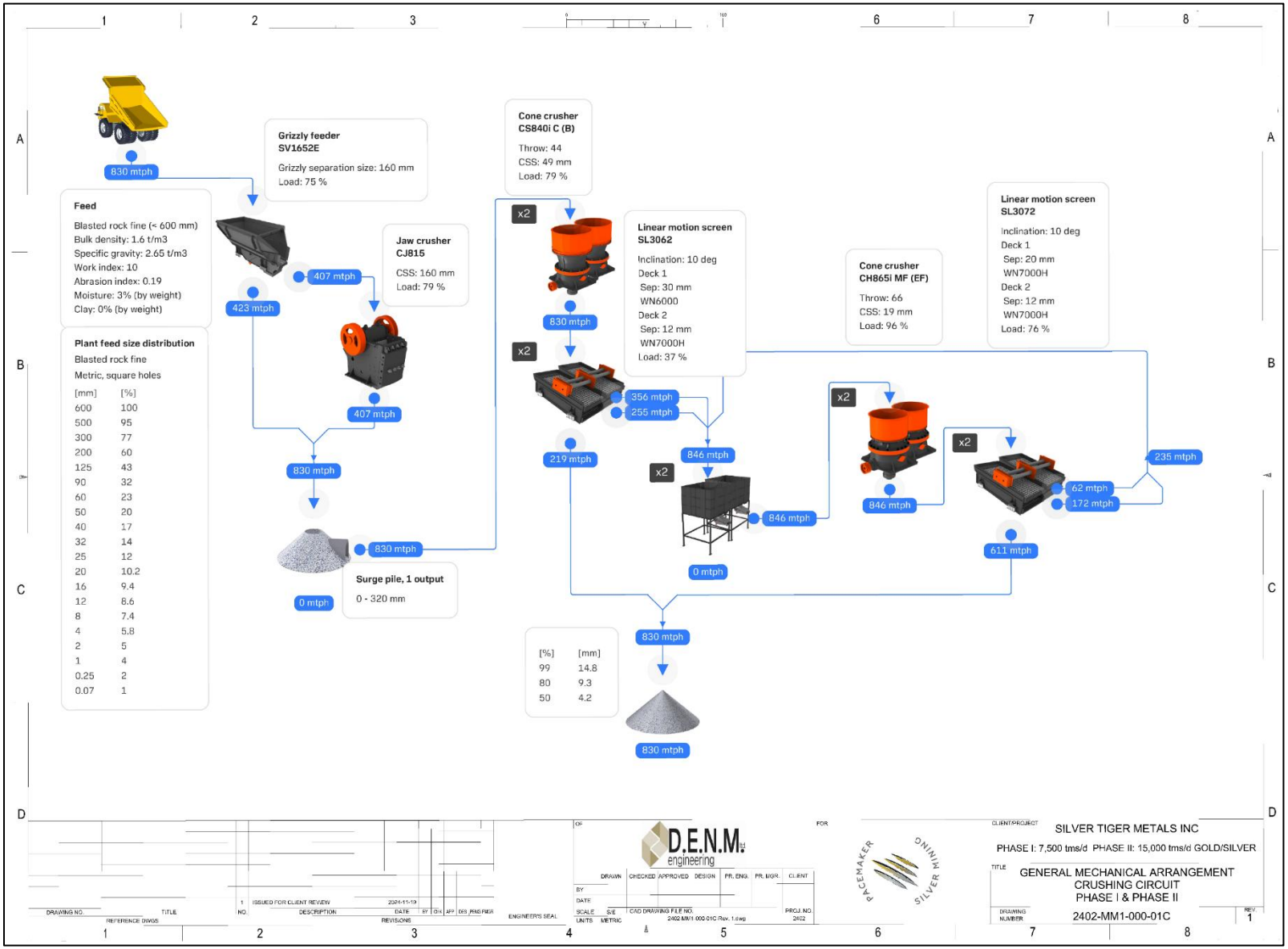


PHASE I		
GROUP	TAGS	DESCRIPTION
		PRIMARY CRUSHER
1	100-BIN-01	DUMP HOPPER OF 50 m3
	100-FED-01	APRON FEEDER AMPCO 1810
	100-GZL-01	SCREEN SCALPER, SANDVIK, MODEL SG1541
	100-CRU-01	JAW CRSUHER, SANDVIK CJ815, OPENING 59"X51"
	100-CON-01	CONVEYOR BELT, 60"X 22m
		SECONDARY CRUSHER
2	100-BIN-02	FEED HOPPER
	100-FED-02	PAN FEEDER AMPCO PF426
	100-CRU-02	CONE CRUSHER, SANDVIK MODEL CS840i
		SECONDARY SCREEN
6	100-SCN-02	VIBRATORY SCREEN, SANDVIK SL3072 OF 10'X24'
	100-CON-06A	CONVEYOR BELT OF 60"x10m
		TERCIARY CRUSHER
4	100-CHU-02	CHUTE
	100-BIN-03	FED HOPPER
	100-FED-03	PAN FEEDER AMPCO PF486
	100-CRU-03	CONE CRUSHER, SANDVIK CH865i
		TERCIARY SCREEN
8	100-SCN-03	VIBRATORY SCREEN, SANDVIK SL3072 OF 10'X24'
	100-CON-06C	CONVEYOR BELT TB06 A 60"X10m
		CONVEYOR BELT AND ADDITIONAL EQUIPMENT
TB02	100-CON-02	CONVEYOR BELT 42" X 70 m
TB03	100-CON-03	CONVEYOR BELT 42" X 36 m
TB04A	100-CON-4A	CONVEYOR BELT 42" X 24 m
TB05A	100-CON-05A	CONVEYOR BELT 30" X 24 m
TB05C	100-CON-05C	CONVEYOR BELT 30" X 24 m
TB07	100-CON-07	CONVEYOR BELT 36" X 40 m
TB08	100-CON-08	CONVEYOR BELT 36" X 53 m
TB09	100-CON-09	CONVEYOR BELT 36" X 40 m
TB10	100-CON-10	CONVEYOR BELT 36" X 80 m
CHU-01	100-CHU-01	CHUTE
BIN-02	100-FED-02	FEED HOPPER
MAG-01	100-MAG-01	TRAMP IRON MAGNET, AMPCO, MODEL EM-42
MED-01	100-MED-01	METAL DETECTOR, BULK-PRO, MODEL 50
MAG-03	100-MAG-03	TRAMP IRON MAGNET, AMPCO, MODEL EM-42
MED-03	100-MED-03	METAL DETECTOR, BULK-PRO, MODEL 50
		INTERMEDIATE ORE STORAGE
FED-04	200-FED-01	FINE ORE RECLAIM CONVEYOR FEEDER
TB11	200-CON-01	CONVEYOR BELT 36" X 36 m
		FINE ORE TO LEACH PAD (EQUIPMENT NOT SHOWN)
TB12	200-CON-02	CONVEYOR BELT 36" X 500 m (HORIZONTAL)
	200-CON-03 TO 200-CON-22	CONVEYOR BELT 36" X 30 m TYPE GRASSHOPPER QUANTITY: 20 UNITS
TB23	200-CON-23	TELESTACKER, SUPERIOR, MODEL TSLP, 36" X 158'

PHASE II		
GROUP	TAGS	DESCRIPTION
		SECONDARY CRUSHER
3	100-BIN-04	FEED HOPPER
	100-FED-04	PAN FEEDER AMPCO PF426
	100-CRU-04	CONE CRUSHER, SANDVIK MODEL CS840i
		SECONDARY SCREEN
7	100-SCN-03	VIBRATORY SCREEN, SANDVIK SL3072 OF 10'X24'
	100-CON-06B	CONVEYOR BELT OF 60"X10m
		TERCIARY CRUSHER
5	100-BIN-05	FED HOPPER
	100-FED-05	PAN FEEDER AMPCO PF486
	100-CRU-05	CONE CRUSHER, SANDVIK CH865i
		TERCIARY SCREEN
9	100-SCN-05	VIBRATORY SCREEN, SANDVIK SL3072 OF 10'X24'
	100-CON-06D	CONVEYOR BELT TB06 A 60"X10m
		CONVEYOR BELT AND ADDITIONAL EQUIPMENT
TB04B	100-CON-4A	CONVEYOR BELT 42" X 24 m
TB05B	100-CON-05A	CONVEYOR BELT 30" X 24 m
TB05D	100-CON-05C	CONVEYOR BELT 30" X 24 m
BIN-04	100-FED-04	FEED HOPPER
MAG-02	100-MAG-02	TRAMP IRON MAGNET, AMPCO, MODEL EM-42
MED-02	100-MED-02	METAL DETECTOR, BULK-PRO, MODEL 50

Source: *D.E.N.M. (2024)*

FIGURE 17.5 CRUSHING CIRCUIT SIMULATION



Source: *D.E.N.M. (2024)*

17.3.4 Heap Leach Pad System and Solution Distribution

The heap leach pad areas cover 168,000 m^2 (Phase I) and 268,000 m^2 (Phase II) of LLDPE material. The stage pad areas are complete with all associated collection piping, geotextile, and supporting items.

17.3.5 Merrill Crowe (Zinc Precipitation) Circuit

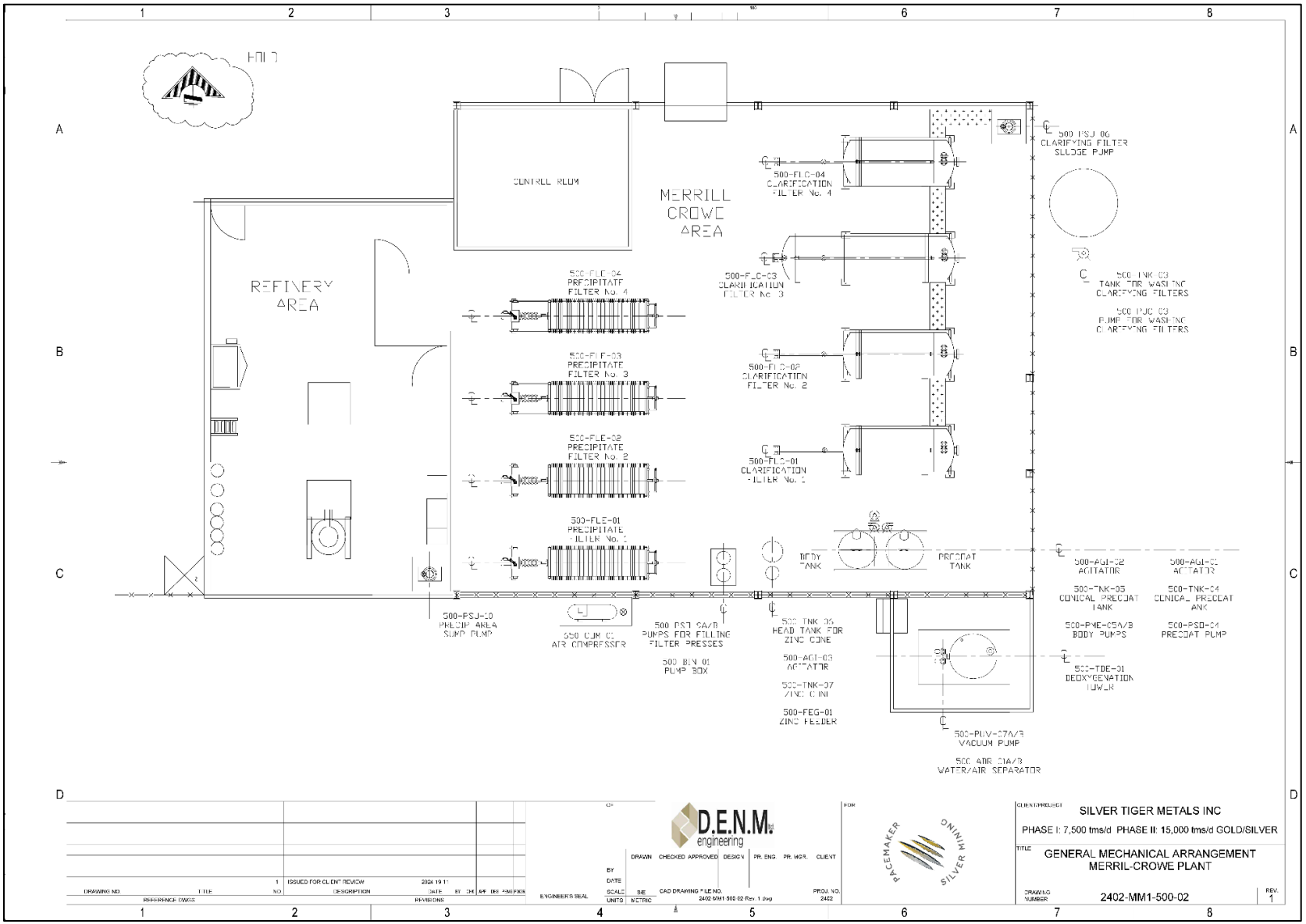
The loaded pregnant solution is pumped from the pond to the Merrill Crowe facility located within a housed building for both safety and security. The location is indicated on the site plan shown in Section 18. The recovered gold and silver precipitate is fluxed and smelted at the on-site refinery.

Equipment includes:

- Three Clarification Filters complete with pumps, and samplers 180 m^2/unit; additional unit for Phase II;
- 4 m diameter Deaeration Tower complete with packing, level control, 45 Hp vacuum pump with separator;
- Zinc Feeding System complete with lead nitrate feed tank;
- Two Precipitate press feed pumps complete with seal 75 Hp;
- Three Precipitate filters; 180 m^2/unit recessed plates, feed and discharge piping, precipitate trolley; additional unit for Phase II;
- Diatamaceous Earth (PreCoat) Skid; mix tank, agitators, feed pumps;
- Secure Refinery Area; 0.30 m^3 induction Bullion Furnace complete with baghouse, power controls, scales and pots. Sized for LOM service; and
- Motor Control Center system (3/60/460V and 110/220) main disconnect, starters, and control package.

The Barren Leach Solution ("BLS") discharges to the barren solution pond after NaCN and lime reagent addition and recirculation, Figure 17.6.

FIGURE 17.6 MERRILL CROWE PLANT



Source: *D.E.N.M. (2024)*

17.3.6 Reagent Handling and Storage

Fresh water pumped from two wells will be utilized for all reagent mixing within the process plant and for make-up water to the heap pads for evaporation and wetting fresh feed material. The process plant also includes a mixing area containment.

Process Plant required reagents include:

- Lime (hydrated) – bulk for silo;
- Sodium Cyanide (NaCN); dry super sacs;
- Diatamaceous Earth (PreCoat); bagged;
- Zinc Dust; canned; and
- Refinery Slagging reagents; bagged.

17.3.7 Assay and Metallurgical Laboratory

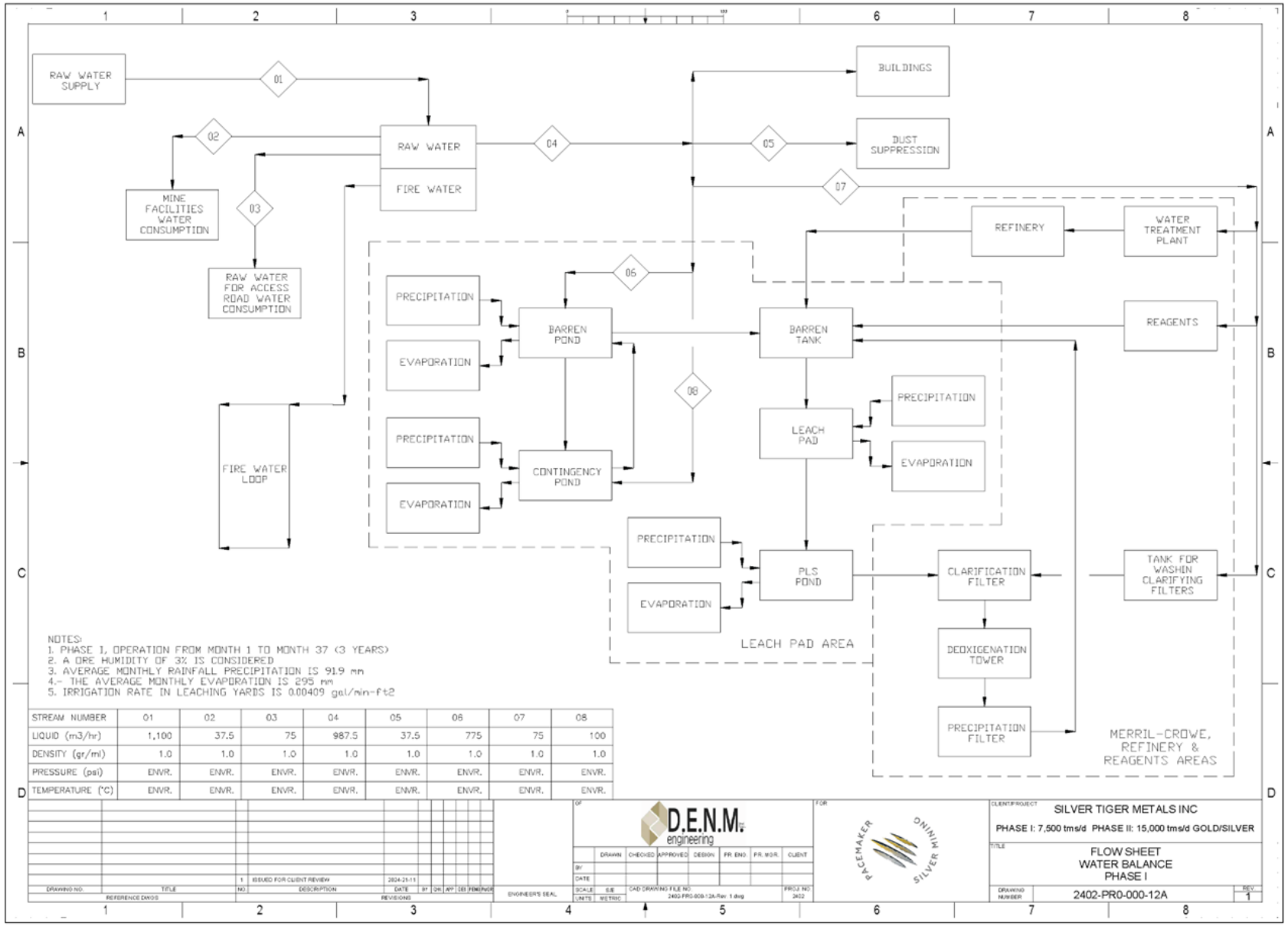
A fully equipped stand-alone laboratory is an integral part of the El Tigre processing system. Located close to the main process facility, it is equipped with the necessary analytics to provide all required data for the mining operation, main process facility, and environmental monitoring requirements.

The laboratory will also play an instrumental role in providing on-time process monitoring of processes, daily production reporting, blast hole sampling, and any and all exploration samples.

17.3.8 Water Supply

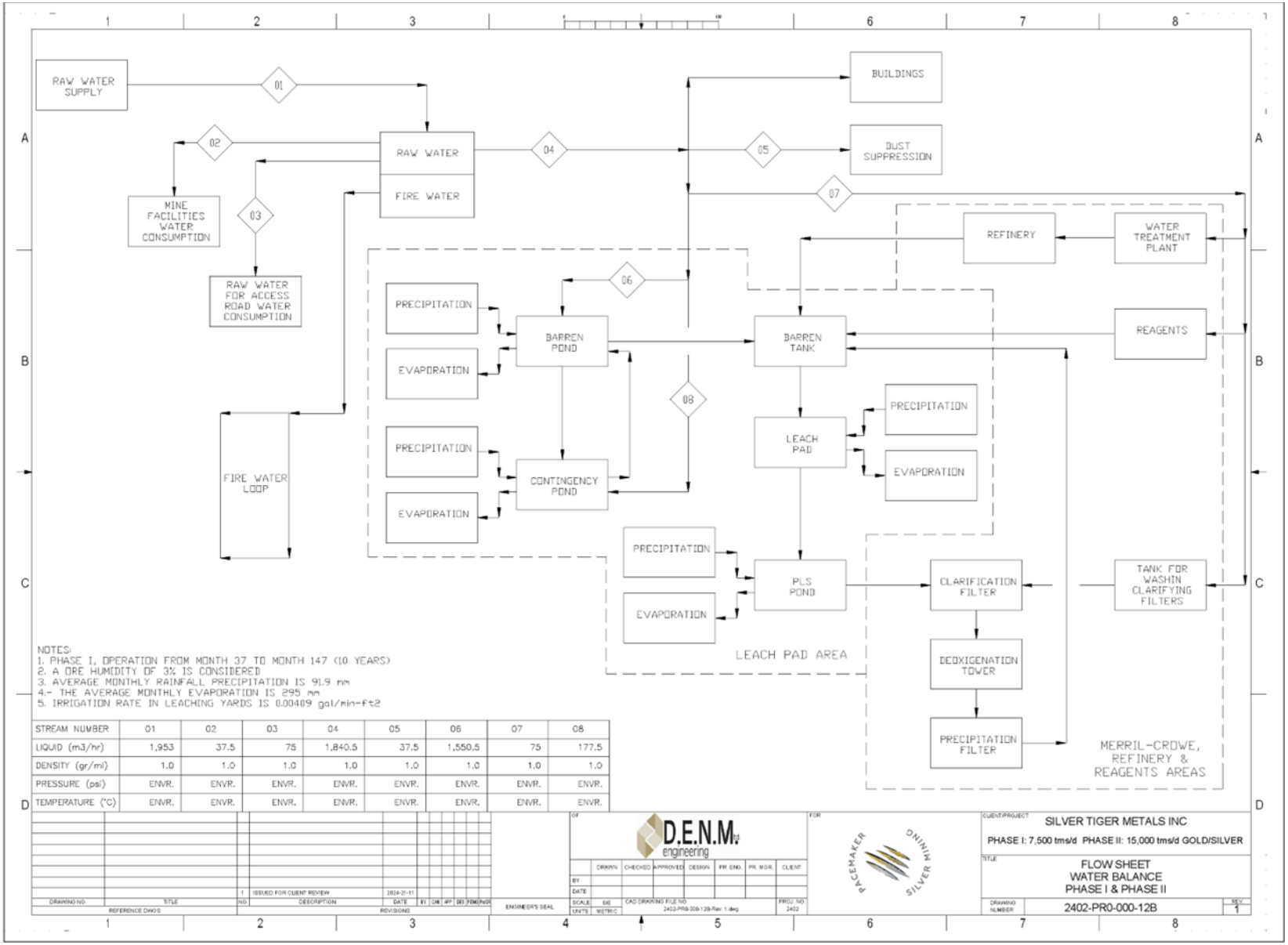
Well water will supply all facets of the Project including make-up water from the process (loss from evaporation), reagent mixing, and emergency water. Treated water will be supplied to a fully operational on-site camp facility. The Project water balances for Phase I and II are illustrated in Figures 17.7 and 17.8, respectively.

FIGURE 17.7 WATER BALANCE PHASE I



Source: *D.E.N.M. (2024)*

FIGURE 17.8 WATER BALANCE PHASE II



Source: *D.E.N.M. (2024)*

In addition to the well water supply, rain and run-off water during the rainy season will also to be diverted and collected. Multiple high head pumps are installed at the water sources to pump water to the process plant's freshwater tank.

17.3.9 Air Supply

An air distribution system is included to supply required process air to the main Merrill Crowe plant facility and instrument air is included for required instrumentation and controls.

17.3.10 Standby Power

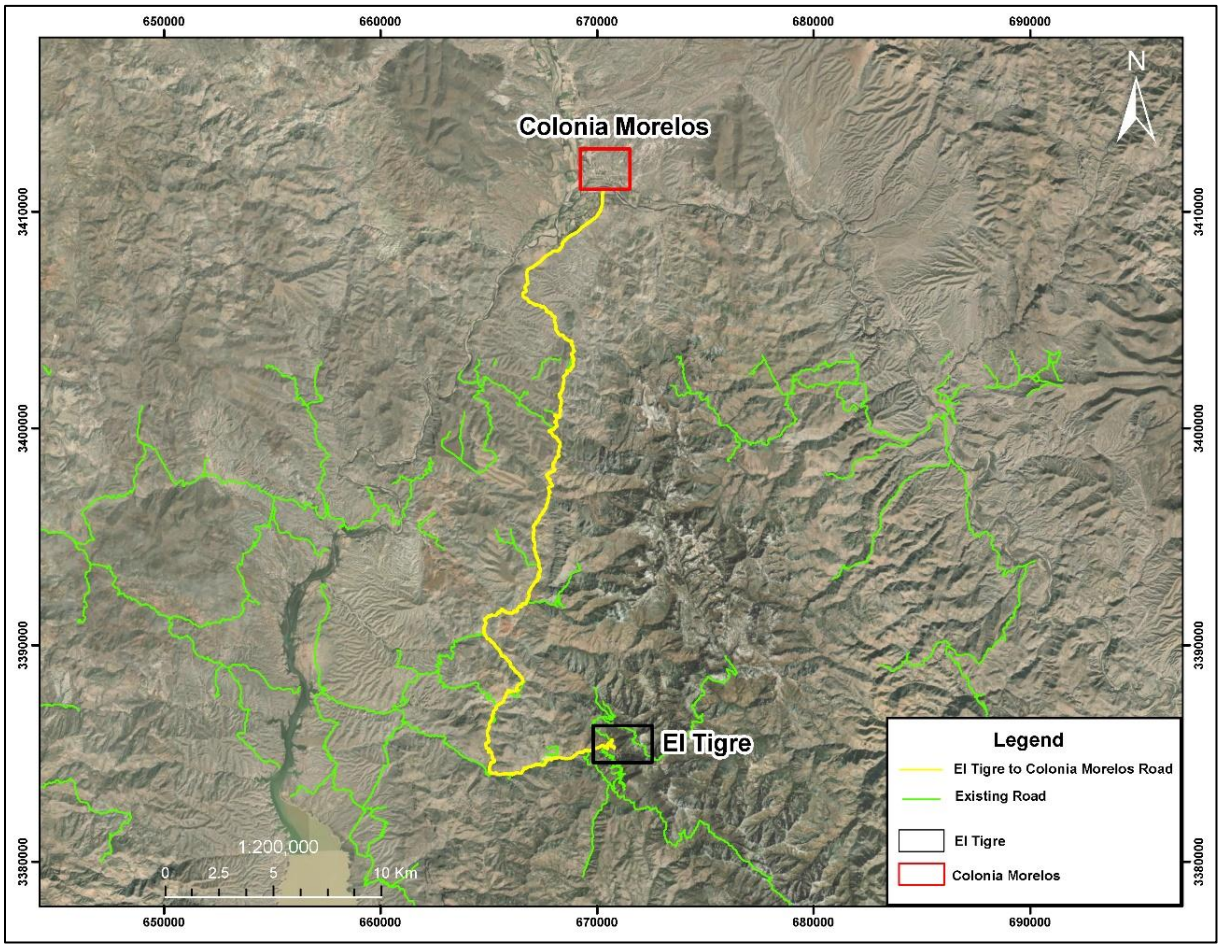
An emergency generator rated for 1,825 kW will supply electrical power for essential process operations during shutdown or system failure (process and solution pumps).

18.0 PROJECT INFRASTRUCTURE

Current infrastructure at the El Tigre Project consists of an exploration camp that has accommodation for drillers and site administration personal. Electricity for the camp is supplied by diesel generators, and water for drilling is supplied by existing underground workings (Level 7). There are existing impounded tailings on site (852 kt Measured and Indicated Mineral Resource with 52 kt Inferred Mineral Resource) and low-grade stockpile material (101 kt Measured and Indicated Mineral Resource with 18 kt Inferred Mineral Resource). While tailings and low-grade stockpile are part of the Project's Mineral Resource, they are not part of the production plan for this PFS. However, both the tailings and low-grade stockpile materials could be processed at a future date if economically viable.

Permanent road access to the site is from Colonia Morelos, located in the Town of Agua Prieta (Figure 18.1). Total distance to the site is 46 km and it traverses the mountain ranges of the Sierra Pilares de Teras, Sierra las Delicias, and the Sierra de Enmedio. The road was upgraded in 2023 making it accessible all year round. Upon completion of final upgrades, the road will be adequate for manpower movement, access for process plant construction equipment and material, mining and drilling equipment, consumables and safety and emergency requirements.

FIGURE 18.1 ACCESS ROUTE



Source: *Pacemaker (2023)*

In early 2023, Silver Tiger commenced road work on the access road with capital improvements to the site, including:

- Improvement of bearing surface;
- Modifications of both horizontal and vertical alignments;
- Curves expanded and increased radius on switchbacks; and
- Reduced slopes to no greater than 15%.

Pending road work required is the installation of water diversion culverts along the route to minimize road wash-outs and deterioration.

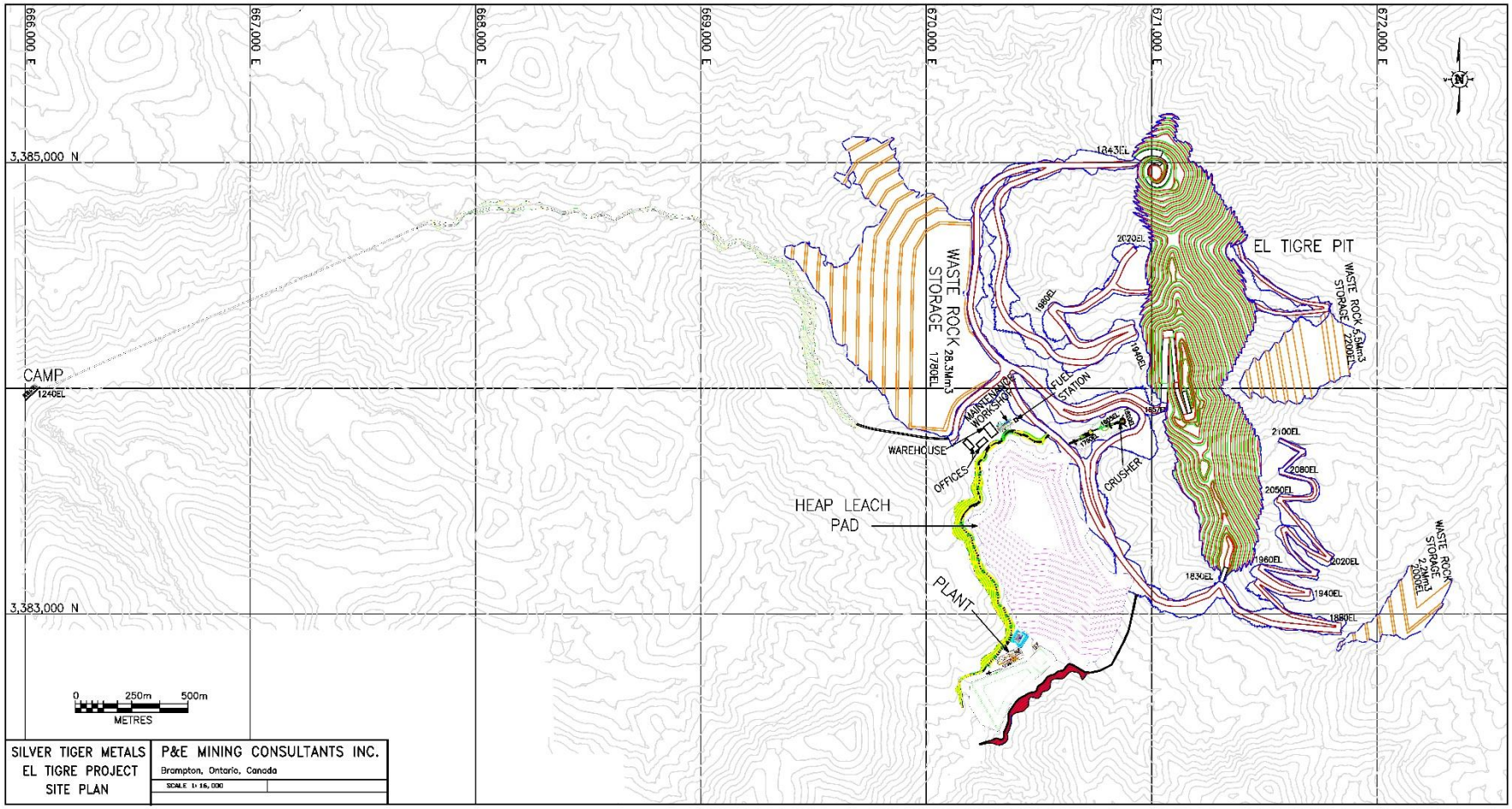
The Authors are of the opinion that there appear to be no obvious impediments to building a surface mine, heap leach processing facility within the area of the El Tigre Project Concessions.

18.1 PLANNED INFRASTRUCTURE

Figure 18.2 shows the major surface infrastructure proposed at the El Tigre Project, which consists of:

- Surface Mine and Waste Rock Storage Facilities;
- Site Access Roads;
- Three-Stage Crushing Circuit with associated material handling components;
- Heap Leach Pad and solution distribution system complete with pumping and piping;
- Heap Leach Ponds - pregnant, barren, and overflow complete with pumping and piping;
- Merrill Crowe (zinc precipitation) Circuit for recovery of gold and silver from pregnant solution stream;
- Building and reagent mixing within the process plant area;
- Refinery - bullion furnaces for doré production including dust collection system;
- Electrical Power Supply from the national grid, and distribution; and
- Assay and Metallurgical Laboratory.

FIGURE 18.2 PROPOSED INFRASTRUCTURE AT THE EL TIGRE PROJECT



Source: *P&E (2024)*

Additional infrastructure to be installed includes:

- Gatehouse and security on the main access road;
- Main office for administration, purchasing, safety and technical personnel;
- Warehouse for all mechanical and process plant parts;
- Fuel storage and distribution facility to be installed by fuel supplier;
- Maintenance workshop to be installed by mining contractor;
- Communications - telephone, cellular, and internet;
- Two water wells drilled from surface;
- Water storage and sewage treatment plant;
- Secure explosives storage facility; and
- 300-person camp to house all site personnel.

Water needed for infrastructure and the process plant will be supplied via two wells. Power for pumping from the wells will be supplied by the process plant's electrical power and distribution system.

The water system includes a main freshwater tank to supply process water and any needed firefighting requirements. Water is also available via the underground workings if needed, however, further testing is required to confirm quality and volume. Water from the underground workings may have mild contamination and would need simple lime and flocculant treatment using mix tanks with settlement in a multi-stage lined basin.

Potable water required for the Project's infrastructure will be supplied by a water filtration treatment system. Details of the water balance for the process plant is included in Section 17 of this Technical Report.

18.2 WATER MANAGEMENT

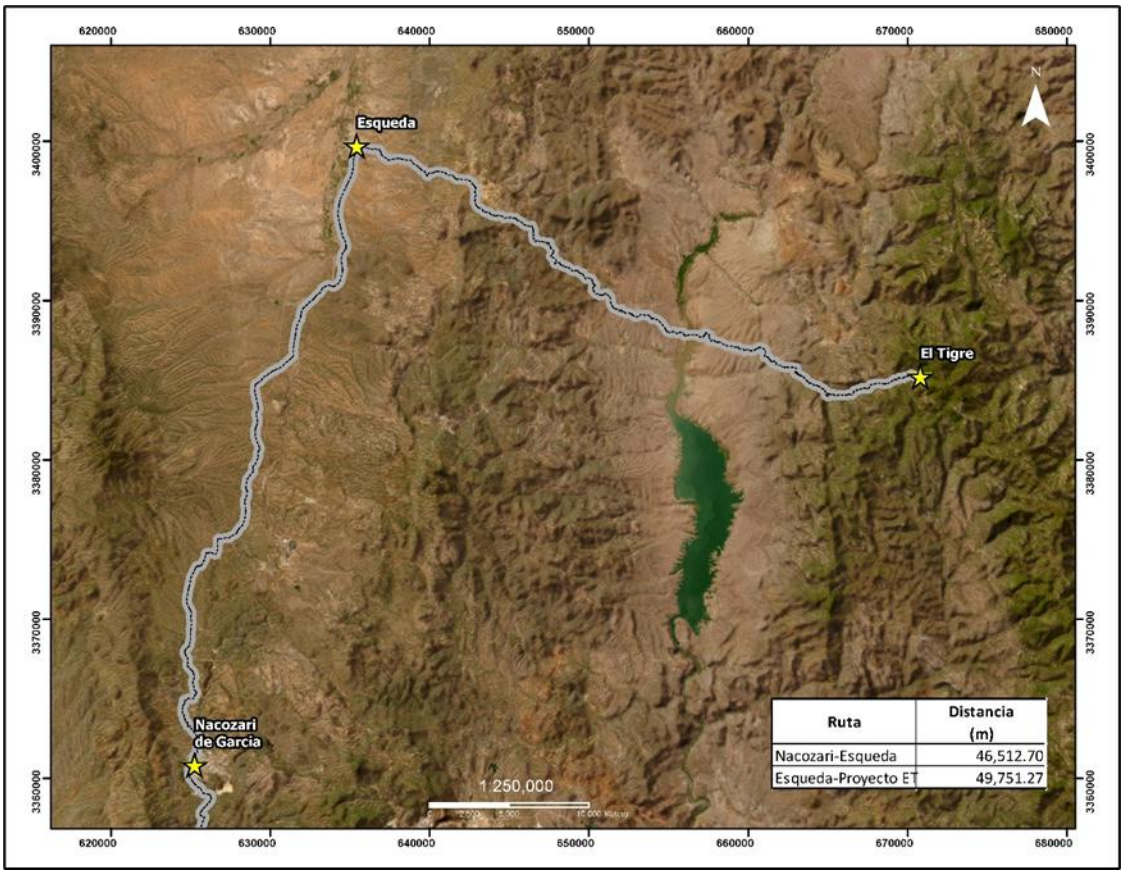
Water usage will be typical of a heap leach operation in the Sonora region of Mexico. The main make-up water demands are determined by the loaded heap pad wetting and irrigation evaporation in the area. The expected evaporation rate in the area is high and has been factored into the water balance outlined in Section 17 of this Technical Report.

Annual precipitation on the area is 500 mm and is high in the summer months with July recording an average 160 mm. Water diversion and management will be important as a means of collection, which will also limit the dilution within the pads and ponds of the gold and silver bearing solution.

18.3 ELECTRICAL POWER

Power to the El Tigre Property will be supplied by the national grid via a 72 km 34.5 kV overhead power line. The routing of the line will be from Nacozari de Garcia (Figure 18.3). Overhead power lines will connect 13.8 kV, three phase and 60 Hz via a sub-station located near the process plant area.

FIGURE 18.3 POWERLINE ROUTE



Source: *Pacemaker (2023)*

Based on the process flowsheet and equipment list, the process electricity consumption for the site is estimated at 3.7 MW for Phase I and 4.8 MW for Phase II after expansion for year 4 of operation.

Onsite accommodations for personnel will be powered by grid power as part of the internal process plant distribution system, and an emergency generator rated for 1,825 kW will supply power for essential process operations during shutdown or system failure (process and solution pumps).

18.4 WASTE ROCK STORAGE FACILITIES

WSP Mexico recommended the following design parameters for waste rock storage at El Tigre:

- The waste rock stockpile will be placed by truck dumping and dozer pushing with little compaction effort. Stacking is assumed to occur in angle of repose lifts, in a benched sloped configuration of 25 m high and 10 m wide crests at 1.5H:1V batter slopes, to achieve an overall slope of 1.9H:1V. For closure conditions, the overall slope would require regrading to achieve flatter slopes for revegetation, and include contour drain construction on any remaining benches tied to downslope drains constructed to deliver rainfall runoff to the toe.

- Alternatively, for a progressive reclamation, bench batter slopes could be constructed at 1.9H:1V, with an overall slope of 2.4H:1V, to reduce the amount of work required for closure implementation.

- The main waste rock storage facility was designed to the progressive reclamation slopes. The facility will hold 28.3 Mm^3 of material, which is approximately 80% of the total 68.3 t of waste rock mined over the LOM.

19.0 MARKET STUDIES AND CONTRACTS

Metal prices are based on an approximate average of December 31, 2025, three-year monthly trailing averages, and Consensus Economics Inc. long term price forecasts, and are presented in Table 19.1. The Mexican Peso:US Dollar exchange rate is based on the approximate past three-year average. The metal prices and exchange rate are subject to spot market conditions. There are no metals streaming or hedging agreements in place.

TABLE 19.1
METAL PRICES AND EXCHANGE RATE

Item	Price
Gold (US$/oz)	3,200
Silver (US$/oz)	38.00
Exchange Rate ($MXN:US$)	20

Note: *$MXN = Mexican peso, US$ = United States dollars.*

Currently there are no contracts in place that are material to the El Tigre Project.

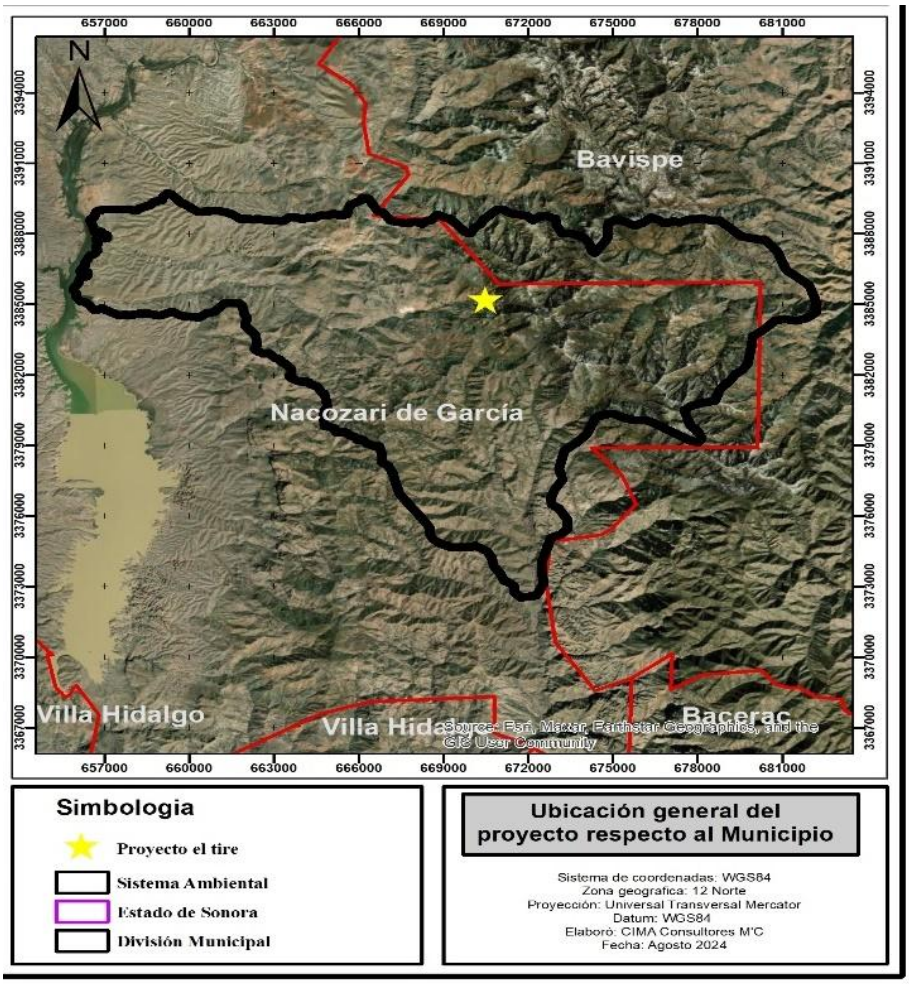
20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The following is a Summary of the original Spanish report "Reporte Técnico 43-101 El Tigre 2024 – Apartado Ambiental" prepared in August 2024 by Consultores Interdisciplinario en Medio Ambiente S.C. ("CIMA"). This document was translated into English by CIMA. The following Summary was prepared by the Authors in November 2024 and has been composed to represent the CIMA detailed report.

20.1 PROJECT LOCATION

The Project is located in the northeast of the state of Sonora, between the limits of the municipalities of Bavispe and Nacozari de García. It is located 240 km east-northeast of the City of Hermosillo in a location known as the as the "El Tigre" mountain range. The proposed Project footprint of 194 ha is only 3.1% of the area identified as Sistema Ambiental (Environment System) which is related to the Project (Figure 20.1).

FIGURE 20.1 EL TIGRE PROJECT LOCATION AND RELATED ENVIRONMENTAL SYSTEM



Source: *CIMA (2024)*
Note: *El Tigre Project Location (yellow star) and Related Environmental System (black line).*

20.2 PROJECT DEVELOPMENT

The El Tigre Project is expected to be developed and operated over 11.5 years in three stages: site preparation and construction (1.5 years); operation and maintenance (9 years); and closure/post-closure (1 year).

20.2.1 Site Preparation and Construction

Once the environmental authorizations and Project permits have been obtained, and base environmental requirements addressed, project site preparation can be initiated. First steps will include project area-identified forest clearing and excavations, removal and stockpile of topsoil, fencing and restriction of industrial areas and the rehabilitation of existing access roads. Deforestation will be carried out gradually to ensure the safe movement of fauna to less disturbed areas. The plant material resulting from the clearing will be chopped and incorporated into the soil stored in the fertile soil storage area, which will be located to the north of principal Project facilities.

In the Project area there are currently no electricity services. High voltage grid power will be installed by the local utility to supply mining, process and infrastructure electrical needs.

Project water supply to the process plant will be provided by pumping from two wells to the process area water distribution system. Later, some process water may be sourced from historical underground mine openings e.g., Level 7. Domestic wastewater from the construction camp and operational facilities will be treated in a Wastewater Treatment Plant ("WWTP").

The construction of industrial infrastructure, auxiliary infrastructure, leach pads, water and solution management facilities will be the major components of Project development.

20.2.2 Operations

The El Tigre Project will be a surface mine and heap-leach operation to recover silver and gold. Crushed ore will be processed for the first three years at a processing rate of 7,500 tpd, increasing to 15,000 tpd by year 4.

The general operations processes include blasting activities in the mine, followed by the extraction and transfer of ore by large mine haulage trucks to a jaw crusher within the Project site. The ore processing will begin with the crushing of run-of-mine ("ROM") material with the jaw crusher, following by two additional cone crushing stages to produce a 10 mm (-3/8 in.) product that will be conveyed to double-lined leach pads by conveyors and distributed by mobile conveyors sometimes called "grasshoppers". A series of pumping and piping systems will allow the irrigation of the stacked heap material with alkaline cyanide solution which extracts the gold and silver from the crushed ore. Subsequent gold and silver containing solution is produced which flows down through the leach pad to a collection pond. The heap irrigation will be performed by a network of dripping pipes laid out on the flat surface of the heap leach pad. Gold and silver containing leachate, known as "pregnant solution" will be collected by sub-pad membranes and pumped to the recovery plant for metal recovery. The recovery technology will involve zinc precipitation of the precious metals from a clarified solution – a process called Merrill Crowe ("MC"). The

combined metal precipitate will be refined in an on-site furnace that produces gold-silver doré bars which are sent to market.

The metal-stripped MC solution, called barren solution, will be recycled to the heap leaching process following replenishment of leaching agents.

The process plant will be located adjacent to the leach pads and associated ponds.

Water supply for the process plant will be provided by pumping from the nearby Bavispe River to the process area water distribution system. High voltage grid power will be installed by the local utility to supply the process operations and infrastructure electrical requirements.

Expansion capital includes the cost to increase the process plant capacity from 7,500 tpd to 15,000 tpd as noted in Year 4 of operation.

20.2.3 Waste Management

The ore will be crushed and treated in the leaching system. Associated waste rock, low in metal content, will be disposed of in a specially designed surface storage area. For this facility, vegetation clearing and top soil removal will be needed. Both the ore and the waste rock will be transported in large trucks with off-road dimensions along dedicated roads and ramps. For these roads and ramps, clearing and removal of the top soil will be required, as well as excavations. The recovered top soil will be stockpiled for restoration purposes in the closure phase.

Other major waste materials include leached and detoxed leach pads, which will left in place and treated when there is excess barren solution due to significant rainfall events. This solution will also be detoxed by oxidation of residual cyanide.

Domestic waste and other specific waste management programs, such as for expired rubber tires and spent oils will be put in place. Where applicable, recycling of some commodities will be put into effect.

20.2.4 Closure

For the post-operational stage of the Project, development of a Closure Plan is required. The Closure Plan will include the following:

- Health and safety goals and objectives;
- Physical stability objectives;
- Land use objectives;
- Hydrological objectives; and
- Social objectives.

The Closure Plan will also include:

- Provision for passive care;
- Environmentally appropriate procedures for the demolition and/or dismantling of facilities;
- A monitoring plan to corroborate closure objectives; and
- Preparation of cost estimates that will be reviewed and updated periodically.

Although a Closure Plan does not currently exist, Closure costs for the Project have been estimated at $6.2M in Section 21.3 of this Technical Report.

20.3 HISTORICAL ACTIVITY AND RESIDUAL INFRASTRUCTURE

The historical mining activity in the area dates back to 1896, the year in which gold was first discovered. Several underground mines were developed and operated.

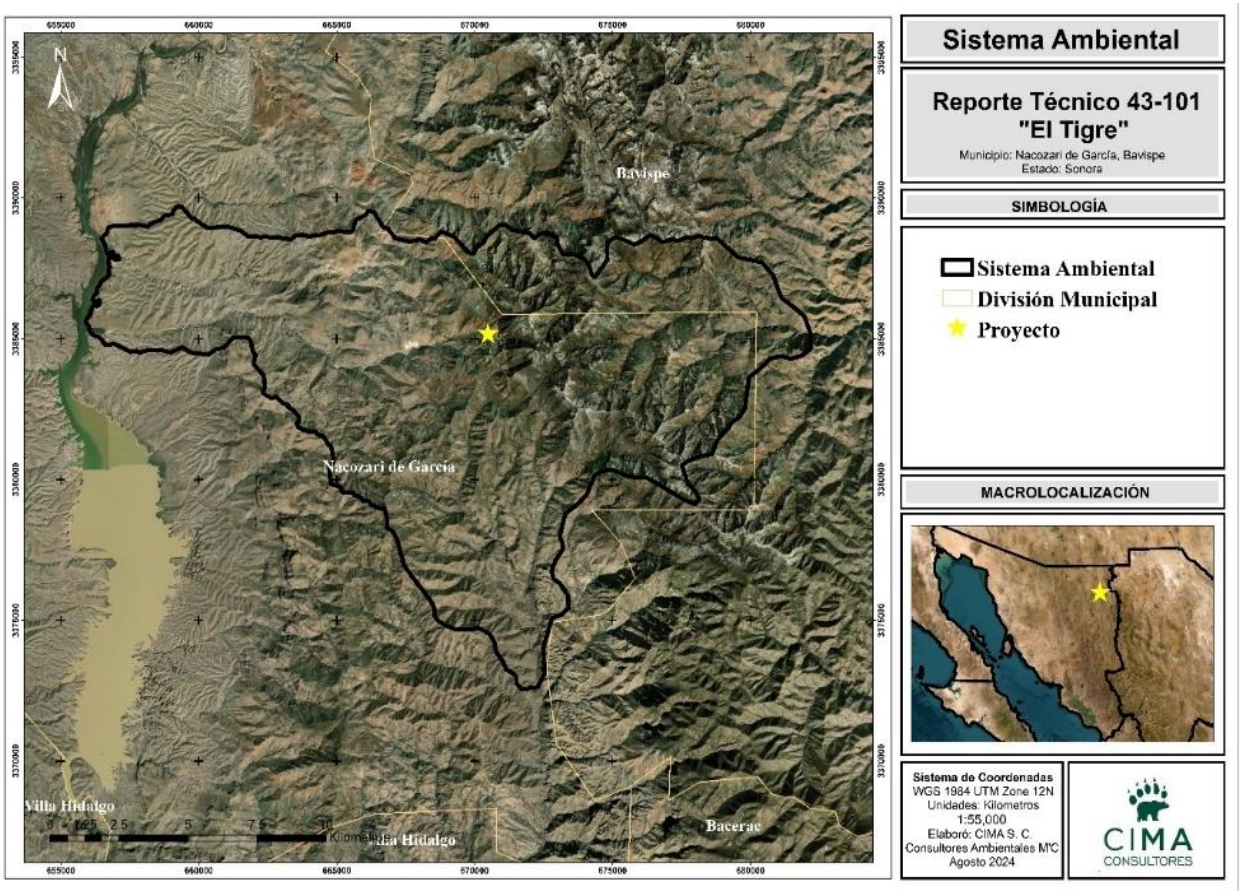
Mining came to an abrupt halt in these veins when the price of silver collapsed to less than 20¢ per ounce with the onset of the Great Depression in 1930. When the mine closed, it was reported to have produced a total of 353,000 ounces of gold and 67.4 million ounces of silver out of 1.87 million tonnes mined. A total of 15.4 ha of disturbed areas are currently in existence (5.8 ha for the camp, main shaft and processing plant), 3.1 ha of waste rock and 6.5 ha of tailings. The tailings are located approximately 2.2 km west of the original mine camp site and 2 km west of the processing site. No environmental liability has been identified that is related to these sites.

Archaeological sites have been located in the state of Sonora, specifically the lower basin of the Bavispe River in the mountainous area of the northwest of the state. The set of archaeological sites are located in the vicinity, however, not within the El Tigre environmental system.

20.4 ENVIRONMENTAL SYSTEM

An environmental system ("ES") was identified according to its components and its significant environmental factors (flora, land use, hydrology, socioeconomic, etc.) with which the Project is expected to interact at some time. It was possible to outline an ES according to the extent of the possible impacts that could be generated. The ES represents a total of 20,900 ha, within which the Project will occupy 194 ha (Figure 20.2).

FIGURE 20.2 EL TIGRE ENVIRONMENTAL SYSTEM (SISTEMA AMBIENTAL)



Source: *CIMA (2024)*

In addition to identification of the boundaries of the ES (as shown in Figure 20.2), the following components in the ES have been studied in some detail:

- Soils;
- Surface and underground hydrology;
- Water quality;
- Climate;
- Flora and fauna;
- Landscape;
- Environmental disturbance and fragility; and
- Cultural and human status.

20.5 ENVIRONMENTAL PERMITTING

20.5.1 Federal Studies, Procedures or Authorizations of the Project

Currently, the execution of mining projects involves extensive laws, regulations and procedures, necessary for the approval of a mining project. Failure to comply with any procedure, permit, authorization, as well as the omission of the terms and conditions of permits previously resolved

in a favorable manner, may generate various sanctions, which range from economic, partial, total, temporary or permanent suspension of works or activities, as well as the revocation of previously authorized permits.

SEMARNAT is the body with the greatest involvement and regulation of mining projects, which are regulated by the LGEEPA and LDFS for the most part, the first being in environmental matters and the second in forestry. However, it is not the only regulatory agency, leaving other important ones such as CONAGUA, CONANP, INAH to mention just a few. These procedures are required according to the type of activities or stages in which the project is developed, mainly divided into exploration, construction, operation, closure and post-closure. Those studies applicable to the project are presented in Table 20.1.

TABLE 20.1
FEDERAL LAWS AND PERMITS

Step	Government Agency
Registration of hazardous waste management	SEMARNAT
Use of federal channels	CONAGUA
Annual Operation Certificate	SEMARNAT
Environmental Risk Study	SEMARNAT
Technical Justification Study for Change of Land Use in Forest Land	SEMARNAT
Feasibility of Electric Energy (Electric Power Contract).	CFE
Preventive Environmental Impact Report	SEMARNAT
Single Environmental License	SEMARNAT
Operating License for Fixed Sources of State Jurisdiction.	SEMARNAT
Environmental Impact Statement - Particular	SEMARNAT
Environmental Impact Statement - Private Party at Risk	SEMARNAT
Environmental Impact Statement - Regional	SEMARNAT
Environmental Impact Statement - Regional with risk	SEMARNAT
Permit for Access and Other Facilities on Free Federal Highways.	SCT
Sewage discharge permit	CONAGUA
Explosives Storage and Use Permit	SEDENA
Permit to Build Hydraulic Works	CONAGUA
Special Handling Waste Management Plan	ECOLOGY
Mining Waste Management Plan	SEMARNAT
Hazardous Waste Plan	SEMARNAT
Accident Prevention Program	SEMARNAT
Registration of the Company with the Ministry of Health (SS) and Municipal Administrations of the Sanitary License, and Sanitary Control Cards	FF
Registration in the Mexican Business Information System (SIEM)	HERSELF
Title of Concession or Assignment of National Water Uses (Surface and Groundwater)	CONAGUA
Concession Title for Extraction of Materials	CONAGUA

TABLE 20.1 FEDERAL LAWS AND PERMITS	
Step	**Government Agency**
Unified procedure for change of land use.	SEMARNAT
Registration of the Business Registry with the IMSS	IMSS
Procedures before CNA for the Installation of a Company Not Connected to the Municipal Network	CNA
Single Environmental License (LAU)	SEMARNAT
Annual Operating Certificate (COA)	SEMARNAT

20.5.2 State and Municipal Permits and Authorizations

The states of the Mexican Republic have different laws and regulations of state that must be considered. These are summarized in Table 20.2.

TABLE 20.2 STATE AND MUNICIPAL PERMITS AND AUTHORIZATION	
Process	**Government Agency**
Zoning Certificate	Municipality
Environmental risk study	State
Preventive Environmental Impact Report	State
Security inspection for explosives	SEDENA
Municipal Building License	Municipality
Operating license	Municipality, if there is no procedure, State
Land Use License	Municipality
Environmental Impact Statement	State
Internal Civil Protection Program	State/Municipal Civil Protection
Special Handling Waste Generator Registration	State

20.5.3 Environmental and Safety Regulations

The exploration, exploitation and processing of materials are subject to a wide range of regulations methodologies and specifications.

These are focused on:

- Air emissions (10 regulations);
- Hazardous waste (9 regulations);
- Water management (4 regulations);
- Noise management and soil remediation (3 regulations);
- Protection of flora and fauna (2 specifications);

- Safety and health (8 regulations); and
- Organization rules (6 specifications).

20.6 COMPLETED ENVIRONMENTAL STUDIES

The El Tigre Project Management has completed an environmental baseline study, as well as a community diagnosis. For its elaboration, it was necessary to compile available documentation, as well as the collection of field data, both environmental and social, in order to describe the physical, environmental and social environment where the El Tigre Project is to be located. The preparation of the study began in 2022 with the collection of information. Annual biodiversity reports have also been produced since 2022.

Regarding procedures specific to mining activity, the Project title holder received the authorization of a Unified Technical Document of the Procedure for Change of Use of Forest Land ("DTU") through the resolution an Official Letter which modifies the DTU project "Presa de Jales" issued by the Federal Delegation of SEMARNAT of the state of Sonora, and which allowed it to carry out Land Use Change activities.

Favorable resolutions were received that allowed a drilling campaign in order to detect those minerals of interest and to explore the mineral bodies of the region.

On February 20, 2021, the Title holder submitted to SEMARNAT of the state of Sonora, the Environmental Impact Statement of the "Tigre Third Stage" project. On May 11, 2022, authorization was received for the Exemption of MIA-P Project "El Tigre" by the Federal delegation of SEMARNAT of the state of Sonora. SEMARNAT permitted that the works and activities for the rehabilitation of disturbed sites (cleaning and mapping of sites) were exempt from the presentation of an environmental impact statement.

On July 27, 2023, the Environmental Impact Statement "El Tigre" Private Modality was submitted and is currently being evaluated by the regulatory authorities.

20.6.1 Environmental Impact Assessment

As usual, the general environment is a dynamic system composed of natural and social elements that can be positively or negatively altered by various activities during the development of a project such as the El Tigre Project. An environmental impact refers to any modification that the Project may cause in the environment in which it is carried out, including (favourable or unfavourable) changes in socioeconomic and health conditions, physical, chemical and biological characteristics, cultural aspects of land use and resources, as well as the preservation of historical and archaeological sites of importance. The potential environmental impact to a wide range of aspects or activities was assessed as either low, medium or high.

The highest number of adverse interactions were identified in the operational stage, totalling 147, since there are a greater number of activities in this stage. The stage of preparation and construction of the site with 82 negative interactions, which, although it is a stage of shorter duration, is where the greatest change to the natural environment is concentrated. In the third stage, for closing and post-closing, 12 negative interactions were found.

20.7 COMMUNITY AND SOCIAL COMMITMENTS

The El Tigre Project represents potential economic growth for the region. This will be realized by the implementation of a community development plan for the localities in the area of influence, with the aim of ensuring good terms between the Company and the local citizens. Priority will be placed on access by local citizens to water, food, employment, communications and housing.

While the individual programs of support and programs can be considered optional, the specific needs of each community will be investigated through community diagnoses. Example programs are listed in Table 20.3.

TABLE 20.3
POTENTIAL SUPPORT AND SOCIAL PROGRAMS TO BE IMPLEMENTED
(To Communities and Residences Near the Project)

Aspect	Type of Support	Description, Methodology
Water	Rehabilitation of water wells	Support with the desilting of private wells in the area near the Project
	Rainwater collecting systems	Implement collecting systems for homes
	Water and soil retention works	By providing temporary work to local persons, carry out water and soil retention works at strategic points in the region
Electric Light	Solar cells in communities without electricity	Install solar cells at homes
	Lighting and light installation	Support and/or link communities and ranches with the CFE for the connection of electricity
Housing	Housing rehabilitation	Support with construction materials (cement, sheet metal) for the rehabilitation of homes, and sites of interest to the residents
Dust & Roads	Dust suppression	Road irrigation in the mine's activity area to prevent dust from affecting nearby residences
	Paving of road sections	Paving of sections near a village to reduce the dispersion of dust in the town
	Regulate traffic speed in the Project area	Implement a maximum speed limit in the mine area to avoid accidents and dust issues
Animal Husbandry	Support for fences and livestock guards	Generate a regime of fences and cattle guards in the Project area
	Pasture support in times of drought	Install drinking troughs for livestock in areas close to the Project
Garbage	Trash and collection bins	Install garbage containers in the community, and organize collection and disposal
Festivities and	In-kind support	Support the villagers in kind for their festivities (bags of sweets, decorations, regular food, drinks) depending on the celebration

TABLE 20.3 POTENTIAL SUPPORT AND SOCIAL PROGRAMS TO BE IMPLEMENTED (To Communities and Residences Near the Project)		
Aspect	**Type of Support**	**Description, Methodology**
celebrations	Organize the celebrations	Support events in schools or reactive areas of the localities (Children's Day, Miner's Day, Environment Day, Mother's Day, etc.)
Food Production	Garden support	Implement a backyard garden campaign that includes training and delivery of garden materials

Consultations, negotiations and agreements with local communities are planned and will take place in the near future to ensure the Project will provide a wide range of benefits, will minimally interfere with traditional activities such as agriculture and forestry, and that the mine site will be closed out safely and in an environmentally responsible manner.

21.0 CAPITAL AND OPERATING COSTS

All costs are presented in Q3 2024 US dollars. No provision has been included in the cost estimates to offset future escalation. A contingency of 12% is added to all capital costs ("CAPEX"). No contingency is added to operating costs ("OPEX").

The total initial CAPEX of the El Tigre Project is estimated at $86.8M. Expansion and sustaining capital costs incurred during the nine production years are estimated to total $20.1M and $6.2M, respectively. Total OPEX over the life-of-mine ("LOM") are estimated at $491.0M, averaging $12.19/t processed. The following subsections provide details of these costs.

21.1 CAPITAL COSTS

Total capital cost over the LOM are estimated at $113.1M and are presented in Table 21.1.

TABLE 21.1
CAPITAL COST SUMMARY

Item	Initial ($k)	Expansion ($k)	Sustaining ($k)	Total ($k)
Process Plant Direct Costs	42,851	13,584	1,600	58,034
Process Plant Indirect Costs	8,121	-	-	8,121
Mining Direct Costs	2,660	4,362	3,956	10,978
Surface Mine Pre-stripping	3,362	-	-	3,362
Infrastructure	20,489	-	-	20,489
Subtotal	**77,483**	**17,946**	**5,556**	**100,985**
Contingencies @ 12%	9,298	2,199	622	12,118
Total	**86,780**	**20,145**	**6,178**	**113,103**

21.1.1 Processing Capital Costs

Estimated processing costs for the El Tigre Project reflect a two-phase development with Phase I processing 7,500 tpd of feed during Years 1 through 3 and Phase II expansion in Year 4 processing 15,000 tpd through to Year 9.

Initial process plant capital costs include a three-stage crushing system, ore storage, Phase I heap leach pad and ponds construction, Merrill Crowe recovery facility, and other related costs such as freshwater system, solution pumping system, reagent systems, and metallurgical laboratory. Specific processing components were selected for Phase I to support expansion in Year 4 to minimize capital costs as well as associated construction time. Details are outlined in Section 17 of this Technical Report.

Phase II expansion in Year 4 will increase capacity to 15,000 tpd and includes crushing circuit and leach pad expansions as well as expansion of the Merrill Crowe system.

21.1.1.1 Process Plant Direct Costs

Capital costs include $16.4M for the Phase I crushing circuit based on the PFS process design criteria outlined in Sections 13 and 17 of this Technical Report. Quotes were provided by three Mexican vendors and the circuit includes a feed system, primary crusher, material handling conveyors, secondary and tertiary cone crushers, intermediate ore storage and reclaim, and screening systems to obtain the desired P_{80} product size for stacking.

Capital costs for the Phase I leach pad and pond are estimated at $5.9M and are based on a third-party supplied take-off and design ensuring sufficient capacity for the operation. Vendor quotes were provided for the excavation and install of 80 and 60 mm LLDPE liner while costs for piping and valving were provided by in-house database.

Vendor quotes were also provided for a Phase I Merrill Crowe system and is estimated at $12.7M including the recovery plant, refinery equipment and a dust collection system.

Total Phase I process plant direct capital costs are estimated at $42.9M and are presented in Table 21.2.

TABLE 21.2
PHASE I PROCESS PLANT DIRECT CAPITAL COSTS

Area	Description	Cost ($k)
100	Crushing Circuit	16,402
300	Heap Leach Pad and Ponds	5,859
500	Merrill Crowe	11,169
520	Refinery and Smelting	1,554
650	Water System	4,218
670	PLS Pumping Systems	350
700	Reagents	2,374
800	Laboratory	925
Total Direct Costs – Phase I		**42,851**

Source: *D.E.N.M. (2024)*

Expansion capital costs for Phase II, as presented in Table 21.3, include $4.93M for the crushing circuit, $1.05M for the Merrill Crowe facility, $7.05M for the heap leach pad expansion, and $0.35M for upgrading the pumping system and distribution.

TABLE 21.3 PHASE II PROCESS PLANT DIRECT CAPITAL COSTS		
Area	**Description**	**Cost ($k)**
100	Crushing Circuit	4,927
300	Heap Leach Pad	7,055
500	Merrill Crowe	1,053
670	PLS Pumping Systems	350
Total Direct Costs – Phase II		**13,584**

Source: *D.E.N.M. (2024)*

Sustaining direct capital costs are reflected in the cash flow with allowance for the crusher and leach pad expansion to support the loading and processing increase to 15,000 tpd. Sustaining capital costs are estimated at $1.6M and are comprised of $0.2M/year over the LOM except for Year 4 when the cost is incorporated into expansion capital.

Process Plant Direct Construction Capital Costs

Direct construction capital costs are primarily associated with the crushing circuit, Merrill Crowe plant and leach pad and are based on factoring as well as quoted installation costs on $/m^2. Capital and construction costs for Phases I and II average $29.30/m^2.

Anticipated construction time for Phase I is 18 months.

21.1.1.2 Process Plant Indirect Costs

Factored costs were used in the process EPCM indirect costs. Engineering and procurement used 5.2% and construction and project management used 7.3% for a total of 12.5% of the fixed cost. The factor was applied to equipment and construction costs in Phase I.

As noted in Table 21.4, indirect capital costs also include factored costs applied to freight, capital spares, start-up and commissioning and first fills, and are estimated at $8.1M.

TABLE 21.4 PROCESS PLANT INDIRECT CAPITAL COSTS		
Factor (%)	**Description**	**Cost ($k)**
7	Freight	1,890
1	Field Indirect	236
2	Start-Up and Commissioning	472
3	Capital Spares	709
1	First Fills	236
Sub-Total		**3,543**

TABLE 21.4 PROCESS PLANT INDIRECT CAPITAL COSTS		
Factor (%)	**Description**	**Cost ($k)**
7.2	Engineering and Procurement	2,637
5.3	Construction Management	1,941
Total Process Plant Indirect Costs		**8,121**

Source: *D.E.N.M. (2024)*

21.1.2 Mining Capital Costs

The mining capital cost incurred in Year -2 and -1 are the costs required to prepare the mine site, build the initial haul roads, and mobilize the mining contractor. Expansion capital costs are expenditures in Year 4 to prepare for an increase in production from 7,500 tpd to 15,000 tpd. Sustaining capital costs are other expenditures in production years to maintain access to open pit mining phases and for contractor demobilization.

The mining capital cost is summarized in Table 21.5 without contingency. The initial mining capital cost is estimated at $6.0M, expansion capital cost is estimated at $4.4M, and mine sustaining capital cost is estimated at $4.0 million. Pre-stripping costs are estimated at $3.36M and are based on mining 1.5 Mt of material at an average unit cost of $2.24/t.

TABLE 21.5 MINING CAPITAL COSTS				
Surface Mine Item	**Cost**	**Initial**	**Expansion**	**Sustaining**
	Total ($k)	**Yr -2 to -1 ($k)**	**Yr 4 ($k)**	**Yr 1 to Yr 9 ($k)**
Phase1 Haul Road to Waste Rock Facility	752.0	752.0		
Phase2 Haul Road to Crusher	1,536.0			1,536.0
Phase2 Haul Road to Waste Rock Facility	2.0			2.0
Phase2 New Road to Waste Rock Facility	12.0			12.0
Phase2 Haul Road to Pit Top	948.0			948.0
Phase3 Haul Road to Waste Rock Facility	328.0		328.0	
Phase3 Haul Road to Pit Exit	1,739.0		1,739.0	
Phase3 Inpit Haul Road to Pit Top	1,737.0		1,737.0	
Clear & Grub - Pit Phase I	58.8	58.8		
Clear & Grub - Pit Phase II	39.2			39.2
Clear & Grub - Pit Phase III	95.2		95.2	
Clear & Grub - Main Waste Rock Facility	170.8	170.8		
Clear & Grub - East Waste Rock Facility	39.2			39.2
Clear & Grub - Southeast Waste Rock Facility	28.0		28.0	

TABLE 21.5 MINING CAPITAL COSTS				
Surface Mine Item	**Cost**	**Initial**	**Expansion**	**Sustaining**
	Total ($k)	**Yr -2 to -1 ($k)**	**Yr 4 ($k)**	**Yr 1 to Yr 9 ($k)**
Clear & Grub - Haul Roads	193.2	193.2		
Bench Access Trails	450.0	75.0	75.0	300.0
Explosive & Emulsion Pad	50.0	50.0		
Explosives Security Fence, Cameras	50.0	50.0		
Contractor Mob / Demob	1,580.0	600.0		980.0
Pre-stripping	3,362.0	3,362.0		
Mine Dewater Systems	100.0			100.0
Mine Truck Shop Laydown Area	100.0	100.0		
Survey Equipment	100.0	100.0		
Office Equipment & Software	100.0	100.0		
Communications Equipment	50.0	50.0		
Pickup Trucks	720.0	360.0	360.0	
Total Surface Mine CAPEX	**14,340.4**	**6,021.8**	**4,362.2**	**3,956.4**

Note: *Yr = year.*

21.1.3 Infrastructure Capital Costs

Infrastructure capital costs are estimated at $20.5M as presented in Table 21.6 and include utilities, access roads, site buildings, camp facilities, water and sewage treatments as well as pre-production processing workforce and Owner's costs. Estimates have been provided by Mexican vendors and from Mexican mining operations that are familiar to the Authors.

TABLE 21.6 INFRASTRUCTURE CAPITAL COSTS	
Infrastructure Item	**Cost ($k)**
Gatehouse	15.3
Warehouse	84.6
Main Office Building	90.0
Service Buildings (Dining, Clinic, Laundry, Food Storage)	196.1
300-Person Camp Facilities (Procsa)	1,213.1
Upgrade Main Access Road	720.0
Site Roads - Upgrade and New	1,000.0
Contractor Laydown Prep	50.0
Reagent Storage	100.0
Generator Backup	400.0

TABLE 21.6 INFRASTRUCTURE CAPITAL COSTS	
Infrastructure Item	**Cost ($k)**
Fire Protection	100.0
Powerline and Right of Way (CFE)	2,307.8
Substation and Power Distribution (Remogal)	1,002.8
Communications	100.0
Potable Water Treatment (Hidrotec)	5.8
Sewage Treatment (Hidrotec)	51.4
Access Bridge	300.0
Pre-production Process Plant Workforce	3,283.4
Pre-production Owner's Costs	9,469.1
Total	**20,489.4**

Notes: *Procsa = PROCSA Ingenieria Electrica Y Sistemas de Aire S.A. de C.V.*
CFE = Comision Federal de Electricidad (Mexican Federal Electrical Commission).
Remogal = REMOGAL Control & Automatizacion.
Hidrotec = HIDROTEC Solucion Integral al Problema del Agua.

The estimate of $9.5M pre-production Owner's costs are presented in Table 21.7 and include general and administrative expenses, concession payments and employee travel.

TABLE 21.7 OWNER'S COSTS	
Item	**Cost ($k)**
G&A and Construction Supervision Labour	1,142.7
G&A Expense Items – Onsite Including Camp Services	5,223.8
Satellite Offices & Off-Site Warehousing Including Concession Payments	2,922.6
Employee Travel - Residence to Site	180.0
Total	**9,469.1**

21.2 OPERATING COSTS

The operating costs estimate includes the cost of mining, processing, and General and Administration ("G&A") services. The LOM average operating costs for the Project during production years are summarized in Table 21.8.

TABLE 21.8 OPERATING COST SUMMARY		
Area	**Unit Cost ($)**	**LOM Cost ($k)**
Surface Mining ($/t mined)	2.24	
Heap Leach and Processing Phase I ($/t processed)	5.79	
Heap Leach and Processing Phase II ($/t processed)	4.74	
OPEX ($/t processed)		
Heap Leach and Processing	4.95	199,604
Surface Mining	5.96	240,120
G&A	1.28	51,300
Total	**12.19**	**491,024**

21.2.1 Process Plant

Table 21.9 and Table 21.10 summarize the estimated process plant operating costs for Phase I at 7,500 tpd and Phase II at 15,000 tpd. The costs do not include mining support or general personnel and are based on the following:

- FX Rate: 1 US$, 20 $MXN;
- CFE Powerline: US$0.113/kWh;
- Bulk Diesel: US$1.11/L;
- Mexico Labour Burden Rate: 37%;
- Phase I Labour: 114 Personnel; and
- Phase II Labour: 130 Personnel.

TABLE 21.9 PROCESSING COST SUMMARY – PHASE I			
Item	**Total $M/Year**	**$/t Processed**	**% of Total**
Labour	2.9	1.05	18
Power and Fuel	4.2	1.56	27
Maintenance and Operating Consumables	7.6	2.79	48
Support Equipment	1.1	0.40	7
Total Phase I	**15.8**	**5.79**	**100**

Source: *D.E.N.M. (2024)*

TABLE 21.10 PROCESSING COST SUMMARY – PHASE II			
Process Operating Cost	**$M/ Year**	**$/t Processed**	**% of Total**
Labour	3.2	0.59	12
Power and Fuel	5.6	1.03	22
Maintenance and Operating Consumables	15.3	2.80	59
Support Equipment	1.8	0.33	7
Total Phase II	**25.9**	**4.74**	**100**

Source: *D.E.N.M. (2024)*

21.2.1.1 Process Plant Personnel

Table 21.11 and Table 21.12 summarize the process plant personnel during Phases I and II and include positions, base salary and burden rate. Total costs are expressed in both US dollars and Mexican Pesos.

TABLE 21.11
PROCESSING PLANT PERSONNEL – PHASE I

Position	Personnel No.	Base Pay $MXN	Overtime $MXN	Burden Rate 37%Base Pay	$MXN Total Per Employee	$US Total FX 20.0	$US Total Cost	$US/t
Supervision	**9**							
Process Clerk	1	292,440		108,203	400,643	20,032		
Process Superintendent	1	960,000		355,200	1,315,200	65,760		
Supervisor/Shift Foreman	3	516,000		190,920	706,920	106,038		
Process Maintenance General Superintendent	1	960,000		355,200	1,315,200	65,760		
Process Maintenance Shift Supervisor	3	380,000		140,600	520,600	78,090	335,680	0.12
Operations	**51**							
Crushing & Reclaim								
Primary Crusher Operator	3	181,080	109,536	67,000	357,616	53,642		
Secondary Crusher Operator	3	181,080	109,536	67,000	357,616	53,642		
Tertiary Crusher Operator	3	181,080	109,536	67,000	357,616	53,642		
Crusher Feed Loader Operator	3	181,080	109,536	67,000	357,616	53,642		
Crusher Helper	6	162,360	97,764	60,073	320,197	96,059		
Heap Leach								
Heap Leach Supervisor	2	516,000		190,920	706,920	70,692		
Heap Leach Operator	3	181,080	109,536	67,000	357,616	53,642		
Reagent Operator	3	181,080	109,536	67,000	357,616	53,642		
Stacking Operator	3	181,080	109,536	67,000	357,616	53,642		
Heap Dozer Operator	3	181,080	109,536	67,000	357,616	53,642		
Piping Crew - Heap Leach	6	181,080	109,536	67,000	357,616	107,285		
Recovery Plant								
Recovery Plant Supervisor	3	516,000		190,920	706,920	106,038		

TABLE 21.11
PROCESSING PLANT PERSONNEL – PHASE I

Position	Personnel No.	Base Pay $MXN	Overtime $MXN	Burden Rate 37%Base Pay	$MXN Total Per Employee	$US Total FX 20.0	$US Total Cost	$US/t
Recovery Plant Operator	3	181,080	109,536	67,000	357,616	53,642		
Recovery Plant Helper	3	162,360	97,764	60,073	320,197	48,030		
Refining Operator	2	181,080	109,536	67,000	357,616	35,762		
Refining Helper	2	162,360	97,764	60,073	320,197	32,020	978,666	0.36
Maintenance	**32**							
Mechanic I	6	216,000	109,536	79,920	405,456	121,637		
Mechanic II	6	181,080	109,536	67,000	357,616	107,285		
Mechanic Helper	6	162,360	97,764	60,073	320,197	96,059		
Planner	1	424,440		157,043	581,483	29,074		
Electrician	6	216,000	108,060	79,920	403,980	121,194		
Electrician Helper	3	162,360	97,764	60,073	320,197	48,030		
Instrumentation Technician	3	216,000	108,060	79,920	403,980	60,597		
Maintenance Clerk	1	292,560		108,247	400,807	20,040	603,916	0.22
Laboratory	**22**							
Metallurgist	1	520,560		192,607	713,167	35,658		
Metallurgical Technician	2	217,800	108,060	80,586	406,446	40,645		
Chief Assayer/Lab	1	520,560		192,607	713,167	35,658		
Assayers	6	181,080	109,536	67,000	357,616	107,285		
Lab Technician	6	181,080	109,536	67,000	357,616	107,285		
Sample Preparation Laborer	6	162,360	97,764	60,073	320,197	96,059	422,590	0.15
Total Personnel	**114**							
Camp Facilities							520,000	0.38
Total Personnel Cost					**16,628,691**	**2,340,851**	**2,860,851**	**1.05**

Source: *D.E.N.M. (2024)*
Note: *$MXN = Mexican peso, FX = foreign exchange rate, US$ = United States dollars.*

TABLE 21.12
PROCESSING PLANT PERSONNEL – PHASE II

Position	Personnel No.	Base Pay $MXN	Overtime $MXN	Burden Rate 37%Base Pay	$MXN Total Per Employee	$US Total FX 20.0	$US Total Cost	$US/t
Supervision	**9**							
Process Clerk	1	292,440		108,203	400,643	20,032		
Process Superintendent	1	960,000		355,200	1,315,200	65,760		
Supervisor/Shift Foreman	3	516,000		190,920	706,920	106,038		
Process Maintenance General Superintendent	1	960,000		355,200	1,315,200	65,760		
Process Maintenance Shift Supervisor	3	380,000		140,600	520,600	78,090	335,680	0.06
Operations	**51**							
Crushing & Reclaim								
Primary Crusher Operator	3	181,080	109,536	67,000	357,616	53,642		
Secondary Crusher Operator	3	181,080	109,536	67,000	357,616	53,642		
Tertiary Crusher Operator	6	181,080	109,536	67,000	357,616	107,285		
Crusher Feed Loader Operator	3	181,080	109,536	67,000	357,616	53,642		
Crusher Helper	6	162,360	97,764	60,073	320,197	96,059		
Heap Leach								
Heap Leach Supervisor	3	516,000		190,920	706,920	106,038		
Heap Leach Operator	6	181,080	109,536	67,000	357,616	107,285		
Reagent Operator	3	181,080	109,536	67,000	357,616	53,642		
Stacking Operator	3	181,080	109,536	67,000	357,616	53,642		
Heap Dozer Operator	3	181,080	109,536	67,000	357,616	53,642		
Piping Crew - Heap Leach	9	181,080	109,536	67,000	357,616	160,927		
Recovery Plant								
Recovery Plant Supervisor	3	516,000		190,920	706,920	106,038		

TABLE 21.12
PROCESSING PLANT PERSONNEL – PHASE II

Position	Personnel No.	Base Pay $MXN	Overtime $MXN	Burden Rate 37%Base Pay	$MXN Total Per Employee	$US Total FX 20.0	$US Total Cost	$US/t
Recovery Plant Operator	3	181,080	109,536	67,000	357,616	53,642		
Recovery Plant Helper	3	162,360	97,764	60,073	320,197	48,030		
Refining Operator	3	181,080	109,536	67,000	357,616	53,642		
Refining Helper	3	162,360	97,764	60,073	320,197	48,030	1,208,829	0.22
Maintenance	**32**							
Mechanic I	6	216,000	109,536	79,920	405,456	121,637		
Mechanic II	6	181,080	109,536	67,000	357,616	107,285		
Mechanic Helper	6	162,360	97,764	60,073	320,197	96,059		
Planner	1	424,440		157,043	581,483	29,074		
Electrician	6	216,000	108,060	79,920	403,980	121,194		
Electrician Helper	3	162,360	97,764	60,073	320,197	48,030		
Instrumentation Technician	3	216,000	108,060	79,920	403,980	60,597		
Maintenance Clerk	1	292,560		108,247	400,807	20,040	603,916	0.11
Laboratory	**26**							
Metallurgist	1	520,560		192,607	713,167	35,658		
Metallurgical Technician	3	217,800	108,060	80,586	406,446	60,967		
Chief Assayer/Lab	1	520,560		192,607	713,167	35,658		
Assayers	6	181,080	109,536	67,000	357,616	107,285		
Lab Technician	6	181,080	109,536	67,000	357,616	107,285		
Sample Preparation Laborer	9	162,360	97,764	60,073	320,197	144,089	490,942	0.09
Total Personnel	**114**							
Camp Facilities							570,000	0.10
Total Personnel Cost					**16,628,691**	**2,639,367**	**3,209,367**	**0.59**

Source: *D.E.N.M. (2024)*
Note: *$MXN = Mexican peso, FX = foreign exchange rate, US$ = United States dollars.*

21.2.1.2 Processing Plant Power and Consumables

Table 21.13 and Table 21.14 summarize estimated costs for power and consumables for the processing plant during Phases I and II.

TABLE 21.13
PROCESSING PLANT POWER AND CONSUMABLES – PHASE I

Item	kWh/Year	$/kWh	kWh/t	Annual Cost $M	$/t
Power	36,693,973	0.113	13.40	4.146	1.51
Fuel and Plant Consumables				0.113	0.04

Source: *D.E.N.M. (2024)*

TABLE 21.14
PROCESSING PLANT POWER AND CONSUMABLES – PHASE II

Item	kWh/Year	$/kWh	kWh/t	Annual Cost $M	$/t
Power	48,294,400	0.113	8.82	5.457	1.00
Fuel and Plant Consumables				0.168	0.03

Source: *D.E.N.M. (2024)*

21.2.1.3 Process Plant Crushing and Process Consumables

Table 21.15 and Table 21.16 summarize estimated costs for crushing and process consumables including reagents for the processing plant during Phases I and II.

TABLE 21.15
PROCESSING PLANT CRUSHING AND PROCESS CONSUMABLES – PHASE I

Crushing and Process Consumables			Material $/t	Annual $M	Total $M	Annual $/t
Crusher			203,626	0.362		
Merrill Crowe Plant			32,831	0.113		
Refinery			1,551,651	1.582		
Total Crushing and Process Consumables					**2.057**	**0.75**
Reagents	**kg/t**	**Tonne /Year**	**$/t**	**Annual $M**	**Total $M**	**Annual $/t**
Lime (CaO)*	1.50	4,106	220	0.903		
Cyanide*	0.25	684	2,600	1.779		

TABLE 21.15 PROCESSING PLANT CRUSHING AND PROCESS CONSUMABLES – PHASE I						
Precoat **	0.25		1,400	0.958		
Zinc Dust**	0.10		6,260	1.714		
Antiscalent**	0.02		4,260	0.233		
Total Reagents					**5.588**	**2.04**

Source: *D.E.N.M. (2024)*
** Cyanide consumption discounted from column testing as noted. No discount for Lime.*
*** Estimated consumptions*

TABLE 21.16 PROCESSING PLANT CRUSHING AND PROCESS CONSUMABLES – PHASE II						
Crushing and Process Consumables			**Material $/t**	**Annual $M**	**Total $M**	**Annual $/t**
Crusher			1,801,734	1.989		
Merrill Crowe Plant			28,549	0.133		
Refinery			1,999,262	2.026		
Total Crushing and Process Consumables					**4.148**	**0.76**
Reagents	**kg/t**	**Tonnes /Year**	**$/t**	**Annual $M**	**Total $M**	**Annual $/t**
Lime (CaO)*	1.50	8,213	220	1.807		
Cyanide*	0.25	1,369	2,600	3.559		
Precoat **	0.25		1,400	1.916		
Zinc Dust**	0.10		6,260	3.427		
Antiscalent**	0.02		4,260	0.466		
Total Reagents					**11.176**	**2.04**

Source: *D.E.N.M. (2024)*
**Cyanide consumption discounted from column testing. No discount for Lime.*
***Estimated consumptions.*

21.2.2 Surface Mining

The El Tigre surface mine operating cost consists of a contract mining and blasting cost and Owner's costs for supervision and technical services. The average LOM unit mining operating cost is estimated at $2.24/t, as shown in Table 21.17. This is the average over the mining production years 1 to 9 and does not include costs from the pre-stripping period (Year -1).

TABLE 21.17
SURFACE MINE OPERATING COST SUMMARY

Operation	Total	Year								
		1	2	3	4	5	6	7	8	9
Total Mined (kt)	107,114	7,500	7,500	9,000	18,000	18,000	18,000	14,000	9,000	6,114
Ore Mined (kt)	40,238	3,001	2,947	3,029	5,472	5,472	5,472	5,472	5,472	3,901
Annual Opex ($k)										
Contract Mining	231,349	15,648	13,519	18,970	35,724	33,088	45,208	38,202	18,502	12,487
Stockpile Reclaim	1,207	-	-	-	-	-	-	-	-	1,207
Owner Team	4,334	510	510	510	510	510	510	510	510	255
Other Costs	3,230	380	380	380	380	380	380	380	380	190
Total	**240,120**	**16,538**	**14,409**	**19,860**	**36,614**	**33,978**	**46,098**	**39,092**	**19,391**	**14,139**
Unit Costs										
$/t Mined	2.24	2.21	1.92	2.21	2.03	1.89	2.56	2.79	2.15	2.31
$/t Ore	5.96	5.51	4.89	6.56	6.69	6.21	8.42	7.14	3.54	3.62

21.2.2.1 Contract Mining

Several Mexican mining contractors were contacted to provide budgetary quotations. These included all surface mining costs, including equipment, personnel, diesel fuel, and maintenance. The lowest quotation was selected and was based on a mining unit rate of $1.44/t, with an overhaul haulage cost of $0.21/t/km. That unit rate did not include explosives or blasting services, which were an extra cost. Table 21.18 provides a breakdown of the mining contractor unit rates.

The explosive supply cost, also shown in Table 21.18 includes three components. The explosive contractor will supply the ammonium nitrate ("AN"), the Owner will provide the diesel fuel to produce the ANFO explosives, and the contractor will provide blasting services at a fixed cost of $49,400 per month. This fixed cost includes personnel and equipment necessary to mix blasting agents, charge holes, and blast. The average powder factor ("PF") estimated by the blasting contractor is 0.21 kg/tonne for both ore and waste.

Mining and blasting costs are both included in the "Contract Mining" cost shown above in Table 21.17. A mining cost model applies the haulage rate to the varying haul distances appropriate for each year.

TABLE 21.18
CONTRACTOR UNIT RATES

Contractor Activity	Unit Cost $/t
Drilling	0.35
Loading	0.30
Hauling 1 km	0.57
Services	0.22
Total	**1.44**
Supplemental Items	**Unit Cost**
Overhaul $/km	0.21
AN supply ($/t)	0.23
Fuel (0.21 PF) ($/t)	0.015
Blasting Service $/Month	$49,400

21.2.2.2 Supervision and Technical Services

The mine supervision and technical services includes salaries for the 16-person Owner's team (Table 21.19), and annual costs for office equipment, vehicle maintenance, outside consultants, and other mine management costs. This annual fixed Owner's cost is estimated to be $890,000 per year.

TABLE 21.19 OWNER'S OPERATING PERSONNEL	
Owners Mining Team	**Number**
Mine Superintendent	1
Mine Superintendent Assistant	1
Mine Supervisor	1
Drill and Blast Supervisor	1
Tech Services Superintendent	1
Chief Mine Engineer	1
Surveyor	2
Surveyor Assistant	2
Geology Supervisor	2
Samplers	2
General Services Assistant	2
Total	**16**

21.2.3 General and Administration

General and Administration ("G&A") costs are estimated at $5.7M annually, as summarized in Table 21.20.

TABLE 21.20 GENERAL AND ADMINISTRATION COSTS		
Item	**Number**	**Annual Cost ($k)**
General Manager	1	187.4
General Manager Assistant	1	14.0
Safety Superintendent	1	65.8
Safety Supervisor	3	74.0
Environment Coordinator	1	41.1
Security Coordinator	1	32.9
Purchasing Chief	1	44.5
Buyer	1	24.7
Warehouse Chief	1	37.0
Warehouse Helper	3	41.9
Controller	1	86.3
Accountant Chief	1	38.8
Accountant	1	37.0
Human Resources Superintendent	1	65.8
Human Resources Coordinator	1	28.8
Community Relations Superintendent	1	58.9

TABLE 21.20 GENERAL AND ADMINISTRATION COSTS		
Item	**Number**	**Annual Cost ($k)**
Community Relations Coordinator	1	28.8
Driver	1	17.9
Regional Office and Concessions	Lump sum	1,461.3
Insurance	Lump sum	250.0
Security Services	Lump sum	117.0
Communications	Lump sum	87.3
Community Assistance and Housing	Lump sum	117.0
Office and Outside Services	Lump sum	523.0
Camp Costs	Lump sum	2,219.0
Total[1]		**5,700.0**

[1] *Totals may not sum due to rounding.*

21.2.4 Mining Tax

The Project is subject to a 1.0% NSR mining duty tax payable to the Mexican government. Total costs associated with this NSR tax over the LOM are estimated at $16.2M.

21.3 CLOSURE COSTS

Costs to be incurred after the LOM plan in this PFS were estimated by CIMA to be $6.2M to close and rehabilitate the Project site, as presented in Table 21.21.

TABLE 21.21 CLOSURE COSTS	
Activity	**Closure Cost ($k)**
Road Rehabilitation	649.6
Small Borrows and Excavations	162.5
Waste Rock Storage Facilities	2,076.5
Surface Mine Rehabilitation	1,854.8
Process Plant, Shops, Warehouse, Buildings	228.8
Heap Leach Pads	1,211.5
Total	**6,183.7**

21.4 CASH COSTS AND ALL-IN SUSTAINING COSTS

Cash costs over the LOM, including Mexican mining taxes, are estimated to average $11.70/oz AgEq or $997/oz AuEq. All-In Sustaining Costs ("AISC") over the LOM are estimated to average $14.53/oz AgEq or $1,238/oz AuEq and include closure costs.

22.0 ECONOMIC ANALYSIS

22.1 SUMMARY

Cautionary Statement - The results of the economic analyses discussed in this Technical Report section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

Information that is forward-looking includes:

- Mining dilution and mining recovery;
- Mineral Reserve Estimates;
- Commodity prices and exchange rates;
- Mine production plan;
- Process plant recovery rates;
- Expansion and sustaining capital costs, and operating costs;
- Closure costs and closure requirements; and
- Environmental, permitting and social risks.

Additional risks to the forward-looking information include:

- Changes to costs of production from what is assumed;
- Unrecognized environmental risks;
- Adverse weather risks and earthquakes;
- Unanticipated reclamation expenses;
- Unexpected variations in quantity of mineralized material, grade or recovery rates;
- Geotechnical or hydrogeological considerations during mining being different from what was assumed;
- Failure of mining methods to continue to operate as anticipated;
- Failure of process plant, equipment or processes to operate as anticipated;
- Changes to assumptions as to the availability of electrical power and its rates used in the operating cost estimates and financial analysis;
- Ability to maintain the social licence to operate;
- Accidents, labour disputes and other risks of the mining industry; and
- Changes to tax rates.

A discounted cash flow model was prepared using the production schedule described in Section 16 and the cost estimates described in Section 21 of this Technical Report. The PFS cash flow model was developed on a pre-tax and after-tax basis. The cash flow model is assumed to commence from the time a production decision is made and does not include time or costs for additional studies.

The El Tigre Project economic evaluation conclusions are summarized in Table 22.1. At base case metal prices of US$3,200/oz Au and US$38/oz Ag the Project has an estimated US$455.6M after-tax net present value ("NPV") at a 5% discount rate ("NPV5%"), and an after-tax internal rate of

return ("IRR") of 66%. The payback period is estimated to be 1.4 years from the start of production.

TABLE 22.1 ECONOMIC EVALUATION SUMMARY		
Item	**Pre-Tax**	**After-Tax**
NPV0% ($M)	1,000.8	625.3
NPV5% ($M)	737.2	**455.6**
NPV7% ($M)	656.5	403.6
IRR (%)	87.7	**65.7**
Payback period (years)	1.2	**1.4**

22.2 BASIC ASSUMPTIONS

A discounted cash flow analysis of the El Tigre Project was prepared based on technical and cost inputs developed by the Authors.

The discounted cash flow analysis was performed on a stand-alone project basis with annual cash flows discounted. The financial evaluation uses a discount rate of 5%, discounting back to the commencement of construction (Year -2) of the Project.

All currency values are expressed in US dollars unless otherwise noted.

Metal Price Assumptions

All metal prices remain constant throughout the LOM of the Project.

Ag: $US38/oz. The sensitivity of the Project return to variations in the actual silver price received was also examined.

Au: $US3,200/oz. The sensitivity of the Project return to variations in the actual gold price received was also examined.

Metallurgical Recoveries

The El Tigre Project's process plant via heap leach metallurgical recovery discounted assumptions are:

Oxide	Ag = 45%, Au = 83%
Sulphide	Ag = 40%, Au = 56%

Capital Costs

Total capital costs during the LOM are estimated to be $113.1M as outlined in the Capital and Operating Cost Section 21. The initial capital costs are incurred over an 18-month construction period and are estimated to be $86.8M. Expansion capital costs for increasing production from 7,500 tpd to 15,000 tpd are estimated at $20.1M, and sustaining costs over the LOM are estimated at $6.2M.

Salvage value was estimated at 15% of the $23.0M process plant mechanical equipment installed over the LOM, equivalent to $3.4M.

Previous Expenses Provision

An amount of $9.16M was considered as a prior expense pool and these monies were deducted from income in production Year 1 when determining the taxable income.

Income Tax Rate

The Mexican income tax is levied at a rate of 30% on the net taxable income. A special duty tax of 8.5% on the net taxable income is also included.

Additional Mining Tax Rate

A Mexican government 1.0% gross revenue mining tax was applied in the cash flow economics and is deductible when determining taxable income.

22.3 CASH FLOW SUMMARY

Based on the assumed metal prices the Project has an after-tax IRR of 66% and a 1.4 year payback of initial pre-production capital costs. The Project is estimated to realize an after-tax NPV of US$455.6M at a discount rate of 5%.

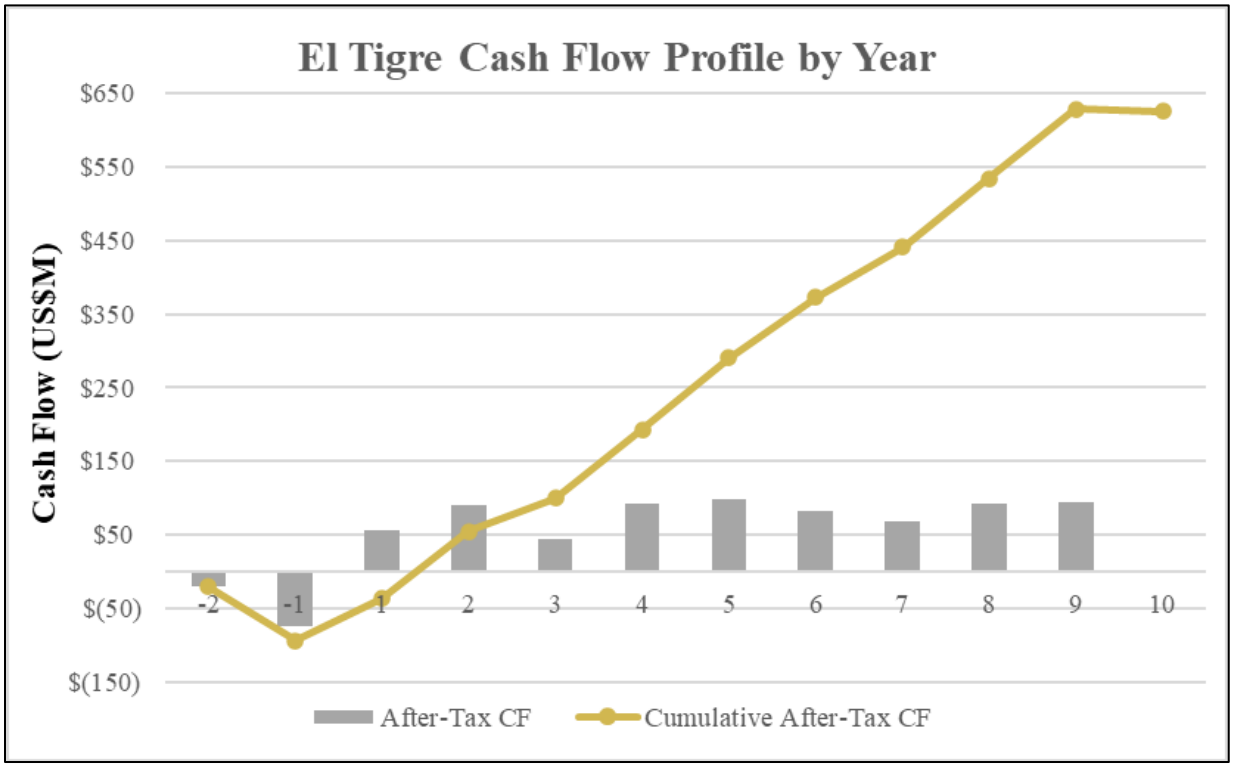
The estimated annual production and LOM cash flows for the El Tigre Project are summarized in Table 22.2 and annual cash flows are presented in Figure 22.1. An annual summary of the cash flow model is provided in Table 22.5.

TABLE 22.2
PROJECT CASH FLOW SUMMARY

Assumption / Result	Unit	Value	Assumption / Result	Unit	Value
Ore Mined	kt	40,292	Net Revenue	US$M	1,623.9
Waste Rock Mined	kt	68,321	Initial Capital	US$M	86.8
Strip Ratio	w:o	1.7:1	Expansion Capital	US$M	20.1
Silver Grade	g/t	14.9	Sustaining Capital	US$M	6.2

TABLE 22.2 PROJECT CASH FLOW SUMMARY					
Assumption / Result	**Unit**	**Value**	**Assumption / Result**	**Unit**	**Value**
Gold Grade	g/t	0.40	Mining Costs	$/t Material	2.24
Silver Recovery (Oxide/Sulphide)	%	45 / 40	Processing Costs (Phase I and Phase II)	$/t Ore	5.79/4.74
Gold Recovery (Oxide/Sulphide)	%	83 / 56	G&A Costs	$/t Ore	1.27
Silver Price	US$/oz	38.00	Operating Cash Cost	US$/oz AgEq	11.7
Gold Price	US$/oz	3,200	All-in Sustaining Cost	US$/oz AgEq	14.5
Payable Silver Metal	Moz	8.57	After-Tax NPV (5% discount)	US$M	455.6
Payable Gold Metal	koz	408	Pre-Tax NPV (5% discount)	US$M	737.2
Payable AgEq	Moz	42.9	After-Tax IRR	%	65.7
Production and Reclamation	years	10	Pre-Tax IRR	%	87.7
Average Material Mined	tpd	30,000	After-Tax Payback Period	years	1.4

FIGURE 22.1 ANNUAL CASH FLOW PROFILE



22.4 SENSITIVITIES

The El Tigre Project economics were examined with a sensitivity analysis for several key variables. The results of the sensitivity analyses on the after-tax NPV with a 5% discount rate are shown in Tables 22.3 and 22.4.

TABLE 22.3
SILVER AND GOLD PRICE SENSITIVITY NPV, IRR AND PAYBACK

Sensitivity	-20%	-10%	Base Case	+10%	+20%
Silver Price (US$/oz)	30	34	**38**	42	46
Gold Price (US$/oz)	2,560	2,880	**3,200**	3,520	3,840
After-Tax NPV (5%) (US$M)	303.9	379.8	**455.6**	531.5	607.4
After-Tax IRR (%)	49.9	58.1	**65.7**	73.0	79.9
After-Tax Payback (years)	1.7	1.5	**1.4**	1.3	1.2

TABLE 22.4
CAPITAL AND OPERATING COST SENSITIVITY OF NPV AND IRR

Sensitivity	-20%	-10%	Base Case	+10%	+20%
Operating Costs – NPV_5 (US$M)	502.0	478.8	**455.6**	432.5	409.3
Operating Costs – IRR (%)	70.7	68.2	**65.7**	63.2	60.6
Capital Costs – NPV_5 (US$M)	470.2	462.9	**455.6**	448.3	441.0
Capital Costs – IRR (%)	77.7	71.2	**65.7**	61.1	57.0

The after-tax base case NPV's and IRR's are most sensitive to metal prices followed by operating costs and capital costs.

TABLE 22.5
CASH FLOW MODEL SUMMARY

Item	Units	Totals	Year											
			-2	-1	1	2	3	4	5	6	7	8	9	10
Ore Processed	**kt**	**40,292**	**-**	**-**	**2,736**	**2,736**	**2,736**	**5,472**	**5,472**	**5,472**	**5,472**	**5,472**	**4,724**	**-**
Grade Au	g/t	**0.40**	-	-	0.56	0.62	0.52	0.41	0.39	0.37	0.37	0.33	0.27	-
Grade Ag	g/t	**14.9**	-	-	14.3	15.0	11.7	10.5	11.9	13.4	8.4	22.5	26.1	-
Recovered Au	kozs	**408**	-	-	30	47	34	60	58	53	47	45	33	-
Recovered Ag	kozs	**8,567**	-	-	447	573	480	755	922	1,030	733	1,520	2,105	-
Revenue	**$M**	**1,623.9**	**-**	**-**	**114.0**	**172.6**	**127.8**	**218.9**	**218.6**	**207.8**	**178.7**	**200.6**	**184.9**	**-**
(-) Operating Cost	$M	**(491.0)**	-	-	(38.1)	(36.0)	(41.4)	(68.3)	(65.6)	(77.7)	(70.7)	(51.0)	(42.2)	-
(-) Mining Tax	$M	**(16.2)**	-	-	(1.1)	(1.7)	(1.3)	(2.2)	(2.2)	(2.1)	(1.8)	(2.0)	(1.8)	-
(-) Capital Spending	$M	**(113.1)**	(19.9)	(66.9)	(3.2)	(0.3)	(15.4)	(5.1)	(0.3)	(0.2)	(0.2)	(0.2)	(1.3)	-
(+) Salvage Value	$M	**3.5**	-	-	-	-	-	-	-	-	-	-	-	3.5
(-) Closure	$M	**(6.2)**	-	-	-	-	-	-	-	-	-	-	-	(6.2)
Pre-Tax Cash Flow	**$M**	**1,000.8**	**(19.9)**	**(73.2)**	**71.6**	**134.6**	**69.7**	**143.4**	**150.5**	**127.8**	**106.0**	**147.3**	**145.8**	**(2.7)**
(-) Income Tax	$M	**(375.5)**	-	-	(14.4)	(43.5)	(24.9)	(50.4)	(52.9)	(45.3)	(37.8)	(54.2)	(51.9)	-
After-Tax Cash Flow	**$M**	**625.3**	**(19.9)**	**(73.2)**	**57.2**	**91.1**	**44.8**	**92.9**	**97.6**	**82.5**	**68.1**	**93.1**	**93.9**	**(2.7)**
Cumul After-Tax Cash Flow	$M	**-**	(19.9)	(93.1)	(36.0)	55.1	99.9	192.8	290.4	372.9	441.0	534.1	628.0	625.3
Disc After-Tax Cash Flow (5%)	**$M**	**455.6**	**(19.9)**	**(71.5)**	**53.1**	**80.6**	**37.8**	**74.6**	**74.6**	**60.0**	**47.3**	**61.5**	**59.1**	**(1.6)**
Disc Cumul After-Tax Cash Flow	$M	**-**	(19.9)	(91.4)	(38.2)	42.4	80.2	154.8	229.4	289.4	336.7	398.2	457.3	455.6

23.0 ADJACENT PROPERTIES

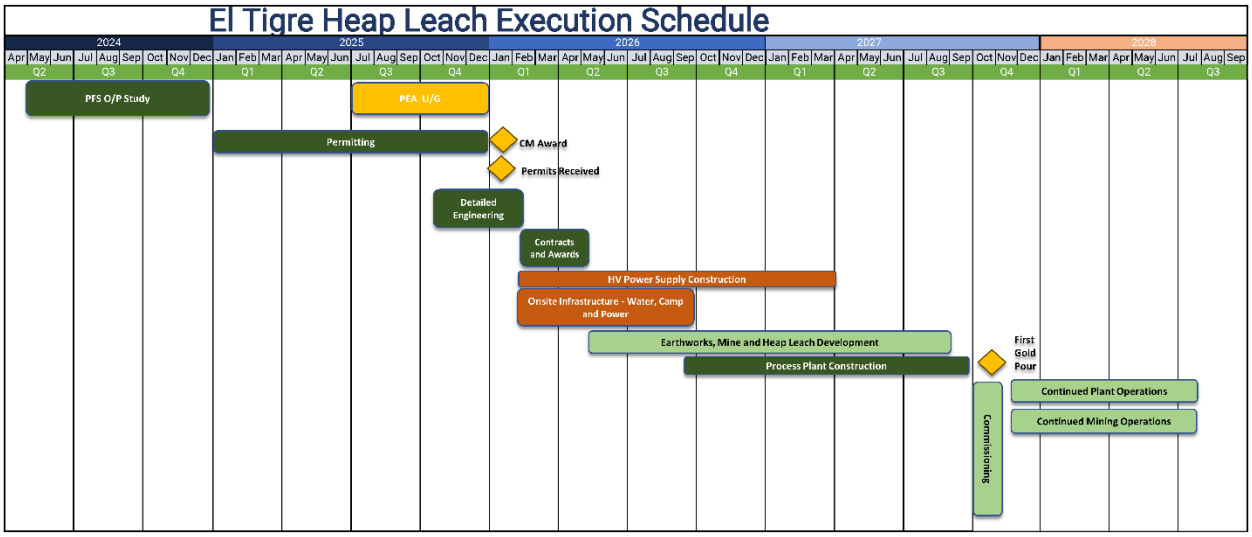
Advanced exploration or operating properties are not known to exist immediately adjacent, or contiguous to, the El Tigre Property that have relevance to this Technical Report.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT EXECUTION SCHEDULE

A simplified Project Execution Schedule is presented in Figure 24.1. On November 7, 2025, the Company announced it has secured all required approvals and permits from the Mexican Federal Environmental Department ("SEMARNAT") to construct the El Tigre Stockwork Silver-Gold Project in Sonora, Mexico. It is anticipated that an 18-month period for FEED, procurement, and construction will be required to construct the mine with all necessary infrastructure to produce the first doré bar and commence production. Several additional approvals/permits for water rights and to allow mining are pending.

FIGURE 24.1 PROJECT EXECUTION SCHEDULE



24.2 PROJECT RISKS AND OPPORTUNITIES

Risks and opportunities have been identified for the Project. The anticipated impact on the Project is listed in brackets after each item, using low-medium-high categories.

24.2.1 Risks

24.2.1.1 Mineral Resource Estimate

- Future metal prices could cause a revision of the Mineral Resource Estimate. (low)

24.2.1.2 Surface Mining

- More detailed preparation of a contractor mining quotation and procurement package could result in higher (or lower) unit costs. The lowest quotation was selected as the basis of the mining costs. (low to medium)

- The veins to be mined are continuous along strike and are relatively wide. The veins exhibit regular geometries (little deformation) as they are young volcanic host rocks. Dilution is estimated to be low at approximately 3%. (low)

- Mining around historical underground workings may create additional productivity, cost and safety impacts. (low)

- More detailed slope geotechnical studies could impact favorably or negatively on the pit designs. Flattening of slopes could have a significant impact on the waste quantity. (low)

- Hydrogeology knowledge is preliminary. Water re-charge rates are likely low, however, are currently unknown. The existing underground workings will tend to drain water away from the surface workings. (low)

24.2.1.3 Process Plant

- Metallurgical recoveries. Existing assumptions (oxide, transition and sulphide zones) could change and heap leach alternatives could lead to reduced metal production, higher OPEX, and circuit changes. Further column testwork optimization and delineation should be revisited and completed in more areas of the starter surface mine and future phases. (medium)

- Surface water inflow to pads and ponds will need to be managed to prevent overflow. (low)

- Presently the main water source to serve the overall site is pumping from two wells. Water right permits are currently taking longer than usual to obtain. (low)

- Suitable process plant site location. The original plant location was planned to be located near the existing impounded tailings Mineral Resource. Due to pumping constraints to this location, the process recovery plant has been situated near the heap leach pond(s) location for ease of operation and to minimize costs. (low)

24.2.1.4 Financial Aspects

- Lower metal prices would decrease the Project economics. However, sensitivity analysis indicates that a 20% decrease in metal prices would still result in a financially attractive Project. (low)

24.2.2 Opportunities

24.2.2.1 Mineral Resource Estimate

- The El Tigre Deposit remains open along strike and down dip. There is an opportunity to extend the veins with additional drilling. (medium)

- Confidence in the Mineral Resource is high since it is based mainly on Measured and Indicated Mineral Resources. There are 4.3 Mt of Inferred Mineral Resources contained in the surface mine design. There is a reasonable expectation that the Inferred Mineral Resource can be upgraded to a Measured or Indicated Mineral Resource with further definition drilling. (medium)

24.2.2.2 Surface Mining

- There is an opportunity to increase the size of the surface mine design, especially using higher metal prices than those used for pit optimizations. (low)

- Further field reconnaissance may identify additional waste rock storage sites close to the surface mine, resulting in improved operating costs via shortened haul distances. (low)

24.2.2.3 Process Plant

- Since the operation is heap leach, there is no tailings storage area to manage and no environmental or risk considerations associated with tailings facilities. (medium)

- Existing testwork assumptions and process flow alternatives could change with metallurgical optimization that would lead to higher metal production and lower OPEX. This will have a definite positive effect to Project overall cash flow. Testwork optimization should be completed in all areas. (low)

- Refurbished equipment available on the market could be inserted into specific areas (e.g. crushers). Capital cost reduction and a decrease in the Project construction timeframe could result. (low)

24.2.2.4 Financial Aspects

- A 40% improvement in the base case metal prices, closer to current spot prices, would provide a $303M increase in After-Tax NPV (using a discount rate of 5%) to $759M with an IRR of 93%. (high)

24.3 PRELIMINARY ECONOMIC ASSESSMENT

24.3.1 Introduction

The 2025 PEA that follows is an alternative development option done at the conceptual level based on Mineral Resources that are considered amenable to underground mining methods.

The 2025 PEA mine plan is partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be classified as Mineral Reserves, and there is no certainty that the 2025 PEA based on these Mineral Resources will be realized.

The information presented in Sections 1 to 14 of this Technical Report also pertains to the 2025 PEA, as do Section 23 and Sections 25 to 27, and therefore are not repeated here. Information relating to Sections 15 to 22 content for the 2025 PEA is provided in the following sub-sections. Years presented in the 2025 PEA are for illustrative purposes only.

24.3.2 Mineral Processing and Metallurgical Testing

24.3.2.1 Summary of Metallurgical Testing (2022-2025)

In August 2022, an initial scoping-level metallurgical testwork program was commenced at SGS Lakefield ("SGS") located in Ontario, Canada. The objectives of testwork were to develop metallurgical data to evaluate and optimize various processes for the recovery of gold and silver, from Black Shale, Deep Sulphide, and Vein underground samples. Details of the SGS program are summarized below.

Concurrently, work was undergoing to investigate reprocessing the existing tailings impounded Mineral Resource and stockpile material at the Company's historical El Tigre Mine. The Company proposed to process at a nominal rate of 800 tpd via conventional recovery methods with material handling, milling, leaching, solid/liquid separation, and Merrill Crowe precipitation. Details of the SGS program are summarized below and referenced in the report titled, "An Investigated into the El Tigre Deposit, Project 19074-01 – Final Report" dated November 6, 2023.

In June 2024, SGS completed additional testwork on underground Vein, Black Shale, and Sulphides samples for mineralogical and metallurgical evaluation. Mineralogical testing included QEMSCAN characterization as well as gold deportment studies and metallurgical testing included sample characterization, comminution testwork, flotation, and scoping gravity separation testing. Details of the SGS program are summarized below and referenced in the report titled, "A Metallurgical Investigation on the El Tigre Underground Sulphide Deposit, Project 19074-03 – Final Report" dated December 9, 2025.

In February 2025, thickening testwork was completed by Metso (Canada) Incorporated ("Metso") on Black Shale, Deep Sulphide and Vein tailing samples. The three samples were prepared by SGS during an SGS flotation pilot plant testwork campaign. Metso completed the thickening testwork to determine the achievable underflow solids content of the samples and aid in confirming sizing and process design criteria. Thickening testwork was completed to also determine flocculant type and dose, overflow clarity, underflow density and underflow yield stress.

Following the thickening testwork, Metso completed filtration testwork on the samples to determine the filtering characteristics of each sample and to generate design data to permit filter sizing and operation. Filtration testwork was completed to also determine filter cloth type, cake thickness, maximum filtration rate, moisture content of the cake and cake handling characteristics. Details of the Metso thickening and filtration programs are summarized below.

The basis for the testing was for the base metal component (copper, lead, zinc) within the mineralized zone to produce saleable concentrate(s) that contained high vales of gold and silver. Upon review of the results and up-to-date mine plan, the desired process pivoted from flotation to

bulk cyanidation to coincide with the previous testing completed on the tailings and stockpile Mineral Resource.

In September 2025, metallurgical testwork (coarse rejects) was completed at Laboratorio Tecnológico de Metalurgia S.A. de C.V. ("LTM"), Hermosillo, Mexico. The objective of the testwork was to evaluate the extraction of precious metals as well as the consumption of key reagents. Details of the LTM program are summarized below and referenced in the report titled, "Estudio Metalúrgico Sobre Muestras de Compositos" dated September 3, 2025.

24.3.2.2 SGS Metallurgy Program Summary, 2022 – 2023

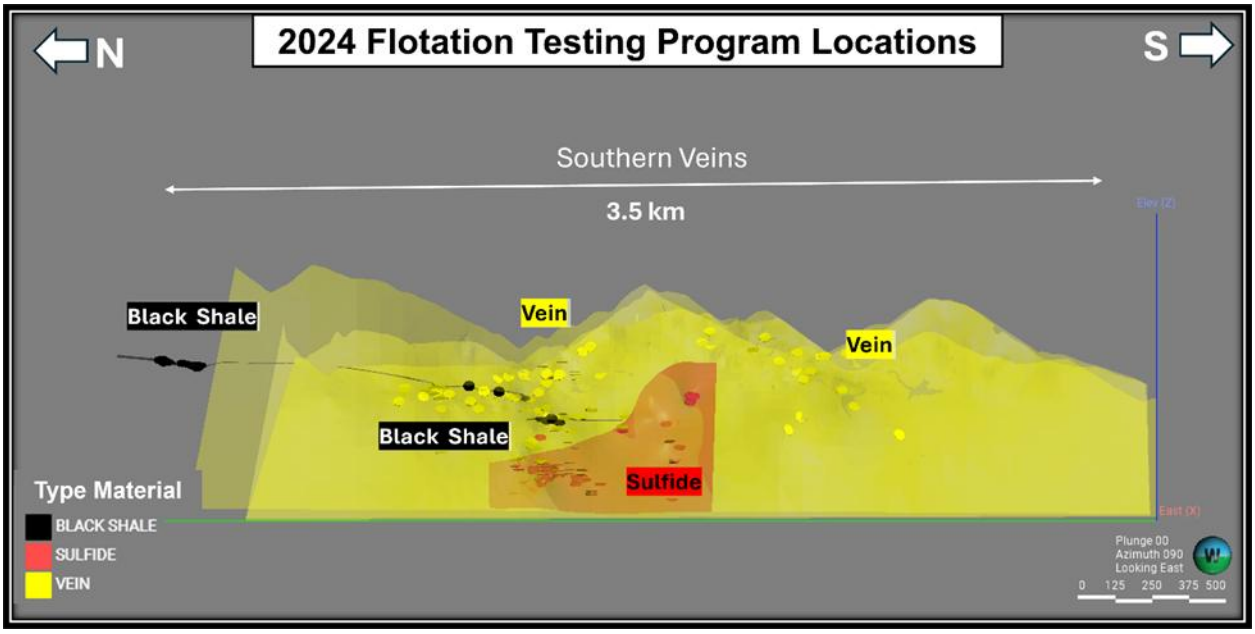
Sample Receipt and Preparation

One shipment of Black Shale and Deep Sulphide mineralization samples was received and utilized to form a Black Shale Composite and a Deep Sulphide Composite. 19 of the interval samples were combined, crushed to -10 mesh and blended to from ~45 kg of Black Shale Composite sample. From the composite, 12 x 2 kg, 12 x 1 kg and 3 x 500 g charges were riffled/rotary split out.

36 interval samples were combined, crushed to -10 mesh and blended to from ~56 kg of Deep Sulphide Composite sample. From the composite, 12 x 2 kg, 12 x 1 kg and 3 x 500 g charges were riffled/rotary split out.

The sample selection was determined and selected by the owner team and are shown in Figure 24.2.

FIGURE 24.2 SCHEMATIC 3-D VIEW OF FLOTATION SAMPLE LOCATIONS



Source: *PSM (2025)*

Mineralogy – XRD Rietveld Analysis

Subsamples of the Black Shale and Deep Sulphide master composites were submitted for a semi-quantitative XRD analysis via the Rietveld method. All minerals identified by X-ray diffraction were reported as a weight percent distribution and grouped into:

- Major (>30%)
- Moderate (10-30%)
- Minor (2-10%)
- Trace (<2%).

The minerals identified as moderate for various samples included orthoclase, muscovite and calcite.

Sample Characterization

Head sample analyses were completed on the composites and variability samples to determine the precious metal concentration as well as the content of other elements. Assays included gold (Au) and silver (Ag) in duplicate by fire assay, sulphur speciation for total sulphur (S_T) and sulphide sulphur (S=) by Leco, copper (Cu) by atomic absorption, and a semi-quantitative Inductively Coupled Plasma ("ICP") scan analysis.

Gold and Silver Analysis

Duplicate sample cuts of 30 g of ~75 µm pulverized material for the samples were submitted for both gold (Au) and silver (Ag) by fire assay.

Black Shale Composite
- Gold: 0.19 g/t Au
- Silver: 405 g/t Ag

Deep Sulphide Composite
- Gold: 0.15 g/t Au
- Silver: 630 g/t Ag

Sulphur Speciation

Representative pulverized (<75 µm) subsamples were submitted for sulphur speciation including total sulphur (S_T), and sulphide sulphur ($S^=$) by Leco analysis.

Black Shale Results
- S_T Composite: 4.93%
- $S^=$ Composite: 4.58%

Deep Sulphide Results
- S_T Composite: 8.90%
- $S^=$ Composite: 8.37%

Carbon Speciation

Representative pulverized (<75 µm) subsamples of the Black Shale and Deep Sulphide composites were submitted for carbon speciation including total carbon (C_T), and total organic carbon (C_{TOC}) by Leco analysis.

Black Shale Results

- C_T Composite: 0.49%
- C_{TOC} Composite: 0.05%

Deep Sulphide Results

- C_T Composite: 2.44%
- C_{TOC} Composite: 0.07%

Individual Copper Analysis

Representative pulverized (<75 µm) subsamples were submitted for individual copper (Cu) analysis by atomic absorption.

Black Shale

- Cu Composite: 0.37%

Deep Sulphide

- Cu Composite: 0.74%

Semi-Quantitative ICP Scan Analysis

Results from the semi-quantitative Inductively Coupled Plasma ("ICP") scan analysis are presented in Table 24.1.

TABLE 24.1
ICP SCAN ANALYSIS SUMMARY NO. 1

Element	Unit	Black Shale Comp.	Deep Sulphide Comp.
Al	g/t	19,200	23,600
As	g/t	< 30	< 30
Ba	g/t	160	676
Be	g/t	2	1
Bi	g/t	< 20	< 20
Ca	g/t	8,530	69,800
Cd	g/t	577	534
Co	g/t	<4	7
Cr	g/t	176	78
Fe	g/t	35,000	49,300

TABLE 24.1 ICP SCAN ANALYSIS SUMMARY NO. 1			
Element	**Unit**	**Black Shale Comp.**	**Deep Sulphide Comp.**
K	g/t	9,680	30,200
Li	g/t	77	30
Mg	g/t	2,520	3,130
Mn	g/t	3,770	11,700
Mo	g/t	25	9
Na	g/t	226	475
Ni	g/t	< 20	< 20
p	g/t	233	255
Pb	g/t	24,000	24,700
Sb	g/t	184	3,660
Se	g/t	< 30	< 30
Sn	g/t	< 20	< 20
Sr	g/t	16	92
Ti	g/t	456	401
Tl	g/t	< 30	< 30
V	g/t	138	6
Y	g/t	16	16
Zn	g/t	65,000	55,400

Source: *SGS (2023)*

Kinetic Rougher Flotation

Flotation is an extensively used and adaptable mineral processing technique capable of producing saleable concentrate grade from both simple and complex mineralization. Kinetic rougher flotation testing was conducted on the Black Shale and Deep Sulphide composites. Testwork included the collection of timed rougher concentrates and a rougher tailing, which were submitted for gold, silver, sulphur, copper, lead iron and zinc assay.

Test Parameters:

- Pre-Flotation P_{80} 75 µm
- 150 g/t potassium amyl xanthate ("PAX") sulphide
- 35 g/t of methyl isobutyl carbinol ("MIBC")
- Natural pH 22-Minutes (froth collection)

One set of tests also includes an addition of 85 g/t of Aerophine 3418A.

The results from Black Shale Composite, test F-1 with PAX and MIBC added, showed a combined rougher concentrate mass pull of 24.3%, yielding a recovery of 86.2% Au, 93.2% Ag, 79.0% S_T, 92.0% Cu, 95.7% Pb, 47.1% Fe and 75.7% Zn. These combined rougher concentrates produced grades of 0.88 g/t Au, 1,560 g/t Ag, 15.6% S_T, 1.44% Cu, 9.65% Pb, 6.96% Fe and 19.9% Zn

compared to the tailing grades of 0.05 g/t Au, 36.3 g/t Ag, 1.33% S_T, 0.04% Cu, 0.14% Pb, 2.51% Fe and 2.05% Zn.

For test F-3, with PAX, MIBC and 3418A added, the combined rougher concentrate had a mass pull of 28.4%, yielding a recovery of 92.3% Au, 94.1% Ag, 93.4% S_T, 94.2% Cu, 96.1% Pb, 52.3% Fe and 94.6% Zn. These combined rougher concentrates produced grades of 1.52 g/t Au, 1,349 g/t Ag, 15.6% S_T, 1.22% Cu, 8.10% Pb, 6.66% Fe and 21.1% Zn compared to the tailing grades of 0.05 g/t Au, 33.7 g/t Ag, 0.44% S_T, 0.03% Cu, 0.13% Pb, 2.41% Fe and 0.48% Zn.

A third test (F-5) was performed with only the addition of PAX and MIBC as with test F-1 to produce flotation concentrate and tailing for cyanidation testing.

The results from Deep Sulphide Composite, test F-2 with PAX and MIBC added, showed a combined rougher concentrate mass pull of 24.0%, yielding a recovery of 91.8% Au, 98.8% Ag, 88.1% S_T, 98.0% Cu, 99.1% Pb, 90.6% Fe and 66.5% Zn. These combined rougher concentrates produced grades of 0.71 g/t Au, 2,509 g/t Ag, 31.1% S_T, 3.10% Cu, 9.98% Pb, 18.7% Fe and 15.2% Zn compared to the tailing grades of 0.02 g/t Au, 10.0 g/t Ag, 1.33% S_T, 0.02% Cu, 0.03% Pb, 0.61% Fe and 2.41% Zn.

For test F-4, with PAX, MIBC and 3418A added, the combined rougher concentrate had a mass pull of 25.9%, yielding a recovery of 85.9% Au, 98.1% Ag, 94.7% S_T, 98.0% Cu, 98.7% Pb, 89.8% Fe and 87.7% Zn. These combined rougher concentrates produced grades of 0.52 g/t Au, 2,257 g/t Ag, 30.5% S_T, 2.80% Cu, 8.89% Pb, 17.4% Fe and 18.2% Zn compared to the tailing grades of 0.03 g/t Au, 15.0 g/t Ag, 0.60% S_T, 0.02% Cu, 0.04% Pb, 0.69% Fe and 0.89% Zn.

A third test (F-6) was performed with only the addition of PAX and MIBC to produce flotation concentrate and tailing for cyanidation testing.

The full rougher flotation summary for testing on the Black Shale Composite and Deep Sulphide Composite are presented in Table 24.2 and Table 24.3.

TABLE 24.2
BLACK SHALE KINETIC ROUGHER FLOTATION SUMMARY

Hole Sample ID	Test ID	Size P_{80} µm	Combined Product	Mass %	Assays							Distribution						
					Au g/t	Ag g/t	S_T %	Cu %	Pb %	Fe %	Zn %	Au %	Ag %	S_T %	Cu %	Pb %	Fe %	Zn %
Black Shale	F-1	88	Rougher Concentrate	24.3	0.88	1,560	15.6	1.44	9.65	6.96	19.9	86.2	93,2	79.0	92.0	95.7	47.1	75.7
			Rougher Tailing	75.7	0.05	36.3	1.33	0.04	0.14	2.51	2.05	13.8	6.8	21.0	8.0	4.3	52.9	24.3
			Head Crade (Calc)	100	0.25	406	4. 79	0.38	2.45	3.59	6.38	100	100	100	100	100	100	100
	F-3	86	Rougher Concentrate	28.4	1.52	1,349	15.6	1.22	8.10	6.66	21.1	92.3	94.1	93.4	94.2	96.1	52.3	94.6
			Rougher Tailing	71.6	0.05	33.7	0.44	0.03	0 13	2.41	0.48	7.70	5.9	66.0	5.8	3.9	47.7	5.4
			Head Crade (Calc)	100	0.47	407	4.75	0.37	2.39	3.62	6.33	100	100	100	100	100	100	100
	F-5	89	Rougher Concentrate	26.8	0.85	1,400	13.8	1.29	8.43	6.77	16.8	88.6	94.1	76.8	92.2	96,0	49.2	71.7
			Rougher Tailing	73.2	0.04	32.4	1.53	0.04	0.13	2.56	2.43	11.4	5.9	23.2	7.8	4.0	50.8	28.3
			Head Crade (Calc)	100	0.26	400	4.82	0.38	2.36	3.69	6..29	100	100	100	100	100	100	100

Source: *SGS (2023)*
Note: *S_T = total sulphur.*

TABLE 24.3
DEEP SULPHIDE KINETIC ROUGHER FLOTATION SUMMARY

Hole Sample ID	Test ID	Size P_{80} µm	Combined Product	Mass %	Assays							Distribution						
					Au g/t	Ag g/t	S_T %	Cu %	Pb %	Fe %	Zn %	Au %	Ag %	S_T %	Cu %	Pb %	Fe %	Zn %
Black Shale	F-1	88	Rougher Concentrate	24.0	0.71	2,509	31.1	3.10	9.98	18.7	15.2	91.8	98.8	88.1	98.0	99.1	90.6	66.5
			Rougher Tailing	76.0	0.02	10.0	1.33	0.02	0.03	0.61	2.41	8,2	1.2	11.9	2.0	0.9	9.4	33.5
			Head Crade (Calc)	100	0.18	610	8.49	0.76	2.42	4.95	5.47	100	100	100	100	100	100	100
	F-3	86	Rougher Concentrate	25.9	0.52	2,257	30.5	2,80	8,89	17,4	18,2	85.9	98.1	94.7	98.0	98.7	89.8	87.7
			Rougher Tailing	74.1	0.03	15.0	0.60	0.02	0.04	0.69	0.89	14.1	1.9	5.3	2.0	1.3	10.2	12.3
			Head Crade (Calc)	100	0.16	597	8.35	0 74	2.34	5.02	5.38	100	100	100	100	100	100	100
	F-5	89	Rougher Concentrate	25.0	0.55	2,410	28.9	2.89	9.06	17.6	14.4	90.2	98.8	88.7	98.0	99.0	90.9	67.4
			Rougher Tailing	750.0	2.00	10.0	1.23	0 02	0 03	0.59	2.32	9 8	1 2	11.3	2.0	1,0	9 1	32.6
			Head Crade (Calc)	100	0.15	610	8.15	0.74	2.29	4.84	5.34	100	100	100	100	100	100	100

Source: *SGS (2023)*
Note: *S_T = total sulphur.*

Sequential Flotation

Sequential flotation testing was explored to determine if saleable flotation products could be achieved. Both sequential rougher and cleaner flotation testing was performed on the Black Shale and Deep Sulphide samples to recover copper, lead and zinc minerals.

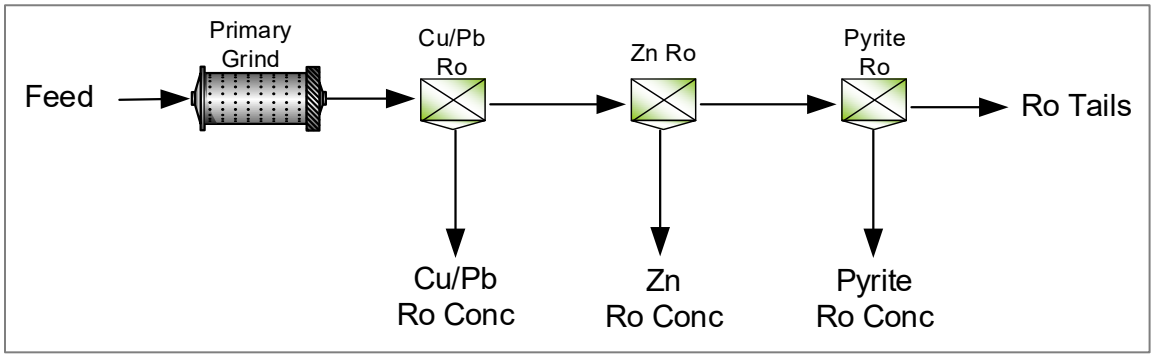
In the rougher circuit, copper and lead were recovered first by depressing zinc from floating, and then conditioning the copper and lead tailings to promote zinc flotation. $ZnSO_4$ and NaCN were used as depressant reagents to prevent zinc minerals from floating and sodium metabisulphite was utilized as a pyrite depressant while collectors such as 5100, 3418A and PEX (potassium ethyl xanthate) were used as collectors to recover the copper and lead minerals.

In the zinc circuit, $CuSO_4$ was used to reactivate depressed zinc minerals and PEX combined with 3418 was used to recover the zinc. Soda was used as a pH modifier in order to not deter galena recovery. Lime was also used as a pH modifier. MIBC was used as a frother on an as-needed basis depending on the physical characteristics of the froth.

Rougher Flotation Testwork

The rougher flotation flowsheet is shown in Figure 24.3.

FIGURE 24.3 ROUGHER FLOTATION FLOWSHEET



Source: *SGS (2023)*

For both the Black Shale and Deep Sulphide samples, optimal copper and lead recoveries were achieved in the Cu/Pb rougher flotation stage when the alternative Zn depression strategy test conditions were employed.

Black Shale Results:

- Cu/Pb Rougher Concentrate
  - Grades: 2.82% Cu, 20.7% Pb, 18.1% Zn
  - Recoveries: 77.0% Cu, 95.7% Pb, 30.6% Zn
  - Mass: 10.8%

- Zn Rougher Concentrate
  - Grades: 0.47% Cu, 0.50% Pb, 36.8% Zn
  - Recoveries: 14.0% Cu, 2.5% Pb, 68.2% Zn

- Mass: 11.8%

Deep Sulphide Results:

- Cu/Pb Rougher Concentrate
  - Grades: 8.18% Cu, 26.8% Pb,12.3% Zn
  - Recoveries: 89.4% Cu, 94.0% Pb, 19.2% Zn
  - Mass: 8.8%

- Zn Rougher Concentrate
  - Grades: 0.23% Cu, 0.20% Pb, 27.2% Zn
  - Recoveries: 4.7% Cu, 1.3% Pb, 78.4% Zn
  - Mass: 16.2%

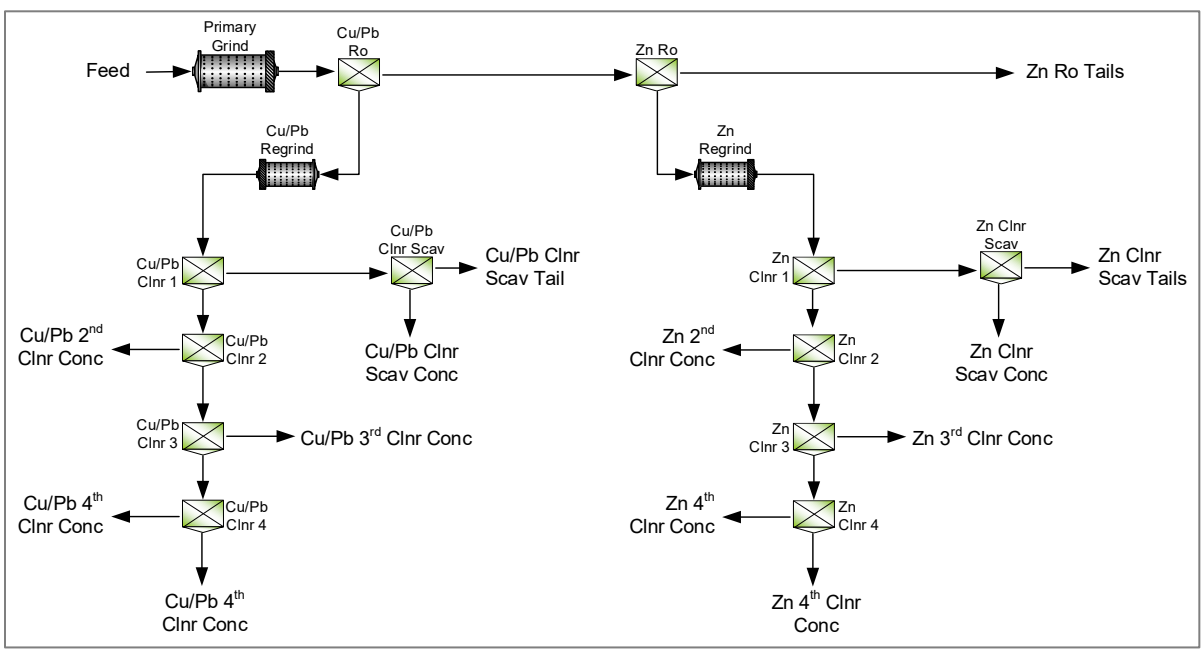
Silver and gold deported with the lead for both the Black Shale and Deep Sulphide materials and gold deported with the zinc for the Deep Sulphide material. The pyrite rougher stage flotation conditions tested were not seen to add value and were omitted in subsequent cleaning tests.

Mineralogical characterization of both samples is recommended to establish associations and liberation sizes of the base metal sulphides as well as precious metals.

Cleaner Flotation Testwork

The open-circuit cleaner flotation flowsheet is shown in Figure 24.4.

FIGURE 24.4 OPEN-CIRCUIT CLEANER FLOTATION FLOWSHEET



Source: *SGS (2023)*

Deep Sulphides Results:

- Cu/Pb Cleaner Concentrate

- Grades: 12.6% Cu, 41.5% Pb, 9.06% Zn
- Recoveries: 87.3% Cu, 95.3% Pb, 9.3% Zn
- Mass: 5.4%

- Zn Cleaner Concentrate
  - Grades: 0.36% Cu, 0.34% Pb, 56.6% Zn
  - Recoveries: 3.5% Cu, 1.1% Pb, 81.0% Zn
  - Mass: 7.6%

Black Shale Results:

- Cu/Pb Cleaner Concentrate
  - Grades: 13.0% Cu, 44.2% Pb, 7.21% Zn
  - Recoveries: 84.4% Cu, 94.7% Pb, 6.7% Zn
  - Mass: 5.1%

- Zn Cleaner Concentrate
  - Grades: 0.34% Cu, 0.26% Pb, 57.8% Zn
  - Recoveries: 3.3% Cu, 0.8% Pb, 79.4% for Zn
  - Mass: 7.5%

The Black Shale material was found to require finer grinding to achieve liberation and full mineralogical analyses for mineral grain sizes is recommended on both samples.

For both samples, closing the circuit is expected to improve the metal recoveries through recycling the middlings streams and locked cycle testwork is recommended to confirm.

Modified Acid Base Accounting

Modified acid-base accounting ("ABA") testing was completed on head samples for Black Shale, and Deep Sulphide composites to help evaluate tailings/waste rock propensity to generate acidic conditions and provide input parameters for kinetics testwork.

ABA testwork also provides quantification of the total sulphur, sulphide sulphur, and sulphate concentrations present, and the potential acid generation ("AP") related to the oxidation of the sulphide sulphur concentration. The test method determines the neutralization potential ("NP") of the sample by initiating a reaction with excess acid, then back titrating to pH 8.3 with NaOH. Carbonate concentrations were also be analyzed, and carbonate NP values were determined. The balance between the AP and NP assists in defining the potential of the sample to generate acid drainage.

Results for the head samples showed the solids are acid generating and have no net acid consumption potential as evidenced by the AP/NP ratio's of less than one and ranging from 0.22 to 0.77.

The net neutralization potential ("Net NP") results ranged from -58.24 to -100.06 t $CaCO_3$/kt of material, respectively, for the head samples. The ABA results summary for the head samples is presented in Table 24.4.

TABLE 24.4 HEAD SAMPLE MODIFIED ACID BASE ACCOUNTING SUMMARY			
Sample ID	**Unit**	**Black Shale Comp. Head**	**Deep Sulphide Comp. Head**
Paste pH	no unit	7.93	8.49
Fizz Rate	no unit	3	4
Sample Weight	g	2.15	1.99
HCl Added	ml	30.00	119.00
HCl	Normality	0.10	0.10
NaOH	Normality	0.10	0.10
NaOH to pH=8.3	ml	17.53	40.32
Final pH	no unit	1.81	1.50
NP	t $CaCo_3$/kt	29.0	198
AP	t $CaCo_3$/kt	129	256
Net NP	t $CaCo_3$/kt	-100.06	-58.24
NP/AP	ratio	0.22	0.77
Sulphur (total)	%	4.93	9.06
Acid Leachable So_4-S	%	0.80	0.87
Sulphide	%	4.13	8.19
Carbon (total)	%	0.485	2.34
Carbon (HCl)	%	2.12	11.7

Source: *SGS (2023)*

Results for the four leach tailing samples showed the solids are acid generating for the samples and have no net acid consumption potential as evidenced by the AP/NP ratio's of less than one and ranging from 0.07 to 0.16. The Net NP results ranged from -18.80 to -25.39 t $CaCO_3$/kt of material, respectively, for the three acid producing samples.

Results for the leach tailing from the Tailing Composite sample showed the solids are non-acid generating with a small net acid consumption potential as evidenced by the AP/NP ratio of greater than one, reported at 1.23. The Net NP results were calculated at 1.54 t $CaCO_3$/kt of material.

The ABA results summary for the leach tailing samples is presented in Table 24.5.

TABLE 24.5 LEACH TAILING SAMPLE MODIFIED ACID BASE ACCOUNTING SUMMARY			
Sample ID	**Unit**	**Heap Leach Composite C-1 Tailing**	**Heap Leach Composite C-2 Tailing**
Paste pH	no unit	8.96	9.57
Fizz Rate	no unit	2	2
Sample Weight	g	2.08	1.98
HCl Added	ml	20.00	20.00
HCl	Normality	0.10	0.10
NaOH	Normality	0.10	0.10
NaOH to pH=8.3	ml	19.24	18.54
Final pH	no unit	1.13	1.21
NP	t $CaCo_3$/kt	1.8	3.7
AP	t $CaCo_3$/kt	27.2	22.5
Net NP	t $CaCo_3$/kt	-25.39	-18.80
NP/AP	ratio	0.07	0.16
Sulphur (total)	%	0.95	0.88
Acid Leachable So_4-S	%	0.08	0.16
Sulphide	%	0.87	0.72
Carbon (total)	%	0.052	0.052
Carbon (HCl)	%	<0.04	0.35

Source: *SGS (2023)*

Net Acid Generation Testing

Net acid generation ("NAG") testing was completed on head samples for the Black Shale and Deep Sulphide composites to determine the balance between the acid producing and acid consuming components of the samples. NAG results confirm acid rock drainage characteristics of each sample based on oxidation of the samples sulphide content (as well as ferrous iron from siderite dissolution).

NAG testing uses hydrogen peroxide to react with the sample's sulphides and the produced acid is consumed by carbonates and/or other acid consuming components of the material. The pH of the solution is measured (NAG pH), and the remaining acid is titrated with standardized NaOH to determine the net acid generated by the reaction.

Results for the head samples showed NAG at pH 4.5 is 0.00 kg H_2SO_4/t for the samples. At NAG pH 7, net acid production potential is 0.00 kg H_2SO_4/t for the Deep Sulphide Composite sample with 4.8 kg H_2SO_4/t for the Black Shale Composite sample.

Results indicate that on complete oxidation of the Deep Sulphide sample, acid is not detected as the net result and thus is non-acid generating and has net acid consumption potential. For the

Black Shale sample, results indicate that on complete oxidation, acid is detected as the net result and thus is acid generating and has no net acid consumption potential. The full NAG summary of results for the head samples is presented in Table 24.6.

TABLE 24.6
HEAD SAMPLE NET ACID GENERATION TESTING SUMMARY

Sample ID	Unit	Black Shale Composite Head	Deep Sulphide Composite Head
Sample Weight	g	1.53	1.56
Vol H_2O_2	mL	150	150
Final pH	no unit	5.80	8.39
NaOH	Normality	0.10	0.10
Vol NaOH to pH 4.5	mL	0.00	0.00
Vol NaOH to pH 7.0	mL	1.49	0.00
NAG (pH 4.5)	kg H_2SO_4/t	0.0	0.0
NAG (pH 7.0)	kg H_2SO_4/t	4.8	0.0

Source: *SGS (2023)*

Results for the leach tailing samples shows the NAG at pH 4.5 ranges from 4.04 kg H_2SO_4/t to 5.89 kg H_2SO_4/t. At NAG pH 7, the net acid production potential ranges from 5.57 kg H_2SO_4/t to 6.16 kg H_2SO_4/t.

Results indicate that on complete oxidation of the samples, acid is detected as the net result and thus is acid generating and has no net acid consumption potential. The full NAG summary of results for the leach tailing samples is presented in Table 24.7.

TABLE 24.7
LEACH TAILING SAMPLE NET ACID GENERATION TESTING SUMMARY

Sample ID	Unit	Heap Leach Composite C-1 Tailing	Heap Leach Composite C-2 Tailing
Sample Weight	g	1.54	1.56
Vol H_2O_2	mL	150	150
Final pH	no unit	2.51	2.57
NaOH	Normality	0.10	0.10
Vol NaOH to pH 4.5	mL	5.89	4.04
Vol NaOH to pH 7.0	mL	5.57	6.16
NAG (pH 4.5)	kg H_2SO_4/t	19	13
NAG (pH 7.0)	kg H_2SO_4/t	18	19

Source: *SGS (2023)*

Conclusions and Recommendations

Black Shale and Deep Sulphide Composite samples were subjected to metallurgical testwork for the potential gold and silver recovery of the El Tigre Deposit.

The gold concentrations in the Black Shale and Deep Sulphide composites were 0.19 g/t and 0.15 g/t, respectively, and silver concentrations of 405 g/t and 630 g/t, respectively. Total sulphur grades were much higher than other samples tested, indicating the samples are refractory in nature. Flotation testing was selected as the main process for examination for these samples.

Rougher kinetic flotation on the Black Shale and Deep Sulphide composites produced good gold, silver, and sulphur rougher concentrates. However, after regrinding/intensive leaching of flotation concentrates combined with flotation tailings leaching, the overall gold and silver extractions were found to be moderate at 79% and 77%, respectively, for the Black Shale Composite and very low at 45% and 12%, respectively, for the Deep Sulphide Composite. Sequential flotation testing was explored to determine if saleable flotation products could be achieved.

Scoping sequential flotation testwork was performed on the Black Shale and Deep Sulphides materials with a typical sequential flowsheet to first float lead then zinc and provided sufficient concentration of each mineral. The Black Shale material required finer grinding to sufficiently liberate the base metals and full mineralogy on both samples was recommended to confirm the exact sizes needed.

Both samples were able to generate a Cu/Pb final concentrate grade of over 12% Cu and over 41% Pb with copper recoveries between 84.4% and 87.3%, lead recoveries between 94.7% and 95.3%, and a Zn final concentrate grade of over 57% with zinc recoveries between 79.4% and 81.0%. Locked cycle testwork was recommended to measure the amount of recovery gain that could potentially be achieved from recycling the middlings streams.

Preliminary environmental assessment of several of the head samples and leach tailing samples by ABA static testing and NAG testing found most of the samples are acid generating and have no net acid consumption potential.

24.3.2.3 SGS Metallurgy Program Summary, 2024

Sample Receipt and Preparation

Approximately 105 samples were received from the El Tigre underground sulphide deposit for the preparation of the Black Shale, Sulphides, and Vein composites. Selected samples were combined, crushed to -6 mesh, sub-sampled for comminution testwork, and then crushed to -10 mesh. From each composite, several 4 kg charges were riffled and split out, along with a subsample for head analyses. Any remaining material was saved and stored in a freezer as reject.

Head Characterization

Chemical Characterization

Head sample analyses were completed on the composites to determine the base and precious metal concentrations as well as the content of other elements. The Vein sample was observed to contained significantly less lead and zinc compared to the other two samples, and significantly more gold.

Bond Ball Mill Grindability

Bond ball mill grindability tests were conducted utilizing a closing screen size of 270 mesh (53 µm). Compared with the extensive SGS database, the Sulphide composite had a Bond Work Index ("BWI") of 16.8 kWh/t and was thus considered moderately hard while the Black Shale and Vein composites had BWI values of 20.1 kWh/t and 21.5 kWh/t respectively, and were considered very hard.

Mineralogical Characterization

Quantitative Evaluation of Minerals by Scanning Electron Microscopy ("QEMSCAN") mineralogical characterization was conducted on the composite samples. Testwork results indicated that the Vein sample contained significantly less sphalerite and galena compared to the Black Shale and Sulphides samples. The Sulphides sample contained the largest amount of pyrite. The Vein sample also contained the largest portion of sericite / muscovite compared to the other samples.

Sphalerite, galena, and Cu-sulphides mineral grains were all significantly finer than the grind particle sizes (80% passing 45 µm) for each of the samples, indicating the necessity for increased liberation in order to separate these minerals.

Flotation Testing

Prior sequential flotation testwork established baseline conditions, including grind size, reagent scheme, and the required flotation retention time for the Black Shale and Sulphides composites. Further flotation testwork was conducted to investigate the potential for optimizing the beneficiation results. Additionally, conditions from these tests were used with the Vein sample to evaluate the flotation response for this composite and investigate the potential for optimization.

Black Shale and Sulphides Composites Flotation

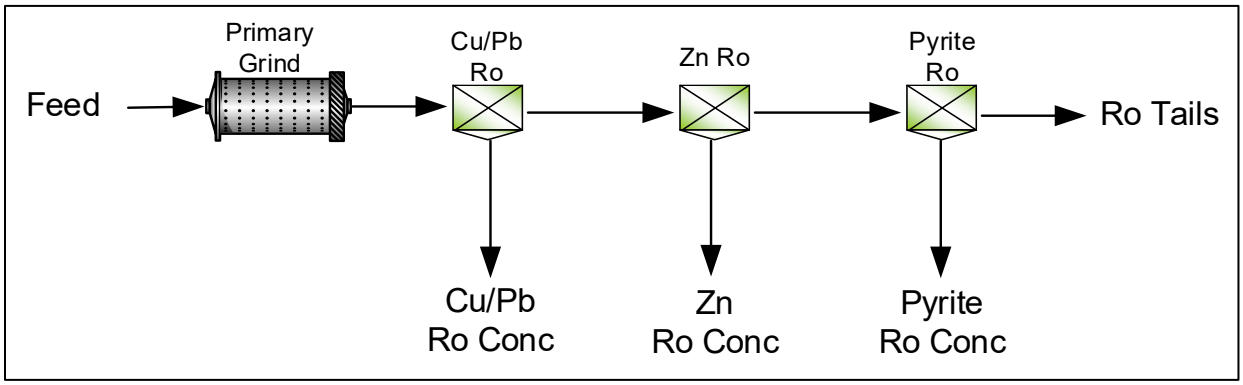
Rougher Flotation

Sequential rougher flotation testwork was completed on the samples to recover a combined Cu/Pb concentrate and a Zn concentrate. In the copper and lead rougher circuit, $ZnSO_4$ and NaCN were added as depressant reagents to prevent zinc minerals from floating and sodium metabisulphite was utilized as a pyrite depressant.

To recover the copper and lead minerals, Aero 5100 (modified thionocarbamate), Aero 3418A (dithiophosphinate), and PEX (potassium ethyl xanthate) were tested as collectors and soda ash was used as a pH modifier to not impede galena recovery.

In the zinc circuit, $CuSO_4$ was used to reactivate the previously depressed zinc minerals and PEX combined with Aero 3418A was used as the zinc collector while lime was used as a pH modifier. In both circuits, MIBC was used as a frother on an as-needed basis depending on the physical characteristics of the froth. The rougher flowsheet is illustrated in Figure 24.5.

FIGURE 24.5 ROUGHER FLOTATION FLOWSHEET



Source: *SGS (2025)*

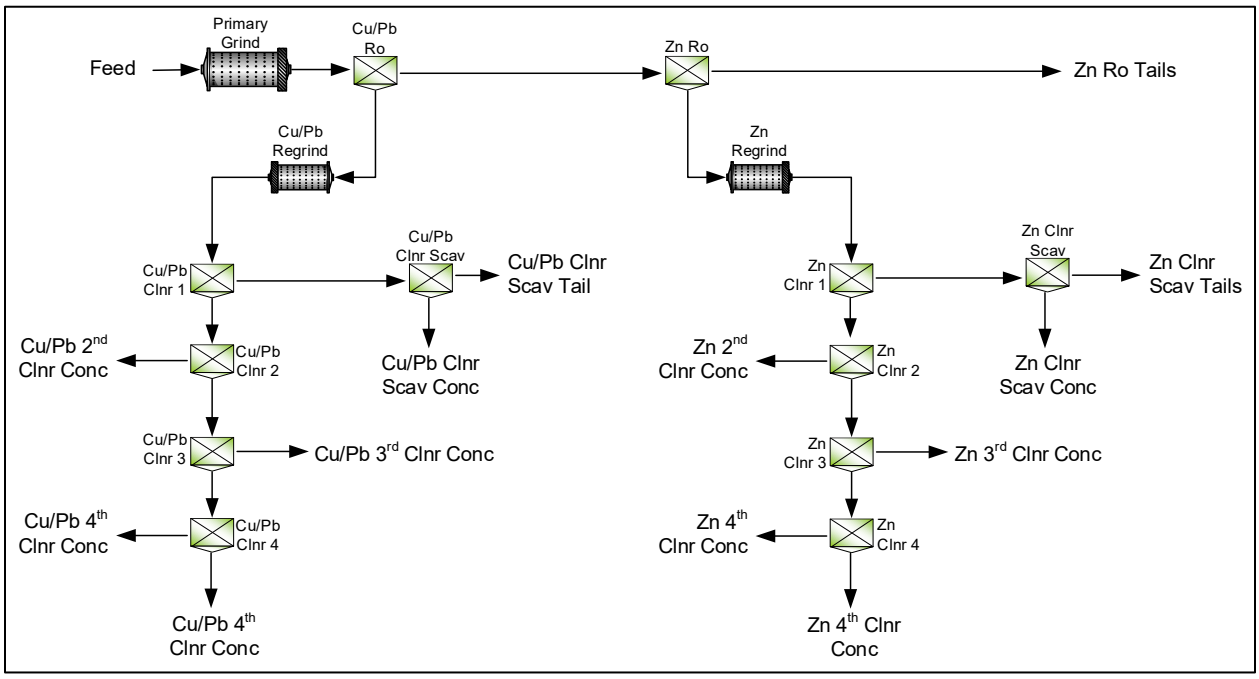
Testwork for the Black Shale sample indicated favorable results at a primary grind size of 80% passing 57 µm. A Cu/Pb rougher concentrate grading 2.05% Cu, 18.2% Pb, 5.26% Zn, 0.72 g/t Au, and 2,633 g/t Ag was obtained at recoveries of 78.6% Cu, 88.2% Pb, 15.7% Zn, 39.9% Au, and 91.0% Ag in 10.4% of the mass. The corresponding zinc concentrate graded 0.33% Cu, 0.82% Pb, 26.8% Zn, 0.55 g/t Au, and 173 g/t Ag at recoveries of 12.6% Cu, 4.0% Pb, 80.5% Zn, 30.5% Au, and 6.0% Ag in 10.4% of the mass.

The best results with the Sulphides sample were achieved at a grind size k_{80} of 42 µm. A Cu/Pb rougher concentrate grading 4.43% Cu, 17.9% Pb, 15.6% Zn, 3.01 g/t Au, and 4,212 g/t Ag was produced in 11.2% of the mass, at recoveries of 89.1% Cu, 89.6% Pb, 42.2% Zn, 70.7% Au, and 95.1% Ag. The corresponding zinc rougher concentrate graded 0.17% Cu, 0.35% Pb, 13.1% Zn, 0.87 g/t Au, and 79.2 g/t Ag at recoveries of 3.6 % Cu, 1.9% Pb, 38.3% Zn, 22.1% Au, and 1.9% Ag in 12.1% of the mass.

Cleaner Flotation

Cleaner flotation testing was performed on the Black Shale and Sulphides samples utilizing an open circuit and, in some tests, a Cu/Pb separation stage was conducted on the Cu/Pb 4th cleaner concentrate. In these tests, sodium metabisulphite and dextrin was utilized to depress the lead minerals and Aero 5100 was utilized as a collector to float the copper minerals. The open circuit flowsheet is illustrated in Figure 24.6.

FIGURE 24.6 OPEN-CIRCUIT FLOTATION FLOWSHEET



Source: *SGS (2025)*

Good results were obtained with the Black Shale material in one test where a 4th cleaner Cu/Pb concentrate grading 4.71% Cu, 63.4% Pb, 4.21% Zn, 1.25 g/t Au, and 8,691 g/t Ag was produced, at recoveries of 36.9% Cu, 69.7% Pb, 2.8% Zn, 16.9% Au, and 68.5% Ag in 2.3% of the mass. This was produced alongside a 1st cleaner Zn concentrate grading 0.72% Cu, 1.46% Pb, 57.6% Zn, 0.30 g/t Au, and 368 g/t Ag at recoveries of 11.9% Cu, 3.4% Pb, 82.3% Zn, 8.4% Au, and 6.1% Ag in 4.9% of the mass.

The 4th cleaner Cu/Pb concentrate produced in another test was separated into a Pb product and Cu 2nd cleaner product. The Pb product graded 1.06% Cu, 57.7% Pb, 7.62% Zn, 0.96 g/t Au, and 2,339 g/t Ag at open circuit recoveries of 5.6% Cu, 35.4% Pb, 3.0% Zn, 6.8% Au, and 10.9% Ag in 1.3% of the mass. The Cu 2nd cleaner product graded 23.0% Cu, 18.4% Pb, 4.43% Zn, 3.02 g/t Au, and 30,165 g/t Ag at open circuit recoveries of 38.0% Cu, 3.5% Pb, 0.5% Zn, 6.7% Au, and 44.2% Ag in 0.4% of the mass.

A cleaner test with the Sulphides composite produced a 4th cleaner Cu/Pb concentrate grading 10.9% Cu, 49.6% Pb, 5.71% Zn, 8.25 g/t Au and 11,200 g/t Ag at recoveries of 66.0% Cu, 78.6% Pb, 5.1% Zn, 61.1% Au, and 75.6% Ag in 3.4% of the mass. This was accompanied by a 1st cleaner Zn concentrate grading 0.50% Cu, 1.55% Pb, 55.7% Zn, 0.58 g/t Au, and 223 g/t Ag with open circuit recoveries of 3.1% Cu, 2.5% Pb, 50.6% Zn, 4.4% Au, and 1.5% Ag in 3.5% of the mass.

The 4th cleaner Cu/Pb concentrate produced in another test was separated into a Pb Product and a 2nd cleaner Cu concentrate. The Pb Product graded 1.75% Cu, 56.5% Pb, 14.1% Zn, and 2.17 g/t Au with open circuit recoveries of 6.6% Cu, 53.8% Pb, 7.3% Zn, and 9.1% Au in 2.0% of the mass. The 2nd cleaner Cu concentrate achieved grades of 24.9% Cu, 20.0% Pb, 4.94% Zn, and

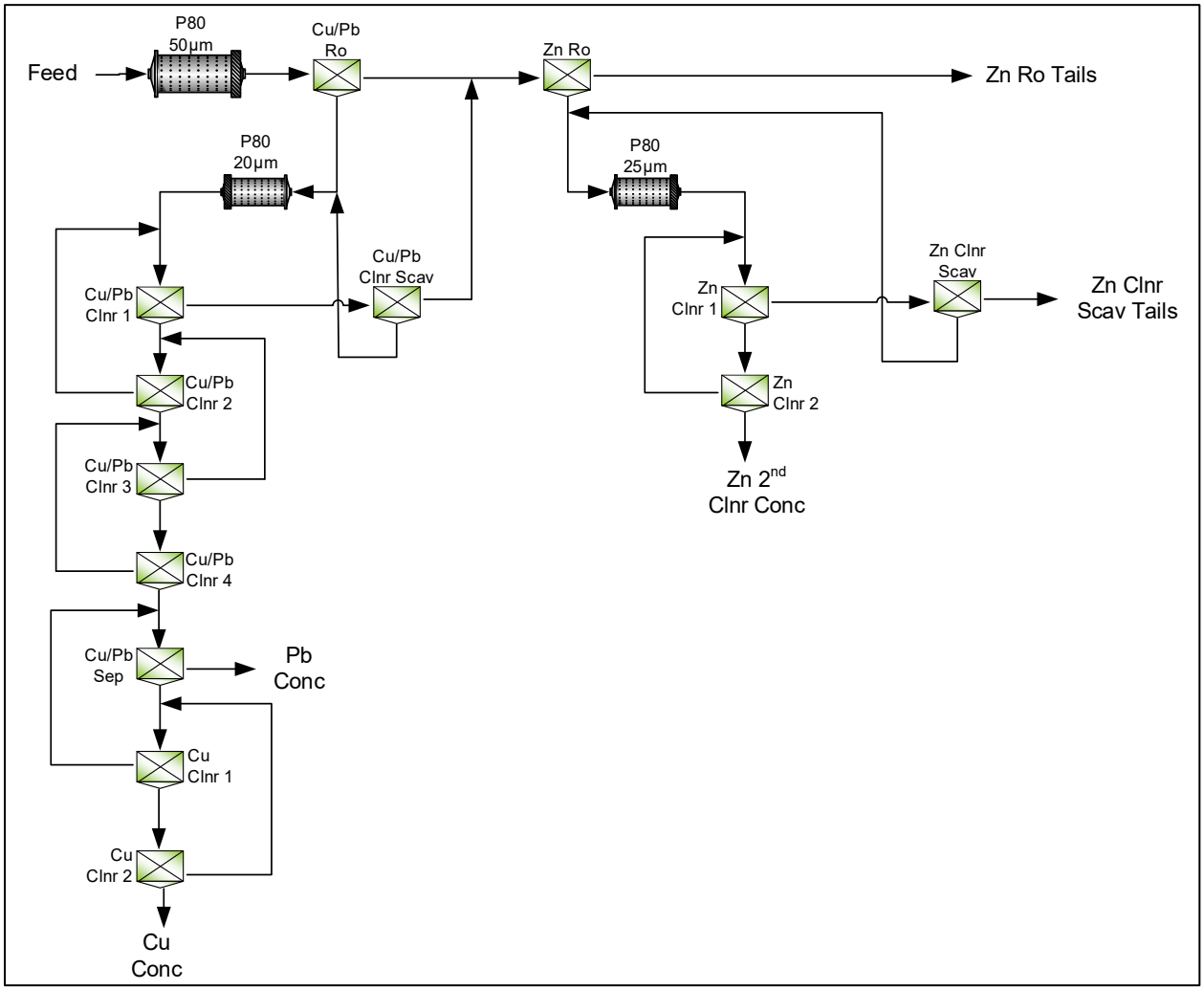
20.1 g/t Au at open circuit recoveries of 59.9% Cu, 12.2% Pb, 1.7% Zn, and 53.7% Au in 1.3% of the mass. There was unfortunately insufficient product mass for silver assays in this test.

Locked Cycle Testing

A Locked Cycle Test ("LCT") was performed on each of the Black Shale and Sulphides samples to simulate a continuous flotation circuit. In each test, a series of consecutive flotation test cycles was conducted, with recycling of intermediate streams from the previous cycle to build a recirculating load. The flowsheet used for each LCT followed the open circuit cleaner test protocol with the tailings stream of each of the cleaning stages reintroduced into the previous stage.

Results from a LCT can be used to predict overall performance of the conceptual flowsheet in a commercial plant in terms of expected grades and recoveries for base metals and precious metals. Six flotation cycles were conducted per test. The LCT flowsheet is illustrated in Figure 24.7.

FIGURE 24.7 LCT FLOWSHEET



Source: *SGS (2025)*

The results for testwork on Black Shale, projected a combined Cu/Pb concentrate grade of 5.35% Cu, 55.9% Pb, 5.65% Zn, 1.28 g/t Au and 7,483 g/t Ag with recoveries of 56.5% Cu, 79.1% Pb, 5.1% Zn, 22.6% Au and 79.0% Ag in 3.2% of the mass. This concentrate was separated into a Pb concentrate and a Cu 2nd cleaner concentrate.

The Pb concentrate graded 0.69% Cu, 64.7% Pb, 6.03% Zn, 0.81 g/t Au and 1,980 g/t Ag with recoveries of 5.3% Cu, 67.1% Pb, 4.0% Zn, 10.5% Au and 15.3% Ag in 3.2% of the mass. The Cu 2nd cleaner concentrate graded 18.1% Cu, 31.7% Pb, 4.61% Zn, 2.56 g/t Au and 22,585 g/t Ag at recoveries of 51.2% Cu, 12.0% Pb, 1.1% Zn, 12.1% Au and 63.6% Ag in 0.8% of the mass. Alongside this, a Zn 2nd cleaner concentrate was produced, grading 1.16% Cu, 3.17% Pb, 52.5% Zn, 0.43 g/t Au and 655 g/t Ag at recoveries of 23.2% Cu, 8.5% Pb, 90.0% Zn, 14.5% Au and 13.1% Ag in 6.0% of the mass.

The circuit obtained good stability with mass accountabilities for copper, lead, zinc, gold, silver, and iron were all between 97% and 101%.

The testwork on Sulphides produced a combined Cu/Pb concentrate grading 10.4% Cu, 47.4% Pb, 6.40% Zn, 8.14 g/t Au, and 10,757 g/t Ag at recoveries of 76.2% Cu, 85.6% Pb, 6.9% Zn, 65.6% Au, and 88.6% Ag in 3.9% of the mass. This concentrate was separated into a Pb concentrate and a Cu 2nd cleaner concentrate.

The Pb concentrate graded 1.36% Cu, 64.3% Pb, 7.70% Zn, 2.37 g/t Au, and 2,287 g/t Ag at recoveries of 6.1% Cu, 71.4% Pb, 5.1% Zn, 11.7% Au, and 11.6% Ag in 2.4% of the mass. The Cu 2nd cleaner concentrate graded 24.8% Cu, 20.5% Pb, 4.32% Zn, 17.3 g/t Au, and 24,273 g/t Ag at recoveries of 70.1% Cu, 14.2% Pb, 1.8% Zn, 53.8% Au, and 77.1% Ag in 1.5% of the mass.

In addition, a Zn 2nd cleaner concentrate was produced grading 0.86% Cu, 2.25% Pb, 56.9% Zn, 0.46 g/t Au, and 491 g/t Ag at recoveries of 9.3% Cu, 6.0% Pb, 90.0% Zn, 5.5% Au, and 6.0% Ag in 5.8% of the mass.

The circuit obtained good stability with mass accountabilities ranging from 97% for Zn to 102% for Pb. Copper and silver stabilities were lower at 95% and 94%, respectively, indicating that some of the copper and silver might have been accumulating in the system, which could affect the grade. Their results should be treated with caution.

Black Shale Composite Gravity Separation

A bench-scale Falcon Concentrator utilizing a fines bowl was tested on the Black Shale LCT rougher tails to investigate the potential for increasing overall gold recovery through incorporating gravity separation. The Falcon Concentrator was able to recover 40.9% of the gold in the rougher tails into 25.1% of the mass, however, the concentrate grade was low (0.19 g/t Au). A pyrite flotation step might be more effective and is recommended for future testwork.

Vein Composite Flotation and Gravity Separation

Vein Composite Rougher Flotation

Scoping sequential flotation testwork also completed on the Vein composite to investigate the amenability of this flowsheet to recover base metals and precious metals into saleable concentrates from the Vein composite.

Testwork demonstrated poor selectivity between copper-lead and zinc concentrates, therefore a combined concentrate was produced in one test grading 1.73% Cu, 2.91% Pb, 4.96% Zn, 8.99 g/t Au, and 3,086 g/t Ag at recoveries of 90.1% Cu, 90.2% Pb, 92.9% Zn, 85.6% Au, and 94.8% Ag in 13.7% of the mass. This was considered promising as a bulk concentrate.

Vein Composite Gravity Separation

The Vein sample was also subjected to gravity separation testwork at a k_{80} grind size of 84 µm utilizing a bench scale Falcon Concentrator with a fines bowl. When comparing the Falcon Testwork to the Rougher flotation testwork on the Vein composite, both methods recovered most of the gold (82.0% and 85.6%, respectively), although copper and silver were not recovered effectively by gravity (53.8% Cu and 57.7% Ag by gravity compared to 90.1% Cu and 94.8% Ag by rougher flotation).

Vein Composite Cleaner Flotation

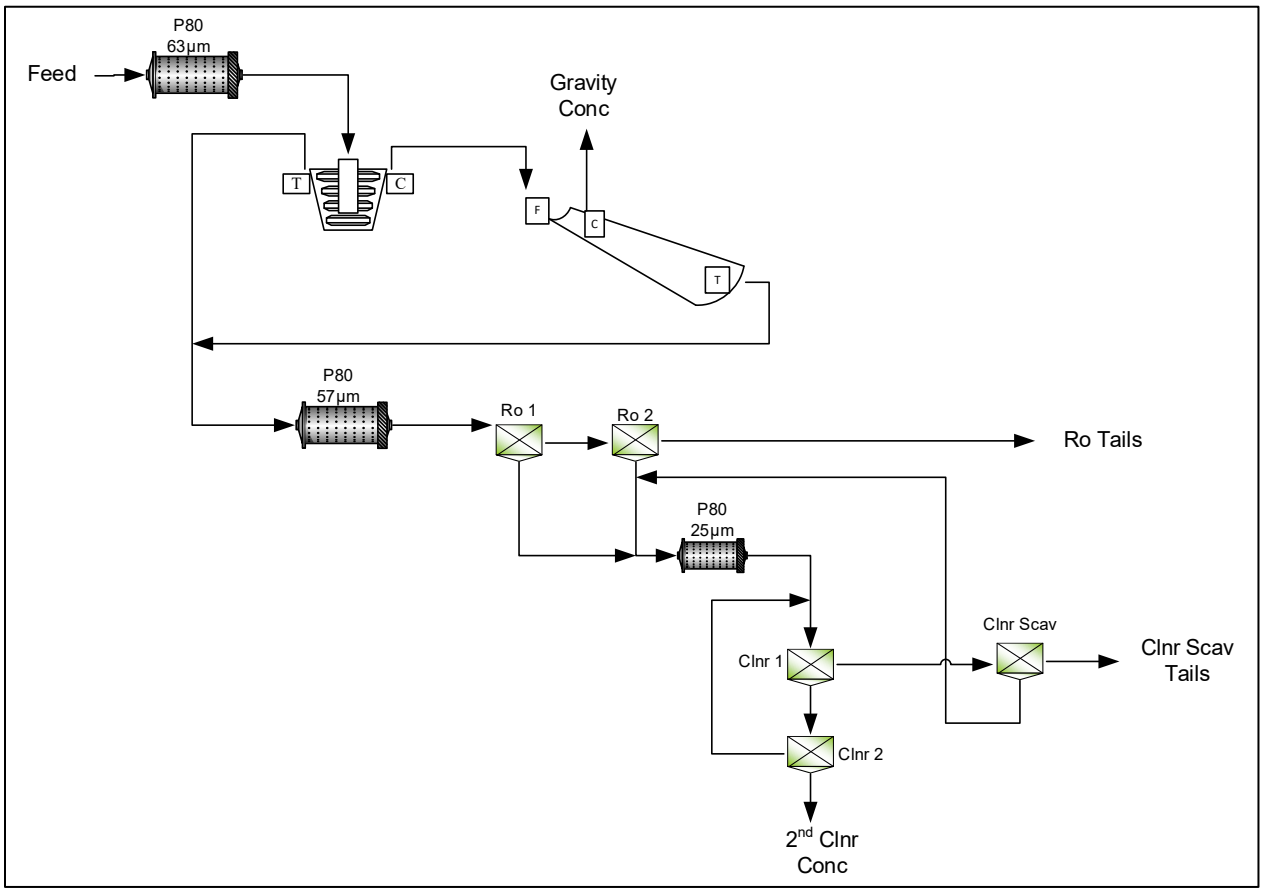
Scoping cleaning flotation testwork performed on the Vein sample included investigating the open-circuit cleaner flotation flowsheet with baseline conditions as determined for the Sulphides sample. In addition, a more traditional bulk flotation approach was investigated as well as evaluating upfront gravity separation prior to bulk sulphide flotation.

Cleaner testwork on the Vein Material demonstrated that optimal overall gold recovery was obtained in a test representing the conditions utilized on the Sulphide composite. In this test, a combined concentrate was produced grading 1.07% Cu, 1.75% Pb, 2.86% Zn, 5.30 g/t Au, and 1,876 g/t Ag at recoveries of 88.0% Cu, 87.3% Pb, 94.0% Zn, 85.3% Au, and 94.5% Ag in 21.5% of the mass.

Vein Composite Locked Cycle Test

LCT testwork was conducted on the Vein composite to simulate a continuous flowsheet. In this test, a series of consecutive test cycles was conducted, with recycling of intermediate streams from the previous cycle to build a recirculating load. The flowsheet utilized followed a test protocol comprising open circuit gravity separation followed by open circuit cleaning flotation, with the tailings stream from each of the cleaning stages reintroduced into the previous stage, as illustrated in Figure 24.8.

FIGURE 24.8 LCT FLOWSHEET ON VEIN



Source: *SGS (2025)*

Testwork yielded a Mozley concentrate grading 164 g/t Au and 5,703 g/t Ag at recoveries of 33.6% Au and 3.6% Ag in 0.3% of the mass. The rougher bulk flotation concentrate graded 1.37% Cu, 1.80% Pb, 3.89% Zn, 4.34 g/t Au, and 2,407 g/t Ag at recoveries of 94.9% Cu, 69.3% Pb, 97.5% Zn, 53.6% Au, and 92.4% Ag in 16.8% of the mass. Combining the rougher concentrate and Mozley concentrate gave overall recoveries of 87.2% Au and 96.1% Ag.

The 2nd cleaner concentrate graded 3.41% Cu, 4.33% Pb, 9.91% Zn, 9.23 g/t Au, and 6,053 g/t Ag at recoveries of 90.1% Cu, 63.6% Pb, 94.3% Zn, 43.3% Au and 88.3% Ag in 6.4% of the mass.

It was observed that while the gravity and 2nd cleaner concentrate had very good grades and good recoveries when combined, 10.3% of the overall gold was lost to the cleaner scavenger tailings and further work is recommended.

The stability of the circuit was very good, with mass accountabilities ranging from 99% for Au and Fe, to 101% for Ag.

Vein Composite Additional Scoping Cleaner Flotation

Additional scoping cleaning flotation testwork was performed to investigate the potential for reducing gold deporting to the cleaner scavenger tailings through the addition of Aero 3418A and increased PEX dosage as well as increased flotation time in this stage and in the 1st cleaner stage.

Testwork demonstrated that the additional flotation time and collector dosages did not significantly reduce the gold content of the cleaner scavenger tailings fraction, which still contained 7.0% of the overall gold. Gold deportment mineralogy on this material is recommended to investigate the liberation and association of gold lost to the cleaner scavenger tailings in order to tailor a more targeted approach.

Concentrate Characterization

Selected concentrate products from the LCT testwork were subjected to ICP-OES, mercury, and fluorine assays. Testwork indicated that the final concentrates contained the following potential penalty elements: <0.12% to 0.97% As, 0.06% to 0.54% Cd, 0.13% to 10.5% Sb, 7.3 ppm to 33.2 ppm Hg, and <50 ppm to 340 ppm F.

Conclusions and Recommendations

Metallurgical testwork was completed on the Black Shale, Sulphides, and Vein composite samples to investigate the potential for recovering base and precious metals into saleable concentrates. Testwork results are as follows:

- Head analyses showed that the Black Shale and Sulphides samples contained significant base metal and silver contents with limited gold content, while the Vein sample contained significant gold and silver with lower base metal contents.

- Bond ball mill grindability testwork placed the composite samples in the "moderately hard" to "very hard" category compared to the SGS database.

- Mineralogy revealed that the Vein sample contained significantly less sphalerite and galena than the Black Shale and Sulphides samples. The Sulphides sample contained the most pyrite while the Vein sample contained a significantly larger amount of sericite / muscovite compared to the other samples. The grain sizes of sphalerite, galena and the Cu-sulphides in all three composites were significantly finer than the grind size evaluated (80% passing 45 µm) indicating that additional grinding (regrinding) would likely be needed to fully liberate these minerals.

- Rougher flotation testwork demonstrated:
  - Promising results with the Black Shale composite at a primary grind k_{80} of 57 µm while the Sulphides sample required finer grinding to a k_{80} of 42 µm.
  - Black Shale and Sulphides composites responded well to a flowsheet with sequential rougher flotation to produce a Pb/Cu concentrate followed by rougher flotation to produce a Zn concentrate.

- Preliminary testwork on conventional processing of the cleaner Pb/Cu concentrate to produce separate Pb and Cu concentrates was also promising.
- Scoping sequential rougher flotation testing on the Vein sample showed that separation of the Pb/Cu and Zn mineralization was not possible. Testwork on the Vein composite therefore focussed on producing a combined Cu/Pb/Zn/Au/Ag concentrate. High recoveries of all the payable metals to a bulk concentrate were achieved.

- Cleaner flotation testwork demonstrated:
  - Baseline cleaning testwork produced positive results on Black Shale and good Cu/Pb separation was obtained utilizing sodium metabisulphite to depress the lead and Aero 5100 to collect the copper.
  - Cleaning of the Sulphides composite rougher concentrate also produced positive results, which responded well to increased zinc depressant dosages, finer regrinding, shorter Cu/Pb rougher flotation times with less collector to enhance selectivity.
  - As with the Black Shale cleaner Pb/Cu concentrate, separation of the lead and copper was possible for the Sulphides composite with sodium metabisulphite to depress the lead and Aero 5100 to collect the copper.
  - Optimal overall recoveries for the Vein composite were obtained utilizing the sequential cleaning conditions approach defined for the Sulphides sample. A combined concentrate was generated with >85% recovery of all the payable metals in one-fifth the mass.
  - A higher-grade precious metal concentrate could be generated by incorporating upfront gravity concentration.

- LCT testwork on the three composites utilized conditions based on the best results achieved in the open circuit cleaning flotation testwork. For the Black Shale and Sulphides samples, flotation time and copper collector were increased in the Cu/Pb 1st cleaner stage to minimize copper losses to the zinc circuit. For the Vein sample zinc depressants were omitted and $CuSO_4$ dosage was decreased while collector dosage was increased to maximize base metal and precious metal recovery. The Vein sample LCT also incorporated gravity separation and some slimes dispersant.

- LCT testwork demonstrated:
  - Black Shale composite sample generated the following products:
    - Lead concentrate grading 65% Pb at 67% recovery
    - Copper concentrate containing 18% Cu at 51% recovery
    - Zinc concentrate containing 53% Zn at 90% recovery

    Major source of copper loss was in the zinc concentrate (23%), and gold distributions to the Pb, Cu, and Zn concentrates were 10%, 12% and 15%, respectively, while silver deportments to the same concentrates were 15%, 64% and 13%, respectively.

  - Sulphides composite sample generated the following products:
    - Lead concentrate grading 64% Pb at 71% recovery
    - Copper concentrate grading 21% Cu at 70% recovery

- Zinc concentrate grading 57% Zn at 90% recovery.

Gold distributions to the Pb, Cu, and Zn concentrates were 12%, 54% and 6%, respectively, while silver deportments to the same concentrates were 12%, 77% and 6%, respectively.

- The Vein composite produced one flotation product, a combined 2nd cleaner concentrate that graded 3.4% Cu, 4.3% Pb, 9.9% Zn, 9.23 g/t Au, and 6,053 g/t Ag at recoveries of 90% Cu, 64% Pb, 94% Zn, 43% Au, and 88% Ag in 6.4% of the mass. This represented a ~16-fold increase in the grade of the mineralization.

- The Vein sample also yielded a Mozley gravity concentrate grading 164 g/t Au and 5,703 g/t Ag at recoveries of 34% Au and 4% Ag in 0.3% of the mass. Overall recovery in combination with the final flotation concentrate increases overall precious metal recovery. The two concentrates could either be combined for sale, or the gravity concentrate could be sold separately.

- Scoping flotation testwork on the Vein LCT cleaner scavenger tailings evaluating more aggressive flotation conditions found gold lost to this fraction was reduced from 10.3% in the LCT to 7.0% in the cleaner flotation scoping test.

- A mineralogical evaluation including gold deportment was recommended on the Vein cleaner scavenger tailings to develop a targeted approach for recovering more gold from this stream.

- Chemical characterization indicated that the final concentrates contained the following potential penalty elements:
  - <0.12% to 0.97% As
  - 0.06% to 0.54% Cd
  - 0.13% to 10.5% Sb
  - 7.3 ppm to 33.2 ppm Hg
  - <50 ppm to 340 ppm F.

The LCT results obtained on the Black Shale and Sulphide samples were promising and an economic evaluation is recommended. Results on the Vein sample were also promising, however, higher-grade products should be targeted. This is likely to lead to higher gold losses in the cleaner scavenger tailings and it is recommended to examine a flowsheet incorporating gravity gold recovery followed by aggressive rougher/cleaner flotation testing. Optimization testwork could examine new flotation technologies, such as the Jameson cell and a mineralogical evaluation to ascertain the cause of gold losses.

24.3.2.4 Metso Thickening Testwork, 2025

Sample Receipt and Preparation

Black shale, Deep Sulphide and Vein tailing samples were provided and prepared by SGS during a flotation pilot plant testwork campaign. The samples were received as a dry solid in 20 L pails and placed in a large barrel and diluted to create a slurry hydrating thoroughly for 24 hours.

Flocculant Selection

Static cylinder testing completed on the samples used a range of SNF products with varying molecular weights and anionic strengths to determine the quickest settling rate with clearest overflow at the lowest dosage. To aid in improving the overflow clarity, SNF Flobeads DB 45 VHM coagulant at 20 g/t was used.

Dynamic Thickening

The samples were dynamically thickened in the 99 mm rig and stock slurry was drawn from an agitated vessel by a variable speed peristaltic pump and diluted as required using a second peristaltic pump. Flocculant was added at a set flow rate at a range of concentrations to deliver the required dosage for each test. Coagulant was added at a set flow rate and diluted using the stock solution that was prepared.

Underflow Rheology

All rheological measurements were carried out using a Thermo Scientific Haake Viscotester iQ rheometer and an "OK600" 4 blade vane. The method measures shear stress versus time with the peak of this curve equating to the yield stress. The underflow yield stress is measured to generate data for use in the sizing of the thickener rake drive.

Feed Dilution

Dilution tests were conducted using Metso's optimum dilution test method to determine the maximum feed solids concentration for efficient flocculation with Metso's Reactor Feedwell. When adding flocculant, the optimum feed slurry concentration for the Black Shale tailing sample was found to be 16.1% solids and for both the Deep Sulphide and Vein tailing samples, the optimum feed slurry concentration was found to be 14.2% solids.

Testwork

Flocculant Screening

All flocculants used during static screening testwork produced visually clear overflow except for 920 VHM.

The flocculant with the best overall performance on the Black Shale tailing sample was the SNF 923 SH which formed floc agglomeration with the most stable at all floc dosages. The SNF 923 SH flocculant also had the best overall performance on the Deep Sulphide tailing sample which formed floc agglomeration with the most stable at all floc dosages and as well as on the Vein tailing sample which formed the largest floc agglomeration at the lower floc dosages and showed no signs of being fragile.

Dynamic Thickening

Multiple dynamic runs were conducted using different solid loading rates and different flocculant dosages to determine the impact on the underflow density and overflow clarities. The results would help aid in thickener sizing and design criteria for the process being considered while feeding the two tailings materials.

Test runs completed on the Black Shale tailing sample returned underflow densities of 59.3 – 60.1% solids (w/w) when using only flocculant and 51.2 – 54.5% when using flocculant and coagulant. Overflow quality ranging from 214 to 481 ppm was achieved using 30 - 50 g/t of flocculant and overflow quality ranged from 107 ppm to 117 ppm when using 40 g/t flocculant and 20 g/t coagulant. Measured underflow yield stresses were 57 - 92 Pa.

Test runs completed on the Deep Sulphide tailing sample returned underflow densities of 58.0 – 59.1% solids (w/w) when using only flocculant and 53.3 – 57% when using flocculant and coagulant. Overflow quality ranging from 219 to 346 ppm was achieved using 30 - 50 g/t of flocculant and overflow quality ranged from 184 ppm to 246 ppm when using 40 g/t flocculant and 20 g/t coagulant. Measured underflow yield stresses were 60 - 104 Pa.

Test runs completed on the Vein tailing returned underflow densities of 53.4 – 57.4% solids (w/w) when using only flocculant and 53.7% when using flocculant and coagulant. Overflow quality ranging from 262 to 1,834 ppm was achieved using 30 - 80 g/t of flocculant and overflow quality was 761 ppm when using 30 g/t flocculant and 20 g/t coagulant. Measured underflow yield stresses were 38 - 108 Pa.

Recommendations

Based on testwork results, the thickener design specifications presented in Table 24.8 are recommended for a Metso High-Rate thickener with Metso Reactor feedwell.

TABLE 24.8
THICKENER DESIGN SPECIFICATIONS

Process Stream	Black Shale	Deep Sulphide	Vein Tails
Solids feed rate (t/h)	36	36	36
Solids loading (t/(m^2·h))	0.7	0.7	0.7
Reporting feed slurry density (% solids (w/w))	25	25	25
Diluted feedwell slurry density (% solids (w/w))	14	14	14
Slurry pH	8.5	8.5	8.5

TABLE 24.8 THICKENER DESIGN SPECIFICATIONS			
Process Stream	**Black Shale**	**Deep Sulphide**	**Vein Tails**
Flocculant dosage (g/t)[1]	40	40	>60
Underflow density (% solids (w/w))	50-55	50-55	50-55
Overflow clarity (mg/L)[2]	<250	<250	250-300
Required thickener diameter (m)	8-9	8-9	8-9

Source: *Metso (2025)*
Notes: *[1] When using SNF Flobeads 45 VHM coagulant at 20 g/t.*
[2] Further mitigation process may be required if frothing issues arise during operation. To reduce frothing in the thickener, froth booms, bridge sprays and launder baffles may be used.
[3] Further testing required to optimize coagulant dosage or blending vein tails sample with either the Black Shale or Deep Sulphide sample to aid in improving the overflow clarity.

Conclusions

Testwork completed on the Black Shale, Deep Sulphide and Vein tailing samples demonstrated that the material can be successfully thickened and achieved underflow values that were within the desired target range of 50-55% solids.

When coagulant was used, the underflow density decreased from 59% to 54% for the Black Shale sample and 59% to 58% solids for the Deep Sulphide tail sample, while for the Vein tail sample the underflow decreased from approximately 57% to 54% solids.

Testwork on both the Black Shale and Deep Sulphide samples demonstrated that the overflow quality improved when using coagulant, however, the addition of coagulant for the Vein tailing sample showed negligible improvement in clarity.

For all samples, mitigation processes may need to be implemented due to frothing issues that may arise in full scale thickener operation. To reduce frothing in the thickener, froth booms, bridge sprays and launder froth baffles may be required.

24.3.2.5 Metso Filtration Testwork, 2025

Following thickening testwork on a Black Shale, Deep Sulphide and Vein tailing samples, Metso conducted filtration testwork using its bench scale Labox 100 pressure filtration unit to determine the filtering characteristics of each sample and to generate design data to permit filter sizing and operation.

Sample Receipt and Preparation

The underflow was mixed until homogeneous and the wt% solids was adjusted to approximately 55% for both the Black Shale and Deep Sulphide tailings samples and approximately 51% for the Vein tailings sample. The samples were pH adjusted to the desired process level of approximately 8.5 using soda ash and all testwork was completed at ambient temperature.

Testwork

A total of six runs were completed on the Black Shale tailing sample using the Labox 100 bench scale test unit in chamber filter ("CFP") mode. Filtrate for all runs contained some ultra-fines when using an FPC 310 filter cloth and was shown to be clear after approximately 10 seconds, returning a value of 512 ppm of solids. When using a 40 mm chamber, the filter cake thickness was 41 mm and cake moistures ranged from 17.4 – 17.6%. When using a 50 mm chamber, the filter cake thickness ranged from 49-52 mm and cake moistures ranged from of 18.1 – 18.8%. When using a 60 mm chamber, the filter cake thickness ranged from 60-61.5 mm and cake moistures ranged from of 18 – 21%.

A total of six runs were completed on the Deep Sulphide tailing sample using the Labox 100 bench scale test unit in CFP mode. Filtrate for all runs contained some ultra-fines when using an FPC 310 filter cloth and was shown to be clear after approximately 10 seconds, returning a value of 367 ppm of solids. When using a 40 mm chamber, the filter cake thickness ranged from 38.6 – 41 mm and cake moistures ranged from 17 – 18.9%. When using a 50 mm chamber, the filter cake thickness ranged from 51.1 – 51.8 mm and cake moistures ranged from of 17.1 – 18.8%. When using a 60 mm chamber, the filter cake thickness ranged from 58.6 – 60.6 mm and cake moistures ranged from of 19–20.5%.

A total of five runs were completed on the Vein tailing sample using the Labox 100 bench scale test unit in CFP mode. Filtrate for all runs contained ultra-fines when using an FPC 310 filter cloth and was shown to be clear after approximately 10 seconds, returning a value of 363 ppm of solids. When using a 40 mm chamber, the filter cake thickness ranged from 29.9 – 35.6 mm and cake moistures ranged from 21 – 23.4%. When using a 50 mm chamber, the filter cake thickness was 43.4 mm and cake moistures of 22.3%. Testwork suggests improvements in cake moisture may be achieved if a pressing diaphragm is used. Cake moisture while using a pressing diaphragm reduced to 18.5% when using a 40 mm chamber.

Conclusions and Recommendations

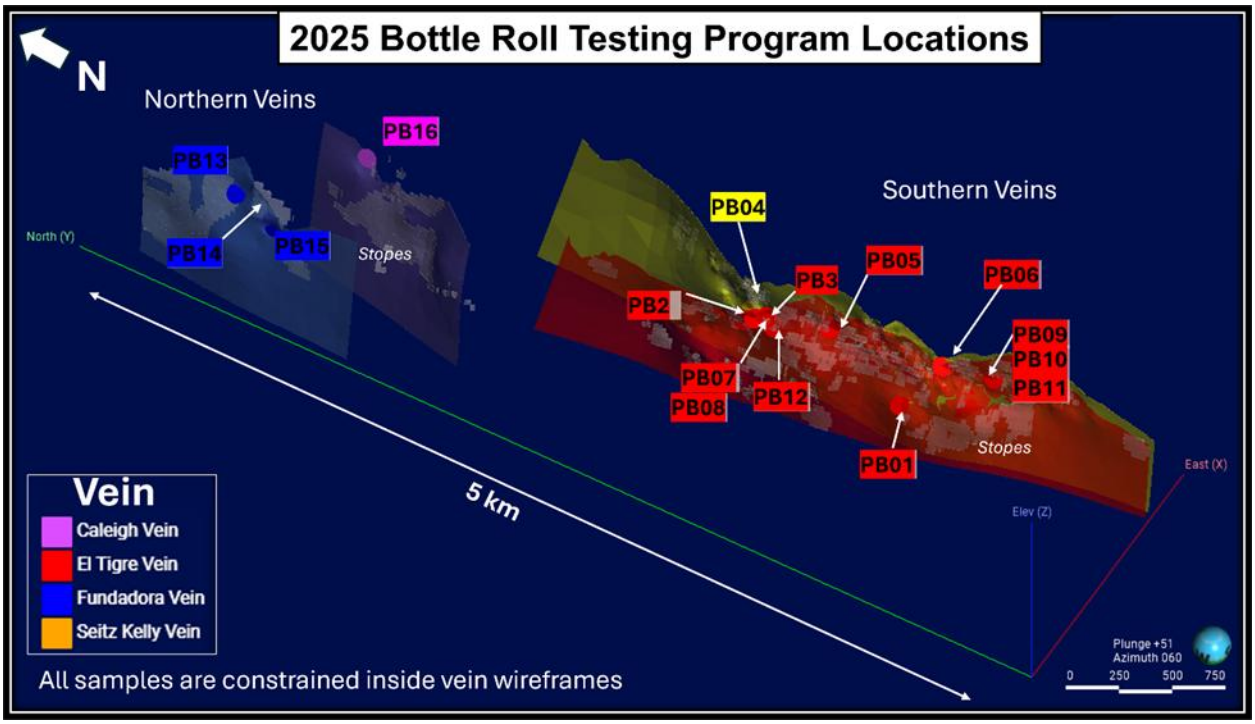
Testwork demonstrated that when using the same filtration parameters as the Black Shale and Deep Sulphide tailing samples, the Vein tailing sample achieves higher cake moistures. Further improvements in cake moisture may be achieved for the Black Shale and Deep Sulphide samples if a pressing diaphragm is used during filtration.

24.3.2.6 LTM Metallurgical Program Summary, 2025

A total of 16 samples coarse reject samples were received by Laboratorio Tecnológico de Metalurgia LTM S.A. de C.V. ("LTM") in Hermosillo and prepared for head analysis and bottle leach testing during 2025. On each sample, a controlled reduction of size to 80% less than 200 meshes was completed. From each sample, one charge was taken for sample characterization, and one load was taken for dynamic bottle leach testing.

The 16 sample locations within the underground vein structure are illustrated in Figure 24.9.

FIGURE 24.9 VEIN SAMPLE LOCATIONS



Source: *PSM (2025)*

Head Analysis

Testwork was completed to determine gold, silver and copper content in the 16 samples. Fire assay and atomic absorption were used to determine gold content while fire assay and acid digestion were used to determine silver content. Inductively Coupled Plasma Optical Emission Spectroscopy ("ICP-OES") was used to determine copper content.

Table 24.9 summarizes the results of the head analysis testwork.

TABLE 24.9
HEAD ANALYSIS

Sample ID		Weight (kg)	Total Au (g/t)	Total Ag (g/t)	Total Cu (g/t)
ETC	29123	0.832	0.215	188.7	1,064.360
ETC	46717	1.050	0.125	216.3	478.657
ETC	50118	1.912	1.179	438.5	64.770
ETC	52708	1.164	0.132	433.8	2,149.060
ETC	91031	1.364	1.550	29.8	212.043
ETC	101681	2.478	2.565	444.7	388.567
ETC	96972	3.002	0.920	658.8	1,180.677
ETC	96973	3.148	2.339	334.5	94.160
ETC	97963	1.592	0.182	107.0	75.167

TABLE 24.9 HEAD ANALYSIS					
Sample ID		Weight (kg)	Total Au (g/t)	Total Ag (g/t)	Total Cu (g/t)
ETC	97964	2.050	0.561	258.6	104.507
ETC	97966	1.256	2.071	597.2	289.490
ETC	102069	1.610	1.091	213.5	558.810
ETC	16851	1.180	1.008	143.8	1,507.073
ETC	17209	0.576	1.402	524.5	6,287.017
ETC	18557	1.592	1.001	7.5	133.100
ETC	20569	1.782	0.016	465.2	11.003

Source: *LTM (2025)*

Test results indicated that gold content in the samples from varied from 0.016 g/t to 2.565 g/t and silver content varied from 7.5 g/t to 658.8 g/t. Copper content had a medium-low concentrate varying from 11.003 g/t to 6,287.017 g/t.

Bottle Leach Testwork

Testwork was performed to determine gold and silver extraction as well as consumption of consumption of NaCN and CaO. Leach time was 72-hours using a load of 1 kg per test, with a solids content between 25% and 45%, at a concentration of 5 g/L NaCN.

Table 24.10 summarizes the results of the bottle leach testwork.

TABLE 24.10 BOTTLE LEACH TESTS				
Sample ID	Extraction Au (%)	Extraction Ag (%)	NaCN (kg/t)	CaO (kg/t)
PB01 ETC-29123	59.15	86.67	5.967	-3.625
PB02 ETC-46717	89.32	94.76	3.035	-2.401
PB03 ETC-50118	92.30	92.45	4.988	-0.375
PB04 ETC-52708	51.88	71.97	6.885	-1.663
PB05 ETC-91031	94.51	69.21	4.672	0.014
PB06 ETC-101681	98.38	95.41	4.678	-0.487
PB07 ETC-96972	94.74	96.07	5.661	-1.220
PB08 ETC-96973	95.86	90.80	2.312	-1.265
PB09 ETC-97963	91.30	52.10	1.654	-1.124
PB10 ETC-97964	94.55	60.21	1.133	-2.163
PB11 ETC-97966	97.90	86.93	3.304	-1.541
PB12 ETC-102069	83.43	85.70	2.858	-1.280
PB13 ETC-16851	79.97	85.80	5.475	0.907

TABLE 24.10 BOTTLE LEACH TESTS				
Sample ID	**Extraction Au (%)**	**Extraction Ag (%)**	**NaCN (kg/t)**	**CaO (kg/t)**
PB14 ETC-17209	73.10	48.24	5.075	-2.897
PB15 ETC-18557	88.13	82.89	2.897	0.115
PB16 ETC-20569	47.04	88.64	2.845	1.175
PB08 ETC-96973 QA/QC	95.28	92.99	2.74	-1.600

Source: *LTM (2025)*

11 of the 16 samples had extraction rates over 85% for both gold and silver. The highest rate of gold extraction was 98.38% and the highest silver extraction rate was 96.07%. The highest consumption of NaCN was 6.885 kg/t and the highest consumption of CaO was 1.175 kg/t. In most tests, total CaO consumption resulted in negative values.

Conclusions

For gold and silver extraction, 15 of the 16 dynamic bottle leaching tests meet LTM's quality control criteria with respect to the correlation between calculated head and tested head. This correlation reflects the accuracy of the tests in conjunction with the accuracy of the analytical processes, both for the samples tested and the metallurgical products obtained, developed in the project activities.

High NaCN consumption levels (unoptimized) are due to the high concentration of the leaching agent focused on the dissolution of precious metals, complex minerals, and cyanide-killing elements. There were negative CaO consumptions levels likely due to the high concentration of NaCN.

The quality assurance and quality control ("QA/QC") testwork confirmed the reliability of the test results. The original sample achieved a cumulative gold and silver extraction of 95.86% and 90.80%, respectively, while the QA/QC sample showed very similar values, with 95.28% Au and 92.99% Ag. The closeness between both results reflects strong analytical consistency, providing high confidence in the representativeness of the data generated during the leaching tests.

An internal review of the LTM samples and testwork was completed to confirm the most economical process and recovery methods to implement for the El Tigre underground mineralization.

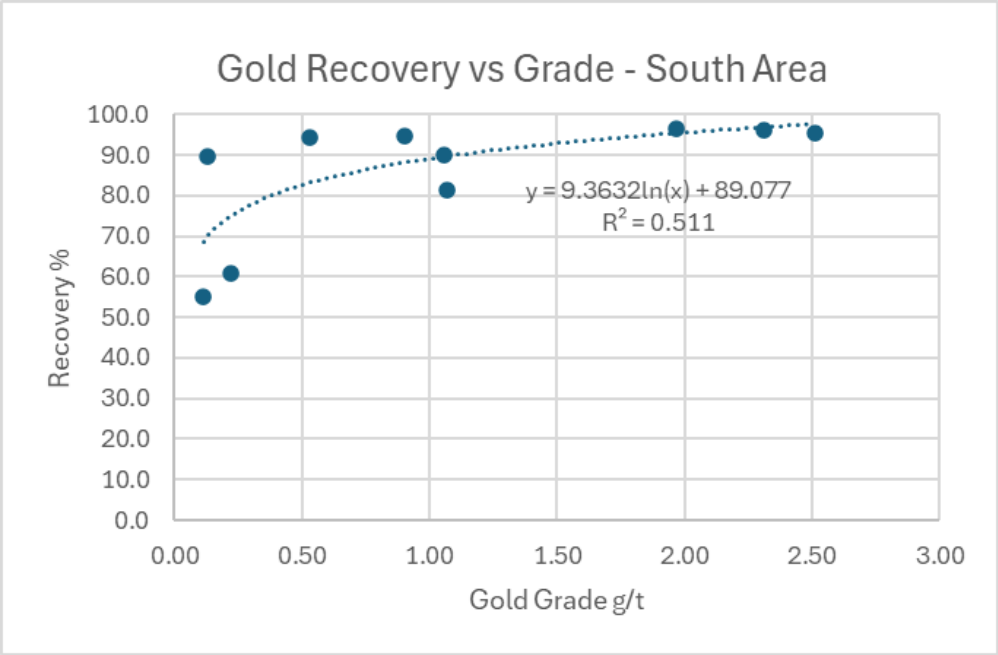
Table 24.11 summarizes test results from work completed by LTM on samples that are considered to be representative of the El Tigre underground South Area.

TABLE 24.11 LTM RESULTS SUMMARY						
Sample ID	**Total Au (g/t)**	**Total Ag (g/t)**	**Recovery Au (%)**	**Recovery Ag (%)**	**NaCN (kg/t)**	**Sample Zone**
PB01	0.22	182	60.7	78.8	6.0	ET Vein
PB02	0.13	217	89.7	96.3	3.0	ET Vein
PB03	1.06	428	90.1	91.1	5.0	ET Vein
PB04	0.11	403	55.3	74.8	6.7	SK Vein
PB06	2.51	452	95.4	93.7	4.7	ET Vein
PB07	0.90	626	94.8	93.1	5.7	ET Vein
PB08	2.31	333	96.0	89.7	2.3	ET Vein
PB10	0.53	268	94.2	58.7	1.1	ET Vein
PB11	1.97	572	96.6	87.4	3.3	ET Vein
PB12	1.07	181	81.4	87.9	2.9	ET Vein
Avg	**1.08**	**366.2**	**89.7**	**84.4**	**4.06**	
Std Dev	0.90	157.6	15.2	11.4	1.8	

Source: *D.E.N.M. (2025)*

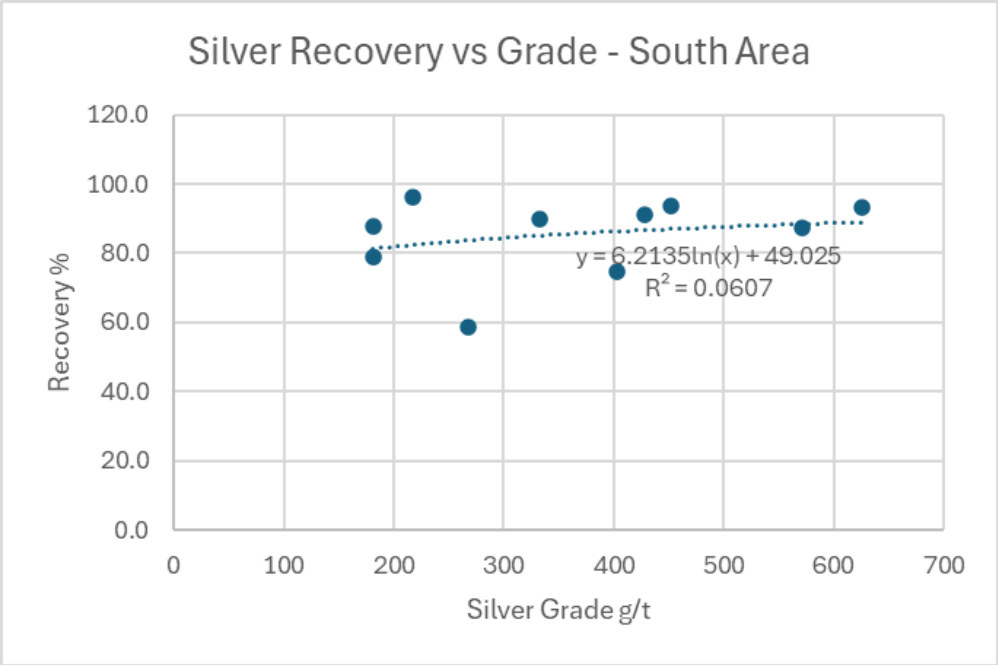
The test result recovery curves are presented in Figure 24.10 and Figure 24.11 for gold and silver, respectively.

FIGURE 24.10 GOLD GRADE RECOVERY CURVE



Source: *D.E.N.M. (2025)*

FIGURE 24.11 SILVER GRADE RECOVERY CURVE



Source: *D.E.N.M. (2025)*

Internal testwork determined that processing a combination of tailings and stockpile together with underground materials would be most economically beneficial. Table 24.12 summaries the combined processing strategy.

TABLE 24.12
COMBINED PROCESSING STRATEGY

Mineralized Samples Zones	Grade		Recovery		Consumption	
	Total Au (g/t)	Total Ag (g/t)	Extraction Au (%)	Extraction Ag (%)	NaCN (kg/t)	CaO (kg/t)
Tailings Material	0.27	78	82	83	1.4	-
Stockpile Material	0.85	176	92	71	1.4	1.4
Underground Material	1.05	293	89.7	84.4	3.9	2.0

Source: *D.E.N.M. (2025)*

24.3.3 Mineral Reserve Estimate

This section is not relevant to the 2025 PEA, as the 2025 PEA mine plan is based on Mineral Resources only.

24.3.4 Mining Methods

24.3.4.1 Introduction

The El Tigre Underground ("UG") Project is primarily a Longhole ("LH") mining operation utilizing cemented paste backfill ("PF") mining on three vein groups (Sooy, El Tigre and Seitz-Kelly) with a minor component of Cut-and-Fill ("C&F") mining. Additionally, there are extant stockpiles ("SPs") of mineralized material on surface, both as broken rock and as previously-processed tailings, which are included in the mine plan. Table 24.13 shows the components of the mine plan by source. The Project is planned to operate at 290 ktpa until the UG mine is exhausted, and then at 365 ktpa during recovery of the Tailings SP afterwards.

TABLE 24.13
MINE PLAN TONNES AND GRADES BY SOURCE

Source	Tonnes (k)	AgEq (g/t)	Ag (g/t)	Au (g/t)	Cu (%)[1]	Pb (%)[1]	Zn (%)[1]
Low-Grade SP[2]	119	101.1	77.6	0.27	N/A	N/A	N/A
UG Mining	4,000	386.2	292.8	1.05	0.041	0.120	0.207
Tailings SP[2]	904	251.1	175.7	0.85	0.022	0.217	0.101
Total	**5,022**	**331.7**	**251.3**	**0.91**	**0.033**	**0.099**	**0.166**

Notes: *1. Base metals provide no economic benefit (payable value) to the Project due to the use of cyanidation for precious metals recovery.*
2. Sampling of the Tailings SP did not include base metals analysis.

Historical Mine Workings ("HMW") exist in all three vein groups, and extend from surface to EL 1,670 at the north end of the mining area, and to EL 1,470 at the south end of the mining area. It is likely that some HMW excavations extend beyond these points, however, extensive mining beyond them is unlikely. Historical extraction was focussed on the El Tigre Vein, and more extensive excavation exists on it than the Sooy and Seitz-Kelly Veins. HMW stoping areas abut many stoping areas in the mine plan and are assumed to be full of free-running backfill material which will need to be excavated and segregated from the mineralized targets in the plan. This backfill material will be stockpiled on surface, and the voids left by its extraction filled with PF. Approximately 437 kt of waste backfill material will be excavated from HMW stopes during the UG life-of-mine ("LOM"), as will 2,438 kt of waste material from underground development. A total of 5,022 of mineralized material will be extracted from UG mining and surface SPs over the same period.

Initial commissioning process plant feed will be sourced primarily from the Low-Grade SP prior to the start of UG production, and the feed in the final three years will be sourced from the Tailings SP after the UG mine is exhausted. The life of the UG mine alone is ~14 years; the combined Project LOM is ~17 years.

While the Deposit is polymetallic, the value is derived from precious metals (Ag and Au) and the cyanidation process used for precious metals recovery negates any economic contribution from base metals. A silver equivalent ("AgEq") formula was utilized during evaluation of mining areas in the study, as follows, and utilizes a unit value (US$ per gram) of AgEq of $0.814:

$$Final_AgEq_{gpt} = Ag_{gpt} + Au_{gpt} * 88.597$$

A total of 76.2 linear km of lateral development is planned, comprised of 32.9 linear km of operating development and 43.3 linear km of capital development. An additional 5.6 linear km of vertical development is required over LOM.

No ramp-up of mining rates is planned in YR 1, since the process plant will be commissioned with surface SP material in YR-1, and multiple independent mining areas will be online by the end of the pre-production period. Peak development rates occur in YR 1 of the schedule when 8.2 linear km of lateral development is required in addition to 1.1 km of vertical development. Peak metal production occurs in YR 4 when the UG mine feed provides 6.2 Moz AgEq (3.6 Moz Ag and 29.1 koz Au) to the process plant.

24.3.4.2 Surface Stockpile Mining

Surface stockpiles will be excavated using standard truck-and-shovel methods. It is assumed that no additional dilution or mining losses will be incurred during these processes, and that the material will be delivered to the process plant in the same fashion as material from UG mining. The Low-Grade SP will be processed during plant commissioning in YR-1, with a small supplement from the Tailings SP (~18 kt) in the same period. The Tailings SP will also be processed after the UG mine is exhausted from YR 14-16 with a small tail into YR 17 of ~78 kt. As the Tailings SP requires minimal, if any, crushing, the expected throughput when the process plant feed is comprised of purely Tailings SP material increases to a nominal 365 ktpa from 290 ktpa.

24.3.4.3 Underground Mining

Underground mining is the source of the vast majority (80%) of tonnes in the mine plan, and the highest grades. Approximately 15% of mineralized UG material mined is derived from development operations, with C&F mining comprising 1% and LH mining comprising the remaining 84% of tonnes. C&F mining is used where flat-lying mineralized structures exist in the Black Shale Formation at insufficient thickness for LH mining. LH mining with PF backfill comprises 99% of stope tonnes, with the remaining portion extracted using C&F methods.

PF is used to backfill all production areas and historical stope voids. Three recipes will be used depending on the stope span (0-10 m, 10-15 m) and the mining direction (overhand/underhand). Total voids of 1.93 Mm^3 are planned to be filled, consuming approximately 2.45 Mt (49%) of the tailings produced during the mine plan (roughly 61% of the tailings produced from UG mining alone).

Methods

LH mining utilizes two different access configurations (longitudinal and transverse) and two different mining directions (overhand and underhand). Generally, longitudinal mining is overhand and uses downholes, and transverse mining is underhand and uses upholes, although exceptions do occur. Over LOM, 52% of LH stope mining utilizes upholes, and 60% of LH stope mining uses longitudinal access. Levels are spaced nominally on 30 m vertical intervals, with sub-levels

at half that height (15 m) utilized around HMW areas to improve drilling geometry and minimize draw cone "rat-holing" of HMW backfill.

C&F mining uses mechanized access either on a single cut or with an undercut and a single overcut. An access drift is driven down the centre of the C&F target area, and alternating crosscuts are driven perpendicularly to the access to the end of the mineralization using normal development methods. Once these crosscuts are complete, they are backfilled with PF and, after curing, the remaining crosscut drifts are driven between the PF pillars to extract the remaining mineralization. Where necessary, this process is repeated on the cut above by slashing the back of the access to create a ramp up to the next cut and running on top of the cured fill.

Mines and Zones

The UG mine has been divided into groups of stopes sharing similar attributes ("Zones") for the purposes of optimizing capital development expenditures. Stopes within a Zone are generally grouped by ease of access from a shared ramp, and minimized length of the access ramp. Most Zones consist of mining on a single main vein, but may access two veins depending on the proximity of the stoping targets.

A total of 16 Zones were originally designed, however, Zone 10 became uneconomic after revisions to the process recoveries and AgEq formulas. The remaining 15 zones are aggregated into 6 separately accessed mining Areas, as shown in Table 24.14.

TABLE 24.14
MINE PLAN TONNES BY AREA AND ZONE

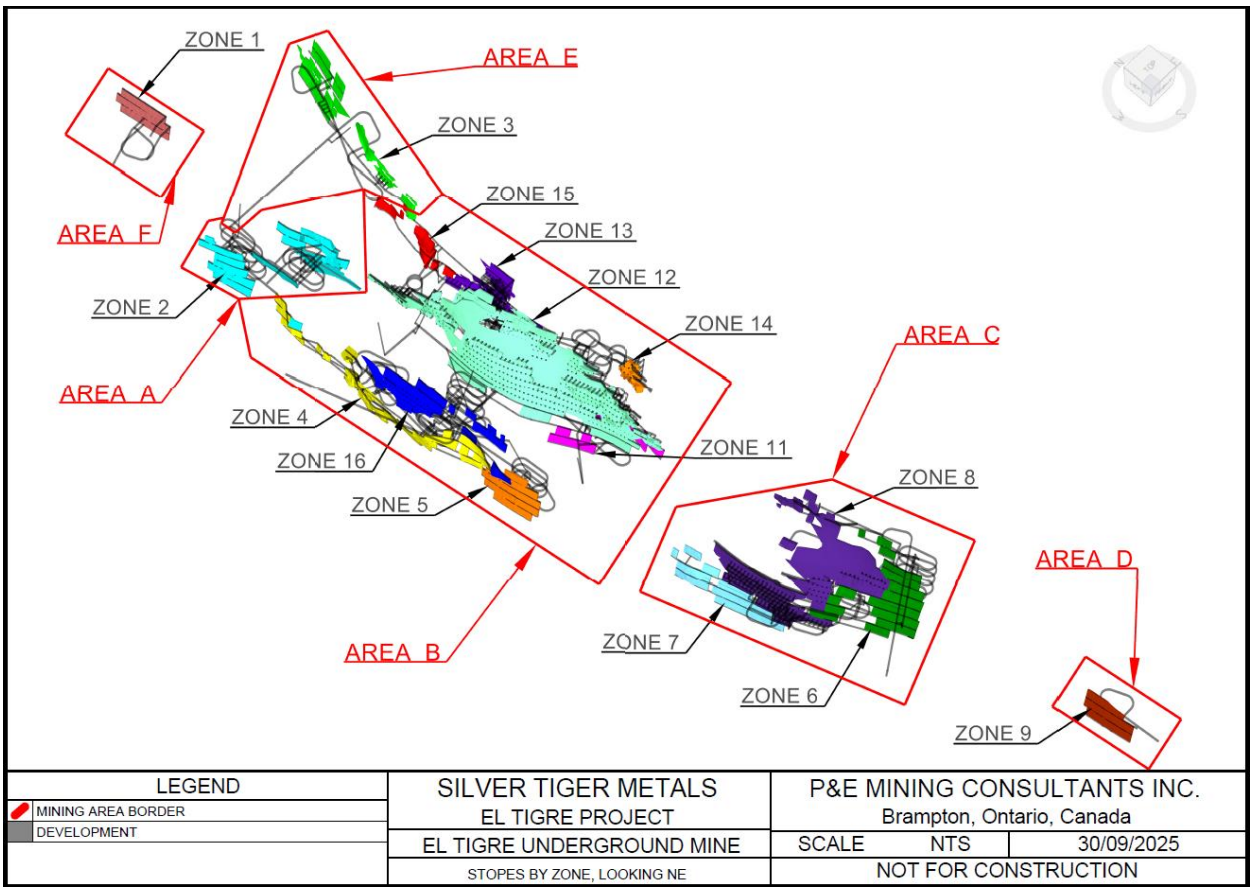
Area	Zone	Mineralized Dev (kt)	Mineralized Prod (kt)	Mineralized Tonnes (k)[1]	Avg. AgEq Grade (g/t)[2]	HMW Backfill (kt)	Waste Dev (kt)
A	2	57.6	313.4	371.0	354.8	-	170.6
B	4	64.9	438.5	503.4	288.1	-	247.4
	5	19.3	125.8	145.1	348.9	-	97.9
	11	34.8	181.7	216.4	278.4	-	72.1
	12	141.2	918.9	1,090.1	528.9	158.7	774.9
	13	41.7	201.7	243.4	303.7	56.8	147.7
	14	18.2	55.0	73.2	499.3	5.7	43.0
	15	12.2	101.8	114.0	315.9	3.8	26.5
	16	32.1	175.6	207.7	293.0	14.1	149.0
	Subtotal[1]	394.4	2,189.9	2,593.3	401.0	239.1	1,558.4
C	6	40.8	210.3	251.0	432.6	1.1	184.0
	7	27.9	136.9	164.8	347.1	-	69.6
	8	32.5	262.6	295.2	413.8	167.2	298.8
	Subtotal[1]	101.2	609.8	711.0	404.9	168.3	552.4
D	9	11.6	70.6	82.2	316.5	-	22.3

TABLE 24.14 MINE PLAN TONNES BY AREA AND ZONE							
Area	**Zone**	**Mineralized Dev (kt)**	**Mineralized Prod (kt)**	**Mineralized Tonnes (k)[1]**	**Avg. AgEq Grade (g/t)[2]**	**HMW Backfill (kt)**	**Waste Dev (kt)**
E	3	19.9	126.5	146.5	244.7	29.8	100.3
F	1	12.7	83.2	95.8	244.5	-	34.8
Total[1]		**597.4**	**3,402.5**	**3,999.8**	**386.2**	**437.1**	**2,438.9**

Notes: *1. Totals may not sum due to rounding.*
2. AgEq grades were recalculated after scheduling from interrogated Au and Ag grades using the final formula from Section 24.3.4.5.

Figure 24.12 shows the mining Zones and mining Areas. The Low Grade and Tailings SPs are not shown.

FIGURE 24.12 3-D ISOMETRIC DRAWING OF EL TIGRE UNDERGROUND MINE AREAS AND ZONES



24.3.4.4 Design Considerations

Production Rate

The production rate for the UG was set at 290 ktpa (nominally 800 tpd) by the Company based on an existing permit and the previous purchase of a used ball mill. Stress-testing of the UG production schedule indicates that higher feed rates can be maintained under different scheduling constraints. Long's modification to Taylor's Rule (2009) indicates a production rate in excess of 1,500 tpd is possible, however, the plan was not tested to this level.

Cut-off Grades and Economics of Stope Selection

Three Cut-off Grades ("COGs") were generated for the mine: Development ("DCOG"), Marginal ("MCOG"), and Economic ("ECOG"). The MCOG used for generating stopes was 150 g/t AgEq, utilizing the AgEq formula from April 16, 2025 and an AgEq value of $0.837 per gram (refer to Section 24.3.4.5 below). Stopes created at this COG generate positive operating margin. These stopes are then grouped together, and development designed to facilitate their extraction. A cross-check is then performed to ensure that the required operating development to recover the stopes does not exceed the operating margin (e.g. that the cost of designed development does not exceed the estimated cost of development). Groups that survive this test are then grouped into mining Zones, and capital development designed to extract the Zone. The same cross-check is then performed on the groups by level and Zone to ensure that they continue to generate positive margin even after the addition of capital development.

This method generates a larger tonnage base for the mine and greater total cash flow than using ECOG alone, however, this can incur some negative opportunity cost when stopes grading between MCOG and ECOG are processed where higher-grade stopes were available (although this is no different to the opportunity cost of processing two stopes that both exceed the ECOG but have appreciably different grades). This opportunity cost can be mitigated through a stockpiling strategy, however, for this Project it is assumed that stopes are generally processed in the order mined, and that stopes on any specific level of any specific Zone all have the same average grade, as it will be largely impossible to segregate mineralized material from specific stopes due to the production rate per face and the number of active faces on a level.

Since the AgEq formula changed after stopes were designed, a similar vetting process to what is described earlier in this section was repeated prior to the final selection of the stopes utilizing the final AgEq formula.

Historical Mine Workings

Extensive historical mining has occurred on the El Tigre Property. HMW areas exist on all three major vein groups (Sooy, El Tigre and Seitz-Kelly) and include both stopes and development. For the purposes of this PEA, stopes are assumed to be backfilled with free-running material, and development is assumed to not be backfilled, but may be collapsed. The Company has provided a 3D modelling solid created from drilling intercepts and historical mapping which has defined the HMW locations sufficiently for a PEA level of study, however, it is likely that the true locations of HMW differ slightly from the solids provided to the Authors.

The mine design generally avoids re-using existing HMW development, however, the intersection of new development and HMW development is unavoidable and occurs in many locations throughout the planned UG mine. Generally, the HMW development is intersected perpendicularly to strike by crosscuts driven from a footwall ("FW") drift, however, there are a few areas where longitudinal drifts intersect them along strike, or where access is from a hanging wall ("HW") drift. In these locations, no allowance for the re-use of the existing void is provided: it is assumed that the full cost of driving a face in virgin ground will apply, and that the full quantity of material from a face in virgin ground will need to be handled.

Intercepts of backfilled HMW areas suggest that the material does contain some mineralization, however, this material is not included as potentially mineable resource due to questions about the consistency of grades and the spacing of intercepts. Additionally, the intercepts suggest that not all of the HMW voids are backfilled. Regardless, the assumptions for this PEA are that HMW stopes are full and HMW development is empty, as stated above. The detailed extraction sequence for mining stopes adjacent to HMW stopes is described in Section 24.3.4.7. The approximate average width of HMW production excavations is 0.97 m.

Geotechnical

WSP was engaged to perform basic geotechnical analyses for the Project, including a review of backfill strengths, crown pillar thicknesses, stope sizing and ground support recommendations. Stope size recommendations vary by area from 15-30 m L for a 30 m H stope, up to a span of 15 m W. In general, stopes will be approximately 25 m L for areas utilizing longitudinal longhole retreat mining, and 10 m L for areas utilizing transverse longhole mining around HMW (this short spacing is driven by access and recovery considerations rather than geotechnical limitations). Further details are provided in Section 24.3.4.7.

WSP also provided guidance on Crown Pillar thickness. Poor quality ground (Q value of 1-4) was assumed, and a thickness-to-span ratio of 2:1 was recommended, with a minimum pillar thickness of 15 m. This guidance encompasses both the natural surface, and any excavated surface (e.g. pit walls).

For longitudinal retreat mining in virgin ground, WSP's stope sizing guidance has been followed, with stopes designed in 5 m L increments along strike, allowing individual increments to be aggregated into larger stopes as necessary to comply with the guidance. Where stopes abut HMW workings they have been limited to a strike of 10 m L due to operational constraints. HMW voids in the UG average less than 1.0 m in width: since the draw-down of HMW material to a relatively level floor is a necessity for the plan of extraction of adjacent stopes, crosscut access have been spaced in close proximity to each other (10 m centres) to facilitate even extraction and mitigate the possibility of a hang-up of HMW backfill. Given that the crosscuts are spaced on 10 m centres and ~50% of transverse stopes are less than 4.0 m W, it is assumed that each stope will be serviced by a single crosscut to maintain effective extraction geometries and allow for a primary-secondary extraction sequence.

Geological

The Deposit is intersected by three major faults: "Fortuna", "Espuelas" and "Bypass". The Fortuna and Bypass Faults trend north-northwest–south-southeast, the Espuelas Fault trends west-northwest–east-southeast. A fourth fault, "Combination Canyon" exists approximately 150 m north of the furthest mining extents of Zone 1, and trends west-southwest to east-northeast, which does not affect the planned UG mine.

Of these faults, the Fortuna Fault has the most impact, since it crosscuts the main access ("Level 7") into the Area B, and intersects the HW of Zone 4 from 1,600 EL to 1,720 EL. It is expected that stopes in the zone of influence of the fault will require additional ground support and possible reduction in strike length as a result.

The Espuelas Fault crosscuts isolated areas of Zone 4 and Zone 15 and forms the northern boundary of Zone 12. The Bypass Fault is located well into the HW of the Deposit and has minimal influence on the mine plan outside of the positioning of the main access adit into the central part of the mine.

Additional geological considerations arise from the mineralogy of the Deposit: appreciable base metal (Pb, Zn, Cu) contents appear in sulphide-rich areas, notably in Zone 4 and the C&F portion of Zone 2 where mining intersects the Black Shale and Sulphide units. While economic recovery of the base metals is possible, the detrimental effects on precious metals recovery outweigh the economic additions of the base metals. As a result, the base metals will not be recovered during processing and provide no economic benefit (see Section 24.3.4.5).

Contractor Versus Owner

The Company has requested that the UG mine be planned to be contractor-operated. A budgetary quote was provided by a large Mexican national mining contractor for the expected sizes of development in the mine plan. The Authors recommend updating this quote with the quantities from the LOM plan to refine the costs at the next stage of study.

Surface Mine Influences

The UG mining areas exclude stopes within the Open Pit ("OP") design described in the technical report titled "Pre-Feasibility Study of the El Tigre Silver-Gold Project, Sonora, Mexico", with an effective date of October 22, 2024, hereafter referred to as the "Planned OP". All development was planned to avoid the Planned OP, with the exception of Drift-and-Fill ("D&F") pillars, which were developed into the Planned OP to a length sufficient to prevent any draw-down of HMW material in the UG from destabilizing the floor of potential operations in the Planned OP. Inevitably, some mineralized material is intersected by the D&F pillars that is currently inside the Planned OP mining areas. Total impact on the Planned OP in-situ material is approximately 100 kt of material containing ~0.9 koz Au and ~178 koz Ag, and essentially zero contained base metals (Cu, Pb, Zn).

Significant additional development, both vertical and lateral, has been included in the UG mine plan relative to that which would have been required if the Planned OP were completed prior to,

or after the end of, the planned UG Mine. The Authors recommend evaluating a combined OP/UG mine plan to provide a more optimized solution for the El Tigre Property as a whole. At present, the UG design described in this PEA can functionally "bolt-on" to the Planned OP at any point in time with virtually no impact to the Planned OP or UG operations and schedules beyond what is described in the previous paragraph.

Backfill and Artificial Pillars

To minimize surface tailings storage requirements and maximize UG mining extraction, cemented PF is planned to be used in both LH and C&F mining. Three recipes are expected to be utilized, with strengths of 0.5, 1.0 and 1.4 MPa. For overhand mining, 0.5 MPa PF provides sufficient strength. For underhand mining, if the span exceeds 10 m, 1.4 MPa strength is required, otherwise 1.0 MPa will suffice. The planned utilization of the three recipes, by volume backfilled, is approximately 45/42/13 percent of backfilled voids. PF also provides additional benefits to the Project in terms of its ability to better seal connections between HMW development workings and stopes than rockfill or cemented rockfill, limiting water inflow into stopes below and reducing ventilation short-circuiting. Additionally, it can be placed in stopes where no overcut access is available via drill holes.

Artificial sill pillars in stopes will be used where a continuous group of stopes in a Zone needs to be segregated into two vertically stacked groups for the purpose of scheduling (increasing the availability of mining faces). These pillars will utilize PF of a minimum strength of 1.0 MPa to allow eventual underhand mining.

As mentioned earlier, artificial sill pillars will also be installed using D&F operations to prevent the drawdown of HMW backfill at UG drawpoints from presenting a risk to any active OP mining operations in the Planned OP, and limit the amount of HMW backfill excavated by the UG mine. A C&F access will be driven approximately perpendicularly to the strike of the HMW voids on certain levels and sub-levels to intersect the HMW stope excavation, and then D&F development will be driven along the strike of the HMW stope, keeping the stope in the HW corner of the back of the drift as much as possible. Extensive ground support, inclusive of spiling/forepoling and in-cycle shotcreting will be required, in addition to very short rounds (nominally 1.5-1.8 m) during this process. This development will continue until it reaches a designated point coincident with a 40° angle of repose to:

- The maximum lateral extent of stopes on levels below that are connected to the HMW void; or
- Another D&F pillar located deeper in the mine; or
- The end of the HMW void.

Once this process is complete, the D&F excavation is backfilled with high-strength (1.0 MPa) PF and, after the PF has cured, the floor of the C&F access is slashed out to drive a ramp down to 4 m below the previous floor, and the process is repeated to create a D&F pillar of 8 m H x 4 m W (2:1 ratio). This second cut should require reduced ground support and spiling/forepoling versus the initial cut, however, it has been scheduled and costed in an identical manner.

Figure 24.13 to Figure 24.15 show the D&F pillars and their locations in the mine relative to stopes, development, and HMW extents. D&F pillars are located predominately on the El Tigre Vein in Zones 8 and 12, and also on the Seitz-Kelly Vein in Zone 14. No D&F pillars are located on the Sooy Vein.

FIGURE 24.13 LONGITUDINAL PROJECTION OF EL TIGRE VEIN MINE PLAN DEVELOPMENT, STOPES, D&F PILLARS, AND HMW STOPES

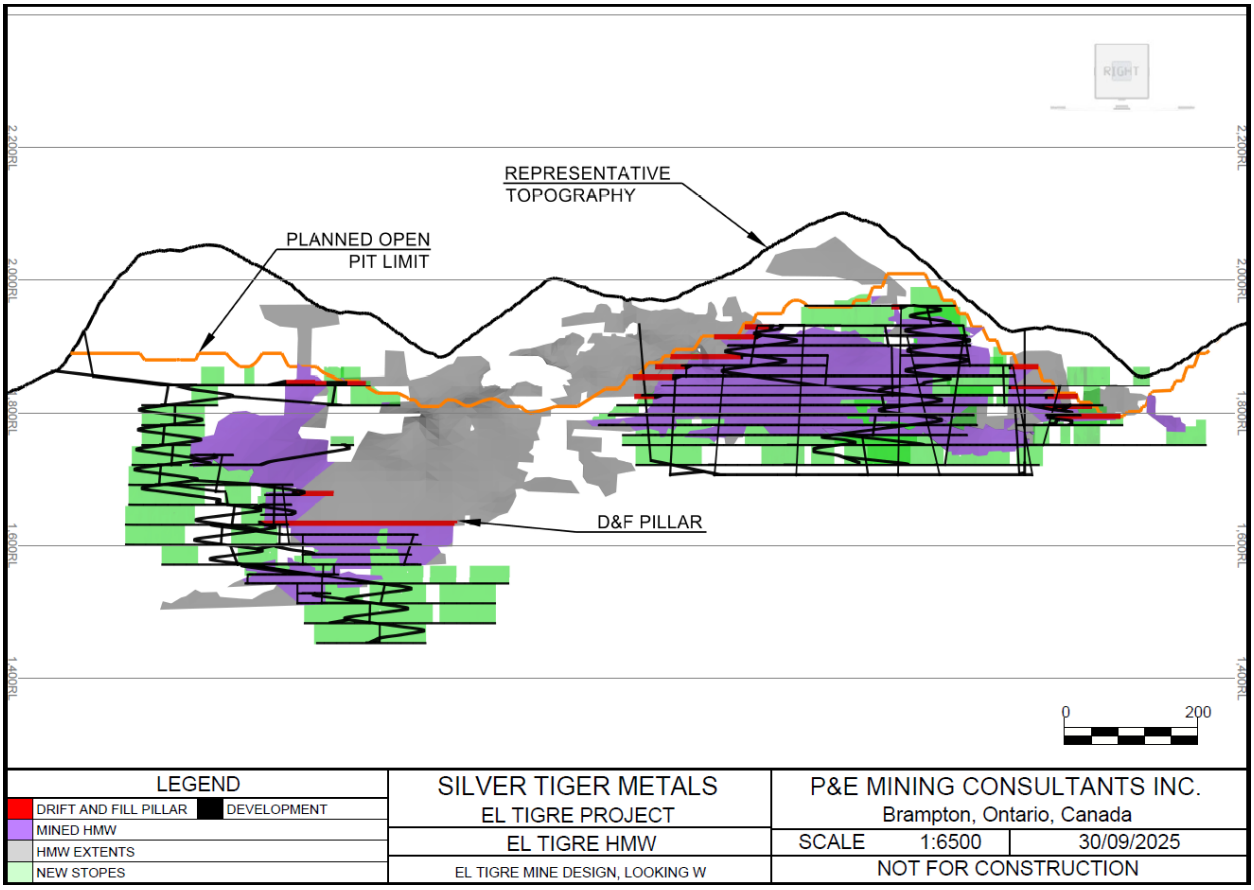


FIGURE 24.14 LONGITUDINAL PROJECTION OF SEITZ-KELLY VEIN MINE PLAN DEVELOPMENT, STOPES, D&F PILLARS, AND HMW STOPES

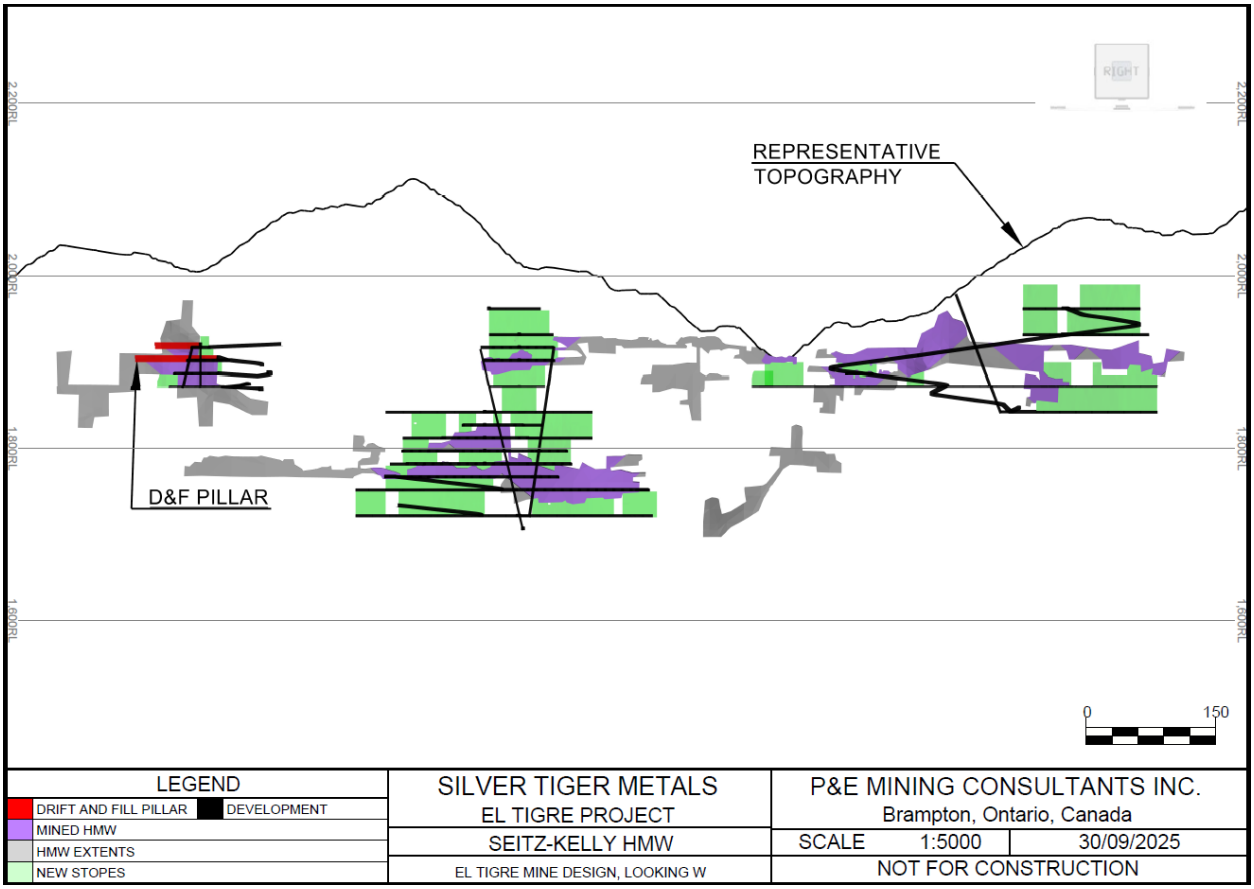
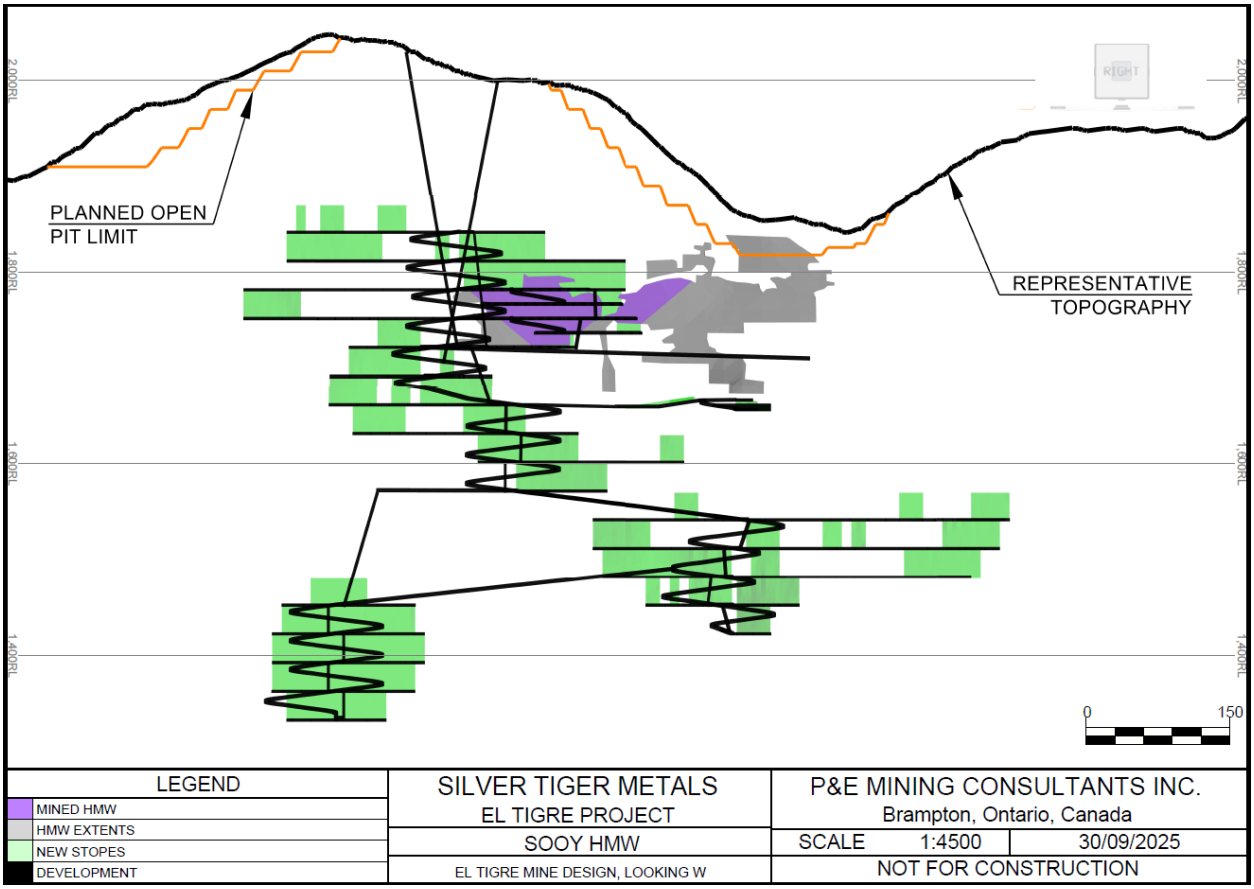


FIGURE 24.15 LONGITUDINAL PROJECTION OF SOOY VEIN MINE PLAN DEVELOPMENT, STOPES AND HMW STOPES



The Authors recommend a thorough geotechnical evaluation of the stability of the D&F pillars in a future study, and also the evaluation of the impact of cable-bolting the walls of the D&F drifts prior to backfilling.

Modularity and Capital Development

The El Tigre UG Project is a brownfields site, with HMW extending through the majority of mining horizons; being contiguous with some of the highest-grade stopes in the mine plan; and requiring significant development to access and recover these stopes. In addition to this, the requirement to avoid the Planned OP extents also places constraints on the positioning of capital development and requires the installation of expensive artificial sill pillars to ensure that the UG operations have no impact on the Planned OP operations.

Initial UG mine designs attempted to combine stopes on each vein into a continuous mining area and then combine those areas into a single UG mine with a single access, however, this resulted in sub-optimal economics and an extended pre-production development period. Therefore, the stopes were grouped into Zones from 1-16, and development was designed such that each zone would be as independent of the sequencing of other zones as reasonably possible. As a result, the functionality of portions of capital development is duplicated between some adjacent/contiguous

Zones, particularly in the ventilation and materials handling systems. Zones scheduled later in mine life are planned to "bolt on" as additional modules to Zones scheduled earlier in mine life, without the earlier Zone having its development increased in order to reduce the amount of development incurred by the later Zone.

This modularity was not tested for economic efficiency after the schedule was completed: it is likely that certain portions of capital development can be shared between Zones effectively. The Authors recommend a review of the schedule and design to see where development that duplicates functionality between two adjacent Zones can be combined into a more efficient combined design, however, the total impact will be minor and well within the expected accuracy of this PEA.

The Authors, for additional clarity, note that Zone 10 was eliminated after becoming uneconomic as a result of changes to the AgEq formula during the study period. Zones 11-16 were not renamed afterwards to account for this. Zone 10 does not appear in any images, schedules, or quantities in this PEA.

24.3.4.5 Cut-off Grades

AgEq Formulas

The El Tigre Deposit contains appreciable quantities of Ag, Au, Cu, Pb and Zn. Silver equivalent ("AgEq") was used to consolidate the values provided from each metal into a single equivalent grade value. This formula was revised several times throughout the 2025 PEA since the planned recovery method changed from a flotation concentrate to leach cyanidation.

Original

At the start of the PEA (April 16, 2025), the AgEq formula for the study was:

$$Orig_AgEq_{gpt} = Ag_{gpt} + Au_{gpt} * 82.028 + Cu_{pct} * 95.127 + Pb_{pct} * 24.266 + Zn_{pct} * 27.386$$

The components of the formula are shown in Table 24.15.

TABLE 24.15
ORIGINAL AGEQ FORMULA COMPONENTS

Commodity	Ag	Au	Cu	Pb	Zn
Price	$28.00/oz	$2,400/oz	$4.25/lb	$0.97/lb	$1.30/lb
Recovery (%)	93	89	85	95	80
Unit Value[1]	0.837	68.674	79.641	20.315	22.928
Formula Factor	1.0	82.028	95.127	24.266	27.386

Note: *[1] Precious metal unit value in $/g, Base metal unit value in $/%*

This formula was used during the initial runs of Deswik Stope Optimizer to generate the mining shapes for the mine plan, with the value of a gram of AgEq set at US$0.837.

Revised

On August 8, 2025, process recovery data for the planned flotation circuit was received that significantly revised the AgEq formula to the following:

$$Rev_AgEq_{gpt} = Ag_{gpt} + Au_{gpt} * 66.645 + Cu_{pct} * 25.417 + Pb_{pct} * 24.604 + Zn_{pct} * 3.915$$

The components of the formula are shown in Table 24.16.

TABLE 24.16
REVISED AGEQ FORMULA COMPONENTS

Commodity	Ag	Au	Cu	Pb	Zn
Price	$30.00/oz	$2,500/oz	$4.00/lb	$1.00/lb	$1.30/lb
Recovery (%)	75.9	60.7	21.1	81.7	10.0
Unit Value[1]	0.732	48.789	18.607	18.012	2.866
Formula Factor	1.0	66.645	25.417	24.604	3.915

Note: *[1] Precious metal unit value in $/g, Base metal unit value in $/%*

This new data indicated that the recovery of base metals was deleterious to the recovery of precious metals, and the plan for the processing circuit was revised to increase precious metals recovery at the expense of base metals recovery. Therefore, cyanidation leach was introduced, resulting in another revision to the AgEq formula. While the cyanidation tests were being performed, however, the UG plan was revised to eliminate stopes that were made uneconomic under the revised AgEq formula and with the unit value of AgEq set at $0.732 per gram. At this point in the study, Zone 10 was eliminated for being uneconomic.

Final

On August 28, 2025, new recovery values from a revised recovery process were received that resulted in a change to the AgEq formula:

$$Final_AgEq_{gpt} = Ag_{gpt} + Au_{gpt} * 88.597$$

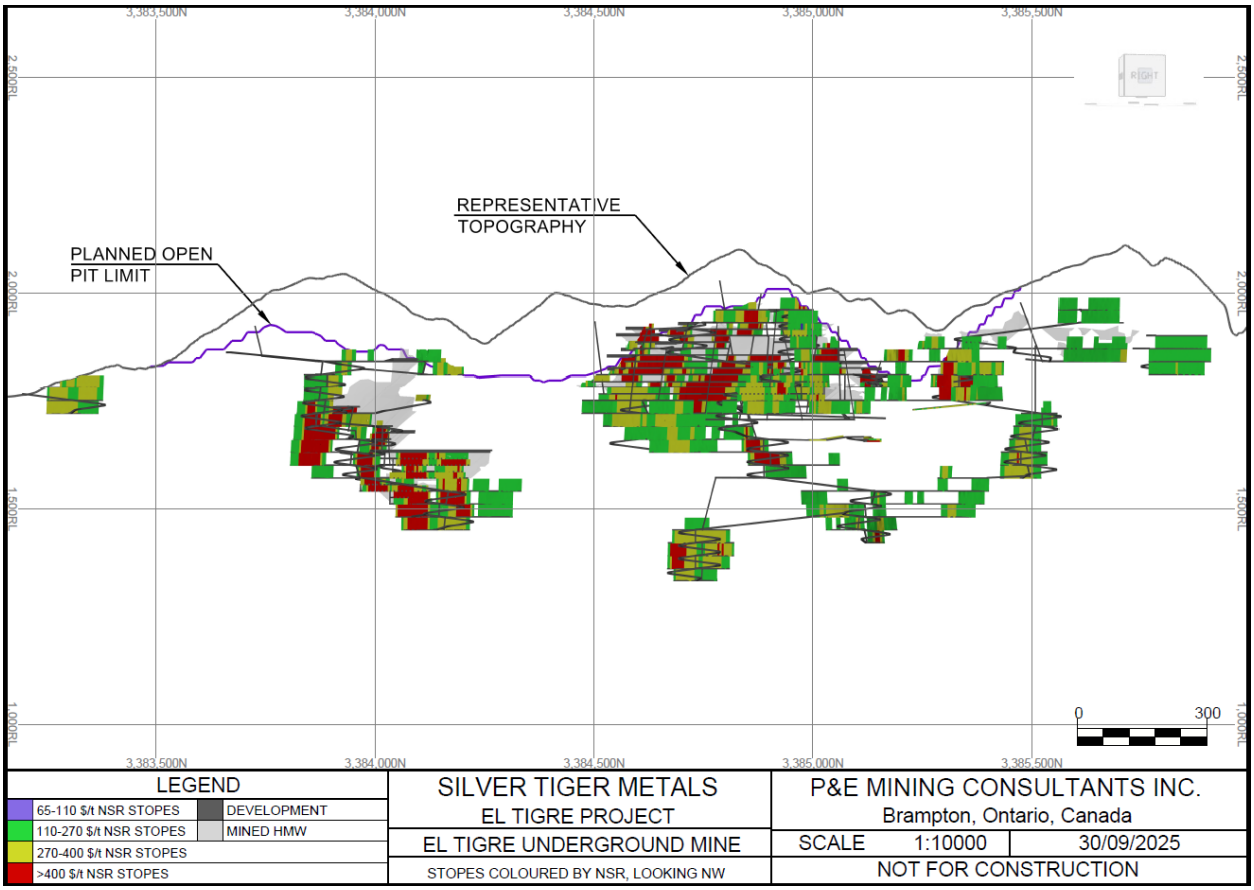
The components of the formula are shown in Table 24.17.

TABLE 24.17 FINAL AGEQ FORMULA COMPONENTS					
Commodity	**Ag**	**Au**	**Cu**	**Pb**	**Zn**
Price	$30.00/oz	$2,500/oz	$4.00/lb	$1.00/lb	$1.30/lb
Recovery (%)	84.38	89.71	0.0	0.0	0.0
Unit Value[1]	0.814	72.106	-	-	-
Formula Factor	1.0	88.597	-	-	-

Note: *[1] Precious metal unit value in $/g*

Based on the above factors, the mining shapes were re-evaluated for inclusion in the plan using a value of AgEq set at $0.814 US$ and the final AgEq formula. Due to the change from flotation to cyanidation leach, base metals have no contribution to the AgEq formula or revenue streams. Zone 10 was re-evaluated for economics at this point, was found to be marginally uneconomic, and was dropped from the plan. Figure 24.16 shows the UG mine stopes by NSR value, where NSR is equal to the AgEq value multiplied by the unit value of AgEq ($0.814/g).

FIGURE 24.16 LONGITUDINAL PROJECTION OF UG STOPES BY NSR



Cut-offs

Three different Cut-offs Grades ("COGs") were used for the mine plan: development, marginal and economic. Due to the changes to the AgEq formula, the cut-off values were revised between the initial generation of stope shapes (April 16, 2025 formula) and the final scheduled stope shapes (August 28, 2025 formula), however, the shapes themselves were not revised. The Authors recommend re-running the Deswik Stope Optimizer ("DSO") models under the revised AgEq conditions to evaluate whether the generated shapes are different than what was originally created. While the expectation is that the shapes will be different, it is the Authors' opinion that the impact on the mine plan would be within acceptable tolerances for a PEA.

Table 24.18 shows the initial values for the COGs, and Table 24.19 shows the final COGs, that were applied to filter out stopes whose economic status was sufficiently impacted by the changes to recoveries as to make them uneconomic under the final conditions.

TABLE 24.18
INITIAL CUT-OFFS USED FOR STOPE GENERATION

AgEq Value ($/g)		Cut-off Type	DCOG	MCOG	ECOG
		Used (AgEq g/t)	65	150	200
0.837		NSR ($/t)	52.39	118.16	159.49
		Calc (AgEq g/t)	64	145	196
Cost Value ($/t)	**AgEq Value ($/g)**	**Cost Item**	**DCOG Multiplier**	**MCOG Multiplier**	**ECOG Multiplier**
42.50	51	Capital Dev't	0	0	1
25.25	30	Operating Dev't	0	1	1
5.00	6	Production	1	1	1
16.50	20	Haulage	0	1	1
10.75	13	Backfill	0	1	1
9.75	12	Indirects	0	1	1
32.75	39	Processing	1	1	1
21.50	26	G&A	0.75	1	1

Note: *DCOG = development cut-off grade, MCOG = marginal cut-off grade, ECOG = economic cut-off grade.*

TABLE 24.19 FINAL CUT-OFFS USED FOR STOPE SELECTION					
AgEq Value ($/g)		Cut-off Type	DCOG	MCOG	ECOG
		Used (NSR $/t)	65.00	135.00	170.00
0.814		Calc (NSR $/t)	65.44	135.50	172.00
		AgEq (g/t)	80	166	211
Cost Value ($/t)	**AgEq Value ($/g)**	**Cost Item**	**DCOG Multiplier**	**MCOG Multiplier**	**ECOG Multiplier**
36.50	45	Capital Dev't	0	0	1
24.50	30	Operating Dev't	0	1	1
5.00	6	Production	1	1	1
16.50	20	Haulage	0	1	1
8.50	10	Backfill	0	1	1
14.75	18	Indirects	0	1	1
43.00	53	Processing	1	1	1
23.25	29	G&A	0.75	1	1

Note: *DCOG = development cut-off grade, MCOG = marginal cut-off grade, ECOG = economic cut-off grade.*

In both cases, the 0.75 multiplier for G&A in the DCOG calculation arises from the fact that the G&A load on these tonnes is lower since the portion of G&A associated with running the UG mine does not need to be incurred since these tonnes are being taken from a surface stockpile into the process feed.

The actual cut-off value used in initial stope generation was rounded up to the nearest 5 g/t AgEq from the true calculated value, and was rounded to the nearest $5.00 NSR/t from the calculated value for the final cut-offs.

Development

The DCOG is utilized to determine which development, that would otherwise be classified as waste, contains sufficient mineralization to justify stockpiling and processing at the end of LOM. This material needs to cover the costs of handling to surface, stockpiling and rehandling, processing and tailings, and G&A. The final DCOG for the stopes in the mine plan was $65/t NSR.

A very small portion of production stoping areas were included to infill isolated holes at a cut-off value of $110/t NSR where both overcut and undercut development costs were already sunk. These tonnes make up less than 1% of the total stope tonnes.

Marginal

The MCOG is utilized to determine which stoping areas have sufficient grade to cover all direct operating costs of their inclusion in the mine plan (operating development, production and backfill,

processing and tailings, and G&A). The final MCOG for the stopes in the mine plan was $135/t NSR.

Economic

Stopes exceeding ECOG cover all average operating and capital costs for the UG mine (total cost divided by total UG mine plan tonnes). The final ECOG for the stopes in the mine plan was $170/t NSR.

24.3.4.6 Development

Table 24.20 shows lateral and vertical development totals by size and category. Actual excavated profiles may vary slightly from nominal profiles in certain cases, such as:

- Remuck bays or mineral pass loadouts will have a slightly higher back;
- Ventilation drifts may be driven larger where it is necessary for trucks to transit them during initial development; and
- The dimensions of C&F accesses will vary dependent on the mineralization, with a minimum of 3.5 m H for equipment transit.

TABLE 24.20
DEVELOPMENT SUMMARY

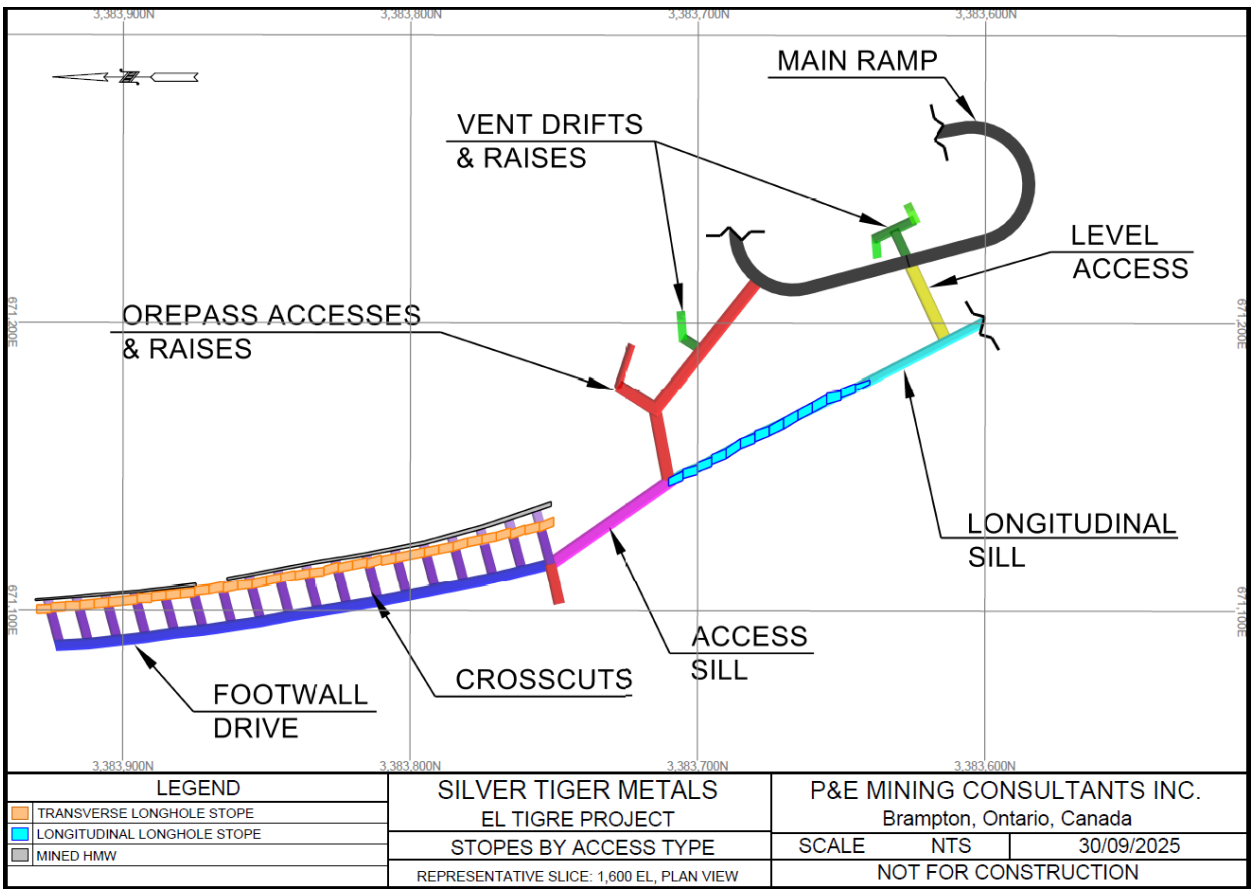
Development Type	Nominal Profile	Capital / Operating and Type	Linear Quantiy (Lin. M)
Main Ramp	4.5 m W x 4.5 m H Arched	Capital, Lateral	16,617
Level Access			4,706
Sublevel Ramp			1,860
Mineral Pass Transfer			781
Remuck Bay			2,168
Vent Access	4.0 m W x 4.0 m H Arched		3,434
Footwall Drift			9,457
Electrical Bay	3.5 m W x 3.5 m H Arched		1,412
Sump/Pump Station			1,638
Mineral Pass Access			1,232
Mineralized or Waste Sills		Operating, Lateral	15,862
Mineralized or Waste Crosscuts			12,796
Pastefill Crosscut			343
Cut and Fill Access	4.0 m W x 4.0 m H Square		1,231
Drift and Fill Access			2,645
Subtotal (Lateral)[1]	**All**	**Lateral**	**76,182**
Mineral Pass Raisebore	2.7 m Ø	Vertical, Capital	1,550

TABLE 24.20 DEVELOPMENT SUMMARY			
Development Type	**Nominal Profile**	**Capital / Operating and Type**	**Linear Quantiy (Lin. M)**
Ventilation Raisebore	3.4 m Ø		2,697
Ventilation Drop Raise	3.0 m W x 3.0 m L		1,348
Subtotal (Vertical) [1]	**All**	**Vertical**	**5,595**

Note: *[1] Totals may not sum due to rounding.*

Figure 24.17 shows a plan view of Area C at 1,600 EL where both longitudinal and transverse mining take place (at separate times), and materials handling and ventilation infrastructure are installed.

FIGURE 24.17 PLAN VIEW OF DEVELOPMENT AT 1,600 EL IN AREA C



Lateral

Lateral development is sized to accommodate the largest piece of equipment expected to operate within the mining area. In general, the LH drills drive the limits for operating development (clearance around the mast during dip/dump and ring drilling), while haul trucks drive the limits

for most capital development, particularly the ramp. Other factors, such as ventilation requirements or speciality services (e.g. mineral pass loadouts or remuck bays) are also taken into account when sizing lateral development.

A total of 76.2 linear km of lateral development and 5.6 linear km of vertical development is planned over LOM. Approximately 43% of lateral development is operating development.

HMW development workings unavoidably intersect certain lateral portions of the new underground workings (new vertical development avoids them entirely) although efforts have been made to avoid these intersections whenever possible. These HMW areas may be open or may be collapsed. Where feasible, the HMW openings can be used as the cut for new development operations, however, the majority of intersections will be perpendicular to strike in transverse access areas (HMW development utilized longitudinal access). No contingency has been added to development quantities to account for these areas. HMW openings are generally smaller than new development openings, so standard bolting and screening practices are expected to be sufficient for ground support requirements.

Vertical

Vertical development in the UG mine is utilized for ventilation raises and mineral passes. Mineral passes are expected to be entirely raisebored by specialist contractors, while ventilation raises are designed as approximately 33% drop raise and 67% raisebore. Escapeways are fitted in ventilation raises as necessary (see Section 24.3.4.10). It is not anticipated that separate escapeway raises will be required.

Raiseboring is used where raise lengths are longer than 40 m. Due to the relatively short level interval, raiseboring is utilized to drive a single raise segment connecting multiple consecutive levels with minimal lateral offset (raisebored raises are designed to have a dip of 70° or higher), whereas drop raises are used to connect two consecutive levels where lateral offsets between levels make raiseboring inefficient.

It is feasible to drive long raises in the UG with either raisebore or Alimak methods. Alimak methods have the advantage of not requiring an overcut to begin development, however, suffer from declining production at the ends of long raises due to the transit time of the deck. Scheduling for the Project has assumed raiseboring from an undercut level to an overcut level.

24.3.4.7 Production

Production operations in the El Tigre UG mine are virtually entirely LH stoping utilizing either transverse or longitudinal access. A small minority (~1%) of production tonnes are extracted utilize C&F methods. LH mining utilizes a combination of top-hammer and In-The-Hole ("ITH") hammer drills. Top-hammer drills are generally used for uphole drilling in transverse mining areas with short level intervals, while ITH drills are used for downhole or uphole drilling in longitudinal access areas and in raises. ITH drills can also be fitted with a large diameter (~0.76 m Ø) reaming head for slot raising in transverse access areas. Approximately 40% of all LH drilling (both drop raises and production) utilizes downhole drilling.

Longhole

Transverse Access

Transverse access mining involves driving an access drift offset from the mineralized vein (in this case, nominally 12-15 m from the contact) and parallel to strike, and then driving perpendicular crosscuts across the vein on a set spacing (in this case, nominally 10 m centres). For the El Tigre UG Project, stopes are drilled using upholes only due to access limitations (the overcut level will be completely backfilled prior to mining). Mining of individual stopes progresses similarly to longitudinal retreat mining, however, where multiple stopes exist on a single crosscut around HMW areas, the sequencing of multiple stopes on a single crosscut can be more complex. Backfilling will generally be from the crosscuts on the level above, or through drill holes from nearby development. Transverse mining maximizes productivity at the cost of additional development.

Generally, for transverse access mining, the optimal sequence of stopes on a single crosscut retreats from HW to FW. Due to the positioning of the mineralized veins and the added complexity of the HMW areas, the sequence is often reversed, progressing from FW to HW and driving through engineered backfill. As the backfill is free-flowing, any attempt to excavate in an overhand fashion would require the extraction of all backfill from the levels above prior to commencement of mining the in-situ mineralization, which could destabilize the HMW voids and result in negative geotechnical and economic consequences. Therefore, underhand mining was selected for stopes adjacent to HMW areas. To safely and economically extract these areas, the following process is used where mineralization exists on both sides of the HMW area, and the PF in the level above has cured to sufficient strength for undermining:

1. Drive a drift parallel to the vein, offset by 12 to 15 m to the nearest stope;

2. Drive crosscuts (roughly perpendicular to the vein or HMW void) on 10 m spacings to just before the HMW contact, using probe drilling to determine the breakthrough location;

3. Prior to breakthrough, drill the stope rings in virgin ground adjacent to the HMW stope. This prevents drilling near open brows;

4. Break through development to the HMW area and draw down the backfill until the brow cracks. Ram the brow closed with a load-haul-dump ("LHD") truck using a "rammer-jammer";

5. Load and prime the drilled holes, pushing the primers into the collars and temporarily plugging them with bottle brushes or cones;

6. Utilizing an LHD on remote control, excavate the brow until backfill reaches the floor level. In some instances, drawing down the backfill may be done from the level below if desired;

7. Blast the stope, using the HMW void as a slot. If void is insufficient to fire the entire stope, excavate the blasted material until the brow cracks, and ram the brow closed. Repeat from Step 5 until the entire stope is blasted and excavated;

8. Install a backfill barricade in the development access;

9. Backfill stope with high-strength PF from the level above and allow to cure required strength;

10. Excavate through cured PF; and

11. Extract mineralization on far side of now-filled HMW void as per normal transverse access stoping practices once the near-side stope has cured to full strength.

Where the HMW void is located on the near side of the in-situ mineralization only, the above sequence is followed with the exception that no drilling and blasting is required. Filling begins after the HMW void is drawn down to the floor of the sill and a barricade is placed.

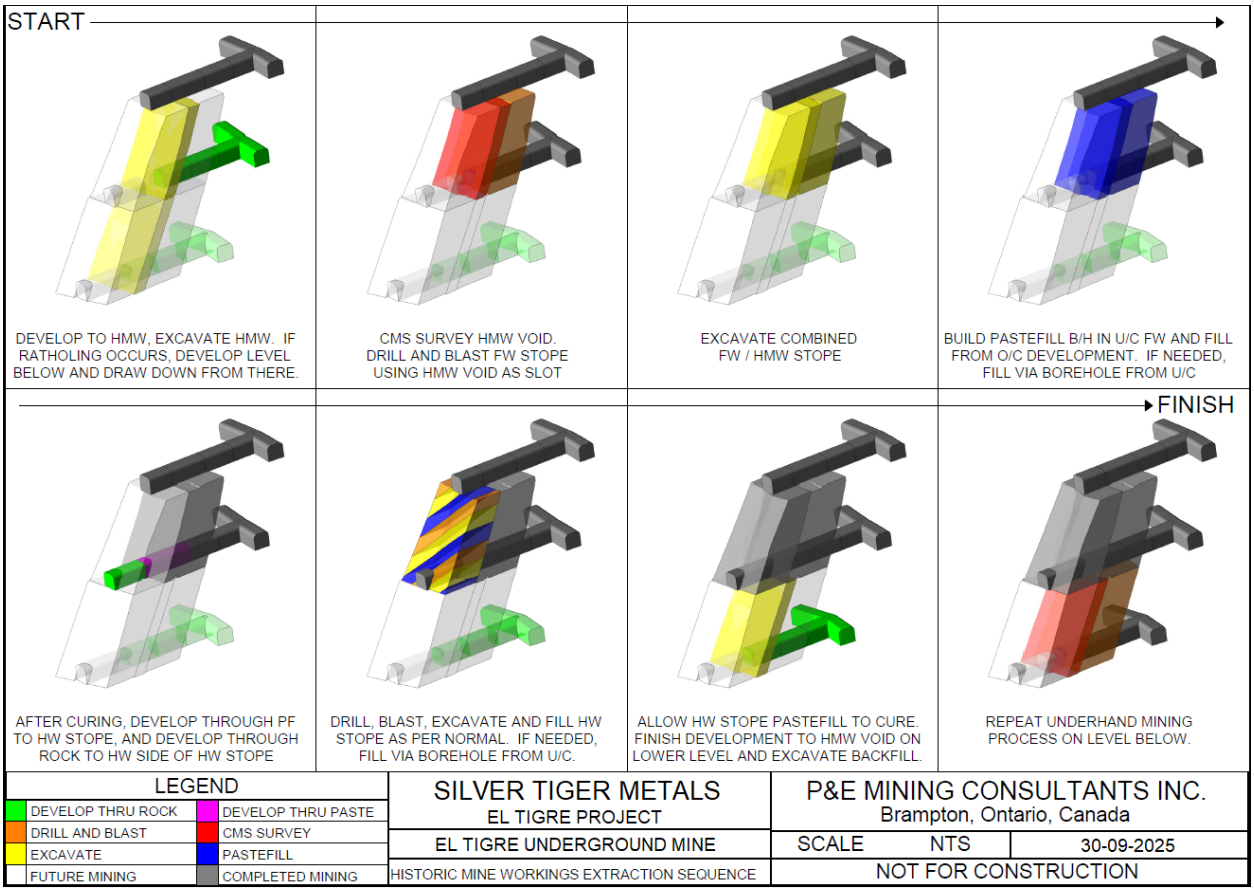
Where the HMW void is located solely on the far side of the in-situ mineralization, the above sequence stops at item 9, and the PF is allowed to cure to full strength for eventual underhand mining on the level below.

It is expected that some PF introduced into stope voids around HMW areas on primary crosscuts will run or be blasted into adjacent HMW voids on secondary crosscuts and increase dilution therein. The average dilution factor applied to all underhand mining stopes accounts for this.

Figure 24.18 shows the extraction sequence for a crosscut that intersects HMW voids.

Alternating crosscuts are extracted in a primary-secondary sequence during transverse access mining. Stopes on primary (odd-numbered) crosscuts are excavated and filled first, prior to stopes on secondary (even-numbered) crosscuts being excavated. This limits sequencing issues and exposure to open stopes in adjacent crosscuts.

FIGURE 24.18 HMW EXTRACTION SEQUENCE



Longitudinal Retreat

Longitudinal retreat mining involves driving an access through mineralization along the strike of the vein to the end of a group of stopes. The stopes are drilled using downholes (or upholes where no top access exists), loaded and blasted, and mineralization is recovered from the bottom access using an LHD. Once the blasted material is excavated and survey is complete, a wall is built to seal the bottom access of the stope and the stope is then filled with PF. After a curing period (nominally 28 days), the cycle repeats on the next stope in the sequence (drilling and loading operations can commence prior to complete curing, however, blasting cannot). Longitudinal retreat mining progresses from the bottom of a mining block to the top, working on top of PF. Backfilling can either be from the sill drift above or through a drill hole from nearby development. Drill holes will be the sole option for the last level in each mining block since there will not be an open sill drift above. Longitudinal retreat mining minimizes waste development by avoiding offset drifts and crosscuts, at the cost of reduced mining faces on a level.

In rare cases (Zone 6 and Zone 15), longitudinal retreat stopes are planned that abut the ends of HMW voids. Development for these stopes will not intersect the HMW voids: only a few production holes will be drilled to breakthrough. In these isolated cases, the expected angle of repose of backfill (40° from horizontal) intersects the extents of the HMW void near to the stope.

While minimal additional dilution is expected, additional dilution has been added to the few stopes where this situation occurs.

Cut and Fill

C&F mining is analogous to development mining with PF backfill. Mining is full-face, with no segregation of mineralization and waste within the face (the grade is diluted over the entire face height). C&F mining occurs on the Black Shale ("BS") Formation, which intersects all the major vein sets, that is the primary host for mining targets only in Zone 2 (where it intersects El Tigre) and Zone 4 (where it intersects Sooy). In Zone 2 the BS Formation is approximately 2 m thick, and in Zone 4 it can reach thicknesses of 10 m. As a result, C&F mining varies in height between Zone 2 and Zone 4, with Zone 2 being at the minimum height (3.5 m H) and Zone 4 being at a slightly larger height (4.0 m H) where the mineralization is thicker. Zone 4 utilizes an overhand mining progression where, once the bottom cut is complete, a second 4 m H sublevel is excavated on top of the fill in the same manner as the bottom cut.

C&F mining in both Zones utilizes a face that is 4.0 m W. Full-face mining with the associated dilution was considered acceptable due to the very small portion of the overall Mineral Resource being recovered in this manner, and the desire to utilize the same types of equipment for C&F production as for development operations. The Authors recommend an evaluation of manual C&F methods with reduced back heights in Zone 2 and reduced dilution at a later stage of study. It is likely practical to vary face heights in Zone 4 slightly to improve recoveries and reduce dilution versus the existing mine plan.

All C&F openings are planned to be backfilled with PF at a strength of 0.5 MPa.

Extraction of Historical Mine Workings

A total of 437 kt of HMW stope material is planned to be excavated over the LOM, based on an assumed loose density of 1.85 t/m^3 derived from a 2.5 t/m^3 in-situ density and a swell factor of 35%. This material is not included in the potentially mineable portion of the Mineral Resource due to insufficient sampling density. The material is assumed to be free-flowing and not require blasting, and will require backfilling with PF. It is planned to be stockpiled separately from development waste for eventual testing for mineralization to determine the economics of eventual processing. The distribution of these HMW tonnes by Zone is shown in Table 24.21.

TABLE 24.21
HMW STOPE MATERIAL BY ZONE

Zone	Tonnes (k)
3	29.8
6	1.1
8	167.2
12	158.7
13	56.8

TABLE 24.21 HMW STOPE MATERIAL BY ZONE	
Zone	**Tonnes (k)**
14	5.7
15	3.8
16	14.1
Total	**437.1**

Stope Sizing

WSP analyzed the various formations in the Deposit and generated a table of LH stope spans with and without HW support, as shown in Table 24.22.

TABLE 24.22 STOPE SIZING RECOMMENDATIONS			
Formation	**Strike Length Without HW Support (m)**	**Strike Length With HW Support (m)**	**Height (m)**
El Tigre Formation First 50 m Depth	15	20	30
El Tigre Formation Deeper than 50 m	20	30	30
Nodular Formation Down to 500 m Depth	30	40	30
Nodular Formation Below 500 m Depth	25	35	30
Basement Formation	30	40	30
All Other Formations Down to 500 m Depth[1]	25	35	30
All Other Formations Below 500 m Depth[1]	20	30	30
Fault Zones / Damaged Vein Areas	15	20	30

Note: *[1] "All other formations" is defined by the Cliff Formation, Flat Formation (includes upper, lower, and black shale), and the Tabular Formation.*

Stope spans of up to 15 m were examined and found to produce stable (unsupported) end walls and stable backs (considering a supported top sill drift). The stope dip was examined down to 70°, the approximate minimum dip of the Deposit, and to a depth of 800 m (maximum planned mining depth below surface is 760 m in Zone 5, with the majority of mining planned for less than 500 m below surface).

Where necessary, 15.2 mm diameter cablebolts will be installed in the overcut and undercut of longitudinal stopes. Preliminary designs suggest that fans of four cables spaced on 2 m centres should be sufficient. To maintain consistent embedment depth, it is expected that fans will comprise two 6 m and two 9 m cables.

Designed stope thicknesses vary by location, as shown in Table 24.23. Functionally all stopes (99.9% of stope tonnes) are less than 30 m H. Transverse stopes, with minor exceptions, are

15 m H and 10 m W. LH stope lengths will vary based on location and are generally assumed to be 25 m L at a maximum.

TABLE 24.23
STOPE THICKNESS BY ZONE AND TYPE

Zone	Longitudinal (m)	Transverse (m)	HMW (m)
1	3.05	-	-
2	4.13	4.72	-
3	3.05	2.56	0.74
4	6.15	-	-
5	2.78	-	-
6	3.25	-	1.88
7	2.95	-	-
8	3.80	3.00	0.86
9	3.06	-	-
11	3.15	-	-
12	4.60	6.97	1.08
13	3.10	2.63	1.23
14	9.02	10.53	0.98
15	6.55	-	0.78
16	3.56	2.61	0.33
Average[1]	**4.13**	**6.01**	**0.97**

Note: *[1] Weighted average based on tonnes*

Wider stopes are generally associated with stockwork mineralization, narrower stopes with veins.

Ground Support

WSP provided empirical benchmarked ground support guidance for the Project. Using NGI Empirical Support Guidelines (2015), the NGI-Q ground support plot assessment suggests most development would only require pattern bolting and mesh, with poor ground areas potentially requiring shotcrete. This analysis was further supported by using the Potvin Ground Support Guidance for mine drifts of 4-6 m Span (2016). Table 24.24 summarizes the guidance.

TABLE 24.24 GROUND SUPPORT GUIDANCE FROM WSP						
Excavation Type	**Rockmass Quality**	**Drift Span**	**Ground Support Element**	**Surface Support**	**Minimum Length (m)**	**Minimum Spacing (m)**
Permanent (CAPEX)	Poor (Q 1- 4)	4 - 4.5 m	20 mm (#6) resin rebar in back and shoulders. Split sets (SS-33 or similar) in side walls.	6 gauge, welded wire mesh	2.4	1.2
	Good (Q 10 - 40)					1.4
Temporary (OPEX)	Poor (Q 1- 4)	3.5 - 4.0 m			1.8	1.2
	Good (Q 10 - 40)					1.4

The Authors have assumed that capital development will utilize resin-grouted rebar on a nominal 1.2 x 1.2 m pattern with lengths of 1.8-2.4 m depending on profile. Operating development is expected to utilize hydraulically inflatable bolts (Swellex or similar) in place of rebar. Welded-wire mesh (6 gauge) is expected to be used in all areas, generally on the back and shoulders, and occasionally further down the walls in areas of poor ground. Fibre-reinforced shotcrete at a thickness of 50 mm will be used where necessary. Vertical development is generally unsupported except in locations where escapeways will be installed.

Backfill

WSP provided recommendations for cemented PF strengths for overhand and underhand mining as follows: overhand mining requires a minimum strength of 0.5 MPa, while underhand mining requires a strength of 1.0 MPa for spans up to 10 m, and 1.4 MPa for spans from 10-15 m. The average span of underhand LH stopes for the El Tigre UG is 6.0 m and for overhand LH stopes is 4.1 m. Some underhand stopes may be additionally extended in span due to the addition of voids from HMW areas. Over 99% HMW areas are less than 2.0 m in span (average width is less than 1.0 m), meaning that only transverse stopes adjacent to HMW areas, and with a span of 8 m or more, will require 1.4 MPa backfill, in addition to any longitudinal stope in excess of 10 m span. Throughout the mine, approximately 13% of LH stope tonnes will require backfill strengths of 1.4 MPa. Practically, this number will be slightly reduced, as some transverse stopes with large spans will be split into two stopes (HW and FW) on a single crosscut access.

No backfill recipes have been generated for the Project, nor have any studies on the suitability of the site tailings for a PF application. The Authors have assumed that the tailings from the process plant will be suitable for use in all UG PF applications, and that standard Portland (Grey) cement will be used as a binder. It as assumed that the recipes shown in Table 24.25 will be utilized.

TABLE 24.25 BACKFILL RECIPES AND QUANTITIES ESTIMATE					
Strength (MPa)	**% Solids**	**Density (t/m^3)**	**Binder by Mass (%)**	**Portion of PF (%)**	**Total m^3 of PF (k)**
0.5	72	1.85	3.0	45	868
1.0			5.4	42	807
1.4			8.0	13	255

Based on these recipes, total Portland cement demand is expected to be approximately 120 kt over LOM. PF is expected to consume 2.45 Mt of tailings over LOM. Additional voids exist from new and historical UG development that could theoretically be backfilled at end of LOM. These voids total approximately 0.5 Mm^3 and would consume an additional 635 kt of tailings if filled.

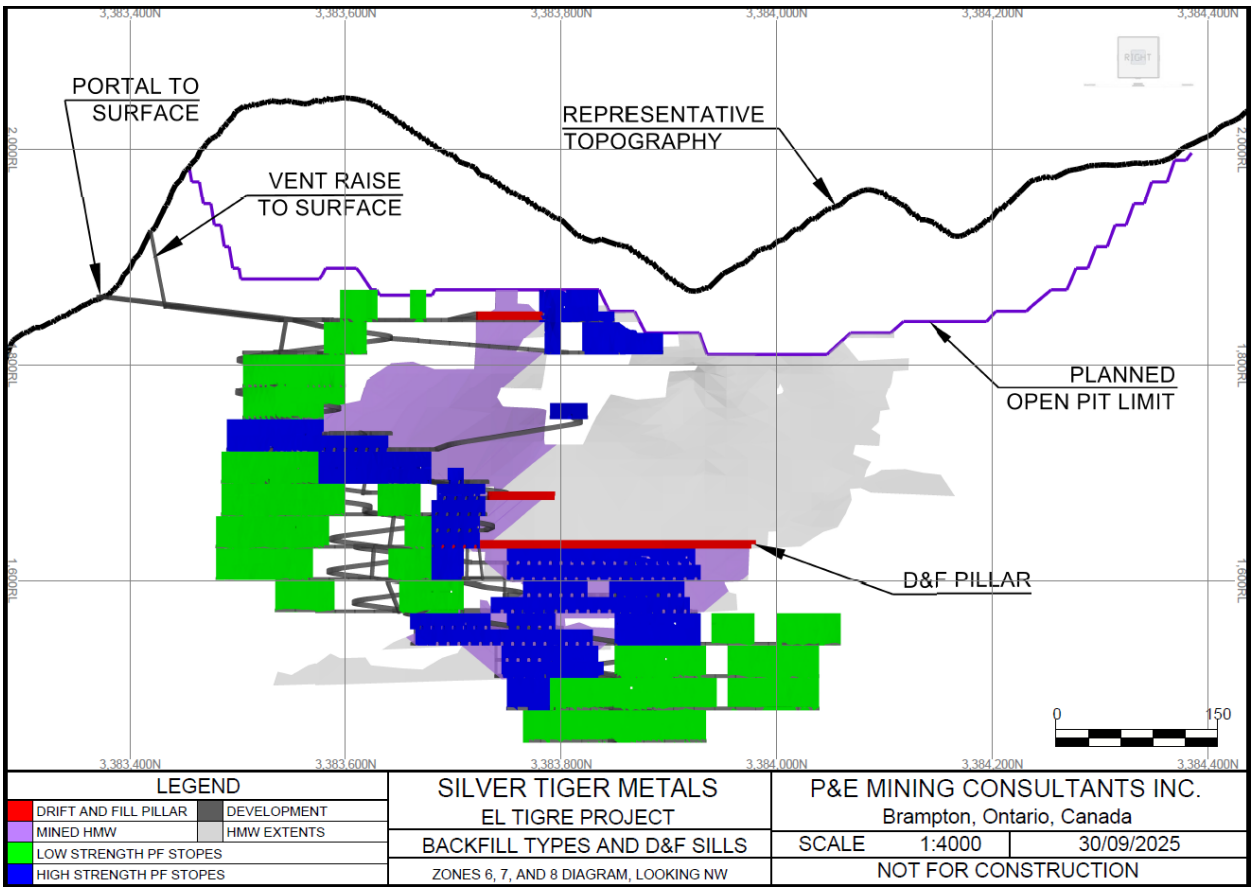
Pillars

Artificial Sill Pillars

A single artificial sill pillar is located in Zone 6 at 1,720 EL. This pillar is 30 m thick and allows for early start-up of the Zone in YR 1 before the ramp reaches the bottom of the Zone. Eventually, mining will progress from the bottom of the Zone up to the sill pillar, and stopes beneath the sill pillar will be excavated using upholes, since no overcut access will be available. This pillar will be filled entirely with 1.0 MPa PF since stope spans will not exceed 10 m in this area.

Figure 24.19 shows the backfill plan, by strength, of Area C (Zones 6-8), including the location of D&F pillars in Zone 8.

FIGURE 24.19 LONGITUDINAL PROJECTION OF AREA C STOPES BY BACKFILL STRENGTH, WITH D&F PILLAR LOCATIONS



Crown Pillars

Stopes within the Crown Pillar have been included in the mining plan. It is the opinion of the Authors that stopes within these areas can be recovered using upholes from the undercut, with backfill placed either from surface or pumped into the stope void from holes drilled from the undercut. Mining recoveries and dilution for these areas have been modified to account for additional difficulties in extraction and filling. Approximately 6.0% of stope tonnes are within these pillars. Of these, only 1.1% of total tonnes are affected by the topographical crown pillar, with the remainder being affected solely by the crown pillar around the Planned OP that would be required if the Planned OP was excavated prior to UG mining.

Stockpiling

Existing Stockpiles

Historical mining operations on the site have left two mineralized stockpiles that are included in the mine plan: the Low-Grade SP and the Tailings SP. The Low-Grade SP comprises approximately 119 kt of material historically excavated in the El Tigre UG Deposit, with grades slightly in excess of the UG MCOG. This material is expected to be recovered using standard

truck-and-shovel methods (front-end loader and highway haul trucks) and will be used for commissioning the process plant.

The Tailings SP comprises approximately 904 kt of previously processed material from historical operations, and is stored on surface approximately 0.5 km from the planned process plant. This material grades slightly above the UG DCOG, and will be re-processed primarily after the end of UG mining, although a small portion of this SP will be processed during commissioning (<20 kt). This material will be recovered in the same fashion as the Low-Grade SP. Due to the reduced demand on the crushing/grinding circuit, it is expected that this material, when it forms the only feed to the process plant, can be processed at 365 ktpa instead of the 290 ktpa throughput rate expected when normal crushing/grinding is required for fresh UG mineralization.

No base metals testing has been performed on the Tailings SP. Due to the selection of cyanidation for the planned process plant, any base metals would have no financial contribution to the Project regardless, and only the precious metals are relevant.

Planned Stockpiles

During UG mining operations, two new SPs will be created in addition to a small Run-of-Mine ("ROM") stockpile at the process plant near the primary crusher. A Development SP will be created to store material from development that exceeds DCOG but falls below MCOG. This material can be used to supplement process plant feed in the event of delays in UG production, or allow for batch processing of certain mineralization types, and is generally expected to be processed after the end of UG mining. This SP is expected to reach a size of 87 kt by the end of YR13, and be processed in YR14. Material from this SP will be recovered in the same fashion as the Low Grade and Tailings SPs, with no additional dilution or mining loss planned beyond those incurred during UG mining.

A second SP will be created to store material extracted from HMW stopes during UG mining. This SP is expected to reach a maximum size of 437 kt by end of LOM.

24.3.4.8 Dilution, Mining Loss and Recovery

Dilution describes the inclusion of sub-economic material in the mining process to recover mineralization. Internal dilution describes the necessary inclusion of sub-economic material within a designed blast shape. Internal dilution may contain some sub-economic mineralization. External dilution describes the unplanned addition of material from outside of the planned mining shape, either as a result of overbreak beyond planned limits, gouging of the floor during mining, or backfill from adjacent stopes or HMW areas running into the open stope. External dilution is assumed to be unmineralized.

Internal dilution for the El Tigre UG mine is calculated by dividing the total tonnes of material in planned stope shapes with an NSR$/t less than MCOG by the total tonnes of material in planned shapes. For development shapes, DCOG is used instead of MCOG. Average internal dilution on development is estimated at 56.3%, average internal dilution on stopes is estimated at 49.3%, and the average internal dilution on all mineralized material in the mine plan is estimated at 50.4%.

Mining loss is defined as a portion of mineralization left behind in an excavation due to any or all of drilling, blasting, excavating, or ground support issues. Recovery is defined as the ratio of mineralization recovered to mineralization planned (the inverse of mining loss). The average recovery on all mineralized tonnes in the mine plan is estimated at 93.8%: 99.0% on development and 93.0% on production. Mining loss is thus 6.2% on all mineralized tonnes: 1.0% on development and 7.0% on production.

Different external dilution factors and mining recovery factors are assigned to stopes based on various modifying factors, namely their proximity to HMW stopes or surface, whether or not there is overcut access, and the direction of drilling. Table 24.26 shows the combinations of mining dilution and mining recovery applied to shapes in the mine plan.

TABLE 24.26
EXTERNAL DILUTION AND MINING RECOVERY FACTORS FOR UNDERGROUND MINING

External Dil. (%)	Mining Rec. (%)	Applies To
0.0	99.0	Development
6.0	99.0	C&F Production
0.0	100.0	HMW stope backfill
3.0	95.0	LH Stoping – Downhole with overcut and undercut access in virgin ground
5.0	92.5	LH Stoping – Uphole with no overcut access or downhole in proximity to HMW stopes
10.0	90.0	LH Stoping – Uphole with no overcut access adjacent to HMW stopes on same level
15.0	85.0	LH Stoping – Uphole with no overcut access with multiple adjacent levels of HMW stopes

External dilution is calculated by taking the weighted average of the external dilution factors and recovered tonnes. The average external dilution on all mineralized tonnes in the mine plan is estimated at 4.8%: 0.0% on development (overbreak is assumed to break into mineralized stoping areas) and 5.6% on production.

For all stockpiling operations (Low-Grade SP, Tailings SP and Development SP), it is assumed that material on the SP can be recovered completely with no additional dilution (100% recovery and 0% dilution).

24.3.4.9 Equipment

As the Project is planned to be operated by contractors, the mining equipment fleet for the UG mine will be determined by the contractor's expected productivities. An estimate of the maximum steady-state quantity of units, and the class of those units, has been performed and is summarized in Table 24.27. Due to the peak in development in YR 1, it is expected that the spare units will be temporarily pressed into service for a short period, prior to being made redundant or used as spares later in mine life. Due to the multiple active mining Areas, the fleet is somewhat oversized for the

planned production rate, as equipment cannot readily be shared among mining Areas (sharing between Zones within an Area is possible), outside of: haul trucks, graders, light vehicles and fuel/lube trucks.

TABLE 24.27
UNDERGROUND MINING FLEET TYPES AND QUANTITIES

Type	Similar To	Quantity Required (incl Spares)
LHD - 10 tonne (Dev't)	Sandvik LH410	2
LHD - 7 tonne (Prod)	Sandvik LH307	3
Haul Truck - 30 tonne	Sandvik TH430	8
Jumbo - LH (Prod)	Sandvik DU311	4
Explosives Loader	MacLean AC3	7
Jumbo (Dev't / C&F)	Sandvik DD321	7
Bolter	Sandvik DS321	7
Scissor Deck (Dev't)	MacLean SL3	7
Shotcreter	MacLean SS3	3
Grader	CAT UG20M	2
Flatbed	MacLean DT3	3
Pipe Truck / Fan Handler	MacLean BT3	2
Telehandler	MacLean LR3	3
Scissor Deck (Constr.)	MacLean SL3	3
Forklift	MacLean FR3	4
Backhoe	CAT 432 Side Pivot	2
Fuel/Lube Truck	MacLean FT3/FL3	2
Light Vehicle	Toyota Hilux	7

Once initial development is complete, expected fleet utilization is approximately 60% for most units except those that can be shared between mining Areas.

24.3.4.10 Services and Infrastructure

Ventilation

Ventilation in the UG mining Zones at the El Tigre Project will utilize exhaust (pull) systems, with fresh air being drawn down ramps and Fresh Air Raises ("FARs"), onto active levels, across footwall drifts ("FWDs"), and into Return Air Raises ("RARs"). In the case of Zone 1 and Zone 9, all ventilation in the Zone is done with auxiliary fans as the Zones are located near surface and have minimal lateral and vertical extents. Zones 11 and 15 utilize the ventilation system from Zone 12 and do not require the installation of primary ventilation fans.

Maximum steady state airflow varies by Area, as do fan operating points. All main fans are planned to use the same diameter fan and hub (1.98 m and 0.80 m diameter, respectively) for

modularity, and are powered by 300, 200, 150 or 60 HP motors. Only RARs are powered. In some cases, booster fans will be required to be installed underground, notably at the 1,705 EL level in Area B, and also for the deeper portions of Zones 4, 5 and 7. Primary fan requirements are shown in Table 24.28. Where possible, fans will be relocated from mined-out areas to active areas to reduce CAPEX expenditures.

TABLE 24.28
PRIMARY VENTILATION FAN REQUIREMENTS BY ZONE

Zone	Primary Fans
1	Auxiliary Only
2	1 x 300 HP + 1 x 150 HP
3	1 x 150 HP
4	1 x 300 HP + 1 x 60 HP
5	1 x 150 HP
6	1 x 300 HP
7	1 x 150 HP
8	2 x 60 HP
9	Auxiliary Only
11	Auxiliary Only
12	2 x 300 HP, 2 x 200 HP, 1 x 150 HP
13	1 x 150 HP
14	1 x 60 HP
15	Auxiliary Only
16	1 x 200 HP

Auxiliary ventilation during ramp and level development utilizes 1.22 m or 1.07 m diameter semi-rigid plastic ducting (similar to Mecanicad or SpeedDuct). Once primary ventilation exists, crosscuts and sills are generally ventilated using 1.07 m or 0.91 m diameter flexible fabric ventilation ducting. Auxiliary fans are generally 150-200 HP for larger capital development and 100 HP for smaller development operating development, however, long single-heading operating development such as D&F pillars will utilize semi-rigid ducting.

Figure 24.20 to Figure 24.22 show the ventilation systems by Zone. Ventilation raises are nominally 3.4 m diameter.

The Authors recommend a detailed study of the required ventilation by Zone and period at a later stage of study. It is likely that ventilation CAPEX and OPEX can be reduced through the optimization of the ventilation system.

FIGURE 24.20 LONGITUDINAL PROJECTION OF ZONES 11 TO 15 VENTILATION SYSTEMS BY ZONE

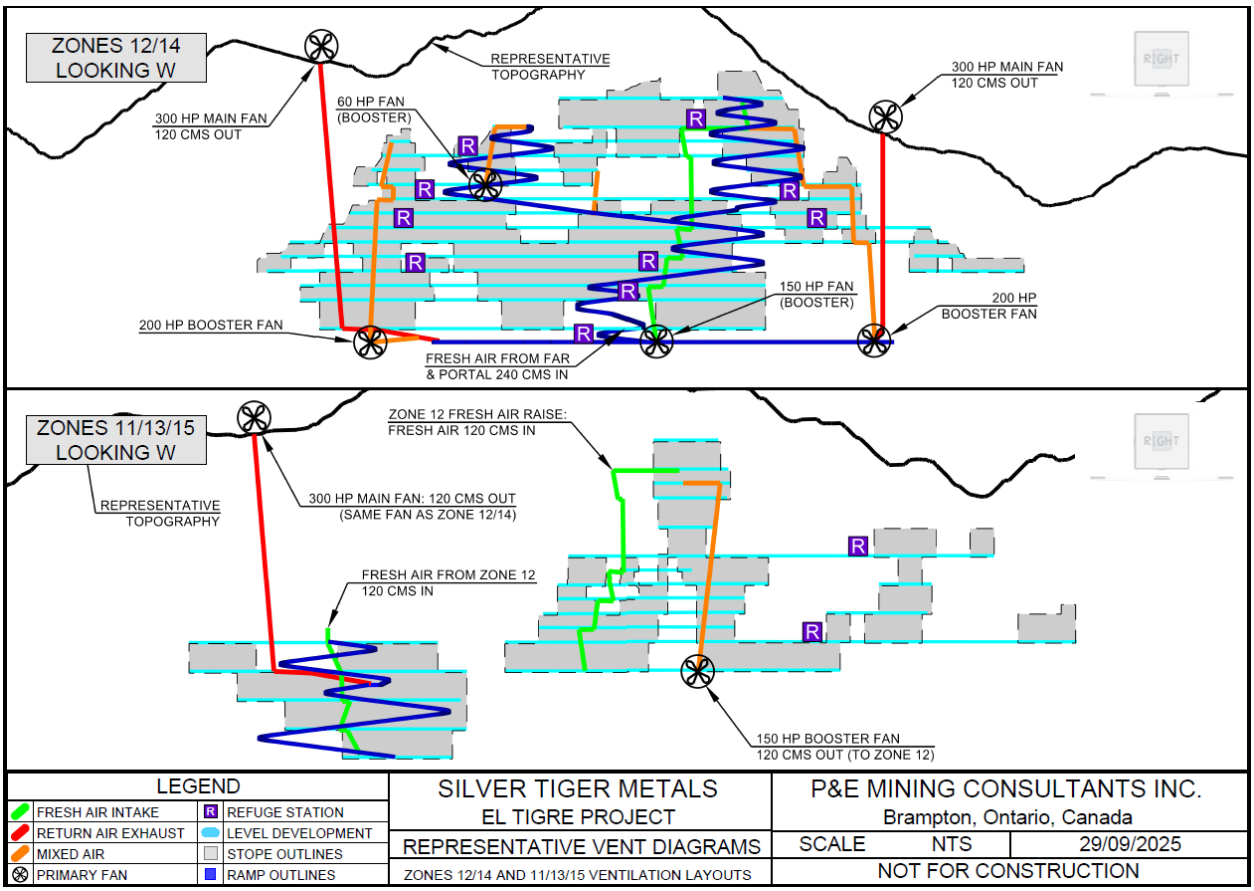


FIGURE 24.21 LONGITUDINAL PROJECTION OF ZONES 4 TO 8 AND 16 VENTILATION SYSTEMS BY ZONE

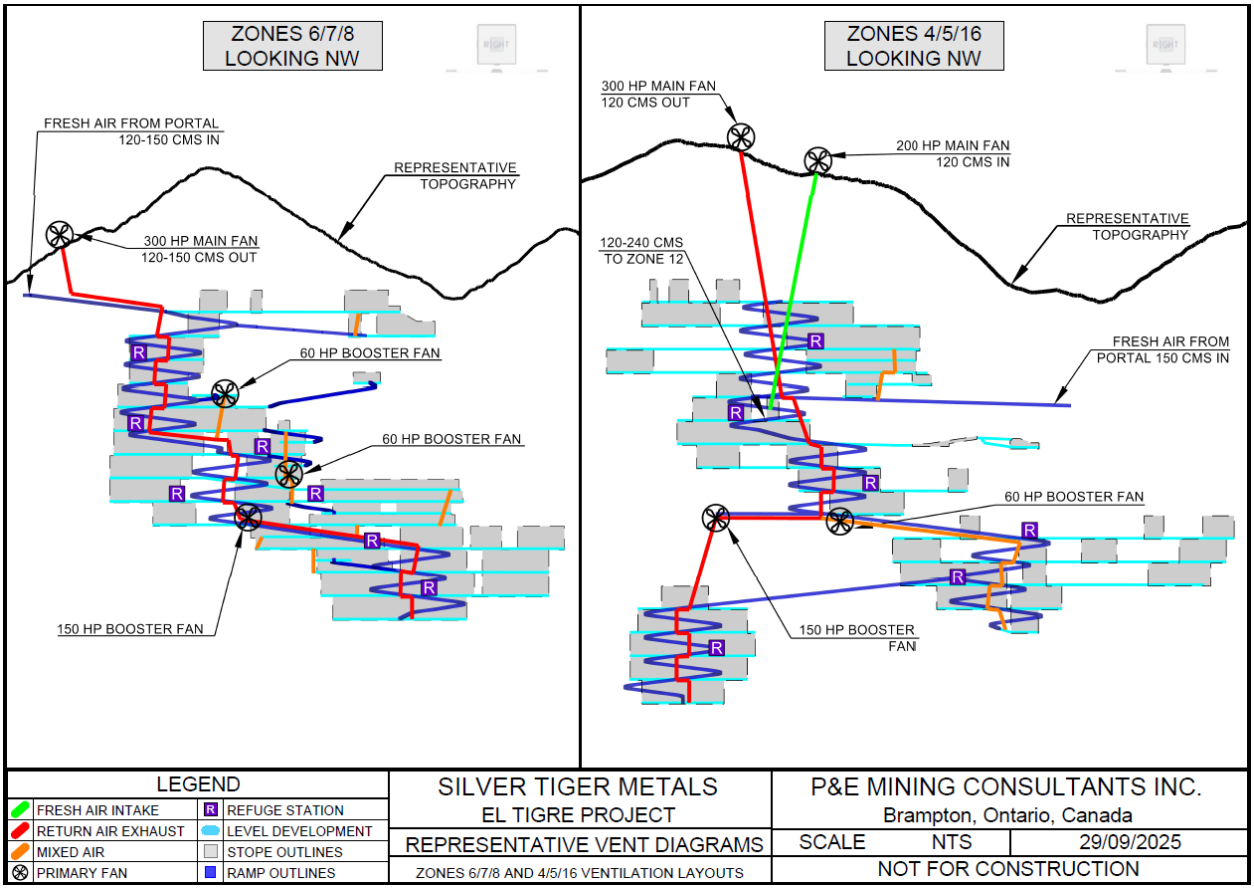
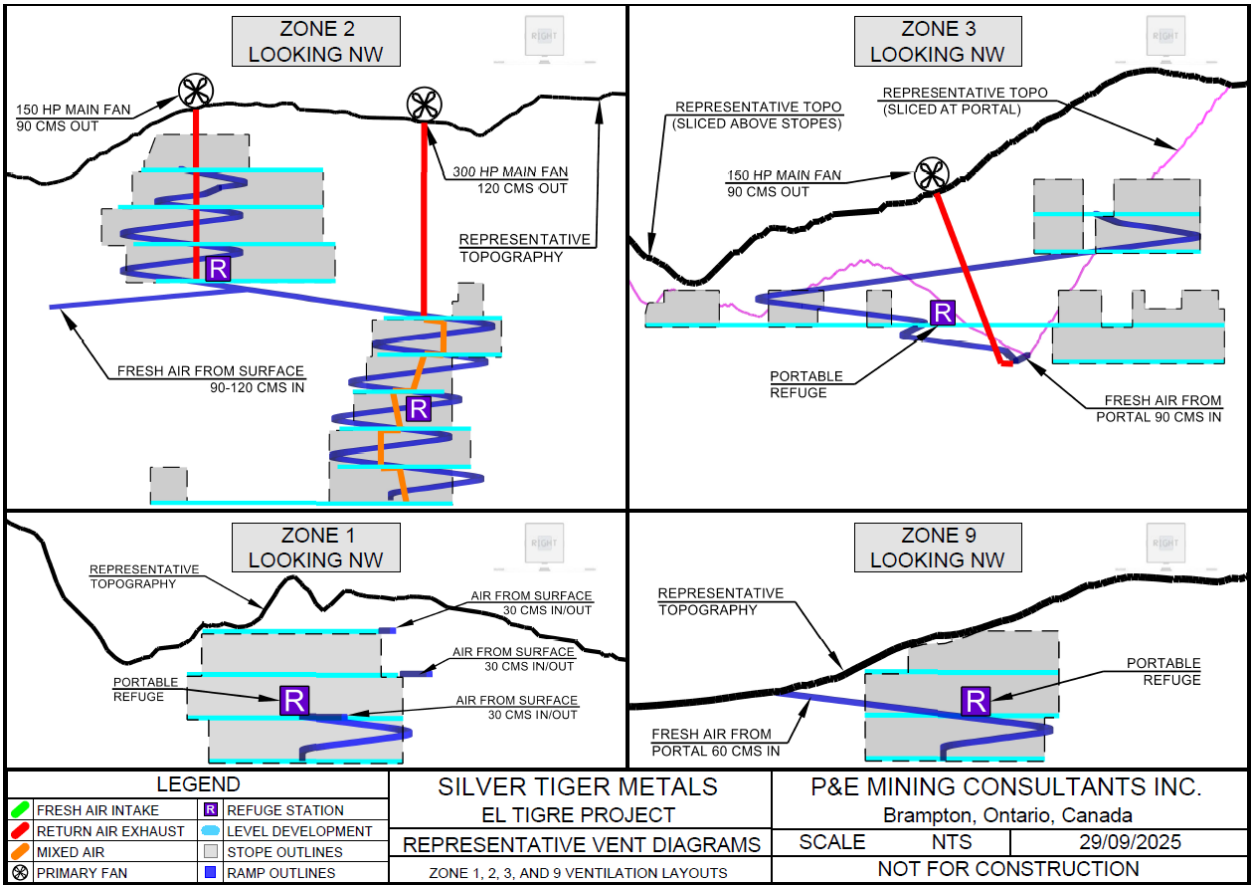


FIGURE 24.22 LONGITUDINAL PROJECTION OF ZONES 1 TO 3 AND 9 VENTILATION SYSTEMS



Dewatering

The UG dewatering system for the Project utilizes the following design:

- Face pumps (submersible, electric, ~5 HP) move dirty water from the face to a level sump;
- Level sumps cascade dirty water via gravity to main pump stations;
- In main pump stations, a settling/clarifying sump segregates solids from the dirty water and diverts clean water back into the mine service water system; and
- Excess water is pumped to surface via pipelines and drill holes.
  - Generally, UG mine outflows are expected to be utilized by the process plant.

The layout of the system varies depending on the Zone being mined. In general, pump stations are spaced on 150 m vertical intervals and are equipped with 250 HP of centrifugal and submersible pumps. Each pump station is designed to run at 10 L/s on a 33% duty cycle, and has a maximum flow rate of 30 L/s. Inflows across the entire UG mine are estimated at 12 L/s, however, multiple

mining areas will be active at one time, meaning that this inflow will be distributed across multiple pump stations.

For Zones with accesses below the highest mining horizon, all mine water flows via gravity to the pump station located closest to the elevation of the access, regardless of the vertical extents of the mining Zone above that elevation (see Zone 12 for an example).

Compressed Air

The compressed air system for the UG mine varies by mining Area. Generally, permanent compressed air distribution systems exist only in Areas B and C. All other Areas are expected to use mobile compressors as necessary given the relatively few active faces required for their operation. For Areas B and C, the compressed air system utilizes 150 mm steel pipes for distribution, with 100 mm diameter HDPE pipes in crosscuts and sills. Compressors are located at 1,705 EL (Area B) and 1,840 EL (Area C) and are expected to supply 1,200 scfm at 8.4 bar (~120 psi). Compressors are expected to be powered by 186 kW motors, and the system has been designed to operate with up to 40% losses. Compressed air lines will generally be run in parallel with PF lines in ventilation raises or drill holes to reduce fittings losses.

Electrical

The electrical distribution system for the UG mine utilizes a nominal 15 kV backbone, with step-down transformers located underground to reduce the supply voltage to a service voltage of a nominal 480 V. Mine Load Centres ("MLCs") containing transformers, starters, and switchgear, will be positioned as necessary throughout the mining Areas, and will generally be sized at either 1,000 kVA or 1,500 kVA depending on their location and expected loads. Development MLCs are generally 1,000 kVA and are expected to contain 4 circuits, production MLCs are generally 1,500 kVA and contain 8 circuits. For areas where surface loads are expected (ventilation fans at collars), development MLCs will be installed in a centralized location to feed up to 3 fans.

It is expected that the MLCs will be relocated within Zones as necessary when levels are exhausted, and between Areas as Zones are exhausted. For production MLCs, it is expected that they will be installed every 3rd level and cables run through drill holes to the adjacent levels to limit the number of necessary relocations. Development MLCs will be regularly relocated as mining faces advance. Figure 24.23 shows the approximate locations of MLCs over LOM. Not all MLCs will be active at once. Total electrical draw for the UG is expected to peak in YR 8 at approximately 4.8 MW, with ventilation (primary and auxiliary) making up slightly more than half of the draw, on average.

Refuges and Egress

Emergency egress from the UG mine will be via ladderway in the ventilation raises. Underground lunchrooms configured to serve as refuge stations will be installed according to applicable laws and regulations.

Due to the large extents and multiple mining faces expected in Zones 11-15 of Area B, each level in Zone 12 will be fitted with a permanent lunchroom, and every second lunchroom will be equipped to double as a refuge station. Table 24.29 shows the numbers of refuge stations per Zone.

A total of 26 permanent refuges will be installed over LOM. Additionally, two portable self-contained refuge stations will be purchased, and will be positioned as required over LOM.

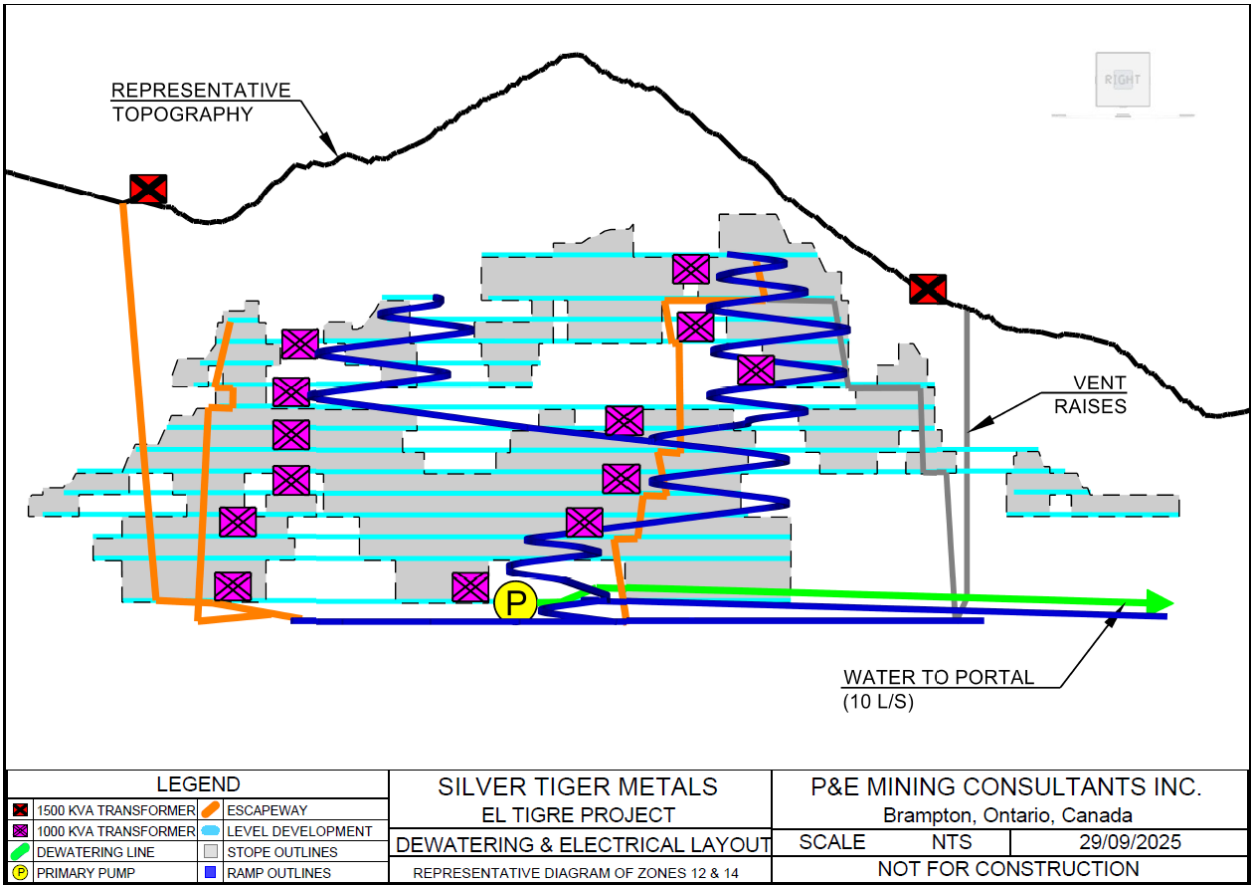
TABLE 24.29
REFUGE STATION QUANTITIES BY ZONE

Zone	No. of Refuge Stations
1	Portable Only
2	2
3	Portable Only
4	4
5	1
6	3
7	1
8	3
9	Portable Only
11	1
12	10
13	Utilizes stations in Zone 12
14	Utilizes stations in Zone 12
15	Utilizes stations in Zone 12
16	1

Escapeways are generally installed in drop raises using timber or prefabricated metal ladders installed after the raises are supported (support will be installed using hand-held drills working off the broken rock pile). Where escapeways are installed in raisebores, prefabricated systems (SafeScape or similar) will be used instead. Raisebored raises will not be supported, however, they may be lined with shotcrete if required. A total of 2.7 km of escapeway ladders will be installed, with roughly 60% of them being installed in drop raises and the remainder in raisebored airways.

Figure 24.23 shows the refuges and escapeways, along with electrical and dewatering services, in Zone 12 of the UG.

FIGURE 24.23 LONGITUDINAL PROJECTION OF ZONE 12 ELECTRICAL AND DEWATERING SERVICES WITH ESCAPEWAY ROUTES



Materials Handling

Mining contractors charge by distance and quantity for haulage operations. As a result, it is more cost-efficient to load on-level than install materials handling infrastructure like mineral passes ("MPs"), waste passes ("WPs") and bins to centralize truck loading. This approach has been followed throughout the majority of the mining Zones, however, MPs and bins are planned for Zone 8 and for the upper portions of Mine B (Zones 11-16). In Zone 8, the MPs are located where mining is transverse and truck accesses are poor. In Zones 12-16, MPs and WPs are used to transport material down to approximately the same elevation as the main portal, since the top 250 m of Area B is above this elevation, and trucking loaded down ramp results in higher costs than using materials handling infrastructure. Figure 24.24 shows the materials handling system in Zones 12-16, and Figure 24.25 shows a plan view of the 1,705 EL transfer level and 1,720 EL level.

FIGURE 24.24 LONGITUDINAL PROJECTION OF ZONES 12 TO 15 MATERIALS HANDLING SYSTEM

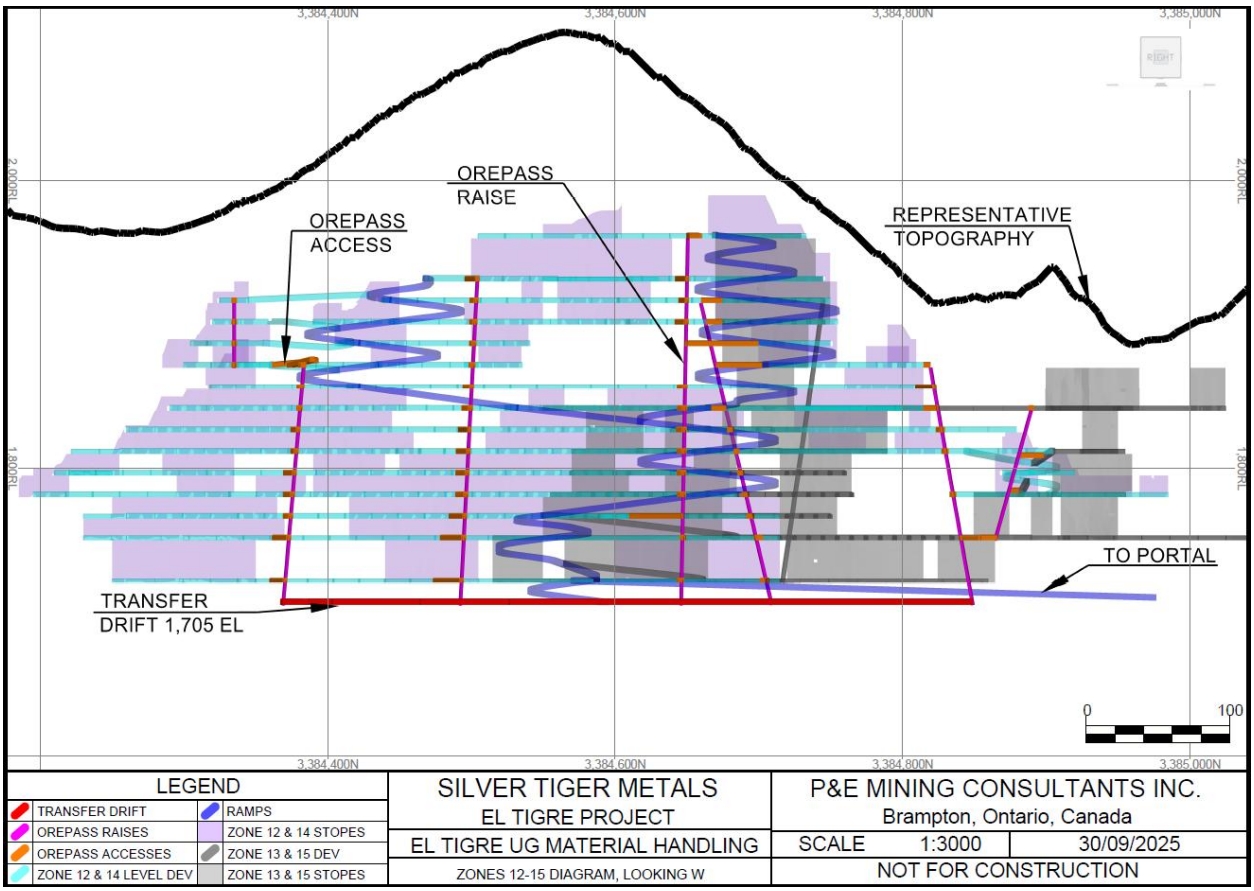
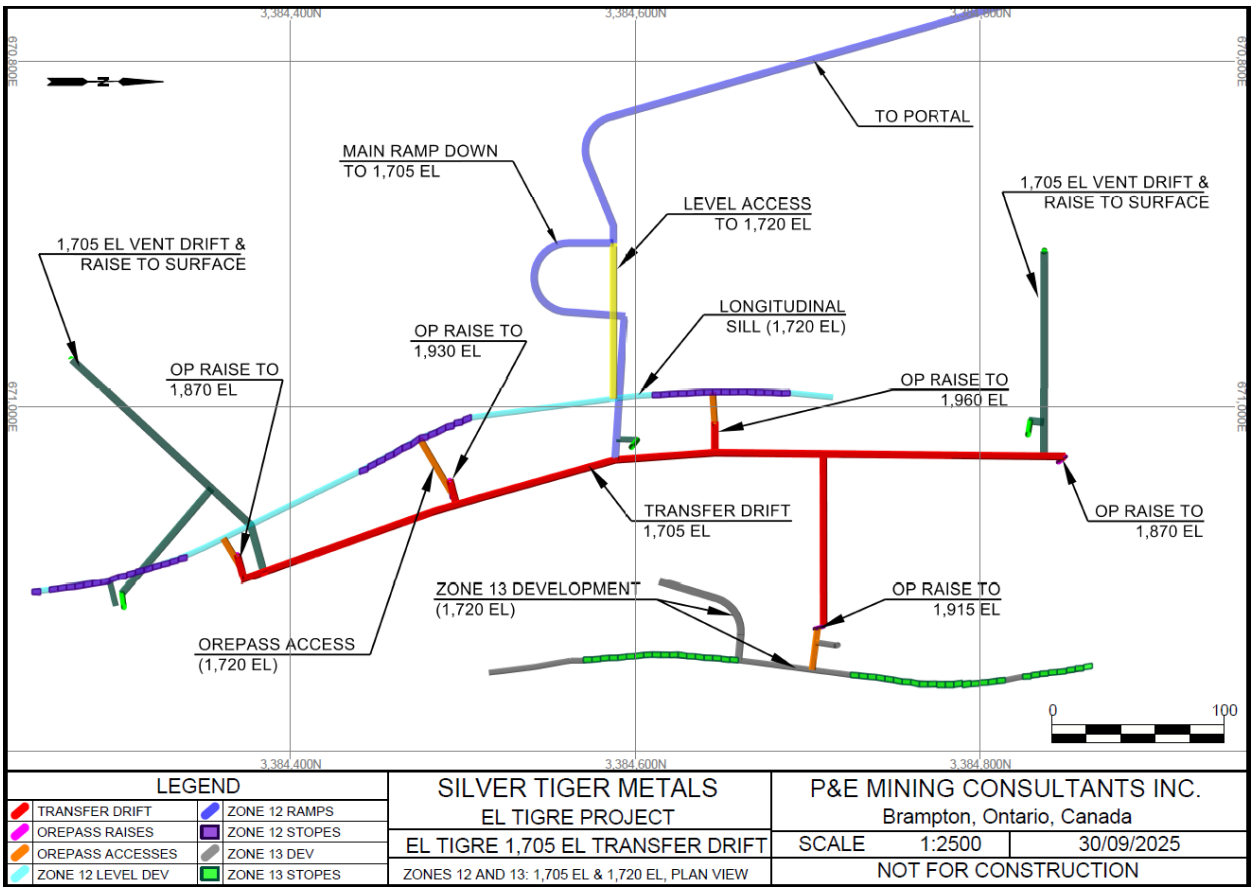


FIGURE 24.25 PLAN VIEW OF 1,705 EL TRANSFER LEVEL AND CONNECTIONS TO 1,720 EL LEVEL AND ABOVE



All passes are assumed to be raisebored at an average 2.7 m diameter (this is the average diameter of the average length mineral pass in the mine if bored at 2.1 m diameter with a 400-t capacity bin excavated at the bottom). Passes with significant longevity (Area B) will be lined with sprayed concrete. Bins will be outfitted with truck dumps to allow trucks to be loaded without the need for LHDs. A total of 9 passes (8 MPs and 1 WP) will be installed over LOM, varying from 45 m in length to 250 m. At each pass access other than the bottom where the bin is located, a grizzly and finger raise/chute will be installed. Grizzlies are expected to last for the life of the level and not be relocated. A total of 78 grizzlies will be installed over LOM. Bins are generally expected to last the life of the mineral pass, with the exception of the bins on the main Zone 12 MPs, which are expected to be replaced once during LOM.

Excavation of material from the surface SPs (Development, Low Grade and Tailings SPs) is expected to be excavated utilizing a front-end loader ("FEL") and 30 tonne highway haul trucks independent of any UG operations.

Backfill Distribution

PF is initially provided through 100 mm diameter steel lines run through the ventilation system. As the mine progresses deeper, drill holes will be completed parallel to the ramps to reduce wear

on the system from elbows in drop raises. Steel piping will be used for all permanent and semi-permanent portions of the system, with 76 mm diameter HDPE pipes used for temporary installations in sills and crosscuts.

The distribution system is planned to be entirely gravity-fed, negating the need for positive displacement pumps within the mine. Initial head may be provided from a positive displacement pump installed at the PF plant for filling of stopes where minimal elevation differences and/or significant lateral transit distances occur relative to the PF plant. It may be more efficient, for the small quantities required, to truck batched PF from the PF plant and dump into surface drill holes for the satellite mines (Areas D, E and F) rather than run surface distribution lines. The Authors recommend a review of this option at a later stage of study.

Peak flow rates within the PF distribution system are ~60 t/h. It is envisioned that up to two stopes can be actively filled at any time.

Maintenance and Fuel/Lube Distribution

Basic preventative maintenance can be performed underground in retrofitted remuck bays, however, major maintenance operations are expected to be performed on surface at a dedicated shop.

The significant lateral extents and multiple accesses for the UG Project make centralized fuel/lube bays impractical. Fuel and lube operations will be performed by mobile vehicles.

24.3.4.11 Personnel Requirements

Since the Project is planned to be operated by contractors, the personnel requirements for the UG operational roles will be determined by the contractor's requirements. National mining contractors generally plan for a 3-shift roster at 12 hours per shift. The Authors estimate a total staff of UG operators of approximately 120 personnel, including operators, labourers/assistants, supervision and maintenance personnel.

The Owner's technical staff, comprised of the engineering and geology departments, is estimated at a peak of 30 personnel, comprised of 10 personnel in geology and grade control, 14 personnel in engineering and surveying, 5 personnel in underground operations management and control, and one technical service manager. This staffing level is higher than normal for a mine of this size due to the need for extensive QA/QC of the contractor workforce to ensure operations are carried out to the standards of the Company across the multiple active mining Zones. Technical staff will work varying rosters and daily hours depending on their specific roles. Night shift coverage of operations management and survey roles is expected. All other roles are expected to be day-shift only.

24.3.4.12 Underground Mining Summary

A total of 5.02 Mt of mineralized material will be excavated over LOM, containing 146.5 koz Au and 40.6 Moz Ag, equivalent to 53.6 Moz AgEq. In addition to the precious metals content, the

mine plan also contains 1.7 kt of Cu, 5.1 kt of Pb and 8.4 kt of Zn. These base metals are not recovered during processing and provide no payable value, however, are provided for reference.

Mineral Resource Recovered in Mine Plan

Table 24.30 shows the mine plan portion of the Mineral Resource.

UG Mining Schedule

Table 24.31 shows the annual UG mining schedule. Figure 24.26 to Figure 24.43 show UG mining by period. The Low-Grade SP and Tailings SP are not shown. Production operations in the UG are scheduled at 290 ktpa of mineralized material with grades above DCOG, therefore overall yearly mineralized material production will slightly exceed 290 ktpa in all years until the end of LOM of the UG mine, with the excess going to the development SP. HMW material on each level is assumed to be 100% excavated prior to the start of production stoping operations of mineralized material. Individual LH faces are expected to produce approximately 150 tpd, whether HMW or mineralized material. A maximum of four faces are expected to be active at any one point on any level. PF rates are based on demand generated from stoping operations and are scheduled for a maximum of 60 m^3/hr and 16 operating hrs/day. Excavation of material from surface SPs is independent of UG production rates. Excavation of material from the Development and Low-Grade SPs is expected to be at 800 tpd, while excavation of material from the Tailings SP is expected to be at 1,000 tpd.

Lateral and vertical development is scheduled using Equivalent Metres (Eq. m), which assumes that development effort is generally proportional to volume excavated, and factors the work required for faces of different sizes to an equivalent effort to advance a standard face size. For the Project, lateral development is scheduled utilizing a nominal 4.0 m W x 4.0 m H face advancing 1.0 m as one Equivalent Metre. Vertical development utilizes a nominal 3.0 m W x 3.0 m L face advancing 1.0 m as one Eq. m. Lateral development rates assume blasted round lengths equivalent to the smallest dimension of profile (e.g. 4.0 m on a 4.0 m W x 5.0 m H face) and one round per day with the exception of: ramps, which are assumed to operate at two 3.0 m L rounds per day; and D&F pillars, which are assumed to operate at one 1.5-1.8 m L round per day. Development rates assume single-face productivity with the exception of transverse mining areas, where dual faces are assumed. Vertical development is assumed to progress at 3.0 m/day for raisebored raises, and 4.0 m/day for drop raises. All vertical development assumes a single face is available in any location.

TABLE 24.30
MINERAL RESOURCES RECOVERED IN MINE PLAN, BY CLASSIFICATION

Item			Tonnes (k)	AgEq (g/t)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
Internally Diluted Resource (In-Situ)	Low-Grade SP	Measured	-	-	-	-	-	-	-
		Indicated	101.0	262.5	0.92	181.0	0.023	0.226	0.104
		Subtotal M&I[1]	**101.0**	**262.5**	**0.92**	**181.0**	**0.023**	**0.226**	**0.104**
		Inferred	18.0	186.8	0.46	146.0	0.016	0.168	0.085
		Waste[2]	-	-	-	-	-	-	-
	UG Mine	Measured	1,338.5	699.0	2.37	489.5	0.070	0.254	0.449
		Indicated	658.8	677.9	1.65	531.7	0.068	0.151	0.295
		Subtotal M&I[1]	**1,997.3**	**692.1**	**2.13**	**503.4**	**0.069**	**0.220**	**0.398**
		Inferred	419.2	653.3	0.71	590.2	0.079	0.152	0.170
		Waste[2]	1,651.5	-	-	-	-	-	-
	Tailings SP	Measured	146.0	104.7	0.29	79.0	-	-	-
		Indicated	706.0	100.0	0.26	77.0	-	-	-
		Subtotal M&I[1]	**852.0**	**100.8**	**0.27**	**77.3**	**-**	**-**	**-**
		Inferred	52.0	104.9	0.27	81.0	-	-	-
		Waste[2]	-	-	-	-	-	-	-
	Total[1]	Measured	1,484.5	640.6	2.16	449.1	0.063	0.229	0.405
		Indicated	1,465.8	371.0	0.93	288.5	0.032	0.083	0.140
		Subtotal M&I[1]	**2,950.3**	**506.6**	**1.55**	**369.3**	**0.048**	**0.157**	**0.273**
		Inferred	489.2	577.8	0.66	519.7	0.069	0.136	0.149
		Waste[2]	1,651.5	-	-	-	-	-	-
Dilution	UG Mine	Waste[2]	197.6	-	-	-	-	-	-
Fully Diluted Mineral Resource	Low-Grade SP	Measured	-	-	-	-	-	-	-
		Indicated	101.0	262.5	0.92	181.0	0.023	0.226	0.104
		Subtotal M&I[1]	**101.0**	**262.5**	**0.92**	**181.0**	**0.023**	**0.226**	**0.104**
		Inferred	18.0	186.8	0.46	146.0	0.016	0.168	0.085
	UG Mine	Measured	2,378.0	393.5	1.33	275.5	0.040	0.143	0.253
		Indicated	1,157.1	386.0	0.94	302.7	0.039	0.086	0.168
		Subtotal M&I[1]	**3,535.1**	**391.0**	**1.20**	**284.4**	**0.039**	**0.124**	**0.225**
		Inferred	730.5	374.9	0.41	338.7	0.045	0.087	0.098
	Tailings SP	Measured	146.0	104.7	0.29	79.0	-	-	-
		Indicated	706.0	100.0	0.26	77.0	-	-	-
		Subtotal M&I[1]	**852.0**	**100.8**	**0.27**	**77.3**	**-**	**-**	**-**
		Inferred	52.0	104.9	0.27	81.0	-	-	-
	Total[1]	Measured	2,524.0	376.7	1.27	264.1	0.037	0.135	0.238
		Indicated	1,964.1	276.8	0.69	215.3	0.024	0.062	0.104
		Subtotal M&I[1]	**4,488.1**	**333.0**	**1.02**	**242.8**	**0.031**	**0.103**	**0.180**
		Inferred	800.5	353.1	0.40	317.6	0.042	0.083	0.091
Mining Loss	UG Mine	Measured	(169.6)	425.2	1.53	290.1	0.025	0.086	0.152
		Indicated	(61.5)	420.5	1.01	330.8	0.033	0.071	0.139
		Subtotal M&I[1]	**(231.1)**	**424.0**	**1.39**	**300.9**	**0.027**	**0.082**	**0.148**
		Inferred	(34.6)	387.0	0.43	348.5	0.044	0.085	0.095
Recovered Mineral Resource	Low-Grade SP	Measured	-	-	-	-	-	-	-
		Indicated	101.0	262.5	0.92	181.0	0.023	0.226	0.104
		Subtotal M&I[1]	**101.0**	**262.5**	**0.92**	**181.0**	**0.023**	**0.226**	**0.104**
		Inferred	18.0	186.8	0.46	146.0	0.016	0.168	0.085
	UG Mine	Measured	2,208.3	391.0	1.32	274.4	0.041	0.148	0.261
		Indicated	1,095.6	384.0	0.94	301.2	0.039	0.087	0.170
		Subtotal M&I[1]	**3,303.9**	**388.7**	**1.19**	**283.3**	**0.040**	**0.127**	**0.230**
		Inferred	695.9	374.3	0.41	338.2	0.046	0.087	0.098
	Tailings SP	Measured	146.0	104.7	0.29	79.0	-	-	-
		Indicated	706.0	100.0	0.26	77.0	-	-	-
		Subtotal M&I[1]	**852.0**	**100.8**	**0.27**	**77.3**	**-**	**-**	**-**
		Inferred	52.0	104.9	0.27	81.0	-	-	-
	Total[1]	Measured	2,354.3	373.3	1.25	262.3	0.038	0.138	0.244
		Indicated	1,902.6	272.2	0.68	211.6	0.024	0.062	0.103
		Subtotal M&I[1]	**4,256.9**	**328.1**	**1.00**	**239.6**	**0.032**	**0.104**	**0.181**
		Inferred	765.9	351.6	0.40	316.2	0.042	0.083	0.091

Notes: *M & I = Measured and Indicated Mineral Resources.*
1. Totals may not sum due to rounding .
2. Waste material is material not classified in the block model. It is distributed into Measured, Indicated or Inferred classifications at the dilution stage using a weighted average by mass.

TABLE 24.31
MINING SCHEDULE

	Item	Unit	YR-1	YR 1	YR 2	YR 3	YR 4	YR 5	YR 6	YR 7	YR 8	YR 9	YR 10	YR 11	YR 12	YR 13	YR 14	YR 15	YR 16	YR 17	Total[1]
Development	OPEX Lateral	Lin. m	-	2,421	3,320	3,464	3,268	2,860	2,870	2,717	2,334	2,529	1,395	2,986	1,080	1,218	415	-	-	-	**32,877**
	CAPEX Lateral		5,731	5,792	4,284	2,680	2,856	3,280	3,113	3,058	3,460	1,657	2,130	1,305	1,732	1,678	550	-	-	-	**43,305**
	CAPEX Vertical		734	1,110	1,278	58	149	239	269	141	740	422	121	67	234	33	-	-	-	-	**5,595**
	OPEX + CAPEX Lateral	*Lin. m/day*	*17.8*	*25.7*	*24.5*	*17.1*	*17.3*	*17.6*	*17.2*	*16.3*	*18.0*	*12.7*	*10.1*	*12.0*	*8.4*	*8.1*	*5.4*	-	-	-	***15.2***
		Eq. m/day	*19.0*	*24.0*	*20.0*	*16.0*	*16.0*	*16.0*	*16.0*	*16.0*	*16.0*	*10.0*	*10.0*	*10.0*	*8.0*	*8.0*	*5.6*	-	-	-	***14.0***
Waste Mined	HMW Stope Backfill	kt	-	1.1	18.2	73.1	40.8	27.0	108.9	29.7	34.1	7.6	48.9	3.8	14.1	27.0	2.8	-	-	-	**437.1**
	Development		280.2	294.8	248.6	179.9	170.0	194.9	162.8	204.2	187.6	106.1	117.0	90.9	87.0	87.2	27.5	-	-	-	**2,438.9**
	Subtotal[1]		**280.2**	**295.9**	**266.7**	**253.0**	**210.8**	**222.0**	**271.7**	**233.9**	**221.7**	**113.7**	**166.0**	**94.8**	**101.1**	**114.2**	**30.3**	-	-	-	**2,876.0**
Mineralization Mined[3]	Low Grade SP	kt	119.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**119.0**
	Dev (<MCOG)		2.6	12.9	5.4	10.2	10.0	4.7	6.2	3.5	5.5	9.9	2.6	7.5	1.4	4.8	-	-	-	-	**87.3**
	Dev (>MCOG)		-	51.2	51.0	35.7	47.9	29.4	61.8	19.9	48.1	36.6	22.5	44.7	29.2	21.0	10.9	-	-	-	**510.1**
	LH Production		-	238.8	239.0	254.3	242.1	260.6	228.2	245.0	224.1	253.4	267.5	245.3	260.8	269.0	131.6	-	-	-	**3,359.6**
	CF Production		-	-	-	-	-	-	-	25.1	17.7	-	-	-	-	-	-	-	-	-	**42.9**
	Tailings SP		17.9	-	-	-	-	-	-	-	-	-	-	-	-	-	77.7	365.0	365.0	78.4	**904.0**
	Subtotal[1]		**139.5**	**302.9**	**295.4**	**300.2**	**300.0**	**294.7**	**296.2**	**293.5**	**295.5**	**299.9**	**292.6**	**297.5**	**291.4**	**294.8**	**220.2**	**365.0**	**365.0**	**78.4**	**5,022.8**
	Contained AgEq[2]	koz	1,028	3,730	4,425	4,977	6,235	4,713	3,670	3,291	3,620	3,044	2,892	2,559	2,861	2,487	1,402	1,186	1,186	255	**53,561**
	Contained Au		3.4	10.8	16.5	20.7	29.2	18.2	11.1	7.3	7.0	2.2	3.0	2.4	3.1	2.9	1.7	3.1	3.1	0.7	**146.5**
	Contained Ag		722.7	2,777.0	2,963.5	3,144.9	3,646.2	3,098.1	2,683.1	2,646.1	2,999.6	2,852.4	2,621.9	2,349.0	2,584.4	2,228.9	1,248.5	910.1	910.1	195.5	**40,582.0**
	Contained Cu	t	26.1	45.6	2.1	10.4	4.2	4.1	18.8	103.9	276.4	413.1	471.8	209.4	64.6	15.1	2.6	-	-	-	**1,668.1**
	Contained Pb		258.5	88.0	2.8	21.9	9.8	7.9	34.9	180.7	733.5	1,496.3	1,327.4	567.4	235.8	82.2	29.4	-	-	-	**5,076.3**
	Contained Zn		120.3	114.2	4.8	23.9	12.1	7.2	41.1	264.4	1,263.3	2,583.2	2,405.8	1,112.4	412.7	46.6	2.2	-	-	-	**8,414.2**
	Ag Eq Grade[2]	g/t	229.2	383.0	465.8	515.6	646.5	497.4	385.4	348.8	381.0	315.6	307.4	267.5	305.4	262.4	198.0	101.1	101.1	101.1	**331.7**
	Au Grade		0.77	1.10	1.74	2.14	3.03	1.92	1.17	0.77	0.74	0.22	0.32	0.25	0.33	0.31	0.24	0.27	0.27	0.27	**0.91**
	Ag Grade		161.1	285.1	312.0	325.8	378.0	327.0	281.7	280.5	315.7	295.8	278.7	245.6	275.9	235.1	176.3	77.6	77.6	77.6	**251.3**
	Cu Grade	%	0.019	0.015	0.001	0.003	0.001	0.001	0.006	0.035	0.094	0.138	0.161	0.070	0.022	0.005	0.001	-	-	-	**0.033**
	Pb Grade		0.185	0.029	0.001	0.007	0.003	0.003	0.012	0.062	0.248	0.499	0.454	0.191	0.081	0.028	0.013	-	-	-	**0.101**
	Zn Grade		0.086	0.038	0.002	0.008	0.004	0.002	0.014	0.090	0.427	0.861	0.822	0.374	0.142	0.016	0.001	-	-	-	**0.168**
Process Plant Feed[3]	Low Grade SP	kt	119.0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	**119.0**
	UG Mine		-	290.0	290.0	290.0	290.0	290.0	290.0	290.0	290.0	290.0	290.0	290.0	290.0	290.0	229.8	-	-	-	**3,999.8**
	Tailings SP		17.9	-	-	-	-	-	-	-	-	-	-	-	-	-	77.7	365.0	365.0	78.4	**904.0**
	Subtotal[1]		**136.9**	**290.0**	**290.0**	**290.0**	**290.0**	**290.0**	**290.0**	**290.0**	**290.0**	**290.0**	**290.0**	**290.0**	**290.0**	**290.0**	**307.5**	**365.0**	**365.0**	**78.4**	**5,022.8**
	Contained AgEq[2]	koz	1,018.6	3,684.1	4,407.7	4,945.2	6,199.5	4,695.9	3,606.5	3,278.3	3,602.0	3,008.6	2,882.5	2,532.2	2,812.5	2,471.1	1,789.5	1,186.0	1,186.0	254.8	**53,561.1**
	Contained Au		3.4	10.6	16.4	20.5	29.1	18.2	10.5	7.3	7.0	2.1	3.0	2.3	3.7	2.9	2.6	3.1	3.1	0.7	**146.5**
	Contained Ag		716.8	2,743.8	2,956.9	3,128.4	3,620.0	3,084.8	2,677.7	2,635.0	2,983.1	2,820.2	2,613.8	2,325.8	2,488.0	2,214.7	1,557.3	910.1	910.1	195.5	**40,582.0**

TABLE 24.31
MINING SCHEDULE

	Item	Unit	YR-1	YR 1	YR 2	YR 3	YR 4	YR 5	YR 6	YR 7	YR 8	YR 9	YR 10	YR 11	YR 12	YR 13	YR 14	YR 15	YR 16	YR 17	Total[1]
	Contained Cu	t	26.1	44.8	2.0	10.3	3.8	4.1	16.4	102.8	274.4	403.6	471.8	208.5	78.6	15.0	5.8	-	-	-	**1,668.1**
	Contained Pb		258.5	174.5	5.6	43.0	18.7	15.7	62.3	359.3	1,462.3	2,963.8	2,654.8	1,131.0	514.4	163.7	66.8	-	-	-	**9,894.2**
	Contained Zn		120.3	111.9	4.6	23.1	11.1	7.0	34.2	260.2	1,255.6	2,531.9	2,405.8	1,104.8	486.8	46.4	10.3	-	-	-	**8,414.2**
	AgEq Grade[2]	g/t	231.5	395.1	472.7	530.4	664.9	503.7	386.8	351.6	386.3	322.7	309.2	271.6	301.7	265.0	181.0	101.1	101.1	101.1	**331.7**
	Au Grade		0.77	1.14	1.76	2.20	3.12	1.95	1.12	0.78	0.75	0.23	0.33	0.25	0.39	0.31	0.27	0.27	0.27	0.27	**0.91**
	Ag Grade		162.9	294.3	317.1	335.5	388.3	330.9	287.2	282.6	320.0	302.5	280.3	249.5	266.8	237.5	157.5	77.6	77.6	77.6	**251.3**
	Cu Grade	%	0.019	0.015	0.001	0.004	0.001	0.001	0.006	0.035	0.095	0.139	0.163	0.072	0.027	0.005	0.002	-	-	-	**0.033**
	Pb Grade		0.189	0.060	0.002	0.015	0.006	0.005	0.021	0.124	0.504	1.022	0.915	0.390	0.177	0.056	0.022	-	-	-	**0.097**
	Zn Grade		0.088	0.039	0.002	0.008	0.004	0.002	0.012	0.090	0.433	0.873	0.830	0.381	0.168	0.016	0.003	-	-	-	**0.168**
Backfill Placed	Volume	m^3 (k)	-	118.4	145.5	183.2	159.6	151.4	179.9	155.7	136.5	122.9	143.3	118.2	120.8	135.2	58.5	-	-	-	**1,928.9**
	Tonnes	k	-	219	269	339	295	280	333	288	253	227	265	219	224	250	108	-	-	-	**3,568**
	Tails Tonnes in PF		-	153	183	231	200	190	227	198	173	157	182	152	155	174	75	-	-	-	**2,450**
	Cement Tonnes in PF		-	5	10	13	13	12	12	9	9	6	9	5	6	6	3	-	-	-	**119**

Notes: *Lin. m = Linear metres, Eq. m = equivalent metres.*

1. *Totals may not sum due to rounding.*
2. *AgEq is calculated using the formula found in Section 24.3.4.5, [AgEq] = [Ag] + [Au] x 88.597*
3. *Copper, zinc and lead were not sampled in the Tailings SP and are shown here as zeroes. Base metals are not payable due to the selection of cyanidation for precious metals recovery.*

FIGURE 24.26 MINE PLAN LONGITUDINAL PROJECTION AT EOY YR -1, ZONES 11 TO 15

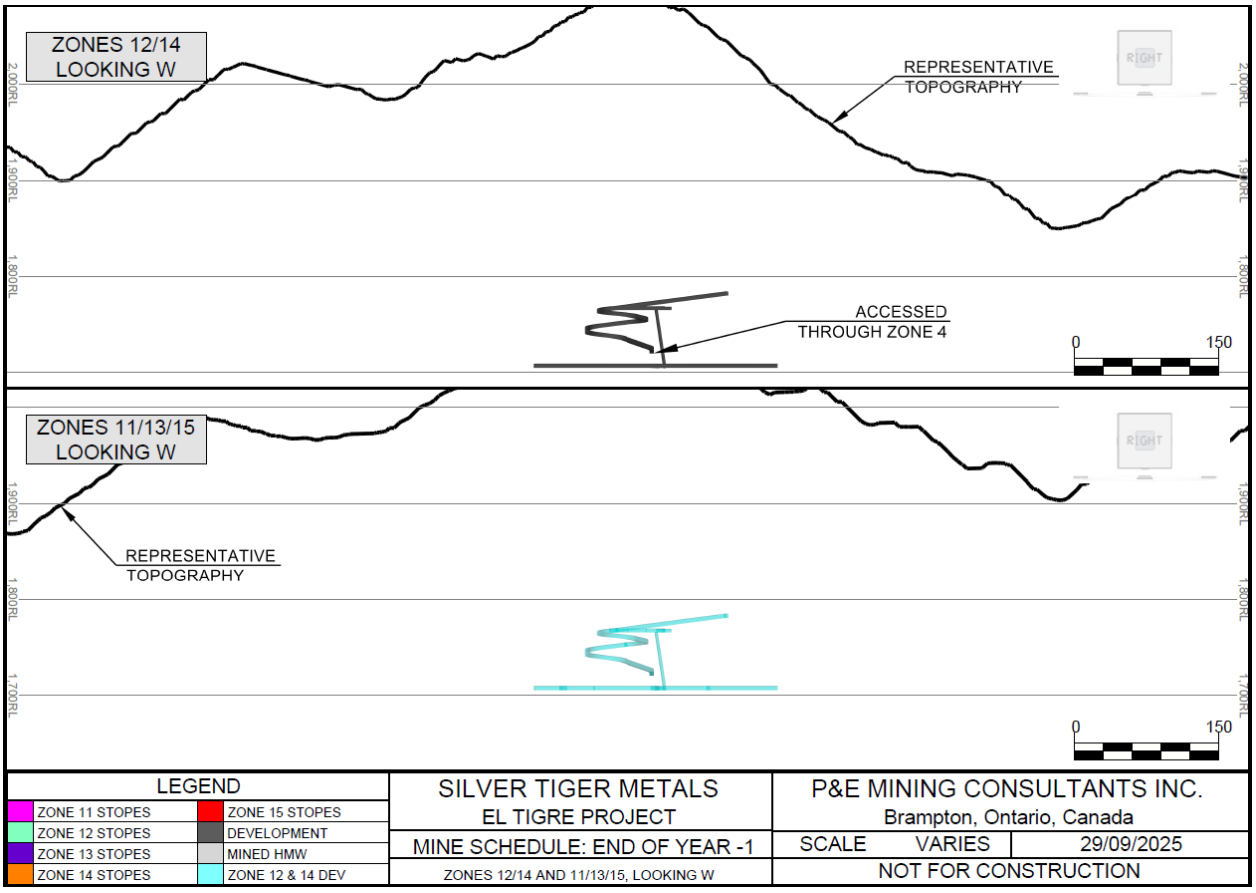


FIGURE 24.27 MINE PLAN LONGITUDINAL PROJECTION AT EOY YR 2, ZONES 11 TO 15

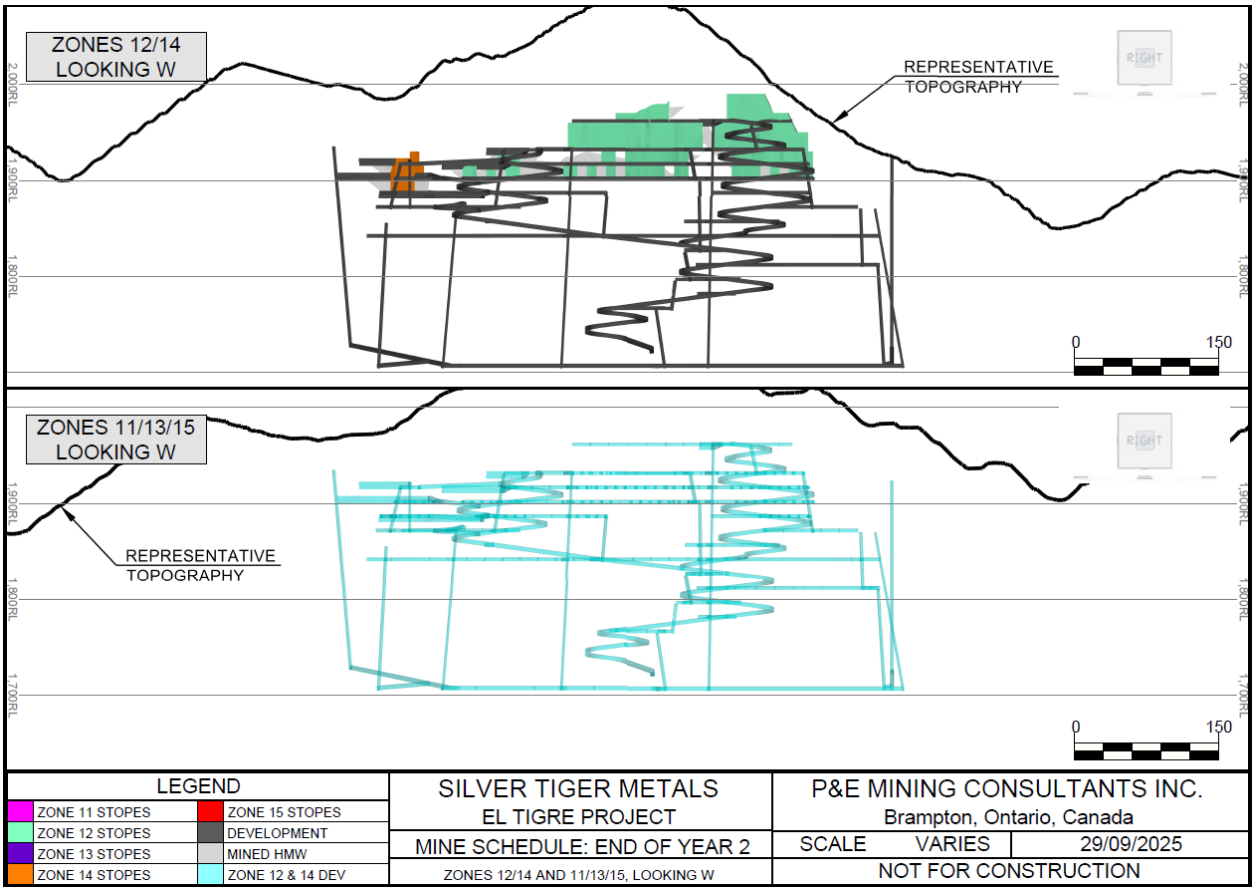


FIGURE 24.28 MINE PLAN LONGITUDINAL PROJECTION AT EOY YR 6, ZONES 11 TO 15

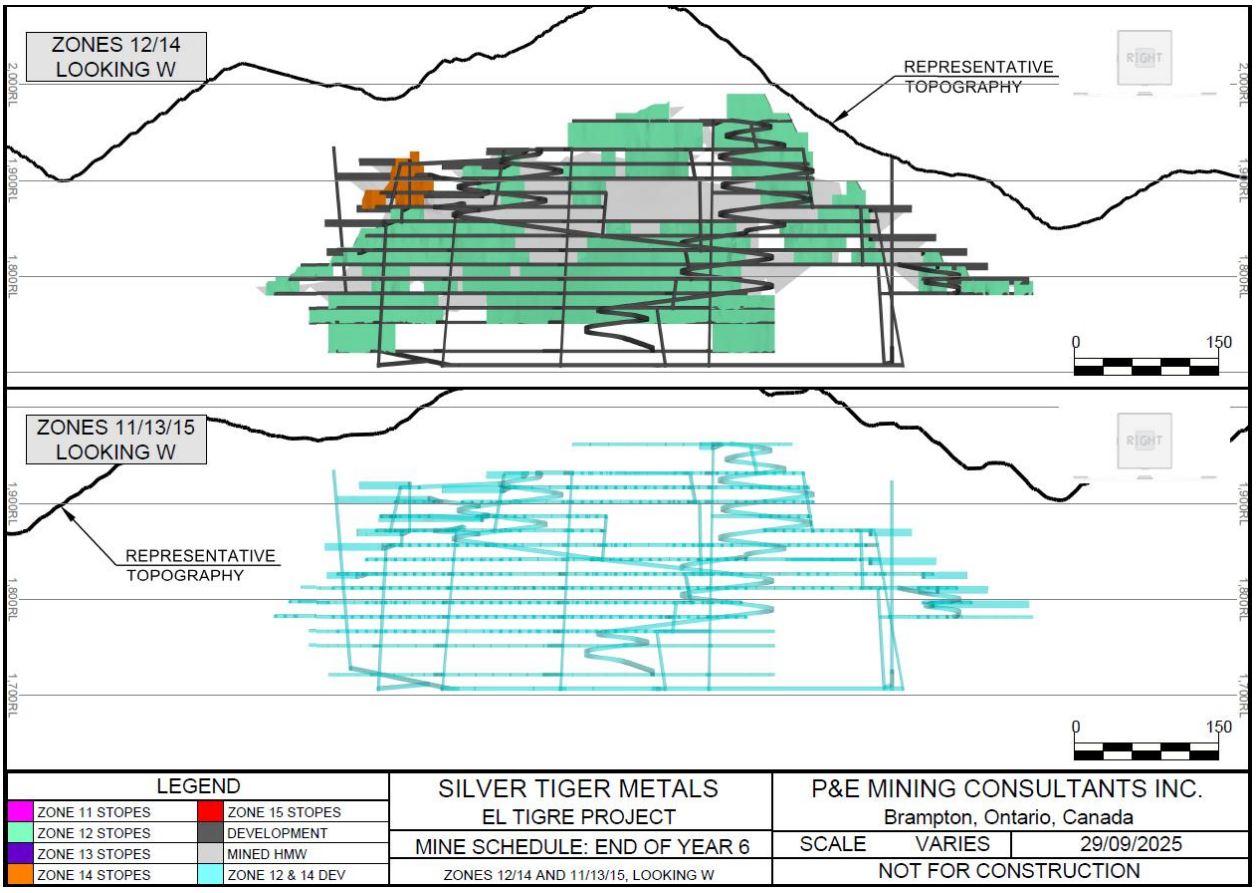


FIGURE 24.29 MINE PLAN LONGITUDINAL PROJECTION AT EOY YR 8, ZONES 11 TO 15

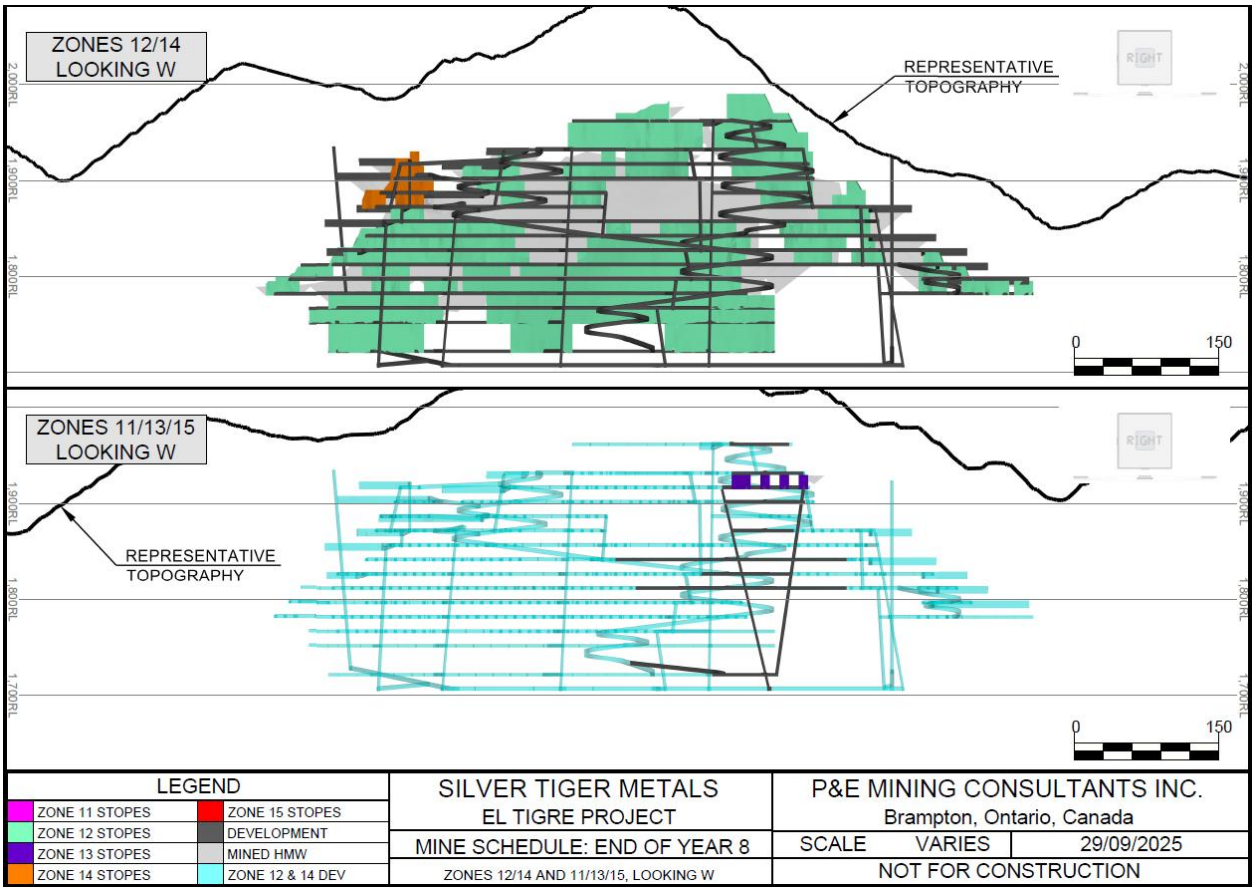


FIGURE 24.30 MINE PLAN LONGITUDINAL PROJECTION AT EOY YR 11, ZONES 11 TO 15

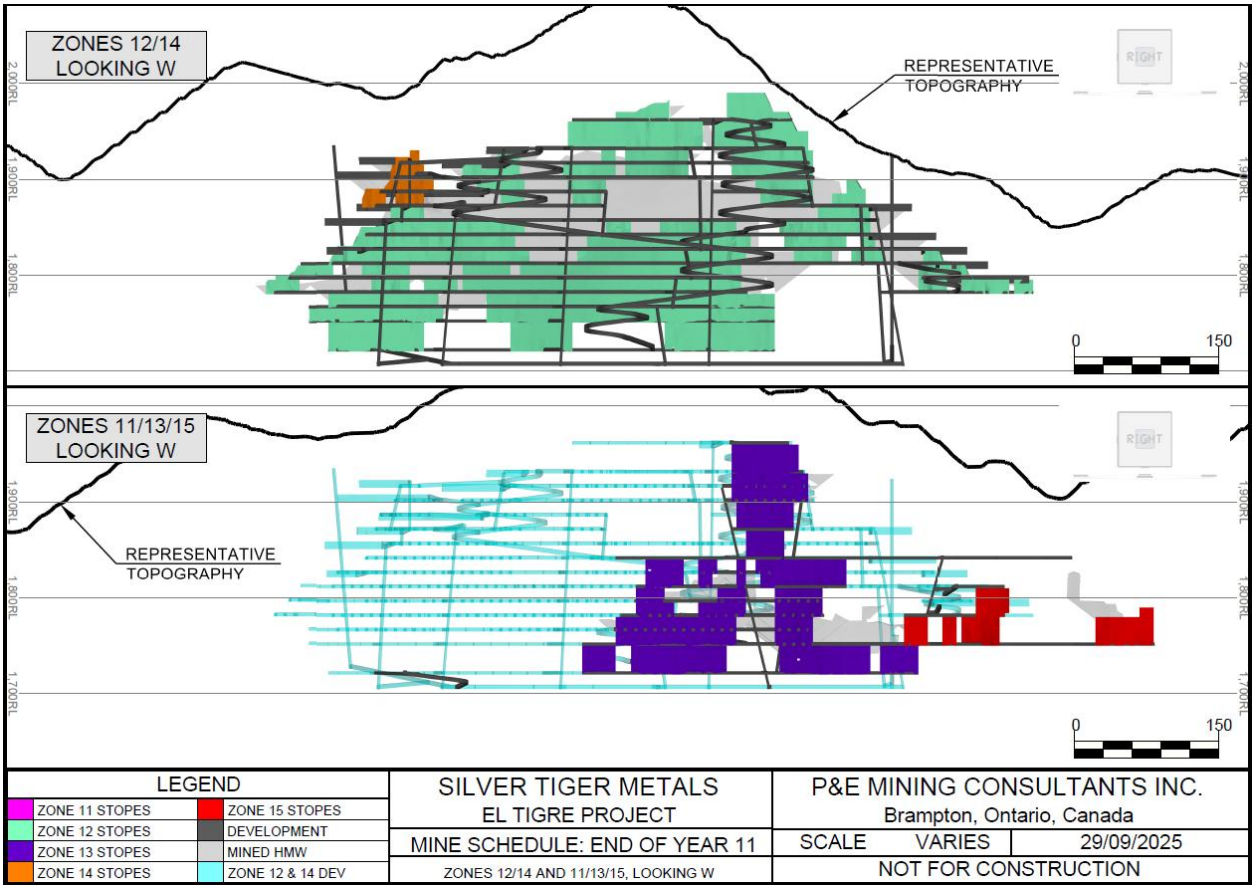


FIGURE 24.31 MINE PLAN LONGITUDINAL PROJECTION AT END OF LOM, ZONES 11 TO 15

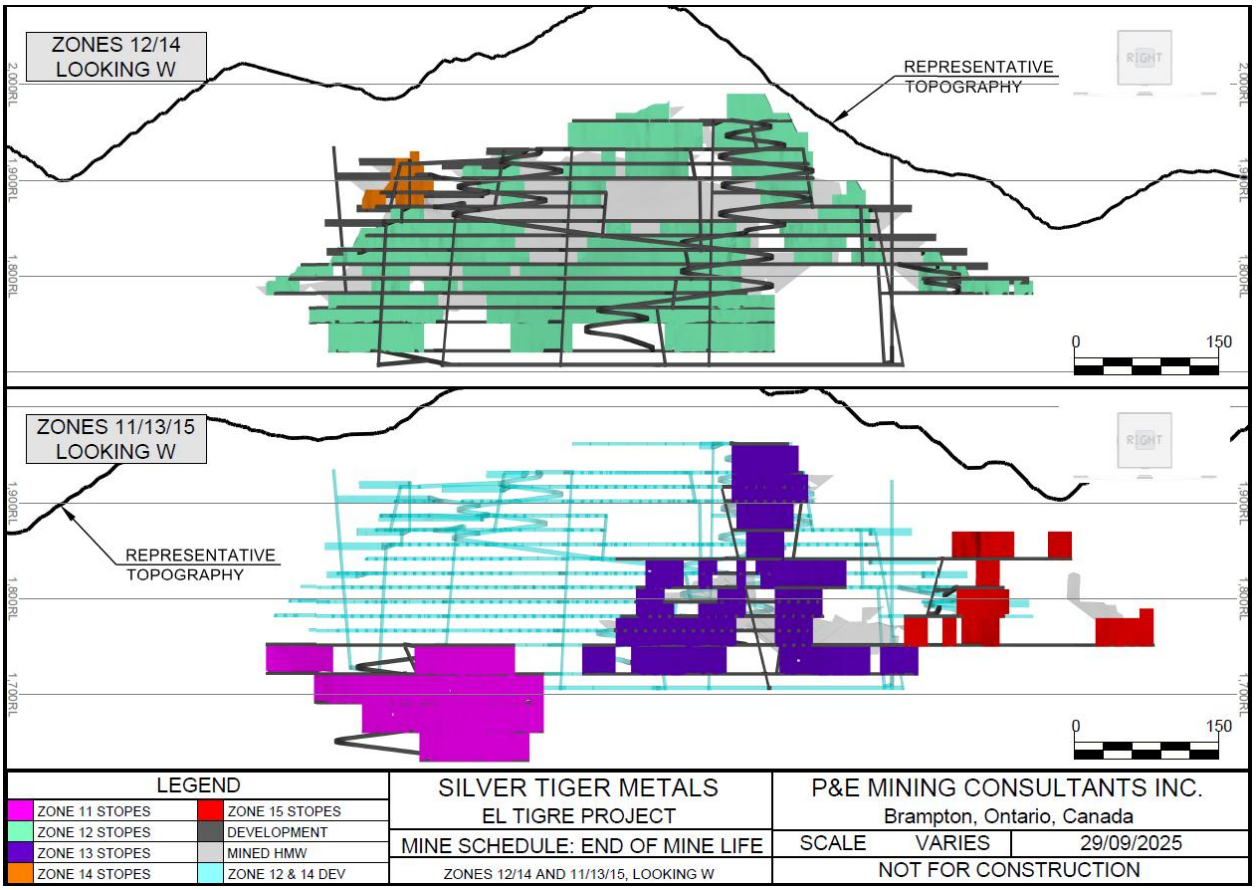


FIGURE 24.32 MINE PLAN LONGITUDINAL PROJECTION AT END OF YR -1, ZONES 4 TO 8 AND 16

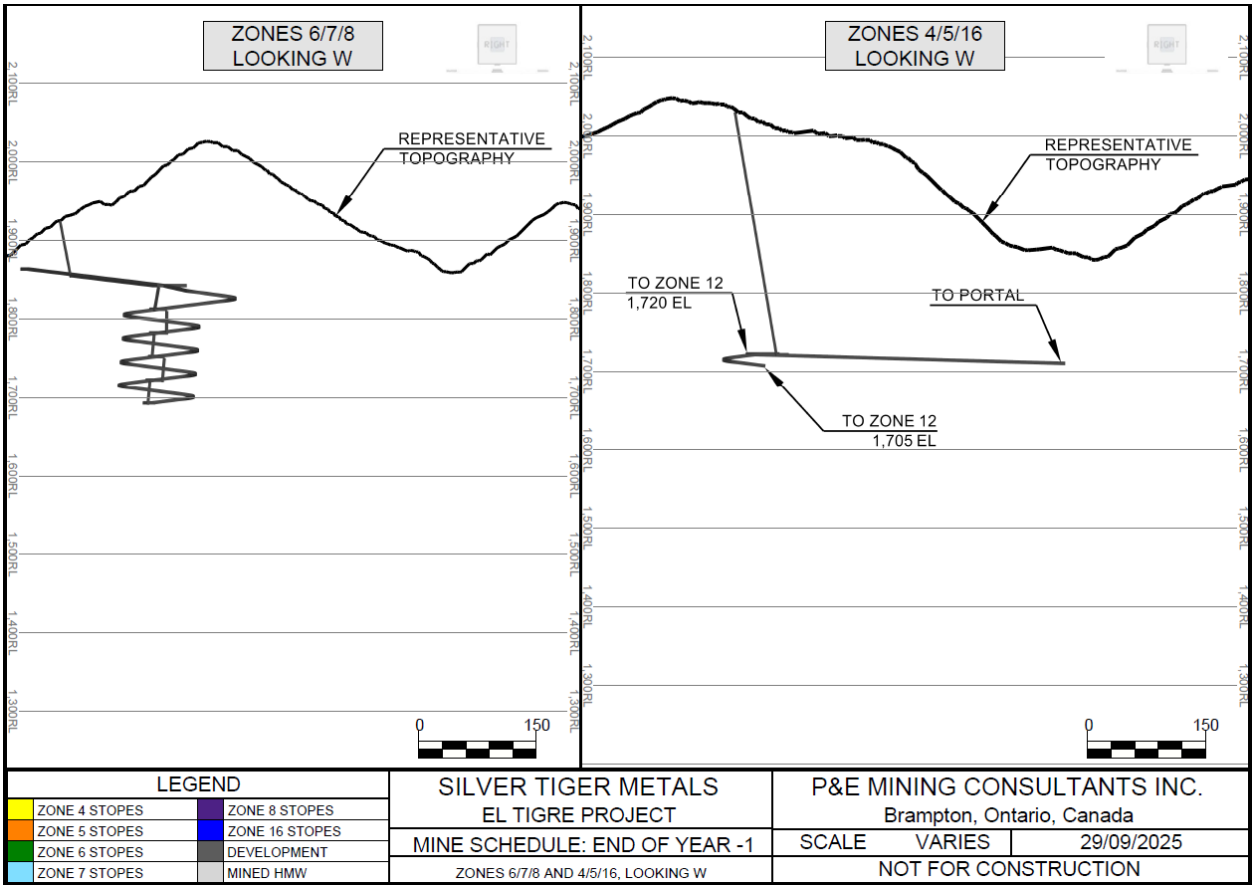


FIGURE 24.33 MINE PLAN LONGITUDINAL PROJECTION AT END OF YR 2, ZONES 4 TO 8 AND 16

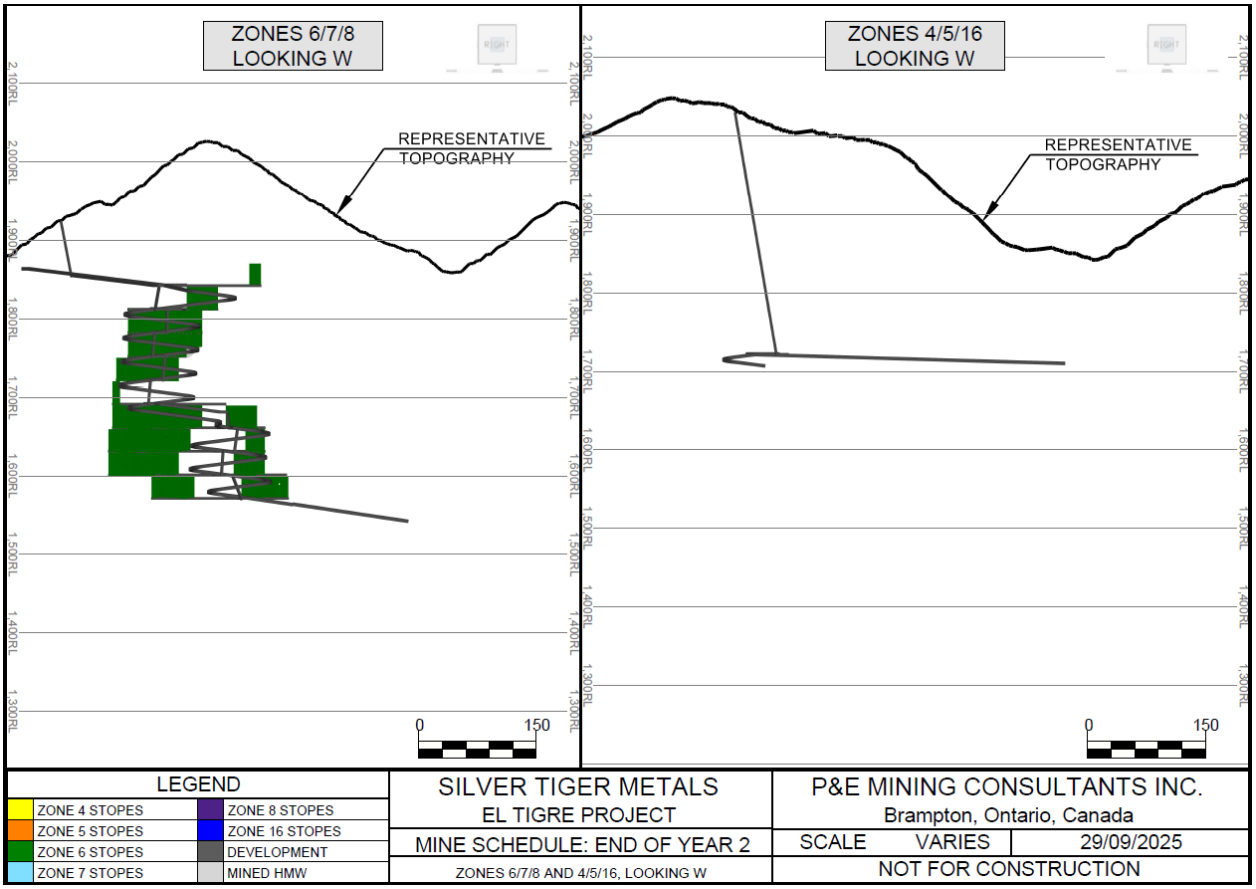


FIGURE 24.34 MINE PLAN LONGITUDINAL PROJECTION AT END OF YR 6, ZONES 4 TO 8 AND 16

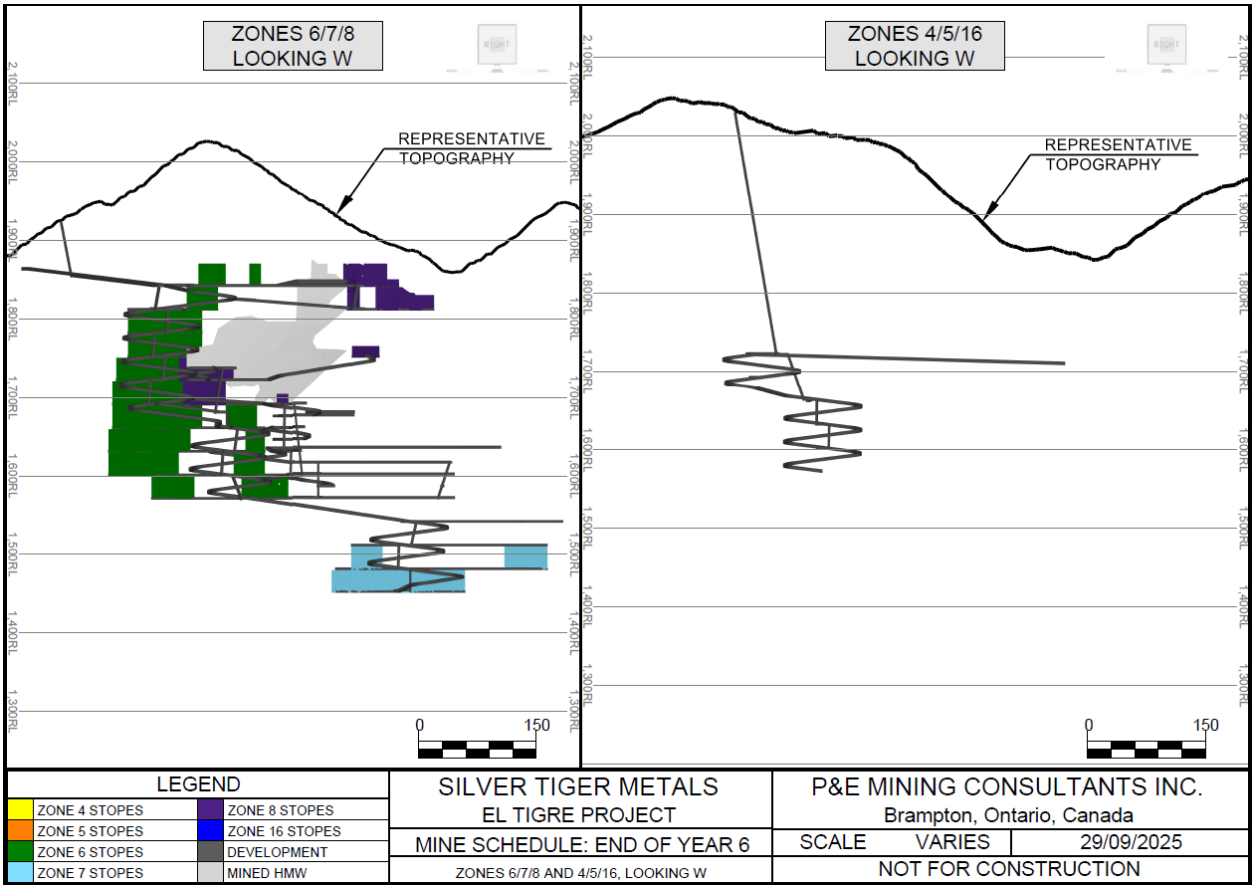


FIGURE 24.35 MINE PLAN LONGITUDINAL PROJECTION AT END OF YR 8, ZONES 4 TO 8 AND 16

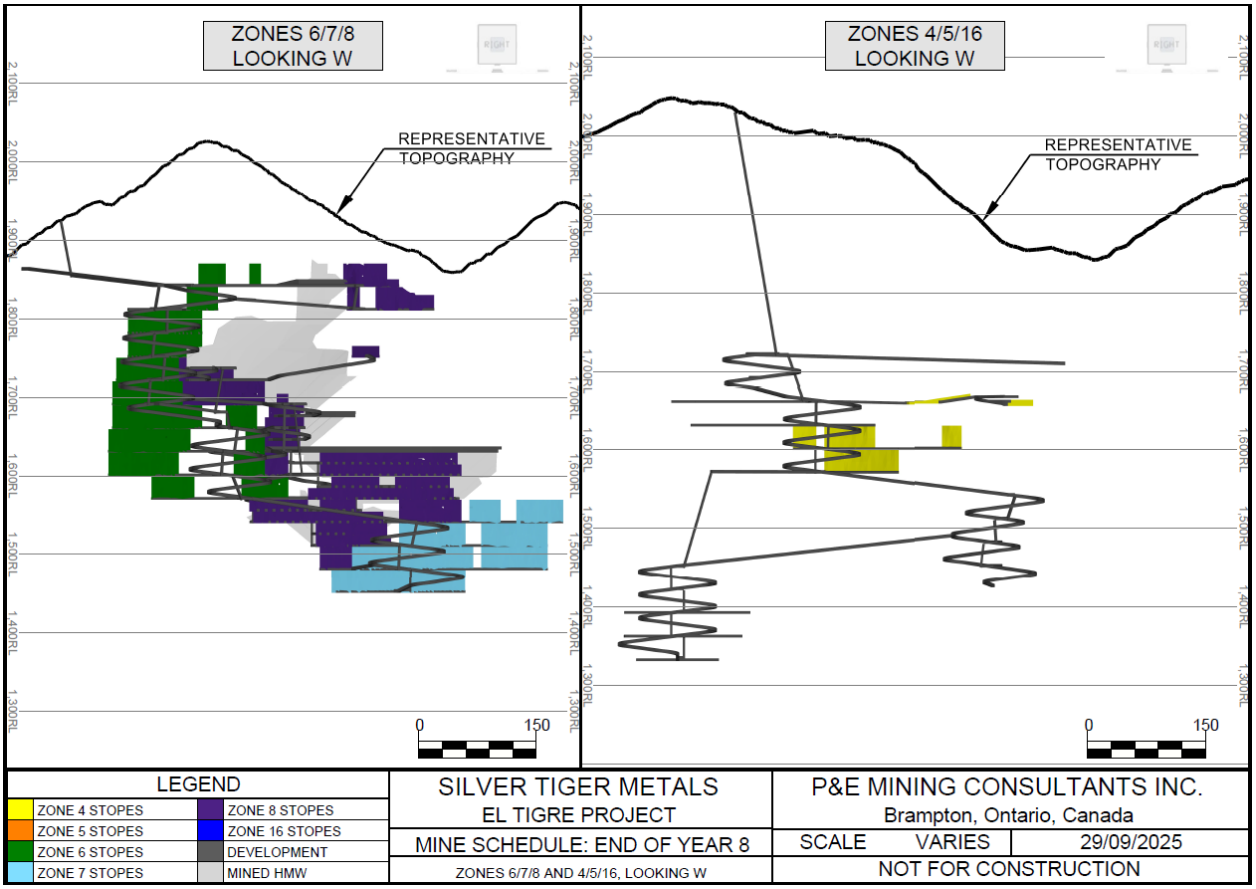


FIGURE 24.36 MINE PLAN LONGITUDINAL PROJECTION AT END OF YR 11, ZONES 4 TO 8 AND 16

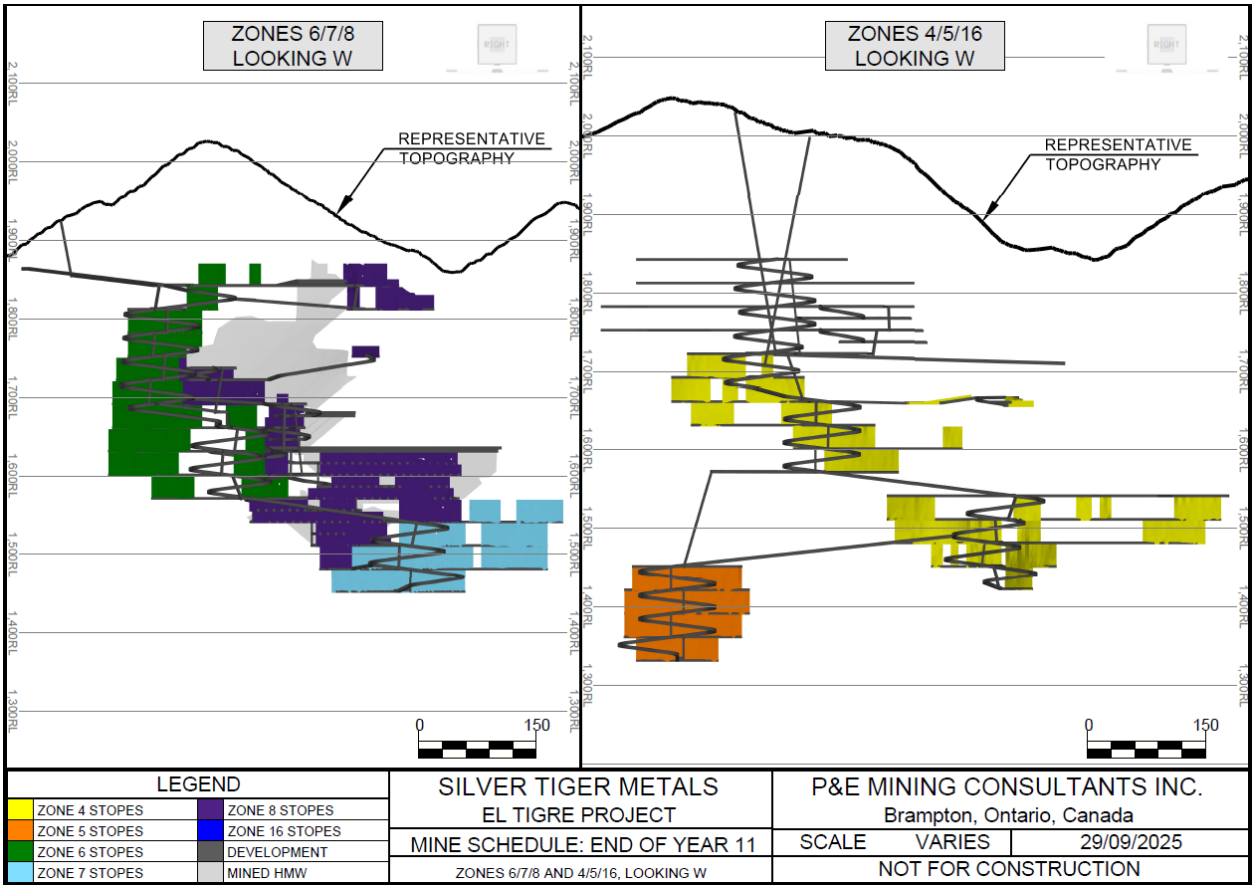


FIGURE 24.37 MINE PLAN LONGITUDINAL PROJECTION AT END OF LOM, ZONES 4 TO 8 AND 16

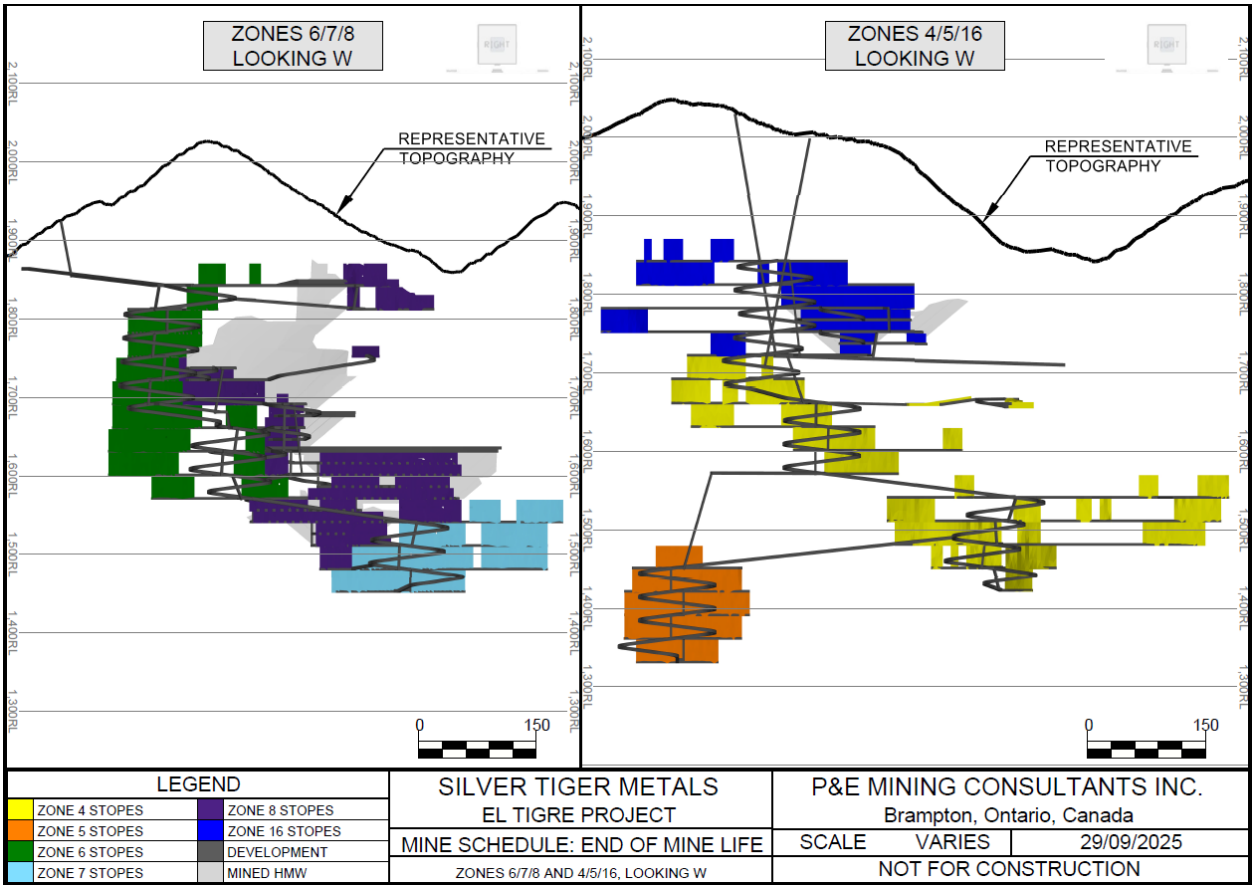


FIGURE 24.38 MINE PLAN LONGITUDINAL PROJECTION AT END OF YR -1, ZONES 1 TO 3 AND 9

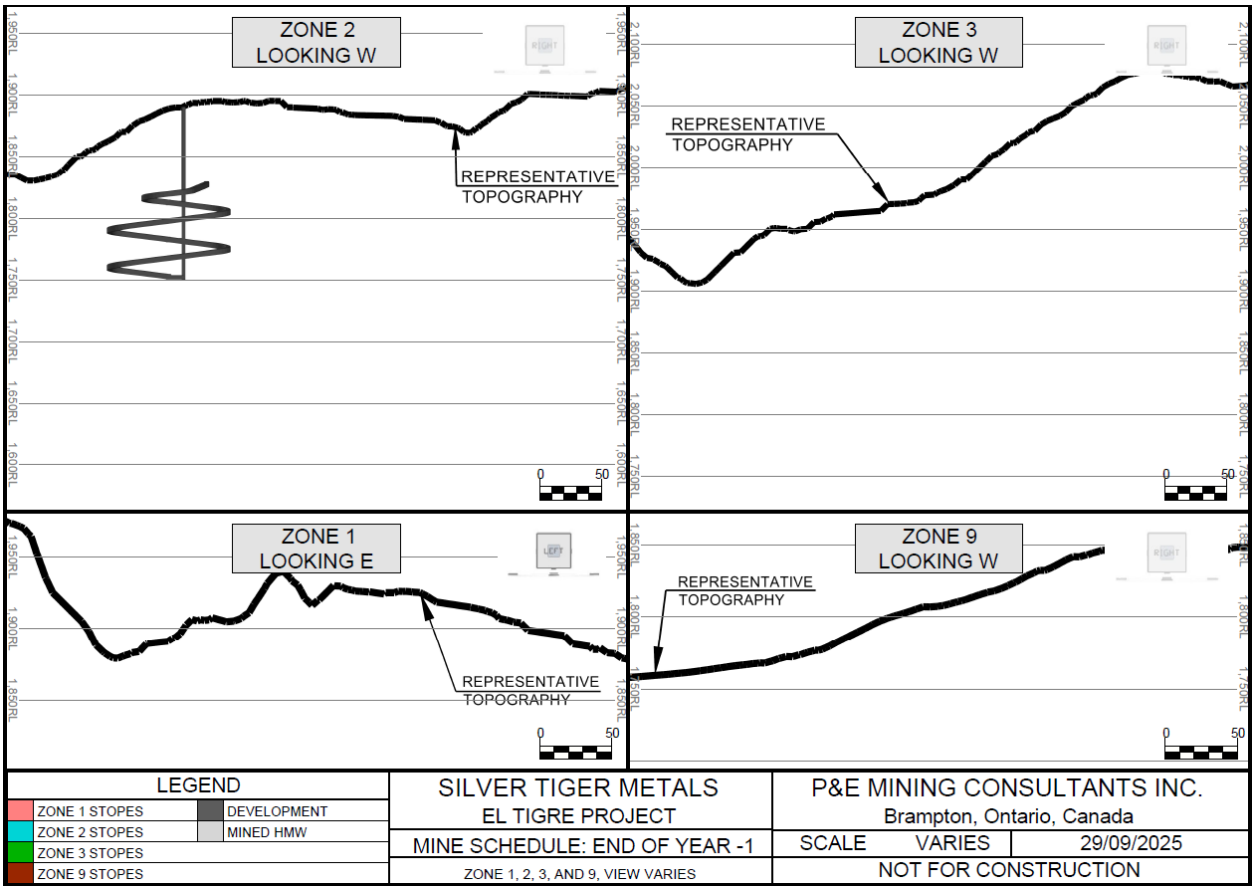


FIGURE 24.39 MINE PLAN LONGITUDINAL PROJECTION AT END OF YR 2, ZONES 1 TO 3 AND 9

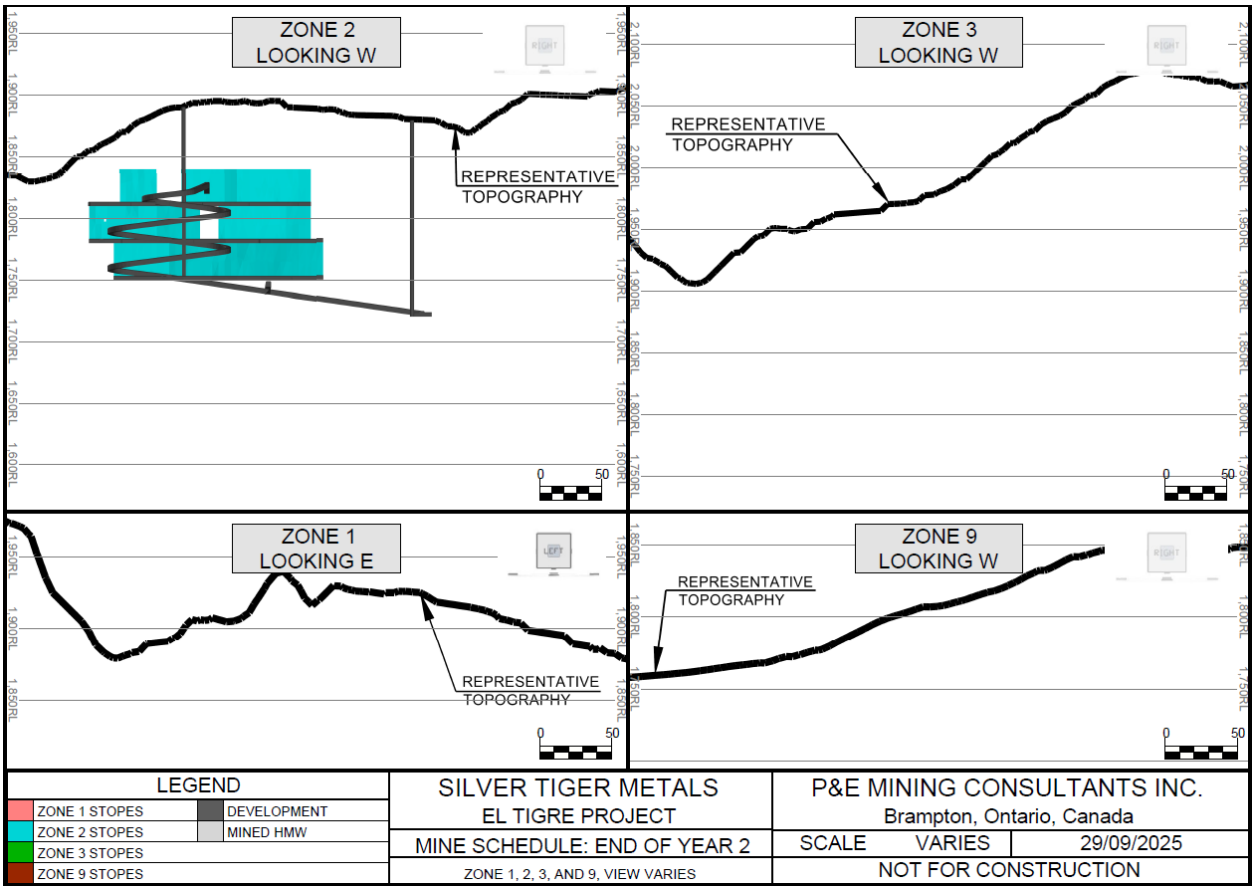


FIGURE 24.40 MINE PLAN LONGITUDINAL PROJECTION AT END OF YR 6, ZONES 1 TO 3 AND 9

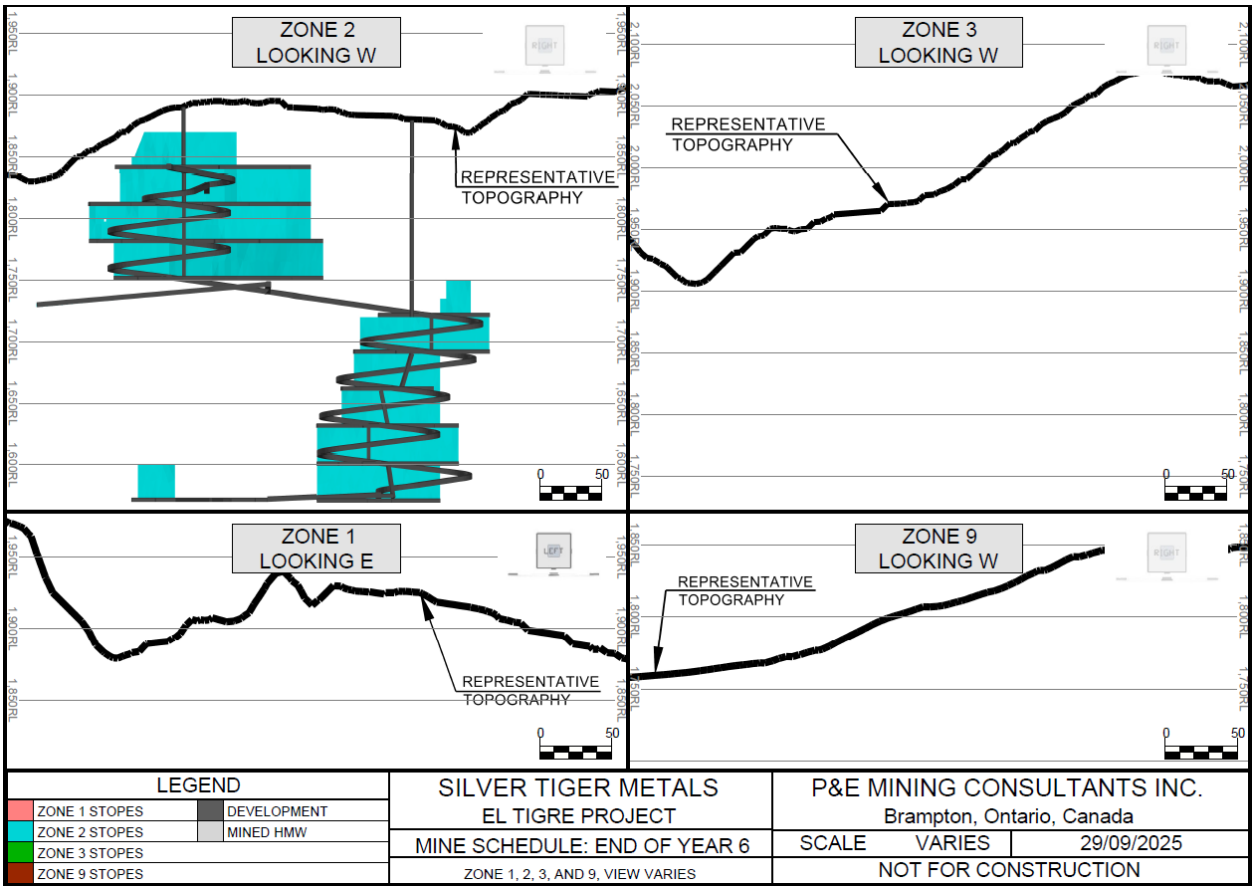
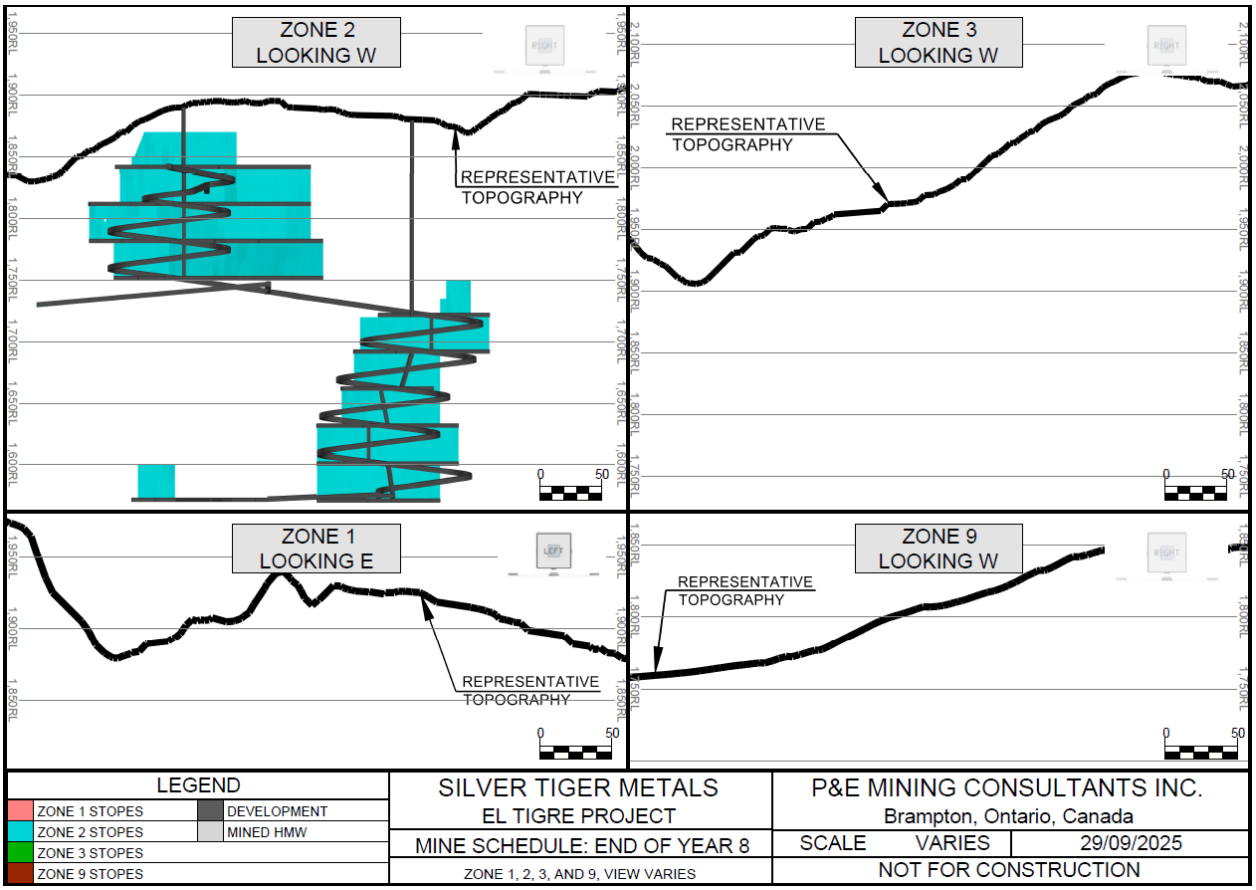


FIGURE 24.41 MINE PLAN LONGITUDINAL PROJECTION AT END OF YR 8, ZONES 1 TO 3 AND 9



Note: *no mining occurs in these zones between YR 6-8.*

FIGURE 24.42 MINE PLAN LONGITUDINAL PROJECTION AT END OF YR 11, ZONES 1 TO 3 AND 9

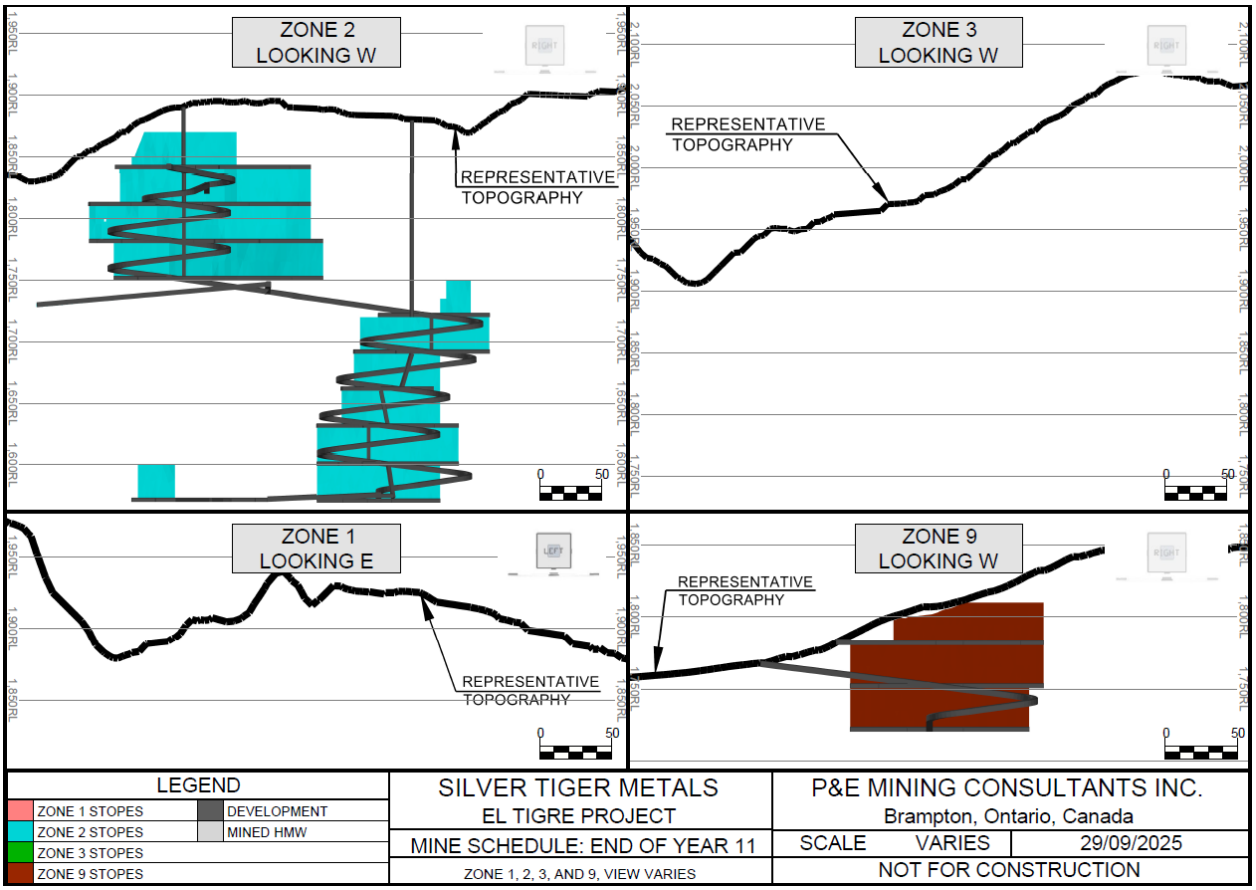
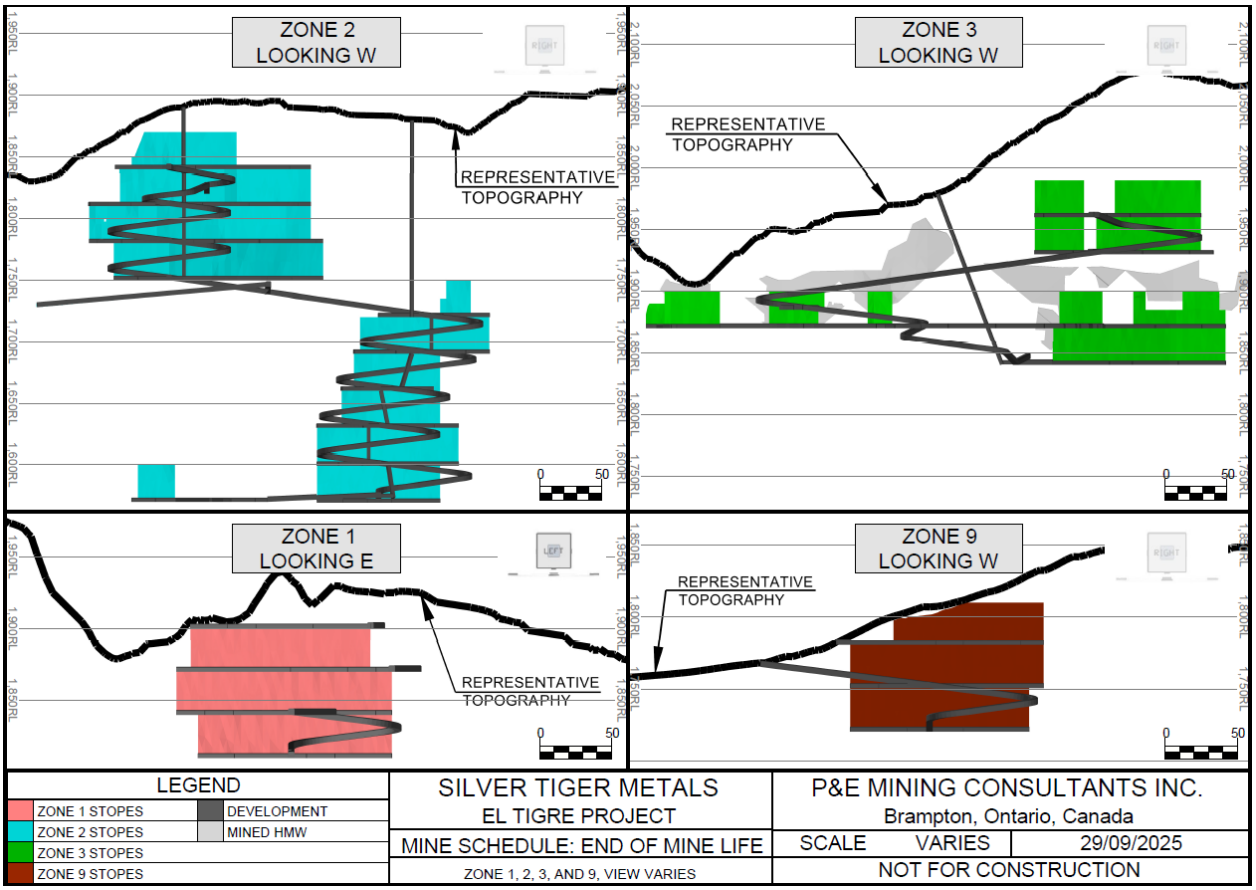


FIGURE 24.43 MINE PLAN LONGITUDINAL PROJECTION AT END OF LOM, ZONES 1 TO 3 AND 9



24.3.5 Recovery Methods and Process Plant

24.3.5.1 Summary

This Section describes the recovery methods to be implemented in the design of the process facility for the El Tigre historical tailings and stockpile mineralization as well as the underground Mineral Resource. Testwork results presented in Section 24.3.2 of this Technical Report form the basis for the flowsheet development and combination design criteria. The proposed processing strategy will first utilize a combination of tailings and stockpile mineralization, followed by underground Mineral Resource, and then back to tailings reprocessing at the end of LOM.

Pre-production development of the underground mine will span two-years when tailings and stockpile materials will be processed at a nominal rate of 1,000 tpd via conventional recovery methods. Pre-production processing includes material handling, milling, leaching, solid/liquid separation, Merrill Crowe precipitation, and dry stack tailings impoundment.

Commencing in year one of production, processing of the underground Mineral Resource will be at a nominal rate of 800 tpd. The underground Mineral Resource is expected to be depleted by year 14 at which time the facility will revert to tailings reprocessing until year 17.

Dewatered tailings are also planned for the Project. To negate high capital costs and stay within the approve permit footprint, the tailings facility was reviewed by Knight Piésold Consulting. It was determined that dewatering tailings would not only minimize storage needs but would also be used in paste backfill for the underground mined zones.

The process plant flowsheet design comprises two stages of conventional crushing (portable), material handling of crushed product, and loading into the fines bin. The grinding circuit comprises two ball mills with a connected load of 746 kW (1,000 Hp) in closed circuit with pumping and classifying cyclones. An extra smaller mill is inserted to ensure proper grind size for the higher hardness of the underground material. The thickened cyclone overflow(s) undergoes a six stage leaching circuit followed by a four stage counter current decantation ("CCD") circuit. The pregnant solution is then treated in a standard Merrill Crowe process followed by refining and production of silver and gold doré bars.

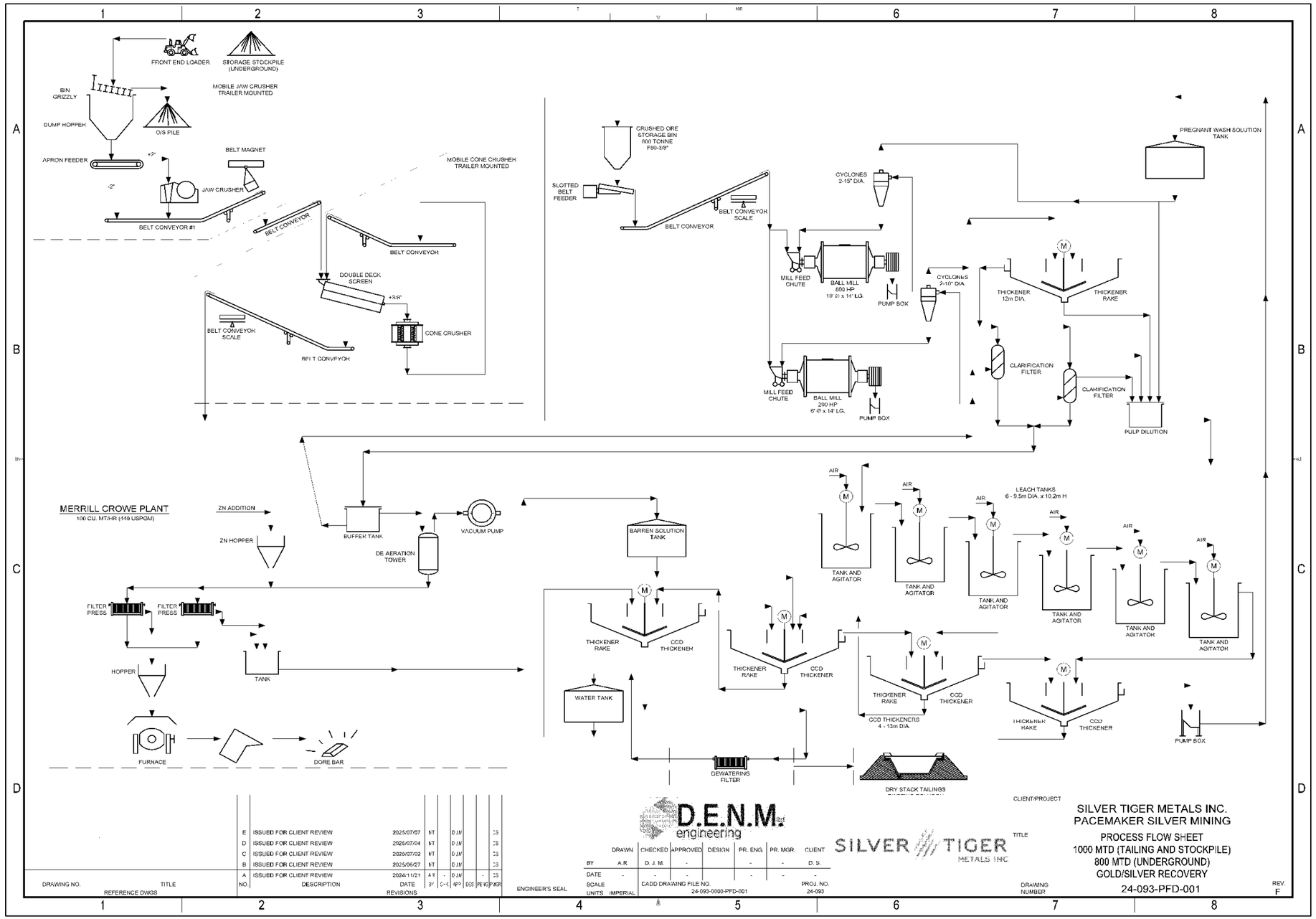
Make-up water for reagent mixing and general process requirements will be pumped to the facility from two surface wells. Power to the EL Tigre Property will be supplied by national grid via a 72 km 34.5 kV overhead power line. Infrastructure details are provided in Section 24.3.6 of this Technical Report.

Unit operations and support facilities include the following:

- Tailings, stockpile, and underground mineralization receiving;
- Two stage mobile crushing circuit with screens;
- Fines bin and mill feed conveyor system;
- Two closed circuit ball mills with cyclones;
- Pre-leach thickener;
- Six stage cyanidation leach circuit;
- Four stage CCD thickeners and solution recovery;
- Complete Merrill Crowe and refinery circuit;
- Dewatering filter system for dry stack;
- Reagent preparation facilities;
- Fresh, recycle, pregnant, barren and pumping distribution solution tanks;
- Metallurgical laboratory; and
- Utilities.

Figure 24.44 illustrates a simplified process flowsheet ("PFD") for the El Tigre process facility.

FIGURE 24.44 SIMPLIFIED FLOWSHEET (PFD)



Source: *D.E.N.M. (2025)*

24.3.5.2 Process Plant Design

Design Criteria

The El Tigre process facility is designed to treat silver-gold bearing mineralized material at a nominal rate of 800 tpd for years one through 14 and at a nominal rate of 1,000 tpd during the pre-production period and final years 15 through 17. The preliminary key process design criteria are shown in Table 24.32.

TABLE 24.32
PROCESS DESIGN CRITERIA

Criteria	Units	Value
Recovery Objectives		
Total Au Recovery Tailings	%	82
Total Ag Recovery - Tailings	%	83
Total Au Recovery - Stockpile	%	92
Total Ag Recovery - Stockpile	%	71
Total Au Recovery - Underground	%	89.7
Total Ag Recovery - Underground	%	84.4
Zone Characteristics		
Tailings Average grade - SILVER	Ag g/t	78
Tailings Average grade - GOLD	Au g/t	0.27
Stockpile Average grade - SILVER	Ag g/t	176
Stockpile Average grade - GOLD	Au g/t	0.85
Underground Average grade - SILVER	Ag g/t	293
Underground Average grade - GOLD	Au g/t	1.05
Average SG	-	2.49
Bulk Density - Crushed	t/m^3	1.90
Bulk Density - Milled	t/m^3	2.39
Moisture in Feed	%	3
Bond ball work index (metric) - Tailings/Stockpile	kWh/t	17.9
Bond ball work index (metric) – Underground avg. (BS and Vein)	kWh/t	20.8
Process Plant Throughput		
Nominal processing rate - Tailings / Stockpile	tpd	1,000
Nominal processing rate - Underground	tpd	800
Operating days per year	dpa	350
Crushing Feed Hopper		
Feed Hopper Capacity	hr	2.0
Storage Capacity, live	t	83.3
Primary Fines Bin		
Feed Storage Requirement	hr	18.0

TABLE 24.32 PROCESS DESIGN CRITERIA		
Criteria	**Units**	**Value**
Storage Capacity, live	t	750
Primary Ball Mills		
Operating Days Per Week	d	7.00
Total Operating Utilization	%	92.0
Average Operating Hours Per Operating Day	hr	22.1
Feed Size - Passing (P_{80})	mm	1
Final Product Size - Passing (P_{80})	microns µm	75
Average Throughput	t/h	45.3
Average Throughput	t/h	36.2
Grinding Mill Solids Density	%w/w	75
Proportion Circulating Load	% dwb	250
Cyclone Feed Density	%w/w	35
Cyclone O/F Solids Density	%w/w	31
Cyclone Feed Density	%w/w	35
Cyclone O/F Solids Density	%w/w	31
Pre-Leach Thickener, with Flocculant		
Flocculation Time	min	10
Thickener Sizing Criteria	$t/m^2/h$	0.40
Mass Flowrate	t/h	45.29
Thickener Area	m^2	113.22
Minimum Diameter	m	12.0
Thickener O/F TSS	mg/L	34.0
Thickener U/F Density	% solids	54
Dilution and Leaching		
Diluted Slurry to Leaching	% solids	50
Leach Slurry		
Leaching time	hours	48
Leaching Volume	m^3	3,771.6
Leach Tank Diameter	m	9.5
PLS Gold tenor	mg/L	0.15
PLS Silver tenor	mg/L	63.4
CCD Thickeners		
Operating Days Per Week	d	7
Total Operating Utilization	%	100.0
Average Operating Hours Per Operating Day	h	24
CCD U/F Density	% solids	47
Number of CCDs	count	4
CCD Sizing Criteria	$t/m^2/h$	0.40

TABLE 24.32 PROCESS DESIGN CRITERIA		
Criteria	**Units**	**Value**
Solids Mass Flowrate	t/h	45.29
CCD area	m^2	113.22
Minimum Diameter	m	12.0
Chemical Consumptions		
Cyanide (NaCN) Application	kg/t	4.00
Cyanide (NaCN) Consumption – Tailings / Stockpile	kg/t	1.40
Cyanide (NaCN) Consumption – Underground	kg/t	3.90
Cyanide Make-up Tank Capacity	hours	24.00
Barren Tank Residence Time	hours	0.50
Lime (CaO eq) Application	kg/t	1.40
Lime (CaO eq) Consumption – Tailings / Stockpile	kg/t	1.40
Lime (CaO eq) Consumption – Underground	kg/t	2.00
Lime Hopper Capacity	days	7.00
Lime Mix-Tank Capacity	days	0.25
Lime Batch Make-up Duration	hours	3.00
Lime Stock-Tank Capacity	days	2.00
Reagent Compositions		
Barren (Leaching) Cyanide Solution Strength	g/t dwb	2.0
Lime Slurry Strength	% solids	20
Tails Pond and Reclaim		
Tails Pond (Dewatered) Capacity	m^3	275,000
Tails Final after Dewatering / Dry Stack	% solids w/w	78
Water and Capacities		
Process Water Tank Capacity	days	0.39
Fresh Water Tank Capacity	days	1

Source: *D.E.N.M. (2025)*

Note: *P_{80} = passing 80%, CCD = counter current decantation circuit, O/F = overflow, U/F = underflow, TSS = total suspended solids.*

Operating Schedule and Availability

The El Tigre process plant is designed to operate for two 12-hour shifts per day, 360 days per year. Utilization expected is 60% for the crushing circuit and 92% for the grinding, leaching, and Merrill Crowe. These factors allow for sufficient downtime for maintenance, both scheduled and unscheduled, within the crushing and processing areas. It also allows flexibility for the processing strategy of three materials over the LOM.

24.3.5.3 Process Plant Description

The proposed process flow diagram ("PFD") incorporates historical tailings material loaded into haul trucks via a front-end loader and delivered to the process facility located within a short distance. The hauled materials ensure a supply of stockpiled historical tailings at the process facility to be processed on a campaign basis. The stockpile material is trucked approximately 1 km downhill to the process facility while the underground material is hauled down approximately 2.6 km from the Level 7 adit to the process area stockpile.

Major process plant equipment includes:

- Mobile Portable Crushing System;
- Front-end Hopper, Feeder, Oversize Screen, and Feed Bin;
- 10.5 ft diameter x 14 ft Long Modular Ball Mill: 800 HP complete with Closed Circuit 15 inch diameter Cyclones and Feed Pumps;
- 6 ft diameter x 14 ft Long Modular Ball Mill: 200 HP complete with Closed Circuit 10 inch Cyclones and Feed Pumps;
- 12 m diameter Pre-Leach Thickener;
- Six 9.5 m diameter x 10.2 m High Leach Tanks: complete with 50 HP Drive Mixers;
- Four 13 m diameter Counter Current Decantation Thickeners;
- Dewatering Filter and Discharge Conveyor System;
- 120 m^3/hr Merrill Crowe System;
- Refinery;
- Solution Tanks, Fresh Water Tanks and Pumping;
- Reagent Mixing Systems: Lime (hydrated) bulk, Sodium Cyanide (NaCN), Dry Super Sacs, Zinc Dust, Diatomaceous Earth (Precoat), Refinery Slagging Reagents, and Flocculant;
- Modular Assay Metallurgical Laboratory;
- Office Buildings;
- Electrical Distribution System: Emergency Generator for Shutdown or System Failure; and
- Air Supply to Plant and Merrill Crowe Systems.

24.3.5.4 Site Location and Details

Images of the historical tailings impound area are shown in Figure 24.45 and Figure 24.46, and Figure 24.47 and Figure 24.48 show the proposed process plant location. The proposed site layout for the El Tigre process plant is presented in Figure 24.49.

FIGURE 24.45 AERIAL VIEW OF EXISTING TAILINGS IMPOUNDMENT

Source: *D.E.N.M. (2023)*

FIGURE 24.46 EXISTING TAILINGS IMPOUNDMENT MINERAL RESOURCE

Source: *D.E.N.M. (2023)*

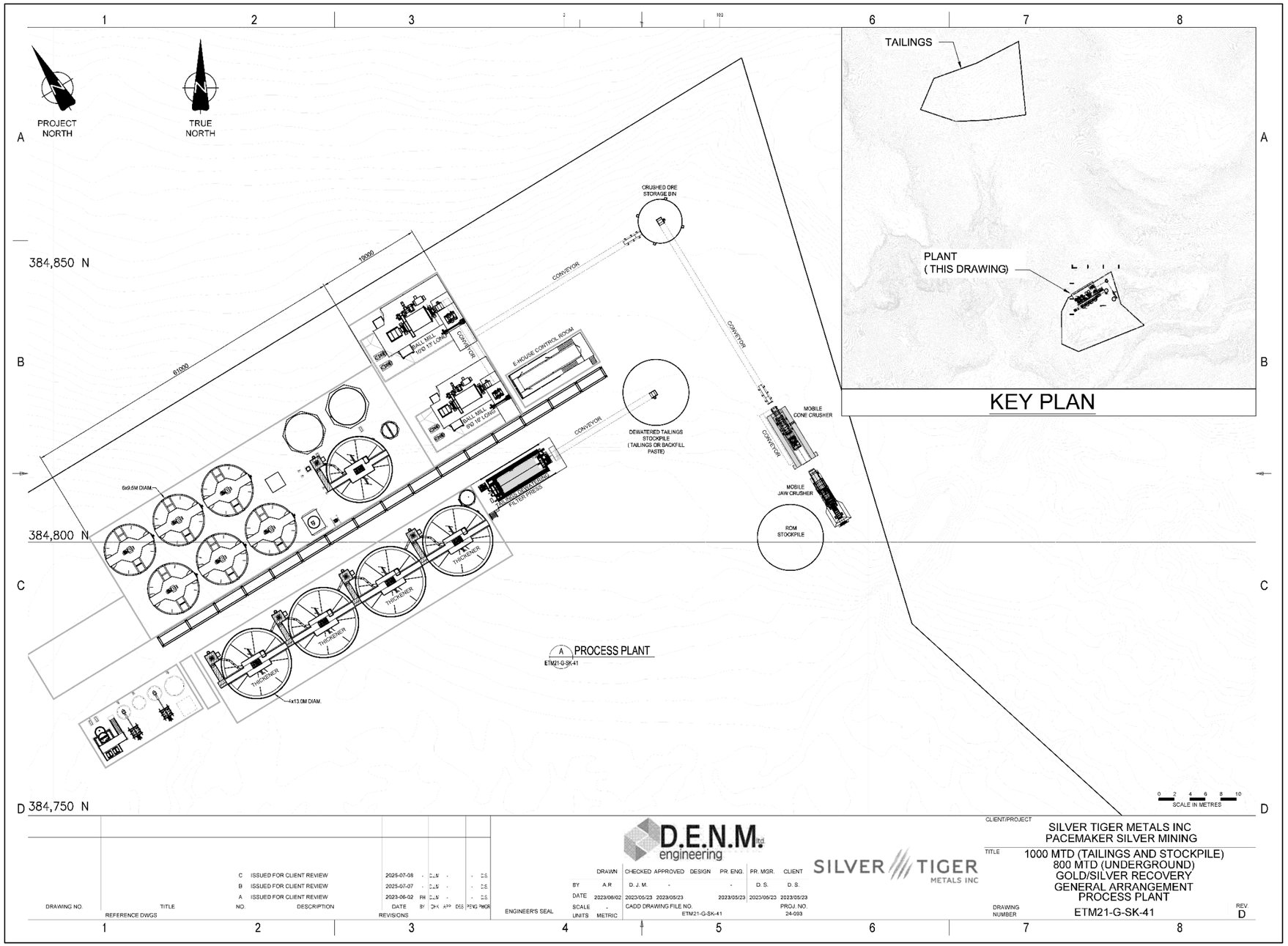
FIGURE 24.47 PROPOSED PROCESS PLANT LOCATION WITHIN PERMITTED POLYGON

Source: *D.E.N.M. (2024)*

FIGURE 24.48 PROPOSED PROCESS PLANT LOCATION

Source: *D.E.N.M. (2024)*

FIGURE 24.49 PROPOSED PROCESS PLANT SITE LAYOUT



Source: *D.E.N.M. (2025)*

24.3.6 Project Infrastructure

Current infrastructure at the El Tigre Project, road access and details of planned electrical power to the Property to be supplied by the national grid via a 72 km 34.5 kV overhead power line, are noted in Section 18 of this Technical Report.

Items specific to the 2025 PEA are noted below.

24.3.6.1 Planned Infrastructure

Figure 24.50 shows the major surface infrastructure proposed at the El Tigre Project, which consists of:

- Portal for Level 7 access to the underground mine workings;
- Low grade mineralization stockpile and waste rock storage facilities near the portal;
- Site access roads;
- Locations of ventilation raises and paste backfill plant;
- Process plant and primary jaw crusher;
- Dry stack tailings storage facility;
- Historical tailings area; and
- Electrical power supply from the national grid.

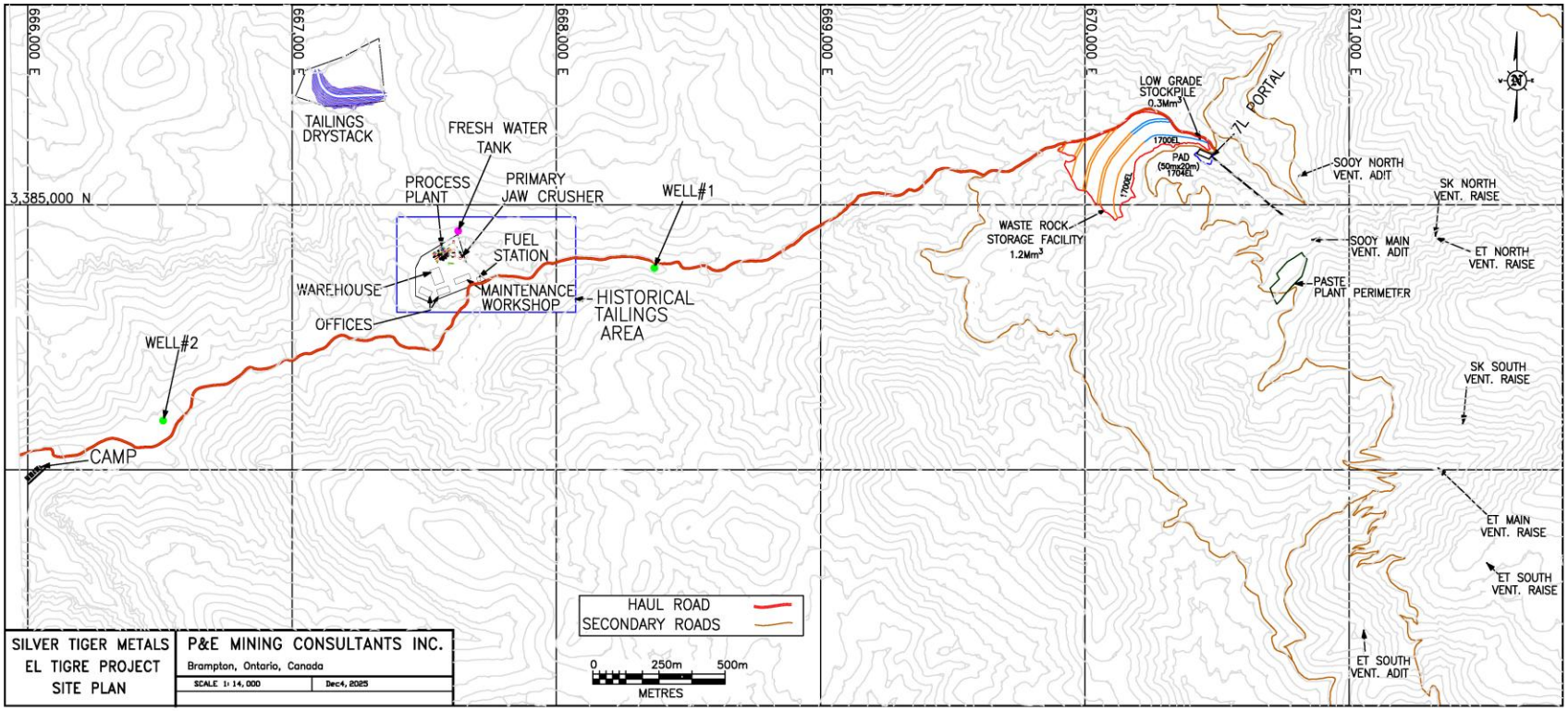
Additional infrastructure to be installed includes:

- Gatehouse and security on the main access road;
- Main office for administration, purchasing, safety and technical personnel;
- Warehouse for all mechanical and process plant parts;
- Fuel storage and distribution facility to be installed by fuel supplier;
- Maintenance workshop to be installed by mining contractor;
- Communications - telephone, cellular, and internet;
- Water wells, storage, potable and sewage treatment plants;
- Secure explosives storage facility; and
- 300-person camp to house all site personnel.

Water needed for the process plant will be supplied by two wells. It is estimated that two wells will be able to meet the process plant freshwater requirements and that water will not need to be pumped from the Bavispe River. The water system will include a main freshwater tank to supply process water and any needed firefighting requirements. Water will also be available from the dry stack tailings collection pond. Additional water is available via the underground workings if needed, however, further testing is required to confirm quality and volume. Water from the underground workings may have mild contamination and would need simple lime and flocculant treatment using mix tanks with settlement in a multi-stage lined basin.

Potable water required for the Project's infrastructure will be supplied by a water filtration treatment system.

FIGURE 24.50 PROPOSED SURFACE INFRASTRUCTURE AT THE EL TIGRE PROJECT



Source: *P&E (2025)*

24.3.6.2 Waste Rock and Low-Grade Storage Facilities

Waste rock will be placed in a storage facility located near the Level 7 portal by truck dumping and dozer pushing with little compaction effort. Stacking is assumed to occur in angle of repose lifts, in a benched sloped configuration of 25 m high and 10 m wide crests at 1.5H:1V batter slopes, to achieve an overall slope of 1.9H:1V. The facility will hold 1.2 Mm^3 of material. For closure conditions, the overall slope would require regrading to achieve flatter slopes for revegetation, and include contour drain construction on any remaining benches tied to downslope drains constructed to deliver rainfall runoff to the slope toe.

Low grade mineralized material will be placed adjacent to the waste rock facility and is planned to contain 0.3 Mm^3 of material. The low-grade facility will be constructed in the same manner as the waste rock facility.

24.3.6.3 Dry Stack Tailings Facilities

Knight Piésold Consulting ("KP") prepared a report titled "Conceptual Filtered Tailings Development and CAPEX" dated July 4, 2025, to summarize a conceptual design and preliminary cost estimate for a filtered tailings storage facility for the El Tigre Project, prepared as an initial alternative based on KP's experience from similar projects. The conceptual assessment did not include site-specific geotechnical or hydrological investigations. Characterization campaigns and further civil, geotechnical, and hydraulic engineering studies will be required to validate or adjust the concepts before implementation.

Two alternatives were considered:

- Alternative 1 initial storage volume of 275,280 m^3 within the currently permitted tailings area; and
- Alternative 2 sustaining storage volume of 565,000 m^3.

Design criteria were:

- Tailings density of 1.43 t/m^3;
- Slope (tailings stack) of 4H:1V;
- Slope (embankment) of 2H:1V;
- Geomembrane thickness of 2 mm HDPE; and
- Surface water management provided by ditches and berms.

24.3.7 Market Studies and Contracts

Metal prices are based on an approximate average of December 31, 2025, three-year monthly trailing averages, and Consensus Economics Inc. long term price forecasts, and are presented in Table 24.33. The Mexican Peso:US Dollar exchange rate is based on the approximate past three-year average. The metal prices and exchange rate are subject to spot market conditions. There are no metals streaming or hedging agreements in place.

TABLE 24.33 METAL PRICES AND EXCHANGE RATE	
Item	**Price**
Gold (US$/oz)	3,200
Silver (US$/oz)	38.00
Exchange Rate ($MXN:US$)	20

Note: *$MXN = Mexican peso, US$ = United States dollars.*

Currently there are no contracts in place that are material to the El Tigre Project.

24.3.8 Capital and Operating Costs

All costs are presented in Q4 2025 US dollars. No provision has been included in the cost estimates to offset future escalation. A contingency of 15% is added to all capital costs ("CAPEX"). No contingency is added to operating costs ("OPEX").

The total initial CAPEX of the El Tigre 2025 PEA is estimated at $83.5M including $10.9M in contingency costs over an expected two-year build. Sustaining capital costs incurred during the ~17 production years are estimated to total $213.6M. Total OPEX over the life-of-mine ("LOM") is estimated at $595.6M, averaging $118.57/t processed. The following subsections provide details of these costs.

24.3.8.1 Capital Costs

Total capital costs over the LOM are estimated at $297.1M and are presented in Table 24.34. Initial capital costs are estimated at $83.5M and sustaining capital costs are estimated at $213.6M. Pre-production revenue of $29.2M is included as a credit against initial capital costs.

TABLE 24.34 2025 PEA CAPITAL COST SUMMARY			
Item	**Initial ($k)**	**Sustaining ($k)**	**Total ($k)**
Process Plant Directs	22,298	-	22,298
Process Plant Indirects	8,811	-	8,811
Underground Mining	49,518	173,508	223,026
Infrastructure	6,449	-	6,449
Dry Stack Tailings	3,978	12,230	16,208
Owner's Costs	10,743	-	10,743
Pre-production revenues[2]	(29,175)	-	(29,175)
Subtotal	**72,622**	**185,738**	**258,360**
Contingencies @ 15%	10,893	27,861	38,754
Total[1]	**83,515**	**213,599**	**297,114**

[1] *Totals may not sum due to rounding.*

[2] *Revenue is generated during the pre-production period from processing historical stockpiled mineralized material.*

Initial Capital Costs

Process Plant Direct Costs

Total process plant direct capital costs are estimated at $22.3M and are presented in Table 24.35.

TABLE 24.35
PROCESS PLANT DIRECT CAPITAL COSTS

Area	Description	Cost ($k)
99	Civil Process Plant	1,365
100	Crushing Circuit	1,185
200	Grinding Circuit	1,795
300	Cyanidation – CCD Circuit	5,312
400	Merrill Crowe Circuit	2,508
500	Tailings – Dewatering Circuit	2,110
600	Reagent Circuit	678
700	Water and Reclaim System	1,901
800	Assay Laboratory	938
850	Generator Power, MMC and PLC System	2,006
875	Mechanical Installation	2,500
Total Direct Costs		**22,298**

Source: *D.E.N.M. (2025)*

Process Plant Direct Construction Capital Costs

Process plant direct capital costs are comprised of total direct equipment costs estimated at $12.8M and total direct construction costs estimated at $9.5M. Direct construction capital costs are primarily associated with the following expenditures:

- Process plant area excavation for civil requirements;
- Process pad grade slab and retaining walls and supports;
- Process equipment mechanical installation;
- Process equipment electrical installation and MCC controls; and
- Process and infrastructure water supply system.

Anticipated construction timeline for the El Tigre process plant is estimated at 18-months. This is consistent with similar projects of size, location and access.

Process Plant Indirect Costs

Factored costs were applied to freight, capital spares, start-up and commissioning as well as first fills for total of 19% of the fixed cost and estimated $5.9M. Factored costs were also used in the process EPCM indirect costs. Engineering and procurement used 5.5% and construction and project management used 7.5% for a total of 13% of the fixed cost and an estimated $2.9M.

Table 24.36 details process plant indirect capital costs for the El Tiger Project.

TABLE 24.36
PROCESS PLANT INDIRECT CAPITAL COSTS

Factor (%)	Description	Cost ($k)
8	Freight	1,025
3	Field Indirect	384
3	Start-Up and Commissioning	384
3	Capital Spares	384
2	First Fills	256
	Commissioning in Year -1 Stockpile Feed	3,478
	Sub-Total	**5,912**
5.5	Engineering and Procurement	1,226
7.5	Construction Management	1,672
Total Process Plant Indirect Costs		**8,811**

Source: *D.E.N.M. (2025)*

Underground Mining Capital Costs

Total capital costs for the underground mine are estimated at $223.0M before contingency. Initial capital costs total $49.5M and sustaining capital costs total $173.5M, as shown in Table 24.37.

TABLE 24.37
CAPITAL COST SUMMARY FOR UNDERGROUND MINE

Area	Initial CAPEX ($M)	Sustaining CAPEX ($M)	Total CAPEX ($M)[1]
Lateral Development	19.4	122.4	141.8
Vertical Development	2.2	13.4	15.6
Backfill Plant & Delivery System	7.3	0.3	7.6
Services	9.6	20.7	30.3
Materials Handling System	0.8	8.5	9.3
Other CAPEX	3.9	8.2	12.1
Other Capitalized OPEX	6.3	0	6.3
Total[1]	**49.5**	**173.5**	**223.0**

Note: *[1] Totals may not sum due to rounding.*

Initial Underground Capital Costs

This subsection addresses the capital cost estimate for engineering, procurement and start-up costs of the UG mine at the El Tigre Project. The UG is expected to be mined with contractors, reducing the overall capital costs versus owner operation. Costs for development were based on a budgetary quote from a major national UG mining contractor with numerous Mexican projects. Costs for production were based on first-principles calculations and benchmarked against recent studies and quotes from national contractors for other similar operations.

Total initial CAPEX cost for the underground is estimated at $49.5M, exclusive of a contingency, as shown above in Table 24.37.

Underground Development

A total of 5.7 km of lateral development and approximately 0.7 km of vertical development will be driven during the pre-production period in YR -1. No operating development is required during the pre-production period, however, some capital development does intersect sufficient mineralization to generate material to be included in the planned feed for the process plant. This material is stockpiled until the start of YR 1.

Table 24.38 provides a summary of development quantities, cost per heading size, and total costs incurred during the pre-production period for development.

TABLE 24.38
INITIAL DEVELOPMENT CAPITAL COSTS

Heading Type	Heading Profile (m W x m H)	Required Development (m)	Development Cost ($/m)	Total CAPEX ($M)[1]
Vent Raise (Drop Raise)[2]	3.0 x 3.0	203	1,882	0.4
Vent Raise (Raisebore)	3.4 dia.	532	3,483	1.9
Mineral Pass (Raisebore)	2.7 dia	-	-	-
Subtotal – Vertical Development[1]		**734**	**3,041**	**2.2**
Ramp	4.5 x 4.5	3,310	3,511	11.6
Level Access		749	3,511	2.6
Sublevel Ramp		-	-	-
Mineral Pass Transfer		269	3,397	0.9
Remuck Bay		387	3,121	1.2
Ventilation Access	4.0 x 4.0	468	3,051	1.4
Footwall Drift		60	3,085	0.2
Electrical Bay	3.5 x 3.5	225	2,841	0.6
Sump/Pump Station		263	2,841	0.7
Mineral Pass Access		-	2,773	-

TABLE 24.38 INITIAL DEVELOPMENT CAPITAL COSTS				
Heading Type	**Heading Profile (m W x m H)**	**Required Development (m)**	**Development Cost ($/m)**	**Total CAPEX ($M)[1]**
Subtotal – Lateral Development[1]		**5,731**	**3,380**	**19.4**
Total[1]		**6,465**	**3,342**	**21.6**

Notes: *[1] Totals may not sum due to rounding.*
[2] Drop raise profile dimensions are in mW x mL.

Backfill

The backfill plant for the El Tigre UG mine is planned to be located roughly centrally to the Deposit on a ridge above the elevation of the highest stopes in the mine and approximately 50 m from the Planned OP limit. It is planned to produce 30 m^3/hr of PF over 16 operating hours per day. The backfill plant is planned to utilize a combination of direct tailings feed from the process plant, and dry-stack tailings from the tailings storage facility.

Backfill distribution within the mine is planned to use 76 and 102 mm diameter pipelines. Larger diameter steel pipelines will be run in capital development areas (ramps, level accesses and FWDs) and smaller diameter HDPE pipelines will be run in operating development areas (sills, crosscuts and C&F/D&F accesses).

Total initial CAPEX for the backfill plant and distribution system is estimated at $7.3M, pre-contingency. The PF plant alone makes up $7.0M of this total.

Services

Underground services initial CAPEX includes:

- Ventilation fans and doors;
- Electrical transformers and substations;
- Dewatering and service water pumps;
- Compressors and accumulators; and
- Communications systems.

Total initial CAPEX for services infrastructure, pre-contingency, is estimated at $9.6M.

Materials Handling System

Materials handling system initial CAPEX includes:

- Bins;
- Liners;
- Chutes; and
- Grizzlies.

It is exclusive of the cost of developing transfers, access and waste/mineral passes: these costs are included in the CAPEX cost for development.

Total initial CAPEX for the materials handling system, pre-contingency, is estimated at $0.8M.

Other CAPEX

Other initial CAPEX includes:

- Fitment of magazines, refuge stations, planned maintenance ("PM") bays, pump stations and escapeways;
- Construction of UG/surface interfaces (portals and ventilation raise collars), including roads to ventilation raises and the paste backfill plant;
- Contractor mobilization; and
- Technical services software purchases, survey equipment and pickup trucks.

Total initial CAPEX in this category, pre-contingency, is estimated at $3.9M.

Other Capitalized Operating Costs

Other capitalized OPEX costs include:

- Delineation drilling;
- Technical services and Company mine supervision staff salaries;
- Consultants;
- UG dayworks and sundries; and
- Electrical power.

Items included in this category would normally be classified as OPEX, however, are categorized as CAPEX in YR -2 and YR -1 since costs are incurred prior to the start of production. The total cost of these items, pre-contingency, is estimated at $6.3M.

Infrastructure Capital Costs

Infrastructure capital costs are estimated at $6.4M as presented in Table 24.39 and include utilities, access roads, site buildings, camp facilities, water and sewage treatment. Estimates have been provided by Mexican vendors and from Mexican mining operations that are familiar to the Authors.

TABLE 24.39 INFRASTRUCTURE CAPITAL COSTS	
Infrastructure Item	**Cost ($k)**
Gatehouse	15
Warehouse	85
Main Office Building	90
Service Buildings (Dining, Clinic, Laundry, Food Storage)	196
300-Person Camp Facilities (Procsa)	930
Upgrade Main Access Road	500
Site Roads - Upgrade	200
Contractor Laydown Prep	50
Fire Protection	100
Two Water Supply Wells	365
Powerline and Right of Way (CFE)	2,308
Substation and Power Distribution (Remogal)	1,003
Communications	250
Potable Water Treatment (Hidrotec)	6
Sewage Treatment (Hidrotec)	51
Access Bridge	300
Total	**6,449**

Notes: *Procsa = PROCSA Ingenieria Electrica Y Sistemas de Aire S.A. de C.V.*
CFE = Comision Federal de Electricidad (Mexican Federal Electrical Commission).
Remogal = REMOGAL Control & Automatizacion.
Hidrotec = HIDROTEC Solucion Integral al Problema del Agua.

Dry Stack Tailings Costs

Initial capital costs for dry stack tailings are based on Alternative 1 estimates by Knight Piesold Consulting as described in Section 24.3.6 of this Technical Report. A total of $4.0M is comprised of:

- Direct Costs (including allowances) $3.3M;
- Indirect Costs (EPCM, QA, etc.) $0.4M; and
- Owner's Costs $0.3M.

Owner's Costs

The estimate of $10.7M pre-production Owner's costs over a two-year period are presented in Table 24.40 and include general and administrative expenses, camp services, one year of underground construction technical team, concession payments, insurance and employee travel.

TABLE 24.40 PRE-PRODUCTION OWNER'S COSTS	
Item	**Cost ($k)**
G&A and Construction Supervision Labour	2,380
G&A Expense Items – Onsite Including Camp Services	5,260
Satellite Offices & Off-Site Warehousing Including Concession Payments	2,923
Employee Travel - Residence to Site	180
Total	**10,743**

Pre-Production Revenues

During the pre-production period, 136,900 t of mineralized material from surface stockpiles will be processed during the commissioning of the process plant. This material is estimated to generate revenues of $29.2M during YR -1, which will be used to offset capitalized OPEX costs and capital spending on development in the UG mine.

Initial Capital Cost Contingency

Contingency allowance for all initial capital costs is estimated at 15% of net pre-production expenditures (costs less revenues). For the pre-production period, this allowance is estimated at $10.9M.

Sustaining Capital Costs

As set out in Table 24.34 above, sustaining capital costs are estimated to total $213.6M over the LOM.

Underground Sustaining Costs

Sustaining capital costs in the UG mine are those capital costs which are incurred during the operational life of the Project. These costs include, however, are not limited to:

- Contractor mob/demob;
- Underground capital development; and
- Underground fixed plant capital.

Total sustaining CAPEX cost for the underground is estimated at $173.5M, pre-contingency, as noted in Table 24.37 above.

Development

Sustaining CAPEX development costs of $135.8M are shown in Table 24.41.

TABLE 24.41 SUSTAINING DEVELOPMENT CAPITAL COSTS				
Heading Type	**Heading Profile (m W x m H)**	**Required Development (m)**	**Development Cost ($/m)**	**Total CAPEX ($M)[1]**
Vent Raise (Drop Raise)[2]	3.0 x 3.0	1,146	1,882	2.2
Vent Raise (Raisebore)	3.4 dia.	2,165	3,483	7.5
Mineral Pass (Raisebore)	2.7 dia	1,550	2,410	3.7
Subtotal – Vertical Development[1]		**4,861**	**2,763**	**13.4**
Ramp	4.5 x 4.5	13,307	3,511	46.7
Level Access		13,354	3,511	13.9
Sublevel Ramp		1,860	3,085	5.7
Mineral Pass Transfer		512	3,397	1.7
Remuck Bay		1,781	3,121	5.6
Ventilation Access	4.0 x 4.0	2,966	3,051	9.0
Footwall Drift		9,397	3,085	29.0
Electrical Bay	3.5 x 3.5	1,187	2,841	3.4
Sump/Pump Station		1,375	2,841	3.9
Mineral Pass Access		1,232	2,773	3.4
Subtotal – Lateral Development[1]		**37,575**	**3,257**	**122.4**
Total (Pre-contingency)[1]		**42,436**	**3,201**	**135.8**

[1] Totals may not sum due to rounding. *[2] Drop raise profile dimensions are in mW x mL.*

Backfill

Sustaining CAPEX for the backfill system is estimated at $0.3M over LOM, pre-contingency.

Services

Sustaining CAPEX for ventilation, electrical, dewatering and compressed air systems is estimated at $20.7M, pre-contingency. Slightly more than half of these expenditures are for the expansion of the electrical distribution system for the UG mine.

Materials Handling System

Sustaining CAPEX for the materials handling system is estimated at $8.5M, pre-contingency. This price includes the necessary expansion of the materials handling system, as well as the replacement of the bins and chutes in Zone 12 once during LOM.

Other Capex

Other sustaining CAPEX items are estimated at $8.2M, pre-contingency.

Dry Stack Tailings Sustaining Costs

Sustaining capital costs for dry stack tailings are based on Alternative 2 estimates provided by KP as described in Section 24.3.6 of this Technical Report. With a portion of tailings being utilized as underground backfill, total LOM tailings storage capacity is equivalent to approximately three times the capacity of the Alternative 2 design. Alternative 2 is estimated by KP at $5.4M. Dry stack tailings sustaining costs are calculated as 3 x $5.4M minus initial CAPEX of $4.0M, equivalent to $12.2M over the LOM.

Sustaining Cost Contingency

Contingency allowance for all sustaining capital costs is estimated $27.9M (15% of total sustaining CAPEX costs).

24.3.8.2 Operating Costs

The operating costs estimate includes the cost of mining, haulage, processing, and General and Administration ("G&A") services. The LOM average operating costs for the Project during production years are estimated at $118.57/t processed and are summarized in Table 24.42.

TABLE 24.42
2025 PEA OPERATING COST SUMMARY

Area	Unit Cost ($/t processed)	LOM Cost ($k)
Processing UG Feed	41.55	166,193
Processing Tailings/Stockpile Feed	24.25	21,489
Subtotal Processing[1]	**37.37**	**187,681**
Underground Mining	76.80	307,200
Haulage UG Portal to Process Plant, Return with Tailings	1.33	6,672
Subtotal Mining and Hauling[2]	**62.49**	**313,872**
General and Administration[3]	18.71	94,000
Total	**118.57**	**595,553**

Note: *1 Processing + 2 Mining and Hauling + 3 G&A = Total.*

Process Plant

Table 24.43 and Table 24.44 summarize the estimated process plant operating costs for processing 800 tpd UG feed and 1,000 tpd tailings/stockpile feed. The costs do not include mining support or general personnel and are based on the following:

- FX Rate: 1 US$, 19 $MXN;
- CFE Powerline: US$0.11/kWh;
- Bulk Diesel: US$1.19/L;

- Mexico Labour Burden Rate: 37%;
- 800 tpd and 1,000 tpd Labour: 63 Personnel;
- Labour totals include operations, maintenance, and technical services all related to the process facility; and
- Excludes G&A and mining costs.

TABLE 24.43
UNDERGROUND FEED PROCESSING COST SUMMARY

Item	$M/ Year	$/t Processed	% of Total
Labour	1.6	5.60	14
Power and Fuel	2.4	8.05	19
Maintenance and Operating Consumables	6.7	22.98	55
Dry Stack Tailings	0.9	3.00	7
Support Equipment	0.6	1.92	5
Total	**12.1**	**41.55**	**100**

Source: *D.E.N.M. (2025)*

TABLE 24.44
TAILINGS AND STOCKPILE FEED PROCESSING COST SUMMARY

Item	$M/ Year	$/t Processed	% of Total
Labour	1.6	4.48	19
Power and Fuel	2.1	5.85	24
Maintenance and Operating Consumables	3.4	9.39	39
Dry Stack Tailings	1.1	3.00	12
Support Equipment	0.6	1.53	6
Total	**8.9**	**24.25**	**100**

Source: *D.E.N.M. (2025)*

Process Plant Personnel

Table 24.45 summarizes the process plant personnel at both 800 tpd and 1,000 tpd production rates, including positions, base salary and burden rate. Staff are scheduled on a 7&7 day rotation and hourly personnel are scheduled on a 4&3 day rotation. All personnel are scheduled at 2,184 hours of work per year. Total costs are expressed in both US dollars and Mexican Pesos.

TABLE 24.45
PROCESSING PLANT PERSONNEL

Position	Personnel No.	Staff/ Hourly	Base Pay ($MXN)	Burden Rate 37% Base Pay ($MXN)	Total Per Employee ($MXN)	Total Annual Cost ($MXN)	Total Annual Cost ($US)	Unit Cost ($US/t)
Operations								
Mill Operations Superintendent	1	Staff	1,500,000	555,000	2,055,000	2,055,000		
Operations Shift Foreman	4	Staff	1,125,000	416,250	1,541,250	6,165,000		
Crushing Plant Operator	4	Staff	180,000	66,600	246,600	986,400		
Ball Mills	4	Hourly	180,000	66,600	246,600	986,400		
CN and CCD Operators	8	Hourly	180,000	66,600	246,600	1,972,800		
Filtering Operator	4	Hourly	180,000	66,600	246,600	986,400		
Reagent Helpers/Operator	4	Hourly	180,000	66,600	246,600	986,400		
Mill Labourer	6	Hourly	120,000	44,400	164,400	986,400		
Subtotal	**35**						**796,042**	**2.18**
Maintenance								
Maintenance Superintendent	1	Staff	1,125,000	416,250	1,541,250	1,541,250		
Mill Maintenance Foreman	2	Staff	720,000	266,400	986,400	1,972,800		
Electrician	3	Hourly	360,000	133,200	493,200	1,479,600		
Instrumentation Technician	3	Hourly	450,000	166,500	616,500	1,849,500		
Millwright	2	Hourly	180,000	66,600	246,600	493,200		
Welder	3	Hourly	180,000	66,600	246,600	739,800		
Subtotal	**14**						**425,061**	**1.16**
Technical Services								
Chief Assayer	2	Staff	800,000	296,000	1,096,000	2,192,000		

TABLE 24.45 PROCESSING PLANT PERSONNEL								
Position	**Personnel No.**	**Staff/ Hourly**	**Base Pay ($MXN)**	**Burden Rate 37% Base Pay ($MXN)**	**Total Per Employee ($MXN)**	**Total Annual Cost ($MXN)**	**Total Annual Cost ($US)**	**Unit Cost ($US/t)**
Assay Technician	6	Hourly	450,000	166,500	616,500	3,699,000		
Laboratory Technician	6	Hourly	240,000	88,800	328,800	1,972,800		
Subtotal	**14**						**413,884**	**1.13**
Total	**63**					**31,064,750**	**1,634,987**	**4.48**

Source: *D.E.N.M. (2025)*
Note: *$MXN = Mexican peso, US$ = United States dollars.*

Processing Plant Power and Consumables

Table 24.46 and Table 24.47 summarize estimated costs for power and consumables for the processing plant at 800 tpd and 1,000 tpd, respectively.

TABLE 24.46
PROCESSING PLANT POWER AND CONSUMABLES AT 800 TPD

Item	Amount (kWh/Year)	Cost ($/kWh)	Amount (kWh/t)	Annual Cost ($M)	Cost ($/t)
Power	20,793,418	0.11	71.0	2.29	7.93
Fuel and Plant Consumables				1.09	3.73

Source: *D.E.N.M. (2025)*

TABLE 24.47
PROCESSING PLANT POWER AND CONSUMABLES AT 1,000 TPD

Item	Amount (kWh/Year)	Cost ($/kWh)	Amount (kWh/t)	Annual Cost ($M)	Cost ($/t)
Power	18,769,281	0.11	51.4	2.06	5.66
Fuel and Plant Consumables				0.54	1.48

Source: *D.E.N.M. (2025)*

Process Plant Crushing and Process Consumables

Table 24.48 and Table 24.49 summarize estimated costs for crushing and process consumables including reagents for the processing plant at 800 tpd and 1,000 tpd, respectively.

TABLE 24.48
PROCESSING PLANT CRUSHING AND PROCESS CONSUMABLES AT 800 TPD

Crushing and Process Consumables				Material ($/t)	Annual ($M)
Crusher				0.57	0.17
Mills and Leaching				3.12	0.91
Reagents	**kg/t**	**Tonne /Year**	**Material ($/t)**	**Annual ($M)**	**Annual ($/t)**
Lime (CaO)	2.0	584	220	0.15	0.51
Cyanide*	3.9	1,139	2,260	3.41	11.66
Precoat **		120	1,400	0.19	0.66
Zinc Dust**		53	6,260	0.38	1.29
Flocculant**	0.04	12	5,560	0.08	0.26

Source: *D.E.N.M. (2025)* ** Unoptimized. ** Estimated.*

TABLE 24.49 PROCESSING PLANT CRUSHING AND PROCESS CONSUMABLES AT 1,000 TPD					
Crushing and Process Consumables				**Material ($/t)**	**Annual ($M)**
Crusher				0.11	0.04
Mills and Leaching				0.88	0.32
Reagents	**kg/t**	**Tonne /Year**	**Material ($/t)**	**Annual ($M)**	**Annual ($/t)**
Lime (CaO)	1.4	511	220	0.13	0.35
Cyanide*	1.4	511	2,260	1.53	4.18
Precoat **		120	1,400	0.19	0.52
Zinc Dust**		53	6,260	0.38	1.03
Antiscalent**	0.04	12	5,560	0.08	0.20

Source: *D.E.N.M. (2025)*
** Unoptimized. ** Estimated.*

Underground Mining

The operating cost estimate addresses the costs of all: development (excluding sustaining CAPEX development); production; backfilling; services and power usage; technical services auxiliary staff roles; and other dayworks and sundry costs associated with operating the underground mine. The El Tigre UG Project is expected to be mined with contractors: costs for development were derived from a budgetary quote from a national mining contractor, all other costs were derived from first-principles. All costs used for underground development and production are sourced directly, or are based on these costs.

Total OPEX for the underground mine over the LOM is estimated at $307M, as shown in Table 24.50. The average OPEX per UG tonne processed is $76.80/t and includes operation of the paste backfill plant.

TABLE 24.50 OPERATING COST SUMMARY FOR UNDERGROUND MINE			
Area	**Unit Cost ($/t)**	**Quantity (kt)**	**Total Cost ($M)[1]**
Operating Development	24.08	4,000	96.3
Production	43.94		175.8
Indirects	8.79		35.1
Total[1]	**76.80**	**4,000[2]**	**307.2**

Notes: *[1] Totals may not sum due to rounding.*
[2] Total tonnes of process plant feed sourced from the UG mine.

Development

Operating development costs are shown in Table 24.51. All operating development is lateral development. Total operating development cost per tonne of feed sourced from the UG mine is $24.08.

TABLE 24.51
DEVELOPMENT OPERATING COSTS

Heading Type	Heading Profile (m W x m H)	Required Development (m)	Development Cost ($/m)	Total OPEX ($M)[1]
Sill	3.5 x 3.5	15,862	2,773	44.0
Crosscut		12,796		35.5
Crosscut through Pastefill		343		1.0
Drift and Fill Access	4.0 x 4.0	2,645	4,371[2]	11.6
Cut and Fill Access		1,231	3,511	4.3
Total[1]		**32,877**	**2,929**	**96.3**

Notes: *[1] Totals may not sum due to rounding.*
[2] Inclusive of the cost of paste backfill.

Production

Underground production costs are shown in Table 24.52.

TABLE 24.52
OPERATING COST SUMMARY FOR UNDERGROUND MINE

Area	Unit Cost ($/t)	Total Cost ($M)[1]
Longhole Stoping	17.03	68.1
Cut and Fill Stoping	0.53	2.1
HMW Mining	1.10	4.4
Delineation Drilling	2.73	10.9
Pastefill (0.5 MPa)	5.82	23.3
Pastefill (1.0-1.4 MPa)	2.83	11.3
Power	13.93	55.7
Total[1]	**43.94**	**175.8**

Notes: *[1] Totals may not sum due to rounding.*
HMW = historical mine workings

Other Underground Operating Costs

Other operating costs for the UG Mine (indirect salaries, dayworks and sundries, etc.) total $35.1M over LOM, equivalent to $8.79/t of feed sourced from the UG mine.

Contingency

No contingency allowance has been added to UG operating costs.

Haulage Between Portal and Process Plant

A contractor will load feed from a small stockpile near the underground portal and transport the feed to the process plant with a fleet of 30-t highway trucks. The operating cost for a truck cycle is estimated at $1.16/t. Once the contractor trucks are at the process plant, there is the opportunity to return to the underground mine with dry stack tailings for the backfill plant. The cost for a front-end loader to load dry stack tailings into the contractor trucks is estimated at $0.41/t. The LOM contactor cost is estimated at $6.7M and averages $1.33/t processed.

General and Administration

General and Administration ("G&A") costs are estimated at $6.0M annually, as summarized in Table 24.53. The average cost over the LOM is estimated at $18.71/t processed. The annual wages include burdens.

TABLE 24.53
GENERAL AND ADMINISTRATION COSTS

Item	Number	Annual Cost ($k)
General Manager	1	197.3
General Manager Assistant	1	14.7
Safety Superintendent	1	69.2
Safety Supervisor	3	77.9
Environment Coordinator	1	43.3
Security Coordinator	1	34.6
Purchasing Chief	1	46.9
Buyer	1	26.0
Warehouse Chief	1	38.9
Warehouse Helper	3	44.1
Controller	1	90.9
Accountant Chief	1	38.9
Accountant	1	40.8
Human Resources Superintendent	1	69.2
Human Resources Coordinator	1	30.3

TABLE 24.53 GENERAL AND ADMINISTRATION COSTS		
Item	**Number**	**Annual Cost ($k)**
Community Relations Superintendent	1	62.0
Community Relations Coordinator	1	30.3
Driver	1	18.8
Regional Office and Concessions	Lump sum	1,446.9
Insurance	Lump sum	250.0
Security Services	Lump sum	117.0
Communications	Lump sum	87.4
Community Assistance and Housing	Lump sum	117.0
Office and Outside Services	Lump sum	445.3
Camp Costs	Lump sum	2,562.3
Total[1]		**6,000.0**

[1] *Totals may not sum due to rounding.*

24.3.8.3 Mining Duty

The Project is subject to a 1.0% NSR Mexican government mining duty. Total costs associated with this NSR duty over the LOM are estimated at $16.7M.

24.3.8.4 Closure Costs

Costs to be incurred after the LOM plan were estimated at $4.0M to close and rehabilitate the Project site, as presented in Table 24.54.

TABLE 24.54 CLOSURE COSTS	
Activity	**Closure Cost ($k)**
Road Rehabilitation	649.6
Small Borrows and Excavations	162.5
Waste Rock Storage Facilities	2,076.5
Process Plant, Shops, Warehouse, Buildings	628.8
UG Accesses and Infrastructure	482.6
Total	**4,000.0**

24.3.8.5 Cash Costs and All-in Sustaining Costs

Cash costs over the LOM, including Mexican mining duties, are estimated to average $16.05/oz AgEq or $1,351/oz AuEq. All-In Sustaining Costs ("AISC") over the LOM are estimated to average $23.98/oz AgEq or $2,019/oz AuEq and include closure costs.

24.3.9 Economic Analysis

24.3.9.1 Summary

Cautionary Statement - The reader is advised that the 2025 PEA summarized in this Technical Report is intended to provide only an initial, high-level review of the Project potential and design options. The 2025 PEA mine plan and economic model include numerous assumptions and the use of Inferred Mineral Resources. Inferred Mineral Resources are considered to be too speculative to be used in an economic analysis except as allowed by NI 43-101 in PEA studies. There is no guarantee the Project economics described herein will be achieved.

A discounted cash flow model was prepared using the production schedule described in Section 24.3.4 and the cost estimates described in Section 24.3.8 of this Technical Report. The 2025 PEA cash flow model was developed on a pre-tax and after-tax basis. The cash flow model is assumed to commence from the time a production decision is made and does not include time or costs for additional studies. The underground mine plan and cash flow model are independent of surface mining described in the PFS sections of this Technical Report.

The El Tigre Project economic evaluation conclusions are summarized in Table 24.55. At base case metal prices of US$3,200/oz Au and US$38/oz Ag the Project has an estimated US$304.0M after-tax net present value ("NPV") at a 5% discount rate ("NPV5%"), and an after-tax internal rate of return ("IRR") of 43%. The payback period is estimated to be 2.6 years from the start of production.

TABLE 24.55
2025 PEA ECONOMIC EVALUATION SUMMARY

Item	Pre-Tax	After-Tax
NPV0% ($M)	753.5	495.6
NPV5% ($M)	487.3	**304.0**
NPV7% ($M)	414.5	252.6
IRR (%)	65.2	**42.8**
Payback period (years)	1.8	**2.6**

24.3.9.2 Basic Assumptions

A discounted cash flow analysis of the El Tigre Project was prepared based on technical and cost inputs developed by the Authors.

The discounted cash flow analysis was performed on a stand-alone project basis with annual cash flows discounted. The financial evaluation uses a discount rate of 5%, discounting back to the commencement of construction (Year -2) of the Project.

All currency values are expressed in US dollars unless otherwise noted.

Metal Price Assumptions

All metal prices remain constant throughout the LOM of the Project.

Ag: $US38/oz. The sensitivity of the Project return to variations in the actual silver price received was examined.

Au: $US3,200/oz. The sensitivity of the Project return to variations in the actual gold price received was examined.

Metallurgical Recoveries

The El Tigre Project's process plant cyanide/leach metallurgical recovery assumptions are:

Underground mine	Ag = 84%, Au = 90%
Historical tailings	Ag = 83%, Au = 82%
Stockpiled mineralization	Ag = 71%, Au = 92%

Capital Costs

Total capital costs during the LOM are estimated to be $297.1M including a 15% contingency, as outlined in the Capital and Operating Cost Section 24.3.8. The initial capital costs are incurred over a two-year construction period and are estimated to be $83.5M. Pre-production revenue of $29.2M is included as a credit against initial capital costs. Sustaining costs over the LOM are estimated at $213.6M.

Salvage value was estimated at 15% of the $9.4M process plant mechanical equipment installed over the LOM, equivalent to $1.4M.

Previous Expenses Provision

An amount of $9.2M was considered as a prior expense pool and these monies were deducted from income in production Year 1 when determining the taxable income.

Income Tax Rate

The Mexican income tax is levied at a rate of 30% on the net taxable income. A special duty tax of 8.5% on the net taxable income is also included.

Additional Mining Tax Rate

A Mexican government 1.0% gross revenue mining tax was applied in the cash flow economics and is deductible when determining taxable income.

24.3.9.3 Cash Flow Summary

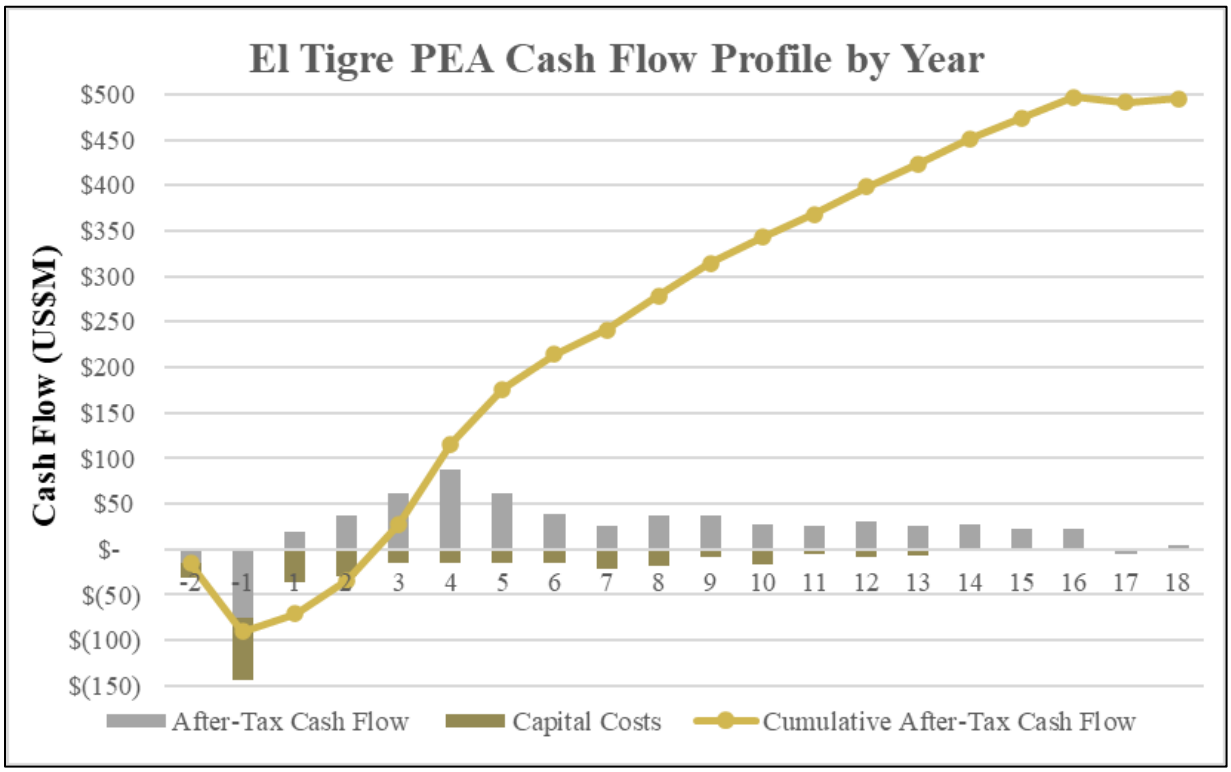
Based on the assumed metal prices the 2025 PEA Project has an after-tax IRR of 43% and a 2.6 year payback of initial pre-production capital costs. The Project is estimated to realize an after-tax NPV of US$304.0M at a discount rate of 5%.

The estimated annual production and LOM cash flows for the El Tigre 2025 PEA are summarized in Table 24.56 and annual cash flows are presented in

Figure 24.51. An annual summary of the cash flow model is provided in Table 24.59.

TABLE 24.56 2025 PEA PROJECT CASH FLOW SUMMARY					
Assumption / Result	**Unit**	**Value**	**Assumption / Result**	**Unit**	**Value**
Feed Processed	kt	5,023	Net Revenue	US$M	1,665.4
Waste Rock Mined	kt	2,439	Initial Capital	US$M	83.5
Silver Grade	g/t	251.3	Sustaining Capital	US$M	213.6
Gold Grade	g/t	0.91	Mining and Haulage Costs	$/t processed	62.49
Silver Recovery	%	84	Processing Costs (UG and Stockpile)	$/t processed	41.55 / 24.25
Gold Recovery	%	90	G&A Costs	$/t processed	18.71
Silver Price	US$/oz	38.00	Operating Cash Cost	US$/oz AgEq	16.0
Gold Price	US$/oz	3,200	All-in Sustaining Cost	US$/oz AgEq	24.0
Payable Silver Metal	Moz	33.6	After-Tax NPV (5% discount)	US$M	304.0
Payable Gold Metal	koz	130	Pre-Tax NPV (5% discount)	US$M	487.3
Payable AgEq	Moz	38.5	After-Tax IRR	%	42.8
Production and Reclamation	years	18	Pre-Tax IRR	%	65.2
Average Feed Processed	tpd	800	After-Tax Payback Period	years	2.6

FIGURE 24.51 2025 PEA ANNUAL CASH FLOW PROFILE



24.3.9.4 Sensitivities

The El Tigre 2025 PEA economics were examined with a sensitivity analysis for several key variables. The results of the sensitivity analyses on the after-tax NPV with a 5% discount rate are shown in Table 24.57 and Table 24.58.

TABLE 24.57
METAL PRICE SENSITIVITY

Sensitivity	-20%	-10%	Base Case	+10%	+20%
Silver Price (US$/oz)	30	34	**38**	42	46
Gold Price (US$/oz)	2,560	2,880	**3,200**	3,520	3,840
After-Tax NPV (5%) (US$M)	153.9	229.1	**304.0**	378.9	453.9
After-Tax IRR (%)	24.8	33.8	**42.8**	51.8	61.0
After-Tax Payback (years)	3.5	3.0	**2.6**	2.2	1.9

TABLE 24.58 CAPITAL AND OPERATING COST SENSITIVITY					
Sensitivity	**-20%**	**-10%**	**Base Case**	**+10%**	**+20%**
Operating Costs – NPV_5 (US$M)	354.9	329.5	**304.0**	278.6	253.2
Operating Costs – IRR (%)	48.2	45.5	**42.8**	40.1	37.4
Capital Costs – NPV_5 (US$M)	337.1	320.6	**304.0**	287.5	271.0
Capital Costs – IRR (%)	55.1	48.4	**42.8**	38.1	34.0

The after-tax base case NPV's and IRR's are most sensitive to metal prices followed by operating costs and capital costs.

24.3.9.5 Comment on Surface and Underground Mining Timing

If the 2025 PEA mine plan was to be implemented after surface mining commenced as detailed in the PFS sections of this Technical Report, there would be a savings in initial capital costs of the 2025 PEA estimate, summarized in Table 24.59, by approximately $17.0M. The savings would consist of infrastructure such as the electrical powerline for grid power, and camp and office facilities, plus much of the owner's costs.

TABLE 24.59
2025 PEA CASH FLOW MODEL SUMMARY

Item	Units	Total	Year																			
			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Process Plant Feed	**kt**	**5,023**	**-**	**137**	**290**	**290**	**290**	**290**	**290**	**290**	**290**	**290**	**290**	**290**	**290**	**290**	**290**	**308**	**365**	**365**	**78**	**-**
Grade Ag	g/t	251.3	-	162.9	294.3	317.1	335.5	388.3	330.9	287.2	282.6	320.0	302.5	280.3	249.5	266.8	237.5	157.5	77.6	77.6	77.6	-
Grade Au	g/t	0.91	-	0.77	1.14	1.76	2.20	3.12	1.95	1.12	0.78	0.75	0.23	0.33	0.25	0.39	0.31	0.27	0.27	0.27	0.27	-
Recovered Ag	kozs	33,574	-	506	2,278	2,455	2,597	3,005	2,561	2,223	2,188	2,477	2,341	2,170	1,931	2,066	1,839	1,290	744	744	160	-
Recovered Au	kozs	130	-	3	9	15	18	26	16	9	6	6	2	3	2	3	3	2	3	3	1	-
Revenue	$M	**1,665.4**	**-**	**-**	**117.0**	**140.1**	**157.3**	**197.4**	**149.3**	**114.5**	**104.0**	**114.2**	**95.2**	**91.3**	**80.2**	**89.1**	**78.3**	**56.5**	**36.6**	**36.6**	**7.9**	**-**
(-) Operating Cost	$M	(595.6)	-	-	(39.6)	(44.8)	(46.5)	(45.4)	(43.4)	(44.6)	(44.3)	(41.7)	(39.7)	(37.9)	(40.5)	(35.8)	(35.0)	(25.7)	(13.3)	(13.3)	(4.0)	-
(-) Mining Tax	$M	(16.7)	-	-	(1.2)	(1.4)	(1.6)	(2.0)	(1.5)	(1.1)	(1.0)	(1.1)	(1.0)	(0.9)	(0.8)	(0.9)	(0.8)	(0.6)	(0.4)	(0.4)	(0.1)	-
(-) Capital Spending	$M	(297.1)	(15.3)	(68.2)	(36.4)	(29.0)	(14.3)	(15.1)	(15.2)	(15.1)	(20.8)	(17.6)	(8.8)	(16.4)	(5.7)	(8.8)	(7.6)	(2.8)	-	-	-	-
(+) Salvage Value	$M	1.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.4
(-) Closure	$M	(4.0)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(4.0)
Pre-Tax Cash Flow	**$M**	**753.5**	**(15.3)**	**(74.8)**	**39.8**	**64.9**	**94.9**	**134.9**	**89.3**	**53.7**	**37.8**	**53.7**	**45.7**	**36.1**	**33.1**	**43.7**	**34.8**	**27.4**	**22.9**	**22.9**	**3.8**	**4.0**
(-) Income Tax	$M	(257.9)	-	-	(20.8)	(28.0)	(32.8)	(47.4)	(28.5)	(15.5)	(11.7)	(16.2)	(9.2)	(8.2)	(7.5)	(13.5)	(9.6)	-	-	-	(9.8)	0.6
After-Tax Cash Flow	**$M**	**495.6**	**(15.3)**	**(74.8)**	**19.0**	**36.9**	**62.2**	**87.5**	**60.9**	**38.3**	**26.1**	**37.5**	**36.5**	**27.9**	**25.6**	**30.1**	**25.2**	**27.4**	**22.9**	**22.9**	**(6.0)**	**4.7**
Cumul After-Tax Cash Flow	$M	-	(15.3)	(90.1)	(71.1)	(34.2)	27.9	115.5	176.4	214.7	240.8	278.3	314.8	342.7	368.3	398.4	423.6	451.1	474.0	496.9	491.0	495.6
Disc After-Tax Cash Flow (5%)	**$M**	**304.0**	**(15.0)**	**(69.5)**	**16.8**	**31.1**	**49.9**	**66.9**	**44.3**	**26.6**	**17.2**	**23.6**	**21.9**	**15.9**	**13.9**	**15.6**	**12.4**	**12.9**	**10.3**	**9.8**	**(2.4)**	**1.8**
Disc Cumul After-Tax Cash Flow	$M	-	(15.0)	(84.5)	(67.6)	(36.5)	13.4	80.3	124.6	151.2	168.4	192.0	213.9	229.8	243.7	259.3	271.8	284.6	294.9	304.7	302.2	304.0

24.3.10 Additional Mineral Resources Amenable to Underground Mining: El Tigre North

Included in the MRE stated in Section 14 of this Technical Report are additional Mineral Resources located approximately 700 m north of the planned mining areas discussed in the 2025 PEA in this Technical Report. An arbitrary division of the Property into "El Tigre South" ("ETS") and "El Tigre North" ("ETN") utilized the Combination Canyon Fault as the cut-off boundary. All mining described in previous sections of this Technical Report targets Mineral Resources on the south side of this fault at ETS. Significant Mineral Resources exist on the north side of the fault at ETN that are amenable to underground mining. A mine design was completed for ETN, however, was not included in the 2025 PEA per the Company's request, and detailed costing/scheduling activities have not been performed. A total of 2.95 Mt of mineralized material at an average grade of 343 g/t AgEq could potentially be recovered from ETN utilizing longitudinal longhole retreat mining with PF, showing the potential for a mine life of slightly in excess of 10 years at 290 ktpa (nominally 800 tpd). It is the opinion of the Authors that the data and information provided in Section 24.3.9 is not as detailed compared to that provided in Section 24.3.4 of this Technical Report, however, it is at similar standards to a PEA.

24.3.10.1 ETN Underground Mining

The mining method selected for ETN stopes is entirely longitudinal longhole retreat mining, on 30 m level intervals, with cemented PF for backfill, and targets five veins (Fundadora, Caleigh, Benjamin, Protectora and Aguila). The vast majority of mining (approximately 93% of stope tonnes) targets the Fundadora and Caleigh Veins. As in ETS, a small amount of mining occurs on the Black Shale structure, however, it is planned to recover this material with LH mining at ETN instead of C&F mining as at ETS. No known HMW excavations exist in the planned ETN mine, negating the need for transverse-access mining. Mining at ETN is expected to utilize contractors, similarly to ETS.

As strike lengths of individual levels can exceed 600 m, FWDs are driven parallel to the vein offset approximately 12-15 m from the FW of the stopes. Crosscuts are driven from the FWD across the vein to the sill drift on nominal 100 m intervals, splitting each level into multiple panels of four 25 m L stopes and limiting the tramming distance for LHDs in the sill drift. Development profiles at ETN are planned to be the same as at ETS.

The vast majority of longhole drilling utilizes downholes (94% of stope tonnes), however, a minority of stopes will utilize upholes, primarily in near-surface areas during pillar extraction. Upholes are also planned in certain areas of stopes where overcut access is not economic and it is expected that reduced recoveries and additional mining losses are more economic than extending overcut development, and in areas where artificial sill pillars are required. ITH drills are expected to be used for all longhole drilling at ETN. Extraction of material from stopes is planned using 7-t LHDs. Backfilling is planned to use PF under the same conditions as mining at ETS. Similarly to ETS, areas where stopes are located above the main haulage access to the mining area utilize a materials handling system comprised of multiple passes (both for mineralization and waste) to transfer material to loadout bins adjacent to the main haulage and reduce haulage costs. Stopes located below the main haulage access utilize on-level loading of 30-t trucks with 10-t LHDs.

Stopes for the ETN mine were designed under the same conditions as the stopes for the ETS mine (identical geometry, cut-offs and sizing limitations), at the same time. While the majority of stopes were not materially impacted by the changes in AgEq formulations throughout the study, certain areas of ETN, particularly the Fundadora Vein, contain significant quantities of sulphide material, and were rendered uneconomic by the planned use of cyanidation in process recovery, and were not included in the mine plan (similar to Zone 10 in ETS). Approximately 115 kt of mineralized material was removed from the mine plan as a result.

Artificial sill pillars are expected to be required at ETN to segregate the large vertical extents of the Caleigh and Fundadora Veins into smaller mining blocks. A total of four artificial sill pillars (two on each vein) are expected to be required. Generally, stopes in ETN are relatively thin (averaging ~3 m W), resulting in the requirement of a 15 m crown pillar (the recommended minimum thickness of this pillar is 15 m H or twice the stope span). Approximately 3% of stope tonnes intersect the crown pillar.

Ground support recommendations for ETN have not been provided by geotechnical consultants. The Authors have assumed similar recommendations to those provided for ETS will apply at ETN. Figure 24.52 shows the ETN mine design at end of LOM.

24.3.10.2 ETN Development

Table 24.60 shows lateral and vertical development totals by size and category. Unlike ETS, no development through PF is required, nor is there any C&F mining or any requirement for D&F pillars. Equivalent metres conversions utilize a 4.0 m W x 4.0 m H arched profile for lateral development, and a 3.0 m W x 3.0 m L square profile for vertical development.

FIGURE 24.52 LONGITUDINAL PROJECTION OF ETN MINE DESIGN AT END OF LOM

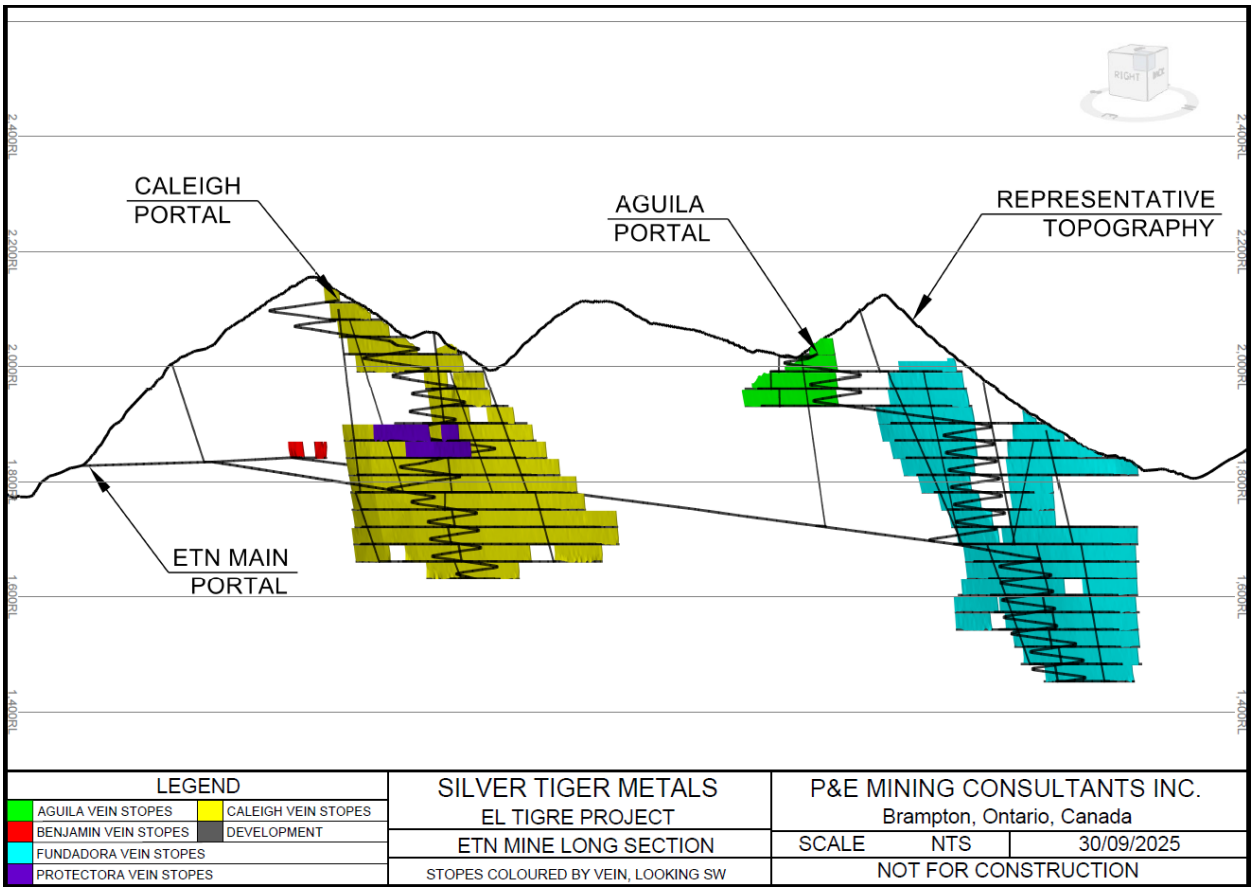


TABLE 24.60
DEVELOPMENT SUMMARY

Development Type	Nominal Profile	Capital/ Operating and Type	Linear Quantity (Lin. m)	Equivalent Metres (Eq. m)
Main Ramp	4.5 m W x 4.5 m H Arched	Capital, Lateral	10,793	13,699
Level Access			1,277	1,621
Footwall Drift			9,309	11,815
Mineral Pass Transfer			185	235
Mineral Pass Access			714	906
Remuck Bay			981	1,245
Vent Access			1,330	1,688
Electrical Bay	3.5 m W x 3.5 m H Arched		383	290
Sump/Pump Station			576	436
Mineralized or Waste Sill	4.0 m W x 4.0 m H	Operating, Lateral	12,694	12,694

TABLE 24.60 DEVELOPMENT SUMMARY				
Development Type	**Nominal Profile**	**Capital/ Operating and Type**	**Linear Quantity (Lin. m)**	**Equivalent Metres (Eq. m)**
Mineralized or Waste Crosscut	Arched		2,415	2,415
Subtotal (Lateral)[1]	**All**	**Lateral**	**40,658**	**47,044**
Mineral Pass Raisebore	2.7 m Ø	Vertical, Capital	1,353	861
Ventilation Raisebore	3.4 m Ø		2,764	2,789
Ventilation Drop Raise	3.0 m W x 3.0 m L		431	431
Subtotal (Vertical)[1]	**All**	**Vertical**	**4,547**	**4,080**

Note: *[1] Totals may not sum due to rounding.*
Lin. m = Linear metres, Eq. m = equivalent metres.

Lateral

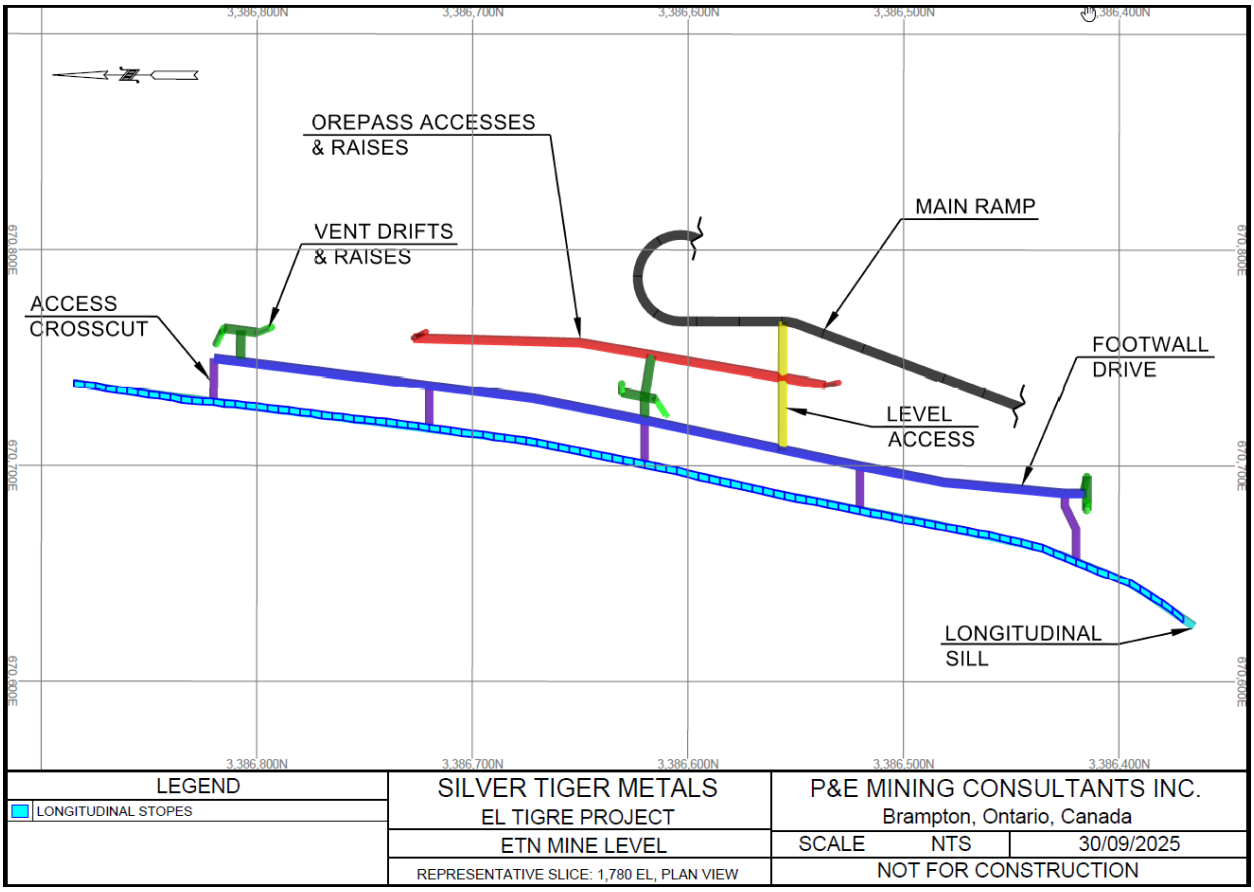
Lateral development is sized to accommodate the largest piece of equipment expected to operate within the mining area. In general, the LH drills drive the limits for operating development (clearance around the mast during dip/dump and ring drilling), while haul trucks drive the limits for most capital development, particularly the ramp. Other factors, such as ventilation requirements or speciality services (e.g. mineral pass loadouts or remuck bays) are also taken into account when sizing lateral development.

A total of 40.7 linear km of lateral development and 4.5 linear km of vertical development is planned over LOM. Approximately 37% of lateral development is operating development.

Development profiles differ slightly from ETS in some cases as level spacings are larger, meaning that longer drill rods are more optimal, which in turn increases the rotational clearance necessary for ITH drill masts and rod handlers. It is likely that some optimizations in development profiles are feasible and, therefore, the Authors recommend a more thorough evaluation of development sizing in future studies on ETN.

Figure 24.53 shows a plan view of the 1,780 EL level at ETN.

FIGURE 24.53 ETN PLAN VIEW, 1,780 EL LEVEL



Vertical

Vertical development in the UG mine is utilized for ventilation raises and mineral passes. Mineral passes are expected to be entirely raisebored by specialist contractors, while ventilation raises are designed as approximately 30% drop raise and 70% raisebore. Escapeways are fitted in ventilation raises as necessary. It is not anticipated that separate escapeway raises will be required.

Raiseboring is used where raise lengths are longer than 40 m. Raiseboring is utilized to drive a single raise segment connecting multiple consecutive levels with minimal lateral offset (raisebored raises are designed to have a dip of 70° or higher), whereas drop raises are used to connect two consecutive levels where lateral offsets between levels make raiseboring inefficient, or where raises need to be advanced incrementally to depth (ramp development below the main haulageways).

It is feasible to drive long raises in the UG with either raisebore or Alimak methods. Alimak methods have the advantage of not requiring an overcut to begin development, however, suffer from declining production at the ends of long raises due to the transit time of the deck. Raiseboring from an undercut level to an overcut level for long raises has been assumed.

24.3.10.3 ETN Production

Production mining at ETN is entirely LH stoping with PF, mined on retreat using longitudinal access.

Longitudinal Longhole Retreat Mining

Longitudinal longhole retreat mining involves driving an access through mineralization along the strike of the vein to the end of a group of stopes. The stopes are drilled using downholes (or upholes where no top access exists), loaded and blasted, and mineralization is recovered from the bottom access using an LHD. Once the blasted material is excavated and a survey is complete, a wall is built to seal the bottom access of the stope and the stope is then filled with PF. After a curing period (nominally 28 days), the cycle repeats on the next stope in the sequence (drilling and loading operations can commence prior to complete curing, however, blasting cannot). Longitudinal retreat mining progresses from the bottom of a mining block to the top, working on top of PF. Backfilling can either be from the sill drift above or through a drill hole from nearby development. Drill holes will be the sole option for the last level in each mining block as there will not be an open sill drift above. Longitudinal longhole retreat mining minimizes waste development by avoiding offset drifts and crosscuts, at the cost of reduced mining faces on a level.

A slight modification to the access method mentioned above is used at ETN. In addition to the sills, a FWD is driven in parallel to the sill on a 12-15 m offset, with crosscut accesses driven on 100 m centres across to the sill. This increases the available faces on a level from two faces total to two faces per crosscut. On the largest levels, this results in a 600% increase in available faces. Additionally, the FWD serves to provide connections to the materials handling system on many levels, and to the primary ventilation system on all levels.

Backfill

No recommendations for PF strengths for mining at ETN have been provided by geotechnical consultants, nor have any backfill recipes been generated for it. The Authors have assumed that the recommendations for ETS will be applicable at ETN. It is expected that a total of 1.2 Mm^3 of PF will be placed over LOM, and approximately 14% of that PF will be at 1.0 MPa strength. No 1.4 MPa high-strength PF is expected to be required. A total of 1.5 Mt of tailings will be consumed in PF placement.

Pillars

Artificial sill pillars will be placed in four locations in ETN: on the Caleigh Vein at EL 1,780 and EL 1,720, and on the Fundadora Vein at EL 1,750 and EL 1,540.

At present there is no surface mining planned at ETN. Therefore, the Crown Pillar is assumed to be 15 m thick and apply only to the existing topographic surface. Stopes intersecting this pillar are extracted using upholes, and PF will be placed from surface as required.

Stockpiling

A small amount of material from development operations that exceeds DCOG, but falls below MCOG, will be stored on surface to supplement process plant feed in the event of delays in UG production, and is generally expected to be processed after the end of UG mining. This SP is expected to reach a maximum size of 35 kt over LOM.

24.3.10.4 ETN Dilution, Mining Loss and Recovery

Dilution and mining recovery factors for ETN stopes and development are the same as for ETS. Internal and external dilution is calculated in the same fashion. Average internal dilution on development is 57.4%, average internal dilution on stopes is 43.4% and the average internal dilution on all mineralized recoverable ETN material is 45.7%. The average recovery on all mineralized recoverable ETN material is 97.0%: 99.0% on development and 94.5% on production. Mining loss is thus 3.0% on those tonnes: 1.0% on development and 5.5% on production. Recovery is generally higher and dilution generally lower at ETN versus ETS as there are fewer uphole stopes and no impacts from HMW workings.

24.3.10.5 ETN Services and Infrastructure

Generally, the basis of the services design for ETN mimics that of ETS wherever possible.

Ventilation

Ventilation at ETN will utilize exhaust (pull) systems, with fresh air being drawn down ramps onto active levels, across FWDs and into RARs. Sills and crosscuts are ventilated using auxiliary ventilation provided through flexible ducting; initial decline ventilation utilizes semi-rigid ducting for reduced resistance. No ventilation modelling of the ETN mine has been completed, however, it is expected that the same types of fans utilized at ETS will be applicable to ETN. It is assumed that 7 fan installations will be required over LOM, and that 200 HP fans will be utilized due to the reduced depths and expected airflow requirements at ETN.

Dewatering

The UG dewatering system for the Project utilizes the same design as used in ETS. Pump stations are spaced on 150 m vertical intervals and equipped with 250 HP of centrifugal and submersible pumps. For levels located above the main haulage access, all mine water flows via gravity to the pump station located closest to the elevation of the access. A total of four pump stations are expected to be required (2 in Fundadora, 1 at the bottom of Caleigh and 1 in the main haulage for use during development and also to act as a final booster station later in mine life).

Compressed Air

The compressed air system for ETN mimics the design used for Areas B and C of ETS. Initially, portable compressors will be used prior to permanent compressors being installed at EL 1,825 at the branch in the main haulage. Compressors are expected to be powered by 186 kW motors, the system is designed to operate with up to 40% losses and supply 1,200 scfm at 8.4 bar (~120 psi).

The system utilizes 150 mm steel pipelines for distribution, with 100 mm diameter HDPE pipelines in crosscuts and sills.

Electrical

The electrical distribution system for ETN utilizes the same structure as for ETS, with a 15 kV supply voltage converted to 480 V service voltage in MLCs of either 1,000 kVA or 1,500 kVa. Generally, for ETN, as there are fewer active faces on any level due to ETN utilizing purely LH retreat mining, the majority of MLCs will be 1,000 kVa. Production MLCs will be installed every third level and cables run through drill holes to adjacent levels to limit the number of necessary MLC relocations. Not all MLCs will be active at once.

Refuges and Egress

Emergency egress from the UG mine will be via ladderway in the ventilation raises. Lunch rooms will be installed approximately every fourth level and configured to additionally serve as refuge stations according to applicable laws and regulations. A total of 7 permanent refuges will be installed, and a portable refuge station will also be provided for use during development. Approximately 1.0 km of escapeways will be installed over LOM. In both the Caleigh and Fundadora areas, a secondary access ramp will be driven from surface to connect with the main haulageway, providing a secondary means of access and limiting the amount of ladderway transit required in the event of an emergency.

Materials Handling

Mining contractors charge by distance and quantity for haulage operations. As a result, it is more cost-efficient to load on-level than install materials handling infrastructure to centralize truck loading when the haulage distance required for material is increased by utilizing the passes. In the case of ETN, more than half (~57%) of tonnes are located above the main haulage route and utilize passes. Five passes totalling roughly 1.3 km of vertical development will be installed over LOM. Each pass will be fitted with a bin and loadout at the bottom. Bins and loadouts are expected to last the entire LOM without replacement. Passes are not expected to require concrete lining due to the limited tonnage throughput required in each (~150 kt maximum). All passes are assumed to be bored to the same diameter as at ETS.

Backfill Distribution

PF will be provided through 100 mm diameter steel pipelines run parallel to the ventilation system. Due to the significant lateral distance from the planned ETS PF plant to the ETN mine, it is likely that the ETS PF plant will need to be modified slightly with the addition of a positive displacement pump to overcome the initial head and to fill the two levels above the planned PF plant elevation of EL 2,070.

Peak flow rates for the PF distribution system at ETN are less than those at ETS due to the lack of HMW voids requiring concurrent filling with stoping.

Maintenance and Fuel/Lube Distribution

Basic preventative maintenance can be performed underground in retrofitted remuck bays, however, major maintenance operations areas expected to be performed on surface at a dedicated shop.

The significant lateral transit distance between the two main mining areas at ETN make centralized fuel/lube bays impractical. Fuel and lube operations will be performed by mobile vehicles.

24.3.10.6 ETN Personnel Requirements

The ETN mine will utilize the same contractor operation as ETS, albeit with a slightly reduced total personnel requirement due to simplified mining and more centralized operations. The Author's estimate a total staff of UG operators of approximately 100 personnel. The Owner's technical staff will be similarly slightly less than at ETS, and is estimated to peak at approximately 24 personnel.

24.3.10.7 ETN Mine Plan Portion of the Mineral Resource

Table 24.61 shows the mine plan portion of the Mineral Resource in the ETN mining area.

TABLE 24.61
ETN MINE PLAN PORTION OF THE MINERAL RESOURCE

Item			Tonnes (k)	AgEq (g/t)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
Internally Diluted Mineral Resource	UG Mine	Measured	66.5	632.7	1.48	501.4	0.101	0.514	0.713
		Indicated	104.9	635.8	2.00	458.5	0.094	0.144	0.165
		Subtotal M&I[1]	**171.4**	**634.6**	**1.80**	**475.1**	**0.097**	**0.288**	**0.378**
		Inferred	1,470.1	648.4	3.00	382.9	0.624	1.382	1.547
		Waste[2]	1,364.0	0.0	0.00	0.0	0.000	0.000	0.000
Dilution	UG Mine	Waste[2]	89.6	0.0	0.00	0.0	0.000	0.000	0.000
Fully Diluted Mineral Resource	UG Mine	Measured	126.7	332.1	0.78	263.1	0.053	0.270	0.374
		Indicated	198.2	336.5	1.06	242.7	0.050	0.076	0.087
		Subtotal M&I[1]	**324.9**	**334.7**	**0.95**	**250.6**	**0.051**	**0.152**	**0.199**
		Inferred	2,770.2	344.1	1.59	203.2	0.331	0.733	0.821
Mining Loss	UG Mine	Measured	(8.0)	305.4	0.63	250.0	0.056	0.372	0.545
		Indicated	(10.1)	341.2	1.03	249.6	0.050	0.083	0.084
		Subtotal M&I[1]	**(18.1)**	**325.4**	**0.85**	**249.8**	**0.053**	**0.210**	**0.287**
		Inferred	(129.0)	348.6	1.61	205.9	0.319	0.710	0.793
Mine Plan	UG Mine	Measured	118.7	333.9	0.79	264.0	0.053	0.263	0.363
		Indicated	188.0	336.2	1.06	242.3	0.050	0.076	0.088
		Subtotal M&I[1]	**306.8**	**335.3**	**0.96**	**250.7**	**0.051**	**0.148**	**0.194**

TABLE 24.61 ETN MINE PLAN PORTION OF THE MINERAL RESOURCE									
Item			**Tonnes (k)**	**AgEq (g/t)**	**Au (g/t)**	**Ag (g/t)**	**Cu (%)**	**Pb (%)**	**Zn (%)**
Mineral Resource		Inferred	2,641.3	343.9	1.59	203.1	0.332	0.735	0.822

Note: *[1.] Totals may not sum due to rounding*
[2.] Waste material is material not classified in the block model. It is distributed into Measured, Indicated or Inferred classifications at the dilution stage using a weighted average by mass.
M&I = Measured and Indicated Mineral Resources.

24.3.11 Historical Mine Workings Mineralization

Assays have been performed on HMW backfill material for Au and Ag, however, have not been taken with sufficient data density to include this material as Mineral Resources. A total of 50 samples were taken, resulting in grades ranging from 0.003 g/t to 2.064 g/t Au and from 2 g/t to 1,044 g/t Ag.

To determine if additional sampling added value to the Project, outlier samples were eliminated from the data set, resulting in a final sample set of 38 samples. These samples have grades ranging from 0.003 g/t to 1.030 g/t Au and from 2 g/t to 151 g/t Ag, with weighted average (weighted by intercept interval length) grades of 0.30 g/t Au and 34.7 g/t Ag. Utilizing an Au:Ag ratio of 88.6:1, these grades give an average AgEq for the HMW backfill material of 61 g/t. Therefore, this material has been classified in the mine plan as waste, however, a total of 437 kt of HMW backfill material will be mined over LOM and segregated onto a separate stockpile for potential evaluation for processing after the end of LOM.

The potential NSR of this material has not been calculated, since the grades of this material are appreciably lower than that of material in the mine plan (lower even than the grades in the Tailings SP), and would likely see reduced recoveries and payables relative to that estimated for the mine plan portion of the Mineral Resource. There is no certainty that the material on the HMW stockpile will ever be sufficiently evaluated to be classified as a Mineral Resource, nor ever be processed.

24.3.12 2025 PEA Risks and Opportunities

Risks and opportunities have been identified for the 2025 PEA. The anticipated impact is listed in brackets after each item, using low-medium-high categories.

24.3.12.1 Risks

Mineral Resource Estimate

- Future metal prices could cause a revision of the Mineral Resource Estimate. (low)

Underground Mining

- Geotechnical analysis is required to determine the safe stand-off distance from historical workings drifts, and the support regime. Hanging wall stability may be poor, meaning that parts of the southern vein mining will need to be revised from hanging wall access to footwall access, which would delay the production schedule. (medium)

- Drift-and-fill pillars may not be developed at the planned rate or may not perform as expected. Possible draw down of backfill material above the pillar would increase excavation requirements and could change the mining sequence or interrupt planned mineralized feed production. This could be significant, particularly since Zone 6 is reliant on the success of this type of pillar early in the mine life. Also, the 40° angle of repose expected for backfill may not be correct, and D&F pillars may need to be longer than planned. (medium)

- Cost quotations from mining contractors could result in higher unit costs. (low to medium)

- Grade estimation of the backfill material in the mined-out historical workings stopes is based on limited data and requires systematic analysis before advancing the Project to the next level of engineering study. (low)

- The amount of caving in the historical workings is unknown. The historical workings voids may have expanded or may be filled in. Most do not appear to be full of backfill. When mining near them, an appreciable delay in the schedule may be incurred if a bulk fill of open void is required. Also, if the walls unravel there is the potential to lose high-grade mineralization into the excavated backfill. The solubility of the historical backfill material is also unknown. (low)

- Hydrogeology is not well understood. Water re-charge rates are likely low, however, are currently unknown. Initial dewatering assumes a minimal refill rate, and it may actually take longer to draw down the water if the re-charge rate is higher. (low)

- Ventilation costs may be higher than expected. One steady-state model for the end of the mine life was created, with no models for the various stages of mine life. (low)

- Bypass Fault impact could be worse than expected, causing a delay, or requiring additional costs to install steelwork for support. (low)

Process Plant and Tailings

- Metallurgical recoveries. The preliminary testwork has indicated high recovery and production from the El Tigre mineralization. Existing assumptions could change and process flow alternatives could lead to reduced metal production, higher OPEX, and circuit changes. Testwork optimization should be conducted in all areas. (medium)

- There is a possibility that fines in dry stack tailings may need to be removed to achieve desirable rheology for paste backfill. (low to medium)

- Ground water inflow to wells needs to be confirmed. Secondary sources are the collection pond at the dry stack tailings facility and from the historical underground mine workings. The volume reported on a yearly basis is sufficient to serve the make-up water demands. Water pumped from the Bavispe River is a more expensive alternative. (low)

Financial Aspects

- Lower metal prices would decrease the Project economics. However, sensitivity analysis indicates that a 20% decrease in base case metal prices would still result in a financially attractive Project, and current spot metal prices are approximately 40% higher than the base case prices. (low)

24.3.12.2 Opportunities

Mineral Resource Estimate

- The El Tigre Deposit remains open along strike and down dip. There is an opportunity to extend the veins with additional drilling. (medium)

- Confidence in the underground portion of the Mineral Resource is high since it is based mainly (85%) on Measured and Indicated Mineral Resources. There are 766 kt of Inferred Mineral Resources contained in the underground mine design. There is a reasonable expectation that the Inferred Mineral Resource can be upgraded to a Measured or Indicated Mineral Resource with further definition drilling. (medium)

- The stockpile of historical mine workings material located near the Level 7 portal could be sampled to determine if it is economic to send to the process plant. (low)

Underground Mining

- El Tigre North could supplement or extend the mine life. Infill drilling is required to upgrade the large amount of Inferred Mineral Resources. (medium)

- It may be possible to slash out some of the historical workings levels at less cost than driving new levels. (low)

- Optimizing the mine design may result in a reduction of development, particularly in the ventilation system. (low)

- An increase in metal prices could improve the economics of Zone 10 and perhaps bring it into the mine plan. This would extend the mine life. (low)

- If there are historical workings that are empty, the mining method can be switched from transverse to longitudinal longhole retreat after initial backfilling, saving lateral development. (low)

Process Plant and Tailings

- Preliminary testwork has indicated high metallurgical recovery and production for the El Tigre underground mineralized material. Existing assumptions and process flow alternatives could change with metallurgical optimization that would lead to higher metal production, lower OPEX, and improved circuit changes. This will have a definite positive effect to Project overall cash flow. Testwork optimization should be conducted in all areas. (low)

- Refurbished equipment available on the market could be inserted into specific areas – i.e. mills, crushers. Capital cost reduction and a decrease in the Project construction timeframe could result. (low)

- There may be a way to recover base metals from the deeper sulphide mineralization (particularly for ETN and to a lesser extent in ETS) on a batch feed basis with modification to the process plant flowsheet. There would be a trade-off in precious metals recovery and the economics would need to be studied. (low)

Financial Aspects

- A 40% improvement in the base case metal prices would provide a $300M increase in After-Tax NPV (using a discount rate of 5%) to $604M with an IRR of 80%. The 40% increase in base case metal prices would result in prices at approximate current spot prices. (high)

25.0 INTERPRETATION AND CONCLUSIONS

25.1 PRE-FEASIBILITY STUDY

The El Tigre Property consists of 59 Mexican Federal mining concessions totalling 21,775 ha that are currently controlled by Silver Tiger. The Property is located in northeastern Sonora State, Mexico, 236 km northeast of the City of Hermosillo, and 230 km southeast of the City of Tucson, Arizona. The Property hosts the past-producing Lucky Tiger Mine that operated intermittently between 1903 and 1938 and produced gold and silver.

The Property is located in the Sierra El Tigre Mountain range that is part of the Basin and Range Province, which extends from Zacatacus and Jalisco in Mexico northwards to northern Nevada in the U.S. The Sierra El Tigre is part of the massif of the Sierra Madre Occidental and was formed during Cenozoic extensional faulting, which consists of northerly-trending horsts and grabens. The silver, gold, lead, zinc, and copper mineralization occur mainly in fissure veins within a narrow, north-trending belt >5 km in length. The silver and gold mineralization in the El Tigre area occurs in both the fissure veins and in a low-grade stockwork halo near the veins. The El Tigre Veins closely resemble those in quartz-adularia, low sulphidation epithermal deposits.

Modern exploration was initiated in 1981 by Anaconda Minerals Company through its wholly-owned subsidiary Cobre de Hercules. Exploration by El Tigre Silver Corporation and Oceanus has included channel sampling of surface mineralization and historical underground workings, sampling of historical tailings, IP geophysics and diamond drilling. Between 1982 and 2013 Anaconda, Mineras Cordilleras and El Tigre Silver Corporation completed a total of 18,114 m of drilling. In 2016 to 2017, Oceanus completed 62 diamond drill holes for a total of 11,923 m. From 2020, Silver Tiger has completed 478 drill holes for 120,007 m to support updated Mineral Resource estimations for the El Tigre Property and provide fresh material for metallurgical testing and geotechnical data and information for advanced project development studies. High-grade mineralization at the El Tigre Property is open to expansion by drilling down-dip and along strike.

The Authors have reviewed drilling procedures, sample preparation, analyses and security and are of the opinion that the drill core logging procedures employed, and the sampling methods used were thorough and have provided sufficient geotechnical and geological information. The Authors consider the data to be of good quality and satisfactory for use in the current Mineral Resource Estimate. Verification of the El Tigre Project data was undertaken by the Authors, and included site visits, due diligence sampling, verification of drilling assay data, and assessment of the available QA/QC data from the historical and recent drilling programs. The Authors consider that there is an adequate correlation between assay values in El Tigre's database and the independent verification samples collected and analyzed, and that the supplied data are of satisfactory quality and suitable for use in this current Mineral Resource Estimate for the El Tigre Project.

Initial metallurgical testwork of the El Tigre Deposit was completed in 2012. The selected process included direct cyanidation followed by Merrill Crowe recovery of Au and Ag at a throughput of 200 tpd with future expansion to 400 tpd. The limited amount of cyanidation testwork was undertaken on three composite tailings samples representing visually distinguishable characteristics. In August 2022, an initial scoping-level metallurgical testwork program was commenced at SGS Lakefield ("SGS") in Ontario, Canada. The objectives of testwork were to

develop metallurgical data to evaluate and optimize various processes for the recovery of gold and silver, including whole ore sample cyanide leaching, Merrill Crowe precipitation, flotation, and heap leach amenability. Mineralogical, environmental, and solid/liquid separation and rheology examinations of fresh mineralized material and leach tailing samples were undertaken to support the testwork program.

Additional metallurgical testing was completed in 2024 at McClelland Laboratories, Inc. ("MLI") in Sparks, Nevada on mineralization from the proposed starter pit at the El Tigre Project site. Testwork from six locations within the starter pit was undertaken to determine variability recovery by heap leach cyanidation processing and to further analyze work completed under the 2023 PEA. The testwork results are the foundation of this PFS and support Project process design criteria and process equipment as well as capital and operating costs and financial modelling. Variability in testwork results indicate starter pit mineralization is comprised of multiple geological zones including oxide, transition, and sulphide. This resulted in determination of variable recoveries. Further testwork to coincide with the MLI testing was completed at Laboratorio Tecnológico de Metalurgia ("LTM"), Hermosillo, Mexico. Testwork investigated the zone variability of the proposed ore for processing. Overall, metal recoveries for this PFS were determined to be 83% for oxide gold, 56% for sulphide gold, 45% for oxide silver and 40% for sulphide silver.

The basis for the PFS is the updated Mineral Resource Estimate which has an effective date of June 20, 2025. The updated Mineral Resource Estimate includes pit-constrained, out-of-pit, tailings and stockpile Mineral Resources. The databases contain a total of 5,875 collar records that contribute directly to the updated Mineral Resource Estimate. A total of 28 individual mineralized domains have been identified through drilling and sampling. The drilling extends ~5 km along strike. The Mineral Resources have been estimated in conformity with the requirements of the CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines as required by the Canadian Securities Administrators' National Instrument 43-101.

Measured and Indicated Mineral Resources are estimated at 58.6 Mt grading 44 g/t Ag, 0.49 g/t Au, 0.09% Cu, 0.36% Pb, and 0.61% Zn (125 g/t silver equivalent ("AgEq") or 0.90 g/t gold equivalent ("AuEq")). Contained metals contents of the Measured and Indicated Mineral Resources are 83,415 koz of Ag and 931 koz of Au plus base metals (235,543 koz AgEq or 1,688 koz AuEq). Inferred Mineral Resources are estimated at 9.1 Mt grading 142 g/t Ag, 0.92 g/t Au, 0.32% Cu, 0.75% Pb and 0.89% Zn (269 g/t AgEq or 2.99 g/t AuEq). Contained metal contents of the Inferred Mineral Resources are 41,589 koz Ag and 271 koz Au plus base metals (78,911 koz AgEq or 874 oz AuEq).

Proven Mineral Reserves are estimated at 24.2 Mt grading 14.7 g/t Ag and 0.43 g/t Au (79.9 g/t AgEq or 0.52 g/t AuEq). Probable Mineral Reserves are estimated at 16.1 Mt grading 15.3 g/t Ag and 0.37 g/t Au (74.9 g/t AgEq or 0.47 g/t AuEq). Total Proven and Probable Mineral Reserves are estimated at 40.3 Mt grading 14.9 g/t Ag and 0.40 g/t Au (77.9 g/t AgEq or 0.50 g/t AuEq). Contained metal contents of the total Mineral Reserve are 19,354 koz Ag and 523 koz Au.

The El Tigre Project will consist of a single surface mine operation using conventional equipment. Based on kinematic analysis and limit equilibrium approach, an updated design of the PFS pit was assessed considering slightly to moderately weathered rock with quadruple benches (20 m) for all sectors. For the west walls a BFA of 70° and IRA of 49° are suggested while for the East Walls a BFA of 65° and IRA of 46° were considered. Slopes of appreciable sizes are not anticipated on the

North and South walls; however, if they are included in a future design, they should follow the East Wall mineralized geometry.

Surface mining will be initiated in Year -1 (pre-stripping period) and will be completed in Year 9. The initial ore heap leaching rate (Years 1 to 3) is 7,500 tpd (2.74 Mta). In Year 4 the crushing rate will increase to 15,000 tpd (5.5 Mtpa) and this rate is maintained to the end of the mine life. The El Tigre Mine is planned to be a contract mining operation. The Mining Contractor will undertake all drilling, loading, hauling, and mine site maintenance activities. A separate Blasting Contractor will provide blast hole loading and charging services using an ammonium nitrate fuel oil mixture. The Owner will provide overall mine management and technical services, such as mine planning, grade control, geotechnical, and surveying services. It is expected that mid-sized hydraulic excavators and diesel-powered front-end loaders will be used. The anticipated truck size is 91 tonnes, similar to the CAT 777 truck, although alternate truck sizes may eventually be used. 68.3 Mt of waste rock will be mined, with 40.3 Mt ore, for a total of 108.6 Mt at a strip ratio of 1.7:1.0.

The process plant design is based on a nominal 7,500 tpd (Years 1-3) and 15,000 tpd (Years 4-9) of ore with average grades of 0.40 g/t Au and 14.9 g/t Ag. The process plant flowsheet design comprises of 3-stage conventional crushing, material handling of crushed product and loading onto lined heap pads. Solution ponds and pumping system allows irrigation of loaded ore and subsequent collection of the pregnant solution. The pregnant solution is pumped to the Merrill Crowe recovery facility for precipitation of gold and silver in solution and filtered accordingly. The resultant precipitate will be smelted at an on-site refinery to produce doré bars for market. The barren solution from the recovery process is recirculated to the heap leach pond (barren) for cyanide addition and pumping to the heaps for leaching. Make-up water for reagent mixing along with water evaporation and general process requirements will be pumped from the Bavispe River.

Current infrastructure at the El Tigre Project consists of an exploration camp that has accommodation for drillers and site administration personal. New major infrastructure will consist of: surface mine and waste rock storage facilities; 3-stage crushing circuit; heap leach pad with pregnant, barren, and overflow ponds; process plant with Merrill Crowe circuit and refinery for doré production, and assay laboratory; buildings such as a maintenance workshop, admin offices, warehouse; diesel fuel storage and distribution station; explosives storage; camp facility; and electrical power supply from the national grid, and distribution. Water needed for infrastructure and the process plant will be supplied via two wells drilled from surface. Power to the El Tigre Property will be supplied by the national grid via a 72 km 34.5 kV overhead power line.

Metal prices of US$3,200/oz Au and US$38/oz Ag are based on an approximate average of December 31, 2025, three-year monthly trailing averages, and Consensus Economics Inc. long term price forecasts. A 20:1 Mexican Peso:US Dollar exchange rate is based on the approximate past three-year average. The metal prices and exchange rate are subject to spot market conditions. There are no metals streaming or hedging agreements in place. Currently there are no contracts in place that are material to the El Tigre Project.

Exploration activities have been completed under annual permits from the Mexican government Secretary of the Environment and Natural Resources ("SEMARNAT"). An extensive list of Federal and State permits will be required before mining can commence, along with environmental impact studies. CIMA has completed an environmental baseline study and a socio-economic

baseline study that considers economic, cultural, social, demographic, and geographical aspects of the local communities. On July 27, 2023, an Environmental Impact Statement on El Tigre was submitted and is currently being evaluated by the regulatory authorities. The mining Project is expected to represent minimal risk to the processes and structure of the local ecosystem, and to the roads, livestock and agricultural activities. The baseline study concluded that there are no archaeological zones within the environmental system.

Total capital costs over the LOM are estimated at $113.1M. The initial capital cost estimate of $86.8M addresses the engineering, procurement, construction and start-up of the El Tigre Project, with an 18-month construction period to develop a surface mine, build a process plant capable of processing 7,500 tpd, prepare a heap leach pad, and install associated ancillary surface facilities. Expansion capital costs estimated at $20.1M are incurred in production years three and four to prepare haul roads to the next surface mine development phase and increase the process plant and heap leach pad capacity to 15,000 tpd. Sustaining capital represents capital expenses for additional costs that are not included in the normal operating costs, equipment purchases that will be necessary during the operating life of the Project, heap leach pad capacity increases, and all capital costs associated with surface mining. LOM sustaining capital is estimated at $6.2M.

The total operating cost over the LOM is estimated at $491.0M, averaging $12.19/t processed. LOM average mining is estimated at $2.24/t mined ($4.95/t processed), processing averages $4.95/t processed, and G&A is estimated at $1.28/t processed. The Project is subject to a 1.0% NSR mining tax payable to the Mexican government. Total costs associated with this NSR royalty mining tax over the LOM are estimated at $16.2M. Cash costs over the LOM, including Mexican mining taxes, are estimated to average $11.70/oz AgEq or $997/oz AuEq. All-In Sustaining Costs ("AISC") over the LOM are estimated to average $14.53/oz AgEq or $1,238/oz AuEq and include closure costs estimated at $6.2M.

Payable metal over the LOM is estimated at 8.57 Moz Ag and 408 koz Au (42.9 Moz AgEq). The El Tigre Project economic evaluation at base case metal prices of US$3,200/oz Au and US$38/oz Ag has an estimated US$455.6M after-tax net present value ("NPV") at a 5% discount rate ("NPV5%"), and an after-tax internal rate of return ("IRR") of 66%. The payback period is estimated to be 1.4 years from the start of production. The after-tax base case NPV's and IRR's are most sensitive to metal prices followed by operating costs and capital costs.

A preliminary risk analysis was undertaken on the Project by the Authors using a low-medium-high ranking system. The highest risk items were identified to be mining contractor cost assumptions and metallurgical recoveries. The main opportunity is that the Deposit remains open along strike and down dip, and there is potential to increase the current Mineral Resource Estimate.

25.2 2025 PRELIMINARY ECONOMIC ASSESSMENT

The El Tigre Underground Project is planned as a contractor operated longhole mining operation with cemented paste backfill supplemented by minor mechanized cut and fill mining and recovery of historical low grade and tailings stockpiles. The underground mine plan consists of approximately 4,000 kt of mineralized material, with 119 kt from a historical low-grade stockpile and 904 kt from a historical tailings stockpile, for a total of 5,023 kt grading over the LOM on average 331.7 g/t AgEq, 251.3 g/t Ag, 0.91 g/t Au, 0.033% Cu, 0.099% Pb and 0.166% Zn; base

metals provide no economic benefit under the cyanidation process. Underground production is scheduled at 290 kt per year (ktpa), giving an underground mine life of approximately 14 years and an overall Project life of approximately 17 years, including commissioning with low-grade stockpile material in Year -1 and processing of the historical tailings stockpile at 365 ktpa after process plant feed from underground mining is depleted.

Longitudinal and transverse longhole stopes, which account for 84% of underground tonnes, are arranged on 30 m level intervals with 15 m sublevels. Typical strike lengths are approximately 25 m for areas utilizing longitudinal retreat mining and 10 m for areas utilizing transverse mining. Stopes are supported by cemented paste backfill with strengths of 0.5, 1.0 or 1.4 MPa depending on span. Average internal dilution is estimated at 50.4%, external dilution at 4.8% and overall mining recovery at 93.8%.

The underground mining fleet comprises 7-t and 10-t load-haul-dump units, 30-t haul trucks, longhole drills, development drill jumbos, rock bolters, shotcrete machines and ancillary support equipment sized for the planned production rate. Approximately 437.1 kt of historical backfill and 2,438.9 kt of development waste, totaling 2,876.0 kt, are mined over the LOM.

Pre-production processing operations will treat 1,000 tpd while production from the underground veins will be processed at 800 tpd within a 350 day/year operating schedule. The flowsheet includes two stage crushing, ball milling, pre-leach thickening, six stage leaching, counter current decantation, and Merrill Crowe precipitation to produce gold and silver doré. Based on metallurgical testing, expected metal recoveries are 82% Au and 83% Ag for tailings, 92% Au and 71% Ag for stockpiles, and 89.7% Au and 84.4% Ag for underground material. Water will be supplied from two surface wells. Reagents include lime and sodium cyanide. Tailings will be dewatered for dry stack placement and will support paste backfill.

Major surface infrastructure proposed at the El Tigre Project will consist of connection to the national electrical grid, a portal for Level 7 access to the underground mine workings, low-grade mineralization stockpile and waste rock storage facilities near the portal, site access roads, ventilation raises, paste backfill plant, a process plant with primary jaw crusher, dry stack tailings storage facility, and historical tailings area. Additional infrastructure will include offices, warehouse, fuel storage and distribution facility, maintenance workshop, two water wells, water treatment plants, explosives storage and a 300-person camp.

Total capital costs over the LOM are estimated at $297.1M. The initial capital cost estimate of $83.5M addresses the engineering, procurement, construction and start-up of the El Tigre Project, with a two-year construction period to develop an underground mine, build a process plant capable of processing 800 tpd, construct a dry stack tailings storage facility, and install associated ancillary surface facilities. Sustaining capital consists of expenses for additional underground development that are not included in the normal operating costs, equipment purchases that will be necessary during the operating life of the Project, and dry stack storage capacity increases. LOM sustaining capital is estimated at $213.6M.

The total operating cost over the LOM is estimated at $595.6M, averaging $118.57/t processed. LOM average underground mining is estimated at $76.80/t mined, haulage between the underground portal and the process plant averages $1.33/t processed, processing underground feed averages $41.55/t processed, processing historical tailings and stockpile feed averages $24.25/t

processed, and G&A is estimated at $18.71/t processed. The Project is subject to a 1.0% NSR mining duty tax payable to the Mexican government, estimated at $16.7M over the LOM. Cash costs over the LOM, including Mexican mining taxes, are estimated to average $16.05/oz AgEq or $1,351/oz AuEq. All-In Sustaining Costs ("AISC") over the LOM are estimated to average $23.98/oz AgEq or $2,019/oz AuEq and include closure costs estimated at $4.0M.

Payable metal over the LOM is estimated at 33.6 Moz Ag and 130 koz Au (38.5 Moz AgEq). The El Tigre Project economic evaluation at base case metal prices of US$3,200/oz Au and US$38/oz Ag has an estimated US$304.0M after-tax net present value ("NPV") at a 5% discount rate ("NPV5%"), and an after-tax internal rate of return ("IRR") of 43%. The payback period is estimated to be 2.6 years from the start of production. The after-tax base case NPV's and IRR's are most sensitive to metal prices followed by operating costs and capital costs.

26.0 RECOMMENDATIONS

26.1 PRE-FEASIBILITY STUDY

The El Tigre Project contains a significant silver-gold Mineral Reserve and the Authors recommend that Silver Tiger proceed to develop the Project. Silver Tiger has indicated that it plans to proceed straight to detailed engineering and project construction as soon as the surface mine and processing permits are in place. Detailed engineering (FEED) is estimated to take nine months to complete at a cost of $1.0M.

Further metallurgical testwork ($0.6M) to confirm expectations based on previous metallurgical investigations of the El Tigre Project Deposits is also recommended. This recommendation has been precipitated by PFS testwork indicating recovery variability in the three domains within the surface mine design. The estimated timeline to complete related mapping, drilling, sample receipt and associated column testing on specific domains is approximately 12 months

A minimal exploration program to potentially upgrade the 1.4 Mt Inferred Mineral Resources contained within the surface mine design to Measured or Indicated Mineral Resources is recommended. Approximately 5,000 m at $200/m is required, at a cost of $1.0M.

The following represents a selection of geotechnical and hydrogeological items to consider for future study at an estimated cost of $0.2M plus any associated drilling costs:

- Silver Tiger should examine the possibility of identifying spatially where lithologies identified in geotechnical drilling as 'Lithic Tuff' and 'Quartz Rhyolite' are located as these two units may have lower intact strengths than the other logged lithological units. Continued collection of point load test ("PLT") data and use of this data for development of a PLT geotechnical block model should also be used to identify areas of potentially lower intact strength.

- The depth of weathering should be reviewed by Silver Tiger considering their exploration drilling to determine where the shallower slope design will be applied. The influence of alteration was not prevalent in the geotechnical drill core logging, contrary to expectations when intersecting the veins. Silver Tiger should consider spatially defining alteration across the site and using this information to potentially plan future geotechnical investigations.

- Use the lithological logging data from exploration drilling to define main lithological units and compare them with the breakdown from the PFS geotechnical drilling campaign. This action will help to understand if there are other lithological units within the El Tigre Formation which should be studied during future phases.

- Additional laboratory direct shear testing is recommended, with samples selected from each primary discontinuity type and rock type. This testing will help better characterize the shear strength of various joint sets and determine if rock type influences shear properties.

- It is recommended to conduct additional oriented core drilling and televiewer analysis to confirm large-scale structural features and major set orientations, particularly on the west wall and along different drill hole orientations to reduce existing blind zones. This proposed geotechnical investigation should include both oriented drill core data collection and televiewer surveys within drill holes that are logged geotechnically with oriented drill core. This will help reduce the uncertainty regarding the discrepancy in the structure data between the data collection methods.

26.2 2025 PRELIMINARY ECONOMIC ASSESSMENT

Based on the results of Silver Tiger's exploration work from 2020 to 2025, and the positive results of the 2025 PEA, the Authors recommend that the Company continue with Project development activities on the El Tigre Property and proceed with an underground Pre-Feasibility Study ("PFS"). To advance the Project and initiate an underground PFS, additional drilling is recommended by the Authors to upgrade Inferred Mineral Resources to Indicated Mineral Resources, and for metallurgical testwork (including mineralogical analyses and locked cycle testwork), geotechnical studies (underground mine, process plant area, and dry stack tailings and waste rock storage locations) and hydrogeological studies. Further study of the process plant and mine design, mine ventilation system, update of mining contractor quotes, infrastructure requirements, environmental/permitting, and Project economics would be part of the PFS.

A work program with an estimated budget of US$7.8M is proposed, as presented in Table 26.1.

TABLE 26.1
RECOMMENDED WORK PROGRAM FOR THE EL TIGRE 2025 PEA PROJECT

Description	Amount (US$)
Infill Drilling to Convert Inferred to Indicated Mineral Resources 15,000 m x $200/m	3,000,000
Rehabilitate Underground Workings for Drill Access	1,000,000
Metallurgical and Geotechnical Drilling	500,000
Sample Preparation and Assay	50,000
Metallurgical Testwork	300,000
Geotechnical and Hydrology Study	200,000
Pre-Feasibility Study	1,500,000
Camp Support and Wages	150,000
Capital Equipment	50,000
Sub-total	**6,750,000**
Contingency (15%)	1,013,000
Total	**7,763,000**

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P&E. 2023. NI 43-101 Technical Report and Updated Mineral Resource Estimate of the El Tigre Silver-Gold Project, Sonora, Mexico. Prepared for Silver Tiger Metals Inc. by P&E Mining Consultants Inc. dated October 27, 2023 (effective date September 12, 2023), 449 pages.

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28.0 CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON

ANDREW BRADFIELD, P. ENG.

I, Andrew Bradfield, P. Eng., residing at 5 Patrick Drive, Erin, Ontario, N0B 1T0, do hereby certify that:

1. I am an independent mining engineer contracted by P&E Mining Consultants.
2. This certificate applies to the Technical Report titled "Stockwork Zone Pre-Feasibility Study and Underground Preliminary Economic Assessment of the El Tigre Silver-Gold Project, Sonora, Mexico", (The "Technical Report") with an effective date of June 20, 2025.
3. I am a graduate of Queen's University, with an honours B.Sc. degree in Mining Engineering in 1982. I have practiced my profession continuously since 1982. I am a Professional Engineer of Ontario (License No.4894507). I am also a member of the National CIM.

   I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

   I have practiced my profession continuously since 1982. My summarized career experience is as follows:
   - Various Engineering Positions – Palabora Mining Company 1982-1986
   - Mines Project Engineer – Falconbridge Limited 1986-1987
   - Senior Mining Engineer – William Hill Mining Consultants Limited 1987-1990
   - Independent Mining Engineer 1990-1991
   - GM Toronto – Bharti Engineering Associates Inc 1991-1996
   - VP Technical Services, GM of Australian Operations – William Resources Inc 1996-1999
   - Independent Mining Engineer 1999-2001
   - Principal Mining Engineer – SRK Consulting 2001-2003
   - COO – China Diamond Corp 2003-2006
   - VP Operations – TVI Pacific Inc 2006-2008
   - COO – Avion Gold Corporation 2008-2012
   - Independent Mining Engineer 2012-Present
4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Sections 2, 3, 16, 19, 22, 24.1, 24.2, 24.3.1, 24.3.6, 24.3.7, 24.3.9, 24.3.12 and co-authoring Sections 1, 15, 18, 21, 24.3.8, 25, 26, and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101. I am independent of the Vendor and the Property.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled "Pre-Feasibility Study of the El Tigre Silver-Gold Project, Sonora, Mexico" with an effective date of October 22, 2024.
8. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 20, 2025
Signing Date: January 22, 2026
{SIGNED AND SEALED}
[Andrew Bradfield]

_____

Andrew Bradfield, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

JARITA BARRY, P.GEO.

I, Jarita Barry, P.Geo., residing at 9052 Mortlake-Ararat Road, Ararat, Victoria, Australia, 3377, do hereby certify that:

1. I am an independent geological consultant contracted by P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Stockwork Zone Pre-Feasibility Study and Underground Preliminary Economic Assessment of the El Tigre Silver-Gold Project, Sonora, Mexico", (The "Technical Report") with an effective date of June 20, 2025.
3. I am a graduate of RMIT University of Melbourne, Victoria, Australia, with a B.Sc. in Applied Geology. I have worked as a geologist for over 20 years since obtaining my B.Sc. degree. I am a geological consultant currently licensed by Engineers and Geoscientists British Columbia (License No. 40875) and Professional Engineers and Geoscientists Newfoundland & Labrador (License No. 08399) and I am registered as a Temporary Registrant with Professional Geoscientists Ontario (Registration No. 3888). I am also a member of the Australasian Institute of Mining and Metallurgy of Australia (Member No. 305397).

   I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:
   - Geologist, Foran Mining Corp. 2004
   - Geologist, Aurelian Resources Inc. 2004
   - Geologist, Linear Gold Corp. 2005-2006
   - Geologist, Búscore Consulting 2006-2007
   - Consulting Geologist (AusIMM) 2008-2014
   - Consulting Geologist, P.Geo. (APEGBC/AusIMM) 2014-Present
4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Section 11, and co-authoring Sections 1, 12, 25, 26, and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled "Pre-Feasibility Study of the El Tigre Silver-Gold Project, Sonora, Mexico" with an effective date of October 22, 2024, and "Technical Report and Updated Mineral Resource Estimate of the El Tigre Silver-Gold Project, Sonora, Mexico" with an effective date of September 12, 2023, and "NI 43-101 Technical Report and Updated Mineral Resource Estimate on the El Tigre Project Sonora, Mexico" for Oceanus Resources Corp., with an effective date of September 7, 2017.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 20, 2025
Signing Date: January 22, 2026

{SIGNED AND SEALED}
[Jarita Barry]

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Jarita Barry, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

FRED H. BROWN, P.GEO.

I, Fred H. Brown, of PO Box 332, Lynden, WA, USA, do hereby certify that:

1. I am an independent geological consultant and have worked as a geologist continuously since my graduation from university in 1987.
2. This certificate applies to the Technical Report titled "Stockwork Zone Pre-Feasibility Study and Underground Preliminary Economic Assessment of the El Tigre Silver-Gold Project, Sonora, Mexico", (The "Technical Report") with an effective date of June 20, 2025.
3. I graduated with a Bachelor of Science degree in Geology from New Mexico State University in 1987. I obtained a Graduate Diploma in Engineering (Mining) in 1997 from the University of the Witwatersrand and a Master of Science in Engineering (Civil) from the University of the Witwatersrand in 2005. I am registered with the Association of Professional Engineers and Geoscientists of British Columbia as a Professional Geoscientist (171602) and the Society for Mining, Metallurgy and Exploration as a Registered Member (#4152172).

   I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:
   - Underground Mine Geologist, Freegold Mine, AAC — 1987-1995
   - Mineral Resource Manager, Vaal Reefs Mine, Anglogold — 1995-1997
   - Resident Geologist, Venetia Mine, De Beers — 1997-2000
   - Chief Geologist, De Beers Consolidated Mines — 2000-2004
   - Consulting Geologist — 2004-2008
   - P&E Mining Consultants Inc. – Sr. Associate Geologist — 2008-Present
4. I have visited the Property that is the subject of this Technical Report on May 22, 2017, and June 19 to 20, 2016.
5. I am responsible for co-authoring Sections 1, 14, 25, 26, and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled "Pre-Feasibility Study of the El Tigre Silver-Gold Project, Sonora, Mexico" with an effective date of October 22, 2024, and "Technical Report and Updated Mineral Resource Estimate of the El Tigre Silver-Gold Project, Sonora, Mexico" with an effective date of September 12, 2023, and "NI 43-101 Technical Report and Updated Mineral Resource Estimate on the El Tigre Project Sonora, Mexico" for Oceanus Resources Corp., with an effective date of September 7, 2017.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 20, 2025
Signing Date: January 22, 2026

{SIGNED AND SEALED}
[Fred H. Brown]

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Fred H. Brown, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

DAVID BURGA, P.GEO.

I, David Burga, P. Geo., residing at 3884 Freeman Terrace, Mississauga, Ontario, do hereby certify that:

1. I am an independent geological consultant contracted by P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Stockwork Zone Pre-Feasibility Study and Underground Preliminary Economic Assessment of the El Tigre Silver-Gold Project, Sonora, Mexico", (The "Technical Report") with an effective date of June 20, 2025.
3. I am a graduate of the University of Toronto with a Bachelor of Science degree in Geological Sciences (1997). I have worked as a geologist for over 20 years since obtaining my B.Sc. degree. I am a geological consultant currently licensed by the Association of Professional Geoscientists of Ontario (License No 1836).

   I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:
   - Exploration Geologist, Cameco Gold 1997-1998
   - Field Geophysicist, Quantec Geoscience 1998-1999
   - Geological Consultant, Andeburg Consulting Ltd. 1999-2003
   - Geologist, Aeon Egmond Ltd. 2003-2005
   - Project Manager, Jacques Whitford 2005-2008
   - Exploration Manager – Chile, Red Metal Resources 2008-2009
   - Consulting Geologist 2009-Present
4. I have visited the Property that is the subject of this Technical Report on May 20 to 21, 2024, and August 5 and 6, 2023, and January 20 to 21, 2016.
5. I am responsible for co-authoring Sections 1, 10, 12, 25, 26, and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled "Pre-Feasibility Study of the El Tigre Silver-Gold Project, Sonora, Mexico" with an effective date of October 22, 2024, and "Technical Report and Updated Mineral Resource Estimate of the El Tigre Silver-Gold Project, Sonora, Mexico" with an effective date of September 12, 2023, and "NI 43-101 Technical Report and Updated Mineral Resource Estimate on the El Tigre Project Sonora, Mexico" for Oceanus Resources Corp., with an effective date of September 7, 2017.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 20, 2025
Signing Date: January 22, 2026

{SIGNED AND SEALED}
[David Burga]

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David Burga, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

D. GRANT FEASBY, P. ENG.

I, D. Grant Feasby, P. Eng., residing at 12,209 Hwy 38, Tichborne, Ontario, K0H 2V0, do hereby certify that:

1. I am an independent metallurgical consultant contracted by P&E Mining Consultants Inc. I am also currently the Owner and President of:

   FEAS - Feasby Environmental Advantage Services
   38 Gwynne Ave, Ottawa, K1Y1W9

2. This certificate applies to the Technical Report titled "Stockwork Zone Pre-Feasibility Study and Underground Preliminary Economic Assessment of the El Tigre Silver-Gold Project, Sonora, Mexico", (The "Technical Report") with an effective date of June 20, 2025.

3. I graduated from Queens University in Kingston Ontario, in 1964 with a Bachelor of Applied Science in Metallurgical Engineering, and a Master of Applied Science in Metallurgical Engineering in 1966. I am a Professional Engineer registered with Professional Engineers Ontario. I have worked as a metallurgical engineer for over 50 years since my graduation from university.

   I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report has been acquired by the following activities:
   - Metallurgist, Base Metal Processing Plant.
   - Research Engineer and Lab Manager, Industrial Minerals Laboratories in USA and Canada.
   - Research Engineer, Metallurgist and Plant Manager in the Canadian Uranium Industry.
   - Manager of Canadian National Programs on Uranium and Acid Generating Mine Tailings.
   - Director, Environment, Canadian Mineral Research Laboratory.
   - Senior Technical Manager, for large gold and bauxite mining operations in South America.
   - Expert Independent Consultant associated with several companies, including P&E Mining Consultants, on mineral processing, environmental management, and mineral-based radiation assessment.

4. I have not visited the Property that is the subject of this Technical Report.

5. I am responsible for authoring Section 20, and co-authoring Sections 1, 25, 26, and 27 of this Technical Report.

6. I am independent of the issuer applying the test in Section 1.5 of NI 43-101.

7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled "Pre-Feasibility Study of the El Tigre Silver-Gold Project, Sonora, Mexico" with an effective date of October 22, 2024, and "Technical Report and Updated Mineral Resource Estimate of the El Tigre Silver-Gold Project, Sonora, Mexico" with an effective date of September 12, 2023.

8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.

9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 20, 2025
Signing Date: January 22, 2026

{SIGNED AND SEALED}
[D. Grant Feasby]

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D. Grant Feasby, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

EUGENE PURITCH, P. ENG., FEC, CET

I, Eugene J. Puritch, P. Eng., FEC, CET, residing at 44 Turtlecreek Blvd., Brampton, Ontario, L6W 3X7, do hereby certify that:

1. I am an independent mining consultant and President of P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Stockwork Zone Pre-Feasibility Study and Underground Preliminary Economic Assessment of the El Tigre Silver-Gold Project, Sonora, Mexico", (The "Technical Report") with an effective date of June 20, 2025.
3. I am a graduate of The Haileybury School of Mines, with a Technologist Diploma in Mining, as well as obtaining an additional year of undergraduate education in Mine Engineering at Queen's University. In addition, I have also met the Professional Engineers of Ontario Academic Requirement Committee's Examination requirement for a Bachelor's degree in Engineering Equivalency. I am a mining consultant currently licensed by the: Professional Engineers and Geoscientists New Brunswick (License No. 4778); Professional Engineers, Geoscientists Newfoundland and Labrador (License No. 5998); Association of Professional Engineers and Geoscientists Saskatchewan (License No. 16216); Ontario Association of Certified Engineering Technicians and Technologists (License No. 45252); Professional Engineers of Ontario (License No. 100014010); Association of Professional Engineers and Geoscientists of British Columbia (License No. 42912); and Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (No. L3877). I am also a member of the National Canadian Institute of Mining and Metallurgy.

   I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

   I have practiced my profession continuously since 1978. My summarized career experience is as follows:
   - Mining Technologist - H.B.M.& S. and Inco Ltd., 1978-1980
   - Open Pit Mine Engineer – Cassiar Asbestos/Brinco Ltd., 1981-1983
   - Pit Engineer/Drill & Blast Supervisor – Detour Lake Mine, 1984-1986
   - Self-Employed Mining Consultant – Timmins Area, 1987-1988
   - Mine Designer/Resource Estimator – Dynatec/CMD/Bharti, 1989-1995
   - Self-Employed Mining Consultant/Resource-Reserve Estimator, 1995-2004
   - President – P&E Mining Consultants Inc, 2004-Present
4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for co-authoring Sections 1, 14, 25, 26, and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled "Pre-Feasibility Study of the El Tigre Silver-Gold Project, Sonora, Mexico" with an effective date of October 22, 2024, and "Technical Report and Updated Mineral Resource Estimate of the El Tigre Silver-Gold Project, Sonora, Mexico" with an effective date of September 12, 2023, and "NI 43-101 Technical Report and Updated Mineral Resource Estimate on the El Tigre Project Sonora, Mexico" for Oceanus Resources Corp., with an effective date of September 7, 2017.
8. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 20, 2025
Signing Date: January 22, 2026

{SIGNED AND SEALED}
[Eugene Puritch]

_____
Eugene Puritch, P.Eng., FEC, CET

CERTIFICATE OF QUALIFIED PERSON

D. GREGORY ROBINSON, P. ENG.

I, David Gregory (Greg) Robinson, P. Eng. (ON), residing at 14 Moorcrest Drive, Aurora, Ontario, Canada, do hereby certify that:

1. I am an independent engineering consultant contracted by P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Stockwork Zone Pre-Feasibility Study and Underground Preliminary Economic Assessment of the El Tigre Silver-Gold Project, Sonora, Mexico", (The "Technical Report") with an effective date of June 20, 2025.
3. I am a graduate of Dalhousie University, Queens University and Cornell University, and Professional Engineer of Ontario (License No. 100216726).

   I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

   I have practiced my profession continuously since 2008. My summarized career experience is as follows:
   - Associate Engineer, P&E Mining Consultants — Aug 2017 - Present
   - Mine Engineer, Lac des Iles Mine, North American Palladium — May 2016 – Jun 2017
   - Senior Underground Engineer, Phoenix Gold, Rubicon Minerals — Sep 2014 – Jan 2016
   - Mine Engineer, Diavik Diamond Mine, Rio Tinto Diamonds — Sep 2011 – Sep 2014
   - Mine Engineer, Bengalla Mine, Rio Tinto Coal and Allied — Dec 2008 – Sep 2011
   - EIT, Creighton Mine, Vale-Inco — May 2008 – Dec 2008
4. I have visited the Property that is the subject of this Technical Report on May 5 to 9, 2025.
5. I am responsible for authoring Sections 24.3.3, 24.3.4, 24.3.10, 24.3.11 and co-authoring Sections 1, 12, 24.3.8, 25, 26, 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101. I am independent of the Vendor and the Property.
7. I have had no prior involvement with the Property that is the subject of this Technical Report.
8. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 20, 2025
Signing Date: January 22, 2026

{SIGNED AND SEALED}
[D. Gregory Robinson]

_____
D. Gregory Robinson, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

WILLIAM STONE, PH.D., P.GEO.

I, William Stone, Ph.D., P.Geo, residing at 4361 Latimer Crescent, Burlington, Ontario, do hereby certify that:

1. I am an independent geological consultant working for P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Stockwork Zone Pre-Feasibility Study and Underground Preliminary Economic Assessment of the El Tigre Silver-Gold Project, Sonora, Mexico", (The "Technical Report") with an effective date of June 20, 2025.
3. I am a graduate of Dalhousie University with a Bachelor of Science (Honours) degree in Geology (1983). In addition, I have a Master of Science in Geology (1985) and a Ph.D. in Geology (1988) from the University of Western Ontario. I have worked as a geologist for a total of 35 years since obtaining my M.Sc. degree. I am a geological consultant currently licensed by the Professional Geoscientists of Ontario (License No 1569).

   I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

   My relevant experience for the purpose of the Technical Report is:
   - Contract Senior Geologist, LAC Minerals Exploration Ltd. 1985-1988
   - Post-Doctoral Fellow, McMaster University 1988-1992
   - Contract Senior Geologist, Outokumpu Mines and Metals Ltd. 1993-1996
   - Senior Research Geologist, WMC Resources Ltd. 1996-2001
   - Senior Lecturer, University of Western Australia 2001-2003
   - Principal Geologist, Geoinformatics Exploration Ltd. 2003-2004
   - Vice President Exploration, Nevada Star Resources Inc. 2005-2006
   - Vice President Exploration, Goldbrook Ventures Inc. 2006-2008
   - Vice President Exploration, North American Palladium Ltd. 2008-2009
   - Vice President Exploration, Magma Metals Ltd. 2010-2011
   - President & COO, Pacific North West Capital Corp. 2011-2014
   - Consulting Geologist 2013-2017
   - Senior Project Geologist, Anglo American 2017-2019
   - Consulting Geoscientist 2020-Present
4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Sections 4, 5, 6, 7, 8, 9, and 23, and co-authoring Sections 1, 10, 25, 26, and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled "Pre-Feasibility Study of the El Tigre Silver-Gold Project, Sonora, Mexico" with an effective date of October 22, 2024, and "Technical Report and Updated Mineral Resource Estimate of the El Tigre Silver-Gold Project, Sonora, Mexico" with an effective date of September 12, 2023.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 20, 2025
Signing Date: January 22, 2026
{SIGNED AND SEALED}
[William Stone]

_____
William E. Stone, Ph.D., P.Geo.

CERTIFICATE OF QUALIFIED PERSON

DAVID SALARI, P.ENG.

I, David Salari, P.Eng., of 125 Bronte Road, Unit 503, Oakville, Ontario, Canada, L6L 0H1, do hereby certify that:

1. I am an independent metallurgical engineer with an office at Suite 300-10, 1100 Burloak Drive, Burlington, Ontario, Canada, L6L 2Y8.
2. This certificate applies to the Technical Report titled "Stockwork Zone Pre-Feasibility Study and Underground Preliminary Economic Assessment of the El Tigre Silver-Gold Project, Sonora, Mexico", (The "Technical Report") with an effective date of June 20, 2025.
3. I am a graduate of The University of Toronto with a Bachelor's of Applied Science (BASc) – Metallurgy and Material Science. I have been actively involved in mining and mineral processing since 1980 with extensive experience in metallurgical and mill testing and design, mill capital and operating costs, construction, commissioning, and mill operations.

   I am a member in good standing of the Professional Engineers Ontario - #40416505 and I am the designated P.Eng. for D.E.N.M. Engineering Ltd. – Certificate of Authorization – Professional Engineers Ontario - #100102038 and Designation as a Consulting Engineer – Professional Engineers Ontario - # 4012.

   I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
4. I have visited the Property that is the subject of this Technical Report on February 1 to 2, 2024 for the purpose to inspect the proposed location of the open pit, heap leach pad(s), ponds, and process recovery facility. Inspection and location of the mineral testing holes was also reviewed and associated drill core. Site access, water source and other associated infrastructure was also addressed and inspected.
5. I am responsible for authoring Sections 13, 17, 24.3.2, 24.3.5 and co-authoring Sections 1, 18, 21, 24.3.8, 25, 26, and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have had prior involvement with the Project that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled "Pre-Feasibility Study of the El Tigre Silver-Gold Project, Sonora, Mexico" with an effective date of October 22, 2024, and "Technical Report and Updated Mineral Resource Estimate of the El Tigre Silver-Gold Project, Sonora, Mexico" with an effective date of September 12, 2023.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 20, 2025
Signing Date: January 22, 2026

{SIGNED AND SEALED}
[David Salari]

_____
David Salari, P.Eng.

CERTIFICATE OF QUALIFIED PERSON JAMES SMITH

I, James Smith, state that:

(a) I am a Senior Geotechnical Engineer at:
WSP Canada Inc.
6925 Century Ave, Suite 600
Mississauga, Ontario
L5N 7K2 Canada

(b) This certificate applies to the technical report titled "Stockwork Zone Pre-Feasibility Study and Underground Preliminary Economic Assessment of the El Tigre Silver-Gold Project, Sonora, Mexico" with an effective date of: June 20, 2025 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of University of Waterloo with a B.ASc. in Geological Engineering from 2012, I am a member in good standing of the Professional Engineers of Ontario (#100523579). My relevant experience after graduation, for the purpose of the Technical Report, includes over 12 years of experience in geotechnical engineering in the areas of mine stability, influences of hydrogeological conditions on rock mass stability, and open pit geotechnical engineering of mineral projects nationally and internationally in a variety of commodities through 12 years of consulting experience with a strong focus on gold and base metals related projects.

(d) I did not complete a personal inspection of the property described in the Technical Report.

(e) I am responsible for co-authoring portions of Sections 1, 2, 15, 25, 26 and 27 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) I have had prior involvement with the Property that is the subject of this Technical Report. I was a "Qualified Person" for a Technical Report titled "Pre-Feasibility Study of the El Tigre Silver-Gold Project, Sonora, Mexico" with an effective date of October 22, 2024.

(h) I have read NI 43-101 and the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101; and

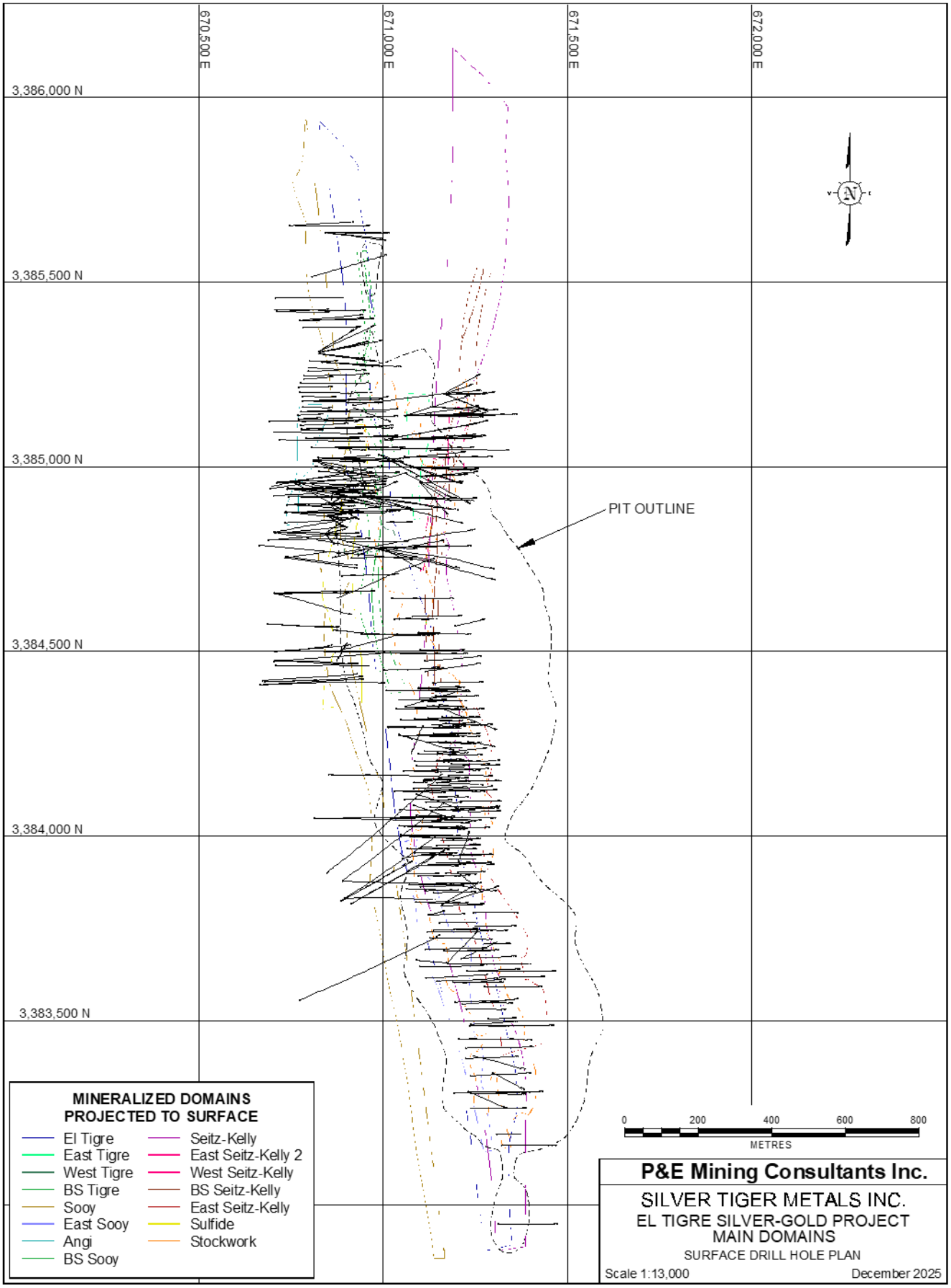
(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Mississauga, Ontario this January 22, 2026.

Signed by James Smith

James Smith; P.Eng.

APPENDIX A DRILL HOLE, CHANNEL AND TRENCH PLANS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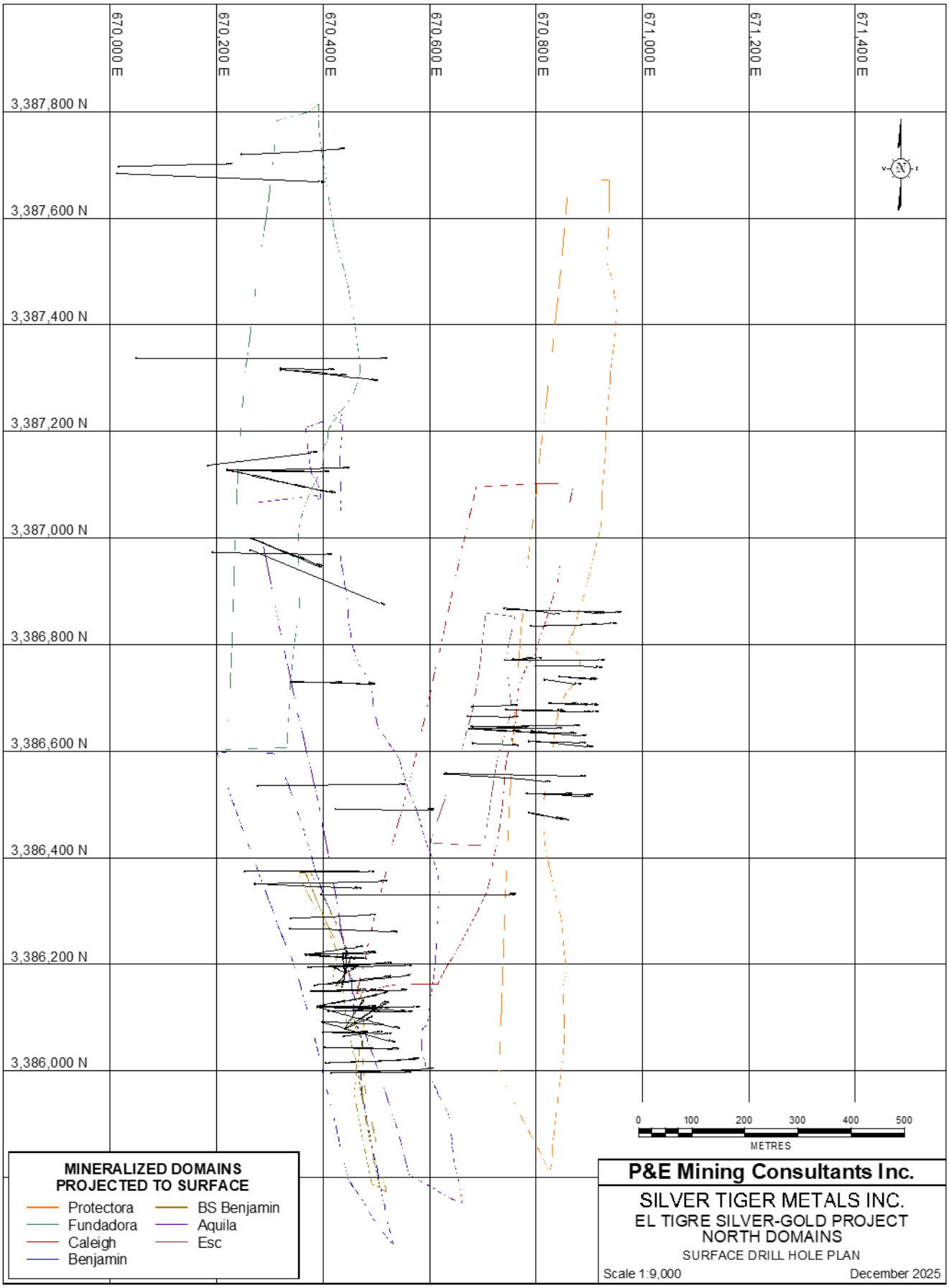
3,383,500 N
PIT OUTLINE
MINERALIZED DOMAINS PROJECTED TO SURFACE
El Tigre
East Tigre
West Tigre
BS Tigre
Sooy
East Sooy
Angi
BS Sooy
Seitz-Kelly
East Seitz-Kelly 2
West Seitz-Kelly
BS Seitz-Kelly
East Seitz-Kelly
Sulfide
Stockwork
0
200
400
600
800
METRES
P&E Mining Consultants Inc.
SILVER TIGER METALS INC.
EL TIGRE SILVER-GOLD PROJECT
MAIN DOMAINS
SURFACE DRILL HOLE PLAN
Scale 1:13,000
December 2025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3,386,800 N
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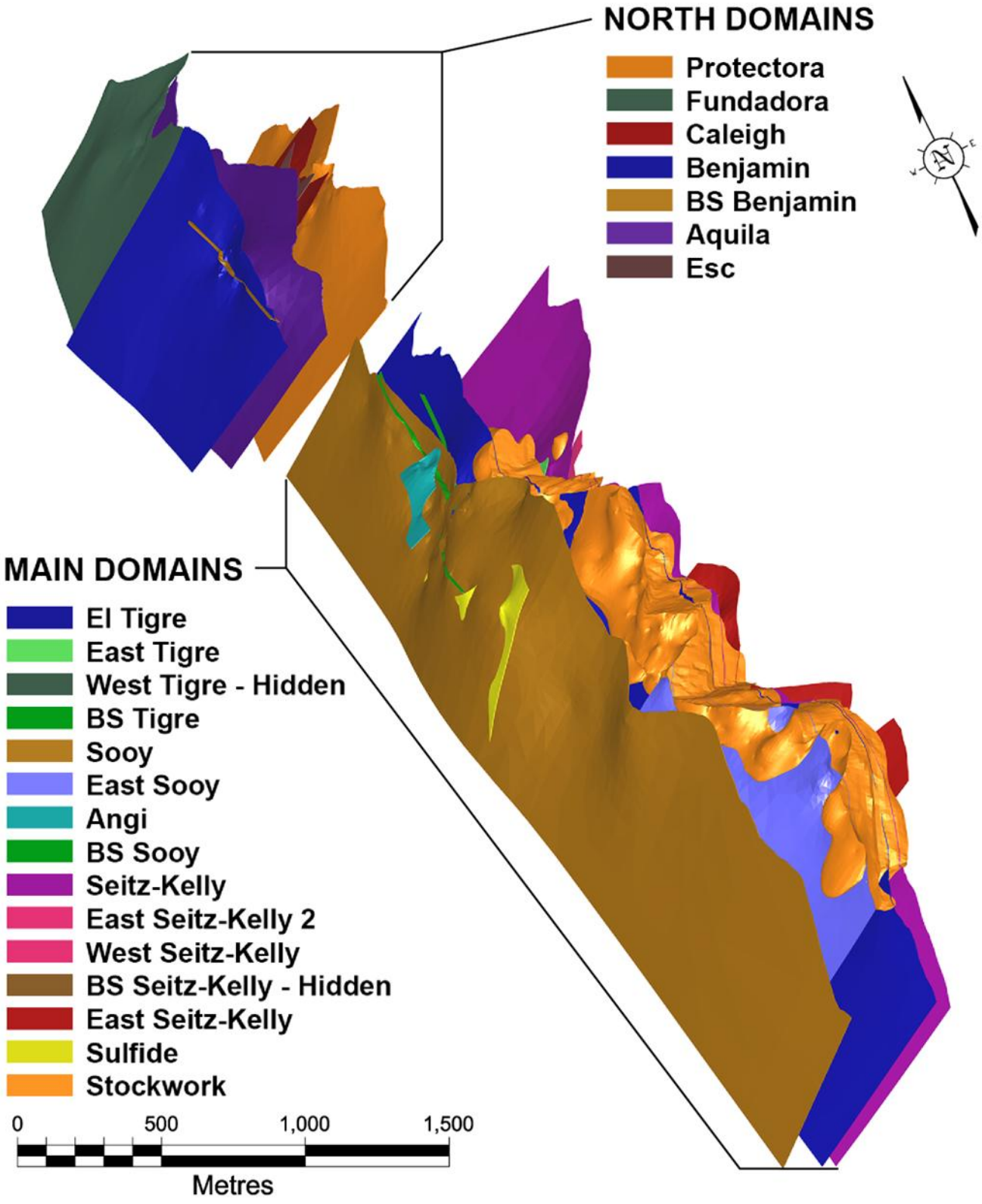
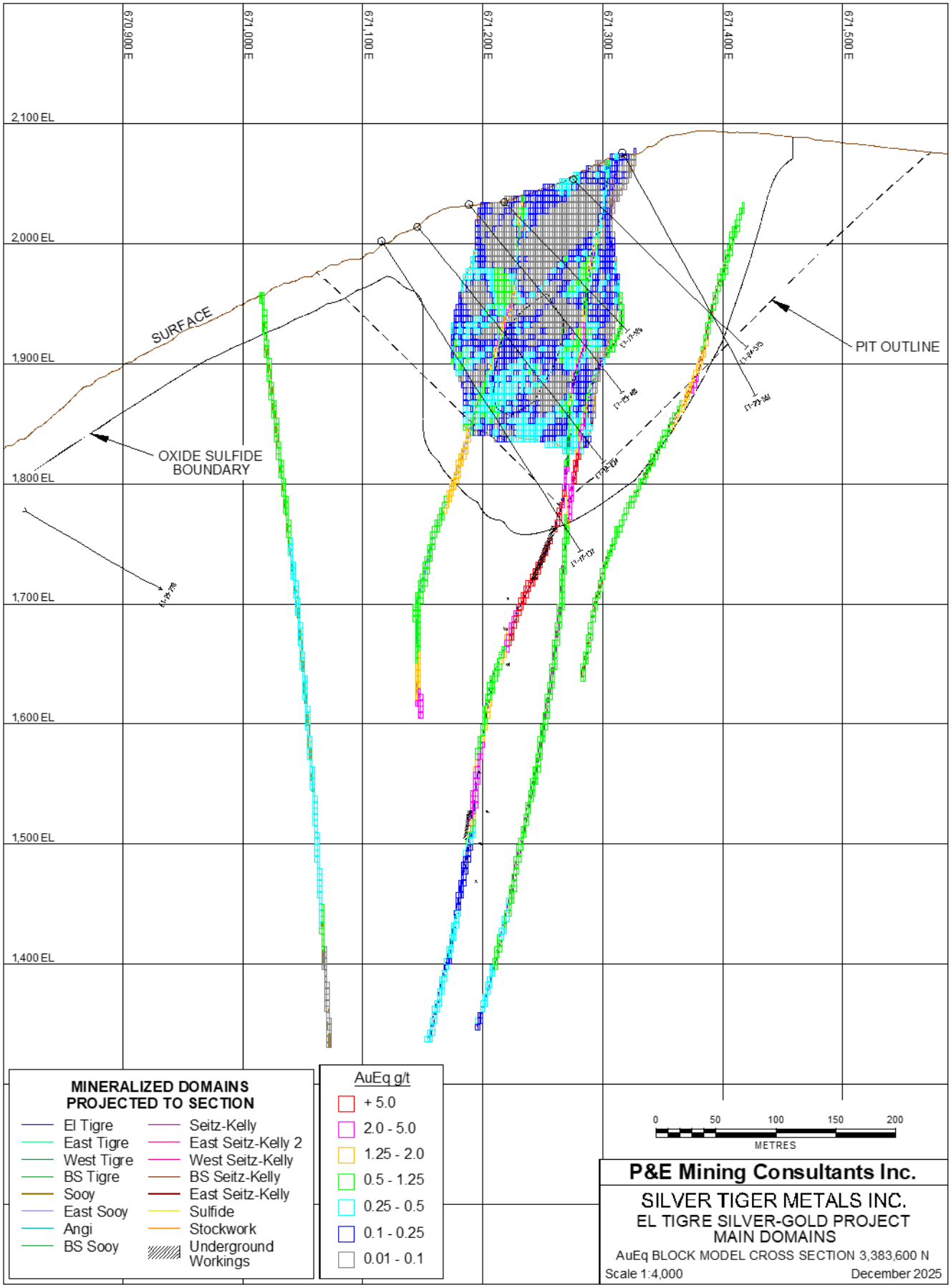
0 100 200 300 400 500
METRES
MINERALIZED DOMAINS PROJECTED TO SURFACE
Protectora
Fundadora
Caleigh
Benjamin
BS Benjamin
Aquila
Esc
P&E Mining Consultants Inc.
SILVER TIGER METALS INC.
EL TIGRE SILVER-GOLD PROJECT
NORTH DOMAINS
SURFACE DRILL HOLE PLAN
Scale 1:9,000
December 2025

APPENDIX B 3-D DOMAINS

EL TIGRE SILVER-GOLD PROJECT
3D DOMAINS



APPENDIX C AUEQ BLOCK MODEL CROSS-SECTIONS AND PLANS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1,800 EL
1,700 EL
1,600 EL
1,500 EL
1,400 EL
SURFACE
OXIDE SULFIDE BOUNDARY
PIT OUTLINE
MINERALIZED DOMAINS PROJECTED TO SECTION
El Tigre
East Tigre
West Tigre
BS Tigre
Sooy
East Sooy
Angi
BS Sooy
Seitz-Kelly
East Seitz-Kelly 2
West Seitz-Kelly
BS Seitz-Kelly
East Seitz-Kelly
Sulfide
Stockwork
Underground Workings
AuEq g/t
+ 5.0
2.0 - 5.0
1.25 - 2.0
0.5 - 1.25
0.25 - 0.5
0.1 - 0.25
0.01 - 0.1
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
SILVER TIGER METALS INC.
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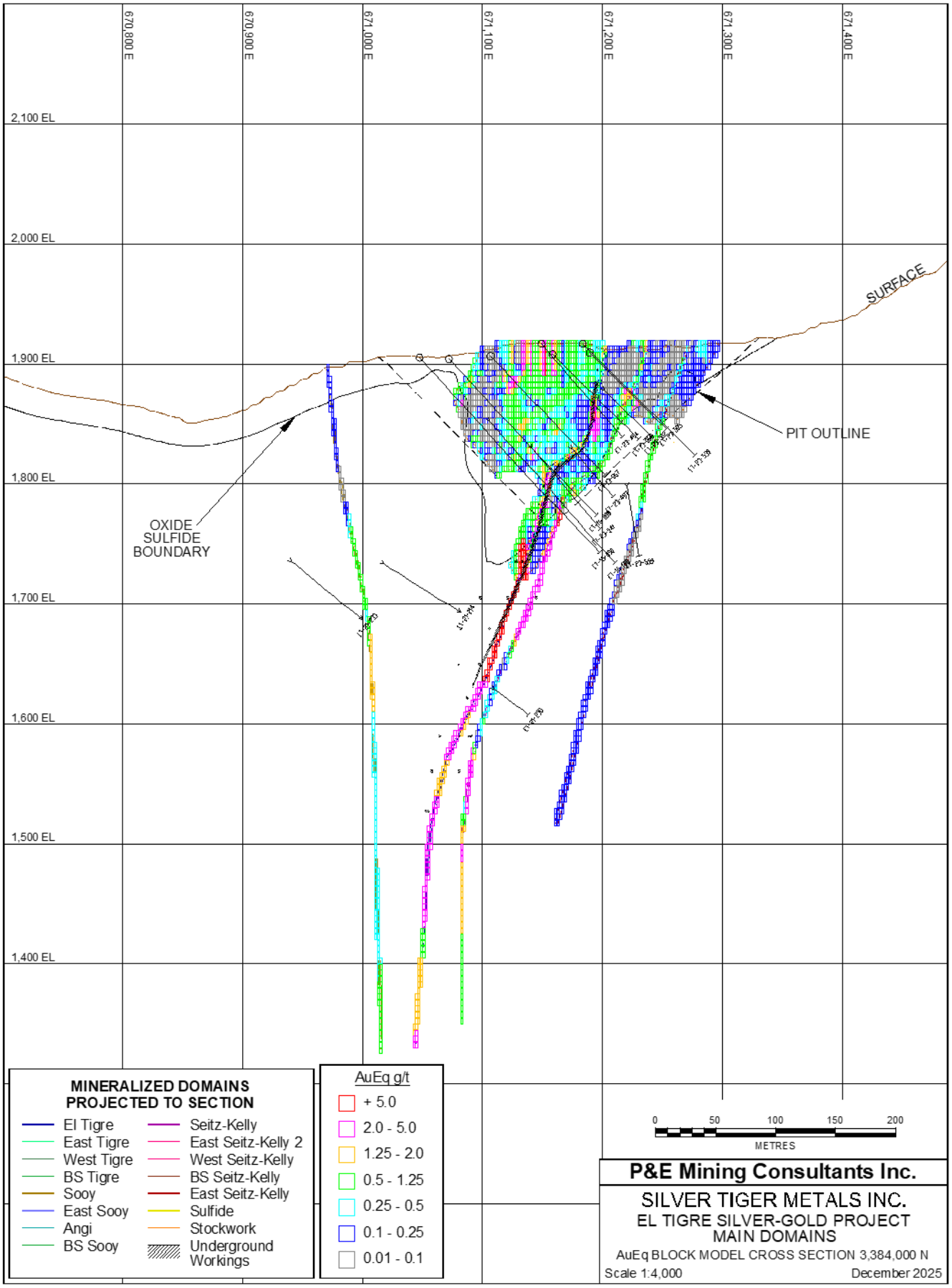
December 2025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1,700 EL
1,600 EL
1,500 EL
1,400 EL
SURFACE
PIT OUTLINE
OXIDE
SULFIDE
BOUNDARY
MINERALIZED DOMAINS
PROJECTED TO SECTION
El Tigre
East Tigre
West Tigre
BS Tigre
Sooy
East Sooy
Angi
BS Sooy
Seitz-Kelly
East Seitz-Kelly 2
West Seitz-Kelly
BS Seitz-Kelly
East Seitz-Kelly
Sulfide
Stockwork
Underground Workings
AuEq g/t
+ 5.0
2.0 - 5.0
1.25 - 2.0
0.5 - 1.25
0.25 - 0.5
0.1 - 0.25
0.01 - 0.1
0
50
100
150
200
METRES
P&E Mining Consultants Inc.
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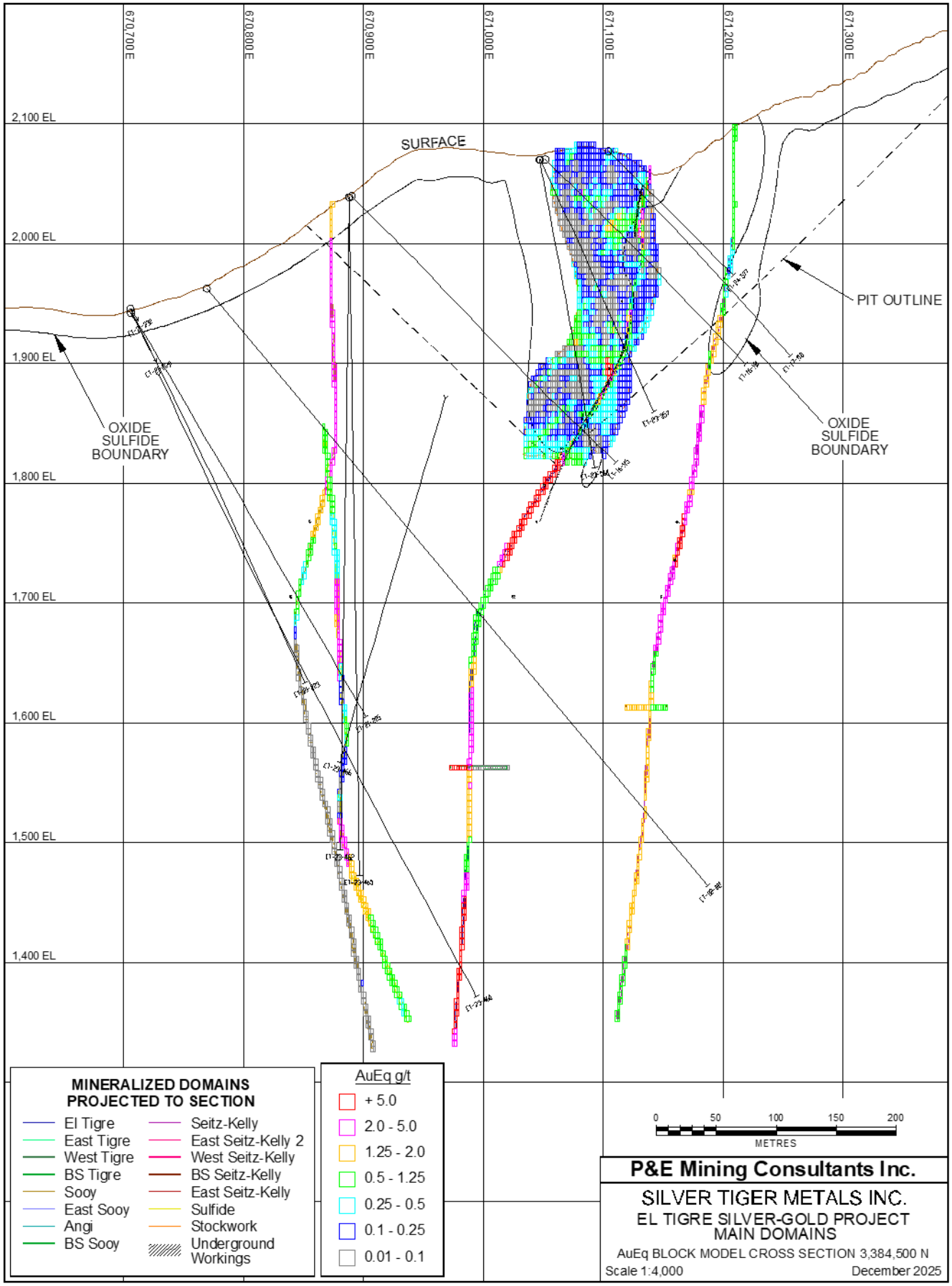
December 2025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SURFACE
PIT OUTLINE
OXIDE
SULFIDE
BOUNDARY
OXIDE
SULFIDE
BOUNDARY
MINERALIZED DOMAINS
PROJECTED TO SECTION
El Tigre
East Tigre
West Tigre
BS Tigre
Sooy
East Sooy
Angi
BS Sooy
Seitz-Kelly
East Seitz-Kelly 2
West Seitz-Kelly
BS Seitz-Kelly
East Seitz-Kelly
Sulfide
Stockwork
Underground Workings
AuEq g/t
+ 5.0
2.0 - 5.0
1.25 - 2.0
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0.25 - 0.5
0.1 - 0.25
0.01 - 0.1
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METRES
P&E Mining Consultants Inc.
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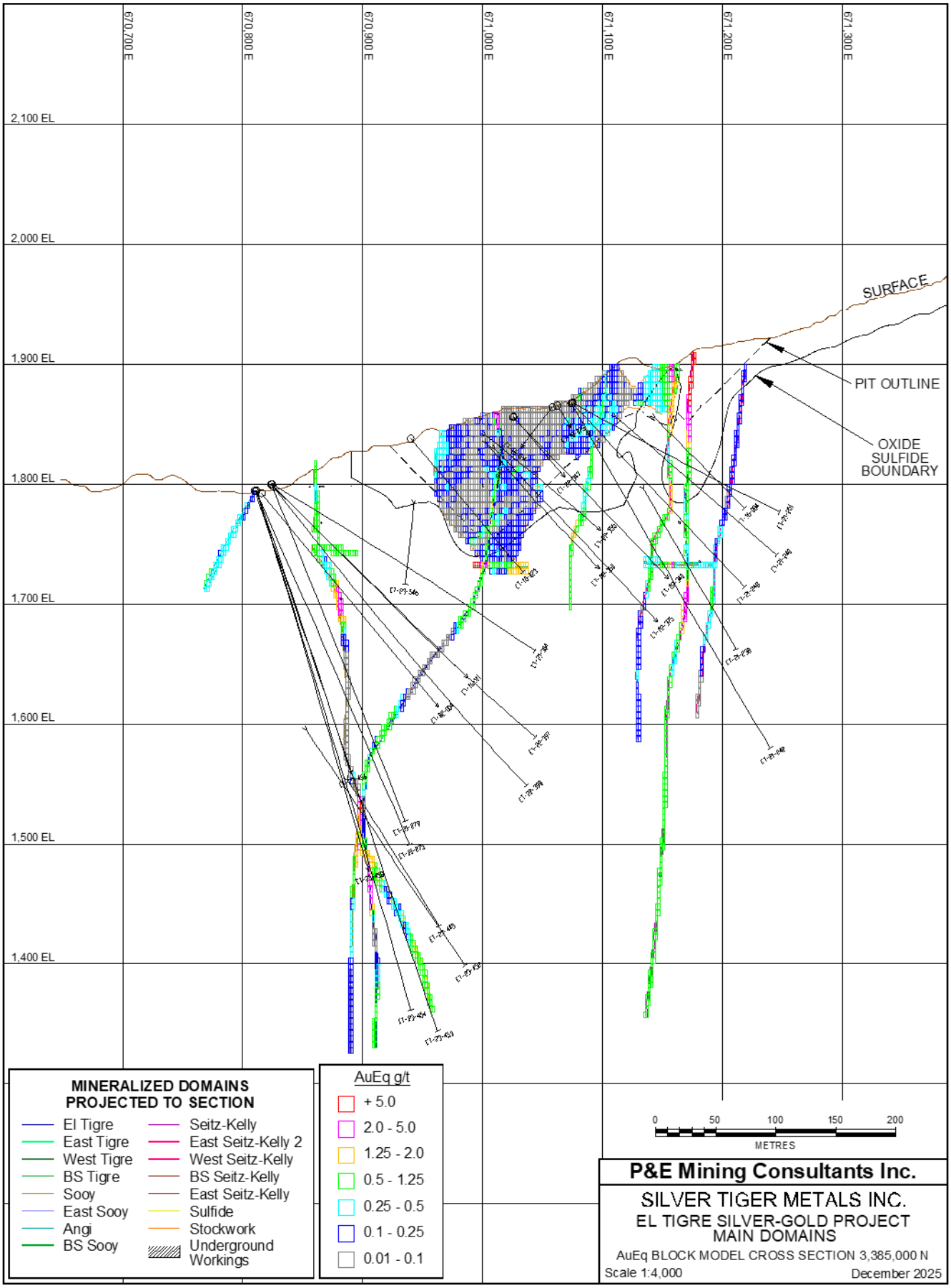
December 2025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SURFACE
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OXIDE
SULFIDE
BOUNDARY
MINERALIZED DOMAINS
PROJECTED TO SECTION
El Tigre
East Tigre
West Tigre
BS Tigre
Sooy
East Sooy
Angi
BS Sooy
Seitz-Kelly
East Seitz-Kelly 2
West Seitz-Kelly
BS Seitz-Kelly
East Seitz-Kelly
Sulfide
Stockwork
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Workings
AuEq g/t
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METRES
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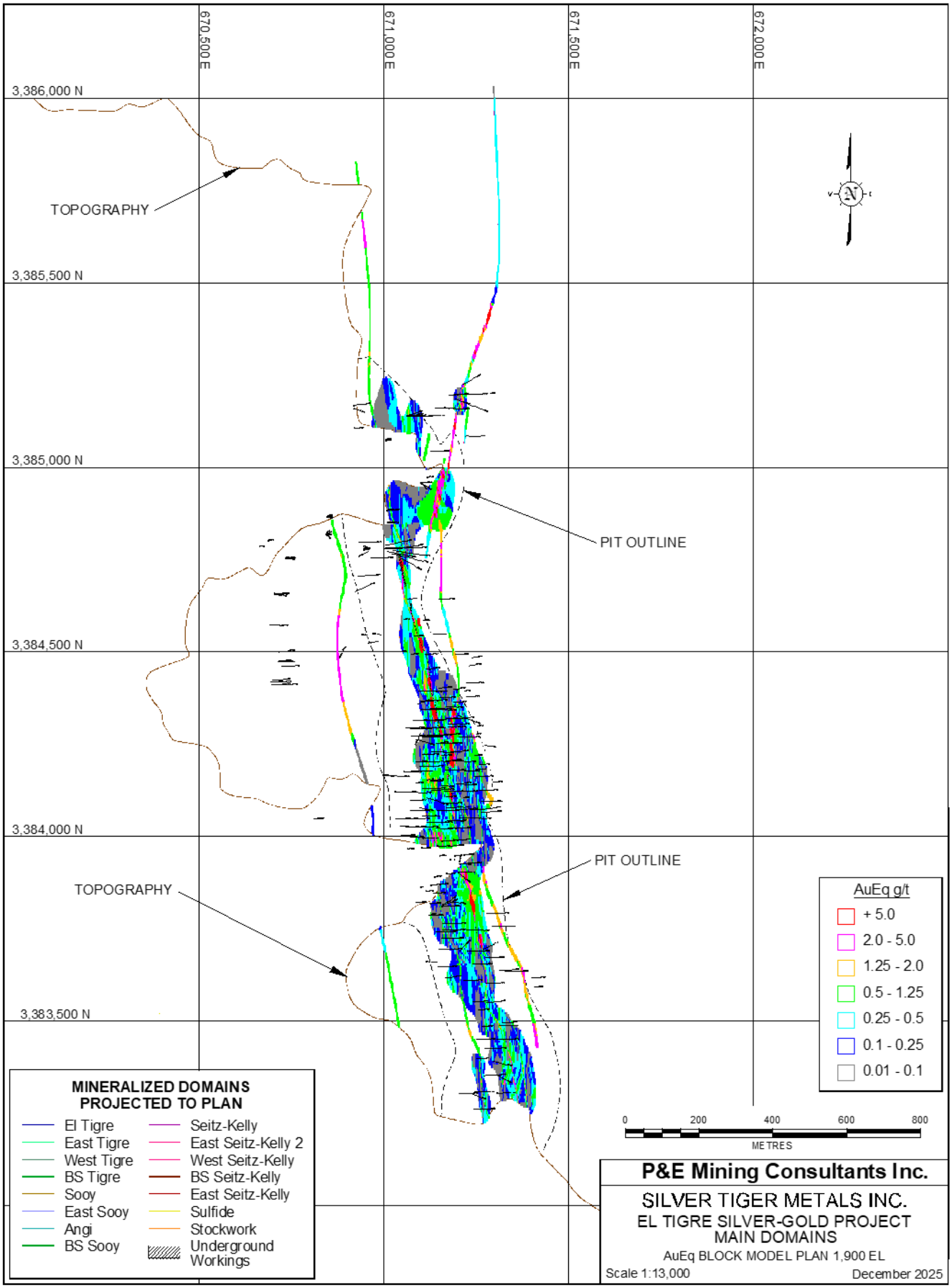
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Sulfide
Stockwork
Underground Workings
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METRES
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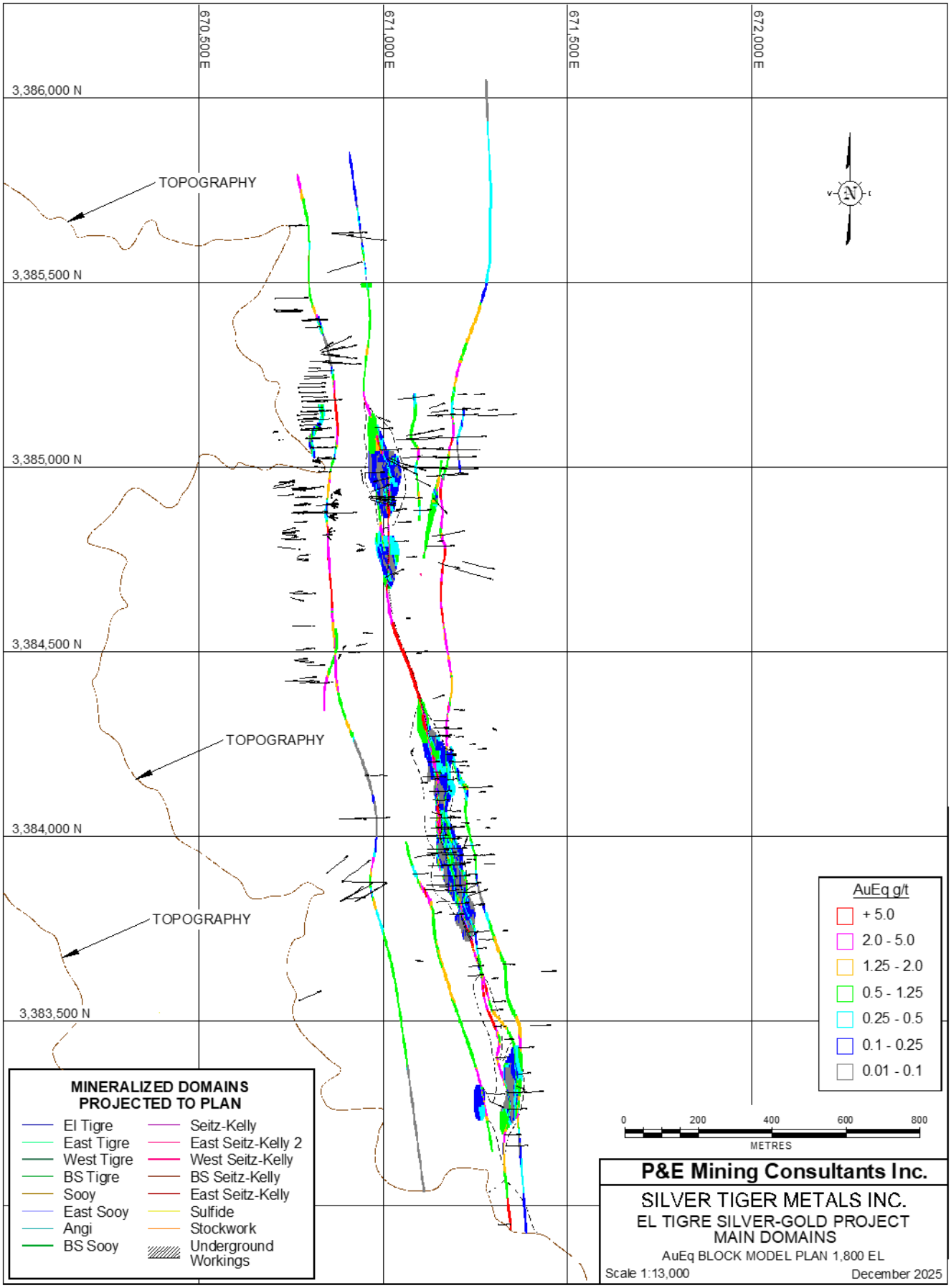
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BS Tigre
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BS Sooy
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West Seitz-Kelly
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East Seitz-Kelly
Sulfide
Stockwork
Underground Workings
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METRES
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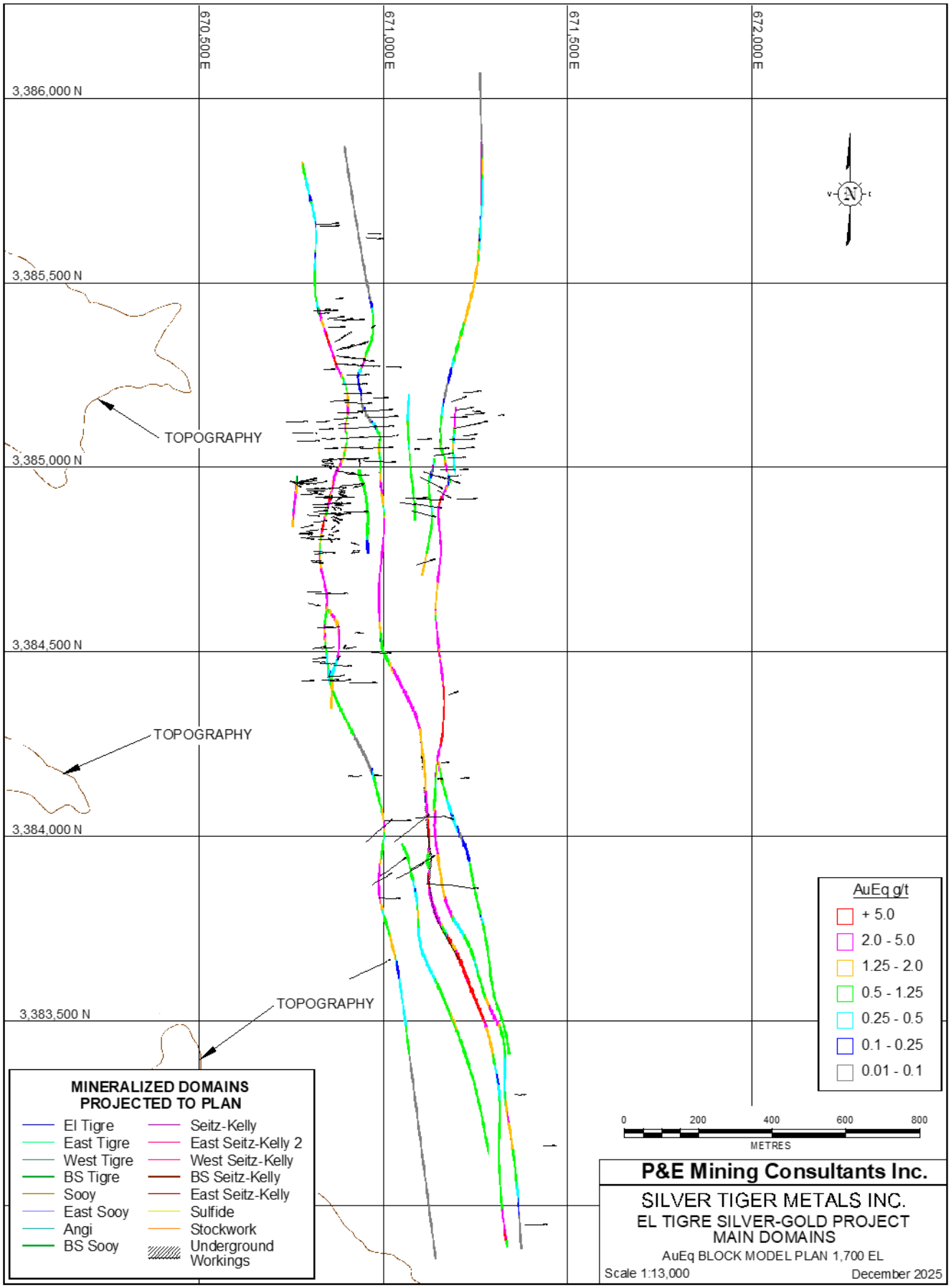
December 2025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BS Tigre
Sooy
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BS Sooy
Seitz-Kelly
East Seitz-Kelly 2
West Seitz-Kelly
BS Seitz-Kelly
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Sulfide
Stockwork
Underground Workings
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600
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METRES
P&E Mining Consultants Inc.
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MAIN DOMAINS
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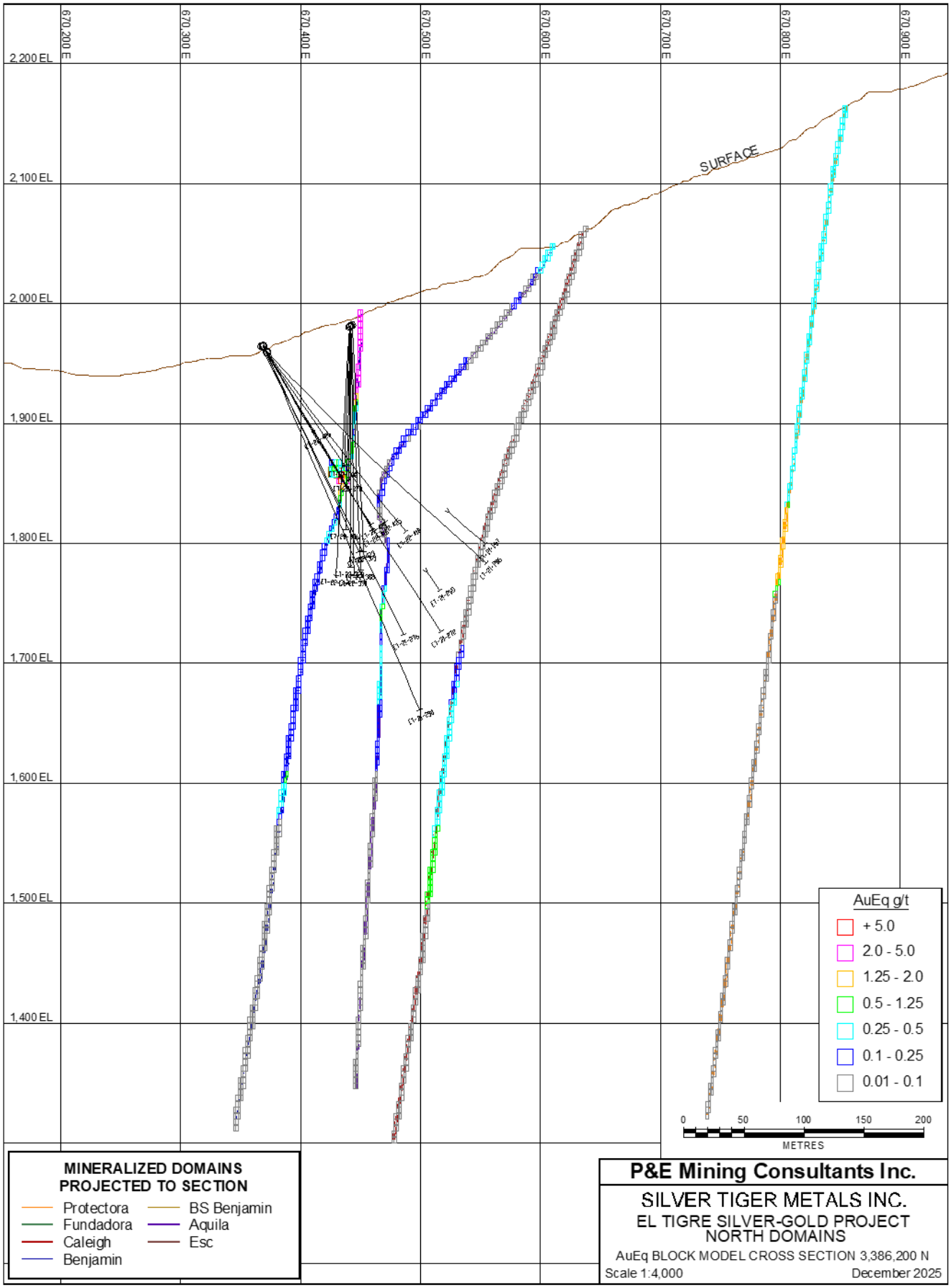
December 2025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0.1 - 0.25
0.01 - 0.1
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50
100
150
200
METRES
MINERALIZED DOMAINS PROJECTED TO SECTION
Protectora
Fundadora
Caleigh
Benjamin
BS Benjamin
Aquila
Esc
P&E Mining Consultants Inc.
SILVER TIGER METALS INC.
EL TIGRE SILVER-GOLD PROJECT
NORTH DOMAINS
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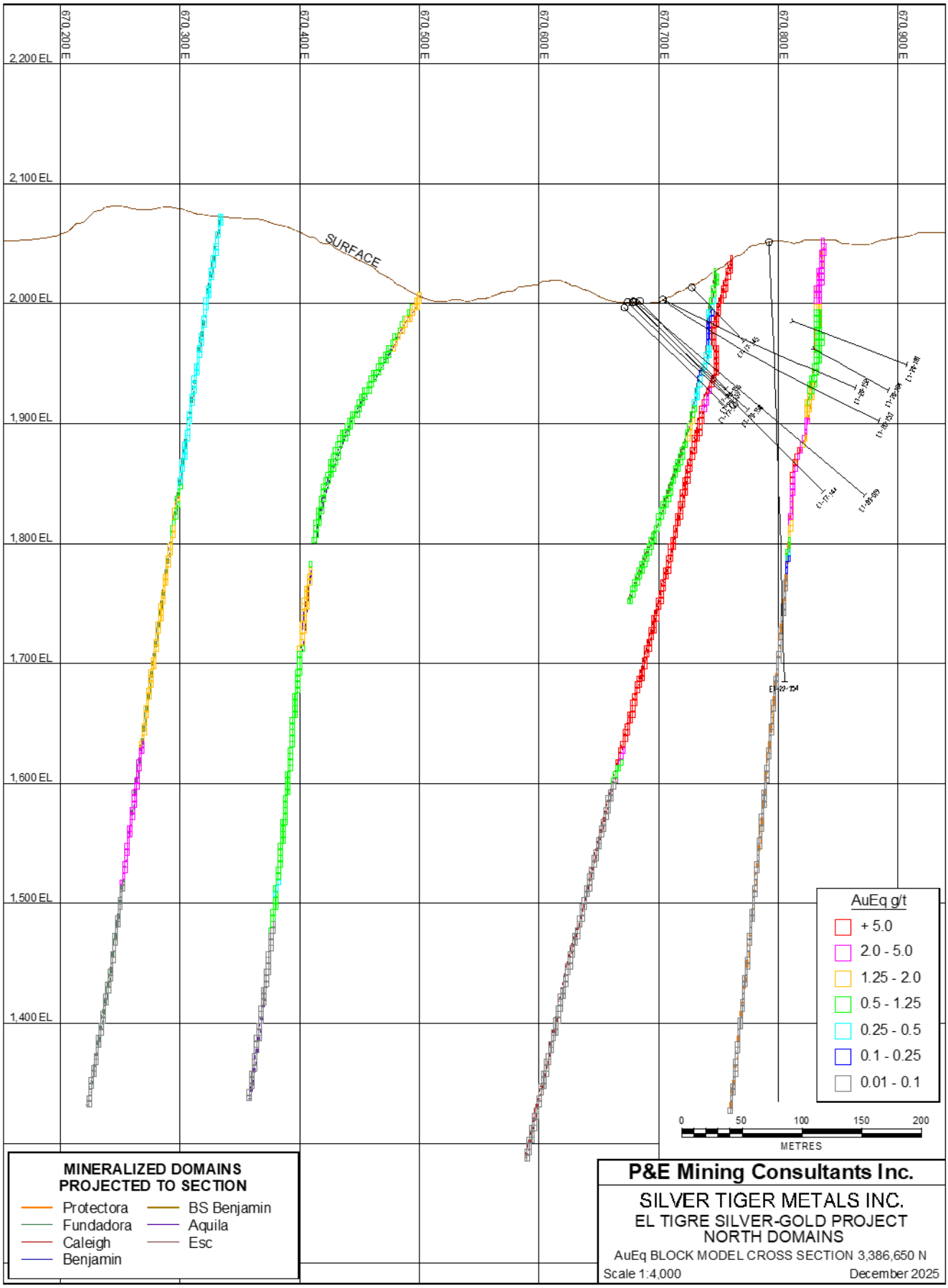
Scale 1:4,000
December 2025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0.1 - 0.25
0.01 - 0.1
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50
100
150
200
METRES
MINERALIZED DOMAINS PROJECTED TO SECTION
Protectora
Fundadora
Caleigh
Benjamin
BS Benjamin
Aquila
Esc
P&E Mining Consultants Inc.
SILVER TIGER METALS INC.
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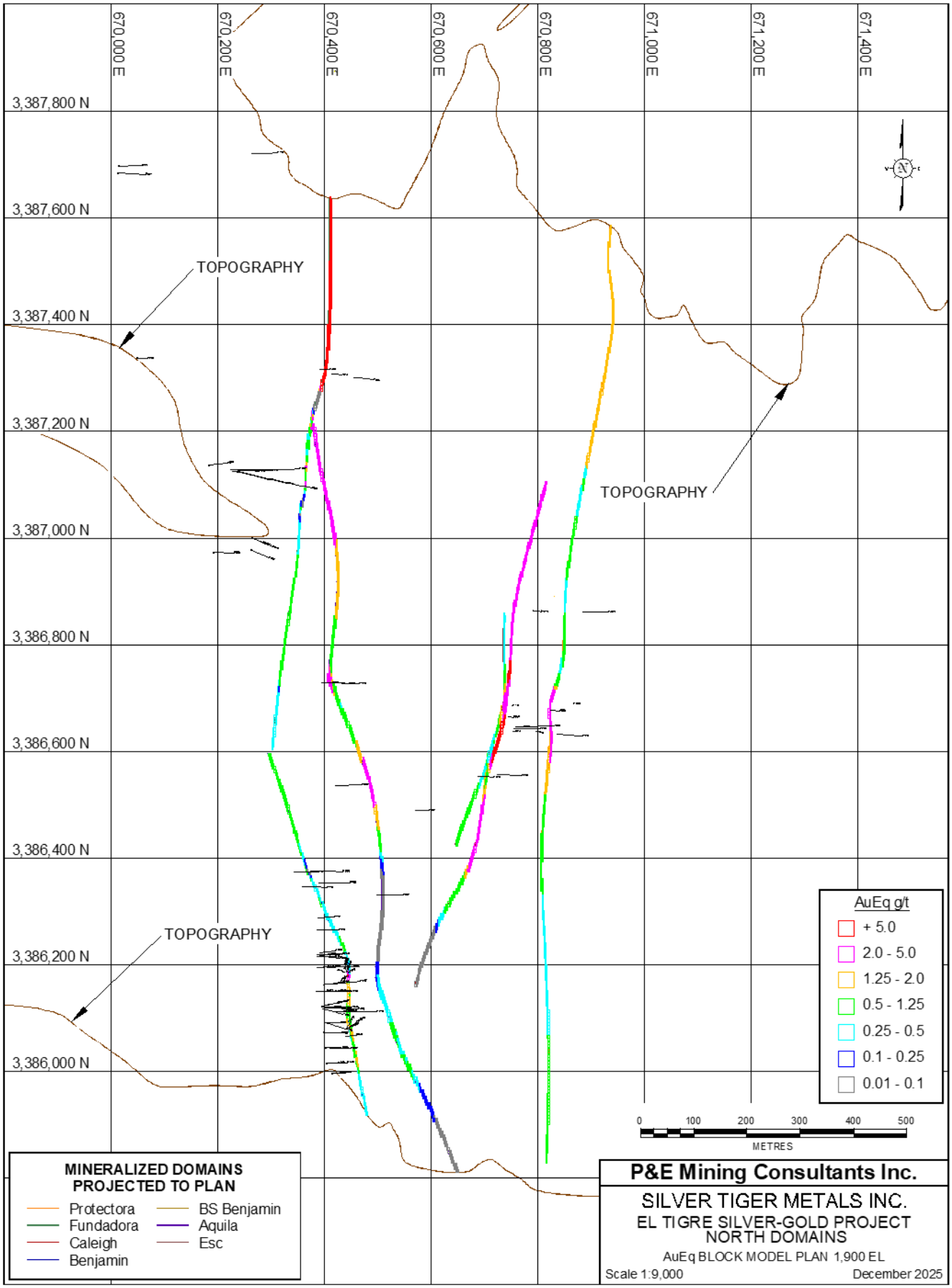
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TOPOGRAPHY
TOPOGRAPHY
TOPOGRAPHY
AuEq g/t
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0.1 - 0.25
0.01 - 0.1
0 100 200 300 400 500
METRES
MINERALIZED DOMAINS PROJECTED TO PLAN
Protectora
Fundadora
Caleigh
Benjamin
BS Benjamin
Aquila
Esc
P&E Mining Consultants Inc.
SILVER TIGER METALS INC.
EL TIGRE SILVER-GOLD PROJECT
NORTH DOMAINS
AuEq BLOCK MODEL PLAN 1,900 EL
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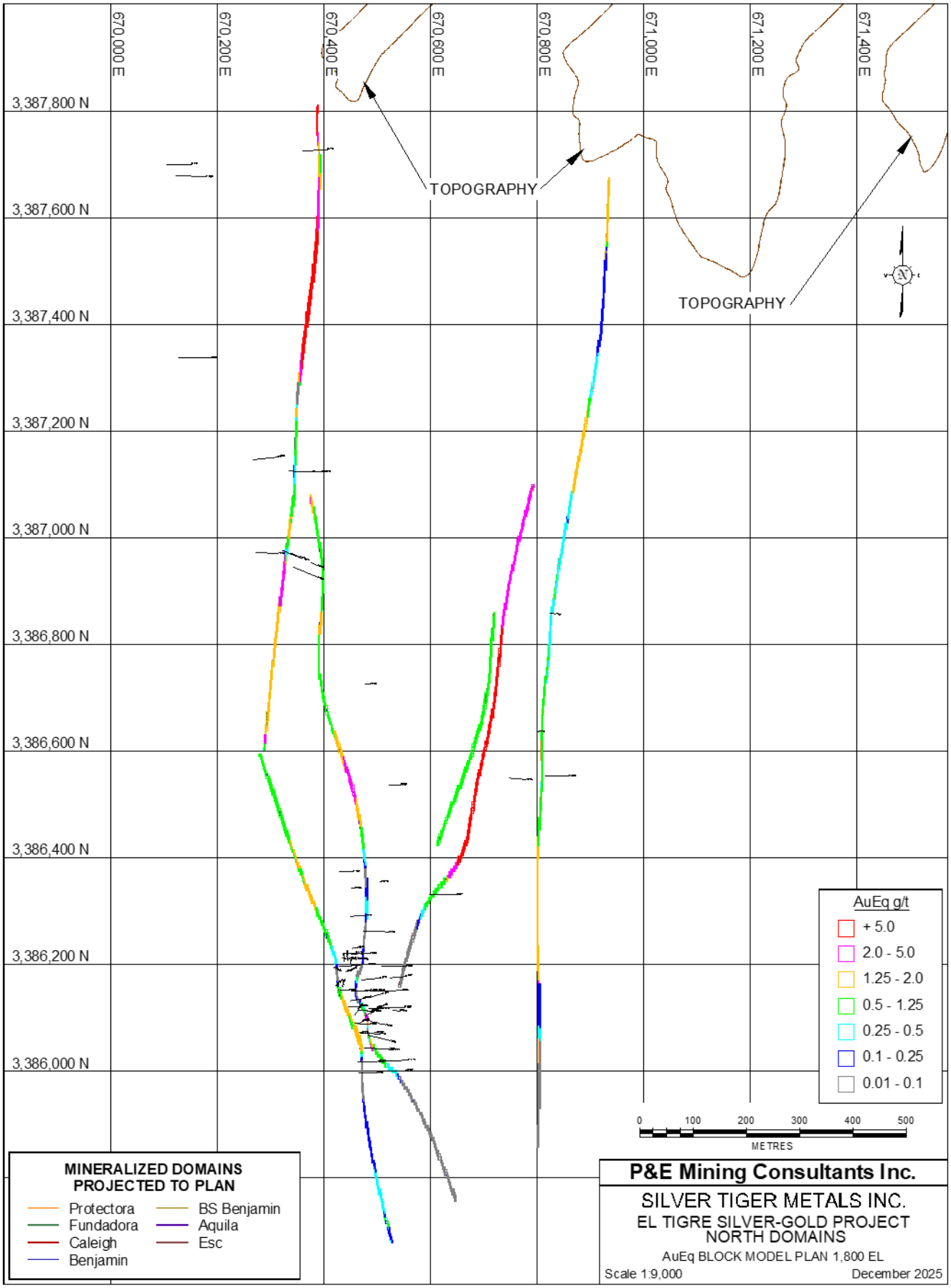
December 2025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TOPOGRAPHY
TOPOGRAPHY
AuEq g/t
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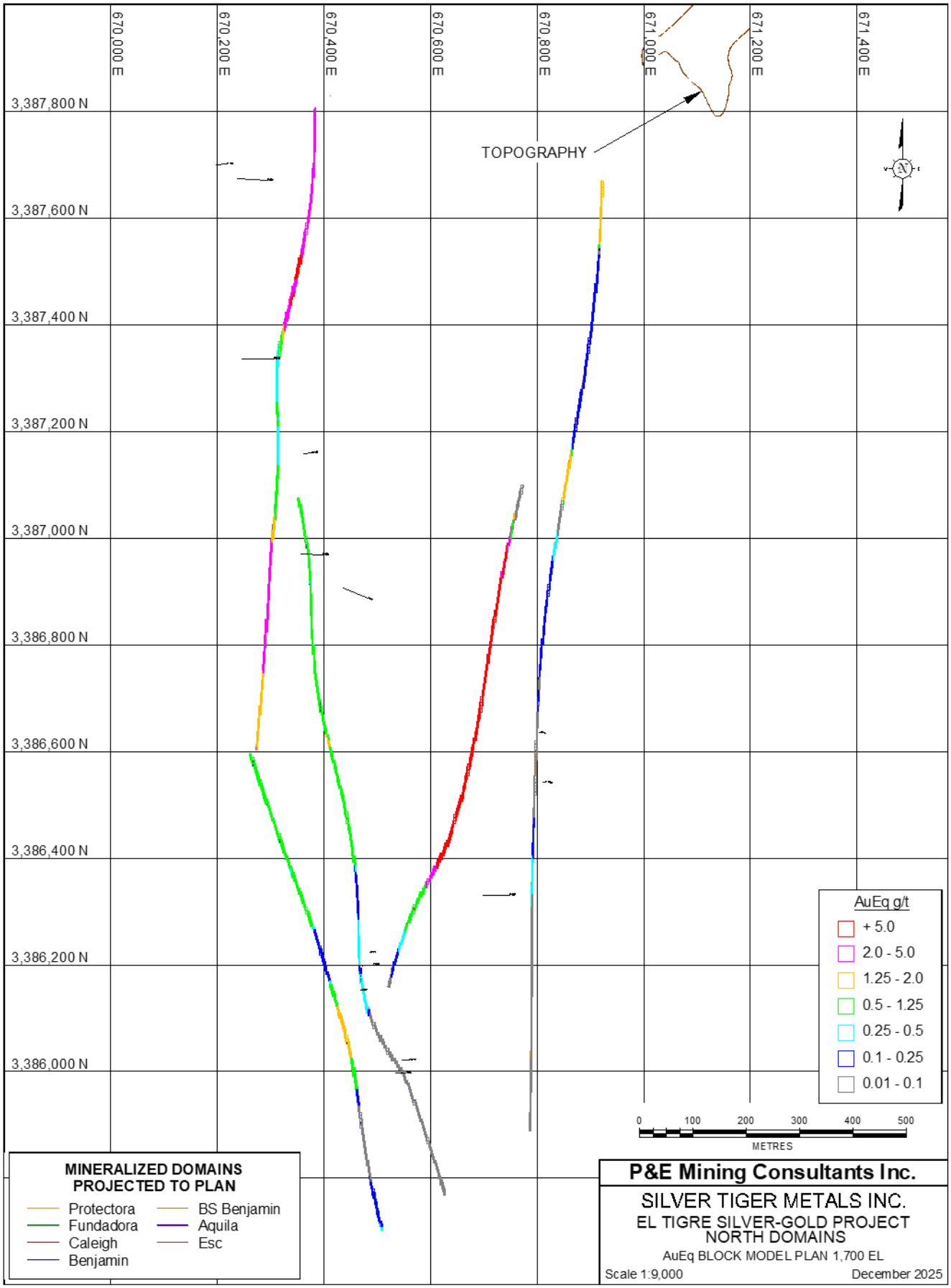
0.01 - 0.1
0 100 200 300 400 500
METRES
MINERALIZED DOMAINS PROJECTED TO PLAN
Protectora
Fundadora
Caleigh
Benjamin
BS Benjamin
Aquila
Esc
P&E Mining Consultants Inc.
SILVER TIGER METALS INC.
EL TIGRE SILVER-GOLD PROJECT
NORTH DOMAINS
AuEq BLOCK MODEL PLAN 1,800 EL
Scale 1:9,000
December 2025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TOPOGRAPHY
AuEq g/t
+ 5.0
2.0 - 5.0
1.25 - 2.0
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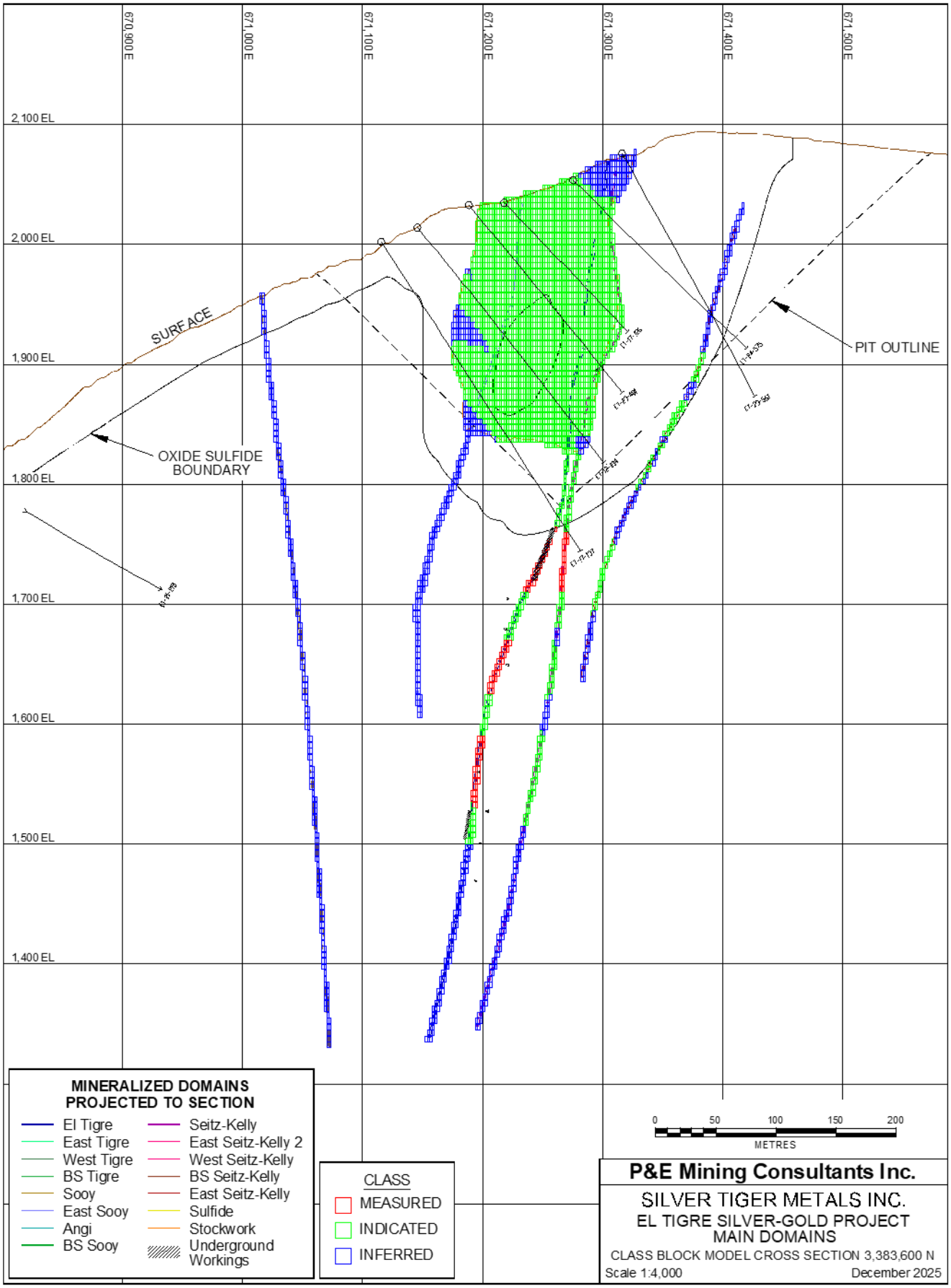
0 100 200 300 400 500
METRES
MINERALIZED DOMAINS PROJECTED TO PLAN
Protectora
Fundadora
Caleigh
Benjamin
BS Benjamin
Aquila
Esc
P&E Mining Consultants Inc.
SILVER TIGER METALS INC.
EL TIGRE SILVER-GOLD PROJECT
NORTH DOMAINS
AuEq BLOCK MODEL PLAN 1,700 EL
Scale 1:9,000
December 2025

APPENDIX D CLASSIFICATION BLOCK MODEL CROSS-SECTIONS AND PLANS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El Tigre
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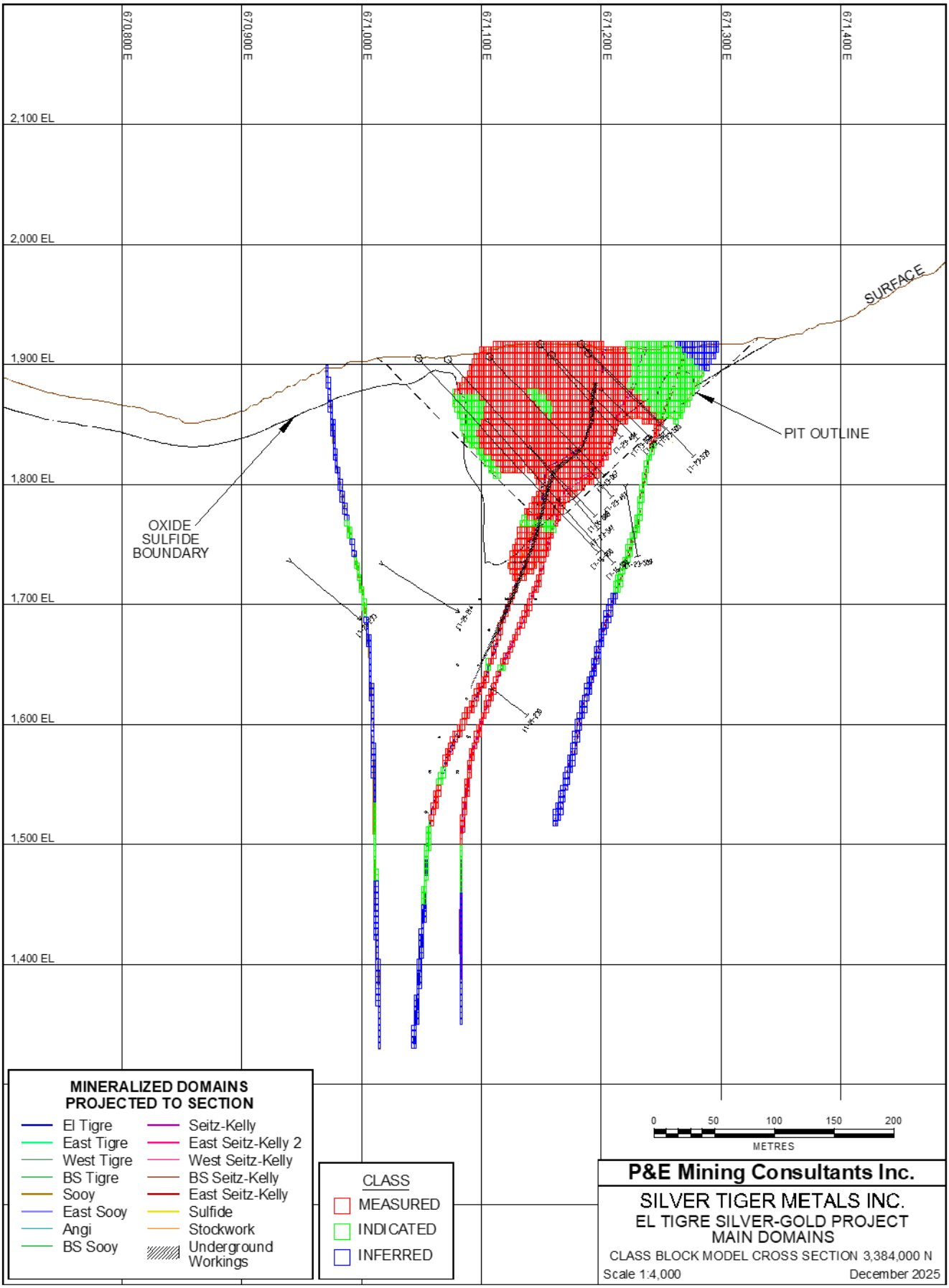
P&E Mining Consultants Inc.
SILVER TIGER METALS INC.
EL TIGRE SILVER-GOLD PROJECT
MAIN DOMAINS
CLASS BLOCK MODEL CROSS SECTION 3,383,600 N
Scale 1:4,000
December 2025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Sulfide
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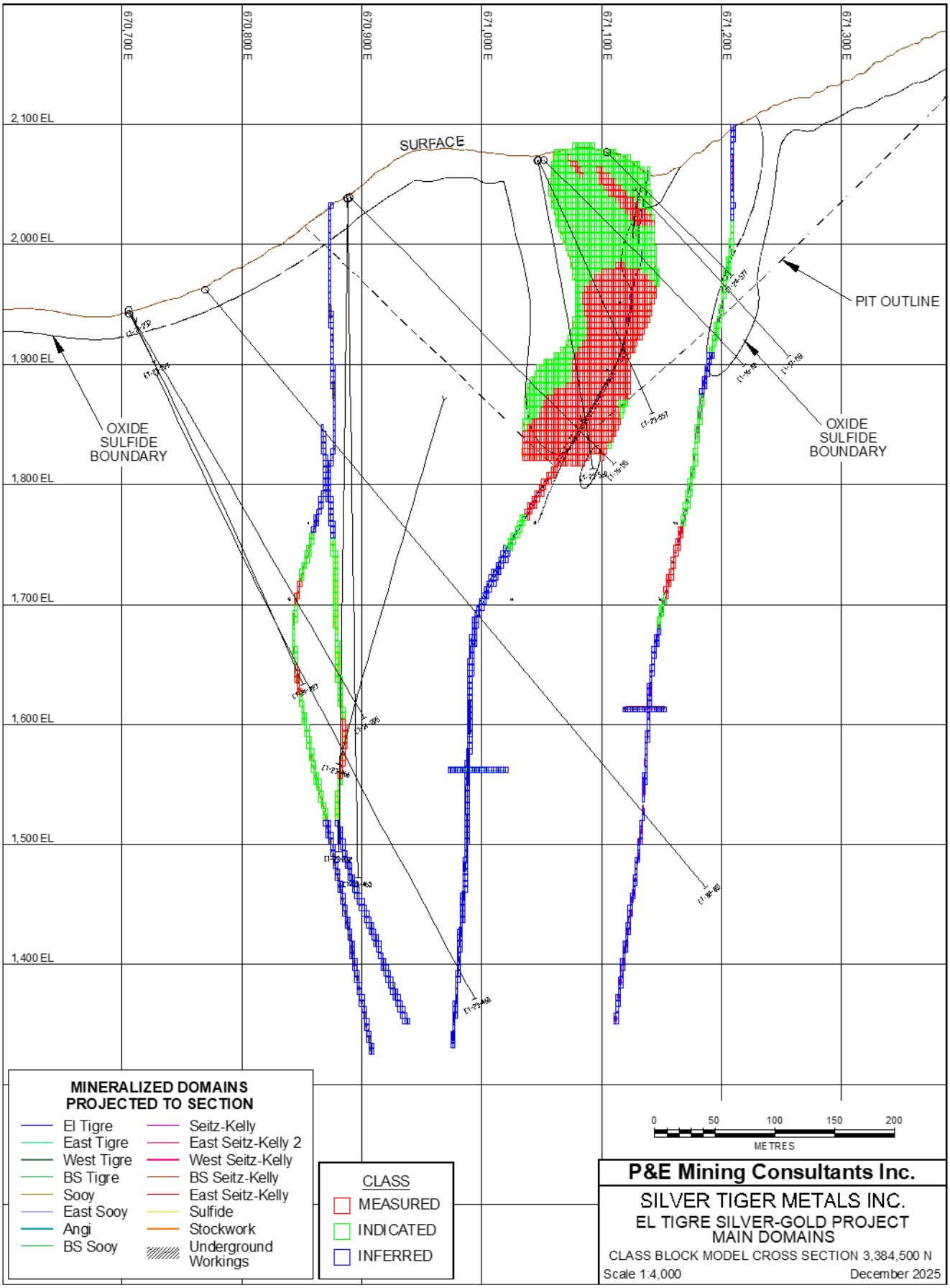
P&E Mining Consultants Inc.
SILVER TIGER METALS INC.
EL TIGRE SILVER-GOLD PROJECT
MAIN DOMAINS
CLASS BLOCK MODEL CROSS SECTION 3,384,000 N
Scale 1:4,000
December 2025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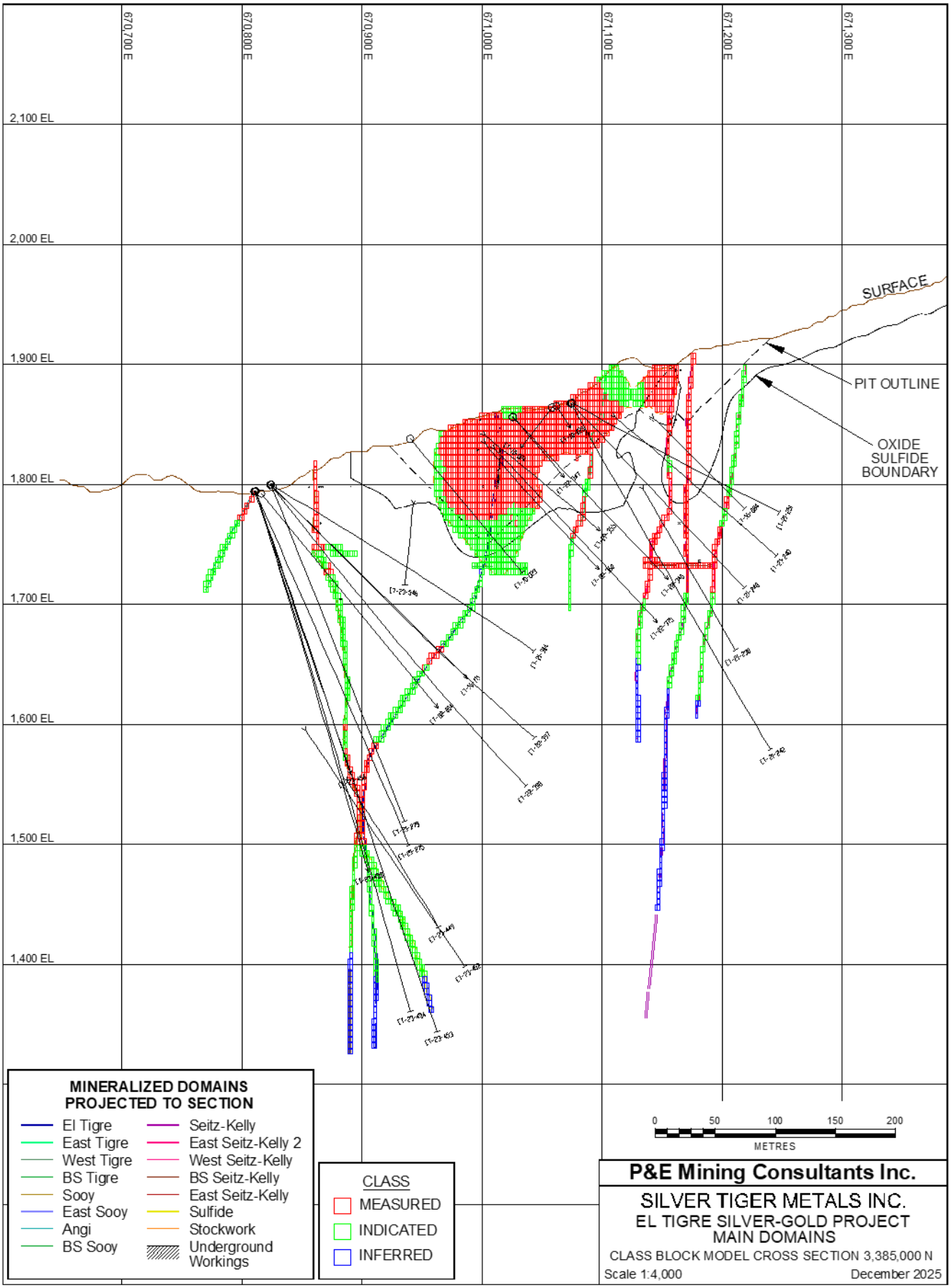
SILVER TIGER METALS INC.
EL TIGRE SILVER-GOLD PROJECT
MAIN DOMAINS
CLASS BLOCK MODEL CROSS SECTION 3,384,500 N
Scale 1:4,000
December 2025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P&E Mining Consultants Inc.
SILVER TIGER METALS INC.
EL TIGRE SILVER-GOLD PROJECT
MAIN DOMAINS
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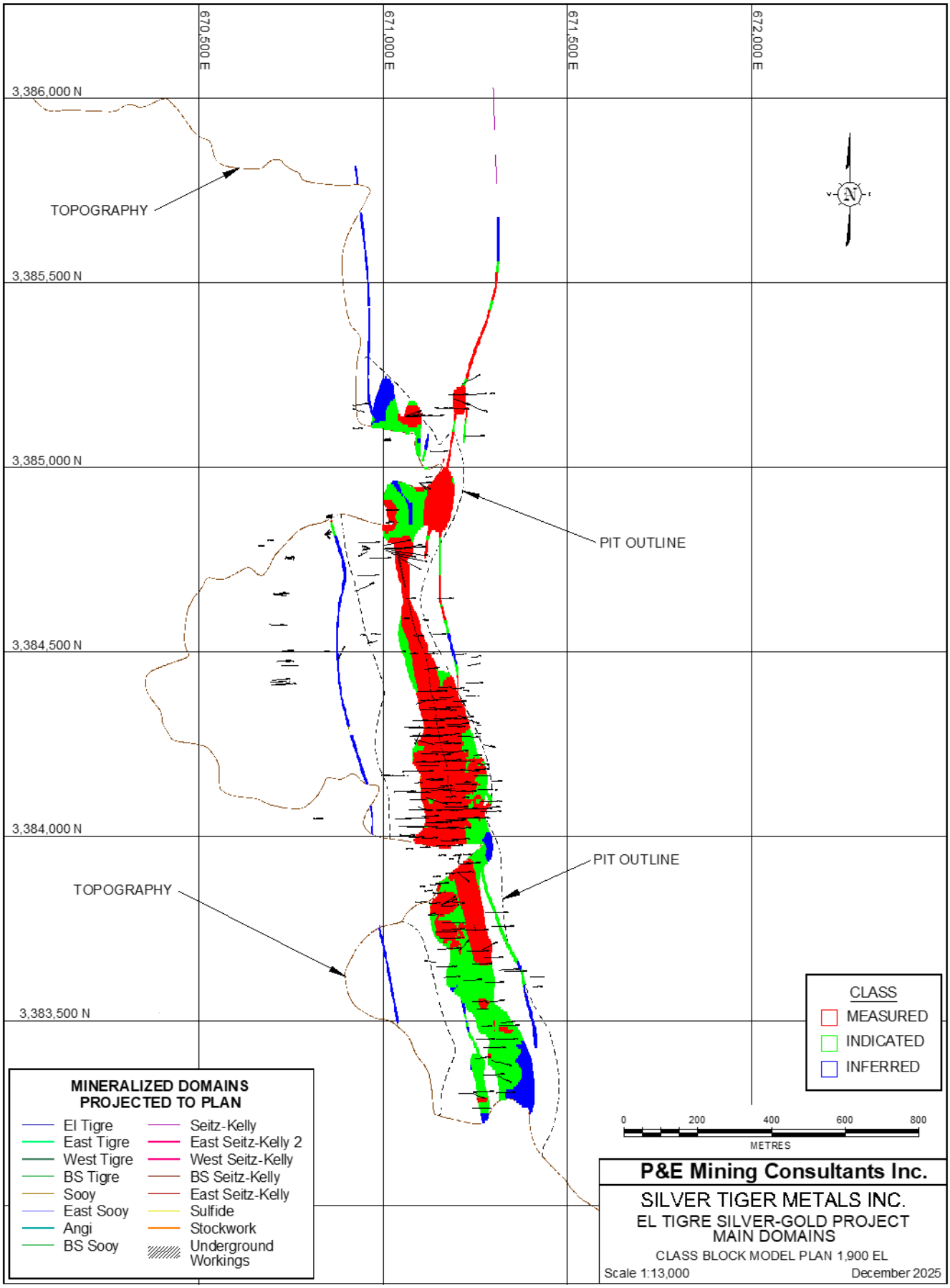
December 2025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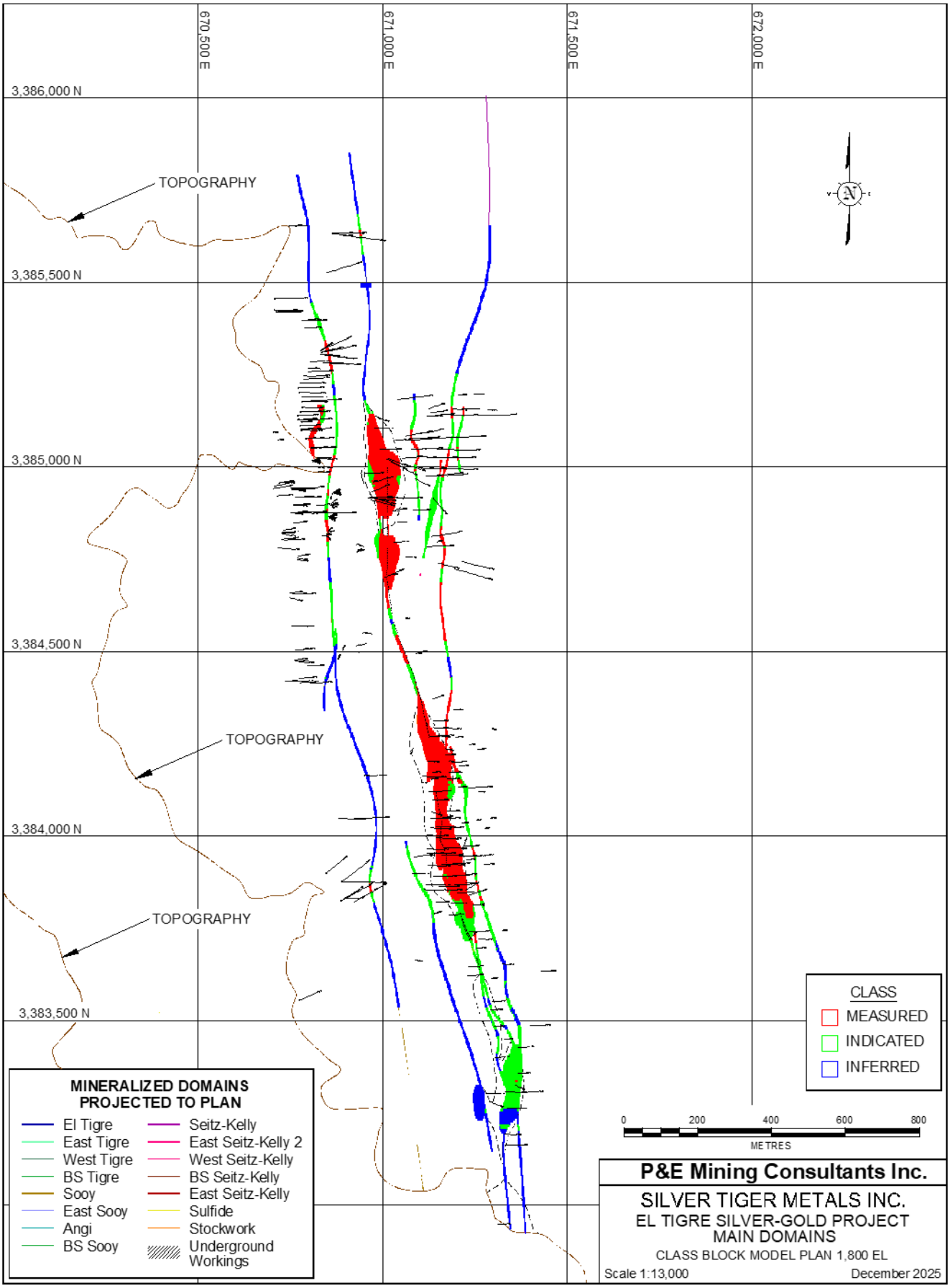
P&E Mining Consultants Inc.
SILVER TIGER METALS INC.
EL TIGRE SILVER-GOLD PROJECT
MAIN DOMAINS
CLASS BLOCK MODEL PLAN 1,900 EL
Scale 1:13,000
December 2025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Sulfide
Stockwork
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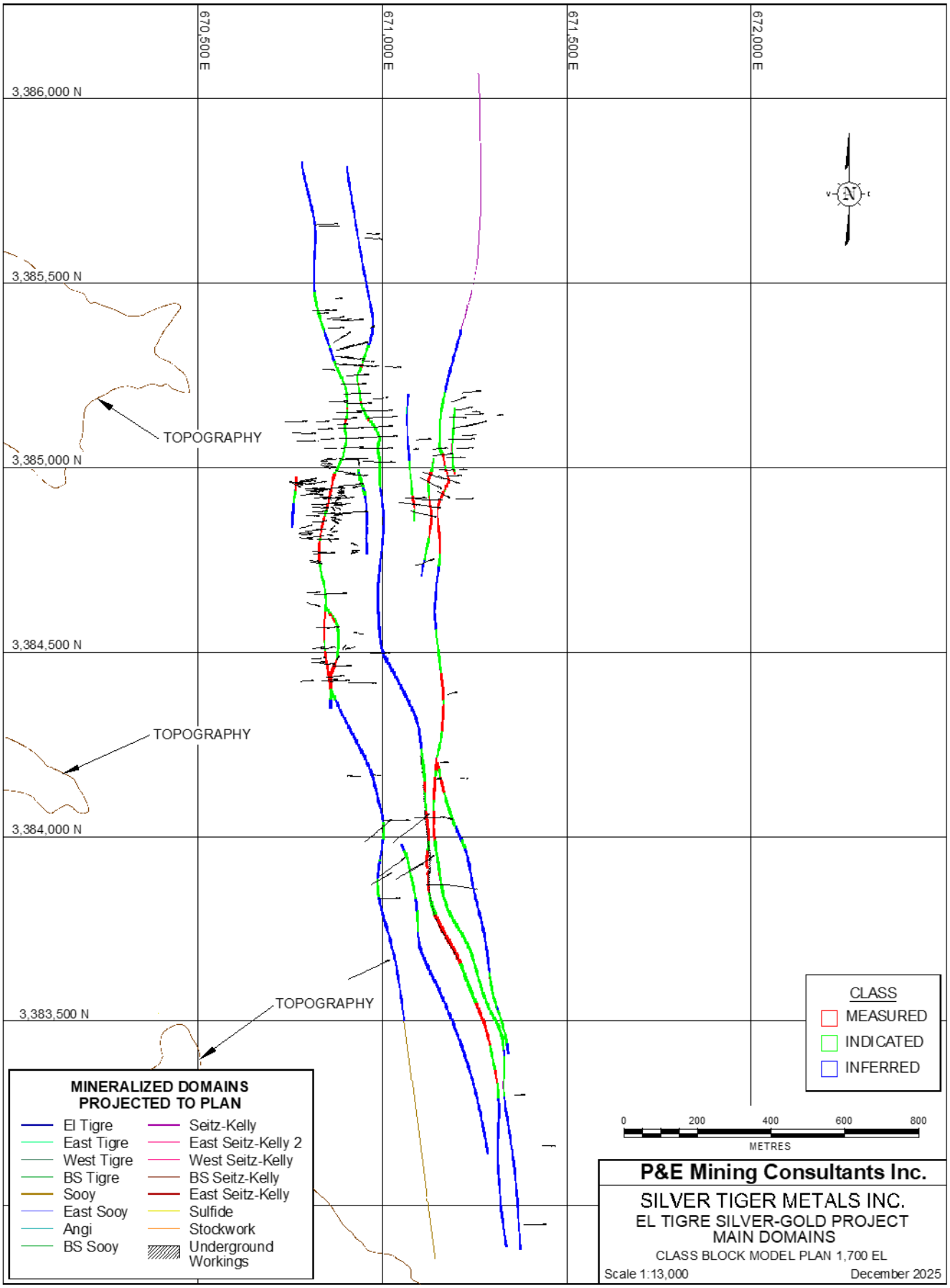
P&E Mining Consultants Inc.
SILVER TIGER METALS INC.
EL TIGRE SILVER-GOLD PROJECT
MAIN DOMAINS
CLASS BLOCK MODEL PLAN 1,800 EL
Scale 1:13,000
December 2025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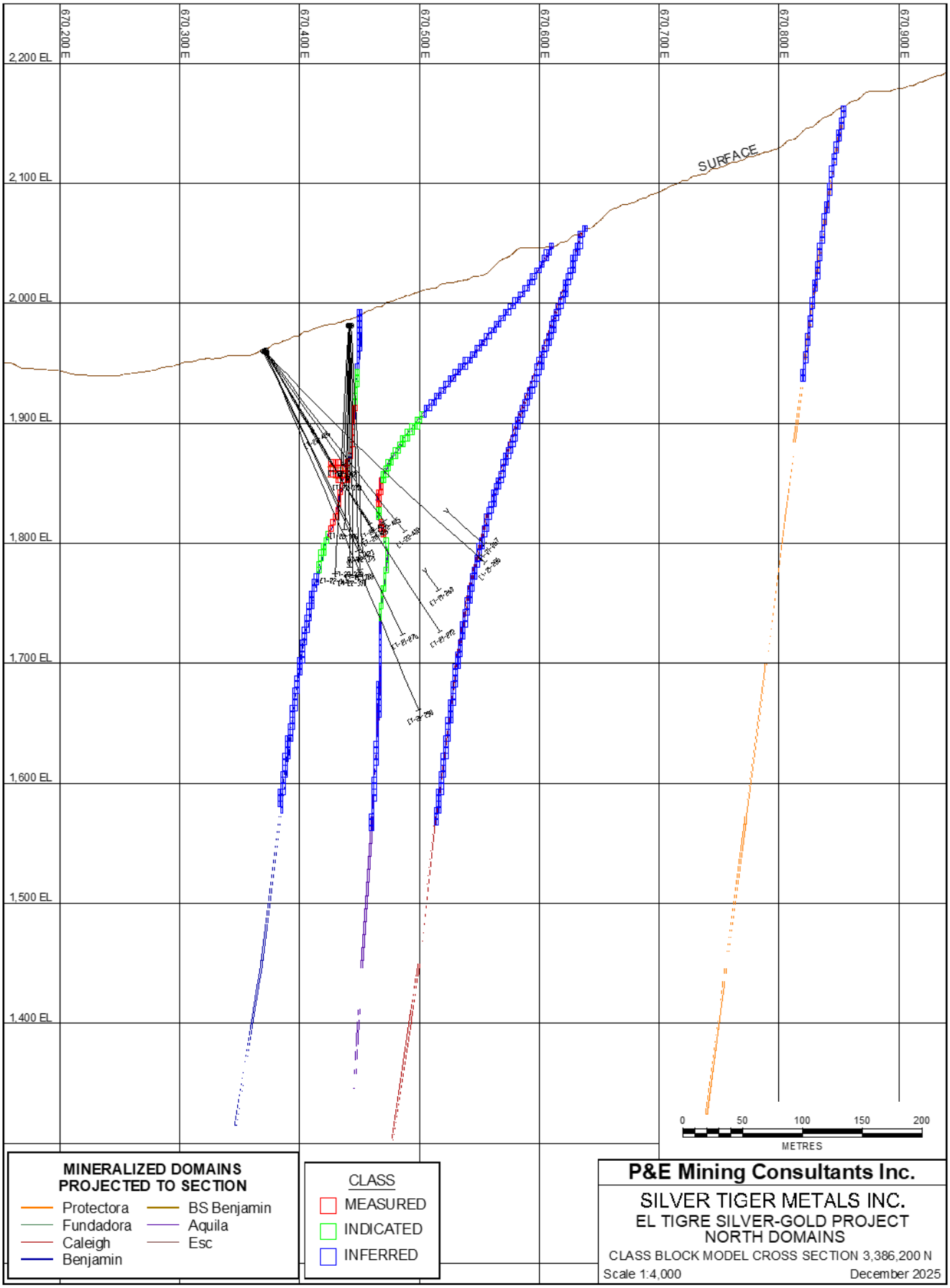
P&E Mining Consultants Inc.
SILVER TIGER METALS INC.
EL TIGRE SILVER-GOLD PROJECT
MAIN DOMAINS
CLASS BLOCK MODEL PLAN 1,700 EL
Scale 1:13,000
December 2025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Fundadora
Caleigh
Benjamin
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P&E Mining Consultants Inc.
SILVER TIGER METALS INC.
EL TIGRE SILVER-GOLD PROJECT
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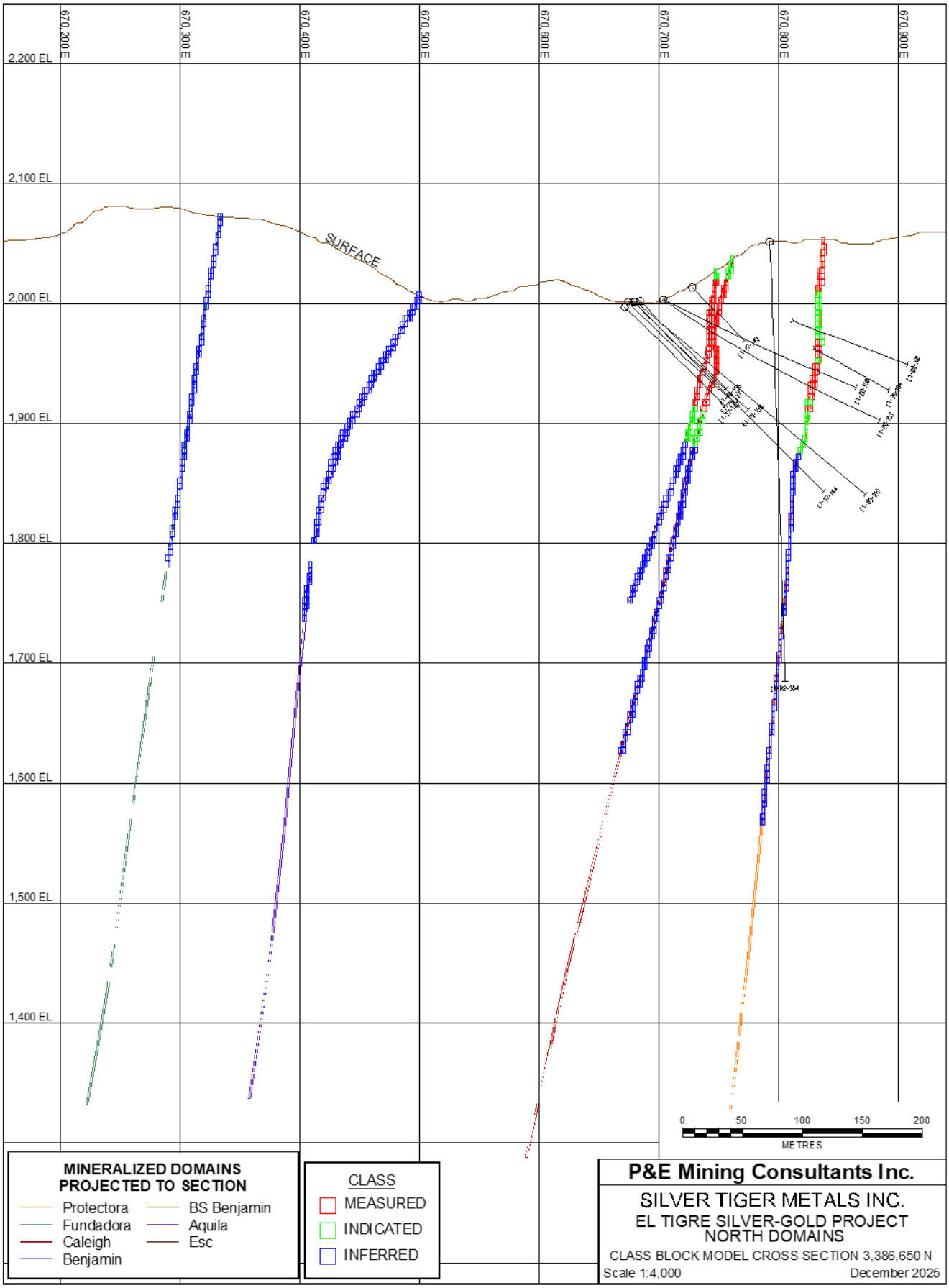
Scale 1:4,000
December 2025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EL TIGRE SILVER-GOLD PROJECT
NORTH DOMAINS
CLASS BLOCK MODEL CROSS SECTION 3,386,650 N
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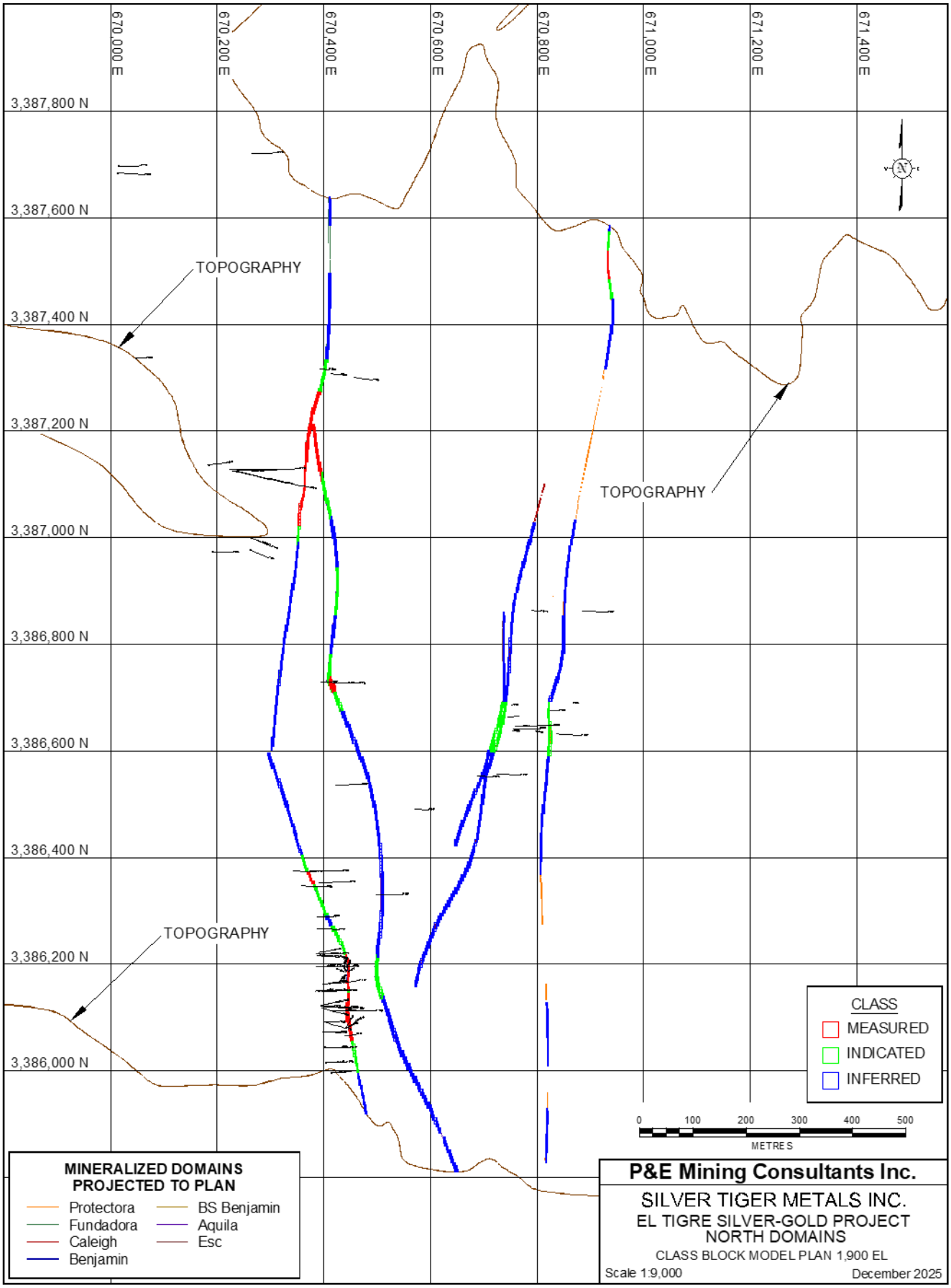
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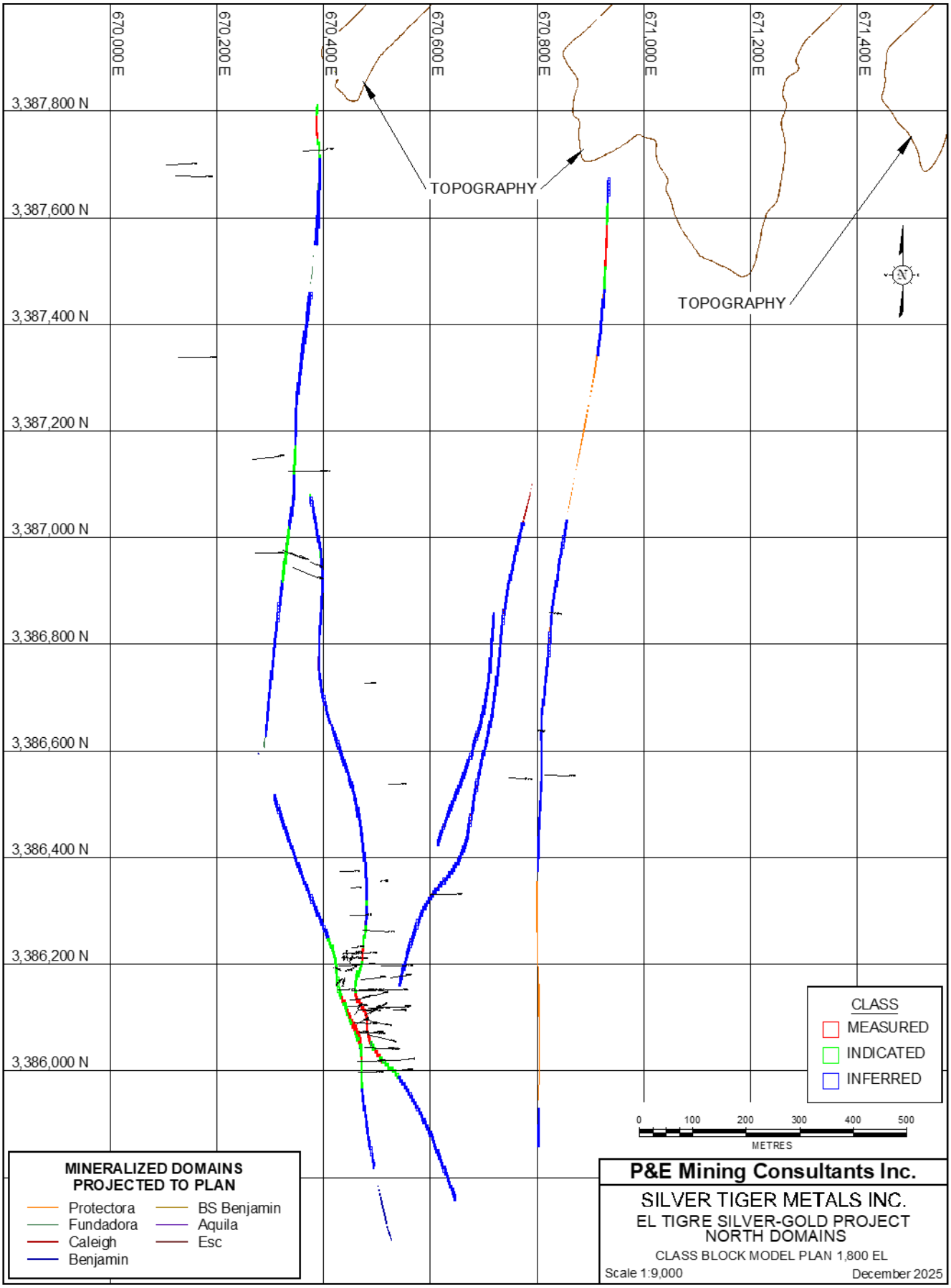
SILVER TIGER METALS INC.
EL TIGRE SILVER-GOLD PROJECT
NORTH DOMAINS
CLASS BLOCK MODEL PLAN 1,900 EL
Scale 1:9,000
December 2025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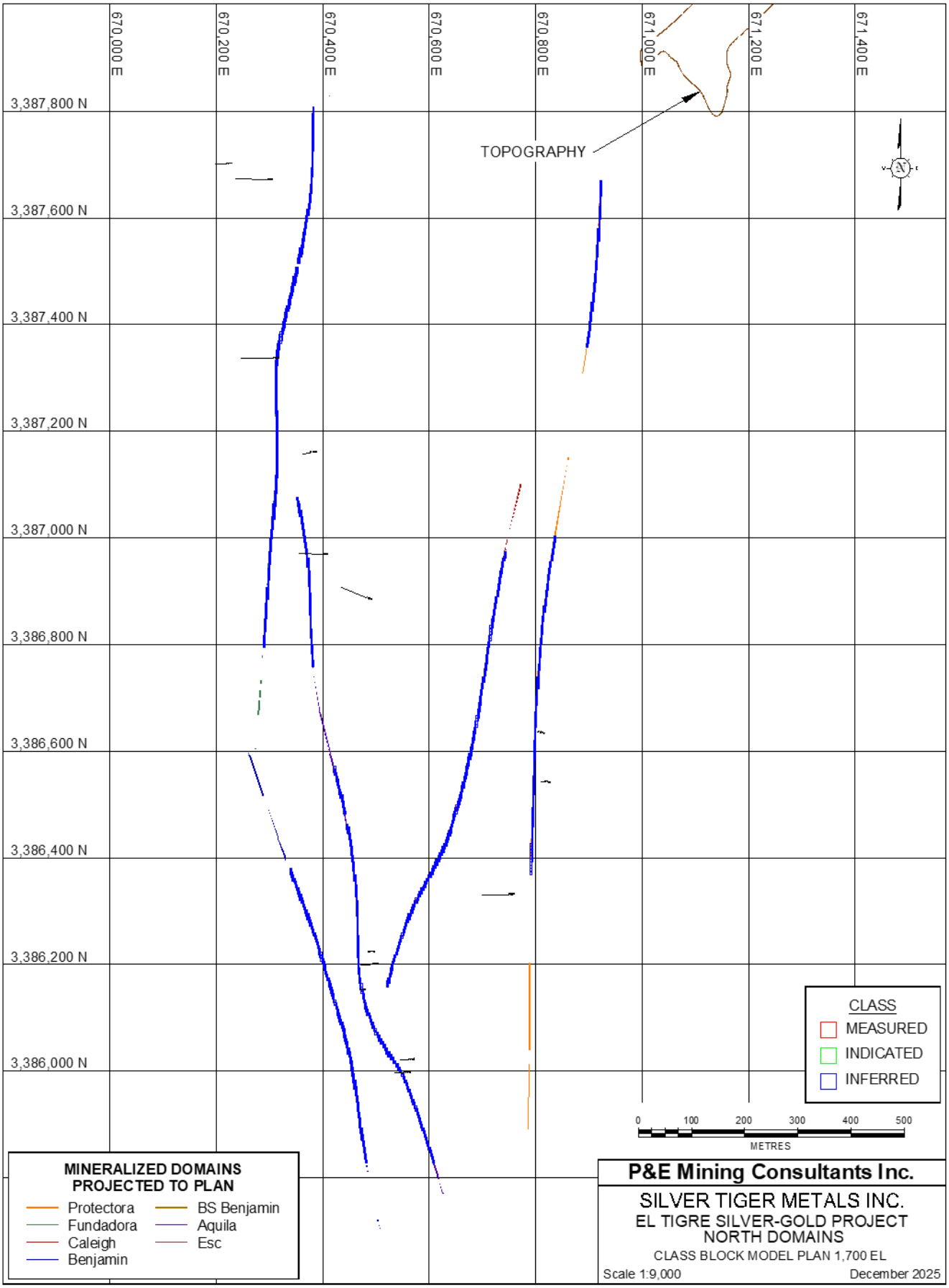
SILVER TIGER METALS INC.
EL TIGRE SILVER-GOLD PROJECT
NORTH DOMAINS
CLASS BLOCK MODEL PLAN 1,800 EL
Scale 1:9,000
December 2025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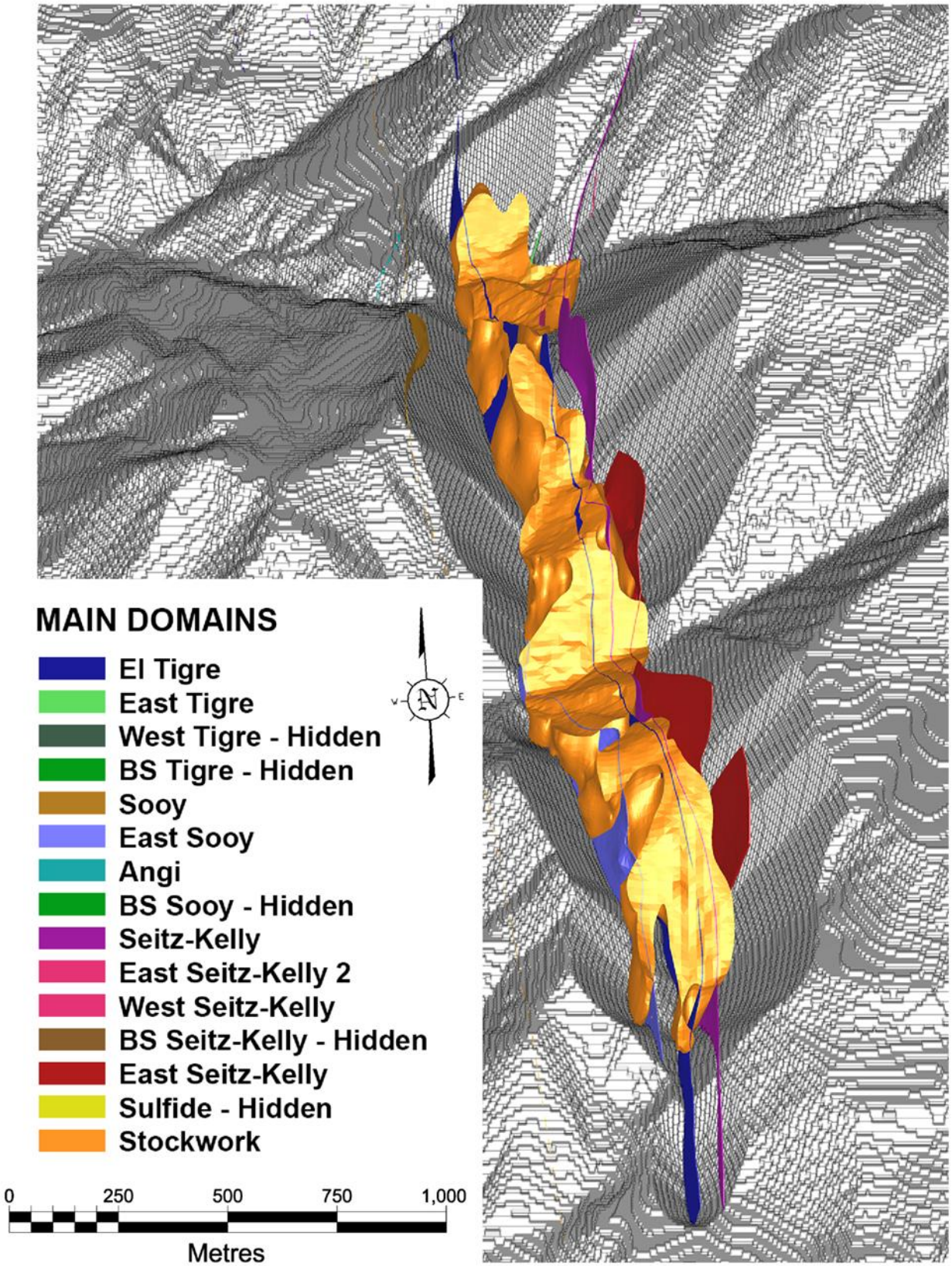
P&E Mining Consultants Inc.
SILVER TIGER METALS INC.
EL TIGRE SILVER-GOLD PROJECT
NORTH DOMAINS
CLASS BLOCK MODEL PLAN 1,700 EL
Scale 1:9,000
December 2025

APPENDIX E OPTIMIZED PIT SHELL

EL TIGRE SILVER-GOLD PROJECT
OPTIMIZED PIT SHELL



APPENDIX F SILVER TIGER DRILL COLLAR LOCATIONS 2020 TO 2024

2020 TO 2024 DRILL HOLE COLLAR LOCATIONS, ORIENTATIONS AND HOLE LENGTHS						
Drill Hole ID	**Easting* (m)**	**Northing* (m)**	**Elevation (m)**	**Hole Length (m)**	**Azimuth (°)**	**Dip (°)**
ET-20-152	670,704	3,386,640	2,000	177	90	-25
ET-20-153	670,703	3,386,640	2,000	207	90	-35
ET-20-154	670,320	3,387,316	2,055	191	90	-55
ET-20-155	670,320	3,387,316	2,055	235	95.8	-50
ET-20-156	670,684	3,386,642	2,001	102	90	-45
ET-20-157	670,674	3,386,642	2,001	114	90	-45
ET-20-158	670,679	3,386,647	1,999	131	90	-45
ET-20-159	670,787	3,386,619	2,056	116	90	-37
ET-20-160	670,219	3,387,129	1,931	217	90	-35
ET-20-161	670,787	3,386,618	2,056	149	90	-47
ET-20-162	670,220	3,387,129	1,931	233	90	-25
ET-20-163	670,828	3,386,690	2,020	61	90	-25
ET-20-164	670,826	3,386,690	2,020	114	90	-45
ET-20-165	670,825	3,386,690	2,019	104	90	-65
ET-20-166	670,219	3,387,129	1,931	241	90	-43
ET-20-167	670,786	3,386,485	2,066	70	100	-20
ET-20-168	670,786	3,386,485	2,066	70	100	-40
ET-20-169	670,220	3,387,128	1,931	207	100	-20
ET-20-170	670,864	3,386,863	2,087	56	90	-20
ET-20-171	670,863	3,386,863	2,086	70	90	-40
ET-20-172	670,219	3,387,128	1,931	217	100	-30
ET-20-173	670,844	3,386,740	2,030	66	90	-20
ET-20-174	670,843	3,386,740	2,030	72	90	-35
ET-20-175	670,340	3,386,730	2,053	250	90	-54
ET-20-176	670,266	3,386,999	1,900	134	112	-41
ET-20-177	670,265	3,386,999	1,900	209	112	-47
ET-20-178	670,843	3,385,634	1,850	198	90	-35
ET-20-179	670,266	3,386,999	1,900	156	112	-30
ET-20-180	670,843	3,385,634	1,850	195	90	-42
ET-20-181	670,745	3,386,678	2,010	174	90	-20
ET-20-182	670,339	3,386,730	2,053	220	90	-64
ET-20-183	670,842	3,385,634	1,850	201	90	-50
ET-20-184	670,744	3,386,678	2,010	169	90	-30
ET-20-185	670,711	3,385,421	1,824	168	90	-39
ET-20-186	670,743	3,386,678	2,010	142	90	-45
ET-20-187	670,711	3,385,421	1,824	174	90	-45

2020 TO 2024 DRILL HOLE COLLAR LOCATIONS, ORIENTATIONS AND HOLE LENGTHS						
Drill Hole ID	**Easting* (m)**	**Northing* (m)**	**Elevation (m)**	**Hole Length (m)**	**Azimuth (°)**	**Dip (°)**
ET-20-188	670,783	3,386,520	2,049	120	90	-25
ET-20-189	670,390	3,386,121	1,936	195	90	-39
ET-20-190	670,783	3,386,521	2,049	136	90	-35
ET-20-191	670,782	3,386,521	2,049	116	90	-52
ET-20-192	670,707	3,385,457	1,820	200	90	-30
ET-20-193	670,390	3,386,121	1,936	186	90	-50
ET-20-194	670,791	3,386,835	2,067	120	90	-25
ET-20-195	670,390	3,386,121	1,936	214	90	-60
ET-20-196	670,790	3,386,835	2,067	180	90	-35
ET-20-197	670,741	3,386,771	2,019	192	90	-25
ET-20-198	670,390	3,386,121	1,936	210	78.7	-60
ET-20-199	670,399	3,386,092	1,928	204	90	-60
ET-20-200	670,800	3,386,760	2,016	125	90	-25
ET-20-201	670,390	3,386,121	1,936	201	78.7	-50
ET-20-202	670,739	3,384,897	1,865	312	90	-60
ET-21-203	670,734	3,384,875	1,865	598	90	-60
ET-21-204	670,389	3,386,120	1,935	235	105	-49
ET-21-205	670,798	3,384,895	1,872	387	90	-65
ET-21-206	670,399	3,386,092	1,927	201	105	-49
ET-21-207	670,706	3,384,821	1,874	351	90	-60
ET-21-208	670,400	3,386,073	1,922	201	90	-55
ET-21-208A	670,400	3,386,073	1,922	125	90	-55
ET-21-209	670,739	3,384,897	1,865	361	90	-63
ET-21-210	670,688	3,384,802	1,878	390	90	-50
ET-21-211	670,734	3,384,848	1,870	351	90	-60
ET-21-212	670,739	3,384,897	1,865	351	82	-63
ET-21-213	670,688	3,384,802	1,878	314	90	-54
ET-21-214	670,890	3,383,878	1,823	378	49	-30
ET-21-215	670,735	3,384,754	1,908	252	82.2	-68
ET-21-216	670,707	3,384,656	1,918	311	94	-61
ET-21-217	670,886	3,383,825	1,822	214	61	-30
ET-21-218	670,773	3,383,555	1,805	454	66	-30
ET-21-219	670,707	3,384,656	1,918	311	90	-57
ET-21-220	670,735	3,384,753	1,908	476	100	-68
ET-21-221	670,685	3,384,574	1,934	329	93	-57
ET-21-222	670,686	3,384,574	1,934	308	93	-53
ET-21-223	670,706	3,384,499	1,942	342	85	-62
ET-21-224	670,680	3,384,418	1,962	332	88	-67
ET-21-225	670,706	3,384,499	1,942	390	85	-58
ET-21-226	670,680	3,384,418	1,962	467	88	-56

2020 TO 2024 DRILL HOLE COLLAR LOCATIONS, ORIENTATIONS AND HOLE LENGTHS						
Drill Hole ID	**Easting* (m)**	**Northing* (m)**	**Elevation (m)**	**Hole Length (m)**	**Azimuth (°)**	**Dip (°)**
ET-21-227	670,913	3,383,816	1,834	345	60	-38
ET-21-228	670,913	3,383,817	1,834	289	60	-30
ET-21-229	670,706	3,384,475	1,945	473	90	-64
ET-21-230	670,886	3,383,824	1,822	372	60	-38
ET-21-231	670,673	3,384,421	1,961	415	88	-52
ET-21-232	670,706	3,384,475	1,945	430	90	-58
ET-21-233	670,848	3,383,900	1,818	461	50	-35
ET-21-234	670,744	3,384,919	1,863	387	90	-50
ET-21-235	670,403	3,386,044	1,915	195	90	-52
ET-21-236	670,744	3,384,919	1,863	397	90	-56
ET-21-237	670,403	3,386,044	1,915	165	90	-40
ET-21-238	671,090	3,384,974	1,874	244	90	-60
ET-21-239	670,404	3,386,015	1,906	253	90	-55
ET-21-240	671,090	3,384,974	1,874	204	90	-40
ET-21-241	670,741	3,384,940	1,852	363	90	-60
ET-21-242	671,074	3,384,998	1,867	332	90	-60
ET-21-243	670,405	3,386,015	1,906	217	90	-40
ET-21-244	670,886	3,383,824	1,821	294	60	-45
ET-21-245	670,415	3,385,997	1,896	241	90	-55
ET-21-246	670,741	3,384,940	1,852	415	90	-50
ET-21-247	670,416	3,385,997	1,896	241	90	-40
ET-21-248	671,074	3,384,998	1,868	210	90	-45
ET-21-249	670,890	3,383,877	1,823	441	90	-25
ET-21-250	670,742	3,384,940	1,851	348	90	-40
ET-21-251	671,075	3,384,998	1,868	195	90	-30
ET-21-252	670,411	3,386,113	1,937	183	90	-35
ET-21-253	670,709	3,384,462	1,945	427	90	-57
ET-21-254	671058	3,385,027	1,870	204	90	-25
ET-21-255	670,378	3,386,149	1,942	235	90	-40
ET-21-256	670,667	3,384,409	1,964	448	90	-45
ET-21-257	670,813	3,384,048	1,878	400	90	-30
ET-21-258	671,057	3,385,027	1,870	230	90	-35
ET-21-259	670,377	3,386,149	1,942	210	90	-55
ET-21-260	671,057	3,385,027	1,870	372	90	-50
ET-21-261	670,377	3,386,149	1,942	238	90	-66
ET-21-262	671,047	3,385,049	1,873	314	90	-28
ET-21-263	670,666	3,384,409	1,964	427	90	-52
ET-21-264	671,047	3,385,049	1,873	262	90	-43
ET-21-265	671,047	3,385,049	1,873	297	90	-58
ET-21-266	670,812	3,384,048	1,878	433	90	-40

2020 TO 2024 DRILL HOLE COLLAR LOCATIONS, ORIENTATIONS AND HOLE LENGTHS						
Drill Hole ID	**Easting* (m)**	**Northing* (m)**	**Elevation (m)**	**Hole Length (m)**	**Azimuth (°)**	**Dip (°)**
ET-21-267	670,384	3,386,161	1,947	226	84.25	-40
ET-21-268	671,006	3,385,072	1,876	281	90	-26
ET-21-269	670,384	3,386,161	1,947	229	82.36	-54.6
ET-21-270	670,694	3,384,941	1,835	409	81	-49
ET-21-271	671,006	3,385,072	1,876	294	90	-35
ET-21-272	670,372	3,386,194	1,959	275	90	-62
ET-21-273	670,740	3,384,939	1,852	494	90	-62
ET-21-274	671,005	3,385,072	1,876	311	90	-45
ET-21-275	670,811	3,385,015	1,794	323	101	-65
ET-21-276	670,367	3,386,219	1,965	268	90	-63
ET-21-277	670,743	3,384,919	1,863	455	90	-61
ET-21-278	671,005	3,385,072	1,876	348	90	-60
ET-21-279	670,824	3,385,024	1,799	302	90	-68
ET-21-280	670,735	3,384,878	1,865	412	90	-55
ET-21-281	670,337	3,386,267	1,990	293	90	-53
ET-21-282	670,937	3,384,920	1,851	369	105	-40
ET-21-283	670,807	3,385,052	1,807	323	90	-66
ET-21-284	670,735	3,384,878	1,865	416	90	-48
ET-21-285	670,736	3,384,878	1,865	397	90	-40
ET-21-286	670,372	3,386,195	1,959	253	90	-48
ET-21-287	670,793	3,385,077	1,809	268	90	-30
ET-21-288	671,006	3,385,074	1,876	326	81	-34
ET-21-289	670,792	3,385,077	1,809	320	90	-43
ET-21-290	670,372	3,386,195	1,959	324	90	-67
ET-21-291	670,791	3,385,077	1,809	305	90	-63
ET-21-292	670,736	3,384,878	1,865	201	90	-27
ET-21-293	671,005	3,385,074	1,876	317	80	-42
ET-21-294	670,971	3,384,781	1,953	227	90	-30
ET-21-295	670,938	3,384,920	1,851	384	90	-40
ET-21-296	670,780	3,385,102	1,813	290	90	-42
ET-21-297	671,059	3,385,138	1,928	332	90	-50
ET-21-298	670,735	3,384,878	1,865	442	96.15	-67.6
ET-21-299	670,971	3,384,781	1,953	336	90	-40
ET-21-300	670,780	3,385,102	1,813	270	90	-30
ET-21-301	671,060	3,385,138	1,928	331	90	-39
ET-21-302	670,340	3,386,287	2,001	284	90	-56
ET-21-303	670,938	3,384,920	1,851	332	96	-40
ET-21-304	670,825	3,385,024	1,799	259	90	-30
ET-21-305	670,779	3,385,102	1,813	330	90	-61
ET-22-306	670,834	3,385,283	1,835	241	95	-32

2020 TO 2024 DRILL HOLE COLLAR LOCATIONS, ORIENTATIONS AND HOLE LENGTHS						
Drill Hole ID	**Easting* (m)**	**Northing* (m)**	**Elevation (m)**	**Hole Length (m)**	**Azimuth (°)**	**Dip (°)**
ET-22-307	670,843	3,385,634	1,848	171	96	-25
ET-22-308	671,060	3,385,138	1,928	372	90	-60
ET-22-309	670,972	3,384,781	1,953	363	97	-30
ET-22-310	670,842	3,385,634	1,848	217	96	-51
ET-22-311	670,940	3,384,920	1,851	339	90	-25
ET-22-312	670,833	3,385,283	1,835	198	95	-42
ET-22-313	671,061	3,385,138	1,928	323	90	-30
ET-22-314	670,807	3,385,514	1,849	233	72	-31
ET-22-315	670,833	3,385,283	1,835	241	95	-51
ET-22-316	670,971	3,384,780	1,953	380	105	-30
ET-22-317	670,833	3,385,283	1,835	241	96	-58
ET-22-318	670,272	3,386,351	2,029	314	90	-40
ET-22-319	670,779	3,385,100	1,812	61	90	-42
ET-22-320	670,776	3,385,113	1,813	156	90	-42
ET-22-321	670,833	3,385,283	1,835	262	96	-64
ET-22-322	671,059	3,385,138	1,928	332	84	-57
ET-22-323	670,939	3,384,920	1,851	360	90	-45
ET-22-324	670,971	3,384,782	1,953	336	83	-40
ET-22-325	670,770	3,385,129	1,813	70	90	-42
ET-22-326	670,833	3,385,283	1,835	371	97	-69
ET-22-327	670,769	3,385,140	1,814	73	90	-40
ET-22-328	670,777	3,385,150	1,815	92	90	-40
ET-22-329	670,442	3,386,186	1,981	201	0	-90
ET-22-330	671,059	3,385,139	1,928	317	72	-56
ET-22-331	670,782	3,385,162	1,817	93	90	-40
ET-22-332	670,828	3,385,306	1,836	174	96	-36
ET-22-333	670,939	3,384,919	1,851	363	96	-49
ET-22-334	670,781	3,385,178	1,823	268	90	-40
ET-22-335	670,919	3,385,167	1,894	387	84	-28
ET-22-336	670,271	3,386,351	2,029	284	90	-50
ET-22-337	670,827	3,385,306	1,836	189	97	-46
ET-22-338	670,827	3,385,306	1,836	192	97	-55
ET-22-339	671,129	3,385,162	1,948	217	90	-39
ET-22-340	671,026	3,385,019	1,856	265	109	-45
ET-22-341	670,827	3,385,306	1,836	210	98	-62
ET-22-342	671,129	3,385,162	1,948	232	90	-54
ET-22-343	670,781	3,385,190	1,825	107	90	-40
ET-22-344	670,827	3,385,306	1,836	265	98	-67
ET-22-345	670,773	3,385,202	1,825	237	90	-40
ET-22-346	671,129	3,385,162	1,948	271	90	-64

2020 TO 2024 DRILL HOLE COLLAR LOCATIONS, ORIENTATIONS AND HOLE LENGTHS						
Drill Hole ID	**Easting* (m)**	**Northing* (m)**	**Elevation (m)**	**Hole Length (m)**	**Azimuth (°)**	**Dip (°)**
ET-22-347	671,026	3,385,021	1,856	290	86	-50
ET-22-348	670,253	3,386,375	2,029	299	90	-40
ET-22-349	670,971	3,384,780	1,953	281	98	-37
ET-22-350	670,771	3,385,215	1,825	107	90	-40
ET-22-351	671,166	3,385,197	1,971	134	90	-20
ET-22-352	670,830	3,385,310	1,836	192	81	-35
ET-22-353	670,253	3,386,375	2,029	310	90	-48
ET-22-354	670,792	3,386,638	2,057	372	90	-87
ET-22-355	670,988	3,385,033	1,854	189	121	-36
ET-22-356	670,775	3,385,227	1,826	236	90	-40
ET-22-357	671,166	3,385,197	1,970	136	90	-40
ET-22-358	670,829	3,385,310	1,836	181	80	-47
ET-22-359	671,165	3,385,197	1,970	203	90	-62
ET-22-360	670,987	3,385,033	1,854	372	121	-48
ET-22-361	670,829	3,385,310	1,836	197	80	-56
ET-22-362	671,165	3,385,197	1,970	226	90	-74
ET-22-363	670,829	3,385,310	1,836	217	79	-63
ET-22-364	670,441	3,386,185	1,981	207	270	-87
ET-22-365	670,628	3,386,557	2,029	366	90	-60
ET-22-366	670,785	3,385,238	1,826	92	90	-40
ET-22-367	670,829	3,385,310	1,836	241	78	-68
ET-22-368	671,165	3,385,197	1,970	153	110	-40
ET-22-369	670,785	3,385,247	1,826	220	90	-40
ET-22-370	671,165	3,385,197	1,970	166	110	-57
ET-22-371	670,443	3,386,185	1,981	189	90	-87
ET-22-372	670,827	3,385,313	1,836	192	66	-35
ET-22-373	670,741	3,386,868	2,069	293	90	-70
ET-22-374	670,796	3,385,259	1,826	93	90	-40
ET-22-375	670,987	3,385,034	1,854	320	110	-45
ET-22-376	671,166	3,385,197	1,970	79	71	-20
ET-22-377	670,826	3,385,313	1,836	241	58	-66
ET-22-378	670,441	3,386,184	1,981	221	182	-85
ET-22-379	670,798	3,385,268	1,826	403	90	-67
ET-22-380	671,164	3,385,197	1,970	128	61	-40
ET-22-381	670,826	3,385,313	1,836	235	60	-61
ET-22-382	670,441	3,386,183	1,981	207	205	-85
ET-22-383	670,782	3,385,378	1,835	268	90	-57
ET-22-384	670,801	3,385,286	1,826	101	90	-52
ET-22-385	670,857	3,384,899	1,869	328	90	-85
ET-22-386	670,441	3,386,186	1,981	171	348	-83

2020 TO 2024 DRILL HOLE COLLAR LOCATIONS, ORIENTATIONS AND HOLE LENGTHS						
Drill Hole ID	**Easting* (m)**	**Northing* (m)**	**Elevation (m)**	**Hole Length (m)**	**Azimuth (°)**	**Dip (°)**
ET-22-387	670,782	3,385,178	1,823	247	90	-30
ET-22-388	670,442	3,386,186	1,981	204	24	-83
ET-22-389	670,781	3,385,178	1,823	332	90	-50
ET-22-390	670,442	3,386,186	1,981	210	5	-83
ET-22-391	670,857	3,384,899	1,868	307	58	-83
ET-22-392	670,806	3,385,053	1,806	339	90	-44
ET-22-393	670,863	3,384,930	1,852	299	105	-81
ET-22-394	670,441	3,386,080	1,932	194	31	-74
ET-22-395	670,857	3,384,899	1,869	458	115	-83
ET-22-396	670,442	3,386,080	1,932	207	52	-68
ET-22-397	670,824	3,385,025	1,799	305	90	-45
ET-22-398	670,824	3,385,025	1,799	329	90	-52
ET-22-399	670,443	3,386,080	1,932	186	61	-64
ET-22-400	670,857	3,384,899	1,869	293	136	-86
ET-22-401	670,439	3,386,065	1,927	156	82	-67
ET-22-402	670,770	3,385,128	1,813	259	90	-42
ET-22-403	670,442	3,386,079	1,932	203	63	-76
ET-22-404	670,862	3,384,928	1,852	252	148	-86
ET-22-405	670,772	3,385,128	1,813	284	90	-27
ET-22-406	670,411	3,386,112	1,937	188	90	-62
ET-22-407	670,849	3,384,865	1,886	284	73	-82
ET-22-408	670,411	3,386,112	1,937	160	90	-59
ET-22-409	670,807	3,385,053	1,807	275	90	-30
ET-22-410	670,378	3,386,150	1,941	183	90	-52
ET-22-411	670,849	3,384,867	1,886	329	92	-83
ET-22-412	670,778	3,385,150	1,815	290	90	-29
ET-22-413	670,378	3,386,150	1,941	183	90	-47
ET-22-414	670,850	3,384,866	1,886	299	112	-82
ET-22-415	670,777	3,385,150	1,815	302	90	-43
ET-22-416	670,369	3,386,218	1,965	180	90	-59
ET-22-417	670,850	3,384,866	1,886	459	120	-84
ET-22-418	670,369	3,386,218	1,965	195	90	-54
ET-22-419	670,773	3,385,396	1,835	253	90	-49
ET-22-420	670,369	3,386,218	1,965	174	94	-60
ET-22-421	670,772	3,385,396	1,835	284	90	-56
ET-22-422	670,848	3,384,865	1,886	503	93	-85
ET-22-423	670,369	3,386,218	1,965	189	94	-64
ET-22-424	670,773	3,385,396	1,836	265	90	-43
ET-22-425	670,369	3,386,218	1,965	178	93	-55
ET-22-426	670,369	3,386,218	1,965	183	80	-59

2020 TO 2024 DRILL HOLE COLLAR LOCATIONS, ORIENTATIONS AND HOLE LENGTHS						
Drill Hole ID	**Easting* (m)**	**Northing* (m)**	**Elevation (m)**	**Hole Length (m)**	**Azimuth (°)**	**Dip (°)**
ET-22-427	670,843	3,384,819	1,917	482	44	-84
ET-22-428	670,845	3,384,819	1,917	461	61	-85
ET-22-429	670,845	3,384,818	1,917	461	89	-83
ET-22-430	670,845	3,384,817	1,917	464	124	-86
ET-22-431	670,850	3,384,868	1,886	473	79	-83
ET-22-432	670,863	3,384,930	1,852	509	100	-82
ET-22-433	670,864	3,384,931	1,852	497	90	-82
ET-22-434	670,863	3,384,930	1,852	470	80	-82
ET-22-435	670,863	3,384,930	1,852	525	70	-81
ET-22-436	670,856	3,384,898	1,869	494	79	-83
ET-22-437	670,857	3,384,898	1,869	485	89	-83
ET-22-438	670,848	3,384,866	1,886	433	107	-83
ET-22-439	670,703	3,384,921	1,843	201	90	-59
ET-22-440	670,741	3,384,942	1,852	555	94	-68
ET-22-441	670,741	3,384,942	1,852	523	95	-64
ET-22-442	670,741	3,384,942	1,852	564	100	-64
ET-22-443	670,741	3,384,943	1,851	555	90	-62
ET-22-444	670,741	3,384,943	1,852	562	90	-68
ET-23-445	670,715	3,384,958	1,834	503	93	-60
ET-23-446	670,715	3,384,959	1,834	509	87	-60
ET-23-447	670,780	3,385,123	1,813	335	90	-65
ET-23-448	670,703	3,385,094	1,775	317	90	-45
ET-23-449	670,711	3,384,956	1,834	477	83	-59
ET-23-450	670,718	3,385,073	1,776	313	90	-45
ET-23-451	670,691	3,385,123	1,772	355	90	-45
ET-23-452	670,700	3,384,947	1,834	524	81	-58
ET-23-453	670,811	3,385,017	1,794	476	99	-71
ET-23-454	670,812	3,385,017	1,794	452	91	-72
ET-23-455	670,704	3,384,823	1,874	530	89	-65
ET-23-456	670,810	3,385,018	1,794	455	84	-71
ET-23-457	670,704	3,384,823	1,874	536	93	-65
ET-23-458	670,811	3,385,016	1,794	476	112	-72
ET-23-459	670,704	3,384,823	1,874	524	99	-65
ET-23-460	670,703	3,384,473	1,946	644	87	-65.5
ET-23-461	670,706	3,384,824	1,874	489	80	-64
ET-23-462	670,888	3,384,500	2,039	545	190	-87.5
ET-23-463	670,968	3,384,780	1,954	584	285	-80.5
ET-23-464	670,968	3,384,780	1,954	599	260	-81.5
ET-23-465	670,888	3,384,501	2,039	566	300	-89
ET-23-466	671,035	3,384,544	2,097	554	253	-73

2020 TO 2024 DRILL HOLE COLLAR LOCATIONS, ORIENTATIONS AND HOLE LENGTHS						
Drill Hole ID	**Easting* (m)**	**Northing* (m)**	**Elevation (m)**	**Hole Length (m)**	**Azimuth (°)**	**Dip (°)**
ET-23-467	670,966	3,384,782	1,954	54	274	-80
ET-23-468	670,968	3,384,780	1,954	612	254	-80
ET-23-469	671,035	3,384,545	2,097	680	272	-76
ET-23-470	671,026	3,384,164	1,912	384	270	-65
ET-23-471	671,025	3,384,164	1,912	381	270	-80
ET-23-472	670,704	3,384,657	1,917	474	105	-64
ET-23-473	670,704	3,384,658	1,917	581	90	-66
ET-23-474	670,737	3,384,757	1,908	572	100	-71
ET-23-475	671,036	3,384,546	2,098	176	90	-60
ET-23-476	671,086	3,384,318	1,978	175.1	90	-45
ET-23-477	671,112	3,384,071	1,958	175.95	90	-45
ET-23-478	671,136	3,383,846	1,894	150.7	90	-45
ET-23-479	671,083	3,384,128	1,944	190	90	-45
ET-23-480	671,153	3,383,765	1,932	122.3	90	-45
ET-23-481	671,189	3,383,605	2,033	202	90	-50
ET-23-482	671,116	3,384,092	1,971	120	90	-45
ET-23-483	671,146	3,384,143	1983	170	90	-50
ET-23-484	671,112	3,384,071	1,958	180	90	-45
ET-23-485	671,055	3,384,176	1,921	141	90	-51
ET-23-486	671,046	3,384,071	1,951	212	90	-45
ET-23-487	671,146	3,383,928	1,868	111	90	-45
ET-23-488	671,095	3,384,223	1,940	165	90	-57
ET-23-489	671,043	3,384,049	1,943	220	90	-45
ET-23-490	671,122	3,383,922	1,862	123	90	-45
ET-23-491	671,089	3,384,191	1,933	180	90	-45
ET-23-492	671,181	3,384,051	1,949	170	90	-47
ET-23-493	671,206	3,383,925	1,894	120	90	-45
ET-23-494	671,137	3,384,022	1,932	122	90	-50
ET-23-495	671,152	3,383,948	1,882	130.5	90	-46
ET-23-496	671,094	3,384,274	1,966	160	90	-63
ET-23-497	671,091	3,384,026	1,927	182	90	-50
ET-23-498	671,118	3,384,124	1,969	210	90	-45
ET-23-499	671,108	3,384,295	1,968	144	90	-66
ET-23-500	671,066	3,383,966	1,879	146	90	-45
ET-23-501	671,160	3,384,322	1,982	105	90	-45
ET-23-502	671,177	3,383,974	1,901	90	90	-45
ET-23-503	671,147	3,384,180	1,973	147	90	-45
ET-23-504	671,084	3,384,202	1,932	190	90	-45
ET-23-505	671,190	3,383,991	1,909	90	90	-45
ET-23-506	671,147	3,384,221	1,967	150	90	-75

2020 TO 2024 DRILL HOLE COLLAR LOCATIONS, ORIENTATIONS AND HOLE LENGTHS						
Drill Hole ID	**Easting* (m)**	**Northing* (m)**	**Elevation (m)**	**Hole Length (m)**	**Azimuth (°)**	**Dip (°)**
ET-23-507	671,138	3,383,890	1,882	120	90	-45
ET-23-508	671,185	3,384,195	1,993	127	90	-45
ET-23-509	671,128	3,384,078	1,965	185.5	90	-45
ET-23-510	671,088	3,384,345	1,990	174.1	90	-49
ET-23-511	671,177	3,384,164	1,995	158	90	-45
ET-23-512	671,151	3,384,070	1,964	186	90	-45
ET-23-513	671,136	3,384,353	1,993	242.67	110	-45
ET-23-514	671,058	3,384,099	1,946	221	90	-56
ET-23-515	671,035	3,384,344	1,995	257	90	-46
ET-23-516	671,129	3,383,868	1,890	150	90	-45
ET-23-517	671,095	3,384,400	2,022	166	90	-45
ET-23-518	671,101	3,383,863	1,876	150	90	-45
ET-23-519	671,142	3,384,392	2,012	116.55	90	-45
ET-23-520	671,043	3,384,369	2,007	166	90	-45
ET-23-521	671,179	3,384,184	1,992	135.55	90	-45
ET-23-522	671,196	3,384,569	2,108	140.99	80	-60
ET-23-523	671,192	3,383,947	1,880	102	90	-45
ET-23-524	671,122	3,384,365	1,993	130.1	90	-45
ET-23-525	671,231	3,384,066	1,959	109.3	90	-45
ET-23-526	671,042	3,384,369	2,007	193	90	-55
ET-23-527	671,182	3,384,278	1,992	99.95	90	-45
ET-23-528	671,093	3,383,920	1,855	140	90	-45
ET-23-529	671,041	3,384,394	2,020	172	90	-45
ET-23-530	671,182	3,384,219	1,990	106.25	90	-45
ET-23-531	671,039	3,384,546	2,098	250.73	90	-60
ET-23-532	671,190	3,384,253	1,994	90.45	90	-44
ET-23-533	671,071	3,383,868	1,868	152	90	-45
ET-23-534	671,190	3,384,301	1,989	96.6	90	-45
ET-23-535	671,187	3,384,322	1,988	90.45	90	-45
ET-23-536	671,045	3,384,272	1,979	228	90	-45
ET-23-537	671,235	3,383,485	2,014	291	90	-70
ET-23-538	671,198	3,384,344	2,004	90.45	90	-45
ET-23-539	671,167	3,384,024	1,933	154.5	90	-45
ET-23-540	671,044	3,384,272	1,979	241	90	-53.6
ET-23-541	671,058	3,384,026	1,926	215.85	90	-49.7
ET-23-542	671,038	3,384,546	2,098	200.02	270	-65
ET-23-543	671,044	3,384,272	1,978	261	90	-60.8
ET-23-544	671,107	3,384,298	1,969	170.35	200	-65
ET-23-545	671,211	3,383,667	2,001	150	65	-50
ET-23-546	670,949	3,384,943	1,853	151.85	350	-65

2020 TO 2024 DRILL HOLE COLLAR LOCATIONS, ORIENTATIONS AND HOLE LENGTHS						
Drill Hole ID	**Easting* (m)**	**Northing* (m)**	**Elevation (m)**	**Hole Length (m)**	**Azimuth (°)**	**Dip (°)**
ET-23-547	671,048	3,384,315	1,984	221	90	-45
ET-23-548	671,197	3,384,073	1,963	150	90	-45
ET-23-549	670,967	3,384,779	1,954	80.77	240	-70
ET-23-550	671,180	3,384,126	2,003	166.5	90	-45
ET-23-551	671,172	3,383,657	2,007	216.1	275	-70
ET-23-552	671,058	3,384,294	1,977	191	90	-45
ET-23-553	671,021	3,384,047	1,938	220	90	-45
ET-23-554	671,014	3,384,294	1,985	276	90	-45
ET-23-555	671,174	3,383,656	2,007	218.2	45	-60
ET-23-556	671,230	3,384,256	2,024	160.26	100	-65
ET-23-557	671,047	3,384,494	2,070	230.8	90	-65
ET-23-558	671,031	3,384,121	1,921	222	90	-45
ET-23-559	671,205	3,383,932	1,893	170.28	20	-65
ET-23-560	671,047	3,384,494	2,070	261	90	-80
ET-23-561	671,316	3,383,618	2,075	230.7	90	-60
ET-23-562	671,297	3,383,359	1,955	179.19	120	-60
ET-23-563	671,202	3,383,903	1,904	90	90	-45
ET-24-564	671,213	3,384,050	1,945	117	90	-47
ET-24-565	671,223	3,383,426	1,977	253.5	90	-45
ET-24-566	671,211	3,384,094	1,979	140	90	-45
ET-24-567	671,094	3,384,273	1,966	150	90	-48
ET-24-568	671,047	3,384,315	1,983	258	90	-60
ET-24-569	671,308	3,383,305	1,918	175.45	90	-50
ET-24-570	671,045	3,384,587	2,099	200.4	90	-70
ET-24-571	671,152	3,384,271	1,972	136.9	90	-45
ET-24-572	671,226	3,383,531	2,008	275	90	-70
ET-24-573	671,218	3,383,950	1,888	102	90	-45
ET-24-574	671,021	3,384,164	1,912	279	90	-45
ET-24-575	671,276	3,383,592	2,053	201	90	-45
ET-24-576	671,009	3,384,395	2,009	218	90	-45
ET-24-577	671,100	3,384,504	2,080	153	90	-45
ET-24-578	671,281	3,383,653	2,048	150	90	-45
ET-24-579	671,013	3,384,296	1,985	250	90	-53
ET-24-580	671,089	3,383,821	1,894	126	90	-50
ET-24-581	671,043	3,384,275	1,979	221	105	-50
ET-24-582	671,230	3,383,928	1901	78	90	-47
ET-24-583	671,145	3,384,203	1,966	150	90	-45
ET-24-584	671,084	3,384,145	1,944	150	90	-80
ET-24-585	671,253	3,384,077	1,968	75	90	-45
ET-24-586	671,203	3,383,902	1,904	103	130	-45

2020 TO 2024 DRILL HOLE COLLAR LOCATIONS, ORIENTATIONS AND HOLE LENGTHS						
Drill Hole ID	**Easting* (m)**	**Northing* (m)**	**Elevation (m)**	**Hole Length (m)**	**Azimuth (°)**	**Dip (°)**
ET-24-587	671,100	3,383,646	1,982	249	90	-50
ET-24-588	671,176	3,383,654	2,007	264	90	-50
ET-24-589	671,043	3,384,275	1,979	56.25	110	-60
ET-24-590	671,038	3,384,544	2,098	251.5	90	-75
ET-24-591	671,232	3,384,202	2,025	105	90	-45
ET-24-592	671,056	3,384,120	1,936	143.25	90	-45
ET-24-593	671,205	3,384,149	2,017	150	90	-45
ET-24-594	671,044	3,384,589	2,099	281.5	90	-85
ET-24-595	671,217	3,384,194	2,016	126	90	-45
ET-24-596	671,308	3,383,192	1,846	76.9	90	-45
ET-24-597	671,010	3,384,850	1,892	46	90	-80
ET-24-598	671,012	3,384,850	1,892	74	90	-45
ET-24-599	671,212	3,384,302	2,004	102	90	-45
ET-24-600	671,011	3,384,850	1,892	137	90	-65
ET-24-601	671,231	3,383,690	2,000	103	90	-45
ET-24-602	671,166	3,383,689	1,989	181.5	90	-45
ET-24-603	671,237	3,383,262	1,932	206.15	90	-45
ET-24-604	670,961	3,385,048	1,857	183	90	-45
ET-24-605	671,025	3,384,644	2,077	245.2	90	-50
ET-24-606	671,024	3,384,644	2,077	222	90	-75
ET-24-607	671,192	3,383,747	1,956	188.4	90	-45
ET-24-608	671,225	3,384,151	2,018	111	90	-45
ET-24-609	671,085	3,384,202	1,932	248.4	90	-70
ET-24-610	671,209	3,385,147	1,976	50.4	90	-45
ET-24-611	671,194	3,385,084	1,954	101.4	90	-45
ET-24-612	671,129	3,385,044	1,922	50.4	90	-45
ET-24-613	670,914	3,384,753	1,976	236.3	90	-45
ET-24-614	671,249	3,383,756	1,971	143.2	90	-45
ET-24-615	671,245	3,383,793	1,958	154.4	90	-45
ET-24-616	670,920	3,385,159	1,894	131.4	110	-45
ET-24-617	671,010	3,384,884	1,896	152.4	90	-45
ET-24-618	671,009	3,384,884	1,896	31.5	90	-80
ET-24-619	671,010	3,384,884	1,896	170.4	90	-70
ET-24-620	671,100	3,384,958	1,886	121.4	90	-30
ET-24-621	671,104	3,384,955	1,886	95.2	120	-30
ET-24-622	671,103	3,384,955	1,886	151.4	140	-50
ET-24-623	671,011	3,384,773	1,960	200	80	-45
ET-24-624	671,016	3,384,770	1,960	125.2	120	-45
ET-24-625	671,274	3,383,708	2,008	84	90	-45
ET-24-626	670,749	3,384,961	1,705	104	-54	330

2020 TO 2024 DRILL HOLE COLLAR LOCATIONS, ORIENTATIONS AND HOLE LENGTHS						
Drill Hole ID	**Easting* (m)**	**Northing* (m)**	**Elevation (m)**	**Hole Length (m)**	**Azimuth (°)**	**Dip (°)**
ET-24-627	670,749	3,384,962	1,705	98	-53	350
ET-24-628	670,748	3,384,961	1,705	118	-53	350

Note: ** coordinates are in WGS 1984 Zone 12.*